Appendix 2 Mine Waste Characterization Documentation and Results

(all Attachments on DVD)

APPENDIX 2

MINE WASTE CHARACTERIZATION

As described in Section 10 of the PTM, over the last ten years, PolyMet has conducted a MDNR-supervised mine waste characterization program to support the Project. This Appendix 2 presents additional information relating to the mine waste characterization program and the results of that program to further demonstrate compliance with the nonferrous mining regulations and support PolyMet's PTM application for the Project. This Appendix provides additional detail regarding the process of developing the mine waste characterization program and the contents of the mine waste characterization program. The attachments to this Appendix include additional documentation of the mine waste characterization program and results. These documents have already been provided to MDNR as part of the collaborative development and implementation of the mine waste characterization program between PolyMet, its consultants, and MDNR, but are being provided as part of, and are incorporated into, the PTM application in order to facilitate MDNR's review of PolyMet's waste characterization program.

2.1. Development of the Mine Waste Characterization Program

Minnesota law requires that an applicant for a PTM meet with the MDNR "to outline chemical and mineralogical analyses and laboratory tests to be conducted for mine waste characterization," which the MDNR will then use in "evaluation of the applicant's mining and reclamation plan" Minnesota Rules, part 6132.1000, subpart 1. In addition, the MDNR must approve the persons conducting the mine waste characterization and these persons must have "demonstrated proficiency in such analysis," Minnesota Rules, part 6132.1000, subpart 2.

In accordance with these regulatory requirements, in 2004, PolyMet initiated a series of characterization conferences and other exchanges with the Lands and Minerals Division of MDNR to develop a mine waste characterization program for the Project. These interactions included PolyMet's consultants, SRK Consulting (SRK) and Barr Engineering Co. (Barr), both of which had demonstrated proficiency in conducting mine waste characterization.[1]

PolyMet's technical experts, including SRK and Barr, worked directly with MDNR from 2004 to 2010 to develop the geochemical characterization plans that served as the primary documents outlining the parameters of the waste characterization program for the Project:

- The Waste Rock and Lean Ore Geochemical Characterization Plan ("Waste Rock Plan"),
- The Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan("Flotation Tailings and Hydromet Plan"),
- The Overburden Geochemical Characterization Plan ("Overburden Plan"), and
- Other targeted studies and analyses to support the waste characterization program (collectively, Characterization Plans).

[1] PolyMet first notified MDNR of its intent to utilize SRK to work with MDNR to develop the mine waste characterization program at a meeting in late 2004.

The individuals who participated in the development of the Characterization Plans are qualified specialists, proficient in their fields. The primary individuals that participated in the conferences and other exchanges and contributed to the Characterization Plans, including representatives from MDNR, are identified in the Characterization Plans. Additionally, other entities that supported the implementation of the waste characterization program, such as laboratories used to perform the analyses, are identified in the Characterization Plans and were subject to the review and approval of MDNR. *See* Attachment 2-3 Waste Rock Plan at §§1.5, 1.6; Attachment 2-4 Flotation Tailings and Hydromet Plan at §§1.5, 1.6; Attachment 2-6 to 2-9 Overburden Plan at §§5.1.5, 5.2. Following development of the Characterization Plans, PolyMet and its consultants, SRK and Barr, continued to work directly with MDNR throughout implementation of the Characterization Plans.

Development of the Characterization Plans was an iterative process with MDNR in which MDNR reviewed the Characterization Plans and evaluated results of preliminary sampling and testing, requested that PolyMet provide supplemental information in certain instances, provided PolyMet with comments and requested revisions to the Characterization Plans, and provided PolyMet with further direction on sampling and testwork. PolyMet addressed MDNR's concerns, and MDNR approved the Characterization Plans and continued to provide PolyMet direction throughout implementation of the waste characterization program. An overview of the timeline and steps involved in this iterative process for each of the Characterization Plans is provided below, with references to supporting documentation, as applicable.

2.1.1. Overview of Consultation with MDNR on the Characterization Plans

Waste Rock and Lean Ore Geochemical Characterization Plan

- **Preliminary Meetings (2004).** A series of meetings with MDNR were held in the second half of 2004 during which PolyMet[2] and MDNR had preliminary discussions related to mine waste management and mine waste characterization. Based on the information exchanged during the preliminary meetings, SRK developed a draft work plan for geochemical characterization, which was submitted to MDNR and other state and federal agency officials. PolyMet also met with these officials to present and discuss the draft work plan. At this meeting, PolyMet notified MDNR that it intended to use SRK to work to develop and finalize the Characterization Plans.

- **Plan Development (2005 – 2006).** PolyMet and MDNR worked together to develop the Waste Rock Plan. Through a series of meetings, teleconferences, and correspondence, MDNR provided comments to PolyMet's proposed Waste Rock Plan, and PolyMet responded to MDNR comments and provided additional information when requested. MDNR and PolyMet developed a sample selection matrix and testwork design, Attachment 2-1. PolyMet submitted the final Waste Rock Plan in May of 2006. Attachment 2-3.

- **Implementation, Adaptation, and Reporting (2005 – 2015).** Implementation of initial testing began as early as mid-2005, as the final details of the Waste Rock Plan

[2] References to PolyMet in these consultation timelines refer to PolyMet employees or their consultants or representatives.

were being finalized. At this time, PolyMet and MDNR also developed outlines to serve as templates for the progress reports to be submitted by PolyMet regarding the work completed under the Waste Rock and other Characterization Plans. PolyMet submitted the first of multiple progress reports regarding the work completed under the Characterization Plans to MDNR in mid-2006, and PolyMet continued to submit additional progress reports and/or updates to progress reports through 2014. During this time, PolyMet and MDNR continued to evaluate and modify the kinetic testing portion of the waste characterization program as testing continued during development of the FEIS. *See* Attachments 2-15 to 2-20.

Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan

- **Plan Development (2005 – 2006).** Building on the work completed in the agency consultation on the Waste Rock Plan, PolyMet and MDNR began to work on the Flotation Tailings and Hydromet Plan in the second half of 2005. As with the Waste Rock Plan, PolyMet and MDNR held a series of meetings, and exchanged comments and responses on drafts of the Flotation Tailings and Hydromet Plan. PolyMet also consulted MDNR on the design of preliminary testing used to inform plan development, and on the sample analysis parameters for the plan. The final version of the Flotation Tailings and Hydromet Plan was submitted concurrently with the final Waste Rock Plan. Attachment 2-4.

- **Implementation and Reporting (2005 – 2015).** PolyMet began pilot plant testing to support the Flotation Tailings and Hydromet Plan in mid-2005, and submitted results of that testing beginning in early 2006. Attachment 2-2. As the Flotation Tailings and Hydromet Plan was being finalized, PolyMet and MDNR also developed outlines to serve as templates for the progress reports to be submitted by PolyMet regarding the work completed under the Flotation Tailings and Hydromet Plan and other Characterization Plans. PolyMet submitted the first of multiple progress reports regarding the work completed under the Characterization Plans to MDNR in mid-2006, and PolyMet continued to submit additional progress reports and/or updates to progress reports through 2015.

Overburden Geochemical Characterization Plan

- **Plan Development & Supplementation (2007 – 2008, 2010).** In the second half of 2007, MDNR requested that PolyMet also prepare a plan for characterizing overburden material. Attachment 2-5. PolyMet consulted with MDNR on development of the Overburden Plan and supporting analytical testing through meetings and exchanges of comments and responses. PolyMet and MDNR agreed initially to develop a plan sufficient to support environmental review. PolyMet submitted the initial Overburden Plan sampling and analytical plans to MDNR in early 2008. Attachment 2-6; Attachment 2-7. Following on this initial effort, PolyMet and MDNR subsequently coordinated on supplementing the Overburden Plan with a sump soil sampling program to collect additional data on the geochemical characteristics of unsaturated overburden for the purposes of supporting project permitting. As a result of this collaboration,

PolyMet provided an updated Overburden Sampling Plan, Attachment 2-8 and an updated Overburden Sampling Analysis Plan to MDNR in early 2010. Attachment 2-9.

- **Implementation, Adaptation, and Reporting (2008 – 2015).** PolyMet began implementation of the initial testing and analysis under the Overburden Plan in 2008, and submitted initial drilling program results in October 2008, Attachment 2-21, and initial pebble chemical analysis results in mid-2009. Attachment 2-22. Following the 2010 supplementation of the Overburden Plan, PolyMet submitted additional characterization results of the sump soil sampling in late 2010. Attachment 2-23.

2.2. Contents of the Mine Waste Characterization Program

Consistent with the regulatory requirements, PolyMet's mine waste characterization program is "based on chemical, physical, and mineralogical analyses and laboratory tests of material generated by exploration, preproduction sampling, and process testing," Minnesota Rules, part 6132.1000, subpart 2. The mine waste characterization program includes: (1) chemical analysis of mine waste; (2) mineralogical and petrological analysis of mine waste; and (3) laboratory tests describing acid generation and dissolved solids released from mine waste.

The design and analyses identified in the Characterization Plans to fulfill the regulatory requirements identified above are summarized below. Throughout implementation, PolyMet and MDNR continued to evaluate the mine waste characterization program data needs, and adjusted or supplemented the analyses as needed to ensure the data collected were sufficient to inform the FEIS, Project design, and management plans.

- Waste Rock and Lean Ore Geochemical Characterization Plan. 58 samples of reactive and non-reactive waste rock, and 24 samples of lean ore were identified using a sampling matrix developed in collaboration with MDNR. In addition, three ore composite samples were selected as part of the same characterization program. These samples were then analyzed using chemical, physical, mineralogical, and laboratory tests, including:

    - optical mineralogical and petrological characterization: reporting includes mineral identification, mineral abundance, grain sizes, and petrologic determination;

    - sub-optical mineral characterization: trace element content of major minerals via electron microprobe analysis;

    - acid-base accounting: total sulfur content, carbonate content, and paste pH;

    - bulk chemical composition of whole rock samples: ICP analyses of 27 elements following four-acid digestion, ICP analyses of 34 elements following aqua regia digestion; whole rock oxides;

    - bulk chemical composition of size-fractionated samples: total sulfur and ICP analyses of 27 elements following four acid digestion;

    - specific gravity (G_s) and particle size distribution; and

- "kinetic" testing: whole rock samples were subjected to humidity cell testing (ASTM Procedure D 5744); size-fractionated portions of splits from five of the whole rock samples were also tested using a laboratory test method developed by the MDNR ("MDNR Reactor" experiments).

See Attachment 2-3, Waste Rock Plan at §4 for additional information.

In addition to the detailed characterization of these 85 samples, the waste rock and ore characterization includes an evaluation of bulk chemical analyses from approximately 18,800 drill core samples. These chemical analyses are used to identify release rates for select chemical constituents, the procedure for which was presented by PolyMet for MDNR review during meetings in early 2011. The total sulfur analyses are used to estimate sulfur content of waste rock that will be produced from the Project, which is the primary driver for rock management. Both of these usages of the drill core chemical data are further described in Attachment 2-24.

- Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan: Ore composites were selected for pilot-plant processing to produce flotation tailings for characterization. The flotation tailings were then analyzed using the following methods:

  - density determinations and size fraction analysis;

  - mineralogical characterization via optical analyses of tailings thin sections;

  - acid-base accounting: total sulfur and sulfur speciation, paste pH, neutralization potential and carbonate content;

  - bulk chemical composition of whole rock samples: ICP analyses of 50 elements following aqua regia digestion; whole rock oxides; and

  - "kinetic" testing: whole rock samples were subjected to humidity cell testing (ASTM Procedure D 5744); in addition, splits from samples were also tested using a laboratory test method developed by the MDNR ("MDNR Reactor" experiments).

  Additionally, hydrometallurgical residue samples were analyzed by the following methods:

  - mineralogical characterization via quantitative X-ray diffraction to identify major and minor crystalline phases and estimate relative abundance of each;

  - acid-base accounting: total sulfur and sulfur speciation, paste pH, neutralization potential and carbonate content;

  - bulk chemical composition of whole rock samples: ICP analyses of 50 elements following aqua regia digestion and whole rock oxides;

- leachate characterization: chemistry of leachate measured after samples are subjected to TCLP, SPLP, and a sequential shake flask leach procedure; and

- "kinetic" testing: samples were subjected to humidity cell testing (ASTM Procedure D 5744); in addition, splits from samples were also tested using a laboratory test method developed by the MDNR ("MDNR Reactor" experiments).

See Attachment 2-4, Flotation Tailings and Hydromet Plan at §3.2-3.3.

- Overburden Characterization Plan: The Overburden Characterization Plan initially identified 16 drilling locations for collection of samples to be analyzed as follows:

  - Physical analysis including: Atterberg limits, grain size, consolidation, permeability, and strength;

  - acid-base accounting: total sulfur content, sulfur speciation, carbonate content, neutralization potential, and paste pH;

  - bulk chemical composition of size-fractionated samples: ICP analyses of 51 elements following aqua regia digestion and ICP analyses of 33 elements following 4-acid digestion;

  - Meteoric Water Mobility Procedure: a standard laboratory test to measure metal leaching; and

  - petrologic characterization via a pebble counting procedure to identify lithologies present.

  After additional access to drilling sites was permitted in 2010, PolyMet collected additional samples of unsaturated overburden for evaluation using the following analyses:

  - Rinse test for pH, oxidation/reduction potential, and specific conductivity;

  - Particle size distribution;

  - Meteoric Water Mobility Procedure: a standard laboratory test to measure metal leaching;

  - bulk chemical composition of samples: ICP analyses of 51 elements following aqua regia digestion and ICP analyses of 44 elements following digestion in 1N nitric acid;

  - Moisture content; and

  - acid-base accounting: total sulfur content, sulfur speciation, carbonate content, neutralization potential, and paste pH.

  See Attachments 2-6 through 2-9.

In addition, PolyMet's mine waste characterization program provides information regarding the reagents associated with tailings and leach residue, including their chemical composition, mass of chemical used, and, where applicable, the degradation and transport characteristics as well as the effects on mineral dissolution of those reagents. The list of reagents associated with the hydrometallurgical and beneficiation processes are provided in Table 8-2 and Table 8-3 of the Application, which includes the chemical composition and mass to be used for each. Only one of these reagents – copper sulfate – is expected to contribute to metal and sulfate concentrations. Process testwork showed that sulfur concentrations in the tailings can be expected to vary in response to changes in process conditions, and that the use of copper sulfate can be expected to lower sulfur content in tailings. Because PolyMet determined to proceed with the use of copper sulfate in the flotation process, MDNR and PolyMet agreed that characterization testwork would primarily be performed on the residues generated by leaching of the concentrate that was produced in the pilot plant with the use of copper sulfate, but both leach residues generated with and without the use of copper sulfate were tested for evaluation of the effects of copper sulfate. The results of this testing, including information regarding the degradation and transport characteristics, and the effects of copper sulfate on mineral dissolution, are provided in Attachments 2-2, 2-11, 2-12, and 2-14.

2.3. Mine Waste Characterization Program Implementation and Results

As noted in Section 2.1 above, PolyMet began implementing the analyses and studies that comprise the mine waste characterization program as early as 2005. As noted in the timeline above, the process of developing the Characterization Plans with MDNR also involved developing outlines for reporting the results of the Characterization Plans. These outlines were finalized in 2006, and were followed in preparing progress reports on the results of the work completed under the Characterization Plans. PolyMet submitted a series of these progress reports and data submittals on the Characterization Plan results beginning in 2006, as work was completed, to facilitate an iterative process between PolyMet and MDNR in which ongoing data needs regarding waste characterization were evaluated and adjusted. This iterative process continued throughout the mine plan development and environmental review processes, and culminated in the submittal of a consolidated data report, the Waste Characterization Data Package, first submitted to MDNR on February 14, 2011. PolyMet continued to update the waste characterization data, and provided MDNR with twelve revisions of the Waste Characterization Data Package, the most recent of which was submitted to MDNR on February 22, 2015. Attachment 2-24. PolyMet used the data provided in these reports and submittals to develop additional studies, plans, and models for the Project, including various management plans.

The results of the mine waste characterization program have been submitted by PolyMet to the MDNR, MPCA, and USEPA as part of the EIS process. As required by Minnesota Rules, part 6132.1000, subpart 3, PolyMet is resubmitting these results as part of the Mining and Reclamation Plan in its Application.

LIST OF ATTACHMENTS – PROVIDED ELECTRONICALLY

Attachment 2-1 SRK Consulting. Sample Selection for Humidity Cells, NorthMet Project. Memorandum to Jennifer Engstrom, MDNR. November 24, 2005.

Attachment 2-2 SRK Consulting. Tailings and Hydromet Residue Testork – Update on Sample Selection from 24 Hour Testwork. Memorandum to Jennifer Engstrom, MDNR. January 6, 2006.

Attachment 2-3 SRK Consulting. Waste Rock and Lean Ore Geochemical Characterization Plan, NorthMet Project, Minnesota. May 2006.

Attachment 2-4 SRK Consulting. Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan, NorthMet Project, Minnesota. May 2006.

Attachment 2-5 MDNR. Letter from Jennifer Engstrom, MDNR, to Jim Scott, PolyMet Mining. August 13, 2007.

Attachment 2-6 SRK Consulting. Overburden Geochemical Characterization Plan in Support of EIS – DRAFT, NorthMet Project. Memorandum to Stuart Arkley, MDNR. February 22, 2008.

Attachment 2-7 SRK Consulting. Analysis of Samples from Overburden Drilling Program – DRAFT, NorthMet Project. Memorandum to Stuart Arkley, MDNR. March 8, 2008.

Attachment 2-8 Kearney, C. PolyMet Overburden Sampling Plan. Email to Stuart Arkley and Jennifer Engstrom, MDNR. February 26, 2010.

Attachment 2-9 Kearney, C. PolyMet Unsaturated Overburden Sampling Analysis Plan. Email to Stuart Arkley and Jennifer Engstrom, MDNR. March 3, 2010 (updated March 4, 2010).

Attachment 2-10 SRK Consulting. RS53A – Waste Rock Characterization Progress Report – DRAFT, NorthMet Project. May 2006.

Attachment 2-11 SRK Consulting. RS33A – Reactive Residues Progress Report – NorthMet Project. May 2006.

Attachment 2-12 SRK Consulting. RS33/RS65 - Hydrometallurgical Residue Characterization and Water Quality Model – DRAFT, NorthMet Project. February 21, 2007.

Attachment 2-13 SRK Consulting. RS53/RS42 - Waste Rock Characteristics/Waste Water Quality Modeling - Waste Rock and Lean Ore – NorthMet Project - DRAFT. March 9, 2007.

Attachment 2-14 SRK Consulting. RS54/RS46 – Waste Water Modeling - Tailings – NorthMet Project - DRAFT. July 20, 2007.

Attachment 2-15 SRK Consulting. RS82 - Update on Use of Kinetic Test Data for Water Quality Predictions – DRAFT 02. Memorandum to Kim Lapakko, MDNR. February 2, 2009.

Attachment 2-16 MDNR. Recommended Kinetic Test Modifications for PolyMet's NorthMet Project. Memorandum to PolyMet. April 22, 2009 (updated May 11, 2009).

Attachment 2-17 SRK Consulting. pH Criteria for Modification to Kinetic Test Program - NorthMet Project. Memorandum to Jim Scott, PolyMet Mining. April 28, 2009.

Attachment 2-18 SRK Consulting. Update on Tailings Humidity Cell Test Data – NorthMet Project – DRAFT. Memorandum to Peter Hinck, Barr Engineering. December 23, 2011.

Attachment 2-19 SRK Consulting. Update on Kinetic Test Data, NorthMet Project – DRAFT. Memorandum to Jim Scott, PolyMet Mining. October 19, 2012 (updated January 25, 2015).

Attachment 2-20 SRK Consulting. NorthMet Project Kinetic Test Program Status and Recommendations. Memorandum to Jim Scott, PolyMet. March 17, 2014.

Attachment 2-21 SRK Consulting. Results of Analysis from Overburden Drilling Program – DRAFT, NorthMet Project. Memorandum to Stuart Arkley, MDNR. October 16, 2008.

Attachment 2-22 SRK Consulting. Overburden Pebble Chemical Analysis – DRAFT. Memorandum to Jim Scott, PolyMet. June 25, 2009.

Attachment 2-23 SRK Consulting. Results of Analysis from Overburden Drilling Program – Update for March 2010 Test Pit Program – DRAFT, NorthMet Project. Memorandum to Stuart Arkley, MDNR. August 13, 2010.

Attachment 2-24 PolyMet. NorthMet Project Waste Characterization Data Package. Version 12. February 13, 2015.

Attachment 2-25 Industrial Chemical Additives (Beneficiation and Hydromet) 3-13-2015.

Attachment 2-1

SRK Consulting. Sample Selection for Humidity Cells, NorthMet Project. Memorandum to Jennifer Engstrom, MDNR. November 24, 2005.

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Memo

To:	Jennifer Engstrom, MDNR	**Date:**	November 24, 2005
cc:	Paul Eger, MDNR Kim Lapakko, MDNR Jim Scott, PolyMet	**From:**	Stephen Day
Subject:	NorthMet Project Sample Selections for Humidity Cells	**Project #:**	1UP005.001

Jennifer,

This memorandum provides a final version of the humidity cell sample characteristics. I plan to incorporate this into the Waste Rock and Lean Ore Plan (if desired) but have provided this separately for convenience.

I attach two tables showing non-reactive waste rock, reactive waste rock and lean ore samples selected for testing in humidity cells. Table 1 is a listing of the sample identification numbers with their corresponding units, rock types and chemical characteristics. The 93 humidity cells (including method blank and duplicates) are currently underway.

Table 2 shows the matrix used for sample selections. It shows the target sulfur concentrations for each percentile, the actual sulfur concentrations in the sample chosen to represent the indicated sulfur percentiles and the ratio of copper and nickel to sulphur (by weight) for the same samples. The sulfur concentrations in Table 2 are from Table 1.

1 Waste Rock and Lean Ore Selections

The 79 samples (25 non-reactive, 32 reactive, 22 lean ore) represent several iterations of sample selection, analysis and re-selection. Samples are being tested in humidity cells for all but one of the DNR's requested and optional samples. A suitable sample for the P95 sulfur concentration (6.06%) in the Virginia Formation could not be located.

As described previously, some deviation from the target sulfur concentrations was expected because (a) samples were selected using weighted average analyses in PolyMet's drill core sample database, (b) suitable samples were not always available; and (c) the actual analyses combine compositing and analytical errors for both the original database values and composite analytical results. Deviations from target values were calculated from:

$$\text{Deviation (\%)} = (S_{Actual} - S_{Target})/S_{Target}).100$$

The resulting deviations (Table 2) show:

- For non-reactive samples:

  - Deviations of 100% are common but they usually represent absolute differences of less than or equal to 0.02% on sulfur concentrations below 0.05% and close to the method detection limit of 0.01%.
  - Two samples (8% of non-reactive samples) with target concentrations of 0.01% had deviations exceeding 100%.

- For reactive samples:
  - 84% of samples selected had deviations below 50%.
  - For four samples (13%), the deviation was between 50 and 100% and only one sample had a deviation of 200%. This latter sample was classified as "optional" in the selections.

- For lean ore samples, only one sample (less than 5% of samples) showed a deviation of 50%.

Based on this low overall incidence of deviation from target values and the large size of the dataset, no further sample selections are necessary for waste rock and lean ore.

2 Deferred Ore

As shown in Table 2, the three ore composite used for metallurgical testing are being tested in humidity cells 66, 67, and 73. These samples contain sulfur concentrations (0.9%) that are bracketed by reactive and lean ore samples being tested, nickel concentrations that are slightly above lean ore grades, and copper concentrations that are above lean ore grades. The copper and nickel to sulfur ratios shown by the ore samples are contained within the range of lean ore samples. Since deferred ore will contain nickel and copper concentrations between ore and lean ore, and sulfur concentrations straddle reactive waste rock and lean ore, the existing distribution of waste rock, lean ore and ore samples will provide sufficient data for characterization of runoff chemistry for temporary deferred ore stockpiles.

No additional sample selection or characterization is needed for deferred ore.

3 Conclusion

The sample distribution for non-reactive waste rock, reactive waste rock, lean ore and ore will provide a more than adequate database for characterization of these materials. No further sampling is proposed for these rock categories or deferred ore.

Attached:
Table 1: Characteristics of Samples in Humidity Cells
Table 2: Summary of Sample Selected for Testing in Humidity Cells for Non-Reactive Waste Rock, Waste Rock and Lean Ore, November 24, 2005.

Table 1
Characteristics of Samples in Humidity Cells

HCT ID	Comment	HCT Full ID (Drill Hole, Footage, HCT)	Waste Type	Geological Unit	Rock Type	Target S Percentile or Level	4-Acid Cu	4-Acid Ni	Leco S	Cu/S	Ni/S
							%	%	%	wt/wt	wt/wt
1		DDH-99-320C(830-850)-1	Reactive	1	Anorthositic	P25	0.0365	0.0174	0.09	0.41	0.19
2		DDH-00-361C(310-320)-2	Reactive	1	Anorthositic	P50	0.0152	0.0208	0.18	0.08	0.12
3		DDH-00-361C(345-350)-3	Reactive	1	Anorthositic	P75	0.0236	0.0276	0.05	0.47	0.55
4		DDH-00-343C(240-250)-4	Reactive	1	Anorthositic	P95	0.0759	0.0199	0.68	0.11	0.03
5		DDH-26030(1047-1052)-5	Reactive	1	Sedimentary Hornfels	P10	0.0098	0.0104	0.24	0.04	0.04
6		DDH-26061(1218-1233)-6	Reactive	1	Sedimentary Hornfels	P25	0.0146	0.0122	0.44	0.03	0.03
7		DDH-00-340C(990-995)-7	Reactive	1	Sedimentary Hornfels	P50	0.0142	0.0144	0.55	0.03	0.03
8		DDH-00-340C(965-974.5)-8	Reactive	1	Sedimentary Hornfels	P75	0.0198	0.012	1.74	0.01	0.01
9	1 Dup	DDH-99-320C(830-850)-9D	Reactive	1	Anorthositic	P25	0.0365	0.0174	0.09	0.41	0.19
10		DDH-00-340C(765-780)-10D	Reactive	1	Troctolitic	P100	0.0707	0.0304	1.68	0.04	0.02
11		DDH-26043&26027(1501&740-1506&745)-11	Reactive	1	Sedimentary Hornfels	P85	0.04	0.02	2.47	0.02	0.01
12	Blank	Method Blank - 12	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
13		DDH-00-340C(595-615)-13	Non-reactive	1	Troctolitic	NR-0.03	0.0141	0.0365	0.04	0.35	0.91
14		DDH-00-334C(580-600)-14	Non-reactive	1	Troctolitic	NR-0.05	0.0278	0.0319	0.06	0.46	0.53
15		DDH-00-334C(640-660)-15	Reactive	1	Troctolitic	P25	0.0311	0.0497	0.07	0.44	0.71
16		DDH-00-347C(795-815)-16	Reactive	1	Troctolitic	P50	0.0446	0.0389	0.07	0.64	0.56
17		DDH-00-350C(580-600)-17	Reactive	1	Troctolitic	P80	0.0423	0.0298	0.19	0.22	0.16
18		DDH-00-327C(225-245)-18	Reactive	1	Troctolitic	P90	0.0324	0.0156	0.44	0.07	0.04
19		DDH-00-371C(435-440)-19	Reactive	1	Troctolitic	P95	0.0563	0.0267	0.88	0.06	0.03
20	10 Dup	DDH-00-340C(765-780)-20	Reactive	1	Troctolitic	P100	0.0707	0.0304	1.68	0.04	0.02
21		DDH-00357C(335-340)-21	Reactive	1	Ultramafic	P25	0.0269	0.0255	0.08	0.34	0.32
22		DDH-00326C(680-685)-22	Reactive	1	Ultramafic	P80	0.105	0.023	0.30	0.35	0.08
23		DDH-00-357C(535-540)-23	Reactive	1	Ultramafic	P85	0.1015	0.0257	0.2	0.51	0.13
24		DDH-99-318C(725-735)-24	Reactive	1	Ultramafic	P90	0.0534	0.0196	0.72	0.07	0.03
25		DDH-99-317C(460-470)-25	Reactive	1	Ultramafic	P95	0.0668	0.0224	1.24	0.05	0.02
26		DDH-00-366C(185-205)-26	Non-reactive	2	Anorthositic	NR-0.01	0.0082	0.0208	0.02	0.41	1.04
27		DDH-00-366C(230-240)-27	Non-reactive	2	Anorthositic	NR-0.03	0.0125	0.0171	0.02	0.63	0.86
28		DDH-99-320C(165-175)-28	Non-reactive	2	Anorthositic	NR-0.05	0.014	0.025	0.03	0.47	0.83
29		DDH-99-318C(250-370)-29	Non-reactive	2	Troctolitic	NR-0.01	0.0139	0.0311	0.04	0.35	0.78
30		DDH-00-373C(95-115)-30	Non-reactive	2	Troctolitic	NR-0.03	0.0226	0.0477	0.04	0.57	1.19
31		DDH-00-373C(75-95)-31	Non-reactive	2	Troctolitic	NR-0.05	0.0286	0.0432	0.06	0.48	0.72
32		DDH-00-357C(110-130)-32	Reactive	2	Troctolitic	P50	0.0427	0.0356	0.08	0.53	0.45
33		DDH-99-320C(315-330)-33	Reactive	2	Troctolitic	P80	0.0355	0.0213	0.07	0.51	0.30
34		DDH-00-369C(335-345)-34	Reactive	2	Troctolitic	P95	0.0433	0.0207	0.18	0.24	0.12
35		DDH-00-368C(460-465)-35	Non-reactive	2	Ultramafic	NR-0.01	0.0413	0.0535	0.06	0.69	0.89
36		DDH-26055(940-945)-36	Non-reactive	2	Ultramafic	NR-0.03	0.0321	0.0495	0.06	0.54	0.83
37		DDH-00-368C(125-145)-37D	Non-reactive	3	Anorthositic	NR-0.03	0.0182	0.0201	0.04	0.46	0.50
38		DDH-00-369C(305-325)-38D	Reactive	3	Troctolitic	P95	0.0349	0.0299	0.25	0.14	0.12
39		DDH-26098&00-337C(145&105-148.5&110)-1	Non-reactive	2	Ultramafic	NR-0.05	0.023	0.055	0.1	0.23	0.55
40		DDH-00-334C(30-50)-40	Non-reactive	3	Anorthositic	NR-0.01	0.0085	0.0262	0.02	0.43	1.31
41	37 Dup	DDH-00-368C(125-145)-41	Non-reactive	3	Anorthositic	NR-0.03	0.0182	0.0201	0.04	0.46	0.50
42		DDH-00-368C(20-40)-42	Non-reactive	3	Anorthositic	NR-0.05	0.022	0.0108	0.04	0.55	0.27
43		DDH-00-366C(35-55)-43	Non-reactive	3	Troctolitic	NR-0.01	0.0054	0.0238	0.02	0.27	1.19
44		DDH-00-334C(110-130)-44	Non-reactive	3	Troctolitic	NR-0.03	0.0135	0.0363	0.04	0.34	0.91
45		DDH-00-347C(155-175)-45	Non-reactive	3	Troctolitic	NR-0.05	0.0162	0.018	0.06	0.27	0.30
46		DDH-00-347C(280-300)-46	Reactive	3	Troctolitic	P50	0.0355	0.0211	0.06	0.59	0.35
47		DDH-00-326C(60-70)-47	Reactive	3	Troctolitic	P85	0.043	0.0413	0.14	0.31	0.30
48	38 Dup	DDH-00-369C(305-325)-48	Reactive	3	Troctolitic	P95	0.0349	0.0299	0.25	0.14	0.12
49		DDH-00-367C(50-65)-49	Non-reactive	4	Troctolitic	NR-0.01	0.0108	0.0192	0.03	0.36	0.64
50		DDH-00-367C(260-280)-50	Non-reactive	4	Troctolitic	NR-0.03	0.018	0.0254	0.04	0.45	0.64
51		DDH-00-367C(290-310)-51	Non-reactive	4	Troctolitic	NR-0.05	0.0198	0.0249	0.04	0.50	0.62
52		DDH-00-370C(20-30)-52	Reactive	4	Troctolitic	P25	0.0177	0.0137	0.08	0.22	0.17
53		DDH-00-369C(20-30)-53	Reactive	4	Troctolitic	P75	0.0577	0.0303	0.21	0.27	0.14
54		DDH-00-367C(170-175)-54	Reactive	4	Troctolitic	P90	0.0331	0.0267	0.51	0.06	0.05
55		DDH-00-367C(395-400)-55	Reactive	4	Troctolitic	P95	0.112	0.0363	0.77	0.15	0.05
56		DDH-26064(44-54)-56	Non-reactive	5	Troctolitic	NR-0.01	0.0104	0.0426	0.02	0.52	2.13
57	51 Dup	DDH-00-367C(290-310)-57D	Non-reactive	4	Troctolitic	NR-0.05	0.0198	0.0249	0.04	0.50	0.62
58		DDH-00-364C(210-229)-58D	Reactive	20	Virginia	P75	0.0155	0.0186	3.79	0.00	0.00
59		DDH-26064(264&146-269&156)-59	Non-reactive	5	Troctolitic	NR-0.05	0.027	0.029	0.06	0.45	0.48
60		DDH-26056(110-125)-60	Non-reactive	6	Troctolitic	NR-0.01	0.0293	0.0401	0.04	0.73	1.00
61		DDH-00-361C(240-245)-61D	Lean Ore	2	Ultramafic	P95	0.0196	0.1075	0.06	0.33	1.79
62		DDH-00-361C(737-749)-62	Reactive	20	Virginia	P25	0.0152	0.0121	2	0.01	0.01
63	58 Dup	DDH-00-364C(210-229)-63	Reactive	20	Virginia	P75	0.0155	0.0186	3.79	0.00	0.00
64		DDH-00-337C(510-520)-64	Reactive	20	Virginia	P90	0.0198	0.0168	5.68	0.00	0.00
65		DDH-26027(616-626)-65	Lean Ore	1	Anorthositic	P95	0.126	0.0528	1.83	0.07	0.03
66		P1-0-66	Ore	#N/A	#N/A	#N/A	0.345	0.0971	0.86	0.40	0.11
67		P2-0-67	Ore	#N/A	#N/A	#N/A	0.349	0.1485	0.9	0.39	0.17
68		DDH-26062&26026(993&565-998&568)-68	Lean Ore	1	Sedimentary Hornfels	P95	0.129	0.047	4.46	0.03	0.01
69		DDH-00-340C(725-745)-69	Lean Ore	1	Troctolitic	P95	0.1005	0.0407	0.91	0.11	0.04
70		DDH-00-344C(515-520)-70	Lean Ore	1	Ultramafic	P95	0.0899	0.0529	1.2	0.07	0.04
71		DDH-00-340C(380-390)-71	Lean Ore	2	Troctolitic	P95	0.0502	0.0342	0.15	0.33	0.23
72	61 Dup	DDH-00-361C(240-245)-72	Lean Ore	2	Ultramafic	P95	0.0196	0.1075	0.06	0.33	1.79
73		P30-73	Ore	#N/A	#N/A	#N/A	0.444	0.0972	0.86	0.52	0.11
74		26029(interval 815-825)-74	Non-reactive	1	Troctolitic	NR-0.01	0.0055	0.0114	0.02	0.28	0.57
75		DDH-26030&26049(291&358-296&362)-75	Lean Ore	3	Troctolitic	P95	0.174	0.071	0.59	0.29	0.12
76		DDH-00-367C(400-405)-76	Lean Ore	4	Troctolitic	P95	0.1	0.0299	1.37	0.07	0.02
77		DDH-26056(302-312)-77	Lean Ore	5	Troctolitic	P50	0.0866	0.0365	0.23	0.38	0.16
78		26056(135-153)-78	Non-reactive	6	Troctolitic	NR-0.03	0.0389	0.0439	0.05	0.78	0.88
80		DDH-26142(360&345-365&350)-80	Lean Ore	6	Troctolitic	P95	0.115	0.048	0.18	0.64	0.27
93		00-331C(255-260)-93	Lean Ore	1	Anorthositic	P75	0.153	0.043	0.86	0.18	0.05
94		00-344C (630-635)-94	Lean Ore	1	Ultramafic	P75	0.155	0.055	0.34	0.46	0.16
95		00-326C(495-505)-95	Lean Ore	1	Ultramafic	P50	0.077	0.065	0.16	0.48	0.41
96		99-318C(325-330)-96	Lean Ore	2	Troctolitic	P85	0.117	0.054	0.17	0.69	0.32
97		00-330C(275-280)-97	Lean Ore	1	Ultramafic	P85	0.148	0.069	0.75	0.20	0.09
98		26056(282-292)-98	Lean Ore	5	Troctolitic	P95	0.141	0.047	0.32	0.44	0.15
99		00-326C (250-265)-99	Lean Ore	1	Troctolitic	P25	0.052	0.034	0.08	0.65	0.43
100		00-340C(910-925)-100	Lean Ore	1	Troctolitic	P50	0.174	0.041	0.36	0.48	0.11
101		26039(310-315)-101	Lean Ore	2	Ultramafic	P95	0.039	0.095	0.06	0.65	1.58
102		00-331C(190-210)-102	Lean Ore	1	Troctolitic	P85	0.175	0.046	0.42	0.42	0.11
103		99-320C(400-405)-103	Lean Ore	1	Anorthositic	P50	0.1	0.068	0.18	0.56	0.38
104		00-326C(225-235)-104	Lean Ore	2	Ultramafic	P80	0.094	0.071	0.12	0.78	0.59
105		00-367C(495-500)-105	Lean Ore	3	Troctolitic	P75	0.193	0.051	0.28	0.69	0.18
106		26058(704-715)-106	Lean Ore	1	Sedimentary Hornfels	P50	0.083	0.03	1.46	0.06	0.02

Table 2
Summary of Sample Selected for Testing in Humidity Cells for Non-Reactive Waste Rock, Waste Rock and Lean Ore
November 24, 2005

Unit	Rock Type	Parameter		Unit Tonnages			Percentiles for Sulfur Contents and Metal to Sulfur Ratios in Test Materials																			
				Non-Reactive	Reactive	Lean ore	Non-reactive			Reactive										Lean Ore							
				M. tons	M. tons	M. tons				P10	P25	P50	P75	P80	P85	P90	P95	P100	P10	P25	P50	P75	P80	P85	P90	P95
1	**Anorthositic**	**Target**	**Total S, %**	0.57	0.99	0.98				0.08	0.1	0.15	0.29				1.09	1.09	0.11	0.26	0.36	0.93				***1.95***
		Characteristics of Test Materials	Total S, %								0.09	0.18	0.05				0.68				0.18	0.86				1.83
			Cu/S, wt/wt								0.41	0.08	0.47				0.11				0.56	0.18				0.07
			Ni/S, wt/wt								0.19	0.12	0.55				0.03				0.38	0.05				0.03
	Sedimentary horr	**Target**	**Total S, %**	0	1.6	0.88				**0.08**	0.35	0.69	2.2	2.32	2.81	3.38	3.5	3.78	0.34	1.37	1.58	1.76				***4.91***
		Characteristics of Test Materials	Total S, %							0.24	0.44	0.55	1.74		2.47						1.46					4.46
			Cu/S, wt/wt							0.04	0.03	0.03	0.01		0.02						0.06					0.03
			Ni/S, wt/wt							0.04	0.03	0.03	0.01		0.01						0.02					0.01
	Troctolitic	**Target**	**Total S, %**	17.2	40.1	35.6	0.01	0.03	0.05	0.06	**0.07**	0.1	0.18	0.21	0.26	0.34	0.62	1.97	0.08	0.15	0.24	0.42	0.49	0.55	0.68	***0.98***
		Characteristics of Test Materials	Total S, %				0.02	0.04	0.06			0.07		0.19		0.44	0.88	1.68		0.08	0.36			0.42		0.91
			Cu/S, wt/wt				0.28	0.35	0.46			0.64		0.22		0.07	0.06	0.04		0.65	0.48			0.42		0.11
			Ni/S, wt/wt				0.57	0.91	0.53			0.56		0.16		0.04	0.03	0.02		0.43	0.11			0.11		0.04
	Ultramafic	**Target**	**Total S, %**	0.21	1.1	2.3				0.07	0.08	0.1	0.13	0.2	0.3	0.5	0.8	1.35	0.07	0.09	0.14	0.33	0.55	0.56	0.65	***0.81***
		Characteristics of Test Materials	Total S, %								0.08			0.3	0.2	0.72	1.24				0.16	0.34		0.75		1.2
			Cu/S, wt/wt								0.34			0.35	0.51	0.07	0.05				0.48	0.46		0.2		0.07
			Ni/S, wt/wt								0.32			0.08	0.13	0.03	0.02				0.41	0.16		0.09		0.04
2	**Anorthositic**	**Target**	**Total S, %**	2.4	0.56	0.25	0.01	0.03	0.05	0.06	0.06	0.08	0.11				0.19		0.09	0.12	0.17	0.21				0.25
		Characteristics of Test Materials	Total S, %				0.02	0.02	0.03																	
			Cu/S, wt/wt				0.41	0.63	0.47																	
			Ni/S, wt/wt				1.04	0.86	0.83																	
	Troctolitic	**Target**	**Total S, %**	16.9	9.7	7	0.01	0.03	0.05	0.06	0.06	**0.07**	0.09	**0.09**	0.1	0.11	**0.12**	0.25	0.05	0.07	0.12	0.18	0.2	0.22	0.26	0.32
		Characteristics of Test Materials	Total S, %				0.04	0.04	0.06			0.08		0.07			0.18							0.17		0.15
			Cu/S, wt/wt				0.35	0.57	0.48			0.53		0.51			0.24							0.69		0.33
			Ni/S, wt/wt				0.06	0.48	0.72			0.45		0.24			0.12							0.33		0.23
	Ultramafic	**Target**	**Total S, %**	0.38	0.25	1.8	0.03	0.04	0.05	0	0	0	0				0		0.04	0.05	0.07	0.12	0.14	0.16	0.19	0.23
		Characteristics of Test Materials	Total S, %				0.06	0.06	0.1														0.12			0.06
			Cu/S, wt/wt				0.69	0.54	0.23														0.78			0.65
			Ni/S, wt/wt				0.89	0.83	0.55														0.59			1.58
3	**Anorthositic**	**Target**	**Total S, %**	9.4	1.2	0.54	0.01	0.03	0.05	0.07	0.08	0.08	0.12				0.14		0.06	0.08	0.12	0.27				0.38
		Characteristics of Test Materials	Total S, %				0.02	0.04	0.04																	
			Cu/S, wt/wt				0.43	0.46	0.55																	
			Ni/S, wt/wt				1.31	0.5	0.27																	
	Troctolitic (augite	**Target**	**Total S, %**	41.2	12.5	1.9	0.01	0.03	0.05	0.06	0.06	0.08	0.1	0.11	0.12	0.14	0.19	0.36	0.05	0.13	0.19	0.32	0.35	0.45	0.48	***0.52***
		Characteristics of Test Materials	Total S, %				0.02	0.04	0.06			0.06			0.14		0.25					0.28				0.59
			Cu/S, wt/wt				0.27	0.34	0.27			0.59			0.31		0.14					0.69				0.29
			Ni/S, wt/wt				1.19	0.91	0.3			0.35			0.3		0.12					0.18				0.12
4	**Troctolitic**	**Target**	**Total S, %**	7	2.2	0.5	0.01	0.03	0.05	0.06	0.07	0.09	0.18	0.19	0.19	0.48	0.92	1.53	0.09	0.15	0.22	0.47				***1.52***
		Characteristics of Test Materials	Total S, %				0.03	0.04	0.04		0.08		0.21			0.51	0.77									1.37
			Cu/S, wt/wt				0.36	0.45	0.5		0.22		0.27			0.06	0.15									0.07
			Ni/S, wt/wt				0.64	0.64	0.62		0.17		0.14			0.05	0.05									0.02
5	**Troctolitic**	**Target**	**Total S, %**	2.5	2	1.1	0.01		0.05	0.07	0.09	0.11	0.16				0.22	0.22	0.12	0.17	***0.26***	0.37				0.45
		Characteristics of Test Materials	Total S, %				0.02		0.06												0.23					0.32
			Cu/S, wt/wt				0.52		0.45												0.38					0.44
			Ni/S, wt/wt				2.13		0.48												0.15					0.15
6	**Troctolitic**	**Target**	**Total S, %**	6.8	0.59	0.33	0.02	0.04		0	0	0	0				0		0.01	0.01	0.03	0.07				0.19
		Characteristics of Test Materials	Total S, %				0.04	0.05																		
			Cu/S, wt/wt				0.73	0.78																		
			Ni/S, wt/wt				1	0.88																		
20	Virginia	**Target**	**Total S, %**	0	10.4	0.054				0.59	1.25	2.98	4.15	4.49	4.85	5.07	6.06	7.45	0	0	0	0				0
		Characteristics of Test Materials	Total S, %								2		3.79			5.68										
			Cu/S, wt/wt								0.01		0.00			0.00										
			Ni/S, wt/wt								0.01		0.00			0.00										

Deviations																										
Non-reactive			Reactive									Lean Ore														
			P10	P25	P50	P75	P80	P85	P90	P95	P100	P10	P25	P50	P75	P80	P85	P90	P95	P100
#N/A	#N/A	#N/A	#N/A	-10%	20%	-83%	#N/A	#N/A	#N/A	-38%	#N/A	#N/A	#N/A	-50%	-8%	#N/A	#N/A	#N/A	-6%	
#N/A	#N/A	#N/A	200%	26%	-20%	-21%	#N/A	-12%	#N/A	#N/A	#N/A	#N/A	#N/A	-8%	#N/A	#N/A	#N/A	#N/A	-9%	
100%	33%	20%	#N/A	#N/A	-30%	#N/A	-10%	#N/A	29%	42%	-15%	#N/A	-47%	50%	#N/A	#N/A	-24%	#N/A	-7%	
#N/A	#N/A	#N/A	#N/A	0%	#N/A	#N/A	50%	-33%	44%	55%	#N/A	#N/A	#N/A	14%	3%	#N/A	34%	#N/A	48%	
100%	-33%	-40%	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	
300%	33%	20%	#N/A	#N/A	14%	#N/A	-22%	#N/A	#N/A	50%	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-23%	#N/A	-53%	
100%	50%	100%	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-14%	#N/A	#N/A	-74%	
100%	33%	-20%	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	
100%	33%	20%	#N/A	#N/A	-25%	#N/A	#N/A	17%	#N/A	32%	#N/A	#N/A	#N/A	#N/A	-13%	#N/A	#N/A	#N/A	13%	
200%	33%	-20%	#N/A	14%	#N/A	17%	#N/A	#N/A	6%	-16%	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-10%	
100%	#N/A	20%	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-12%	#N/A	#N/A	#N/A	#N/A	-29%	
100%	25%	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	
#N/A	#N/A	#N/A	#N/A	60%	#N/A	-9%	#N/A	#N/A	12%	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	

Notes

Target sulfur levels for samples tested in humidity cells (selected by DNR).

0.07 Optional target samples selected by DNR.

1.52 Additional low grade ore samples.

* Approximate sulfur concentrations

Notes:

Greater than 100% deviation for non-reactive samples and between 50% and 100% deviation for reactive and lean ore samples.

Greater than 100% deviation for reactive and lean ore samples.

Attachment 2-2

SRK Consulting. Tailings and Hydromet Residue Testwork – Update on Sample Selection from 24 Hour Testwork. Memorandum to Jennifer Engstrom, MDNR. January 6, 2006.

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Memo

To:	Jennifer Engstrom, MDNR	**Date:**	January 6, 2006
cc:	John Borovsky, Barr	**From:**	Stephen Day
	Jim Scott, PolyMet		
	Don Hunter, PolyMet		
Subject:	NorthMet Project Tailings and Hydromet Residue Testwork – Update on Sample Selection from 24 Hour Testwork	**Project #:**	1UP005.001

Jennifer

We have now received the 2-hourly tailings total sulfur analyses from the pilot plant testwork. These results allow final recommendations to be made for the selection of samples for tailings and hydrometallurgical testwork.

1 Results and Implications of 2-Hourly Sulfur Analyses

Results of the 2-hourly sulfur analyses are shown by the coloured solid lines in Figure 1. The broken horizontal lines are the concentration of sulfur in the composite tailings samples currently being tested in humidity cells. The sulfur content of the ore composites was very uniform (Parcel 1, 2 and 3, 0.86%, 0.9% and 0.86%, respectively) as shown by the solid black lines in Figure 1.

The trend in sulfur results in tailings is explained by the chronology of the testwork and evaluation of addition of copper sulfate as a reagent:

- Flotation testwork began on July 17 with Parcel 2 without the use of copper sulfate. Parcel 2 was processed entirely without using copper sulfate. As shown, sulfur concentrations varied from 0.05% to 0.25% reflecting adjustment of the process conditions early in the testwork. The average was 0.19%. The composite tailings sample has a sulfur content of 0.2% closely representing the average.

- Testwork continued with Parcel 1 without using copper sulphate. Processing was continuous so one point is shared between Parcel 2 and Parcel 1. The range of sulfur concentrations was 0.19% to 0.28% with an average of 0.24%. The composite sample was 0.23% and is close to the average.

- Pilot plant testwork was suspended on July 19 to allow for further bench scale testing on recovery of metals.

- The pilot plant resumed on August 8 using Parcel 1. Addition of copper sulfate was evaluated. This reagent causes activation of the sulfide mineral surfaces and improves bulk sulfide flotation. The effect of copper sulfate on tailings characteristics was immediately apparent for Parcel 1.

Total sulfur concentrations decreased to a range of 0.09% to 0.13% (average 0.1%) and the resulting tailings composite was 0.1%.

- Processing continued with Parcel 3 using the copper sulfate additive. Sulfur content of the tailings varied over a wider range (0.09% to 0.25%, average 0.18%) though the range was comparable to the total range indicated by processing of other ore packages. The resulting composite had a total sulfur content of 0.15%.

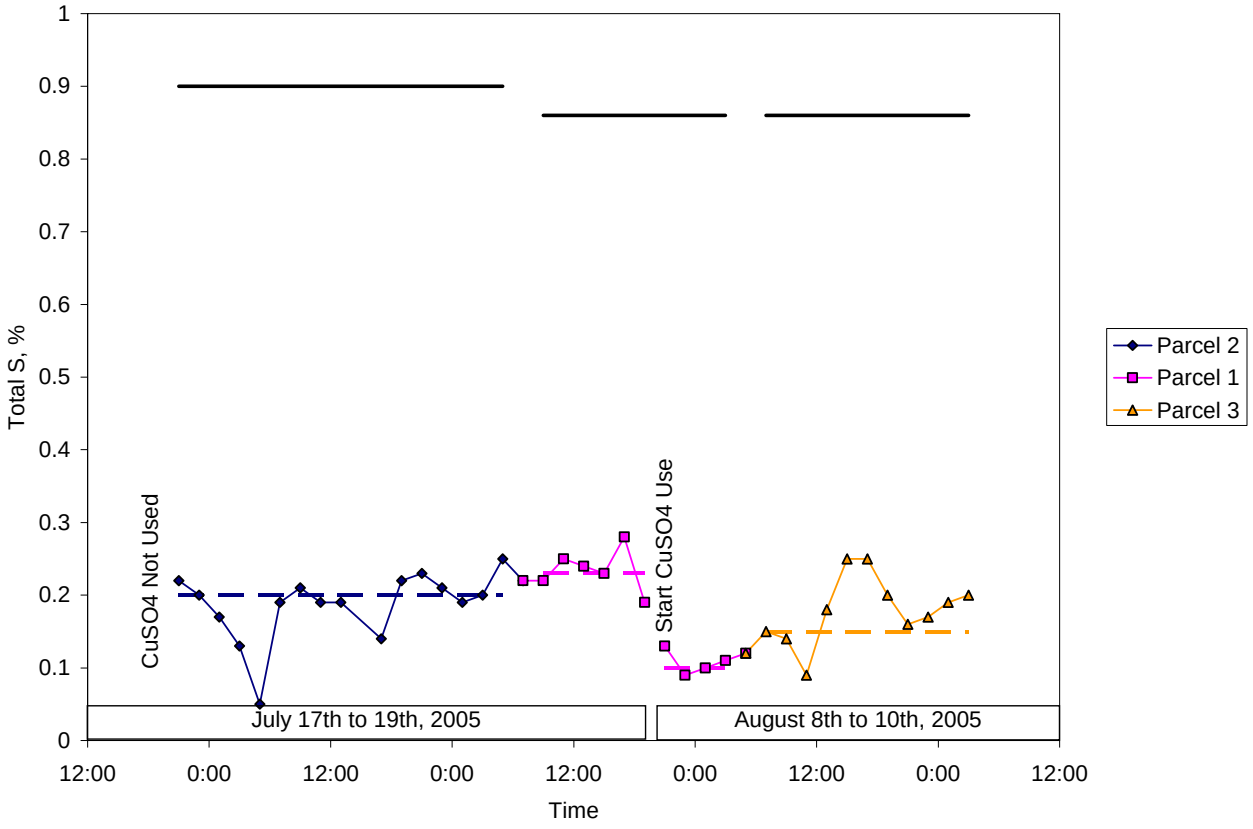

Figure 1. Results of 2-Hourly Total Sulfur Analyses. Solid lines and points connect 2-hourly results. Broken solid lines are sulfur concentrations in composite tailings samples representing each stage of testwork. Solid horizontal lines are the respective ore composite sulfur contents.

Based on the process testwork, Polymet has the made decision to advance the project with the use of copper sulfate to optimize overall sulfide mineral flotation. This decision is beneficial for the tailings since it is expected to lower the overall sulfide content.

The process testwork showed that sulfur concentrations in the tailings can be expected to vary in response to changes in process conditions including the use of copper sulfate. Parcel 3 showed that the use of copper sulfate may not always result in low sulfur content in tailings, and therefore there is need to capture sulfur concentrations approaching 0.25% in the kinetic testwork. The samples generated without copper sulfate provide the required range and can be tested to represent the potential for higher sulfur concentrations in the tailings. The lack of copper sulfate for the Parcel 2 and 1 samples is not expected to have significantly affected the reactivity of the residual sulfide minerals in the tailings:

It is therefore concluded that:

- Kinetic testing of all four tailings samples should be continued.
- No additional samples are needed to represent the range of sulfur content expected in tailings.

2 Testing of HydroMet Residues

2.1 Source of Sulfide Concentrate for Hydromet Process Evaluation

Evaluation of the hydromet process was performed using two bulk sulfide concentrates produced by processing of ore parcels 2 and 1 (without copper sulfate) and ore parcels 1 and 3 (with copper sulfate). The sulfide concentrates contained the following total sulfur concentrations:

- No Copper Sulfate
  - Parcel 2 – 23.6%
  - Parcel 1 – 21.3%
- With Copper Sulfate
  - Parcel 1 – 22.1%
  - Parcel 3 – 21.6%

It is apparent that the sulfur content of the concentrates does not vary significantly though the effect of copper sulfate on concentrate sulfur content for Parcel 1 is apparent and corresponds with the matching decrease in sulfur content of the tailings. Since the decision has been made to proceed with the use of copper sulfate, only the residues produced from sulfide concentrate generated using copper sulfide should be tested.

2.2 HydroMet Residues

All the expected HydroMet Residues were produced by processing of the sulfide concentrate generated using copper sulfate.

A difference exists between the way that the residues were recovered in the pilot test compared to actual operating conditions.

To summarize, the first step in the process is the leaching of the sulfide concentrate to produce a low pH pregnant solution containing all the commodity metals. Subsequent recovery of the metals involves a series of pH adjustments to the leach solution that results in precipitation of products and residues. The products are then refined to recover the contained metals (copper, nickel, cobalt, PGM, zinc). The residues contain entrained leach solutions that have to be recovered to optimize recovery of commodity metals. Under full-scale operating conditions, recovery of the leach solutions from the residues will occur by rinsing the residue cakes with pH-adjusted re-cycled final process water to displace the leach solutions. The pH adjustment is required to ensure that metals in the leach solution are not lost to the solids.

However, under pilot plant conditions, the recycled process water was not available because the processing of the leach solutions occurred in a stepwise rather than continuous fashion. Each metal recovery step was performed and completed before proceeding to the next. The final process solution that will be used for rinsing at full-scale was only generated at the end of the pilot plant and was therefore not available for the residue rinsing steps. The difference between full-scale and pilot plant conditions represents a practicality of metallurgical testing in that operation the pilot plant continuously is not an option with the available quantity feed concentrate.

Residues generated by the pilot plant were rinsed with locally obtained river water. No additional rinsing of the residues is proposed for the dissolution testwork. Rinsing with river water was less aggressive in displacing metal-laden leach solutions than can be expected with pH-adjusted process water. The residues can therefore be expected to contain higher metal content than under operating conditions and indicate greater leachable metals in dissolution tests. The testwork will tend to over-estimate rather than under-estimate water quality for water management planning and impact assessment.

3 Conclusions

The following actions are proposed:

- Kinetic testing of all four existing tailings samples in dissolution tests will continue.
- Testing of residues produced by hydromet testing of sulfide concentrate (with copper sulfate) will be started as described in the "Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan"

Attachment 2-3

SRK Consulting. Waste Rock and Lean Ore Geochemical Characterization Plan, NorthMet Project, Minnesota. May 2006.

Waste Rock and Lean Ore Geochemical Characterization Plan NorthMet Project, Minnesota

Prepared for

PolyMet Mining Corporation

Prepared by

SRK Consulting
Engineers and Scientists

May 2006

Waste Rock and Lean Ore Geochemical Characterization Plan NorthMet Project, Minnesota

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1 Introduction

1.1 Background

PolyMet Mining (PolyMet) is proposing to develop the NorthMet Project (Dunka Road Project of US Steel) near Babbitt, Minnesota. As a part of the Minnesota Department of Natural Resources (MDNR) "Permit to Mine" process a complete "mine waste characterization" will be required (Minnesota Rules Chapter 6132.1000). This document describes the plan developed for selection and testing of waste rock samples for the NorthMet Project, and the context for interpretation of the results.

The issues associated with waste rock at the NorthMet are expected to include acid rock drainage (ARD) and leaching of some heavy metals. The latter in particular are expected to include nickel and cobalt both of which do not require acidic conditions to be mobilized at elevated concentrations.

The specific objectives of this program include:

- Refinement of preliminary waste rock management criteria developed by PolyMet and MDNR.
- Development of mass-loading rates for input into water quality predictions for impact assessment and mitigation design.

1.2 Geological Setting

The NorthMet Deposit is located in the intrusive Duluth Complex of northern Minnesota. Disseminated copper-nickel-iron sulfides (chalcopyrite, cubanite, pentlandite and pyrrhotite) with associated platinum group element (PGE) mineralization will be extracted from several igneous stratigraphic horizons.

In the vicinity of the NorthMet deposit, the Duluth Complex intruded and assimilated the Virginia Formation, which consists of argillite and greywacke with minor interbeds of siltstone, graphitic argillite, chert, and carbonate. This formation is the stratigraphic footwall of the NorthMet deposit, but also occurs as xenoliths ("inclusions") within the deposit.

1.3 Agency Consultation and Design Process

This document was developed in consultation with staff from the Minnesota Department of Natural Resources (MDNR). The consultation included the following steps:

- December, 2004. PolyMet submitted a draft "Work Plan for Geochemical Characterization of Rock and Concentrator Flotation Tailings". The plan was presented to MDNR representatives.
- January 31 and February 1, 2005. Meetings were held by teleconference between SRK and MDNR representatives to further discuss the variables potentially affecting water chemistry from waste stockpiles.

- March 17, 2005. MDNR requested additional information on the tonnages of the major units and rock types, and the distribution of sulfur and minerals.
- March 28, 2005. PolyMet provided the requested information.
- April 12, 2005. MDNR provided a sample selection matrix. This matrix was accepted by PolyMet and is the basis for the selection of samples described in this document.
- May 15, 2005. MDNR provided a design for specific testwork.
- May 17, 2005. MDNR provided a design for specific testwork.
- June 6, 2005. A draft of this sampling plan was submitted to MDNR.
- June 15, 2005. MDNR provided comments on the draft plan.
- June 22, 2005. SRK provided responses and discussion of the MDNR comments in a letter to MDNR which were discussed during a teleconference on June 27, 2005.
- July 5, 2005. SRK provided results of candidate samples selected for kinetic testing to in a memorandum to MDNR.
- July 13, 2005. MDNR provided comments on the July 5, 2005 SRK memorandum.
- July 15, 2005. SRK provided clarification on sample selection in a memorandum to MDNR.
- July 20, 2005. MDNR notified SRK and PolyMet that kinetic testing on the majority of waste rock samples could be initiated. It was recognized that analysis of a few candidate samples was ongoing.
- August 4, 2005. MDNR Provided recommendations for lean ore characterization.
- August 29, 2005. As requested by SRK, MDNR provided additional rationale for the recommendations on lean ore sampling selection.
- September 14, 2005. Lean ore sample selection was further discussed during a conference call which were provided the basis for completion of this plan.

The plan was fully implemented in October 2005.

This document has been prepared to conclude the design process and seek MDNR approval of PolyMet's plans to respond to the waste rock characterization component of requirements under Minnesota Rules 6132.1000.

1.4 Organization of this Document

This document describes:

- Section 2. Design basis for the program.
- Section 3. Sample selection. This section describes the methods used to select samples from the NorthMet Project drill hole database.

- Section 4. Analytical methods. This section describes methods used to analyse solids and leachates.
- Section 5. Use of the results in the context of water chemistry predictions.

1.5 Acknowledgements

The following individuals cooperated in the preparation of this plan:

- John Borovsky, Barr Engineering Company.
- Stephen Day, SRK Consulting.
- Paul Eger, MDNR.
- Jennifer Engstrom, MDNR.
- Steve Geerts, PolyMet.
- Don Hunter, PolyMet.
- Kim Lapakko, MDNR.
- Richard Patelke, PolyMet.
- Jim Scott, PolyMet.

1.6 Analytical Laboratues

The following laboratories are performing the procedures described in this document (contact names for each laboratory are shown):

- ALS Chemex, North Vancouver, British Columbia – solids analysis listed in Section 4.1.1 (Bill Anslow).
- Optical – PolyMet (Richard Patelke).
- Sub-Optical Lab – McSwiggen and Associates (Pete McSwiggen).
- Canadian Environmental and Metallurgical Inc (CEMI), North Vancouver, British Columbia - kinetic testing (Rik Vos).
- Cantest Inc.. Vancouver, British Columbia - Kinetic test leachate analysis (Richard Jornitz).

2 Characterization Design

2.1 Background

Sample selection for this project was based on the December 2004 PolyMet geologic and assay database (assembled by PolyMet) and the February 2005 block model by Dr. Phil Hellman of Hellman & Schofield. Ultimately, the rock characterization data from these tests will be linked to the mine plan through this database and block model. Current and anticipated future geochemical data collection from drilling is described in documents submitted to MDNR by PolyMet on August 23, 2004 and September 15, 2004.

2.2 Design Basis

Based on discussions between SRK and MDNR on January 31 and February 1, 2005, the following critical variables were identified that potentially could affect drainage quality from waste rock stockpiles:

- Sulfur content.
- Sulfide mineral type.
- Rock type.
- Fragment particle size.

Other important variables include, mineral content, mineral grain size, mineral chemistry, and mode of mineral occurrence.

Sulfur content is considered the primary factor affecting the potential for acid generation and metal mobility based on existing research conducted by MDNR. At higher sulfur concentrations, it appears that acid generation starts earlier and results in lower pHs resulting in increased metal leaching. At lower sulfur concentrations, acid generation is not expected to occur, but sulfur content is correlated with metal content and is therefore expected to be related to metal release.

Sulfide mineral type can be important in terms of rate of reaction and metal release. For example, it is well know that pyrrhotite is more reactive than pyrite. Chalcopyrite and pentlandite are sources of copper and nickel, respectively in drainage. However, since the dominant sulfide mineral in the waste rock appears to be pyrrhotite (based on distribution of metal content) and the commodity sulfide minerals (chalcopyrite, pentlandite and cubanite) are expected to be present at low concentrations, sulfide mineral type was not considered as a primary variable for sample selection. As described below, concentrations of copper, nickel, cobalt and zinc were used as secondary factors for sample selection which is expected to capture variations in sulfide mineralogy. Lean ore characterization is considered separately. All samples are being characterized to evaluate assumptions about the mineralogical occurrence of the important metals.

All rock types in the Duluth Complex are variants of troctolite and to a lesser extent ultramafic rocks. Carbonate minerals are absent or occur at very low concentrations in this rock type. Therefore, the variation in silicate content of these rocks is considered to be an important variable controlling drainage pH.

MDNR also considered that igneous layer in the intrusive complex may be a significant variable. because the reactivity of the minerals may be different in each of the layers. SRK and PolyMet did not agree with this position based on evidence from MDNR's past testwork. Nonetheless, this variable has been carried through the sampling design.

Finally fragment particle size is an important factor because it controls exposure of the reactive minerals and the overall surface area available for reaction.

2.3 Sampling Matrix for Waste Rock

A sampling matrix for waste rock characterization (Table 1) was developed by MDNR, SRK and PolyMet through a series of discussions and exchange of relevant data. Table 1 shows how the main variables have been translated to a sampling design. The table also provides estimates of the tonnages of each major rock type within each unit. Reading from left to right, the columns in the table show the following:

- Unit. This refers to the stratigraphic igneous layers in the complex (number 1 to 7). Unit 20 refers to the footwall of the deposit composed of Virginia Formation and localized igneous intrusions
- Rock Type. This refers to a generalized rock description in the associated unit.
- Estimated Rock Tonnages. These tonnages indicate the estimated amounts of each rock type within each layer and therefore their relative importance. The categories were developed by MDNR and PolyMet to indicate rock with sulfur less than 0.05%[1] ("non-reactive"), sulfur greater than 0.05% but not likely ore grade ("reactive"), and rock with marginal ore grade (lean ore).

Selection of samples for each unit and rock type combination was based on sulfur concentrations in order to develop correlations between reactivity and bulk characteristics such as sulfur and metal content. This provides a basis for water chemistry predictions using bulk characteristics, prediction of waste management criteria (based on sulfur content and metal content) and ultimately for the selection of easily-measured parameters that can be used for waste management during mining (see Section 5.2 below, for additional discussion).

[1] Note that the non-reactive classification is a temporary criterion which will be refined by this testwork program.

Table 1: Matrix for Sample Selection in Waste Rock Types

Unit	Rock Type	Non-Reactive	Reactive	Approximate sulfur contents																		Reactive Rock
				Non-reactive[1]			Reactive[1]								Lean Ore[1]							P size
		M. tons	M. tons	NR1	NR2	NR3	P10	P25	P50	P75	P80	P85	P90	P95	P100	P10	P25	P50	P75	P80	P85	P90	P95	
1	Anorthositic	0.57	0.99				0.08	***0.1***	***0.15***	***0.29***				***1.09***	1.09	0.11	0.26	***0.36***	***0.93***				***1.95***	***4***
1	Gabbroic	0	0.68				0.06	0.07	0.07	0.08				0.19	0.5	0.09	0.1	0.18	0.37				0.5	
1	Sedimentary hornfels	0	1.6				***0.08***	***0.35***	***0.69***	***2.2***	2.32	***2.81***	3.38	3.5	3.78	0.34	1.37	***1.58***	1.76				***4.91***	
1	Troctolitic	17.2	40.1	***0.01***	***0.03***	***0.05***	0.06	**0.07**	***0.1***	0.18	***0.21***	0.26	***0.34***	***0.62***	***1.97***	0.08	***0.15***	***0.24***	0.42	0.49	***0.55***	0.68	***0.98***	***4***
1	Ultramafic	0.21	1.1				0.07	***0.08***	0.1	0.13	***0.2****	***0.3****	***0.5****	***0.8****	1.35	0.07	0.09	***0.14***	***0.33***	0.55	***0.56***	0.65	***0.81***	
1	Vein	0.055	0.022				0	0	0	0				0										
2	Anorthositic	2.4	0.56	***0.01***	***0.03***	***0.05***	0.06	0.06	0.08	0.11				0.19		0.09	0.12	0.17	0.21				0.25	
2	Basalt inclusions	0.28	0													0	0	0	0				0	
2	Gabbroic	0	0.082				0	0	0	0				0										
2	Troctolitic	16.9	9.7	***0.01***	***0.03***	***0.05***	0.06	0.06	0.07	0.09	0.09	0.1	0.11	0.12	0.25	0.05	0.07	0.12	0.18	0.2	***0.22***	0.26	***0.32***	
2	Ultramafic	0.38	0.25	***0.03***	***0.04***	***0.05***	0	0	0	0				0		0.04	***0.05***	0.07	0.12	***0.14***	0.16	0.19	***0.23***	
3	Anorthositic	9.4	1.2	***0.01***	***0.03***	***0.05***	0.07	0.08	0.08	0.12				0.14		0.06	0.08	0.12	0.27				0.38	
3	Fault-Breccia	0	0.055				0	0	0	0				0										
3	Gabbroic	0.2	0.72				0.09	0.15	0.18	0.27				0.29		0	0	0	0				0	
3	Noritic	0.11	0																					
3	Sedimentary hornfels	0.38	0.46				0.12	1.42	1.67	1.97				2.22		1.66	1.77	1.85	2.43				3.26	
3	Troctolitic (augite)	41.2	12.5	***0.01***	***0.03***	***0.05***	0.06	0.06	***0.08***	0.1	0.11	***0.12***	0.14	***0.19***	0.36	0.05	0.13	0.19	***0.32***	0.35	0.45	0.48	***0.52***	***4***
3	Ultramafic	0.24	0.071				0	0	0	0				0		0	0	0	0				0	
4	Anorthositic	0.16	0.055				0	0	0	0				0										
4	Sedimentary hornfels	0	0.055				0	0	0	0				0										
4	Troctolitic	7	2.2	***0.01***	***0.03***	***0.05***	0.06	***0.07***	0.09	***0.18***	0.19	0.19	***0.48***	***0.92***	1.53	0.09	0.15	0.22	0.47				***1.52***	***4***
4	Vein	0.055	0																					
5	Troctolitic	2.5	2	***0.01***		***0.05***	0.07	0.09	0.11	0.16				0.22	0.22	0.12	0.17	0.26	0.37				***0.45***	
6	Chlorite	0.16	0													0	0	0	0				0	
6	Fault-Breccia	0.055	0.055				0	0	0	0				0										
6	Troctolitic	6.8	0.59	***0.02***	***0.04***		0	0	0	0				0		0.01	0.01	0.03	0.07				***0.19***	
6	Ultramafic	0	0.11				0	0	0	0				0										
7	Ultramafic	0.082	0													0	0	0	0				0	
20	Troctolitic	0	0.27				3.18	3.45	3.76	4.3				4.31										
20	Virginia	0	10.4				0.59	***1.25***	2.98	***4.15***	4.49	4.85	5.07	***6.06***	7.45	0	0	0	0				0	***4***

Notes:
1. Non-reactive rock categories (lower, medium and higher sulfur contents).
2. Sulfur percentiles for reactive rock types calculated by PolyMet are shown. "*" indicates approximate percentiles.
3. Grey – sampling plan.
4. Bold and italic – samples obtained.
5. Bold border – duplicate cell in operation.

Separate sulfur ranges were defined for the “non-reactive” and “reactive” categories. Within the non-reactive category, three sulfur concentrations were selected to represent the lowest possible sulfur concentration in the rock type (typically 0.01%), the upper limit to this category (0.05%) and an intermediate (0.03%). The need for samples was identified for relatively abundant rock types contributing more than 1,000,000 tons (ie more than 1% of the rock mass). For the reactive category, sample selections were based on sulfur concentration percentiles calculated by Polymet. Again, rock types contributing more than 1,000,000 tons were identified for testing.

Sulfide mineral variability was considered by preferring samples with higher concentrations of Ni and Co, Cu and Zn.

The search for suitable samples included all candidate sulfur values indicated in grey shading Table 1.

2.4 Characterization of Lean Ore

Lean ore is defined as rock containing grades of commodity minerals below that at which processing can currently be justified, but may eventually be processed if project economics improve. In terms of sulfur content, lean ore mainly overlaps the “reactive” waste rock category and also some to degree the non-reactive catagory but contains higher nickel and copper concentrations than waste rock. Therefore, the main difference between lean ore and waste rock is expected to be in the mineralogical occurrence of sulfur. In waste rock, sulfur occurs mainly as iron sulfide but in lean ore the commodity minerals pentlandite, chalcopyrite and cubanite are expected to be more important. This has important implications for drainage chemistry. In particular, oxidation and leaching of pentlandite is expected to release more nickel than pyrrhotite due to the higher Ni/Fe ratio in pentlandite. This limits the co-precipitation of nickel with iron oxhydroxides during oxidation.

The overall approach to selection of samples was similar to that of waste rock.

3 Sampling

3.1 Methods

3.1.1 Sample Selection for Non-Reactive and Reactive Waste Types

Samples were selected from the NorthMet Project drill hole database. The following sequence was used to select samples:

1. The database was reduced to rock core. Only diamond drill holes were considered because this method of drilling results in the best characterised samples by eliminating mixing of intervals and allowing accurate logging.
2. Separate datasets were created for “waste rock” (non-reactive and reactive) and “lean ore”.
3. Due to ease of recovery from the core archives, core obtained by PolyMet drilling was preferred. In cases where suitable samples could not be found from this source, US Steel drillholes were considered.
4. To select samples, 20-foot moving-averages for consistent rock types were calculated for all parameters. This interval was selected to reduce the potential for characterizing small-scale local heterogeneity, and provide a sample width that relates to mining scale. The wider interval also ensured that sufficient sample is available for all the different tests. In practice, the need to obtain samples with specific unit, rock type and ulphur combinations resulted in sampling intervals of 5 to 20 feet. In three cases, sample intervals from two different locations had to be selected to yield sufficient material for testing.
5. Within each rock type and unit combination, samples were identified based on a ulphur range straddling the desired concentrations. The range was calculated as between the midpoint concentration to the two nearest ulphur concentrations. For example, a target concentration of 0.03% between 0.01% and 0.05% could be selected from intervals with ulphur concentrations between 0.02% and 0.04% (with a preference for 0.03%). This approach was necessary to provide candidate samples in all ranges and incorporate the need to target higher metal concentrations for tests.
6. If more than one candidate interval was identified, the interval with the highest nickel and cobalt concentrations was selected with a primary focus on nickel content. If several samples could still be chosen, copper and zinc concentrations were considered. This approach resulted in selection of samples with nickel concentrations approaching the highest values in the database. For example, the highest nickel concentration in a non-reactive sample is 0.043% compared to a maximum value of 0.043% in the database. Similarly, the highest reactive nickel concentration in the database is 0.042%, and the highest sample is 0.038%.

The sample lists (Table 2 and 3) were re-generated several times due to limitations of core availability. The first pass attempted to use PolyMet core only. Subsequent lists included US Steel core.

Lean ore sample selection was performed in two passes. The first pass targeted samples containing ulphur concentrations at the P95 level. The second pass selected samples for the remaining samples. To ensure that delays did not occur in the second pass, a large number of intervals were selected to ensure that at least one sample would be available for each target ulphur concentration.

3.1.2 Core Recovery

Samples were recovered from archived core boxes by PolyMet personnel. Since the core had previously halved for initial analysis, the remaining half core was quartered for this sampling.

All required intervals were sampled on a continuous basis and placed in plastic bags for shipment to the analytical laboratory.

3.2 Sample List

The resulting lists of waste rock samples and lean ore samples shipped to the laboratory are shown in Tables 2 and 3.

Table 2: List of Non-Reactive and Reactive Samples Selected

Unit	Rock Type	S Percentile – Non-Reactive and Reactive Waste Rock[1]	Selected Interval				Weighted Average Characteristics Calculated from Drill Hole Database				
			DDH	From	To	Length	S	Ni	Co	Cu	Zn
				Feet	Feet	Feet	%	%	mg/kg	%	mg/kg
1	Anorthositic	P25	99-320C	830	850	20	0.12	0.015	35.8	0.035	72
1	Anorthositic	P50	00-361C	310	320	10	0.16	0.018	36.0	0.013	51
1	Anorthositic	P75	00-361C	345	350	5	0.33	0.026	53.0	0.038	68
1	Anorthositic	P95	00-343C	240	250	10	0.67	0.020	38.0	0.069	69
1	Sedimentary Hornfels	P10	26030	1047	1052	5	0.17	0.006	10.0	0.006	34
1	Sedimentary Hornfels	P25	26061	1218	1233	15	0.43	0.011	19.3	0.012	193
1	Sedimentary Hornfels	P50	00-340C	990	995	10	0.62	0.012	24.0	0.031	148
1	Sedimentary Hornfels	P75	00-340C	965	974.5	15	1.49	0.015	23.7	0.031	249
1	Sedimentary Hornfels	P85	26043	1501	1506	5	2.76	0.024	36.0	0.037	344
1	Sedimentary Hornfels	P85(a)	26027	740	745	5	2.59	0.035	67.0	0.051	48
1	Troctolitic	NR1	26029	815	825	10					
1	Troctolitic	NR2	00-340C	595	615	20	0.03	0.025	49.8	0.013	71
1	Troctolitic	NR3	00-334C	580	600	20	0.05	0.024	52.0	0.024	83
1	Troctolitic	P25	00-334C	640	660	20	0.08	0.038	63.8	0.024	102
1	Troctolitic	P50	00-347C	795	815	20	0.09	0.034	70.8	0.035	94
1	Troctolitic	P80	00-350C	580	600	20	0.22	0.027	55.5	0.041	115
1	Troctolitic	P90	00-327C	225	245	20	0.44	0.015	50.5	0.032	73
1	Troctolitic	P95	00-371C	435	440	5	0.65	0.014	32.0	0.036	54
1	Troctolitic	P100	00-340C	765	780	15	1.72	0.022	78.0	0.064	69
1	Ultramafic	P25	00-357C	335	340	5	0.08	0.015	35.0	0.026	68
1	Ultramafic	P80	00-326C	680	685	5	0.21	0.016	33.0	0.085	82
1	Ultramafic	P85	00-357C	535	540	5	0.26	0.026	34.0	0.095	62
1	Ultramafic	P90	99-318C	725	735	10	0.44	0.011	23.0	0.032	45
1	Ultramafic	P95	99-317C	460	470	10	1.10	0.018	33.0	0.056	86
2	Anorthositic	NR1	00-366C	185	205	20	0.01	0.018	35.0	0.008	47
2	Anorthositic	NR2	00-366C	230	240	10	0.02	0.014	32.0	0.011	51
2	Anorthositic	NR3	99-320C	165	175	10	0.04	0.023	46.0	0.012	63
2	Troctolitic	NR1	99-318C	250	270	20	0.02	0.022	42.3	0.011	64
2	Troctolitic	NR2	00-373C	95	115	20	0.03	0.036	64.0	0.019	84
2	Troctolitic	NR3	00-373C	75	95	20	0.05	0.031	55.3	0.020	75
2	Troctolitic	P50	00-357C	110	130	20	0.07	0.024	53.5	0.029	88
2	Troctolitic	P80	99-320C	315	330	15	0.09	0.017	39.3	0.026	65
2	Troctolitic	P95	00-369C	335	345	10	0.16	0.021	43.0	0.046	68
2	Ultramafic	NR1	00-368C	460	465	5	0.03	0.042	70.0	0.033	98
2	Ultramafic	NR2	26055	940	945	5	0.04	0.037	69.0	0.025	88
2	Ultramafic	NR3	26098	145	148.5	3.5	0.05	0.037	76.0	0.014	112

Table 2: List of Non-Reactive and Reactive Samples Selected (Cont'd).

Unit	Rock Type	S Percentile – Non-Reactive and Reactive Waste Rock[1]	Selected Interval				Weighted Average Characteristics Calculated from Drill Hole Database				
			DDH	From	To	Length	S	Ni	Co	Cu	Zn
				Feet	Feet	Feet	%	%	mg/kg	%	mg/kg
2	Ultramafic	NR3(a)	00-337C	105	110	5	0.05	0.043	71.0	0.023	116
3	Anorthositic	NR1	00-334C	30	50	20	0.01	0.022	47.0	0.009	64
3	Anorthositic	NR2	00-368C	125	145	20	0.03	0.016	38.0	0.015	62
3	Anorthositic	NR3	00-368C	20	40	20	0.04	0.010	25.8	0.022	47
3	Troctolitic	NR1	00-366C	35	55	20	0.01	0.019	45.3	0.005	50
3	Troctolitic	NR2	00-334C	110	130	20	0.03	0.028	57.3	0.012	73
3	Troctolitic	NR3	00-347C	155	175	20	0.04	0.015	49.5	0.015	67
3	Troctolitic	P50	00-347C	280	300	20	0.08	0.018	45.3	0.035	61
3	Troctolitic	P85	00-326C	60	70	10	0.12	0.031	51.0	0.034	91
3	Troctolitic	P95	00-369C	305	325	20	0.27	0.030	51.5	0.037	57
4	Troctolitic	NR1	00-367C	50	65	15	0.02	0.015	38.7	0.010	59
4	Troctolitic	NR2	00-367C	260	280	20	0.04	0.024	53.8	0.018	78
4	Troctolitic	NR3	00-367C	290	310	20	0.04	0.021	41.3	0.018	64
4	Troctolitic	P25	00-370C	20	30	10	0.07	0.010	37.0	0.016	67
4	Troctolitic	P75	00-369C	20	30	10	0.14	0.021	39.0	0.043	60
4	Troctolitic	P90	00-367C	170	175	5	0.48	0.023	45.0	0.034	54
4	Troctolitic	P95	00-367C	395	400	5	0.92	0.028	76.0	0.080	96
5	Troctolitic	NR1	26064	44	54	10	0.01	0.035	62.0	0.009	58
5	Troctolitic	NR3	26064	264	269	5	0.04	0.017	42.0	0.005	56
5	Troctolitic	NR3(a)	26064	146	156	10	0.05	0.032	67.0	0.031	78
6	Troctolitic	NR1	26056	110	125	15	0.02	0.034	66.7	0.025	85
6	Troctolitic	NR2	26056	135	153	18	0.04	0.037	62.7	0.032	89
20	Virginia	P25	00-361C	737	749	15	1.91	0.015	22.3	0.039	225
20	Virginia	P75	00-364C	210	229	19	4.11	0.018	26.8	0.017	872
20	Virginia	P90	00-337C	510	520	10	5.12	0.016	29.5	0.019	492

Notes:

1. Designation (a) indicates that the sample will be mixed with the previous sample in the list to make a composite for testing.

Table 3: List of Lean Ore Samples Selected

Unit	Rock Type	S Percentile	Selected Interval				Weighted Average Characteristics Calculated from Drill Hole Database				
			DDH	From	To	Length	S	Ni	Co	Cu	Zn
				Feet	Feet	Feet	%	%	mg/kg	%	mg/kg
1	Anorthositic	P50	99-320C	400	405	5	0.32	0.06	63	0.07	68
1	Anorthositic	P75	00-331C	255	260	5	0.95	0.05	38	0.16	42
1	Anorthositic	P95	26027	616	626	10	2.31	0.054	77.0	0.144	75
1	Sedimentary Hornfels	P50	26058	704	715	11	1.59	0.04	40	0.08	175
1	Sedimentary Hornfels	P95	26062	993	998	5	5.49	0.049	81.0	0.153	166
1	Sedimentary Hornfels	P95	26026	565	568	3	3.83	0.033	70.0	0.074	88
1	Troctolitic	P25	00-326C	250	265	15	0.15	0.04	49	0.07	73
1	Troctolitic	P50	00-340C	910	925	15	0.33	0.04	50	0.15	77
1	Troctolitic	P85	00-331C	190	210	20	0.54	0.04	41	0.17	43
1	Troctolitic	P95	00-340C	725	745	20	1.06	0.045	116.0	0.114	95
1	Ultramafic	P50	00-326C	495	505	10	0.14	0.06	91	0.06	128
1	Ultramafic	P75	00-344C	630	635	5	0.33	0.05	54	0.13	82
1	Ultramafic	P85	00-330C	275	280	5	0.60	0.06	123	0.10	86
1	Ultramafic	P95	00-344C	515	520	5	1.10	0.057	72.0	0.090	130
2	Troctolitic	P85	99-318C	325	330	5	0.21	0.05	63	0.14	90
2	Troctolitic	P95	00-340C	380	385	5	0.30	0.04	56	0.13	84
2	Ultramafic	P80	00-326C	225	235	10	0.13	0.06	87	0.10	122
2	Ultramafic	P95	00-361C	240	245	5	0.20	0.085	103.0	0.017	130
3	Troctolitic	P75	00-367C	495	500	5	0.28	0.05	52	0.16	58
3	Troctolitic	P95	26049	358	362	4	0.55	0.058	73.0	0.151	60
3	Troctolitic	P95	26030	291	296	5	0.50	0.055	78.0	0.149	104
4	Troctolitic	P95	00-367C	400	405	5	1.52	0.031	79.0	0.116	86
5	Troctolitic	P95	26056	302	312	10	0.45	0.04	50	0.11	67
6	Troctolitic	P95	26142	360	365	5	0.15	0.043	50.0	0.109	80

4 Sample Preparation and Analysis

4.1 Solids Characterization

4.1.1 Sample Preparation

Samples were shipped to Canadian Environmental and Metallurgical Inc (CEMI) as whole core pieces. The following procedures were used for sample preparation:

- Upon arrival, each sample was weighed and its weight recorded. Specific gravity was determined.
- Each sample was crushed to pass a 0.25 inch screen.
- A 3 kg split of crushed sample was split and saved for humidity cell testing. This provides sufficient sample for duplicate kinetic testing if needed
- A 200 g split was used for solids characterization
- A 50 g split was saved for additional archive and petrographic analysis

4.1.2 Chemical Analysis

A split of each sample was submitted for an extensive suite of analysis, as follows:

- Acid base accounting (total S, carbonate, paste pH). Sulfur as sulfate is not needed because previous work shows that sulfur occurs exclusively as sulfide. Carbonate rather than neutralization potential is being determined because neutralization potential determinations on rocks containing reactive silicates are ambiguous and do not reflect field capacity to neutralize acid. Carbonate indicates the field reactive component of acid neutralization potential.
- 27 elements by ICP scan following four-acid (nitric-hydrochloric-perchloric-hydrofluoric) digestion (near total) (0.5 g).
- 34 elements by ICP scan following aqua regia (nitiric-hydrochloric acid) digestion (0.5 g).
- Whole rock oxides (0.5 g).

These methods were selected to provide continuity with the earlier work and will therefore allow the samples selected to be compared with the existing project database.

Method detection limits are provided in Appendix A.

In addition, 200 g of all samples were split into four size fractions (-100+270 mesh, -35+100 mesh, -10+35 mesh and -0.25”+10 mesh) for analysis of total S and 27 elements by four acid digestion.

Chemical analysis of all samples was completed prior to implementation of kinetic testing.

4.1.3 Optical Analysis

Two pieces of typical core from each interval sampled were taken for preparation of polished thin sections to confirm the rock type and quantify reactive minerals. Optical mineralogy reports will indicate mineral types, mineral abundance, grain sizes and mineral occurrence.

4.1.4 Sub-Optical Analysis

Sub-optical analysis included determination of the trace element content of major minerals on selected samples using microprobe analyses.

4.2 Kinetic Test Methods

4.2.1 Humidity Cell

Humidity cell testing is being performed using ASTM Procedure D 5744 – 96 (Reapproved 2001). This procedure was selected for the following reasons:

- Similar procedures have been in use under different names since the late 1980s (e.g. MEND 1991). The results can therefore be evaluated in the context of more than a decade of experience using the procedure.
- It is a standard procedure approved by the ASTM and is therefore defensible as a method.

The ASTM procedure provides some options for varying the test procedure. Appendix B provides a detailed listing of the requirement of the ASTM procedure, options chosen and any variances from the ASTM procedure

4.2.2 MDNR Reactor

To evaluate size fraction effects, four size fractions (-100 mesh, -35+100 mesh, -10+35 mesh and -0.25”+10 mesh) from five samples are being tested using a procedure referred to as the “MDNR Reactor” experiment. The two smallest size fractions are being tested in a specifically designed apparatus designed by MDNR (Appendix C) to contain 75 g. The two coarser fractions are being tested in cells with the same configuration as ASTM Procedure D 5744–96. Details of the construction of the smaller MDNR reactors as provided by MDNR are attached in Appendix C.

For the small reactors, a weekly volume of 200 mL is being used. For the larger samples, the leachate volume is 300 mL.

4.2.3 Leachate Analysis

Leachates from kinetic tests are being analyzed for the parameters indicated in Table 4 every four weeks beginning on the first rinsing cycle (week 0). Every four weeks on weeks 2, 6, 10 etc. the leachates are analysed for a higher level scan to evaluate trends in major elements. Based on experience, testing of non-reactive rock samples with very low sulfur concentrations is expected to

result in very dilute leachates containing low concentrations of the metals of interest. Back-calculation of metal concentrations from other testwork performed by DNR indicates that nickel and cobalt concentrations could be as low as 0.0002 mg/L (200 ng/L) and 0.00001 (10 ng/L), respectively. Quantification of these low metal concentrations is needed to provide reasonably constrained estimates of metals concentrations in waste rock seepage.

A number of different approaches are available to quantify low levels of metals:

- The routine leachate analysis will achieve a detection level of 0.0001 mg/L (100 ng/L). Should concentrations be undetected, detection limits of 50 ng/L can be obtained with additional processing effort using the same routine method.
- Specialist methods can achieve lower detection limits. These are non-routine (for example, evaporation to increase concentrations) and will need to be developed as the need arises. In order to generate a 10 times decrease in detection limit, the samples would need to be concentrated at 1east 10 times. A composite leachate sample would be prepared from several cycles.
- Existing testwork demonstrates that good correlations exist between cobalt and nickel concentrations in leachates. Detectable nickel concentrations can be used to estimate cobalt concentrations if this relationship can be demonstrated.
- The particle size experiments provide a larger surface area and provide greater likelihood that lower concentrations will be detected.
- In the event of undetectable low levels, detection limit values would be used in subsequent calculations. A scale-up methodology will be agreed upon with MDNR to translate non-detectable concentrations to waste rock seepage concentrations. Section 5.2 provides discussion of possible scale-up approaches.

Table 4: List of Parameters for Humidity Cell Leachate Analyses. Concentrations in mg/L except where indicated

Parameter	Limit	Parameter	Limit
pH (standard units)	-	Acidity	1
Conductivity (µS/cm)	1	Alkalinity	1
Chloride	0.2	Sulfate	0.5
Fluoride	0.05	Total Inorganic Carbon	1
ORP (mV)	-		
Dissolved Elements			
Aluminum	0.001	Molybdenum	0.00005
Antimony	0.0001	Nickel	0.0001 (0.00005)[1]
Arsenic	0.0001	Potassium	0.02
Barium	0.0001	Selenium	0.0002
Beryllium	0.0002	Silicon	0.05
Bismuth	0.0002	Silver	0.00005
Boron	0.005	Sodium	0.01
Cadmium	0.00004	Strontium	0.0001
Calcium	0.01	Tellurium	0.0002
Chromium	0.0002	Thallium	0.00002
Cobalt	0.0001 (0.00005)[1]	Thorium	0.0001
Copper	0.0001	Tin	0.0001
Iron	0.01	Titanium	0.0002
Lead	0.00005	Uranium	0.00005
Lithium	0.0002	Vanadium	0.0002
Magnesium	0.005	Zinc	0.001
Manganese	0.00005		

Notes:

1. Low detection limits are available for cobalt and nickel as shown.

4.3 Quality Assurance/Quality Control

To summarize, QA/QC includes the following components:

- Roughly 10% of all solids analyses are performed in duplicate.
- Roughly 10% of all cell and reactor tests are run as duplicates.
- A blank cell and reactor containing no sample is being operated to check for contamination of leachates by construction materials.
- Individual leachate results are reviewed.
- Ion balances on leachate results are reviewed. In general, imbalances of ±10% are considered acceptable. Re-analysis if requested depending on the nature of the imbalance.
- Data trends in kinetic test leachates are analysed to check for anomalies.

5 Analytical Results for Samples Selected

Table 5 compares analytical results for all samples selected with concentrations calculated from individual intervals in the database. This table reflects the final sample selection following review of data to locate potential replacements, and replacement of one sample. The final three columns indicate the difference between calculated and analytical results using:

$$\% \text{ Difference} = \frac{\text{Target- Actual}}{\text{Target}} \times 100\%$$

Sulfur results showed the greatest percentage differences for samples in the non-reactive category. However, these differences reflect small absolute differences (0.01 to 0.02%) resulting from analytical variability near the detection limit. One sample was replaced in this category because it had 0.05% sulphur compared to target of 0.01%.

A few samples in the reactive class had differences between 50% and 100%. Review of the database indicated no suitable replacements.

One sample in the lean ore class showed a difference of 50%.

Based on these results, MDNR, SRK and PolyMet agreed that the sample selections mostly provided good coverage with respect to the targeted sample ranges indicated in Table 2 and that set up of kinetic tests could proceed.

Table 5: Comparison of Weighted Sample Characteristics Used for Sample Selection with Analytical Results for Interval Composites

Unit	Rock Type	Parameter		Percentiles for Sulfur Contents																			
				Non-reactive			Reactive									Lean Ore							
				NR1	NR2	NR3	P10	P25	P50	P75	P80	P85	P90	P95	P100	P10	P25	P50	P75	P80	P85	P90	P95
1	Anorthositic	Target	Total S, %				0.08	0.1	0.15	0.29				1.09	1.09	0.11	0.26	0.36	0.93				1.95
		Test Material	Total S, %					0.09	0.18	0.05				0.68				0.18	0.86				1.83
	Sedimentary hornfels	Target	Total S, %				0.08	0.35	0.69	2.2	2.32	2.81	3.38	3.5	3.78	0.34	1.37	1.58	1.76				4.91
		Test Material	Total S, %				0.24	0.44	0.55	1.74		2.47						1.46					4.46
	Troctolitic	Target	Total S, %	0.01	0.03	0.05	0.06	0.07	0.1	0.18	0.21	0.26	0.34	0.62	1.97	0.08	0.15	0.24	0.42	0.49	0.55	0.68	0.98
		Test Material	Total S, %	0.02	0.04	0.06			0.07		0.19		0.44	0.88	1.68		0.08	0.36			0.42		0.91
	Ultramafic	Target	Total S, %				0.07	0.08	0.1	0.13	0.2	0.3	0.5	0.8	1.35	0.07	0.09	0.14	0.33	0.55	0.56	0.65	0.81
		Test Material	Total S, %					0.08			0.3	0.2	0.72	1.24				0.16	0.34		0.75		1.2
2	Anorthositic	Target	Total S, %	0.01	0.03	0.05	0.06	0.06	0.08	0.11				0.19		0.09	0.12	0.17	0.21				0.25
		Test Material	Total S, %	0.02	0.02	0.03																	
	Troctolitic	Target	Total S, %	0.01	0.03	0.05	0.06	0.06	0.07	0.09	0.09	0.1	0.11	0.12	0.25	0.05	0.07	0.12	0.18	0.2	0.22	0.26	0.32
		Test Material	Total S, %	0.04	0.04	0.06			0.08		0.07			0.18							0.17		0.15
	Ultramafic	Target	Total S, %	0.03	0.04	0.05	0	0	0	0				0		0.04	0.05	0.07	0.12	0.14	0.16	0.19	0.23
		Test Material	Total S, %	0.06	0.06	0.1														0.12			0.06
3	Anorthositic	Target	Total S, %	0.01	0.03	0.05	0.07	0.08	0.08	0.12				0.14		0.06	0.08	0.12	0.27				0.38
		Test Material	Total S, %	0.02	0.04	0.04																	
	Troctolitic (augite)	Target	Total S, %	0.01	0.03	0.05	0.06	0.06	0.08	0.1	0.11	0.12	0.14	0.19	0.36	0.05	0.13	0.19	0.32	0.35	0.45	0.48	0.52
		Test Material	Total S, %	0.02	0.04	0.06			0.06			0.14		0.25					0.28				0.59
4	Troctolitic	Target	Total S, %	0.01	0.03	0.05	0.06	0.07	0.09	0.18	0.19	0.19	0.48	0.92	1.53	0.09	0.15	0.22	0.47				1.52
		Test Material	Total S, %	0.03	0.04	0.04		0.08		0.21			0.51	0.77									1.37
5	Troctolitic	Target	Total S, %	0.01		0.05	0.07	0.09	0.11	0.16				0.22	0.22	0.12	0.17	0.26	0.37				0.45
		Test Material	Total S, %	0.02		0.06												0.23					0.32
6	Troctolitic	Target	Total S, %	0.02	0.04		0	0	0	0				0		0.01	0.01	0.03	0.07				0.19
		Test Material	Total S, %	0.04	0.05																		
20	Virginia	Target	Total S, %				0.59	1.25	2.98	4.15	4.49	4.85	5.07	6.06	7.45	0	0	0	0				0
		Test Material	Total S, %					2		3.79			5.68										

Unit	Rock Type	Parameter		Deviations																			
				Non-reactive			Reactive									Lean Ore							
				NR1	NR2	NR3	P10	P25	P50	P75	P80	P85	P90	P95	P100	P10	P25	P50	P75	P80	P85	P90	P95
1	Anorthositic	Target	Total S, %																				
		Test Material	Total S, %					-10%	20%	-83%				-38%				-50%	-8%				-6%
	Sedimentary hornfels	Target	Total S, %																				
		Test Material	Total S, %				200%	26%	-20%	-21%		-12%						-8%					-9%
	Troctolitic	Target	Total S, %																				
		Test Material	Total S, %	100%	33%	20%			-30%		-10%		29%	42%	-15%		-47%	50%			-24%		-7%
	Ultramafic	Target	Total S, %																				
		Test Material	Total S, %					0%			50%	-33%	44%	55%				14%	3%		34%		48%
2	Anorthositic	Target	Total S, %																				
		Test Material	Total S, %	100%	-33%	-40%																	
	Troctolitic	Target	Total S, %																				
		Test Material	Total S, %	300%	33%	20%			14%		-22%			50%							-23%		-53%
	Ultramafic	Target	Total S, %																				
		Test Material	Total S, %	100%	50%															-14%			-74%
3	Anorthositic	Target	Total S, %																				
		Test Material	Total S, %	100%	33%	-20%																	
	Troctolitic (augite)	Target	Total S, %																				
		Test Material	Total S, %	100%	33%	20%			-25%			17%		32%					-13%				13%
4	Troctolitic	Target	Total S, %																				
		Test Material	Total S, %	200%	33%	-20%		14%		17%			6%	-16%									-10%
5	Troctolitic	Target	Total S, %																				
		Test Material	Total S, %	100%		20%													-12%				-29%
6	Troctolitic	Target	Total S, %																				
		Test Material	Total S, %	100%	25%																		
20	Virginia	Target	Total S, %																				
		Test Material	Total S, %					60%		-9%			12%										

Notes
Target sulfur levels for samples tested in humidity cells (selected by DNR).

Notes:
Greater than 100% deviation for non-reactive samples and between 50% and 100% deviation for reactive and lean ore samples.
Greater than 100% deviation for reactive and lean ore samples.

6 Implementation Schedule

The majority of waste rock testwork and some lean ore test work was implemented beginning August 8, 2005 following approval by MDNR on July 20, 2005. The balance of lean ore testwork was started on October 28, 2005. Based on the agreement between PolyMet and MDNR that 26 weeks will provide sufficient data for initial analysis, and allowing 8 weeks for reporting and quality assurance evaluations of metals results, the timing of 26 weeks of available data is as follows:

- Waste Rock –Early April, 2006.
- Lean Ore – Late June, 2006.

Based on these time frames, waste rock data will be reported in mid-May and lean ore in early August. These reports will contain recommendations for modifications to the test program. Termination of testwork will consider the following factors:

- Observation of stable trends for all monitored parameters. Stable is defined as a either flat or steady decrease in metal release.
- Demonstration that results for tests are similar and results can be grouped. Selected tests represent the groups of tests will be continued to demonstrate stability of trends.
- Similarity of results with previous DNR testwork.

It is recognized based on MDNR long term experience with kinetic testing of waste rock from other locations in the Duluth Complex that the pH of initial leachates may be elevated compared to long term pH, and that this may result in under-estimation of metal release. This factor will be considered when selecting samples for continuation.

7 Use of Data for Water Quality Predictions

7.1 Purpose of this Section

This section of the plan describes how the data obtained from kinetic tests are used as inputs into prediction of water chemistry for the NorthMet Project. Section 5.2 describes how water quality predictions fit into the overall mine planning process. Section 5.3 provides discussion on scaling up data obtained from small lab experiments to full scale site stockpiles and waste dumps.

7.2 General Context to Water Quality Predictions in Mine Planning

The ultimate objectives of geochemical characterization are to obtain data that can be used as inputs to:

1. Waste management planning (for example, is the rock/tailings acid generating and/or metal leaching?); and

2. Impact assessment (what concentrations of metals and other components might leach from rock/tailings?).

Figure 1 illustrates the general flow of data collection to achieve the above objectives. The bulk geological and geochemical characteristics (indicated by the geological and lithogeochemical models) are interpreted in the context of release rates and geochemical waste classification criteria, and are input into waste scheduling. The resulting waste composition allows release rate information to be used in scale-up calculations, which in turn are used to develop water chemistry predictions.

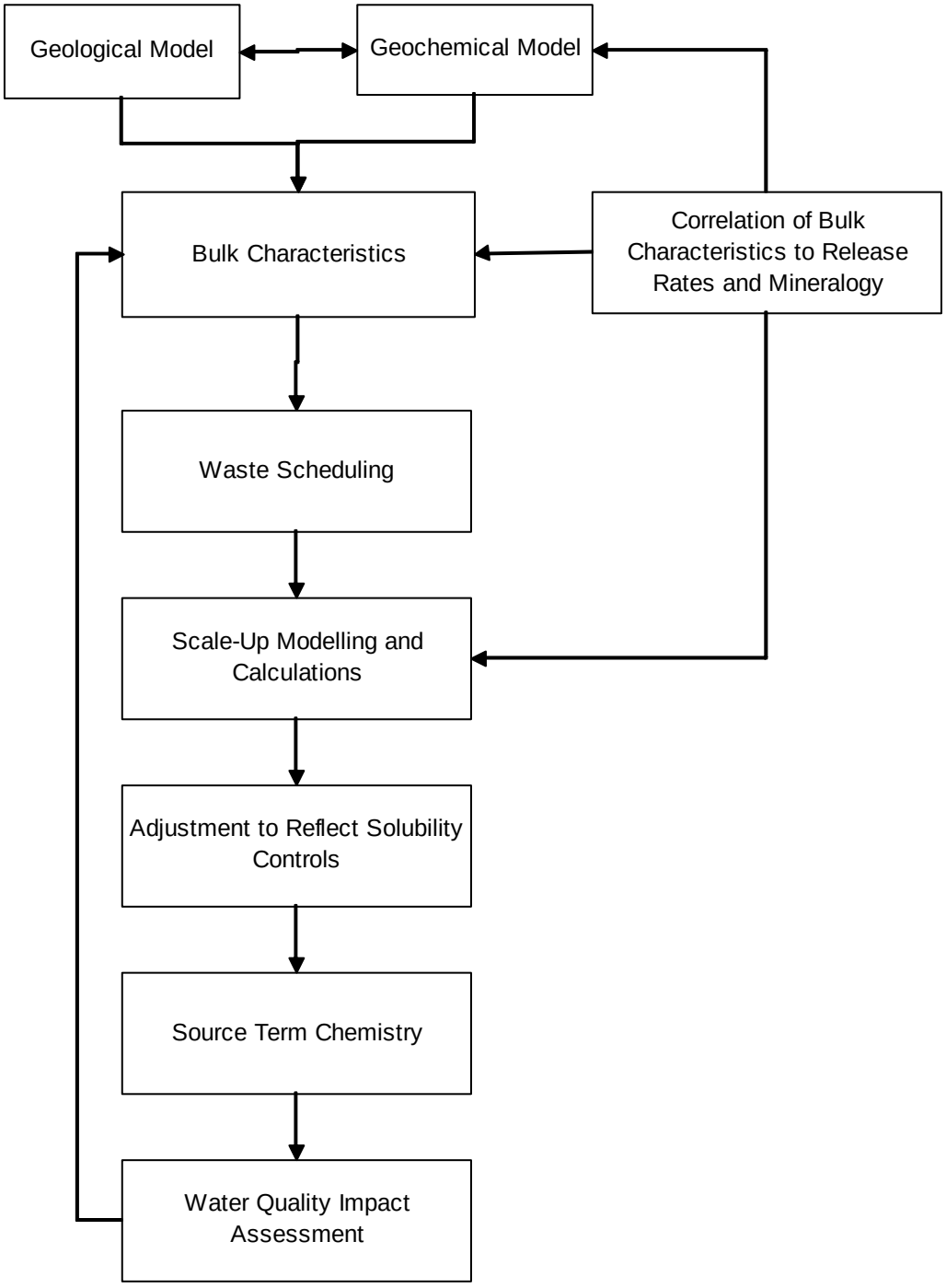

Figure 1: Flow of Information for Water Quality Predictions During Mine Planning

The overall components of geochemical characterization therefore include:

1. Bulk characterization of the rock mass using geological and/or geochemical variables that can be used to predict the waste characteristics for the purpose of waste management planning.
2. Correlation of the characteristics used for bulk characterization with relevant ML/ARD (metal leaching/acid rock drainage) variables and development of criteria based on that correlation (e.g. correlate sulfur content with acid generation and correlate metal leaching rates with bulk metal content).
3. Prediction of contaminant release rates on a mass basis from rock and tailings under various disposal scenarios.
4. Determination of water quality controls (e.g. solubility limits, attenuation effects etc.) for prediction of source term concentrations for individual facilities. Data obtained for this component will be used to adjust water quality predictions obtained from scale-up of laboratory kinetic tests.

All four components are relevant to both objectives and the process is iterative. For example, the last component may indicate parameters that should be used for classification of waste leading to requirements for waste modeling in the first component (Figure 1).

7.3 Approach to Developing Water Quality Predictions

A number of general approaches are available to obtain water quality predictions. These include:

- Theoretical (“First Principals”);
- Site comparisons; and
- Empirical

Discussion of each of these approaches is provided in the following sections.

7.3.1 Theoretical Approach

The theoretical approach involves working from first principals with reaction kinetics and thermodynamics for the processes involved in sulfide oxidation, acid neutralization and metal leaching and attenuation (MEND 2000). There are a number of limitations to this approach which include the difficulty of modeling processes for site-specific conditions. For example, sulfide mineral reactivity can vary widely due to differences in mineral type, occurrence, crystallinity and trace element content. To address these limitations, practitioners typically introduce site-specific calibrations for some processes resulting in predictions that contain empirical aspects. These calibrations may involve actual measurements of oxygen consumption rates, heat generation and seepage chemistry, most of which require that a waste rock dump exists. The ability to make predictions for completely new facilities is therefore limited using purely theoretical approaches (MEND 1995) and should not be pursued further for the NorthMet Project

7.3.2 Site Comparisons

Site comparisons are based on the assumption that mineralogy has a strong influence on water quality, and therefore that comparison of mineral deposits with similar mineralogy is a legitimate approach to making water quality predictions for facilities in the same geological setting.

For example, Caruccio and Ferm (1974) first proposed that paleo-environment is an important factor in determining water quality for coal mines because coal seams formed in salt water environments have higher initial sulfur content and are therefore more prone to generation of acid due to the formation of pyrite during lithification.

Recently, Red Chris Development Co. (2005) compiled data for six porphyry copper mine sites in western Canada and found strong similarities between geographically scattered sites despite variations in host rock geology and climate (for example Figure 2). Porphyry deposits form by interaction of hot water with volcanic or plutonic rocks typically in a sub-volcanic environment. The similarities in drainage chemistry reflected the relatively simple sulfide mineralogy of these deposits and the formation of common alumino-silicate alteration minerals.

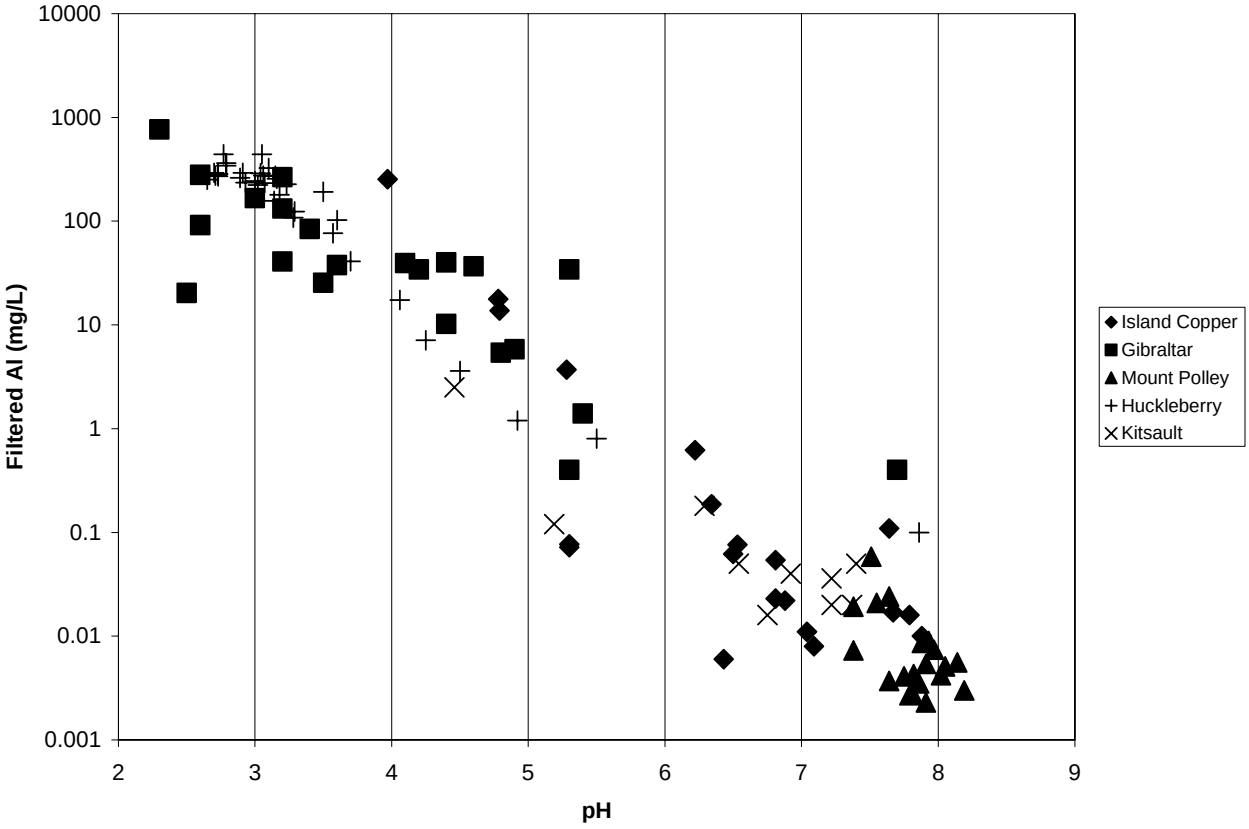

Figure 2: Strong Correlation between Aluminum Concentrations and pH for Porphyry Copper Deposits in Western Canada (Red Chris Development Co. 2004)

A limitation of these comparisons is that geographical proximity does not always guarantee geological similarity because ore forming processes can vary over short lateral and vertical distances, especially as a result of interaction with different rock types. An example of this limitation is seen at the Mount Washington mine site on Vancouver Island where two nearby pits have strongly acidic copper-bearing and non-acidic arsenic-bearing drainages (SRK 2000). The host rock geology is clearly different though for the two pits.

Based on the experience with site comparisons, there are a number of reasons to indicate that mineral deposits within the Duluth Complex can be compared including:

- Uniform, troctolitic to ultramafic composition of the mineralization and metamorphosed siliclastic footwall rocks (Virginia Formation);
- Relatively simple iron sulfide mineralogy; and
- Magmatic rather than hydrothermal mineral deposit formation.

The latter is particularly important because the ore-forming process in the NorthMet Deposit and nearby occurrences have not altered the associated primary silicate minerals. It is concluded therefore that the water quality data collected from nearby full-scale facilities (such as the Dunka Pit Duluth Complex waste rock dumps) and testwork should be factored into the water quality predictions for the NorthMet Project.

7.3.3 The Empirical Method

Introduction

The Empirical Method is also sometimes referred to as “scale-up calculations” because it involves translation of results from small laboratory or field tests to full-scale facilities. The attraction of this approach is that it involves the use of site-specific laboratory and field data, and does not rely on theoretical calculations. The results are transparent and easily explained. However, a significant issue is that the resulting concentrations are typically excessively conservative. This may be attractive for environmental assessment purposes but the resulting predictions may unreasonably over-predict the need for mitigation measures to address potential water quality impacts. A necessary component of the Empirical Method is the adjustment of resulting predictions to reflect basic geochemical controls and experience from other sites.

There are three main steps in the method:

1. Design of a laboratory program to collect rate information;
2. Calculation of concentrations based on rock mixtures, scale-up factors, and hydrological considerations; and
3. Adjustment of calculated concentrations to reflect geochemical constraints indicated by testwork, thermodynamic constraints and experience.

Additional description of these steps is provided in the following sections.

Laboratory Program

The laboratory program is designed to obtain weathering rates, typically expressed as mass of component released per mass of rock per week. Rates are obtained for all rock types and a range of the characteristics for each rock type. Generally, the objective is to obtain rates that can be correlated with bulk characteristics of the rock so that overall rates can be calculated for mixtures. Examples of strong correlations of sulfur content with sulfate release are common and include MDNR's humidity cell data (Figure 3). When good correlations are established, the data can be interpolated between points. For example, in Figure 3, although no rate was specifically measured at a sulfur concentration of 0.9%, it is reasonable to use the overall trend to interpolate a rate. Figure 3 also shows that since the correlation indicates no sulfate release if no sulfur is present, it is also reasonable to extrapolate between 0 and 0.2% sulfur.

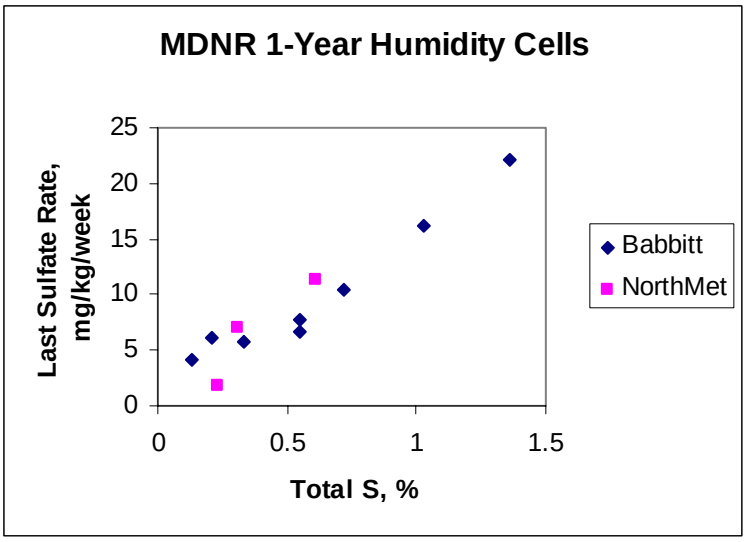

Figure 3: Correlation of Sulfur Content and Sulfate Release for MDNR ASTM Procedure Humidity Cells.

Calculation of Concentrations Using Scale-Up Factors

The purpose of the scale-up calculation is to convert laboratory measured generation rates (R) (for example in mg/kg/week) to seepage concentrations (C) (in mg/L). The scale-up calculations need to consider the rock type mixture, temperature effects, grain size, rock mass, flow path development, and water volume. Pore water concentrations are calculated for each rock type, then mixed according to the proportion of rock types indicated by the mine planners.

Temperature should be considered because oxidation rates decrease as temperatures decreases and vice versa. This correction is typically applied based on the average annual site temperature, and can be calculated using the Arrhenius equation. This equation provides a good approximation of actual

rate decrease observed in laboratory experiments. The laboratory rate (R) is therefore adjusted using a constant factor (k_T) to obtain the adjusted rate (R_a):

$R_a = R.k_T$

This correction should be applied cautiously for reactive materials because the sulfide oxidation reaction is exothermic and will offset cooler site conditions.

The next step is to consider particle size effects. There are two issues to consider:

- Oxidation is a surface area phenomenon. A larger surface area provides a greater reactive surface area, and
- Reactive minerals encapsulated in large rock types do not oxidize at the same rate as exposed reactive particles because oxygen must diffuse through a solid rather than a gas.

Figure 4 illustrates the relationship between particle size and surface area for particles occurring as cubes. The graph shows that below a particle size of 0.1 cm, the available surface area increases exponentially. For larger particles, the area contribution is insignificant. Therefore, a standard humidity cell containing -¼" (0.6 cm) material provides a good representation of the surface area of a rock mixture containing much larger particles. For example, in a typical rock mixture containing 5% by weight finer than this size, the particles finer than 0.6 cm can account for 95% of the surface area.

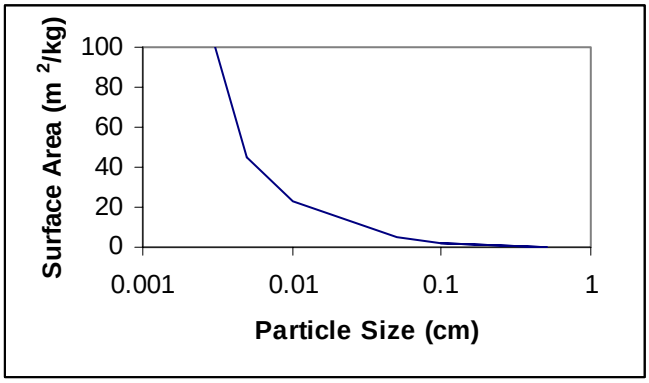

Figure 4: Particle Surface Area as a Function of Particle Size for Cubic Particles

The correction for particle size then becomes a ratio of the fine-grained reactive mass (M_r) to the total mass (M):

$R_a = R.k_T.(M_r/M)$.

The scale up of rate to full scale is then obtained by multiplying by M to obtain:

$R_a' = R.k_T.M_r$ (in mg/week).

R_a' is the scale-up of laboratory rate to field rate for total mass; however, it represents production rather than release because humidity cells are designed to be fully flushed. Under field conditions, the entire rock mass is not flushed due to flow path development. For thin waste rock dumps, flushing is likely to be relatively thorough but as the flow path length increases the degree of flushing decreases (eg Morin 1991; Morin and Hutt 1997). Simple calculations for long flow paths indicate that the proportion flushed may be as low as 20% (Day and Harpley 1992). R_a' can therefore be converted to leached mass (L) by multiplying by a flushed proportion (k_f):

$L = R.k_T.M_r. k_f$ (mg/week).

This leached loading can then converted to a concentration (C) by dividing by the volume of infiltrating water (Q):

$C = R.k_T.M_r.k_f/Q$ (mg/L).

The application of this method to calculation of actual concentrations is illustrated by the following example.

Adjustment of Calculated Concentrations – Example Calculation

Example Dataset

As noted previously, calculation of concentrations using the empirical method often results in unusually high concentrations and it is therefore necessary to evaluate the individual concentrations with consideration of chemical principals and experience from other sites. In order to explain the approach, a recently released dataset from the environmental assessment of a porphyry copper project is used (Red Chris Development Company 2004). For that project, fourteen ASTM-style humidity cells were operated on several rock types which included mineralized quartz diorite and andesitic volcanics and unmineralized siltstones. The resulting calculated average rates from two tests representing a range of conditions are provided in Table 6. The calculation applies to non-acidic drainage.

As an example calculation, concentrations were calculated using an input infiltration rate of 240 mm/year acting on a waste rock dump 50 m high. No temperature correction was applied (k_T), but it was assumed that the bulk of the surface area is contained in 10% of the waste rock and that 20% of the rock is actually flushed.

Evaluation and Adjustment of Major Parameters

The first step in evaluation of the calculated scale-up concentrations is to examine the major parameters (sulfate, calcium, magnesium, etc). An appropriate approach is to enter the data into a thermodynamic equilibrium model (such as MINTEQ, PHREEQE). These models can assist with identifying concentrations that are not supportable thermodynamically. For example, when dissolving common salt in a container of water, only a finite amount can be dissolved after which

any additional salt remains as solid in the bottom of the container. The water is said to be saturated with respect to salt, and the resulting sodium and chloride concentrations in solution can be no greater than when the salt stops dissolving. The reverse is not always true though. It is possible for a solution to be over-saturated with respect to a solid during evaporation. In this case, the energy required to start forming (or nucleating) the first crystals is not available. The thermodynamic models must therefore be used cautiously.

Table 6: Example of Empirical (Scale-up) Calculation of Waste Rock Seepage Chemistry for pH Neutral Drainage Using Humidity Cell Data (from Red Chris Development Co. 2004)

Parameter	Unit	Typical Release for Specific Cells Rates		Calculated Concentrations		P_{95} Concentrations at Other Porphyry Mine Sites (pH>6)
		Major Parameters Low	Major Parameters High	Low	High	
		mg/kg/wk	mg/kg/wk	mg/L	mg/L	mg/L
SO_4		4	73	1633	26978	1526
Mo		0.0004	0.017	0.2	6	0.3
Cu		0.0003	0.0015	0.1	1	2
Pb		0.00005	0.0002	0.018	0.08	0.0002
Zn		0.0004	0.0051	0.2	1.9	0.7
Ni		0.00005	0.0013	0.017	0.50	0.07
Co		0.0001	0.0005	0.020	0.19	0.167
Mn		0.007	0.067	3	25	4
Fe		0.01	0.06	5	23	0.1
As		0.0002	0.0005	0.08	0.20	0.01
Cd		0.0000	0.0001	0.004	0.03	0.002
Ca		3.4	8.8	1254	3252	727
Mg		1.9	1.3	689	477	101
Al		0.040	0.094	15	35	0.1
Na		0.9	32	322	11772	53
K		1.1	0.7	397	275	37
Se		0.0002	0.0089	0.08	3.3	0.2
Si		0.28	0.62	104	229	34
Infiltration	mm/a			240	240	
Density	t/m^3			1.7	1.7	
k_T	-			1	1	
k_f	-			0.2	0.2	
M_r/M	-			0.1	0.1	
Dump Height	m			50	50	

In the case of this dataset, the initial evaluation would allow the following example incompatibilities to be assessed:

- Sulfate and calcium concentrations are much higher in the "high" case than would be expected based on the solubility of gypsum. Since gypsum is well known to form readily, the calcium and sulfate concentrations could be adjusted to reflect precipitation of gypsum.
- Aluminum is not soluble at these levels at neutral pH and could be adjusted to reflect the solubility of basic aluminum sulfates.
- Iron may be present at these concentrations but not under well-oxygenated conditions as would be present in a coarse waste rock pile. Iron hydroxides would be expected to form significantly lowering dissolved iron concentrations.
- Alkalinity was not calculated but can be estimated by assuming a carbonate mineral is present.

Downward adjustment of sulfate obviously impacts the charge balance of the water probably leaving a positive imbalance due to the high sodium and potassium concentrations. While this can be rectified by adding another anion (like chloride), the source of that anion needs to be justified (ie. chloride may not be present in the rock).

In summary, the result of the empirical calculation is a set of concentrations for major ions that typically exceed expected values. This indicates that some products of the weathering reactions remain stored in the rock and are not leached by infiltrating processes. This fact can be applied to adjustment of minor and trace parameters.

Evaluation and Adjustment of Minor and Trace Parameters

Evaluation of these parameters is treated separately because they occur at concentrations that are not a major component of the ion balance and with some exceptions occur at concentrations below limits implied by saturation controls. Copper and manganese concentrations in Table 5 are expected to be close to saturation limits for their carbonates and oxides at neutral pH and can therefore be adjusted to reflect the solubility of these minerals.

Other elements, including for example, lead, zinc, nickel cobalt, cadmium and selenium are predicted to be released at concentrations which seem to be "high" based on experience. In some cases, the concentrations are result of scaling up of detection limit values (for example, arsenic and lead). To refine the predictions for the project in the example, a database of seepage chemistry for other similar porphyry copper mine sites was evaluated. The 95^{th} percentile concentrations from the database are shown in Table 7. Comparison of these concentrations to the calculated concentrations indicated that the calculated low-end lead, arsenic, cadmium and selenium concentrations are higher than the database concentrations, and all calculated high end concentrations are greater than the database concentrations. In other words, the empirical calculation is most likely over-estimating the concentrations of the main trace parameters.

The explanation for this effect is probably that these elements are being released as part of weathering processes but remain stored in the rock. The expected retention of iron represents a sink for these elements through sorption processes. Pyrite is the dominant source of iron, and is likely also the source for many trace elements. The ratio of iron to other metals is very high and represents a significant source of sorptive capacity. Since this process is pH dependent, it is expected that metal concentrations would be negatively correlated with pH. An example of this type of relationship used as part of this example is shown in Figure 5. Data for two sites show that nickel concentrations are strongly related to pH. For example, between pH 4 and 7.5, the data from Huckleberry Mine shows a negative correlation with pH. Likewise, the Island Copper Mine dataset shows a good correlation throughout the pH range.

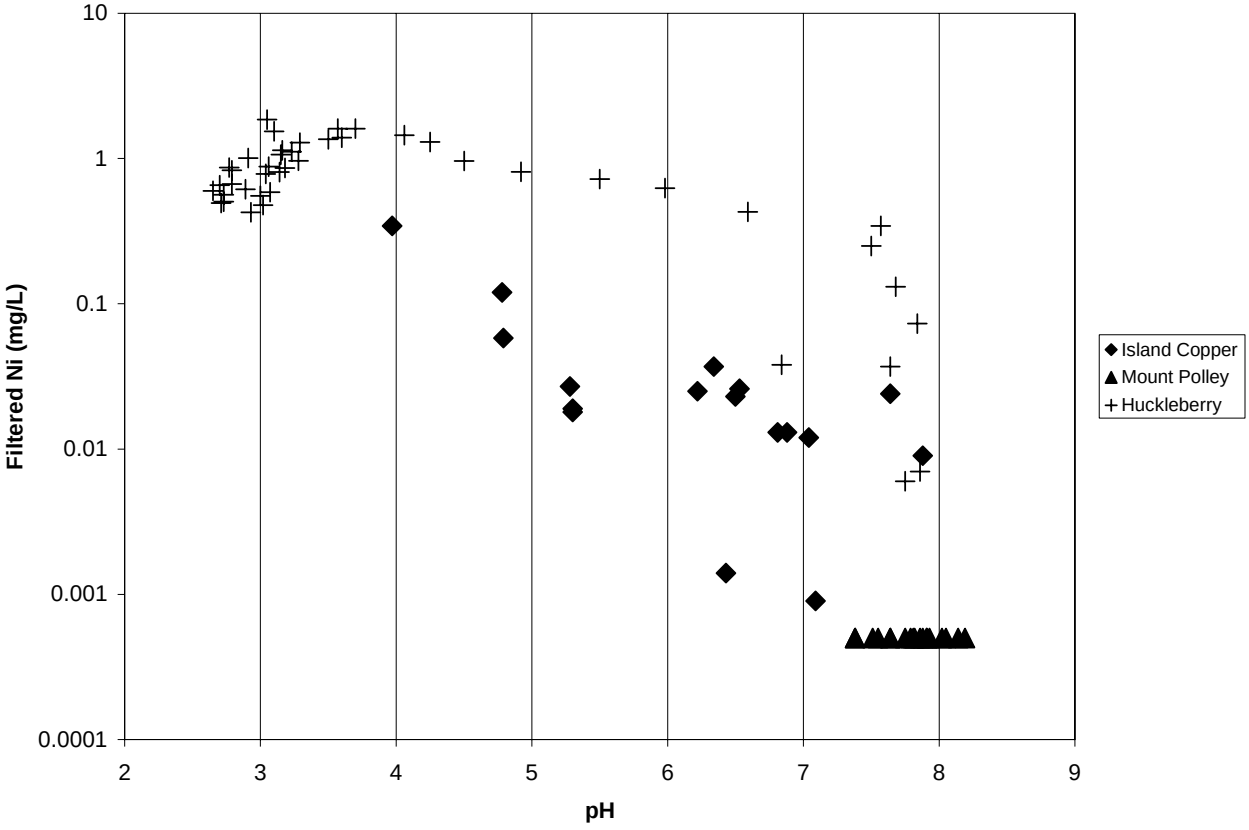

Figure 5: Example of Relationship Between Nickel Concentrations and pH (Red Chris Development Company 2004).

Example of Combined Empirical and Site Comparison Approach in Alaska

Water quality predictions waste rock and dry stack tailings for the recently permitted Pogo Project in Alaska were obtained using a combination of empirically-calculated concentrations scaled-up from humidity cells adjusted to reflect concentrations observed in groundwater, surface water, leach columns, meteoric water extraction procedures and seepage from a pile of waste rock from underground development.

Documentation from the project can be obtained from:

http://www.dnr.state.ak.us/mlw/mining/largemine/pogo/

7.3.4 Implementation for the NorthMet Project

The proposed overall approach is comparable to the combined empirical and site comparisons approach described in the example above. The initial empirical calculation will be based on interpolation and extrapolation of humidity cell results with adjustments for major parameters based on thermodynamic equilibrium calculations and reference to concentrations measured in test pile and waste rock pile drainage.

Scale-up of low concentrations of nickel and cobalt in the non-reactive rock category (sulfur less than 0.05%) is expected to require additional data since the majority of testwork to date has been focused on "reactive" rock containing higher concentrations of sulfur. Since drainage from non-reactive materials is expected to be non-acidic, reliable relationships that indicate correlations between metal concentrations and pH (for example, as shown in Figure 3 and see also Norecol Dames & Moore 1996) may be used to predict metal concentrations. The limitation of the current dataset is that nickel can be expected to be very mobile under non-acidic conditions when the metal to iron ratio is high (for example, if pentlandite is present). A distinctive water quality dataset is needed for low sulfur rock piles. The following approaches may be considered:

- Sampling of seepage from existing Duluth Complex waste rock piles or rock exposures known to contain low concentrations of sulfur.
- Evaluation of oxide coatings to understand the attenuation of metals and comparison with loads leached from humidity cells. Generally speaking, sulfate is conservative in slowly reactive humidity cells, therefore if the nickel to sulfur ratio is lower in the leachate than the sulfide minerals, then nickel is being attenuated (assuming that nickel originates from oxidation of sulfides). Likewise, comparison of iron and nickel release with iron and nickel ratios in sulfides and oxides coatings will indicate how nickel is attenuated relative to iron.

7.4 Conclusions

An empirical scale-up approach is proposed to translate weathering rates observed in humidity cells to full-scale concentrations. The resulting predicted concentrations will be evaluated and adjusted based on solubility constraints and data from existing monitoring. The incorporation data from past or existing waste rock facilities (AMAX test piles, Dunka Pit waste rock) ensures that the predictions are consistent with large scale operational experience. Additional testing may be designed to evaluate mobility and attenuation of metals such as nickel and cobalt.

8 References

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Appendix A

Parameter Lists and Detection Limits for Analysis of Solids

ALS/CHEMEX METHOD CODE	ITEM	UINTS	CHEMEX DETECTION LIMIT
ME-ICP61 (four acid)	CU%	%	0.001
ME-ICP61 (four acid)	NI%	%	0.001
S-IR08 (LECO SULFUR)	S%TOT	%	0.01
ME-ICP61 (four acid)	S%ICP	%	0.01
PGM-ICP23 (30 GRAM)	PT_PPB	PPB	5
PGM-ICP23 (30 GRAM)	PD_PPB	PPB	1
PGM-ICP23 (30 GRAM)	AU_PPB	PPB	1
ME-ICP61 (four acid)	CO_PPM	PPM	1
ME-ICP61 (four acid)	AG_PPM	PPM	0.5
ME-ICP61 (four acid)	ZN_PPM	PPM	2
ME-ICP61 (four acid)	CD_PPM	PPM	0.5
ME-ICP61 (four acid)	MO_PPM	PPM	1
ME-ICP61 (four acid)	PB_PPM	PPM	2
ME-ICP61 (four acid)	AS_PPM	PPM	5
ME-ICP61 (four acid)	CR_PPM	PPM	1
ME-ICP61 (four acid)	V_PPM	PPM	1
ME-ICP61 (four acid)	TI%	%	0.01
ME-ICP61 (four acid)	AL%	%	0.01
ME-ICP61 (four acid)	CA%	%	0.01
ME-ICP61 (four acid)	FE%	%	0.01
ME-ICP61 (four acid)	K%	%	0.01
ME-ICP61 (four acid)	NA%	%	0.01
ME-ICP61 (four acid)	MG%	%	0.01
ME-ICP61 (four acid)	MN_PPM	PPM	5
ME-ICP61 (four acid)	P_PPM	PPM	10
ME-ICP61 (four acid)	BA_PPM	PPM	10
ME-ICP61 (four acid)	BE_PPM	PPM	0.5
ME-ICP61 (four acid)	BI_PPM	PPM	2
ME-ICP61 (four acid)	SB_PPM	PPM	5
ME-ICP61 (four acid)	SR_PPM	PPM	1
ME-ICP61 (four acid)	W_PPM	PPM	10

ALS/CHEMEX METHOD CODE	ITEM	UINTS	CHEMEX DETECTION LIMIT
ME-ICP41 (Aqua regia digestion)	CU%	%	0.001
ME-ICP41 (Aqua regia digestion)	NI%	%	0.001
ME-ICP41 (Aqua regia digestion)	S%ICP	%	0.01
ME-ICP41 (Aqua regia digestion)	CO_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	AG_PPM	PPM	0.2
ME-ICP41 (Aqua regia digestion)	ZN_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	CD_PPM	PPM	0.5
ME-ICP41 (Aqua regia digestion)	MO_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	PB_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	AS_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	CR_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	V_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	TI%	%	0.01
ME-ICP41 (Aqua regia digestion)	AL%	%	0.01
ME-ICP41 (Aqua regia digestion)	CA%	%	0.01
ME-ICP41 (Aqua regia digestion)	FE%	%	0.01
ME-ICP41 (Aqua regia digestion)	K%	%	0.01
ME-ICP41 (Aqua regia digestion)	NA%	%	0.01
ME-ICP41 (Aqua regia digestion)	MG%	%	0.01
ME-ICP41 (Aqua regia digestion)	MN_PPM	PPM	5
ME-ICP41 (Aqua regia digestion)	P_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	B_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	BA_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	BE_PPM	PPM	0.5
ME-ICP41 (Aqua regia digestion)	BI_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	GA_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	HG_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	LA_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	SB_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	SC_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	SR_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	W_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	TL_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	U_PPM	PPM	10

ALS/CHEMEX METHOD CODE	ITEM	UINTS	CHEMEX DETECTION LIMIT
ME-ICP06--whole rock geochemisrty by ICP-AES	SIO2	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	AL203	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	TIO2	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	FE2O3	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	CAO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	MGO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	MNO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	NA2O	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	K2O	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	P2O5	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	BAO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	SRO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	LOI	%	0.01

Appendix B

Options and Variance in ASTM Humidity Cell Procedure

9. Sample Preparation

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
9.1	Air dry as-received bulk samples of solid material to prevent the additional oxidation of reactive minerals or compounds. If air drying is not practicable, oven dry the solid material at a maximum temperature of 50 ± 2°C for 24 h, or until a constant weight is reached.	Samples were air-dried at room temperature (~ 20 °C).	
9.1.1	If exploration-generated or run-of-mine solid material samples are not readily available, archived dried and crushed samples from geological exploratory or development drilling programs may be used for preliminary evaluations of ore and waste rock from new operations; this is provided that the available solid material samples are not significantly finer than 95 % passing a No. 12 (1.7-mm) sieve. Document the sample drying and preparation procedures used during the drill sampling program in order to interpret the results properly. Evaluate the effects of drying temperature on metals volatilization (for example, mercury in cinnabar vaporizes at temperatures exceeding 80 to 90°C) and mineral morphology and chemistry modifications (for example, on heating at temperatures exceeding 100°C, chalcocite changes crystal form and is oxidized subsequently from Cu_2S to CuO, $CuSO_4$, and SO_2). Especially ensure that the effects of particle size distribution changes resulting from the more finely crushed sample are considered in the interpretation (this is, the potential for increased liberation of acid-producing and acid-consuming minerals with an attendant increase in mineral surface area).	NA	
9.1.2	In mining waste evaluations, the particle size for mill tailings will be significantly finer (commonly less than 150 µm/100 mesh) than the particle size distributions from ore and waste rock. Pilot plant tailings should be used if mill tailings are not available.	NA	
9.2	Screen the air-dried bulk samples through a 6.3-mm (¼-in.) screen in accordance with Test Method E 276. Crush any oversize material so that 100 % passes the screen.	ASTM	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
Note 7	Caution: Recent accelerated weathering studies of run-of mine waste rock from metal mines demonstrate that crushing a bulk sample so it passes a 6.3-mm (¼-in.) screen may change the character of the sample by artificially increasing liberation and consequent surface areas of acid-producing and acid-consuming minerals contained in the + 6.3-mm (¼-in.) material. A suggestion for avoiding this problem is to segregate the - 6.3-mm (¼-in.) fraction by screening rather than crushing, and to test that fraction according to the protocol and equipment described in this text. The + 6.3-mm (¼-in.) material can be tested separately (for example, Brodie, et al (10) describe a large-scale humidity cell test that would accommodate – 75-mm material). Samples from the drill core and cuttings also present material sizing problems, which must be considered when interpreting drill core and cuttings accelerated data. The drill core must be crushed to -6.3-mm (¼-in.) to fit the cell described in this test method. The resulting size distribution from crushing will differ from that of run-of-mine due to differences in fracture patterns inherent to blasting practices that produce run-of-mine material. By contrast, drill cuttings size fractions are commonly less than 6.3-mm (¼-in.) due to the rotary-percussive nature of obtaining the sample.	NA	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
9.3	Mix and divide the bulk sample to obtain a representative test unit with a weight in the range of 8 to 10 kg, using a riffle splitter with 1-in. (2.54-cm) chutes. Divide the test unit into eight nominal 1-kg specimens. Seal each test specimen in a moisture-barrier bag.	Samples mixed by riffle splitter, but different sample weights were available (see column at the right).	All material available for each sample was mixed by the riffle splitter (0.552 kg – 10.54 kg). The test unit was divided into the following amounts: Samples received June 06, 2005 (lean ore): 150 – 200 g for Chemex Assay; Reject for archive, screen assay, etc. Samples received May 20, 2005 (waste rock): 200 g for Chemex Assay, 50 g of crushed archive, 100 g for screen assay, store rejects for HC
Note 8	The dried sample should be mixed through the riffle splitter at least once before making any splits; recombine the splits resulting from the sample mixing exercise by pouring individual splits either over each other or through the splitter again. Once the actual split is made, it is wise to re-mix it (according to the above procedure) prior to making the next split.		Samples were mixed through the riffle splitter once.
9.4	Select one test specimen at random, and determine the moisture content by weighing and drying to constant weight at 80 ±5°C.		Determined at 20 °C
9.4.1	Crush the dried test specimen so that at least 95 % passes a 1.7-mm (10-mesh) screen, in accordance with Test Method E 276.		See 9.3
9.4.2	Divide the crushed test specimen in half twice, using a riffle splitter with 6.35-mm (¼-in.) chutes, and select a ¼ subsample at random.		See 9.3
9.4.3	Transfer the selected subsample to a ring and puck grinding mill and grind to a nominal of 95 % passing a 150-µm (100-mesh) screen, in accordance with Test Method E 276. Use the subsample for chemical and mineralogical characterization of the test unit.		See 9.3

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
9.5	Select one test specimen at random, and determine the particle size distribution in accordance with Test Method E 276.		100 g were removed after 9.3 for screen assays: -1/4" + 10 mesh, -10 mesh + 35 mesh, -35 mesh + 100 mesh, -270 mesh; each size was weighed and submitted to Chemex for Total S (S-IR08) and four acid digestion (ME-ICP61).
9.6	Select one test specimen at random for use in the accelerated test method. Divide the test specimen into four nominal 250-g subsamples using the riffle splitter with 25.4-mm (1-in.) chutes, and label and store in vapor-barrier bags until it is time to load the humidity cells.	See 9.3 - variance column	
9.7	Reserve the remaining test specimens for replicated testing or to resolve disputed results.	See 9.3 - variance column	

10. Apparatus Assembly

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
10.1	The humidity cells are table-mounted at a height sufficient to accommodate the placement of both the humidifier and one Erlenmeyer flask for effluent collection from the bottom of each cell. During the water-saturated and dry-air portions of each weekly cycle, feed air is metered to the bottom of each cell at the selected rate (1 to 10L/min). Feed air for the three-day dry–air portion is routed first through a desiccant column and then to each of the cells through a dry-air manifold. Feed air for the water-saturated air portion is routed through a water-filled humidifier by means of aeration stones or gas dispersion fritted cylinders/disks, and then to each humidity cell lid air exit port to prevent the short circuiting of air through cells containing more permeable solid material samples. A separatory funnel rack is mounted on the table that holds the cells if the weekly water leach is applied dropwise (drip trickle). Multiple separatory funnels (one for each cell) are held in the rack during the drip trickle leach that is performed on the seventh day of each weekly cycle. The separatory funnel can be used to meter the required water volume slowly down the sides of the cell wall until the sample is flooded if the weekly leach is to be a flooded leach.	Humidity cells are constructed of acrylic tubing with an inside diameter of four inches and an overall height of twelve inches, with an acrylic base plate. The base plate is glued to the tube and threaded with a nylon hose adapter to which a length of tubing is attached to allow for leachate drainage into a collection container. A perforated PVC support plate is positioned inside the cell, one inch above the base plate and covered with six layers of nylon mesh. A nylon adapter is threaded into the side of the cell between the support plate and the base plate and a length of tubing was connected from the side adapter to the humidifier to facilitate the inflow of humid air to the cell. A dry air line is also connected to each cell. Each cell is covered with a removable acrylic lid.	Approximately 16 cells per humidifier Flood leaching: peristaltic pump using a peristaltic pump Temperature: 20 ± 2°C. Feed air rate to be determined.

11. Procedure

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.1	Cell Loading:		
11.1.1	If more than one humidity cell is used at one time, label each with a sequential number, and use the same number for the matching collection vessel (Erlenmeyer flask).	ASTM	
11.1.2	Weigh each humidity cell (without its lid) and each collection vessel; record the tare weights of each to the nearest 0.1 g.	ASTM	
11.1.3	Cut the filter media (such as 12-oz/yd^2 polypropylene described in 6.11) to the humidity cell's inside diameter dimensions so that it fits snugly yet lies flat on the perforated support.		PVC perforated disk & nylon mesh
11.1.4	Re-weigh the humidity cell, and record the resulting tare to the nearest 0.1 g; the original cell tare (11.1.2) minus the new cell tare is the weight of the filter media.	ASTM	
11.1.5	Transfer the contents from each of the four bags containing the 250-g samples (9.6) into the humidity cell. Prior to the transfer, mix the contents of each bag by gentle rolling to eliminate possible stratification that may have occurred during sample storage.	ASTM	
11.1.6	Re-weigh the loaded cell, and record the weight to the nearest 0.1 g; the loaded cell weight minus the combined cell and filter-media tare weight is the weight of the sample charge.	ASTM	
11.2	First Leach:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.2.1	The first leach (whether drip trickle or flooded), designated as the Week 0 leach, initiates the 20-week long humidity cell test and establishes the starting or initial characteristics of the leachate. Either a 500-mL or 1-L volume of water may be used for the weekly leachates, depending on the weekly pore volume desired or the quantity of solution required for analytical purposes; however, once a weekly volume has been selected, that weekly volume must remain constant throughout the 20-week testing period. A centrifuged cell culture of *Thiobacillus ferrooxidans* may be used in the first leach in order to ensure that optimum conditions for accelerates weathering are present at the beginning of the test.	500 mL Flood Leach	
Note 9	In the testing of mining wastes, cation (including metals and trace metals) and anion loadings are commonly high in the Week 0 leachate due to the dissolution of pre-existing soluble oxidation salts present in the sample prior to sample collection. The average number of weekly accelerated weathering cycles required to flush these pre-existing salts ranges from 3 to 5 weeks. Oxidation products observed during these 3 to 5 weeks are principally from the pre-existing salts, while those products observed after this period are considered to be solely a function of the accelerated weathering procedure. A method for estimating the amount of pre-existing oxidation salts present in a solid material sample is described by Sobek, et al (6). A comparison of estimated salt storage data obtained using this method with the first thee weeks of humidity cell effluent loadings from three different samples is describes by White and Jeffers (7).	NA	
11.2.2	Fill a separatory funnel with for each cell with de-ionized water using a volumetric flask. If the leach is to be performed using the drip trickle method, set each separatory funnel above its corresponding cell, and adjust the drip rate (approximately 3 to 4 L/min) so that the solid material sample is wetted thoroughly but not flooded.	NA	
11.2.3	A minimum of 2 to 3 h is commonly required to complete the drip trickle leach.	NA	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.2.4	If the leach is to be performed by flooding, the separatory funnel can be used to meter the selected water volume slowly down the sides of the cell wall until the sample is flooded. This application method reduces hydraulic agitation of the sample surface commonly caused by pouring liquid from an open-mouthed vessel. Alternatively, flooding may be accomplished by any application apparatus (for example, a peristaltic pump) that supplies the selected volume of leachant at a reasonable rate without causing agitation and suspension of the finer fractions contained in the sample charge.	ASTM	
11.2.4.1	Allow the flooded cell to sit for a period of 1 h before draining the leachate into the Erlenmeyer collection flask. The 1-h leach time commences after all of the leachant has been placed in the cell. The solid material sample should be saturated and covered with leachant to a depth sufficient to maintain sample saturation. In testing mining wastes, the observed depth of leachant cover from a 500-mL flooded leach performed in 10.2-cm (4.0-in.) ID cells is approximately 2.5 cm (1.0 in.).	ASTM	
11.2.5	The following is performed once the leaching process has been completed: to reduce the effects of evaporation, and to prevent the contamination of each cell by airborne contaminants, place the lids on their corresponding cells and let the cells complete the leachate draining process for the remainder of the leaching day and overnight.	ASTM	
11.2.6	Disconnect the cells on the day following the leach, and weigh and record the weight of each cell and Erlenmeyer collection flask. Set each filled collection flask aside for leachate analyses. (Measurements of pH and Eh and sample preservation procedures must be performed as soon as possible after leachate collection.) Return each cell, replace the filled collection flasks with clean, tared Erlenmeyer flasks, hook up all connections, and begin the dry-air cycle.	ASTM	
11.3	Dry-Air Cycle:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.3.1	The commencement of the three-day dry-air period marks the beginning of each new weekly cycle of the accelerated weathering humidity cell test; the first full-week cycle after the first leaching is designated Week 1; subsequent weeks (commencing with the second dry-air period) are designated as Week 2, Week 3 … . Week n, etc.	ASTM	
11.3.2	To perform the dry-air cycle, feed air is metered to the humidity cell array with a flowmeter (see 6.3) set at a target rate in the range of 1 to 10 L/min per cell, depending on the objectives of the testing. The air flow rate must be checked daily and adjusted to the target value ± 0.5 L/min.	ASTM	
11.3.3	Feed air from the flowmeter is routed first through a desiccant column and then to each of the sells through a dry-air manifold. Air exiting the desiccant column should have a relative humidity of less than 10 % as measured with a hygrometer (see 6.23).	ASTM	
11.3.4	To maintain similar positive air pressure through the cells, attach a water-bubbling vessel to each humidity cell air exit port coming out of the humidity cell lid; a 50-mL Erlenmeyer flask with a rubber stopper containing a vent and air inlet tube serves as a simple and efficient bubbler.	ASTM	
11.3.5	The dry air is passed through each humidity cell for three days. Air flow rates from each of the cells should be checked each day, recorded, and adjusted, if necessary. See also Note 10.	ASTM	
11.4	Wet-Air Cycle:		
11.4.1	The three-day wet-air period commences on the fourth day of each weekly cycle.	ASTM	
11.4.2	To perform the wet-air cycle of the method, feed air is routed through a water-filled humidifier via aeration stones or gas dispersion fritted cylinders/disks and then to each humidity cell.	ASTM	
11.4.3	The water temperature in the humidifier is maintained at 30 ± 2°C to ensure that the sparged air maintains a relative humidity of approximately 95 % as measured with a hygrometer (see 6.23) from one of the humidifier exit lines. Air flow rates to each of the cells should be checked each day, recorded, and adjusted, if necessary.	ASTM	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
Note 10	It is good practice to measure the air flow rates and relative humidity of the air exiting each humidity cell during each day of the three-day dry- and wet-air periods; the measurements should be taken at the same time each day from the humidity cell air exit port; these measurements can be accomplished by installing a quick-disconnect fitting in the tubing that connects the air exit port to the bubbler.	NA	
Note 11	Coals spoils in eastern states are commonly saturated; Caruccio (10) has suggested the following geographic control alternative to the dry-air versus saturated-air scheduling: (1) Eastern States Samples – Six days of saturated air (versus three days dry/three days wet); and (2) Western States Samples – Three days dry/three days wet.	NA	
11.5	Subsequent Weekly Leaches:		
11.5.1	A second leach with water is performed on the day following the end of the three-day wet-air period (that is, day seven of the first weekly cycle). This leach marks the end of the first weekly cycle and is designated as the Week 1 leach.	ASTM	
11.5.2	Subsequent leaches are designates as Week2, Week 3 … Week n, and they mark the end of the weekly cycle for that numbered week. Perform each weekly leach as described in 11.2.2 – 11.2.5. Weekly weighing of the test cells is optional.	ASTM	No weekly weighing of the cells.
11.6	It is recommended that the weekly accelerated weathering cycles described in 11.2, 11.3, 11.4 and 11.5 be performed for a minimum of 20 weeks.	ASTM	
Note 12	Additional weeks of accelerated weathering may be required to demonstrate the nature of the material, depending on the chemical composition of the solid material. For some metal mining wastes, researchers have shown that as much as 60 to 120 weeks of accelerated weathering data may be required to demonstrate the complete weathering characteristics of a particular sample (7, 12). The criteria for ending the testing may be site specific and should be agreed before initiating the testing.	ASTM	
11.7	Leachate Analyses:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.7.1	Analyze the leachates for specific constituents or properties, or use them for biological testing procedures as desired, using (1) appropriate ASTM test methods or (2) methods accepted for the site where disposal will occur. Where no appropriate ASTM test method exists, other test methods may be used and recorded in the report, provided that they are sufficiently sensitive to assess potential water quality impacts at the proposed disposal site. Suggested minimum weekly analyses should include pH, Eh, conductivity, and selected metals could be analyzed less frequently (for example, at Weeks 0, 1, 2, 4, 8, 12, 16, and 20), especially if changes in leachate chemistry are slow. Whether visible phase separation during storage of the leachates occurs or not, appropriate mixing should be used to ensure the homogeneity of the leachates prior to their use in such analyses.	At the end of weekly cycle the volume of leachate collected is recorded. The leachate is filtered through a Gelman magnetic filter funnel fitted with a membrane filter with pore size of 0.45 microns and analyzed for the parameters listed in Table 2 of the RFP. Filtered leachate samples will be submitted to ALS Environmental/Cantest Ltd. for dissolved metals analysis as requested in Table 4 of the Waste Rock and Lean Ore Geochemical Characterization Plan. Conductivity, Eh, and pH are measured in the CEMI laboratory using standard procedures. An aliquot of filtered leachate is titrated with standardized sulphuric acid to pH 4.5 to calculate total alkalinity. Standardized sodium hydroxide is used to titrate an aliquot of leachate to pH 4.5 and to pH 8.3 to calculate total acidity. Analysis frequency: pH, cond, Eh every cycle; SO4, Cl, F, alkalinity, TIC, acidity cycle 0, 2, 4, 6 etc.; ICP-MS including Hg and Si cycle 0, 4, 8, 12, etc., ICP-ES including Si cycle 2, 6, 10, 14, etc.	
11.7.2	Table 1 is an example of a spreadsheet format used for recording 20 weeks of leachate analytical data.	ASTM	
11.7.3	Fig. 5 is an example of a method used to plot the temporal variation (by week) of leachate pH, sulfate load, and cumulative sulfate load from 21 weeks of accelerated load and release rates).	ASTM	
11.8	Weathered Solid Material Analyses:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.8.1	Weigh the humidity cell after collection of the final effluent and completion of a three-day dry-air period.	ASTM	
11.8.2	Transfer the weathered residue and filter media to a clean drying pan, and dry to constant weight at 50 ± 5°C. Record the final weight.	ASTM	
Note 13	Perform any gross sample examination (for example, sample texture and weathering product mineralogic characterization) desired for the weathered residues prior to pulverization. To facilitate such an examination, empty the humidity cell contains into a clean drying pan carefully by pushing gently on the bottom of the perforated plate with a wooden dowel until the sample exits the cell mouth. The perforate plate is accessed through the humidity cell drain port.	NA	
11.8.3	Identify and mark the top versus bottom portions of the sample for gross sampling purposes. Formations of cemented lumps of sample termed “ferricrete” that result from the accelerated weathering process arte common in iron-sulfide-mineral rich samples. Depending on the sample mineralogy, the degree of “ferricrete” cementation may vary vertically within the sample, and the investigator may wish to segregate the sample into upper, middle, and lower thirds to document and characterize such changes.	Procedure to be determined	
11.8.4	After drying to constant weight and prior to splitting, use an instrument such as a rolling pin to break up cemented lumps in the sample (if the cemented lumps cannot be sufficiently reduced to pass through the chutes of a riffle splitter, remove, record, and weigh separately):	ASTM	
11.8.4.1	Split the sample into halves using a riffle splitter with 2.54-cm (1-in.) chutes, and reserve one half to determine the particle size distribution in accordance with Test Method E 276.		Repeat same screen assay method as for pre-test characterization (s.9.5)
11.8.4.2	Split the remaining half sample into two quarters using a riffle splitter with 2.54-cm (1-in.) chutes, and submit one quarter for mineralogical characterization; pulverize the other quarter in either a ring-and-puck or disk-pulverizing machine to 95 % passing a 150-µm (100-mesh) screen in accordance with Test Method E 276.	Procedure to be determined	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.8.5	Mix the pulverized residue in a blender or on a rolling cloth. Use the prepared residue for chemical characterization and for comparison with the pre-weathered solid material sample.	Procedure to be determined	

Appendix C

Design of MDNR Reactor

Day, Stephen

From: Kim Lapakko [kim.lapakko@dnr.state.mn.us]
Sent: Tuesday, May 17, 2005 9:55 AM
To: Stephen Day
Cc: Dave Antonson; Jennifer Engstrom; Paul Eger
Subject: RE: Small reactor

Attachments: MN DNR psize methods 050517.doc

MN DNR psize methods 050517.do...

Steve,

Attached is a description of the reactors, masses, and rinse volumes used for various size fractions of Duluth Complex rock in our particle size experiment. As indicated in the attachment, I won't have access to the trace metal data from that experiment until tomorrow. I will need to examine this to help evaluate the expected metal concentrations in drainage relative to detection limits. I'm not sure it will give us as much as hoped because the sulfur contents of the samples typically were on the order of 0.9% to 1.3%. This may make extrapolation by more than an order of magnitude tenuous. It will be another pertinent piece of information.

Kim

>>> "Stephen Day" <sday@srk.com> 5/17/2005 11:18:50 AM >>>
Dave

A design drawing should be fine along with description of the procedure.

The main question is what do you do to scale-up the sample mass as the particle size increases? I want to copy your procedure exactly.

Thanks
Steve.

-----Original Message-----
From: Kim Lapakko [mailto:kim.lapakko@dnr.state.mn.us]
Sent: Tuesday, May 17, 2005 8:38 AM
To: Stephen Day
Cc: Dave Antonson
Subject: Small reactor

Steve,

Dave Antonson will email a figure depicting our small reactor, along with some design details (perforated plate, adehesive, filter). He could also send a reactor. Please contact him directly, with an address to send it, if you think that would be helpful.

Kim

17 May 2005

Steve,

In our particle size tests we used a small reactor and 75-g mass for particle sizes of –270, +270/-100, and +100/-35 mesh. We used the ASTM cell and 1000-g mass for +35/-10, +10/-0.25 inch, and +0.25/-0.75 inch particle sizes. For rinse volumes, we used 200 mL for the 75-g samples and 300 mL for the 1000-g samples. The 300-mL rinse volume was determined as the quantity of water, rounded up to the nearest 100 mL, required to submerge the solids.

I won't have access to the metal release data for the particle size experiment until tomorrow. As mentioned on the phone, sulfate release rates appear to vary linearly with surface area. It seems likely that nickel release rates will vary similarly, and I'll look into this further tomorrow. Hopefully this information will shed some light on the maximum particle size question.

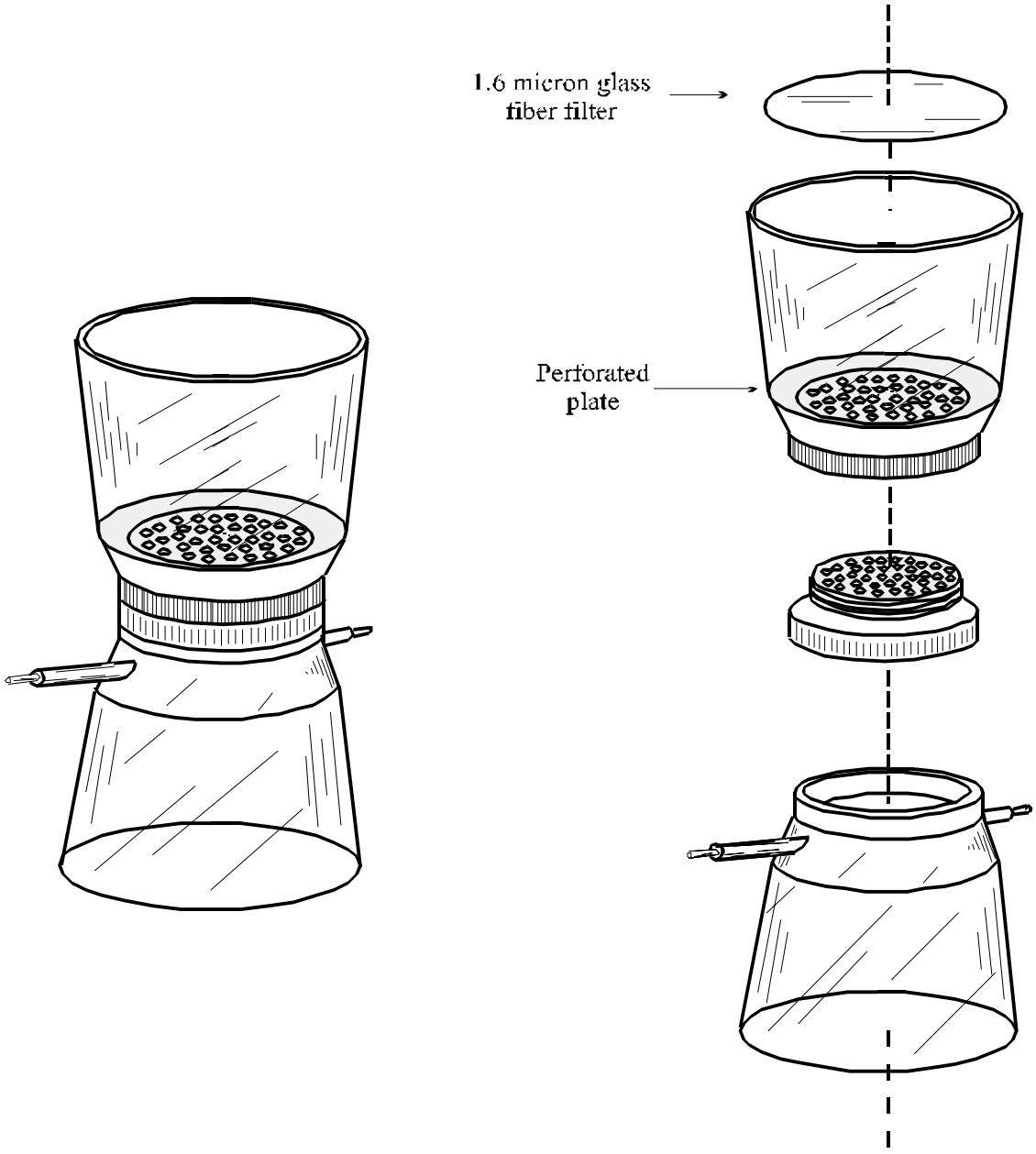
Day, Stephen

From: Dave Antonson [dave.antonson@dnr.state.mn.us]
Sent: Tuesday, May 17, 2005 11:53 AM
To: Kim Lapakko
Subject: reactor

Attachments: small reactor.doc

small reactor.doc
(271 KB)

see if this makes any sense. you can edit it if you want. if it seems adequate you can forward it to steve. maybe he doesn't need a sample of the base.


1.6 micron glass
fiber filter
Perforated
plate

The reactors were purchased from Millipore Corporation (1-800-645-5476). They are 47 mm Sterifil aseptic systems. You will need the 250 ml receiver flask, 250 ml funnel (top), silicone o-rings, and the filter holder base and support screen.

The perforated acrylic plastic base was purchased as flat stock and fabricated to fit the top funnel. The plates are 1/8" thick, 2 1/4" in diameter and tapered to fit into the reactor top. Approximately sixteen 1/16" holes were drilled in the plate. The plate was glued into the reactor using acrylic solvent cement purchased from United States Plastics (1-800-537-9724). Catalog # 44629 for 5 oz. tube. The acrylic flat stock was also purchased from United States Plastics.

After the plate is glued into the top of the reactor there should be approximatly a 3/8" gap between the bottom of the perforated plate and the top of the support screen of the filter unit.

The filter that rests on the perforated plate is a 55 mm Whatman GF/A glass microfibre filter (catalog # 1820 055).

Note: Before adding the solids to the filter you should wet the filter slightly with distilled water so no solids escape around the filter.

Attachment 2-4

SRK Consulting. Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan, NorthMet Project, Minnesota. May 2006.

Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan NorthMet Project, Minnesota

Prepared for:

Polymet Mining Inc.
#2350 – 1177 West Hastings Street
Vancouver, BC
V6E 2K3

Prepared by:

SRK Consulting
Engineers and Scientists

Project Reference Number
SRK 1UP005.001

May 2006

Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan NorthMet Project, Minnesota

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SRK Project Number 1UP005.001

May 2006

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1 Introduction

1.1 Background

PolyMet Mining Inc (PolyMet) is proposing to develop the NorthMet Project (Dunka Road Project of US Steel) near Babbitt, Minnesota. As a part of the Minnesota Department of Natural Resources (MDNR) "Permit to Mine" process a complete "mine waste characterization" will be required (Minnesota Rules Chapter 6132.1000). This document describes the plan developed for testing of flotation tailings and hydrometallurgical residue samples for the NorthMet Project.

The issues associated with tailings and residues at the NorthMet are expected to include acid rock drainage (ARD) and leaching of some heavy metals. The latter in particular are expected to include nickel and cobalt both of which do not require acidic conditions to be mobilized at elevated concentrations.

The objective of this program is to predict the reactivity of tailings and residues in their respective disposal areas for input into waste and water management planning, and environmental impact assessment. A separate Environmental Sampling and Analysis Plan describes characterization methods for effluents and emissions as metallurgical testing proceeds.

1.2 Geological Setting

The NorthMet Deposit is located in the intrusive mafic Duluth Complex of northern Minnesota. Disseminated copper-nickel-iron sulfides (chalcopyrite, cubanite, pentlandite and pyrrhotite) with associated platinum group element (PGE) mineralization will be extracted from several igneous stratigraphic horizons.

1.3 Agency Consultation and Design Process

This document was developed in consultation with staff from the Minnesota Department of Natural Resources (MDNR). The consultation included the following steps:

- June 22, 2005. A draft of the plan was prepared for MDNR Review.
- July 21, 2005. MDNR provided initial comments focussed primarily on flotation tailings.
- August 16, 2005. MDNR provided further comments on the characterization of hydrometallurgical wastes.
- September 9, 2005. SRK responded to the July 21, 2005 letter.
- September 14, 2005. A conference call was held to discuss the July 21, August 16 and September 9 letters.

This document includes responses to comments provided by MDNR, and has been prepared to conclude the design process and seek MDNR approval of PolyMet's plans to respond to the tailings/residue characterization component of requirements under Minnesota Rules 6132.1000.

1.4 Organization of This Document

This document describes:

- Section 2. Design basis for the program.
- Section 3. Analytical methods. This section describes methods used to analysis solids and leachates.
- Section 4. Use of the results in the context of water chemistry predictions.

1.5 Acknowledgements

The following individuals cooperated in the preparation of this plan:

- John Borovsky, Barr Engineering Company;
- Stephen Day, SRK Consulting;
- Paul Eger, MDNR;
- Jennifer Engstrom, MDNR;
- Don Hunter, PolyMet;
- Kim Lapakko, MDNR;
- Richard Patelke, PolyMet; and
- Jim Scott, PolyMet.

1.6 Laboratory Selection

The following laboratories will perform the procedures described in this plan (contact names for each laboratory are shown):

- ALS Chemex, North Vancouver, British Columbia – solids analysis listed in Section 4.1.1 (Bill Anslow);
- Optical – PolyMet or a Contractor (Richard Patelke);
- Sub-Optical Lab – McSwiggen and Associates (Peter McSwiggen);
- Canadian Environmental and Metallurgical Inc, North Vancouver, British Columbia – kinetic testing (Rik Vos); and
- Cantest Inc.. Vancouver, British Columbia - Kinetic test leachate analysis (Richard Jornitz).

2 Characterization Design

2.1 Objective of the Program

The overall objective of the program is to provide geochemical characterization information that can be used as inputs to design of management plans for the tailings and process residues and inputs into the environmental impact study (EIS) for the project.

2.2 Metallurgical Process Background

The processing and recovery of NorthMet Project ore to recover commodity metals will involve conventional flotation to produce a sulfide concentrate followed by hydrometallurgical treatment of the concentrate. The process flow sheets are provided in Appendix A.

Processing will result in the generation of the following waste products:

- Flotation tailings (low sulfide);
- Leach residue (silicates);
- Gypsum residue (calcium sulfate);
- Raffinate neutralization residue (calcium sulfate);
- Fe/Al residue (oxide); and
- Mg residue (oxide).

2.3 Tailings and Metallurgical Residue Disposal

Tailings and metallurgical residues are proposed for disposal in the existing impoundments at the former LTV Steel Mining Company (LTVSMC) operation (Figure 1). Review of the history of deposition at the LTVSMC tailings area indicates that the tailings are a result of processing ore feed from several iron ore pits and working faces in those pits. The ore was blended by truck delivery to loading pockets and then train delivery to crusher. Once in the plant ore was further blended in the coarse ore bins by a coarse ore tripper which continuously spread coarse ore across seven fine crushing lines and in the fine ore bins by a fine ore tripper which continuously spread fine ore across 34 mill lines. The fine ore was then processed through 34 mill lines in parallel. The tailings are a recombination from the 34 mill lines pumped to the basin and discharged at many spigots at the periphery of the basin. The tailings were deposited over many years as many layers in the basin.

Flotation tailings produced by conventional extraction of commodity-bearing sulfide minerals will be disposed in the existing Cells 1E, 2E and 2W. For the first five years of operation, cell 2W will be lined. Discharge methodology has not been determined but will most likely involve conventional discharge from one or more spigots. Hydrometallurgical residues produced by leaching of the sulfide concentrate will be disposed in lined basins within Cell 2W. The method used to transport the

residues to the cells has not been determined. The residues may be combined for disposal, or disposed separately depending on factors such as the possibility of selling some by-products.

The possibility of constructing tailings dams using cycloned tailings is being considered and has been incorporated in this test program.

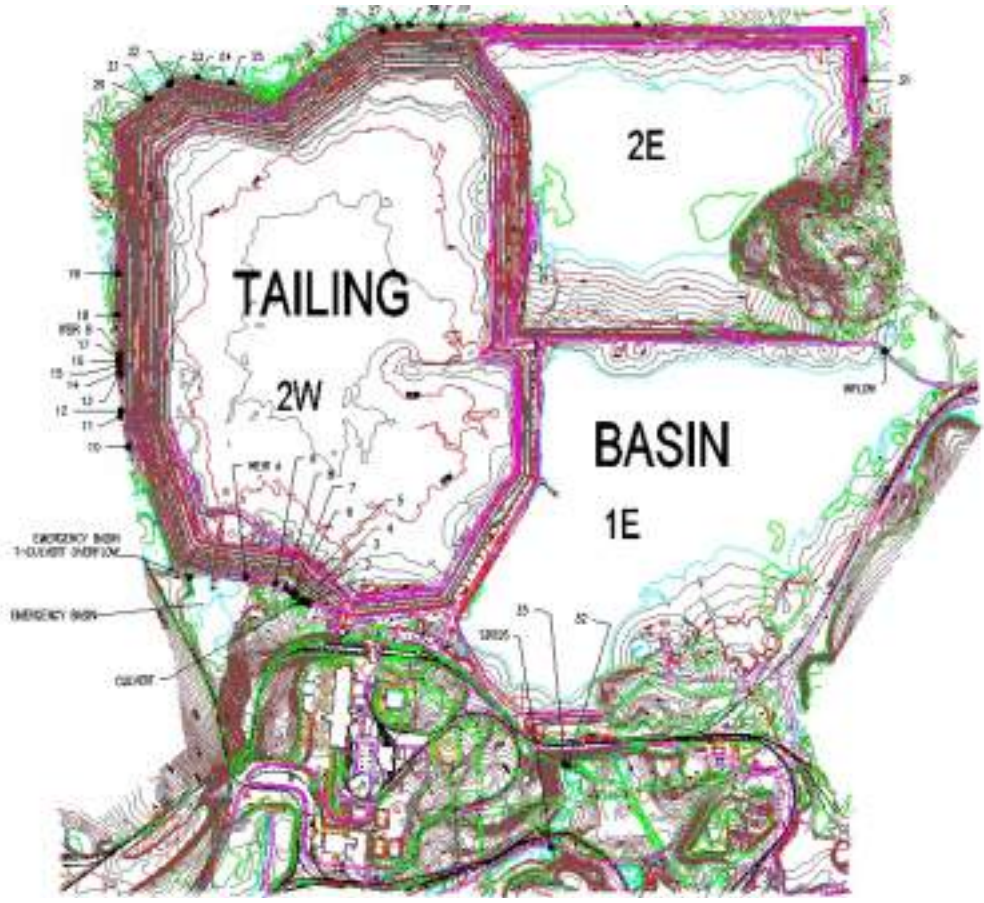

Figure 1: LTV Steel Mining Company Tailings Basins

2.4 Metallurgical Testing

Three ore composites were prepared under PolyMet's direction from diamond drill hole core bracketing head grades expected during mining. Pilot-scale metallurgical testwork was begun in July 2005 and continued into September. Flotation tailings testing was completed in August and included assessment of process alternatives. Addition of copper sulfate to improve sulfide concentrate recovery was evaluated for two ore composites.

A separate Environmental Sampling and Analysis Plan has been prepared to describe monitoring of air, water and solid emissions as the testwork proceeds. That document explains the rationale for preparation of the three ore composites representing copper grades of 0.3%, 0.35% and 0.4%. It also contains details of pilot plant monitoring designed to evaluate variations in tailings geochemical characteristics potentially produced by variations in ore characteristics and process performance. For example, the pilot testing program included frequent (every two hours) monitoring of tailings characteristic.

2.5 Design Basis

2.5.1 Flotation Tailings

Flotation tailings will consist primarily of silicate minerals with small amounts of residual sulfide minerals not recovered by flotation.

The number of variables expected to control reactivity is relatively small. Ore processing naturally results in a well-blended tailings product because the process requires a uniform feed to maximize recovery of commodities. Therefore, variables such as rock type, sulfide mineral type, silicate mineral type and source of ore within the layers of the Duluth Complex will not be significant. Preparation of the ore composite, which has been documented by PolyMet mimics mining at several faces and will result in composites each containing similar distributions of the main silicate and sulfide minerals. Variations will occur reflecting the distribution of commodity-containing minerals. Further, the requirement for grinding to optimize beneficiation of the commodity minerals limits the importance of variables such as mineral particle size and degree of liberation. The remaining variables are therefore expected to be:

- Sulfur content;
- Metal content; and
- Particle size (where separations occur at the deposition site as a result of hydraulic factors) resulting in differences in chemical and mineralogical composition.

The testing of tailings from the three ore composites will allow these variables to be evaluated. It is expected that recovery of sulfide minerals from ore will vary during testwork resulting in tailings containing variable concentrations of sulfur and metals. Results of two-hourly testing were provided in a memorandum to DNR dated January 6, 2006 (Appendix B). The composition of the four tailings samples initiated concurrently with preparation of this plan are shown in Table 1 along with the range of sulfur concentrations indicated by the two-hourly testing.

Table 1: Composition of Four Tailings Samples

Ore Composite	Total Sulphur Content of Ore	Copper Sulfate Used in Flotation	Range of Total Sulphur Concentrations in Tailings	Total Sulphur Content of Tailings Under Test
1	0.86	NO	0.19% to 0.28%	0.23%
		YES	0.09% to 0.13%	0.10%
2	0.90	NO	0.05% to 0.25%	0.20%
3	0.86	YES	0.09% to 0.25%	0.15%

2.5.2 Hydrometallurgical Residues

The residues are mainly chemical products in which the original concentrate components are oxidized to sulfates and hydroxides, and dissolved.

- Sulfur from the sulfide minerals is oxidized to sulfate and precipitated as calcium sulfate through the neutralization process;
- Iron released from sulfide and silicate minerals is precipitated as hydroxide; and
- Magnesium and aluminum released from silicate minerals are precipitated as hydroxides.

Copper is recovered in the process by electro-winning to produce copper anodes. Nickel, cobalt and zinc are recovered as a mixed hydroxide product.

Incomplete dissolution of the silicate and sulfide minerals results in generation of a fourth mineralogical leach residue.

Since these products represent near-complete dissolution, oxidation and neutralization of the concentrate, the chemistry of contact waters is expected to be controlled mainly by the simple dissolution of the compounds. Oxidation of residual sulfide minerals may occur from the leach residue but this effect is expected to minor.

Overall, the process of dissolution of neutralization products is unrelated to the composition of the ore and is not expected to vary in the long term except by depletion.

Pilot scale hydrometallurgical testing will be completed on two composite concentrate samples produced from processing of the three ore samples prepared with and without the use of copper sulfate in the flotation process.

3 Sample Handling and Analysis

3.1 Sample Shipping and Storage

SGS/Lakefield in Lakefield, Ontario, Canada is performing the metallurgical testing. Products will be shipped to Canadian Environmental and Metallurgical Inc. (Vancouver, British Columbia, Canada). Samples will be shipped and stored prior to testing as follows:

- Ore Feed Samples – Refrigerated;
- Sulfide Concentrate – Refrigerated;
- Flotation Tailings – Slurry in sealed pails with sufficient (6 cm) supernatant to ensure the samples are covered by water;
- Leach Residue – Cake, refrigerated; and
- Hydrometallurgical Residues – Cake, refrigerated.

Residual materials remaining after testing will be stored in the same condition as shipped.

3.2 Solids Characterization

3.2.1 Introduction

Table 1 summarizes the types of materials generated by testwork and the chemical testing procedures for each one. Physical testing of these products (including particle size determinations) is described in Environmental Sampling and Analysis Plan for the pilot plant test program. The DNR made several requests for multiple tests on some material types. In reality, the quantity of materials generated by metallurgical testing was limited and restricted the number of tests that could be performed. The 2-hourly testing showed that the sulfur content of the tailings did not vary widely and that the bulk samples under test will characterize the range of sulfur content of tailings.

Details of the test procedures are provided in the following sections.

Table 2: Procedures and Numbers of Samples for Testing

Material	Sulfur Forms	Neutralization Potential	Carbonate	Metals	TCLP	SPLP	Shake Flask	Optical Mineralogy	XRD	Sub-Optical	HCT	Sequential Shake Flask	Column	DNR Reactor Test	Layered Column
Ore feed	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore				
Sulfide Concentrate	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore						
Bulk Flotation Tailings – Without $CuSO_4$[1]	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore			1/Ore	
Bulk Flotation Tailings – With $CuSO_4$[1]	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore			1/ore	Multiple
Cyclone Sands	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore			1/ore	
Tailings Slimes	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		2	1/ore	
Tailings Beaches	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		2	1/ore	
Leach Residue	1	1	1	1	1	1	1	1	1	1	1	1		1	
Gypsum Residue	1	1	1	1	1	1	1		1	1	1	1		1	
Raffinate Neutralization Residue	1	1	1	1	1	1	1		1	1	1	1		1	
Fe/Al Residue	1	1	1	1	1	1	1		1	1	1	1		1	
Mg Residue	1	1	1	1	1	1	1		1	1	1	1		1	
Reactive Residues[2]	1	1	1	1	1	1	1		1	1	1	1		1	
Reactive Residue without Gypsum[3]	1	1	1	1	1	1	1				1	1		1	

Notes:

1. Flotation tailings samples were produced with and without the use of copper sulfate in the process to enhance recovery of sulfide minerals to the sulfide concentrate.

2. Combination of leach, gypsum, raffinate, Fe/Al and Mg residues.

3. Combination of leach, raffinate, Fe/Al and Mg residues.

3.2.2 Bulk Chemical Characterization

A split of each sample will be submitted for an extensive suite of analyses, as follows:

- Sulfur forms (total S, S as sulfate).
- Paste pH.
- Neutralization potential and carbonate.
- 50 elements (mostly metals by ICP scan following aqua regia (nitric and hydrochloric acids) digestion.
- Whole rock oxides. This is provides total concentrations of major elements.

Method detection limits are provided in Appendix C.

3.2.3 Regulatory Leachate Characterization

All samples will be submitted for regulatory leach tests (EPA 1311; EPA 1312) to provide data for waste classification purposes should this be needed. Testing of the ore samples will provide a baseline for comparison to effects from processing.

3.2.4 Shake Flask

A third leach leachate procedure was used as the first step of the sequential shake flask leach procedure described in Section 3.3.3 for hydrometallurgical residues. This procedure, developed by Price (1997) for the British Columbia Ministry of Energy Mines and Petroleum Resources, uses a lower leach ratio (1:3) to improve detection of low levels of metals. The lixiviant is deionized water which typically has a pH between 5 and 6. The leachate is not fixed as in the TCLP and SPLP methods. The extraction is performed by shaking in a glass or plastic container for 24 hours, after which the leachate is extracted and analyzed.

3.2.5 Mineralogical Characterization

Mineralogical characterization will include:

- Optical Analysis on feed, product and mineral wastes (Flotation Tailings, Leach Residue). Other residues will not be examined optically since they are precipitates;
- X-Ray Diffraction on all samples to determine crystalline compounds; and
- Sub-Optical Analysis on ore feed, mineral wastes, and if practical residues to determine the distribution of trace elements in individual minerals prior to and following processing.

3.3 Kinetic Test Methods

3.3.1 Humidity Cell

Humidity cell testing will be performed on ore feed (to characterize three types of ore stockpiles), flotation tailings and leach residues using ASTM Procedure D 5744 – 96 (Reapproved 2001). This procedure was selected for the following reasons:

- Similar procedures have been in use under different names since the late 1980s (e.g. MEND 1991). The results can therefore be evaluated in the context of more than a decade of experience using the procedure.
- It is a standard procedure approved by the ASTM and is therefore defensible as a method.

The ASTM procedure provides some options for varying the test procedure. Appendix D provides a detailed listing of the requirements of the ASTM procedure, options chosen and any variances from the ASTM procedure.

3.3.2 MDNR Reactor

To allow comparison with previous MDNR studies, bulk tailings samples will be tested using a procedure referred to as the "MDNR Reactor" experiment. An apparatus specifically designed by MDNR (Appendix E) contains 75 g of solids.

3.3.3 Sequential Shake Flask Test

All residues are being tested using a sequential leach procedure consisting of weekly repetition of the leach procedure developed by Price (1997) in which the solid to liquid ratio is 1:3 (Section 3.2.4). The procedure involves weekly leaching of roughly 300 g of solids in a plastic bottle. The leaching step consists of addition of deionized water and agitation for 24 hours. The leachate is then decanted for analysis. Between leach steps, the bottle remains open to the atmosphere.

This procedure was requested by the DNR to provide a more aggressive evaluation of hydrometallurgical residue dissolution than occurs in humidity cells.

3.3.4 Leach Columns

A procedure to evaluate the interaction between leachate from NorthMet tailings and LTV tailings was designed and presented to the DNR (Appendix F). The procedure provides for two subaerial columns to generate leachate from the NorthMet tailings.

3.3.5 Leachate Analysis

Leachates from kinetic tests will be analyzed for the parameters indicated in Table 2, which also shows reporting limits. These limits are higher than the detection limits for the analytical instruments. Reporting limits represent the level at which the analytical laboratory (CANTEST) is

confident that the concentrations are quantifiable to an acceptable level. The instrument is able to detect much lower levels but these concentrations carry a very high degree of uncertainty which includes “undetectable”.

Low level leachate analyses for dissolved elements as shown in Table 2 are performed every four weeks (weeks 0, 4, 8 etc). On the intervening even numbered weeks (2, 6, 10 etc.), an ICP scan is performed using a higher detection limit primarily to determine the trend in major ions. pH and conductivity are determined every week. Acidity, alkalinity, inorganic carbon, sulphate, fluoride and chloride are determined every other week.

It is expected that testing of flotation tailings will result in very dilute leachates containing low concentrations of the metals of interest. Back-calculation of metal concentrations from other testwork performed by DNR indicates that cobalt and nickel concentrations could be in the tens of nanograms per litre (ng/L) for nickel and near nanograms per litre for cobalt. Quantification of these low metal concentrations is needed to provide reasonably constrained estimates of metals concentrations in the tailings storage facility.

A number of different approaches are available to quantify low levels of nickel and cobalt:

- The routine leachate analysis will achieve a reporting level of 0.0001 mg/L (100 ng/L). Should concentrations be undetected, detection limits of 50 ng/L can be obtained with additional processing effort using the same routine method.
- Specialist methods can achieve lower detection limits. These are non-routine (for example, evaporation to increase concentrations) and will need to be developed as the need arises.
- Existing testwork demonstrates that good correlations exist between cobalt and nickel concentrations in leachates. Detectable nickel concentrations can be used to estimate cobalt concentrations if this relationship can be demonstrated.
- In the event of undetectable low levels, a scale-up methodology will be agreed upon with MDNR to translate non-detectable concentrations to tailings concentrations. Detection limit values will be used in modeling calculations.

Table 3: List of Parameters for Low Level Analysis of Humidity Cell Leachates

Parameter	Reporting Limit	Parameter	Reporting Limit
pH (standard units)	-	Acidity	1
Conductivity (µS/cm)	1	Alkalinity	1
Chloride	0.2	Sulfate	0.5
Fluoride	0.05	Total Inorganic Carbon	1
ORP (mV)	-		
Dissolved Elements (mg/L)			
Aluminum	0.001	Mercury	0.00005[2]
Antimony	0.0001	Molybdenum	0.00005
Arsenic	0.0001	Nickel	0.0001 (0.00005)[1]
Barium	0.0001	Potassium	0.02
Beryllium	0.0002	Selenium	0.0002
Bismuth	0.0002	Silicon	0.05
Boron	0.005	Silver	0.00005
Cadmium	0.00004	Sodium	0.01
Calcium	0.01	Strontium	0.0001
Chromium	0.0002	Tellurium	0.0002
Cobalt	0.0001 (0.00005)[1]	Thallium	0.00002
Copper	0.0001	Thorium	0.0001
Iron	0.01	Tin	0.0001
Lead	0.00005	Titanium	0.0002
Lithium	0.0002	Uranium	0.00005
Magnesium	0.005	Vanadium	0.0002
Manganese	0.00005	Zinc	0.001

Notes: 1. Low detection limits are available for cobalt and nickel as shown.
2. Lower level mercury analyses will be performed on selected samples.

3.4 Analysis of Remaining Sample Following Dissolution Tests

Analyses of the remaining sample following dissolution tests will be considered depending on the results obtained from the tests. Generally, these analyses can be of value if the test has undergone a major chemical change during the procedure (e.g. change from alkaline to acidic leachate) or if calculations indicate that a large quantity of one or more minerals or elements has been depleted. These is little value in post-test analysis if the depletion quantity is less than the uncertainty that can be expected from sampling of the test residue analysis. Experience also indicates that mineralogical analyses are also of little value unless weathering processes have had a detectable effect on the sample.

Therefore, residue analyses will be performed if:

- A large drop in pH has occurred (for example, from above 7 to below 5).
- Depletion calculations indicate that more than 10% of an important component was removed during the procedure.

Residue analyses will consist of the same procedures performed prior to the test.

3.5 Quality Assurance/Quality Control

A Quality Assurance Project Plan (QAPP) is being prepared for this project. To summarize, QA/QC includes the following components:

- Roughly 10% of all solids analyses will be performed in duplicate as sample availability permits.
- Roughly 10% of all cell and reactor tests will be run as duplicates if sufficient test material is available.
- A blank cell and reactor containing no sample will be operated to check for contamination of leachates by construction materials.
- Individual leachate results will be reviewed.
- Ion balances on leachate results will be reviewed. In general, imbalances of ±10% are considered acceptable. Re-analysis if requested depending on the nature of the imbalance.
- Data trends in kinetic test leachates will be analysed to check for anomalies.

4 Use of Data for Water Quality Predictions

4.1 Introduction

The data obtained from these programs will be used to estimate water quality during operation and closure. The following sections describe the application of the data to making water chemistry predictions.

4.2 Operational Water Quality

4.2.1 Flotation Tailings

Operational water quality in flotation tailings impoundments tends to be dominated by process water since this is the largest volume of water moving into and out of the impoundment by discharge and reclaim. Processes resulting from oxidation are not usually significant because continual placement of fresh tailings covers up older tailings before extensive weathering is initiated. The tailings pond water and trapped pore waters therefore reflect re-circulating process water. Seepage from the impoundments typically has a process water signature modified by anoxic conditions in the saturated tailings and interaction with LTVSMC taconite tailings.

Seepage chemistry will also be influenced by dam construction, particularly whether drains will be needed for stability. If drains are needed, water will be drained horizontally away from the dams, rather than the normal vertical seepage thru the coarse fraction of tailings near the dam.

The method used to estimate tailings pond water during operations is typically a coupled water and load balance that evaluates the effect of build-up of solutes in the tailings pond water due to interactions (e.g. mineral dissolution), reagent addition in the process, unintended additions in the process (e.g. Mo from lubricants), dilution due to rain and snowfall, dilution by run-in, and solute load loss due to encapsulation. If kinetic testing shows a short term leaching effect from beaches and dam faces, these loads are included.

The effect of interaction of saturated tailings with LTVSMC tailings will be evaluated directly by column tests (Appendix F). These tests will indicate whether any significant losses or additions occur as process water moves through the taconite tailings.

4.2.2 Leach Residues

Rinsed leach residues are expected to be relatively soluble and water chemistry associated with the residues will be dominated by equilibration of rinse. Water quality may be affected by the method of disposal (pumped slurry vs truck hauled solids). Since the residues will be continually accumulated, the operational water chemistry will be a result of mixing of rinsate water with precipitation and run-in. The chemistry of contact water during operation will be estimated directly from dissolution test results. Pore water chemistry indicated by testwork will be evaluated using MINTEQA2 or similar thermodynamic equilibrium models.

The effect of small amounts of sulfide minerals in the leach residues will be evaluated.

4.3 Water Quality at Closure and Post Closure

4.3.1 Flotation Tailings

At closure, the main effect is removal of inflows of process water and on-set of oxidation of tailings resulting in metal and possibly acidity loadings. The water and load balance developed for operational conditions is typically modified to evaluate these effects.

The long term closure modeling will need to consider evolution of the tailings profile in response to oxidation. This type of modelling uses humidity cell weathering rates, tailings physical characteristics, and moisture profiles predicted by modelling (HELP and HYDRUS-2D) as inputs. The propagation of the oxidation front and acidity front through the tailings is predicted and used to model the movement of solutes. The migration of the fronts can be used to estimate changes in solute loading in the future due to arrival of chemical fronts at the base of tailings. This type of modeling can be coupled with groundwater models to predict the chemistry of groundwater leaving the site.

4.3.2 Leach Residues

Water associated with the leach residues is expected to evolve as rinsate is displaced by dilute water from rainfall and snowmelt. Humidity cells will show whether this results in changing water quality. Concentrations indicated from humidity cells may be suitable for direct prediction of water chemistry or may be adjusted using MINTEQA2 or similar thermodynamic equilibrium models.

5 References

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Appendix A

Process Flow Sheets

BATEMAN METALS

ORE CAR TIPPER

COARSE CRUSHER DUMP POCKET

PRIMARY CRUSHER (1)

PRIMARY CRUSHER PRODUCT POCKET

SECONDARY CRUSHER (4)

APRON FEEDER (4)

CONVEYOR

CONVEYOR

COARSE ORE BIN

FEED CHUTE (18)

BELT FEEDER (6)

RETRACTABLE CONVEYOR (3)

TERTIARY CRUSHER (3)

VIBRATING SCREEN FEEDER (6)

VIBRATING SCREEN (6)

QUARTERNARY CRUSHER (6)

CONVEYOR

CONVEYOR

FINE ORE BIN

FEEDER (132)

RODS

BALLS

ROD MILL (12)

PRIMARY CYCLONE (36)

BALL MILL (12)

PAX

AIR

MIBC

ROUGHER FLOTATION (4)

ROUGHER CONDITIONING TANK (2)

AIR

MIBC

PAX

SCAVENGER FLOTATION (12)

REGRIND CYCLONE (9)

FLOCCULANT

FLOTATION TAILS THICKENER (2)

WW1572

PAX

CLEANER 1 CONDITIONING TANK (1)

MIBC

AIR

CLEANER 1 FLOTATION (8)

PAX

REGRIND HOPPER (2)

REGRIND MILL (3)

FLOCCULANT

CONCENTRATE THICKENER (1)

PROCESS WATER TANK (1)

FLOAT TAILINGS

AIR

MIBC

WW1572

CLEANER 2 FLOTATION (4)

PAX

CLEANER 3 CONDITIONING TANK (1)

CLEANER 3 FLOTATION (4)

(2)

(2)

CONCENTRATE GRINDING

FINAL CONCENTRATE SURGE TANK (2)

TO AUTOCLAVE FEED TANKS

BAB403-P-DX-003

FIGURE 1
POLYMET FEASIBILITY STUDY
COMMINUTION AND FLOTATION
SCHEMATIC FLOW DIAGRAM

BATEMAN METALS

1ST STAGE HYDROXIDE PRECIPITATION TANK (3)
AIR
MgO
1ST STAGE HYDROXIDE THICKENER (1)
FLOCCULANT
SERVICE WATER
HYDROXIDE FILTER (1)
MIXED HYDROXIDE PRODUCT
2ND STAGE HYDROXIDE PRECIPITATION TANK (3)
LIME
FLOCCULANT
2ND STAGE HYDROXIDE THICKENER (1)
TO AUTOCLAVE LET DOWN TANK
Mg REMOVAL TANK (4)
LIME
Mg REMOVAL THICKENER (1)
TO HM PROCESS WATER TANK
TO REACTIVE RESIDUES
STEAM
N_2
NaHS
Cu PRECIPITATION TANK (3)
CuS THICKENER (1)
FROM FINAL CONCENTRATE SURGE TANKS
HCl
AUTOCLAVE FEED TANK (3)
OXYGEN
AUTOCLAVE (3)
BLOWDOWN TO FLOTATION TAILINGS
SERVICE WATER
COOLING TOWER
SERVICE WATER
AUTOCLAVE GAS SCRUBBER (3)
HEAT EXCHANGER
TO ATMOSPHERE
SERVICE WATER
VENTS FROM ALL TANKS
FINAL GAS SCRUBBER
AUTOCLAVE FLASH VESSEL (3)
FLOCCULANT
LEACH RESIDUE THICKENER (1)
LIMESTONE
AIR
STEAM
BLEED
Fe REMOVAL TANK (4)
FLOCCULANT
Fe REMOVAL THICKENER (1)
HM PROCESS WATER
LIMESTONE
Fe/Al REMOVAL THICKENER FILTER
Fe/Al RESIDUE TO REACTIVE RESIDUES
Al REMOVAL TANK (2)
FLOCCULANT
Al REMOVAL THICKENER (1)
PRE NEUTRALISATION THICKENER (1)
FLOCCULANT
GYPSUM FILTER (1)
HM PROCESS WATER
GYPSUM RESIDUE TO REACTIVE RESIDUES
LIMESTONE
PRE NEUTRALISATION TANK (4)
FLOCCULANT
THICKENER (1)
SERVICE WATER
PGM FILTER (1)
NaHS
PGM PRECIPITATION (3)
HM PROCESS WATER
LEACH RESIDUE FILTER (3)
SERVICE WATER
HM PROCESS WATER
HM PROCESS WATER TANK
REACTIVE RESIDUES
ORGANIC MAKE-UP
LOADED ORGANIC TANK
COPPER EXTRACTION (3)
COPPER STRIPPING (2)
WASH (1)
SERVICE WATER
GUAR GUM
$CoSO_4$
H_2SO_4
COPPER ELECTROWINNING (160)
PRODUCT COPPER CATHODES
H_2SO_4
OXYGEN
RELEACH VESSEL (2)
TO ATMOSPHERE
SERVICE WATER
EW MIST SCRUBBER
DISCHARGE TO SOLVENT EXTRACTION
SERVICE WATER
PGM CONCENTRATE FILTER (1)
PRODUCT PGM CONCENTRATE
LIMESTONE
FLOCCULANT
RAFFINATE NEUTRALISATION THICKENER (1)
HM PROCESS WATER
RAFFINATE NEUTRALISATION (3)
RAFFINATE NEUTRALISATION FILTER (1)
GYPSUM RESIDUE
CRUD REMOVAL
SOLIDS TO REACTIVE RESIDUES
LIQUID TO LOADED ORGANIC TANK
BAB403-P-DX-004

FIGURE 2
POLYMET FEASIBILITY STUDY
HYDROMET PROCESS PLANT
OPTION 1 - MIXED Ni/Co
HYDROXIDE PRECIPITATION
SCHEMATIC FLOW DIAGRAM

Appendix B

Update on Sample Selection from 24 Hour Testwork

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Memo

To:	Jennifer Engstrom, MDNR	**Date:**	January 6, 2006
cc:	John Borovsky, Barr	**From:**	Stephen Day
	Jim Scott, PolyMet		
	Don Hunter, PolyMet		
Subject:	NorthMet Project Tailings and Hydromet Residue Testwork – Update on Sample Selection from 24 Hour Testwork	**Project #:**	1UP005.001

Jennifer

We have now received the 2-hourly tailings total sulfur analyses from the pilot plant testwork. These results allow final recommendations to be made for the selection of samples for tailings and hydrometallurgical testwork.

1 Results and Implications of 2-Hourly Sulfur Analyses

Results of the 2-hourly sulfur analyses are shown by the coloured solid lines in Figure 1. The broken horizontal lines are the concentration of sulfur in the composite tailings samples currently being tested in humidity cells. The sulfur content of the ore composites was very uniform (Parcel 1, 2 and 3, 0.86%, 0.9% and 0.86%, respectively) as shown by the solid black lines in Figure 1.

The trend in sulfur results in tailings is explained by the chronology of the testwork and evaluation of addition of copper sulfate as a reagent:

- Flotation testwork began on July 17 with Parcel 2 without the use of copper sulfate. Parcel 2 was processed entirely without using copper sulfate. As shown, sulfur concentrations varied from 0.05% to 0.25% reflecting adjustment of the process conditions early in the testwork. The average was 0.19%. The composite tailings sample has a sulfur content of 0.2% closely representing the average.

- Testwork continued with Parcel 1 without using copper sulphate. Processing was continuous so one point is shared between Parcel 2 and Parcel 1. The range of sulfur concentrations was 0.19% to 0.28% with an average of 0.24%. The composite sample was 0.23% and is close to the average.

- Pilot plant testwork was suspended on July 19 to allow for further bench scale testing on recovery of metals.

- The pilot plant resumed on August 8 using Parcel 1. Addition of copper sulfate was evaluated. This reagent causes activation of the sulfide mineral surfaces and improves bulk sulfide flotation. The effect of copper sulfate on tailings characteristics was immediately apparent for Parcel 1.

Total sulfur concentrations decreased to a range of 0.09% to 0.13% (average 0.1%) and the resulting tailings composite was 0.1%.

- Processing continued with Parcel 3 using the copper sulfate additive. Sulfur content of the tailings varied over a wider range (0.09% to 0.25%, average 0.18%) though the range was comparable to the total range indicated by processing of other ore packages. The resulting composite had a total sulfur content of 0.15%.

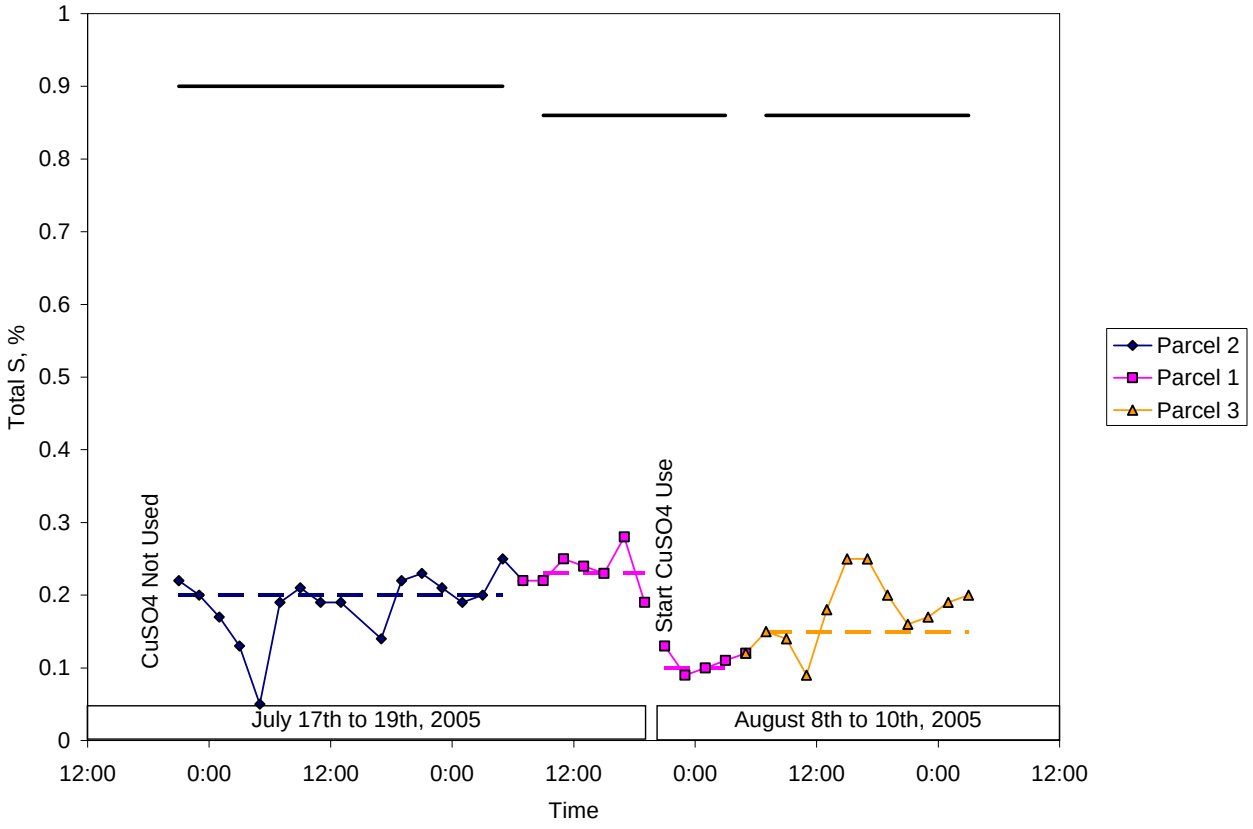


Figure 1. Results of 2-Hourly Total Sulfur Analyses. Solid lines and points connect 2-hourly results. Broken solid lines are sulfur concentrations in composite tailings samples representing each stage of testwork. Solid horizontal lines are the respective ore composite sulfur contents.

Based on the process testwork, Polymet has the made decision to advance the project with the use of copper sulfate to optimize overall sulfide mineral flotation. This decision is beneficial for the tailings since it is expected to lower the overall sulfide content.

The process testwork showed that sulfur concentrations in the tailings can be expected to vary in response to changes in process conditions including the use of copper sulfate. Parcel 3 showed that the use of copper sulfate may not always result in low sulfur content in tailings, and therefore there is need to capture sulfur concentrations approaching 0.25% in the kinetic testwork. The samples generated without copper sulfate provide the required range and can be tested to represent the potential for higher sulfur concentrations in the tailings. The lack of copper sulfate for the Parcel 2 and 1 samples is not expected to have significantly affected the reactivity of the residual sulfide minerals in the tailings:

It is therefore concluded that:

- Kinetic testing of all four tailings samples should be continued.
- No additional samples are needed to represent the range of sulfur content expected in tailings.

2 Testing of HydroMet Residues

2.1 Source of Sulfide Concentrate for Hydromet Process Evaluation

Evaluation of the hydromet process was performed using two bulk sulfide concentrates produced by processing of ore parcels 2 and 1 (without copper sulfate) and ore parcels 1 and 3 (with copper sulfate). The sulfide concentrates contained the following total sulfur concentrations:

- No Copper Sulfate
  - Parcel 2 – 23.6%
  - Parcel 1 – 21.3%
- With Copper Sulfate
  - Parcel 1 – 22.1%
  - Parcel 3 – 21.6%

It is apparent that the sulfur content of the concentrates does not vary significantly though the effect of copper sulfate on concentrate sulfur content for Parcel 1 is apparent and corresponds with the matching decrease in sulfur content of the tailings. Since the decision has been made to proceed with the use of copper sulfate, only the residues produced from sulfide concentrate generated using copper sulfide should be tested.

2.2 HydroMet Residues

All the expected HydroMet Residues were produced by processing of the sulfide concentrate generated using copper sulfate.

A difference exists between the way that the residues were recovered in the pilot test compared to actual operating conditions.

To summarize, the first step in the process is the leaching of the sulfide concentrate to produce a low pH pregnant solution containing all the commodity metals. Subsequent recovery of the metals involves a series of pH adjustments to the leach solution that results in precipitation of products and residues. The products are then refined to recover the contained metals (copper, nickel, cobalt, PGM, zinc). The residues contain entrained leach solutions that have to be recovered to optimize recovery of commodity metals. Under full-scale operating conditions, recovery of the leach solutions from the residues will occur by rinsing the residue cakes with pH-adjusted re-cycled final process water to displace the leach solutions. The pH adjustment is required to ensure that metals in the leach solution are not lost to the solids.

However, under pilot plant conditions, the recycled process water was not available because the processing of the leach solutions occurred in a stepwise rather than continuous fashion. Each metal recovery step was performed and completed before proceeding to the next. The final process solution that will be used for rinsing at full-scale was only generated at the end of the pilot plant and was therefore not available for the residue rinsing steps. The difference between full-scale and pilot plant conditions represents a practicality of metallurgical testing in that operation the pilot plant continuously is not an option with the available quantity feed concentrate.

Residues generated by the pilot plant were rinsed with locally obtained river water. No additional rinsing of the residues is proposed for the dissolution testwork. Rinsing with river water was less aggressive in displacing metal-laden leach solutions than can be expected with pH-adjusted process water. The residues can therefore be expected to contain higher metal content than under operating conditions and indicate greater leachable metals in dissolution tests. The testwork will tend to over-estimate rather than under-estimate water quality for water management planning and impact assessment.

3 Conclusions

The following actions are proposed:

- Kinetic testing of all four existing tailings samples in dissolution tests will continue.
- Testing of residues produced by hydromet testing of sulfide concentrate (with copper sulfate) will be started as described in the "Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan"

Appendix C

Parameter Lists and Detection Limits for Analysis of Solids

ALS/CHEMEX METHOD CODE	ITEM	UINTS	CHEMEX DETECTION LIMIT
ME-ICP61 (four acid)	CU%	%	0.001
ME-ICP61 (four acid)	NI%	%	0.001
S-IR08 (LECO SULFUR)	S%TOT	%	0.01
ME-ICP61 (four acid)	S%ICP	%	0.01
PGM-ICP23 (30 GRAM)	PT_PPB	PPB	5
PGM-ICP23 (30 GRAM)	PD_PPB	PPB	1
PGM-ICP23 (30 GRAM)	AU_PPB	PPB	1
ME-ICP61 (four acid)	CO_PPM	PPM	1
ME-ICP61 (four acid)	AG_PPM	PPM	0.5
ME-ICP61 (four acid)	ZN_PPM	PPM	2
ME-ICP61 (four acid)	CD_PPM	PPM	0.5
ME-ICP61 (four acid)	MO_PPM	PPM	1
ME-ICP61 (four acid)	PB_PPM	PPM	2
ME-ICP61 (four acid)	AS_PPM	PPM	5
ME-ICP61 (four acid)	CR_PPM	PPM	1
ME-ICP61 (four acid)	V_PPM	PPM	1
ME-ICP61 (four acid)	TI%	%	0.01
ME-ICP61 (four acid)	AL%	%	0.01
ME-ICP61 (four acid)	CA%	%	0.01
ME-ICP61 (four acid)	FE%	%	0.01
ME-ICP61 (four acid)	K%	%	0.01
ME-ICP61 (four acid)	NA%	%	0.01
ME-ICP61 (four acid)	MG%	%	0.01
ME-ICP61 (four acid)	MN_PPM	PPM	5
ME-ICP61 (four acid)	P_PPM	PPM	10
ME-ICP61 (four acid)	BA_PPM	PPM	10
ME-ICP61 (four acid)	BE_PPM	PPM	0.5
ME-ICP61 (four acid)	BI_PPM	PPM	2
ME-ICP61 (four acid)	SB_PPM	PPM	5
ME-ICP61 (four acid)	SR_PPM	PPM	1
ME-ICP61 (four acid)	W_PPM	PPM	10

ALS/CHEMEX METHOD CODE	ITEM	UINTS	CHEMEX DETECTION LIMIT
ME-ICP41 (Aqua regia digestion)	CU%	%	0.001
ME-ICP41 (Aqua regia digestion)	NI%	%	0.001
ME-ICP41 (Aqua regia digestion)	S%ICP	%	0.01
ME-ICP41 (Aqua regia digestion)	CO_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	AG_PPM	PPM	0.2
ME-ICP41 (Aqua regia digestion)	ZN_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	CD_PPM	PPM	0.5
ME-ICP41 (Aqua regia digestion)	MO_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	PB_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	AS_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	CR_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	V_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	TI%	%	0.01
ME-ICP41 (Aqua regia digestion)	AL%	%	0.01
ME-ICP41 (Aqua regia digestion)	CA%	%	0.01
ME-ICP41 (Aqua regia digestion)	FE%	%	0.01
ME-ICP41 (Aqua regia digestion)	K%	%	0.01
ME-ICP41 (Aqua regia digestion)	NA%	%	0.01
ME-ICP41 (Aqua regia digestion)	MG%	%	0.01
ME-ICP41 (Aqua regia digestion)	MN_PPM	PPM	5
ME-ICP41 (Aqua regia digestion)	P_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	B_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	BA_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	BE_PPM	PPM	0.5
ME-ICP41 (Aqua regia digestion)	BI_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	GA_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	HG_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	LA_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	SB_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	SC_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	SR_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	W_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	TL_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	U_PPM	PPM	10

ALS/CHEMEX METHOD CODE	ITEM	UINTS	CHEMEX DETECTION LIMIT
ME-ICP06--whole rock geochemisrty by ICP-AES	SIO2	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	AL203	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	TIO2	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	FE2O3	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	CAO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	MGO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	MNO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	NA2O	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	K2O	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	P2O5	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	BAO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	SRO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	LOI	%	0.01

Appendix D

Options and Variance in ASTM Humidity Cell Procedure

9. Sample Preparation

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
9.1	Air dry as-received bulk samples of solid material to prevent the additional oxidation of reactive minerals or compounds. If air drying is not practicable, oven dry the solid material at a maximum temperature of 50 ± 2°C for 24 h, or until a constant weight is reached.	Samples were air-dried at room temperature (~ 20 °C).	
9.1.1	If exploration-generated or run-of-mine solid material samples are not readily available, archived dried and crushed samples from geological exploratory or development drilling programs may be used for preliminary evaluations of ore and waste rock from new operations; this is provided that the available solid material samples are not significantly finer than 95 % passing a No. 12 (1.7-mm) sieve. Document the sample drying and preparation procedures used during the drill sampling program in order to interpret the results properly. Evaluate the effects of drying temperature on metals volatilization (for example, mercury in cinnabar vaporizes at temperatures exceeding 80 to 90°C) and mineral morphology and chemistry modifications (for example, on heating at temperatures exceeding 100°C, chalcocite changes crystal form and is oxidized subsequently from Cu_2S to CuO, $CuSO_4$, and SO_2). Especially ensure that the effects of particle size distribution changes resulting from the more finely crushed sample are considered in the interpretation (this is, the potential for increased liberation of acid-producing and acid-consuming minerals with an attendant increase in mineral surface area).	NA	
9.1.2	In mining waste evaluations, the particle size for mill tailings will be significantly finer (commonly less than 150 µm/100 mesh) than the particle size distributions from ore and waste rock. Pilot plant tailings should be used if mill tailings are not available.	NA	
9.2	Screen the air-dried bulk samples through a 6.3-mm (¼-in.) screen in accordance with Test Method E 276. Crush any oversize material so that 100 % passes the screen.	NA	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
Note 7	Caution: Recent accelerated weathering studies of run-of mine waste rock from metal mines demonstrate that crushing a bulk sample so it passes a 6.3-mm (¼-in.) screen may change the character of the sample by artificially increasing liberation and consequent surface areas of acid-producing and acid-consuming minerals contained in the + 6.3-mm (¼-in.) material. A suggestion for avoiding this problem is to segregate the - 6.3-mm (¼-in.) fraction by screening rather than crushing, and to test that fraction according to the protocol and equipment described in this text. The + 6.3-mm (¼-in.) material can be tested separately (for example, Brodie, et al (10) describe a large-scale humidity cell test that would accommodate – 75-mm material). Samples from the drill core and cuttings also present material sizing problems, which must be considered when interpreting drill core and cuttings accelerated data. The drill core must be crushed to -6.3-mm (¼-in.) to fit the cell described in this test method. The resulting size distribution from crushing will differ from that of run-of-mine due to differences in fracture patterns inherent to blasting practices that produce run-of-mine material. By contrast, drill cuttings size fractions are commonly less than 6.3-mm (¼-in.) due to the rotary-percussive nature of obtaining the sample.	NA	
9.3	Mix and divide the bulk sample to obtain a representative test unit with a weight in the range of 8 to 10 kg, using a riffle splitter with 1-in. (2.54-cm) chutes. Divide the test unit into eight nominal 1-kg specimens. Seal each test specimen in a moisture-barrier bag.	NA	
Note 8	The dried sample should be mixed through the riffle splitter at least once before making any splits; recombine the splits resulting from the sample mixing exercise by pouring individual splits either over each other or through the splitter again. Once the actual split is made, it is wise to re-mix it (according to the above procedure) prior to making the next split.		Samples were mixed through the riffle splitter once.
9.4	Select one test specimen at random, and determine the moisture content by weighing and drying to constant weight at 80 ±5°C.		Determined at 20 °C
9.4.1	Crush the dried test specimen so that at least 95 % passes a 1.7-mm (10-mesh) screen, in accordance with Test Method E 276.	NA	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
9.4.2	Divide the crushed test specimen in half twice, using a riffle splitter with 6.35-mm (¼-in.) chutes, and select a ¼ subsample at random.	NA	
9.4.3	Transfer the selected subsample to a ring and puck grinding mill and grind to a nominal of 95 % passing a 150-µm (100-mesh) screen, in accordance with Test Method E 276. Use the subsample for chemical and mineralogical characterization of the test unit.	NA	
9.5	Select one test specimen at random, and determine the particle size distribution in accordance with Test Method E 276.	ASTM	
9.6	Select one test specimen at random for use in the accelerated test method. Divide the test specimen into four nominal 250-g subsamples using the riffle splitter with 25.4-mm (1-in.) chutes, and label and store in vapor-barrier bags until it is time to load the humidity cells.	NA	
9.7	Reserve the remaining test specimens for replicated testing or to resolve disputed results.	NA	

10. Apparatus Assembly

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
10.1	The humidity cells are table-mounted at a height sufficient to accommodate the placement of both the humidifier and one Erlenmeyer flask for effluent collection from the bottom of each cell. During the water-saturated and dry-air portions of each weekly cycle, feed air is metered to the bottom of each cell at the selected rate (1 to 10L/min). Feed air for the three-day dry–air portion is routed first through a desiccant column and then to each of the cells through a dry-air manifold. Feed air for the water-saturated air portion is routed through a water-filled humidifier by means of aeration stones or gas dispersion fritted cylinders/disks, and then to each humidity cell lid air exit port to prevent the short circuiting of air through cells containing more permeable solid material samples. A separatory funnel rack is mounted on the table that holds the cells if the weekly water leach is applied dropwise (drip trickle). Multiple separatory funnels (one for each cell) are held in the rack during the drip trickle leach that is performed on the seventh day of each weekly cycle. The separatory funnel can be used to meter the required water volume slowly down the sides of the cell wall until the sample is flooded if the weekly leach is to be a flooded leach.	Humidity cells are constructed of acrylic tubing with an inside diameter of four inches and an overall height of twelve inches, with an acrylic base plate. The base plate is glued to the tube and threaded with a nylon hose adapter to which a length of tubing is attached to allow for leachate drainage into a collection container. A perforated PVC support plate is positioned inside the cell, one inch above the base plate and covered with six layers of nylon mesh. A nylon adapter is threaded into the side of the cell between the support plate and the base plate and a length of tubing was connected from the side adapter to the humidifier to facilitate the inflow of humid air to the cell. A dry air line is also connected to each cell. Each cell is covered with a removable acrylic lid.	Approximately 16 cells per humidifier Flood leaching: peristaltic pump using a peristaltic pump Temperature: 20 ± 2°C. Feed air rate to be determined.

11. Procedure

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.1	Cell Loading:		
11.1.1	If more than one humidity cell is used at one time, label each with a sequential number, and use the same number for the matching collection vessel (Erlenmeyer flask).	ASTM	
11.1.2	Weigh each humidity cell (without its lid) and each collection vessel; record the tare weights of each to the nearest 0.1 g.	ASTM	
11.1.3	Cut the filter media (such as 12-oz/yd^2 polypropylene described in 6.11) to the humidity cell's inside diameter dimensions so that it fits snugly yet lies flat on the perforated support.		Shark Skin filter paper (320mm)
11.1.4	Re-weigh the humidity cell, and record the resulting tare to the nearest 0.1 g; the original cell tare (11.1.2) minus the new cell tare is the weight of the filter media.	ASTM	
11.1.5	Transfer the contents from each of the four bags containing the 250-g samples (9.6) into the humidity cell. Prior to the transfer, mix the contents of each bag by gentle rolling to eliminate possible stratification that may have occurred during sample storage.	ASTM	
11.1.6	Re-weigh the loaded cell, and record the weight to the nearest 0.1 g; the loaded cell weight minus the combined cell and filter-media tare weight is the weight of the sample charge.	ASTM	
11.2	First Leach:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.2.1	The first leach (whether drip trickle or flooded), designated as the Week 0 leach, initiates the 20-week long humidity cell test and establishes the starting or initial characteristics of the leachate. Either a 500-mL or 1-L volume of water may be used for the weekly leachates, depending on the weekly pore volume desired or the quantity of solution required for analytical purposes; however, once a weekly volume has been selected, that weekly volume must remain constant throughout the 20-week testing period. A centrifuged cell culture of *Thiobacillus ferrooxidans* may be used in the first leach in order to ensure that optimum conditions for accelerates weathering are present at the beginning of the test.	500 mL Flood Leach	
Note 9	In the testing of mining wastes, cation (including metals and trace metals) and anion loadings are commonly high in the Week 0 leachate due to the dissolution of pre-existing soluble oxidation salts present in the sample prior to sample collection. The average number of weekly accelerated weathering cycles required to flush these pre-existing salts ranges from 3 to 5 weeks. Oxidation products observed during these 3 to 5 weeks are principally from the pre-existing salts, while those products observed after this period are considered to be solely a function of the accelerated weathering procedure. A method for estimating the amount of pre-existing oxidation salts present in a solid material sample is described by Sobek, et al (6). A comparison of estimated salt storage data obtained using this method with the first thee weeks of humidity cell effluent loadings from three different samples is describes by White and Jeffers (7).	NA	
11.2.2	Fill a separatory funnel with for each cell with de-ionized water using a volumetric flask. If the leach is to be performed using the drip trickle method, set each separatory funnel above its corresponding cell, and adjust the drip rate (approximately 3 to 4 L/min) so that the solid material sample is wetted thoroughly but not flooded.	NA	
11.2.3	A minimum of 2 to 3 h is commonly required to complete the drip trickle leach.	NA	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.2.4	If the leach is to be performed by flooding, the separatory funnel can be used to meter the selected water volume slowly down the sides of the cell wall until the sample is flooded. This application method reduces hydraulic agitation of the sample surface commonly caused by pouring liquid from an open-mouthed vessel. Alternatively, flooding may be accomplished by any application apparatus (for example, a peristaltic pump) that supplies the selected volume of leachant at a reasonable rate without causing agitation and suspension of the finer fractions contained in the sample charge.	ASTM	
11.2.4.1	Allow the flooded cell to sit for a period of 1 h before draining the leachate into the Erlenmeyer collection flask. The 1-h leach time commences after all of the leachant has been placed in the cell. The solid material sample should be saturated and covered with leachant to a depth sufficient to maintain sample saturation. In testing mining wastes, the observed depth of leachant cover from a 500-mL flooded leach performed in 10.2-cm (4.0-in.) ID cells is approximately 2.5 cm (1.0 in.).	ASTM	
11.2.5	The following is performed once the leaching process has been completed: to reduce the effects of evaporation, and to prevent the contamination of each cell by airborne contaminants, place the lids on their corresponding cells and let the cells complete the leachate draining process for the remainder of the leaching day and overnight.	ASTM	
11.2.6	Disconnect the cells on the day following the leach, and weigh and record the weight of each cell and Erlenmeyer collection flask. Set each filled collection flask aside for leachate analyses. (Measurements of pH and Eh and sample preservation procedures must be performed as soon as possible after leachate collection.) Return each cell, replace the filled collection flasks with clean, tared Erlenmeyer flasks, hook up all connections, and begin the dry-air cycle.	ASTM	
11.3	Dry-Air Cycle:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.3.1	The commencement of the three-day dry-air period marks the beginning of each new weekly cycle of the accelerated weathering humidity cell test; the first full-week cycle after the first leaching is designated Week 1; subsequent weeks (commencing with the second dry-air period) are designated as Week 2, Week 3 … . Week n, etc.	ASTM	
11.3.2	To perform the dry-air cycle, feed air is metered to the humidity cell array with a flowmeter (see 6.3) set at a target rate in the range of 1 to 10 L/min per cell, depending on the objectives of the testing. The air flow rate must be checked daily and adjusted to the target value ± 0.5 L/min.	ASTM	
11.3.3	Feed air from the flowmeter is routed first through a desiccant column and then to each of the sells through a dry-air manifold. Air exiting the desiccant column should have a relative humidity of less than 10 % as measured with a hygrometer (see 6.23).	ASTM	
11.3.4	To maintain similar positive air pressure through the cells, attach a water-bubbling vessel to each humidity cell air exit port coming out of the humidity cell lid; a 50-mL Erlenmeyer flask with a rubber stopper containing a vent and air inlet tube serves as a simple and efficient bubbler.	ASTM	
11.3.5	The dry air is passed through each humidity cell for three days. Air flow rates from each of the cells should be checked each day, recorded, and adjusted, if necessary. See also Note 10.	ASTM	
11.4	Wet-Air Cycle:		
11.4.1	The three-day wet-air period commences on the fourth day of each weekly cycle.	ASTM	
11.4.2	To perform the wet-air cycle of the method, feed air is routed through a water-filled humidifier via aeration stones or gas dispersion fritted cylinders/disks and then to each humidity cell.	ASTM	
11.4.3	The water temperature in the humidifier is maintained at 30 ± 2°C to ensure that the sparged air maintains a relative humidity of approximately 95 % as measured with a hygrometer (see 6.23) from one of the humidifier exit lines. Air flow rates to each of the cells should be checked each day, recorded, and adjusted, if necessary.	ASTM	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
Note 10	It is good practice to measure the air flow rates and relative humidity of the air exiting each humidity cell during each day of the three-day dry- and wet-air periods; the measurements should be taken at the same time each day from the humidity cell air exit port; these measurements can be accomplished by installing a quick-disconnect fitting in the tubing that connects the air exit port to the bubbler.	NA	
Note 11	Coals spoils in eastern states are commonly saturated; Caruccio (10) has suggested the following geographic control alternative to the dry-air versus saturated-air scheduling: (1) Eastern States Samples – Six days of saturated air (versus three days dry/three days wet); and (2) Western States Samples – Three days dry/three days wet.	NA	
11.5	Subsequent Weekly Leaches:		
11.5.1	A second leach with water is performed on the day following the end of the three-day wet-air period (that is, day seven of the first weekly cycle). This leach marks the end of the first weekly cycle and is designated as the Week 1 leach.	ASTM	
11.5.2	Subsequent leaches are designates as Week2, Week 3 … Week n, and they mark the end of the weekly cycle for that numbered week. Perform each weekly leach as described in 11.2.2 – 11.2.5. Weekly weighing of the test cells is optional.	ASTM	No weekly weighing of the cells.
11.6	It is recommended that the weekly accelerated weathering cycles described in 11.2, 11.3, 11.4 and 11.5 be performed for a minimum of 20 weeks.	ASTM	
Note 12	Additional weeks of accelerated weathering may be required to demonstrate the nature of the material, depending on the chemical composition of the solid material. For some metal mining wastes, researchers have shown that as much as 60 to 120 weeks of accelerated weathering data may be required to demonstrate the complete weathering characteristics of a particular sample (7, 12). The criteria for ending the testing may be site specific and should be agreed before initiating the testing.	ASTM	
11.7	Leachate Analyses:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.7.1	Analyze the leachates for specific constituents or properties, or use them for biological testing procedures as desired, using (1) appropriate ASTM test methods or (2) methods accepted for the site where disposal will occur. Where no appropriate ASTM test method exists, other test methods may be used and recorded in the report, provided that they are sufficiently sensitive to assess potential water quality impacts at the proposed disposal site. Suggested minimum weekly analyses should include pH, Eh, conductivity, and selected metals could be analyzed less frequently (for example, at Weeks 0, 1, 2, 4, 8, 12, 16, and 20), especially if changes in leachate chemistry are slow. Whether visible phase separation during storage of the leachates occurs or not, appropriate mixing should be used to ensure the homogeneity of the leachates prior to their use in such analyses.	At the end of weekly cycle the volume of leachate collected is recorded. The leachate is filtered through a Gelman magnetic filter funnel fitted with a membrane filter with pore size of 0.45 microns and analyzed for the parameters listed in Table 2 of the RFP. Filtered leachate samples will be submitted to ALS Environmental/Cantest Ltd. for dissolved metals analysis as requested in Table 4 of the Waste Rock and Lean Ore Geochemical Characterization Plan. Conductivity, Eh, and pH are measured in the CEMI laboratory using standard procedures. An aliquot of filtered leachate is titrated with standardized sulphuric acid to pH 4.5 to calculate total alkalinity. Standardized sodium hydroxide is used to titrate an aliquot of leachate to pH 4.5 and to pH 8.3 to calculate total acidity. Analysis frequency: pH, cond, Eh every cycle; SO4, Cl, F, alkalinity, TIC, acidity cycle 0, 2, 4, 6 etc.; ICP-MS including Hg and Si cycle 0, 4, 8, 12, etc., ICP-ES including Si cycle 2, 6, 10, 14, etc.	
11.7.2	Table 1 is an example of a spreadsheet format used for recording 20 weeks of leachate analytical data.	ASTM	
11.7.3	Fig. 5 is an example of a method used to plot the temporal variation (by week) of leachate pH, sulfate load, and cumulative sulfate load from 21 weeks of accelerated load and release rates).	ASTM	
11.8	Weathered Solid Material Analyses:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.8.1	Weigh the humidity cell after collection of the final effluent and completion of a three-day dry-air period.	ASTM	
11.8.2	Transfer the weathered residue and filter media to a clean drying pan, and dry to constant weight at 50 ± 5°C. Record the final weight.	ASTM	
Note 13	Perform any gross sample examination (for example, sample texture and weathering product mineralogic characterization) desired for the weathered residues prior to pulverization. To facilitate such an examination, empty the humidity cell contains into a clean drying pan carefully by pushing gently on the bottom of the perforated plate with a wooden dowel until the sample exits the cell mouth. The perforate plate is accessed through the humidity cell drain port.	NA	
11.8.3	Identify and mark the top versus bottom portions of the sample for gross sampling purposes. Formations of cemented lumps of sample termed “ferricrete” that result from the accelerated weathering process arte common in iron-sulfide-mineral rich samples. Depending on the sample mineralogy, the degree of “ferricrete” cementation may vary vertically within the sample, and the investigator may wish to segregate the sample into upper, middle, and lower thirds to document and characterize such changes.	Procedure to be determined	
11.8.4	After drying to constant weight and prior to splitting, use an instrument such as a rolling pin to break up cemented lumps in the sample (if the cemented lumps cannot be sufficiently reduced to pass through the chutes of a riffle splitter, remove, record, and weigh separately):	ASTM	
11.8.4.1	Split the sample into halves using a riffle splitter with 2.54-cm (1-in.) chutes, and reserve one half to determine the particle size distribution in accordance with Test Method E 276.		Repeat same screen assay method as for pre-test characterization (s.9.5)
11.8.4.2	Split the remaining half sample into two quarters using a riffle splitter with 2.54-cm (1-in.) chutes, and submit one quarter for mineralogical characterization; pulverize the other quarter in either a ring-and-puck or disk-pulverizing machine to 95 % passing a 150-µm (100-mesh) screen in accordance with Test Method E 276.	Procedure to be determined	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.8.5	Mix the pulverized residue in a blender or on a rolling cloth. Use the prepared residue for chemical characterization and for comparison with the pre-weathered solid material sample.	Procedure to be determined	

Appendix E

Design of MDNR Reactor

Day, Stephen

From: Kim Lapakko [kim.lapakko@dnr.state.mn.us]
Sent: Tuesday, May 17, 2005 9:55 AM
To: Stephen Day
Cc: Dave Antonson; Jennifer Engstrom; Paul Eger
Subject: RE: Small reactor

Attachments: MN DNR psize methods 050517.doc

MN DNR psize methods 050517.do...

Steve,

Attached is a description of the reactors, masses, and rinse volumes used for various size fractions of Duluth Complex rock in our particle size experiment. As indicated in the attachment, I won't have access to the trace metal data from that experiment until tomorrow. I will need to examine this to help evaluate the expected metal concentrations in drainage relative to detection limits. I'm not sure it will give us as much as hoped because the sulfur contents of the samples typically were on the order of 0.9% to 1.3%. This may make extrapolation by more than an order of magnitude tenuous. It will be another pertinent piece of information.

Kim

>>> "Stephen Day" <sday@srk.com> 5/17/2005 11:18:50 AM >>>
Dave

A design drawing should be fine along with description of the procedure.

The main question is what do you do to scale-up the sample mass as the particle size increases? I want to copy your procedure exactly.

Thanks
Steve.

-----Original Message-----
From: Kim Lapakko [mailto:kim.lapakko@dnr.state.mn.us]
Sent: Tuesday, May 17, 2005 8:38 AM
To: Stephen Day
Cc: Dave Antonson
Subject: Small reactor

Steve,

Dave Antonson will email a figure depicting our small reactor, along with some design details (perforated plate, adehesive, filter). He could also send a reactor. Please contact him directly, with an address to send it, if you think that would be helpful.

Kim

17 May 2005

Steve,

In our particle size tests we used a small reactor and 75-g mass for particle sizes of –270, +270/-100, and +100/-35 mesh. We used the ASTM cell and 1000-g mass for +35/-10, +10/-0.25 inch, and +0.25/-0.75 inch particle sizes. For rinse volumes, we used 200 mL for the 75-g samples and 300 mL for the 1000-g samples. The 300-mL rinse volume was determined as the quantity of water, rounded up to the nearest 100 mL, required to submerge the solids.

I won't have access to the metal release data for the particle size experiment until tomorrow. As mentioned on the phone, sulfate release rates appear to vary linearly with surface area. It seems likely that nickel release rates will vary similarly, and I'll look into this further tomorrow. Hopefully this information will shed some light on the maximum particle size question.

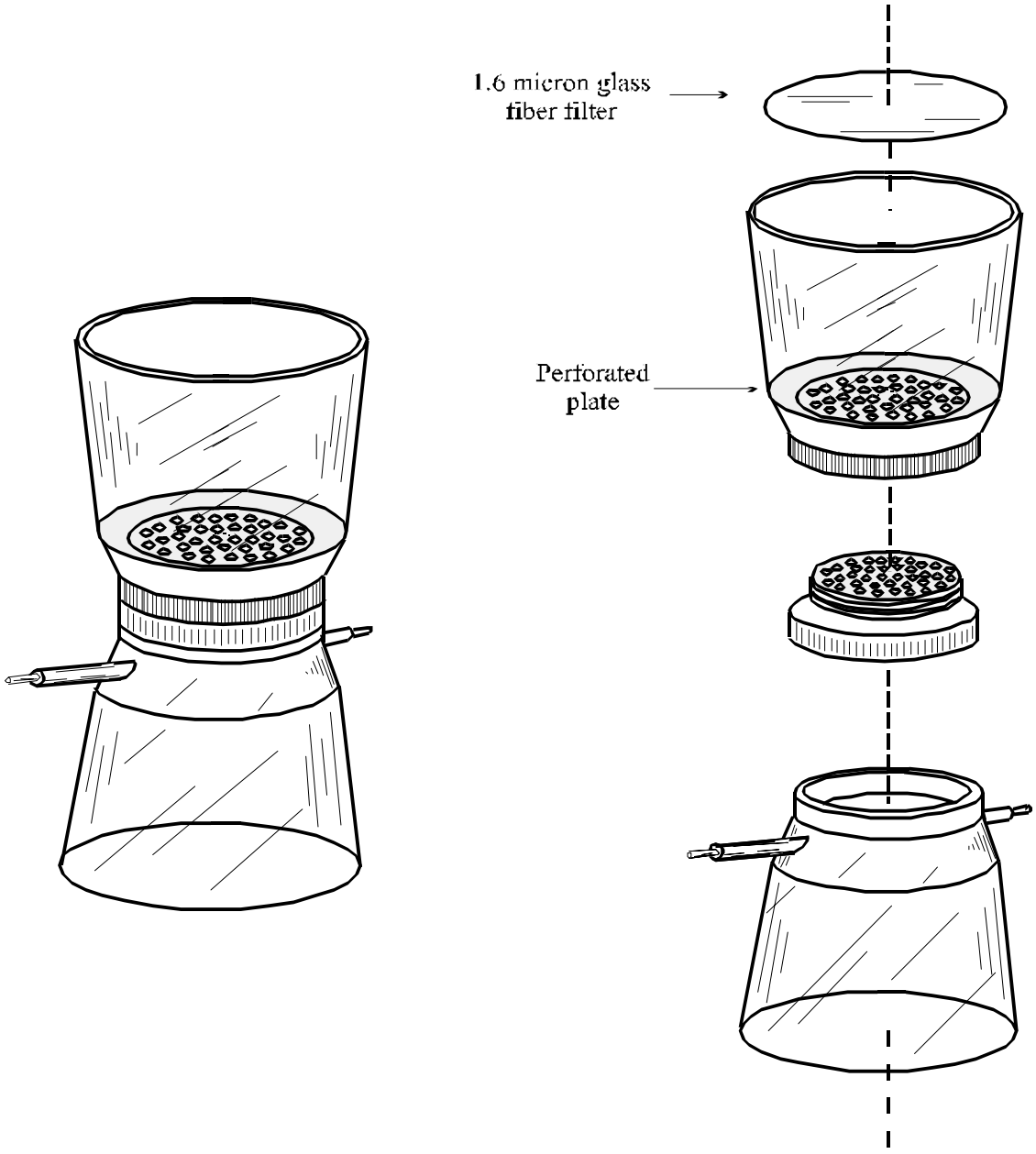
Day, Stephen

From: Dave Antonson [dave.antonson@dnr.state.mn.us]
Sent: Tuesday, May 17, 2005 11:53 AM
To: Kim Lapakko
Subject: reactor

Attachments: small reactor.doc

small reactor.doc
(271 KB)

see if this makes any sense. you can edit it if you want. if it seems adequate you can forward it to steve. maybe he doesn't need a sample of the base.


1.6 micron glass
fiber filter
Perforated
plate

The reactors were purchased from Millipore Corporation (1-800-645-5476). They are 47 mm Sterifil aseptic systems. You will need the 250 ml receiver flask, 250 ml funnel (top), silicone o-rings, and the filter holder base and support screen.

The perforated acrylic plastic base was purchased as flat stock and fabricated to fit the top funnel. The plates are 1/8" thick, 2 1/4" in diameter and tapered to fit into the reactor top. Approximately sixteen 1/16" holes were drilled in the plate. The plate was glued into the reactor using acrylic solvent cement purchased from United States Plastics (1-800-537-9724). Catalog # 44629 for 5 oz. tube. The acrylic flat stock was also purchased from United States Plastics.

After the plate is glued into the top of the reactor there should be approximatly a 3/8" gap between the bottom of the perforated plate and the top of the support screen of the filter unit.

The filter that rests on the perforated plate is a 55 mm Whatman GF/A glass microfibre filter (catalog # 1820 055).

Note: Before adding the solids to the filter you should wet the filter slightly with distilled water so no solids escape around the filter.

Appendix F

Design of Column Testing on Interaction between NorthMet and LTV Tailings

SRK Consulting (Canada) Inc.
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Memo

To:	Jennifer Engstrom, MDNR	**Date:**	December 8, 2005
cc:	Kim Lapakko, MDNR Paul Eger, MDNR	**From:**	Stephen Day
Subject:	Design of Column Testing on Interaction Between NorthMet and LTV Tailings	**Project #:**	1UP005.001

Jennifer

During our conference call on Oct 25, 2005, I agreed to provide design details for column testing to evaluate the effect of contact of leach waters from the NorthMet Tailings with LTV tailings when we received characterization data for the samples obtained from the LTV impoundment in September. Complete results have now been received.

This memorandum describes:

- Characteristics of LTV tailings.
- Characteristics of available tailings sample material.
- Proposed testwork.

The original proposed program was described in a memorandum dated September 23, 2005. SRK is seeking comments from MDNR on this proposal. It is acknowledged, as we discussed during conference call that this column testwork may represent a preliminary assessment. Additional testing may be required as the design for the tailings basin progresses.

1 Characteristics of LTV Tailings

1.1 Sample Analysis

Seven holes were drilled in the tailings to a depth exceeding 60 feet using a geoprobe. Samples have been analyzed from five holes with tailings having the following textural characteristics:

- GP-1 – Mainly coarse sand.
- GP-2 – Interlayered fine sand and slimes.
- GP-3 – Coarse sand grading into fine sand and slimes.
- GP-4 – Interlayered coarse and fine sands.
- GP-5 – Interlayered fine sand and slimes.

Samples were obtained as core and shipped whole to Canadian Environmental and Metallurgical Inc. At the laboratory, all discrete textural layers were tested for rate of HCl reaction (ie a “fizz” test) and qualitative magnetism as an indicator of magnetite content. Samples were selected from each hole to to represent the surface material (ie potentially weathered) and two samples of each textural type from each hole. These samples were submitted for relative density, moisture content and particle size determinations, quantitative mineralogy by x-ray diffraction and chemical analysis.

1.2 Results

Table 1 shows selected data sorted by the main textural groups (coarse sand, fine sand and slimes).

The dominant mineral in all samples was quartz which varied from 58 to 79% (by weight) but was not different in the three textural groups. Hematite and magnetite were present as expected. Magnetite was lower in the slimes samples likely resulting from density segregation as the tailings were deposited. Carbonates were a significant mineralogical component varying from 5 to 14%. Total carbonate content was greater in the slimes fraction compared to the coarse sands. Ankerite and siderite dominated and occurred in about equal amounts. The calcite content was lower than either ankerite or siderite.

Pyrite was detected in most samples but at very low levels. The sulphur content of the samples varied from 0.02 to 0.04% equivalent to pyrite content of 0.04 to 0.08%.

Silicates occurring in all samples were hydrobioitite, kaolinite, amphibole (cummingtonite ± grunerite), diopside, ferripyrophyllite (possibly minnesotaite) and albite. Other minerals occurring in a few samples were pyrophyllite, muscovite and hydroxylapatite. There was no evidence that the mineral distribution was related to particle size.

Distribution of metals was also unrelated to particle size with the possible exception of manganese which appeared to be elevated in the slimes. This is consistent with the higher carbonate content and indicates that manganese is associated with the carbonates.

2 Characteristics of PolyMet Tailings

Four tailings samples have been tested. These samples originated from processing of three ore composites nominally containing 0.35%, 0.4% and 0.45% copper (Parcels 1, 2 and 3, respectively). Flotation testing also considered addition of copper sulphate as an activator of pyrite. For Parcel 1 and 3, copper sulphate was evaluated. Also for Parcel 1, and Parcel 2 processing without the addition of copper sulphate was evaluated. The addition of copper sulphate does not materially affect the copper and sulphur content of the tailings directly but changes the mineralogical composition due to the selective recovery of pyrite to the bulk sulfide concentrate.

The characteristics of the four samples are shown in Table 2. Very little difference between the tailings was apparent. As expected, addition of copper sulphate in the process resulted in lower sulphur content in the tailings (0.1% and 0.15%, Parcels 1 and 3, respectively) compared to 0.23% and 0.2% (Parcels 1 and 2, respectively) without copper sulphate. Copper showed the widest variation in concentration (223 to 527 mg/kg). Variations of cobalt, nickel and zinc were small.

Table 1. Characteristics of LTV Tailings Samples

Core	Sampled		Material	Q	Py	Cal	Ank	Sid	Hem	Mag	Bio	Kao	Fe-Pyr	Albite low	As	Cd	Co	Cr	Cu	Fe	Mn	Ni	P	Pb	S	Zn
	Start ft	Finish ft		%	%	%	%	%	%	%	%	%	%	%	%	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	ppm
Coarse Sand																										
GP-1	8	12	Coarse Sand	72	0.2	0.3	3	3	3	4	1	1	2	2	31	-0.5	12	100	14	15.85	5970	5	240	7	0.02	14
GP-1	20	40	Coarse Sand	71	0.2	0.2	5	8	2	2	2	1	2	2	36	-0.5	14	90	25	15.5	7110	8	250	5	0.04	13
GP-3	8	12	Coarse Sand	58	0	0.4	6	6	2	3	5	1	6	4	16	-0.5	7	42	7	13.7	3420	3	250	4	0.02	9
GP-4	4	16	Coarse Sand	59	0.1	0.7	2	4	2	3	10	2	2	3	19	-0.5	7	57	9	15.45	3890	1	240	7	0.02	9
GP-4	20	24	Coarse Sand	79	0	0.1	3	2	2	3	3	1	2	1	21	-0.5	10	45	8	12.85	4010	3	250	5	0.02	34
n				5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Min				58	0	0.1	2	2	2	2	1	1	2	1	16	-0.5	7	42	7	12.85	3420	1	240	4	0.02	9
Median				71	0.1	0.3	3	4	2	3	3	1	2	2	21	-0.5	10	57	9	15.45	4010	3	250	5	0.02	13
Max				79	0.2	0.7	6	8	3	4	10	2	6	4	36	-0.5	14	100	25	15.85	7110	8	250	7	0.04	34
Fine Sand																										
GP-1	60	72	Fine Sand	72	0.1	0.7	3	2	1	1	2	2	2	4	22	-0.5	11	77	20	12.05	7010	3	330	3	0.04	14
GP-2	0	1	Fine Sand	60	0	0.2	6	7	2	2	11	4	3	0	15	-0.5	9	27	7	14.4	4270	-1	490	-2	0.02	10
GP-2	24	28	Fine Sand	62	0.4	1	6	3	1	2	5	2	3	3	22	-0.5	11	61	13	14.55	5340	3	550	7	0.02	14
GP-3	44	60	Fine Sand	68	0.2	0.5	4	3	3	2	4	1	3	1	15	-0.5	12	45	17	13.4	8510	4	400	7	0.03	12
GP-4	18	20	Fine Sand	73	0	0.2	7	3	2	2	4	2	2	1	43	-0.5	10	41	12	13.3	4020	2	290	8	0.02	13
GP-5	8	20	Fine Sand	78	0.1	0.1	4	2	3	3	3	1	2	1	14	-0.5	7	42	7	13.45	3630	2	270	2	0.02	7
GP-5	36	48	Fine Sand	73	0.4	0.4	4	2	2	3	2	1	3	2	14	-0.5	9	53	14	13.55	5820	2	290	4	0.02	8
n				7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7
Min				60	0	0.1	3	2	1	1	2	1	2	0	14	-0.5	7	27	7	12.05	3630	-1	270	-2	0.02	7
Median				72	0.1	0.4	4	3	2	2	4	2	3	1	15	-0.5	10	45	13	13.45	5340	2	330	4	0.02	12
Max				78	0.4	1	7	7	3	3	11	4	3	4	43	-0.5	12	77	20	14.55	8510	4	550	8	0.04	14
Slimes																										
GP-2	28	32	Slimes	62	0	1	4	4	2	1	5	2	3	5	18	-0.5	14	31	10	14.2	7390	3	530	4	0.02	13
GP-3	60	72	Slimes	62	0	0.7	8	5	3	1	7	2	2	1	15	-0.5	14	33	20	12	10050	4	590	2	0.04	14
GP-5	20	24	Slimes	70	0.1	0.3	6	4	3	2	6	2	3	1	25	-0.5	9	38	7	13.75	4830	1	460	4	0.02	8
GP-5	48	52	Slimes	72	0	0.6	4	6	2	1	4	1	2	1	16	-0.5	16	40	19	13.1	12400	3	550	4	0.03	11
n				4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Min				62	0	0.3	4	4	2	1	4	1	2	1	15	-0.5	9	31	7	12	4830	1	460	2	0.02	8
Median				66	0	0.65	5	4.5	2.5	1	5.5	2	2.5	1	17	-0.5	14	35.5	14.5	13.425	8720	3	540	4	0.025	12
Max				72	0.1	1	8	6	3	2	7	2	3	5	25	-0.5	16	40	20	14.2	12400	4	590	4	0.04	14

Notes
Minerals Q = Quartz, Ank = Ankerite, Hem = Hematite, Mag = Magnetite, Sid = siderite, Bio = hydrobiotite, kao = kaolinite, Fe-Pyr = Ferriprophyllite, Cal = Calcite, py =pyrite

Table 2. Characteristics of NorthMet Tailings Samples

Sample Number	Source and Process	Total S %	As ppm	Cd ppm	Co ppm	Cr ppm	Cu ppm	Mn ppm	Ni ppm	P ppm	Zn ppm
PIS-ACS TO DSC COMP	Parcel 1, $CuSO_4$ Added	0.10	<5	<0.5	59	186	223	1130	319	790	95
PISA TO D COMP	Parcel 1, $CuSO_4$ Not Added	0.23	<5	<0.5	62	188	248	1140	329	800	97
P2SA TO D COMP	Parcel 2, $CuSO_4$ Not Added	0.20	<5	<0.5	61	199	527	1125	385	750	93
P3S-A TO D COMP	Parcel 3, $CuSO_4$ Added	0.15	<5	<0.5	60	175	418	1110	372	770	92

3 Proposed Column Testwork

3.1 Design Basis

Based on the characteristics of the NorthMet and LTV tailings, the following chemical processes can be expected to occur within the layered tailings basins:

- In NorthMet Tailings
  - Near surface oxidation of residual sulfide minerals resulting in release acidity, iron, sulphate and trace elements (copper and nickel).
  - Development and migration of an oxidation front due to consumption of oxygen near the surface.
  - Attenuation of metals as a result of interaction between pore fluids and mineral grains.
- LTV Tailings
  - Enhanced dissolution of ankerite and siderite under saturated conditions resulting in release of calcium, magnesium, ferrous iron, reduced manganese and bicarbonate alkalinity.
  - Localized re-precipitation of ferric hydroxides and manganese oxides due to variations in pH and oxidation-reduction potential.
- Interaction Between NorthMet Tailings Pore Water and LTV Tailings
  - Possible sorption of metals by ferric hydroxides and manganese oxides, particularly in the immediate contact zone where LTV tailing are probably partially oxidized.
  - Precipitation of metal carbonates due to alkaline conditions.

Testing of the LTV tailings indicates little significant variation in mineralogical and chemical content. The primary variable expected to influence the degree to metal attenuation occurs, if at all, is particle size which will control the availability of adsorption sites, oxidation-reduction conditions and contact time.

3.2 Proposed Testwork

The proposed leach column design includes the following main features (Figure 1):

- In series leach columns designed to generate NorthMet tailings pore water as feed into LTV tailings.

- First column containing 10 kg of unsaturated NorthMet Tailings and second receiving column containing 5 kg of LTV tailings. The larger volume of NorthMet tailings is intended to optimize development of near equilibrium pore water chemistry below an oxidation zone in the NorthMet tailings. The smaller volume of LTV tailings may allow breakthrough of NorthMet tailings water chemistry to be observed.
- NorthMet tailings open to atmosphere.
- Connecting pipe between columns and sampling "T" operated to exclude oxygen.
- Side sampling port in LTV tailings column to enable sampling of pore water just below the entry point.
- Sampling of final effluent.
- Application of 2 L of deionized water every week to allow withdrawal of up to 250 mL of water from each of the two intermediate location and 1.5 L of the final effluent. This application rate represents approximately one pore volume every 4 weeks.
- Analysis of intermediate sampling points for pH and Eh every week and composite sample for anions and cations every other week (including sulfur).
- Analysis of final effluent for same parameters, sulphate, alkalinity and anion scan.

Approximately 11 kg of each of the tailings samples shown in Table 2 are available for additional testing. The following matrix summarises six proposed tests (Table 3). It is preferred to test slimes and coarse sands as two extreme characteristics of LTV tailings. From experience, it is unlikely that the slimes will transmit sufficient water for the experiment. Therefore, a fine sand composite would be used instead.

Each "X" in Table 3 represents an in-series column pair. The two control experiments will operate without NortMet tailings in the first column to evaluate leachate chemistry from LTV tailings. This will allow comparison with seepage chemistry in the existing basin.

Table 3. Matrix of Proposed Tests

		NorthMet Tailings Samples		
		Control No NorthMet Tailings	**Lower S Composite (Parcels 1 and 3, $CuSO_4$ Added)**	**Higher S Composite (Parcels 1 and 2, $CuSO_4$ added)**
LTV Samples	Fine sand or Slimes Composite	X	X	X
	Sand Composite	X	X	X

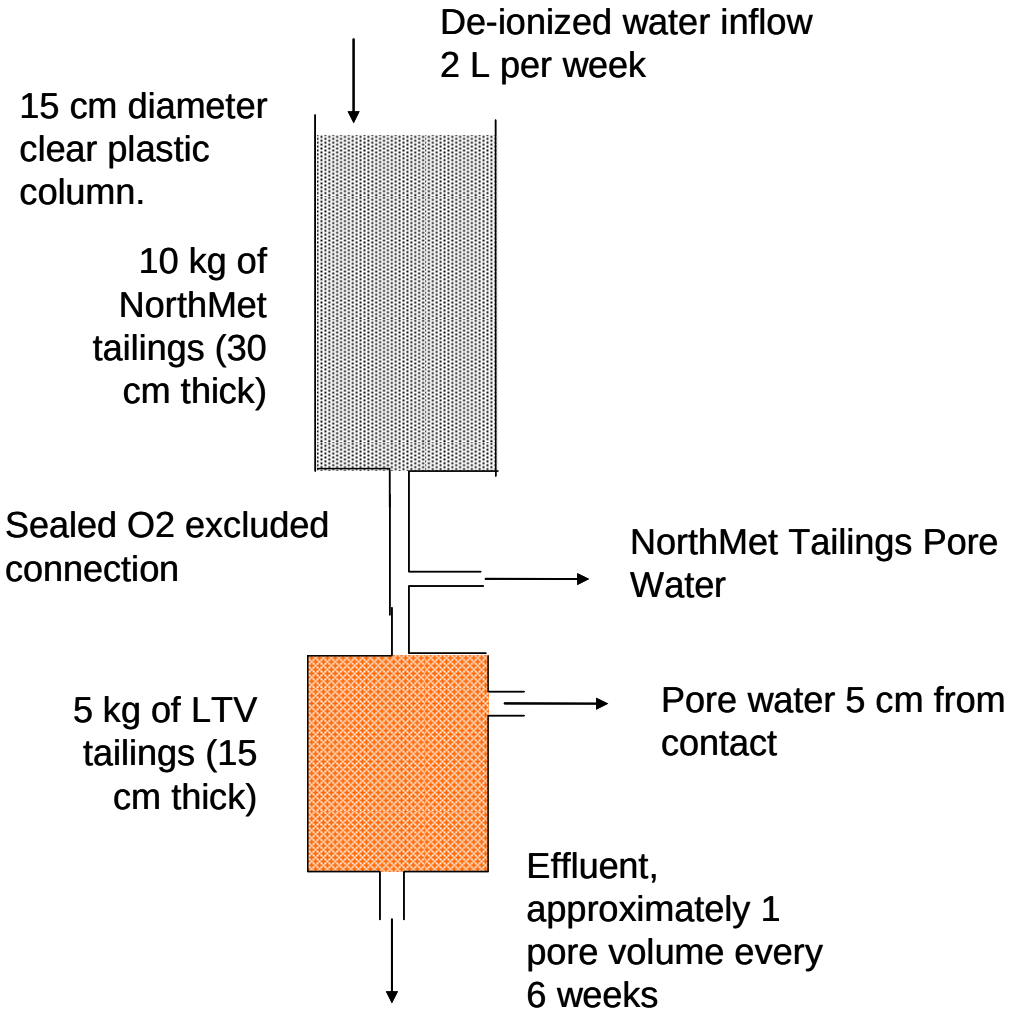

Figure 1. Schematic of In-Series Column Design

Attachment 2-5

MDNR. Letter from Jennifer Engstrom, MDNR, to Jim Scott, PolyMet Mining. August 13, 2007.

Minnesota Department of Natural Resources
Division of Lands and Minerals
500 Lafayette Road
St. Paul, Minnesota 55155-4045

August 13, 2007

Jim Scott
PolyMet Mining, Inc.
PO Box 475
Hoyt Lakes, MN 55750-0475

Dear Jim,

The Division of Lands and Mineral's comments on RS documents (RS553/42, RS257, RS22, RS18) mentioned that overburden at the NorthMet site should be characterized. The quantity of material indicated in the detailed project description (January 2007) is larger than initially anticipated. It needs to be characterized so that the quality of drainage from the overburden can be accounted for prior to mining. The attached informal summary provides perspective on concerns with the overburden projected to be excavated and stockpiled. Based on this information and other information or experience at PolyMet's disposal, we are requesting a plan for characterizing the overburden material. We can discuss the initial plan and work toward a mutually agreeable characterization plan.

It might be efficient to include hydrological and geotechnical characterization in this plan as well. Additional data on soil properties have been requested in our comments on the stockpile design reports. Data are needed for stockpile foundations, covers, and liners. Careful design of a sampling program might provide a single set of representative samples that could be subjected to a wide range of analyses. Please lend this concept consideration.

Sincerely,

Jennifer Engstrom

Enclosures

Attachment 2-6

SRK Consulting. Overburden Geochemical Characterization Plan in Support of EIS – DRAFT, NorthMet Project. Memorandum to Stuart Arkley, MDNR. February 22, 2008.

SRK Consulting (Canada) Inc.
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vancouver@srk.com
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Tel: 604.681.4196
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Memo

To:	Stuart Arkley, MDNR	**Date:**	February 22, 2008
cc:	John Borovsky, Barr Engineering Jim Scott, PolyMet	**From:**	Stephen Day Richard Patelke, PolyMet Nancy Dent, Barr Engineering
Subject:	Overburden Geochemical Characterization Plan in Support of EIS - DRAFT NorthMet Project	**Project #:**	1UP005.001

1 Objective

Mining and stockpile construction at the NorthMet Project will involve pre-stripping of an estimated 32 million tons of overburden (glacial drift) deposited by glacial action and subsequently formed soils in upland areas and depressions. It is currently estimated that 97% of overburden will be dominantly mineral material with the balance of 3% composed mainly of peat-type materials. Mineral overburden will be needed for waste rock stockpile foundation construction, road construction and soil covers on stockpiles for a total of about 17 million tons. The remaining balance will be placed in the waste rock stockpiles depending on its chemical reactivity to be determined by pre-mining characterization and monitoring.

To the extent practicable, peat will be stripped and stockpiled in a manner to allow for future reclamation use.

Overburden is typically composed of a mixture of locally- and distally-derived material comprised of rock derived by erosion and re-working of bedrock and pre-existing surficial material. In areas of bedrock mineralization it is possible, but not certain that the glacial drift could contain mineralized materials which could be a source of immediately or potentially soluble contaminants and acidity.

Therefore, the objective of this Overburden Geochemical Characterization Plan is to initiate data collection on the influence of mineralized bedrock on overburden composition in support of subsequent further characterization for permitting, development of plans for excavation, final disposal and water management of stockpile drainage.

At this stage, sufficient information is needed for the preparation of the Draft Environmental Impact Statement (DEIS). The information needed includes:

- estimation of potential reasonable worst case requirements for selective management of components of the overburden; and
- reasonable worst case prediction of immediate and future water quality from the stockpiles and other locations where overburden is used.

A higher level of monitoring information may be required to implement management of the overburden during operation. It is expected that management plans will be developed to support application for the Permit to Mine and could include characterization of overburden by drilling in advance of stripping to allow annual calculation of financial assurance payments, and on-site characterization of samples at the dig face. The scope of these plans will be determined following completion of the current characterization program for the DEIS.

2 Background

2.1 Regional Surficial Geology

2.1.1 Surficial Geology

Olcott and Siegel (1978) described the surficial geology of the copper-nickel study as being the result of two south-westward advances of the Rainy Lobe of the Laurentian ice sheet. The direction of ice movement was in part controlled by regional geologic structures. The lithological composition of rocks in the glacial deposits showed a contribution of the Duluth Complex units to ice contact and till deposits and less to outwash deposits.

Surficial geology in the area has been mapped by the Minnesota Geological Survey at a scale of 1:100,000 (Jennings and Reynolds 2005). Four units were described in approximate decreasing order of significance:

- Qrt – Till. This unit chiefly has a sandy loam matrix texture with common pebbles, cobbles and boulders. Where massive, compact layers are interpreted to have been deposited beneath moving ice, whereas stratified deposits were more likely deposited at the ice margin during moraine formation and retreat. The mapping indicates it occurs mainly in the eastern part of the project (Figure 1).

- Qrp – Till, re-sedimented till and sorted sediment. This unit marks the transition from a glacial to a proglacial setting (for example, ice contact delta fronts). It is interpreted as being created by deposition of basal Rainy lobe till at the ice front (Qrt) followed by re-working by gravity and slope processes down a steep morainal front. Due to the environment of formation, facies are not laterally continuous. The mapping indicates that this unit is mainly present in the proposed West Pit area.

- Qrm – Till. This unit is the same as Qrt but it is interpreted as being eroded by water concentrating coarse-grained clasts as a lag deposit at the surface.

- Qp – Peat. This unit is composed of organic material in various stages of decomposition. These are interpreted as formed in shallow depressions of glacial origin. Due to the scale of mapping, peat is shown as less extensive than actually occurs at the site.

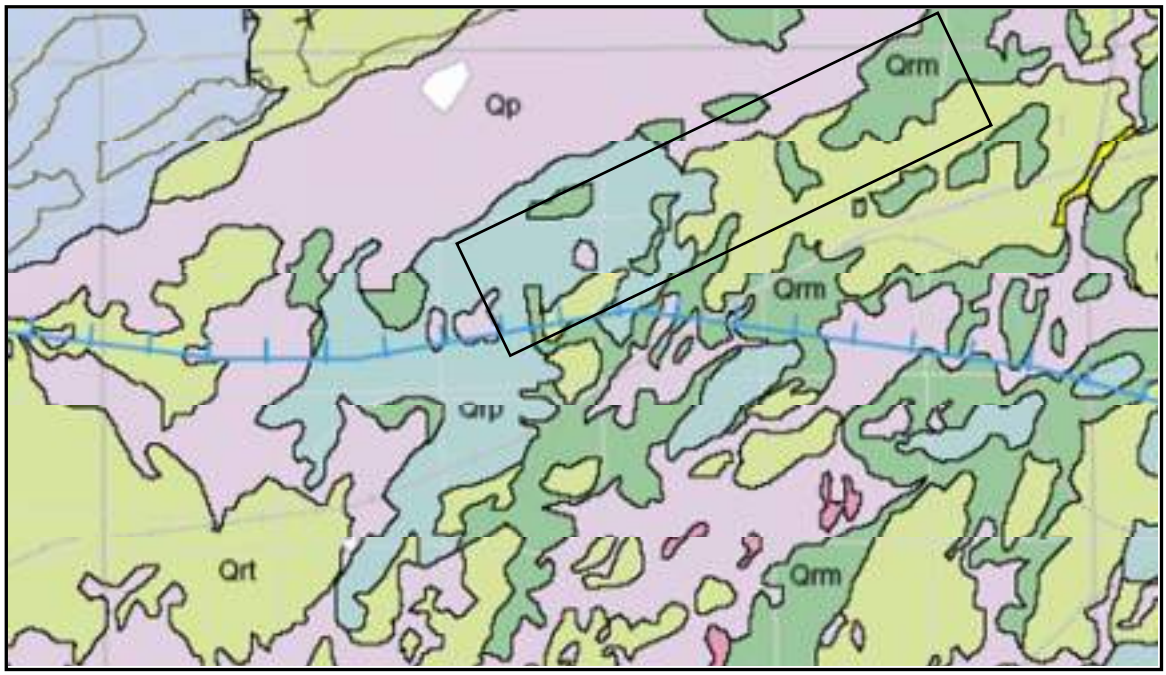

Figure 1. Excerpt from MN Geological Survey Map (M-164) (Jennings and Reynolds 2005). The blue line indicates an interpreted ice margin. The approximate footprint of the project is outlined.

Based on presence of rare Duluth Complex boulders north of the Virginia Formation-Duluth Complex contact, and ice flow directions referenced in Lehr and Hobbs (1992) PolyMet's geologists conclude that ice of the Rainy lobe generally moved from roughly east to north of northeast toward west to south of southwest. In other words, ice flow roughly paralleled the strike of units in the Partridge River intrusion of the Duluth Complex.

2.1.2 Overburden Geochemistry

Thorleifson et al (2007) describe a state-wide regional overburden sampling program. Two samples were collected from the Rainy lobe tills overlying the Duluth Complex (R12 and R13, 14 and 20 miles from the project site respectively). Due to the distance from the site, these samples only provide general information on the regional characteristics of tills. These samples were characterized by the presence of reddish volcanics (indicating Superior basin provenance), regionally relatively low carbonate content (less than 25th percentile, or 2.9% carbonate as dolomite and calcite), and regionally elevated nickel content (greater than 90th percentile, or 51 mg/kg). The latter, although present in the Duluth Complex, is regionally elevated in the northeast part of the state and appears to originate from beyond the local influence of the Duluth Complex.

Olcott and Siegel (1978) did not analyze their overburden samples but instead analyzed lake sediment samples. They noted no agreement between metal concentrations and proximity to the Duluth Complex.

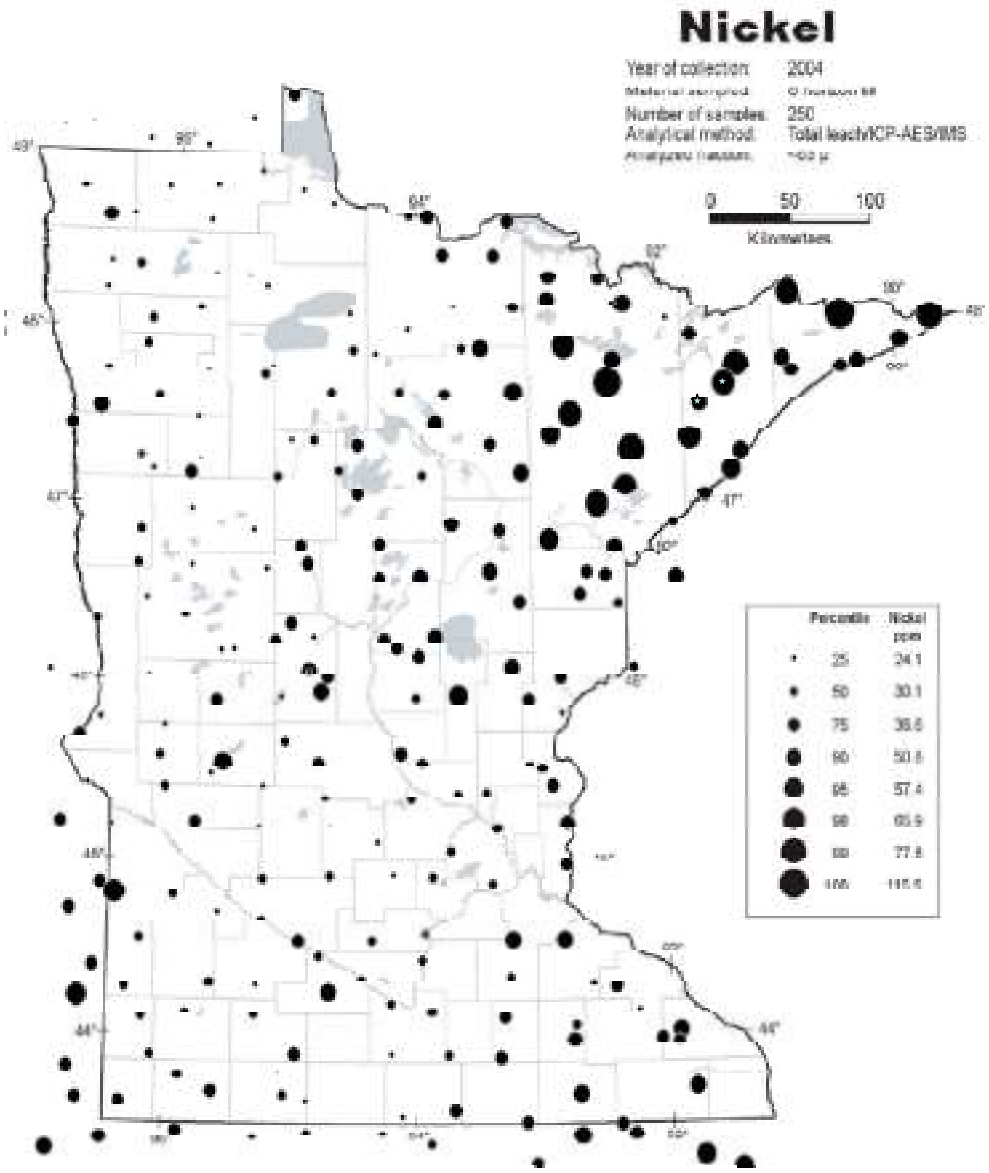

Figure 2. Nickel concentrations in regional overburden samples (Thorleifson et al 2007). The stars indicate samples collected nearest to the project site.

Mafic to ultramafic complexes like the Duluth Complex can exert a significant influence on regional geochemistry at a more local scale. Day and Matysek (1989) reported that stream sediments collected at a density of roughly 1 sample per 10 km^2 derived from an ultramafic complex in northern British Columbia contained average nickel and cobalt concentrations of 1356 mg/kg and 46 mg/kg (respectively) compared to nearby intermediate intrusions (4 to 11 mg/kg and 8 to 9 mg/kg, respectively).

3 Site Overburden Characteristics

Existing information relevant to the overburden characterization plan includes the thickness of overburden, the chemical characteristics of groundwater, and the chemistry of rock immediately in contact with the overburden deposits.

PolyMet has determined overburden thickness using exploration drilling records, regional and local outcrop mapping, test pits, and geophysical soundings. An overburden thickness map is provided in Figure 3. This mapping will be used as basis to estimate the volumes of different types of overburden indicated by the characterization work.

No data are available for the geochemical characteristics of overburden in the project area. However, shallow groundwater data in the mine area provides some indication of natural weathering characteristics (see RS02 and RS10A). Groundwater is non-acidic (lowest pH 6.6) but sulfate is variable (undetected to 66 mg/L in 21 samples) and at least two wells had elevated nickel (0.13 to 0.14 mg/L), copper (0.07 to 0.10 mg/L and/or cobalt (0.007 to 0.009 mg/L) compared to near detection (0.002, 0.002 and 0.001 mg/L respectively) in many other wells).

Figures 4 and 5 show the composition of rock at the overburden contact inferred from block modeling. Ore blocks are shown outside of the pit area because these are inferred rather than measured or indicated and therefore cannot be included in the ore reserve and pit design.

These maps confirm the presence of rock containing elevated sulfur and metal concentrations in these blocks though the degree to which this rock was transported into the overburden will be evaluated by the characterization plan.

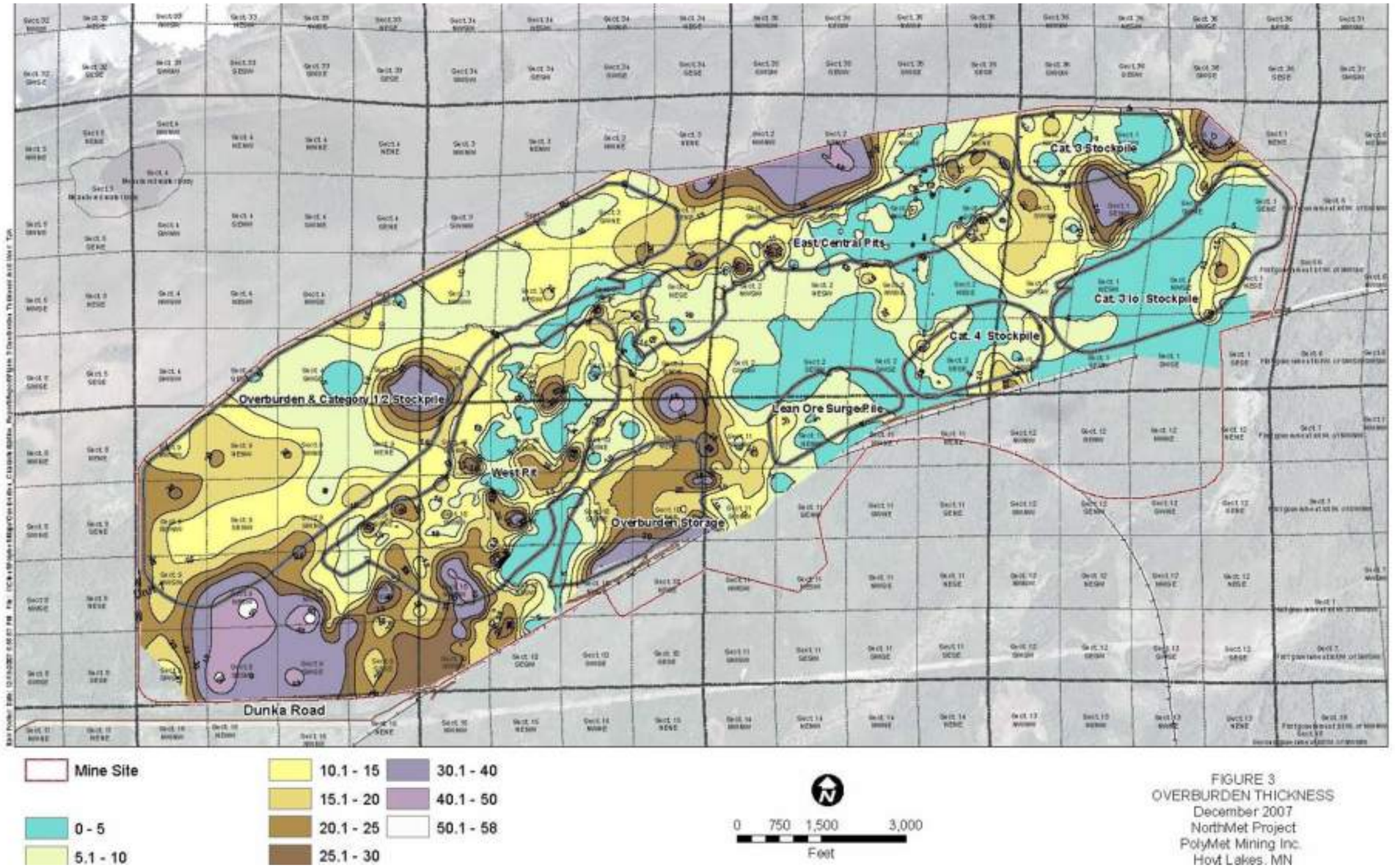

Figure 3. Overburden Thickness Map (in feet)

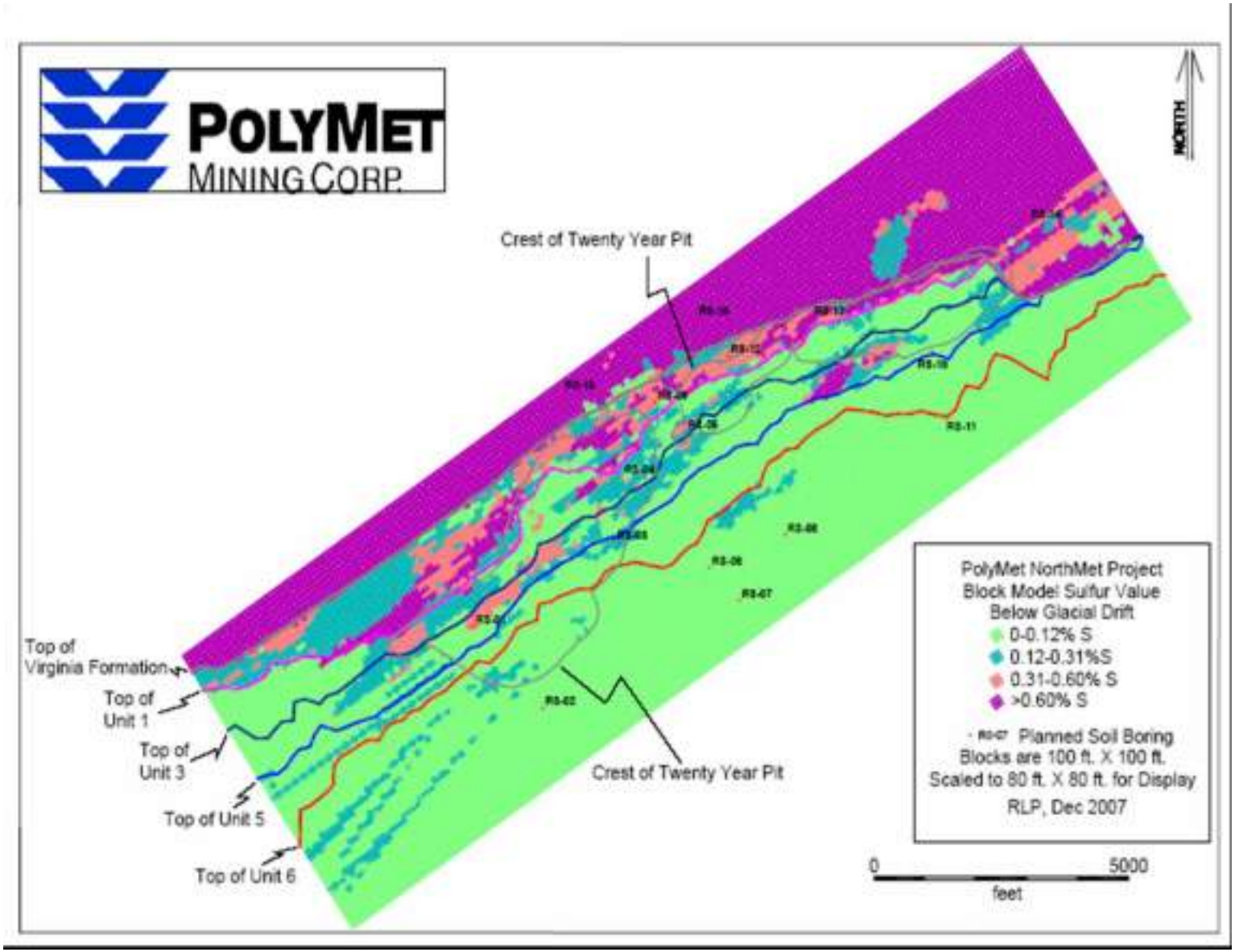

Figure 4. Sulfur Content in Blocks in Contact with Overburden

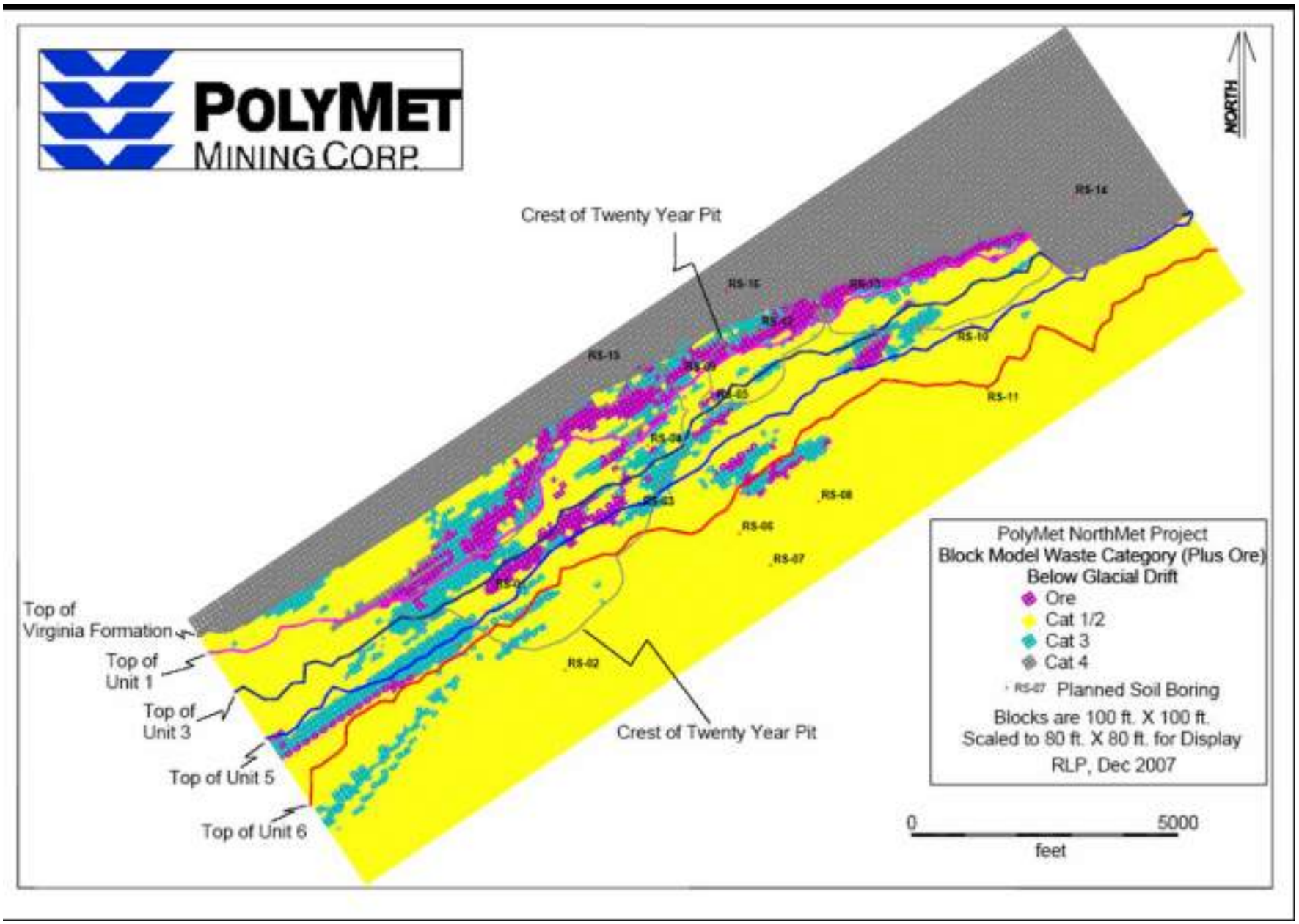

Figure 5. Waste and Ore Categorization of Blocks in Contact with Overburden

4 Program Design

4.1 Conceptual Geological and Geochemical Model

4.1.1 Glacial Processes

A conceptual model has been developed from first principles to guide design of the sampling program (Figure 6).

Based on the findings of Olcott and Siegel (1978), the bedrock in the area can be expected to have influenced the composition of glacial till in the area. The conceptual model is relatively simple. As ice contacted different units, rock was plucked from the near surface materials (which may have been already partially oxidized) and entrained in the glacial till as a wide range of particle sizes including clays to boulders. This leads to a dispersion train in the till which becomes progressively "diluted" by mixing with rock sourced further downstream (Figure 6a). As the ice direction parallels or nearly parallels the geological structure, the degree of dilution may be relatively small (Figure 6b).

The above concept appears to apply to the eastern two-thirds of the project site. Jennings and Reynolds (2005) describe a local transition to "re-sedimented" till in the western part of the property. In this area, periglacial processes can be expected to have resulted in locally highly variable deposits possibly resulting in disruption of the dispersion trains at the property scale shown in Figure 6 but also locally concentrating indicator minerals.

4.1.2 Post-Glacial Processes

Following retreat of the ice, surficial materials begin to weather and oxidize depending on the material characteristics (physical and geochemical), overall thickness of the deposit, and the degree of saturation. Clayey materials can be expected to oxidize slower than sandy materials due to the lower oxygen entry. Tills below the water table can also be expected to oxidize much less rapidly than unsaturated tills. The weathering processes can be expected to result in conversion of silicate minerals to clays, dissolution of carbonate minerals and oxidation of sulfide minerals. As with weathering of waste rock, some of these weathering products may be retained as secondary minerals whereas others will be leached into groundwater. There is some evidence of the latter in existing groundwater data.

Leaching of weathering products to groundwater may result in local accumulation of metals in low points formed where groundwater emerges. If metals are enriched in groundwater but under less-oxidizing conditions, iron may precipitate from emergent seepage and cause co-precipitation of metals. Similarly, peats may contain elevated metals as sulfides where reduction of sulfate has occurred.

4.1.3 Effect of Stripping

During stripping of overburden, rock particles will be exposed to leaching by incident precipitation. Drift, whether un-oxidized or oxidized, will then be exposed to conditions allowing greater oxidation rates. When exposed and placed in the overburden stockpile, existing soluble weathering products will be leached resulting in an initial flush of water containing elevated dissolved solids. Subsequently, oxidation and weathering will control water chemistry in much the same way that the same processes control water chemistry during waste rock weathering. Therefore, variables such as sulfur and carbonate mineral content are expected to control water chemistry by controlling pH and metal leaching.

Exposure of peat could result in oxidation of syngenetic (formed at same time as surrounding material, or formed in place) sulfide minerals, sorbed ions or organic compounds resulting in release of metals. Metal hydroxide precipitates may be exposed as peats are excavated resulting in exposure to leaching.

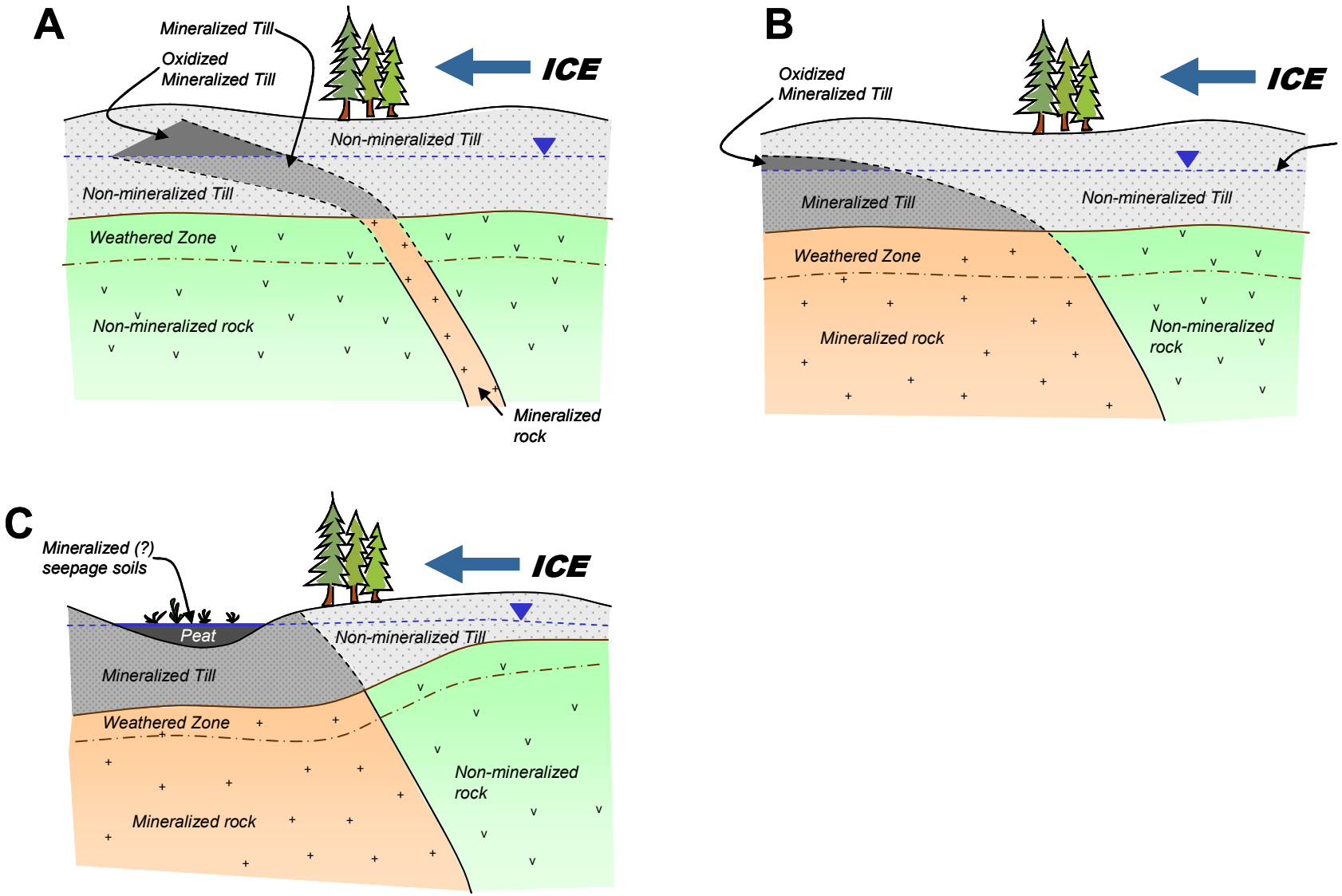

Figure 6. Conceptual Models for Dispersion of Bedrock Material in Glacial Overburden

4.2 Application of the Conceptual Model to Characterization Design

The conceptual model indicates variables in the natural system that could be considered in the characterization program (Table 1).

Table 1. Variables Considered in Experimental Design

	Variable	Effect	Values	Characterization Component
1	Up-ice origin of glacial till	• Regional mineralogical composition of till. • Acid buffering.	• Rainy Lobe.	• Up-ice sampling locations
2	Underlying bed rock geology and large scale mineralization	• Overall reactivity. • Source of leachable salts when exposed to precipitation during mining. • Potential for long term oxidation	• Sulfide mineralized Virginia Formation • Sulfide mineralized Unit 1 of the Partridge River Intrusion • Other less to unmineralized units of the Partridge River Intrusion	• Sampling locations within each of the three major units.
3	Distance from source of sulfur and metals (mineralized Duluth Complex and host rocks).	• Dilution of source effect	• Distance of sampling point from known bedrock mineralization indicated by drilling.	• Down-ice sampling locations
4	Glacial and Periglacial Depositional environment	• Mixing of till. • Degree of heterogeneity of till. • Potential for syngenetic sulfides	• Till • Re-sedimenting of till • Peat	• Samples in areas of both types of till. • Samples in wetlands.
5	Groundwater level during deposition	• Degree of oxidation of till.	• Oxidized below current water level. • Un-oxidized below current water level. • Above current water level.	• Description and sampling of oxidation state during drilling
6	Current position of water table	• Degree of oxidation of till • Presence of seepage soils	• Oxidized • Un-oxidized	• Mapping of water level while sampling. • Observation of mottling, • Samples above and below water table. • Sampling of peat

Consideration of these variables leads to the waste characterization plan design which at this stage is intended to evaluate the variability in acid generation and metal leaching associated with each variable (last column of Table 1).

Analytical methods have been selected to evaluate potential for immediate release of acidity and metals when exposed to meteoric water, and potential for long term leaching due to the oxidation of sulfide minerals and change in pH conditions.

5 Sampling and Analysis Plan

5.1 Field Program

5.1.1 Locations

Sampling locations (Figure 7) were selected to provide information to address each of the variables indicated in Table 1. Due to requirements of the US Forest Service, drilling locations are along existing roads in the project area and the number of locations has been fixed. This placed some constraints on the design of the program including lack of suitable sites to characterize till up-ice on the property in the Partridge River intrusion, and limited opportunities to characterize till overlying Virginia Formation at depth. However, the program as described provides sufficient coverage of other variables in the context of the overall objectives of the program.

Table 2. Selection of Drill Holes and Variables Characterized

DH	Depth	Up-ice origin of glacial till	Underlying bed rock geology and large scale mineralization			Dispersion Train[1]			Glacial and Periglacial Depositional environment			Groundwater level during deposition	Current position of water table	
	ft	-	VF	Unit 1	Others[2]	VF	Unit 1	Others[2]	Till	Re-sedimented Till	Peat	-	Saturated	Unsaturated
1	23				X			Down		X		X	X	X
2	13				X			Down		X		X	X	X
3	25?				X			Mid		X	X	X	X	
4	12				X			Mid		X	X	X	X	
5	16				X			Mid	X			X	X	X
6	23				X			Mid		X		X	X	X
7	16		X					Mid	X		X	X	X	
8	18				X			Mid	X			X	X	X
9	17			X			Down			X	X[3]	X	X	X[3]
10	17				X			Mid	X			X	X	X
11	15				X			Mid	X			X	X	X
12	15			X			Mid		X		X[3]	X	X	X[3]
13	14			X			Mid		X		X[3]	X	X	X[3]
14	7	X	X			Up			X			X	X	X
15*	27		X			Mid			X					X
16*	19		X			Down				X				X

Notes:

* - Hand borings

1. Units 1, and other (Units 2 to 7).

2. Relative dispersion distances – up = up-ice; mid = mid station; down = down-ice.

3. One of borings 9, 12 and 13 will be adjusted in the field to obtain peat samples. Movement to peat will result in these locations representing saturated conditions.

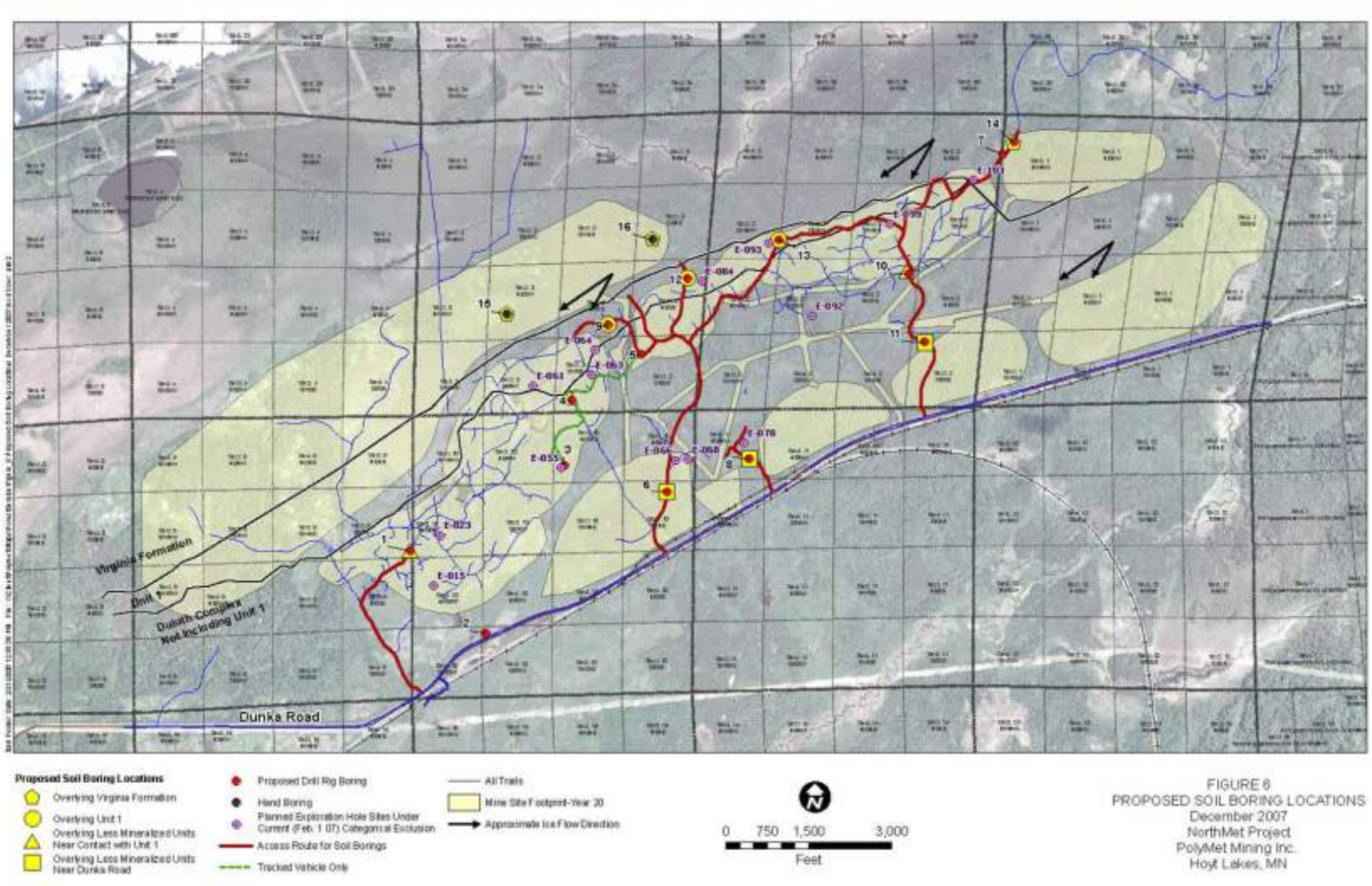

Figure 7. Proposed drilling locations in the context of the mine footprint, geology and wetlands.

5.1.2 Drilling Method

Locations 1 to 14 will be drilled using a rotosonic rig. See pdf files at www.geotechinc.com/equip_rotosonicdrilling.html for description of rotosonic process.

The drilling method is capable of drilling through rocky material; however, in the event of refusal before intersecting bedrock (as indicated by current understanding of overburden thickness in Figure 3), the drill rig will be moved a short distance and the hole re-started.

Locations 15 to 16 will be drilled using manual equipment due to access limitations. A hand auger will be used. Since the ground is frozen and rocks may be encountered, the hand boring is likely to penetrate to a maximum depth of about 5 feet.

5.1.3 Field Observations

Field logs prepared by Barr Engineering and PolyMet Mining staff will include the following information:

- Color of matrix and mottles (Munsell).
- Estimate percentage of lithic materials in coarse clasts.
- Sulfide mineralogy and quantity of each mineral of coarse clasts.
- Texture of materials.
- Percentage of organic matter.
- Relative moisture content.
- Oxidation features.

5.1.4 Field Analysis

Field analysis on all visibly distinct layers indicated by textural, color and compositional changes will include:

- Semi-quantitative rating of reaction of 10% hydrochloric acid with matrix and selected coarse clasts.
- Field rinse test for soluble salts. The procedure will involve stirring of 50 g of the matrix with 50 mL of de-ionized water to form a slurry. The resulting pH and specific conductivity of the supernatant will be recorded.

Field observations and sampling intervals will be compiled into logs and spreadsheets by Barr Engineering staff. These will be transferred to SRK for interpretation and selection of samples for laboratory analysis.

5.1.5 Sample Collection

Samples will be collected roughly every five feet and stored in plastic bags. Actual sampling intervals will be determined in the field from the core logs. In each hole, specific intervals will be selected by field personnel to characterize expected variations which will include changes in observed lithology, degree of oxidation and saturation. Barr Engineering will conduct the field sampling activities with support from SRK. Representatives of the DNR have made arrangements to observe the field sampling activities and collect split samples if desired.

As field conditions and core recovery allow, the goal will be to obtain 10 lbs of sample from each interval. This quantity of material may prove impractical, particularly at manual sampling locations.

Samples will be shipped in coolers to Canadian Environmental and Metallurgical Inc. (CEMI) in Vancouver, British Columbia for analysis. Samples will be refrigerated until the analysis plan is agreed with the DNR to minimize weathering and oxidation of unstable components.

5.2 Physical Analyses

A portion of each sample will be shipped to Barr Engineering for physical analysis which include:

- Atterberg limits (ASTM D 4318 - Liquid Limit, Plastic Limit, and Plasticity Index of Soils)
- Grain Size (ASTM D 421 - Dry Preparation of Soil Samples for Particle-Size Analysis and Determination of Soils or ASTM D 2217 - Wet Preparation of Soil Samples for Particle-Size Analysis and Determination of Soil Constants, and ASTM D 422 - Standard Test Method for Particle-Size Analysis of Soils)
- Consolidation (ASTM D 2435 - One-Dimensional Consolidation Properties of Soils)
- Permeability (ASTM D 5084 - Measurement of Hydraulic Conductivity of Saturated Porous Materials Using a Flexible Wall Permeameter)
- Strength (ASTM D 2850 - Unconsolidated, Undrained Compressive Strength of Cohesive Soils in Triaxial Compression, and/or ASTM D 2166 - Unconfined Compressive Strength of Cohesive Soil, and/or ASTM 4767 - Consolidated Undrained Triaxial Compression Test for Cohesive Soils)

5.3 Chemical Analysis

5.3.1 Sample Processing

Upon arrival at the laboratory, all samples of inorganic soils will be weighed and a sub-sample obtained for moisture content. These samples will be dried and screened to obtain three particles sizes nominally to represent the coarse (greater than 2 mm) and sand (2 mm to 74 μm) and silt to to clay (less than 74 μm) components. The latter will represent the potentially more reactive component of the soils.

Peat samples will be handled to minimize loss of syngenetic components formed under chemically reducing conditions. As indicated above, all samples will be refrigerated prior to analysis.

5.3.2 Selection of Samples for Analysis

SRK will select a subset of samples for analysis in consultation with the MDNR based on the field observations and field analysis results. In addition to testing samples to characterize the variables shown in Table 2, the field measured pH of the samples will be used to select a range of pH for evaluation of metal leachability.

5.3.3 Acid Generation Potential

Acid generation potential will be estimated using acid-base accounting on matrix and clast materials of overburden materials. The acid-base accounting procedure will include:

- Total sulfur (Leco Furnace)
- Sulfur as sulfate (HCl soluble, Sobek et al 1978)
- Neutralization potential (Sobek et al 1978)
- Total carbon (Leco Furnace)
- Total inorganic carbon (direct determination by evolved CO_2 by reaction with hydrochloric acid)

5.3.4 Metal Leaching Potential

Metal leaching potential will be evaluated using two methods:

- Total metal content of the matrix and clast materials using aqua regia and four acid digestions.
- Meteoric water mobility procedure (MWMP, NDEP 1996) will be performed on whole overburden samples.

Initial samples for this procedure will be selected in consultation with the MDNR based on the results of the field screening procedures. A second set of analyses may be completed in consultation with the MDNR based on the initial results.

5.4 Sample Archive

All samples obtained for this program will be archived for future analysis.

6 Data Analysis

The data obtained from this program will be used for two specific applications:

- **Evaluate the differences in geochemical characteristics for overburden collected over different rock units and the effect of variations in depositional environment and existing oxidation condition.** The evaluation will be performed by grouping the data to evaluate the various combinations of variables (for example, bedrock type, degree of saturation, distance along ice flow direction, depth) and will indicate the potential need for segregation of overburden and what the segregation criteria might need to be. At an initial level, comparative averages and range measures (percentiles, confidence limits) will be compared. If the data permit, more sophisticated methods (e,g, analysis of variance) may be used to evaluate the significance of different variables)
- **Predict water chemistry for different segregated materials.** This evaluation will be based on leaching tests but depending on the characteristics observed will also include the use of data obtained from the waste rock and lean ore characterization programs.

7 References

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Thorleifson, L. H., K. L. Harris, H. C. Hobbs, C. E. Jennings, A. R. Knaeble, R. S. Lively, B. A. Lusardi, G. N. Meyer, 2007, Till geochemical and indicator mineral reconnaissance of Minnesota. Minnesota Geological Survey Open File Report OFR-07-01, 512 p., 15 pdf digital files, 5 digital images.

Attachment 2-7

SRK Consulting. Analysis of Samples from Overburden Drilling Program – DRAFT, NorthMet Project. Memorandum to Stuart Arkley, MDNR. March 18, 2008.

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Memo

To:	Stuart Arkley, MDNR	**Date:**	March 18, 2008
cc:	John Borovsky, Barr Jim Scott, PolyMet	**From:**	Stephen Day Rich Patelke, PolyMet Nancy Dent, Barr
Subject:	Analysis of Samples from Overburden Drilling Program NorthMet Project – DRAFT	**Project #:**	1UP005.001

1 Introduction

The purpose of this memorandum is to provide the proposed analytical program for overburden samples collected from the NorthMet Project in the area of the proposed mine site. The design of the overburden characterization program was provided to the MDNR on February 22, 2008[1]. As requested in the characterization plan, the analytical proposal is provided for MDNR approval.

This memorandum is not intended to be an exhaustive description of the data obtained from the field program but instead forms the basis for selection of samples for chemical testing. Testing of samples for geotechnical purposes is described elsewhere.

It is acknowledged that additional testing of the samples may be warranted following completion of the proposed program, and additional sample collection may be needed to support permitting and operational monitoring.

2 Summary of Field Program

2.1 Drilling Results

Completed holes are compared to the proposed drilling program in Table 1. Of the sixteen holes proposed, fifteen were completed. Hole 2 was not drilled because access was not available. Except for two hand-drilled holes (15 and 16), an attempt was made to advance all holes to bedrock. The expected depth to bedrock was estimated prior to drilling using an overburden thickness map calculated based on the previous bedrock drilling programs. The similarity of expected depths and actual depths achieved, and lithology expected compared to observed indicated that the end of most of the holes was in bedrock rather than float. Hole 11 was considerably longer than expected and encountered bedrock at 33' compared to the expected depth of 15'.

A total of 225' of drilling was completed for geochemical characterization of overburden.

[1] SRK Consulting, PolyMet Mining and Barr Engineering. 2008. Overburden Geochemical Characterization Plan in Support of EIS – DRAFT NorthMet Project. February 22, 2008.

Table 1. Comparison of Proposed and Actual Drilling Program

DH	Location	Planned Depth	Actual Depth	Local Bedrock	Bedrock Encountered
		Feet	Feet		
1	As planned	23	20.5	Unit 2 and Higher Duluth Complex	Troctolite
2	Not drilled	13	-	Unit 2 and Higher Duluth Complex	-
3	As planned	25?	22	Unit 2 and Higher Duluth Complex	Troctolite
4	As planned	12	26	Unit 1 Duluth Complex	Sulfidic Troctolite
5	As planned	16	13	Unit 2 and Higher Duluth Complex	Troctolite
6	As planned	23	21	Unit 2 and Higher Duluth Complex	Troctolite
7	As planned	16	14.5	Virginia Formation	Graphitic Argillite
8	As planned	18	11	Unit 2 and Higher Duluth Complex	Troctolite
9	As planned	17	8	Unit 1 Duluth Complex	Troctolite
10	As planned	17	16	Unit 2 and Higher Duluth Complex	Troctolite
11	As planned	15	33	Unit 2 and Higher Duluth Complex	Troctolite
12	As planned	15	22	Unit 1 Duluth Complex	Sulfidic Troctolite
13	Located about 1000' west of planned	14	10	Unit 1 Duluth Complex	Sulfidic Troctolite
14	As planned	7	5	Virginia Formation	Argillite
15*	As planned	-	0.5	Virginia Formation	-
16*	As planned	-	2	Virginia Formation	-

2.2 Field Observations

Field observations, including visual observations and chemical measurements were obtained as described in the characterization plan. Drill hole logs are attached in Appendix A. Table 2 provides an interpreted summary of the stratigraphy.

As expected, drift in the area has complex lithology. The majority of intervals (75%) were characterized as dominantly sandy till with varying quantities of gravels and silts. A few intervals were dominated by gravels (21'). Dominantly silt intercepts were unusual (two intervals totaling 5'). The total intersection of peaty materials was 25'.

The main feature of the overburden profile was the presence of oxidized (brown) and unoxidized (olive and grey) tills corresponding roughly to the presence of the water table. Of the thirteen mechanically-drilled holes, only two were in a fully unsaturated profile (holes 10 and 14) while the others were either completely saturated (four holes) or were unsaturated near surface (seven holes). Measurements of oxidation reduction potential (ORP) showed a strong negative correlation with depth (Figure 1). Near surface samples had typical ORPs of 100 to 300 mV, whereas deeper samples had ORPs below 100 mV and as low as -200 mV. Loggers recorded the presence of what appeared to be secondary iron sulfides in the chemically-reduced overburden. Visual observations were supported by the evolution of hydrogen sulfide gas when 10% hydrochloric acid was applied.

Surface tills appeared to be weakly acidic (pH less than 6.5) as shown by the correlation of rinse pH with depth. Deeper tills had pHs greater than 6. The presence of acidic conditions generally did not correlate with conductivity (Figure 2) indicating that the variation of pH was not significantly related to the presence of acidic salts as would be produced by oxidation of sulfide minerals. In fact, conductivities for samples showing pHs less than 5.5 (the typical pH of deionized water) were mostly low. The exceptions were two samples with conductivity above 100 µS/cm and pHs below 5.6. These measurements did not correspond to the presence of mineralized rock.

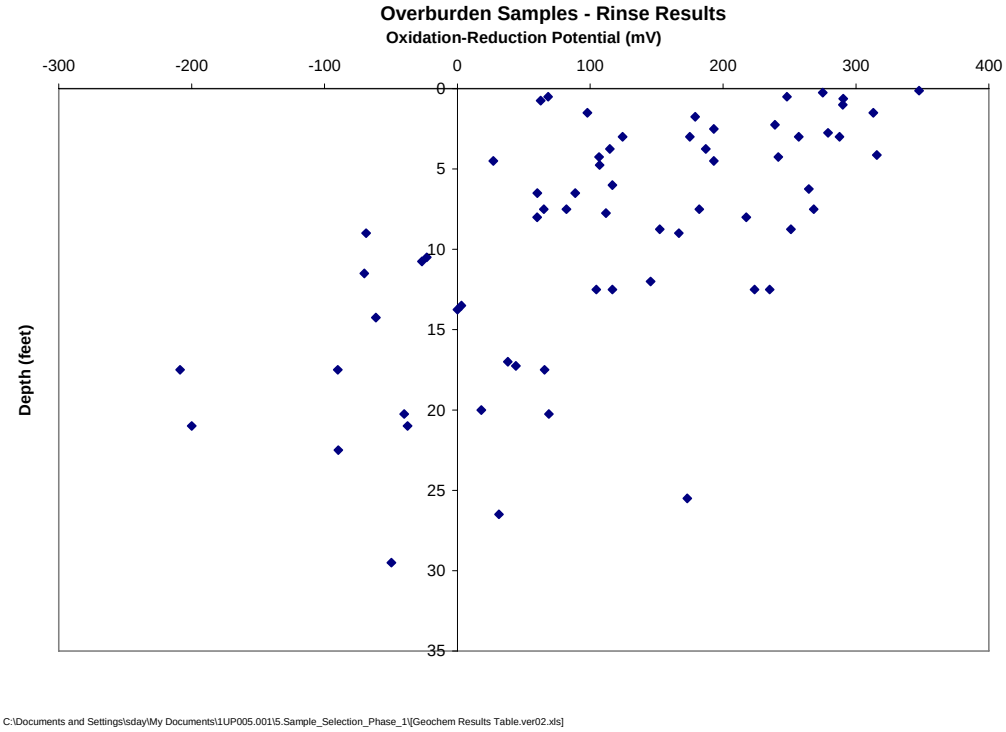

Figure 1. Variation of ORP with Depth

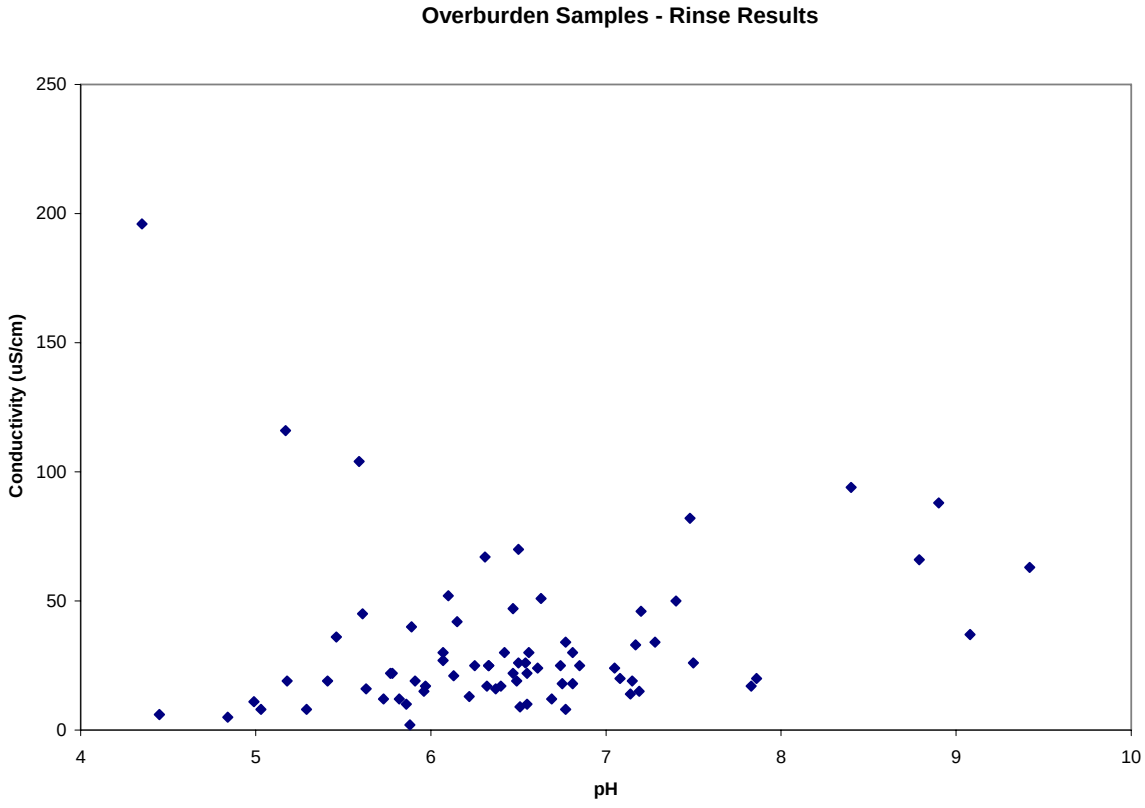

Figure 2. Comparison of Rinse pH and Conductivity

Other than brown coatings related to weathering of iron-bearing components of the overburden chemical precipitates were uncommon. White cement and lenses were observed in drill hole 10, but they did not react with dilute hydrochloric acid.

The overburden rarely reacted with hydrochloric acid which indicated low concentrations of carbonate minerals.

3 Sample Selection and Analysis

Samples were selected primarily for an element scan (including sulfur) on the basis that the distribution of components such as sulfur, copper and nickel will indicate the presence of the rock components derived from the Duluth Complex and Virginia Formation in the overburden. The majority of overburden layers have been selected with some compositing within layers to result in intervals of about 5 feet. The sample selection has resulted in 29 samples from 73% of the core obtained.

As described in the characterization plan, each sample will be separated into particle sizes. The coarse fraction (>2 mm) will be examined to determine if the number of identifiable pebbles is sufficient to warrant lithological pebble counts. If less than 200 pebbles are present, this step will not be performed.

In addition to the element scan on each particle fraction, the intermediate fraction (-2+0.074 mm) will be analyzed for acid-base account (ABA). No ABAs will be performed on peat samples.

Mobile metal content will be evaluated using the Meteoric Water Mobility Procedure. Factors expected to control metal leaching are pH and the presence of primary and secondary sulfide minerals. The basis for sample selection for the MWMP was as follows:

- One sample from each hole containing mineralized rock (2 samples).
- One sample from each hole containing secondary sulfides (6 samples).
- Unsaturated sandy sample with lowest and highest pH (2 samples).
- Saturated sandy sample with lowest and highest pH (2 samples).
- Peat with lowest and highest pH (2 samples).

As described in the characterization plan, additional samples may be selected for the MWMP based on the results obtained.

The sample analysis list is provided in Attachment B.

Table 2. Summarized Stratigraphy

Hole	From	To	Length	Description
RS-01B	0	1	1	PEAT
RS-01B	1	20	19	UPPER TILL
RS-01B	20	20.5	0.5	LOWER TILL
RS-01B	20.5			BEDROCK
RS-03	0	10	10	PEAT
RS-03	10	20	10	UPPER TILL
RS-03	20	22	2	LOWER TILL
RS-03	22			BEDROCK-TROCTOLITE
RS-04	0	1	1	PEAT
RS-04	1	5	4	SOIL
RS-04	5	18	13	UPPER TILL
RS-04	18	25	7	LOWER TILL
RS-04	25	26	1+	BEDROCK-TROCTOLITE W/ SULFIDES
RS-05A	0	13	13	UPPER TILL
RS-05A	13			BEDROCK-TROCTOLITE
RS-05B	0	5	5	UPPER TILL
RS-06A	0	2	2	SOIL
RS-06A	2	21	19	UPPER TILL
RS-06A	21			BEDROCK
RS-06R	0	2	2	PEAT
RS-06R	2	21	19	UPPER TILL
RS-06R	21	21.5	.5+	BEDROCK
RS-07	0	1	1	PEAT
RS-07	1	3	2	SOIL
RS-07	3	10	7	UPPER TILL
RS-07	10	11	1	LOWER TILL
RS-07	11			BEDROCK
RS-07R	0	1	1	PEAT
RS-07R	1	3	2	SOIL
RS-07R	3	9.5	6.5	UPPER TILL
RS-07R	9.5	11	1.5	LOWER TILL
RS-07R	11	14.5	3.5+	BEDROCK
RS-08	0	1	1	SOIL
RS-08	1	11	10	UPPER TILL
RS-08	11			BEDROCK
RS-09	0	1	1	SOIL
RS-09	1	7	6	UPPER TILL
RS-09	7	8	1	LOWER TILL
RS-09	8			BEDROCK-TROCTOLITE
RS-10	0	1	1	SOIL
RS-10	1	14	13	UPPER TILL
RS-10	14	16	2+	BEDROCK
RS-11	0	9.5	9.5	PEAT
RS-11	9.5	17	7.5	UPPER TILL
RS-11	17	28	11	OUTWASH
RS-11	28	33	5	LOWER TILL
RS-11	33			BEDROCK
RS-12	0	2	2	SOIL
RS-12	2	5.5	3.5	OUTWASH
RS-12	5.5	19.5	14	UPPER TILL
RS-12	19.5	20.5	1	OUTWASH
RS-12	19.5	22	2.5	LOWER TILL
RS-12	22			BEDROCK
RS-13	0	1.5	1.5	SOIL
RS-13	1.5	8	6.5	LOWER TILL
RS-13	8	10	2+	BEDROCK
RS-14A	0	1.5	1.5	SOIL
RS-14A	1.5	5	3.5	UPPER TILL
RS-14A	5			BEDROCK
RS-14B	0	1.5	1.5	SOIL
RS-14B	1.5	5	3.5	UPPER TILL
RS-14B	5			BEDROCK-BLACK BIOTITE ARGILLITE
RS-15A-E	0	0.25	0.25	PEAT
RS-15A-E	0.25	0.5	0.25	SOIL-REFUSAL 1.5
RS-16A-C	0	2	2	SOIL-REFUSAL 2

Attachment A
Drill Hole Logs

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/15/08 | Ended 1/15/08

Location NorthMet Mine Site | Logged By MMB/REE

LOG OF Boring RS-01B

DRAFT

SHEET 1 OF 3

Elevation 1613.0

Total Depth 20.5

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH-ORP-Specific Cond.	%GR/SA/FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
			7.05 248.1 24		Frozen	10YR 2/1 Black	PT		Peat	Fibrous Peat; 90-100% organic matter, mostly woody material. Up to 10% mineral soil.	
											1612
2		None	5.89 256.9 10	10/75/15 (Visual)	Dry to Moist	10YR 4/4 Dark Yellowish Brown	SM			Silty sand, homogeneous, very fine- to fine-grained, angular to subrounded, fine to coarse gravel. Sand fraction is 80% quartz, 15% lithics, and 5% feldspars. Cobbles are 80% granitic rock, 15% black fine-grained metasediment (Virginia Formation?), and 5% other (foliated gneiss).	
4											1610
											1608
6		None	6.55 268.1 10	15/75/10 (Visual)	Moist	10YR 3/2 Very Dark Grayish Brown	SP-SM		Upper Till	Sand with silt and gravel, homogeneous, medium dense, fine- to medium-grained, gravel is fine- to coarse-grained, angular to subrounded. Cobbles are 70% granitoids, 20% black fine-grained metasediment, and trace schist. Rust-colored coatings along fractures and cobble interfaces, dark red brown (7.5YR 3/4). Less than 2% dendritic or irregular mottles, fine to medium size - dark reddish brown (5YR 3/4).	
											1606
8			5.97 258.0 17							9-10': 10% dark red (2.5YR 3/6) mottles associated with tiny fractures within matrix.	1604
										(continued)	

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 0-1', 1-5', 6-7', 14-15', 18-20', 20-20.5'; Geotechnical samples: 0-1', 1-5', 5-10', 10-15', 12.5-15', 15-17.5', 18-20', 20-20.5'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 3/10/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/15/08 | Ended 1/15/08

Location NorthMet Mine Site | Logged By MMB/REE

LOG OF Boring RS-01B

DRAFT

Elevation 1613.0

Total Depth 20.5

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
				15/65/20 (Visual)	Very Moist					Silty sand with gravel, homogeneous, medium dense, fine- to medium-grained, gravel is fine- to coarse-grained, angular to subrounded. Cobbles are 70% granitoids, 20% black fine-grained metasediment, and trace schist. Rust-colored coatings along fractures and cobble interfaces, dark red brown (7.5YR 3/4). Less than 2% dendritic or irregular mottles, fine to medium size - dark reddish brown (5YR 3/4).	1602
12		None			Wet	2.5Y 4/3 Olive Brown					1600
14			6.37 223.7 16	25/60/15 (Visual)			SM		Upper Till	Abundant dark red (2.5YR 3/6) staining on coarse clasts. Brownish yellow (10YR 6/6) weathering or precipitate along fractures of black, fine-grained metasediment clasts.	1598
16											1596
18		None	7.28 65.6 34								1594
										(continued)	

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POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 3/10/08

LOG OF Boring RS-01B

DRAFT

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/15/08 | Ended 1/15/08 | Elevation 1613.0

Location NorthMet Mine Site | Logged By MMB/REE | Total Depth 20.5

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH-ORP-Specific Cond.	%GR/SA/FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
			8.79 -40.0 66	10/50/40 (Visual)	Moist	Gley1 3/10Y Very Dark Greenish Gray	SM		Lower Till	Silty sand, homogeneous, dense, very fine- to fine-grained sand. Gravel is fine- to coarse-grained, angular to subrounded. Cobbles are black, fine-grained metasediment and granitoid. Olive brown (2.5Y 4/3) color at bottom of borehole, irregular contact with above. Bedrock at 20.5'. End of Boring - 20.5 feet	1592
22											1590
24											1588
26											1586
28											1584

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POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 3/10/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/16/08 | Ended 1/16/08 | Elevation 1595.5

Location NorthMet Mine Site | Logged By REE/JAM2 | Total Depth 22.0

LOG OF Boring RS-03

DRAFT

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
0					Very Moist	2.5YR 2.5/1 Reddish Black				Fibrous peat; wood and other organic material. Note: Low recovery	1595.5
2											1594
4											1592
4					Wet	10YR 2/1 Black	PT		Peat	Fibrous and amorphous peat, composed of primarily muddy material with trace leaf and woody organic material.	
6			5.17 65 116								1590
8											1588
											1586
										(continued)	

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Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 3/10/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/16/08 | Ended 1/16/08 | Elevation 1595.5

Location NorthMet Mine Site | Logged By REE/JAM2 | Total Depth 22.0

LOG OF Boring RS-03

DRAFT

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
				10/5/85 (Visual)	Wet					Silt with gravel, loose, homogeneous, up to 5% organic matter from 10-12'. Sand is fine- to medium-grained, gravel is fine-grained, subangular to subrounded. Cobbles are black, fine-grained metasediment and troctolite.	1584
12			5.46 3 36			Gley1 5/10Y Greenish Gray	ML			12-15': No organic matter, increased gravel and sand, cobbles as above.	
				15/10/75 (Visual)	Wet						1582
14											
				15/45/40 (Visual)					Upper Till	Silty sand with gravel, homogeneous, loose, fine-grained, gravel is fine- to coarse-grained, subangular to subrounded. Cobbles are as above, also some magnetic cherty iron formation, and one pyrite-bearing rock (possibly greenstone).	
			7.4 -208.7 50								1580
16											
				15/40/35 (Visual)	Wet		SM				1578
18			9.08 -27 37								
											1576
										(continued)	

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Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/16/08 | Ended 1/16/08

Location NorthMet Mine Site | Logged By REE/JAM2

LOG OF Boring RS-03

DRAFT

Elevation 1595.5

Total Depth 22.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
			9.42 -200 63	40/10/50 (Visual)	Moist	Gley1 2.5/10Y Greenish Black	ML		Lower Till	Gravelly silt, homogenous, gravel is fine- to coarse-grained, subangular to subrounded. Cobbles are magnetic cherty iron formation, granitoid.	1574
22										Bedrock at 22.0', troctolite. End of Boring - 22 feet	
											1572
24											
											1570
26											
											1568
28											
											1566

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Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/16/08 | Ended 1/18/08 | Elevation 1600.0

Location NorthMet Mine Site | Logged By REE/JAM2 | Total Depth 26.0

LOG OF Boring RS-04

DRAFT

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
				95% organics	Wet	10YR 2/2 Very Dark Brown	PT		Peat	Fibrous peat, composed primarily of woody material with some fine-grained organic material.	
2		None	5.71 124.3 22	30/30/40 (Visual)	Wet	2.5Y 3/3 Dark Olive Brown	SM		Soil	Silty sand with gravel, homogeneous, up to 10% organic material, sand is fine- to coarse-grained, gravel is subangular to subrounded. Matrix has dark reddish brown (2.5YR 3/4) mottles.	1598
4											1596
6										Silty sand with gravel, homogeneous, fine- to coarse-grained. Gravel is fine- to coarse-grained. Cobbles are fine-grained black metasediment, magnetic cherty iron formation, and granitoid.	1594
8		None	5.91 82 19	30/50/20 (Visual)	Wet	10YR 4/3 Brown	SM		Upper Till		1592
										(continued)	

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Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 3/10/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/16/08 | Ended 1/18/08

Location NorthMet Mine Site | Logged By REE/JAM2

LOG OF Boring RS-04

DRAFT

Elevation 1600.0

Total Depth 26.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
12				30/50/20 (Visual)	Wet		SM			Silty sand with gravel, homogeneous, same as the 5-10' interval.	1588
14		None	6.33 104.5 25			Transitional Mottling			Upper Till	13-15': Gradational change in color and texture to 15-20' interval.	1586
16		None	6.74 -90 25							Silty sand with gravel, homogeneous, fine- to coarse-grained. Gravel is fine- to coarse-grained, subangular to subrounded. Cobbles as above.	1584
18		None	6.85 -81.6 25	25/55/20 (Visual)	Wet	10YR 3/1 Very Dark Gray	SM		Lower Till	19-20': Matrix contains possible sulfide flakes or secondary mineralization. 20': Several troctolite cobbles with sulfide minerals.	1582

(continued)

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 0-1', 1-5', 5-10', 10-15', 15-20', 20-25', 25-26'; Geotechnical samples: 1-5', 5-10', 10-15', 15-20', 20-25'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 3/10/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/16/08 | Ended 1/18/08

Location NorthMet Mine Site | Logged By REE/JAM2

LOG OF Boring RS-04

DRAFT

SHEET 3 OF 3

Elevation 1600.0

Total Depth 26.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH-ORP-Specific Cond.	%GR/SA/FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
										Silty sand with gravel, homogeneous, fine- to coarse-grained. Gravel is fine- to coarse-grained, subangular to subrounded. Matrix has possible secondary sulfide mineralization. Cobbles are sulfide-bearing troctolite, fine-grained black metasediment, magnetic cherty iron formation, and granitoid.	
22		None	7.83 -87.6 17	30/50/20 (Visual)	Wet	10YR 3/1 Very Dark Gray	SM		Lower Till		1578
24		None	8.10 173.0 94	70/20/10 (Visual)	Dry	Gley1 2.5/N Black to Gley1 6/1 Greenish Gray	GP-GM			Gravel with silt and sand, fine- to coarse-grained. Cobbles are as above.	1576
									Bed-rock	Bedrock at 25'. Sulfide-bearing troctolite.	
26										End of Boring - 26 feet	1574
28											1572

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 0-1', 1-5', 5-10', 10-15', 15-20', 20-25', 25-26'; Geotechnical samples: 1-5', 5-10', 10-15', 15-20', 20-25'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 3/10/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/18/08 | Ended 1/18/08

Location NorthMet Mine Site | Logged By REE

LOG OF Boring RS-05A

DRAFT

SHEET 1 OF 2

Elevation 1605.0

Total Depth 13.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
										Low recovery on RS-05A for 0-5'. See R5-05B log for description.	
		None	6.42 124.5 30		Moist	7.5YR 3/3 Dark Brown					1604
2							SM				1602
4									Upper Till		1600
6		None	6.55 88.7 22	20/60/20 (Visual)	Moist	10YR 3/4 Dark Yellow Brown	SM			Silty sand with gravel, homogeneous, fine- to coarse-grained. Gravel is fine- to medium- grained, subangular to subrounded. Up to 1% organic matter. Cobbles are 60% granitoid, 30% black fine-grained metasediment, 5% cherty iron formation, and trace greenstone. Rust-colored staining on some clast surfaces.	1598
8		None	6.49 166.6 19	40/40/20 (Visual)	Moist	2.5Y 4/2 Dark Gray Brown	SM			Silty sand with gravel, transitional color change with above. Cobbles are same lithologies as above.	1596
										(continued)	

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 5-10', 10-13'; Geotechnical samples: 0-1', 5-6', 6-11.5', 10-11.5', 11.5-13'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 3/10/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/18/08 | Ended 1/18/08 | Elevation 1605.0

Location NorthMet Mine Site | Logged By REE | Total Depth 13.0

LOG OF Boring RS-05A

DRAFT

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
						2.5Y 4/2 Dark Gray Brown	GP-GM			Gravel with silt and sand, fine- to coarse-grained, subangular to subrounded. Cobbles are 60% troctolite, 30% granitoid, 5% magnetic cherty iron formation with rust-colored staining, and 5% black fine-grained metasediment with rust-colored staining.	1594
				70/20/10 (Visual)	Wet				Upper Till	As above, increased clay content, gray.	
12		None	8.9 -70 88			2.5Y 5/1 Gray	GP-GM				1592
										Bedrock at 13.0', troctolite. End of Boring - 13 feet	
14											1590
16											1588
18											1586

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 5-10', 10-13'; Geotechnical samples: 0-1', 5-6', 6-11.5', 10-11.5', 11.5-13'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 3/10/08

Client PolyMet Mining Corporation
Project Name Polymet Overburden Characterization
Number 23/69-B75 INV
Location NorthMet Mine Site

Drill Contractor Boart Longyear
Drill Method Rotasonic
Drilling Started 1/18/08 Ended 1/18/08
Logged By REE

LOG OF Boring RS-05B

DRAFT

SHEET 1 OF 1

Elevation 1605.0
Total Depth 5.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
				30/50/20 (Visual)		10YR 4/4 Dark Yellowish Brown				Silty sand with gravel, homogeneous, fine- to coarse-grained. Gravel is fine- to coarse-grained, angular to subrounded. Cobbles are 50% granitoid, 30% fine-grained, black metasediment, 20% magnetic cherty iron formation, and trace greenstone or silica rocks (possible Archean).	1604
2		None	6.13 179.0 21								
				30/50/20 (Visual)	Moist		SM		Upper Till		1602
		None	6.54 187.0 26			10YR 4/2 Dark Grayish Brown				3.5-4': Lens of dark grayish brown silty sand with gravel.	
4		None	6.25 193.0 25			SA 1-3.5'					
										End of Boring - 5 feet	1600
6											1598
8											1596

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 1-5'; Geotechnical samples: 1-3.5', 3.5', 3.5-4'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/26/08 Ended 1/26/08

Location NorthMet Mine Site | Logged By MMB/MJD/REE

LOG OF Boring RS-06A

DRAFT

SHEET 1 OF 3

Elevation 1611.0

Total Depth 21.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
		None	4.45 290.3 6	10/50/40 (Visual)		10YR 4/4 Dark Yellowish Brown	SM		Soil	Silty sand, up to 20% organic matter, homogeneous, sand is fine- to coarse-grained, gravel is fine- to coarse- grained, subrounded to subangular. Matrix is magnetic. Sand fraction is 70% quartz, 10% feldspar, and 20% white fragments. Cobbles are 75% black fine-grained metasediment, 20% magnetic iron formation, and 5% granitoid.	1610
		None		5/65/30 (Visual)		7.5YR 3/2 Dark Brown	SM			Silty sand, up to 30-40% organic matter, homogeneous, sand is fine- to coarse-grained. Matrix has dark-brown to black organic masses and lenses. Sand fraction is 40% quartz, 50% feldspar, and 10% lithic fragments. Cobbles are 90% granitoid, 5% fine-grained black metasediment, and 5% magnetic iron formation.	
2			4.84 313.0 5		Moist						
		None	4.99 279 11	20/65/15 (Visual)		7.5YR 3/4 Dark Brown	SM			Silty sand with gravel, homogeneous, sand is fine- to medium-grained, gravel is fine- to coarse grained. Matrix has less than 5% mottles, black (5YR 2.5/1) and yellowish red (5YR 4/6), and is magnetic. Sand fraction is 50% quartz, 40% feldspar, and 10% lithic fragments. Cobbles are 70% granitoid, 30% gabbroic (or possibly recrystallized metasediment) - abundant, rust staining. Large granitoid boulder from 3.5-4.5'.	1608
4					Dry						
			5.03 316 8							Clay with sand, firm, laminated, sand is fine- to medium-grained, gravel is fine- to medium-grained. Matrix is magnetic and has abundant mottles (30-40%), dark yellowish gray (10YR 4/6) and grayish brown (2.5YR 5/2). Sand fraction is 70% quartz, 20% feldspar, and 10% lithic fragments. Cobbles are 80% magnetic chert iron formation, 10% granitoid, and 10% fine-grained black metasediment.	1606
6		None	5.82 264 12	5/20/75 (Visual)			CL		Upper Till		1604
						10YR 4/3 Brown					
8			6.32 251 17	15/70/15 (Visual)			SM			Silty sand with gravel, dense, homogeneous, sand is fine- to coarse-grained, gravel is fine- to coarse-grained. Matrix is slightly magnetic, has less than 5% disseminated mottles, very dark gray (10YR 3/1), dark brown (7.5YR 3/4), dark yellowish brown (10YR 4/6), and black mottles associated with rootlets. Increased mottles at 10-12'. Matrix has a faint rotten egg odor below 15', increasing odor with depth. Sand fraction lithology transition from 70% quartz, 10% feldspar, and 20% lithic fragments to 15% quartz, 65% feldspar, and 20% lithic fragments at 10'. Cobbles are 70% iron formation rocks (magnetic and non-magnetic), 25% granitoid, 5% other (troctolite, gabbroic).	1602

(continued)

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 0.5-2', 2-4', 5-7.5', 7.5-10', 10-15', 15-19', 19-21'; Geotechnical samples: 0-1', 1-2', 2-3.5', 3.5-7.5', 7.5-10', 10-15', 15-21'; Shelby tubes: 6-7', 15-16', 16-18'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/26/08 Ended 1/26/08

Location NorthMet Mine Site | Logged By MMB/MJD/REE

LOG OF Boring RS-06A

DRAFT

Elevation 1611.0

Total Depth 21.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
					Moist					Silty sand with gravel, dense, homogeneous, sand is fine- to coarse-grained, gravel is fine- to coarse-grained. Matrix is slightly magnetic, has less than 5% disseminated mottles, very dark gray (10YR 3/1), dark brown (7.5YR 3/4), dark yellowish brown (10YR 4/6), and black mottles associated with rootlets. Increased mottles at 10-12'. Matrix has a faint rotten egg odor below 15', increasing odor with depth. Sand fraction lithology transition from 70% quartz, 10% feldspar, and 20% lithic fragments to 15% quartz, 65% feldspar, and 20% lithic fragments at 10'. Cobbles are 70% iron formation rocks (magnetic and non-magnetic), 25% granitoid, 5% other (troctolite, gabbroic).*(continued)*	1600
12		None	6.81 235 17	15/65/20 (Visual)							1598
14					Moist to Wet	10YR 4/3 Brown	SM		Upper Till		1596
16											1594
18			6.75 38 18								1592
			7.86 18.0 20		Wet					*(continued)*	

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 0.5-2', 2-4', 5-7.5', 7.5-10', 10-15', 15-19', 19-21'; Geotechnical samples: 0-1', 1-2', 2-3.5', 3.5-7.5', 7.5-10', 10-15', 15-21'; Shelby tubes: 6-7', 15-16', 16-18'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation
Project Name Polymet Overburden Characterization
Number 23/69-B75 INV
Location NorthMet Mine Site

Drill Contractor Boart Longyear
Drill Method Rotasonic
Drilling Started 1/26/08 Ended 1/26/08
Logged By MMB/MJD/REE

LOG OF Boring RS-06A

DRAFT

Elevation 1611.0
Total Depth 21.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH-ORP-Specific Cond.	%GR/SA/FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
							SM		Upper Till		
										End of Boring - 21 feet	1590
22											1588
24											1586
26											1584
28											1582

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 0.5-2', 2-4', 5-7.5', 7.5-10', 10-15', 15-19', 19-21'; Geotechnical samples: 0-1', 1-2', 2-3.5', 3.5-7.5', 7.5-10', 10-15', 15-21'; Shelby tubes: 6-7', 15-16', 16-18'

Additional data may have been collected in the field which is not included on this log.

Client PolyMet Mining Corporation
Project Name Polymet Overburden Characterization
Number 23/69-B75 INV
Location NorthMet Mine Site

Drill Contractor Boart Longyear
Drill Method Rotasonic
Drilling Started 1/29/08 Ended 1/29/08
Logged By MMB

LOG OF Boring RS-06R

DRAFT

Elevation 1611.0
Total Depth 21.5

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH-ORP-Specific Cond.	%GR/SA/FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
							SM		Soil	See RS-06A, 0-1' for description.	
							SM			See RS-06A, 1-2' for description.	1610
2							SM			See RS-06A, 2-4.75' for description.	1608
4							CL			See RS-06A, 4.75-7.5' for description.	1606
6							SM		Upper Till	See RS-06A, 7.5-21.0' for description.	1604
8											1602

(continued)

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. See RS-06A log for sampling intervals.

Additional data may have been collected in the field which is not included on this log.

Client PolyMet Mining Corporation
Project Name Polymet Overburden Characterization
Number 23/69-B75 INV
Location NorthMet Mine Site

Drill Contractor Boart Longyear
Drill Method Rotasonic
Drilling Started 1/29/08 Ended 1/29/08
Logged By MMB

LOG OF Boring RS-06R

DRAFT

Elevation 1611.0
Total Depth 21.5

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH-ORP-Specific Cond.	%GR/SA/FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
										See RS-06A, 7.5-21.0' for description. *(continued)*	
12											1600
											1598
14											
							SM		Upper Till		1596
16											
											1594
18											
											1592
										(continued)	

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. See RS-06A log for sampling intervals.

Additional data may have been collected in the field which is not included on this log.

Client PolyMet Mining Corporation
Project Name Polymet Overburden Characterization
Number 23/69-B75 INV
Location NorthMet Mine Site

Drill Contractor Boart Longyear
Drill Method Rotasonic
Drilling Started 1/29/08 Ended 1/29/08
Logged By MMB

LOG OF Boring RS-06R

DRAFT

Elevation 1611.0
Total Depth 21.5

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH-ORP-Specific Cond.	%GR/SA/FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
							SM		Upper Till	See RS-06A, 7.5-21.0' for description.*(continued)*	
									Bed-rock	Bedrock at 21.0'. Troctolite piece, 4" thick.	1590
										End of Boring - 21.5 feet	
22											1588
24											1586
26											1584
28											1582

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. See RS-06A log for sampling intervals.

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/24/08 | Ended 1/24/08 | Elevation 1608.0

Location NorthMet Mine Site | Logged By MMB/MJD/REE | Total Depth 11.0

LOG OF Boring RS-07

DRAFT

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH-ORP-Specific Cond.	%GR/SA/FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
					Frozen	10YR 2/2 Very Dark Brown	PT		Peat	Fibrous peat; grass, roots, twigs.	
		None	5.61 97.8 45	5/65/30 (Visual)	Wet	10YR 2/2 Very Dark Brown	OL/OH		Soil	95% organic material (roots, grass, branches). Mineral component is silty sand. Less than 5% dark brown (10YR 3/3) mottles from 1.5-2'.	
2		None		5/85/10 (Visual)	Moist	2.5Y 3/3 Dark Olive Brown	SP-SM			Sand with silt, 5% organic material, sand is fine- to medium-grained. Less than 5% mottles and layers, dark brown (7.5YR 3/3).	1606
			6.10 27.0 52							Sand with silt and gravel, homogeneous, trace organic matter, sand is fine- to coarse-grained, gravel is fine- to coarse-grained, subrounded to subangular. Matrix is mottled: irregular, very dark brown (7.5YR 2/2) and minor strong brown (7.5YR 5/8) mottles. Sand fraction is 10% quartz, 10% feldspar, and 80% lithic fragments. Cobbles are 90% fine-grained black metasediment, 5% black cherty iron formation, and 5% granitoid.	
4				20/70/10 (Visual)		7.5YR 3/3 Dark Brown					1604
		None			Moist		SP-SM				
				30/60/10 (Visual)							
6											1602
			6.40 60.0 17						Upper Till	Sand with gravel, homogeneous, sand is fine- to coarse-grained, gravel is fine- to coarse-grained, subrounded to subangular. Sandier and slightly drier toward 10'. Sand fraction and cobble lithologies are same as 3-6' interval.	
		None		30/65/5 (Visual)		5Y 2.5/1 Black					
8					Moist		SP				1600
			6.61 38.0 24								
										(continued)	

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 1-2', 2-3', 3-5', 5-6', 6-10', 10-11'; Geotechnical samples: 0-2', 2-5', 8-10', 10-11'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 3/10/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/24/08 | Ended 1/24/08

Location NorthMet Mine Site | Logged By MMB/MJD/REE

LOG OF Boring RS-07

DRAFT

Elevation 1608.0

Total Depth 11.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH-ORP-Specific Cond.	%GR/SA/FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
		None	7.15 -23.0 19	50/30/20 (Visual)	Wet	Gley1 2.5/10Y Greenish Black	GM		Lower Till	Silty gravel with sand, homogeneous, sand is fine-grained, gravel is fine- to coarse-grained, angular to subrounded. Matrix has a rotten egg odor after HCL, and a very dark brown (10YR 2/2) layer from 10-10.25'. Sand fraction is 50% quartz, 10% feldspar, and 40% lithic fragments.	
										Bedrock at 11.0'. End of Boring - 11 feet	
12											1596
14											1594
16											1592
18											1590

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 1-2', 2-3', 3-5', 5-6', 6-10', 10-11'; Geotechnical samples: 0-2', 2-5', 8-10', 10-11'

Additional data may have been collected in the field which is not included on this log.

Client PolyMet Mining Corporation
Project Name Polymet Overburden Characterization
Number 23/69-B75 INV
Location NorthMet Mine Site

Drill Contractor Boart Longyear
Drill Method Rotasonic
Drilling Started 1/29/08 Ended 1/29/08
Logged By MMB

LOG OF Boring RS-07R

DRAFT

Elevation 1608.0
Total Depth 14.5

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH-ORP-Specific Cond.	%GR/SA/FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
							PT		Peat	See RS-07 for description.	
							OL/OH		Soil		
2							SP-SM				1606
							SP-SM				
4											1604
									Upper Till		
6											1602
							SP				
8											1600
									Lower Till		

(continued)

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 10-12', 13.5-14.5'; Geotechnical samples: 1-2', 2-3', 3-6', 6-10', 10-14.5'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation
Project Name Polymet Overburden Characterization
Number 23/69-B75 INV
Location NorthMet Mine Site

Drill Contractor Boart Longyear
Drill Method Rotasonic
Drilling Started 1/29/08 Ended 1/29/08
Logged By MMB

LOG OF Boring RS-07R

DRAFT

Elevation 1608.0
Total Depth 14.5

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
			6.63 -125 51						Lower Till	Possible fractured bedrock at 9.5' or boulders on bedrock. Soil in fractures. Sample is 0.5-4" thick core pieces of biotite argillite of Virginia formation. Rinse test at 14' has silver metallic sheen (floating graphite from graphite-bearing Virginia formation rocks?).*(continued)*	
12									Bed-rock		1596
14			7.48 -152 82								1594
										End of Boring - 14.5 feet	
16											1592
18											1590

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 10-12', 13.5-14.5'; Geotechnical samples: 1-2', 2-3', 3-6', 6-10', 10-14.5'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/26/08 Ended 1/26/08

Location NorthMet Mine Site | Logged By MMB/MJD

LOG OF Boring RS-08A

DRAFT

SHEET 1 OF 2

Elevation 1591.0

Total Depth 11.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH-ORP-Specific Cond.	%GR/SA/FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
		None	4.35 347.5 196	15/55/30 (Visual)	Moist	7.5YR 3/4 Dark Brown	SM		Soil	Silty sand with gravel, with up to 20% organic material, homogeneous, dense, sand is fine-grained, gravel is fine-grained, subangular to subrounded. Matrix has 2-5% dark reddish brown (2.5YR 3/4) mottles associated with disseminated rootlets and pebbles. Also less than 1% gray (5YR 5/1) mottles and layer at 1'. Sand fraction is 65% quartz, 10% feldspar, and 15% lithic fragments. Cobbles are fine-grained black metasediment, black chert/iron formation, less than 5% green-black crystalline rock with quartz veins (possibly Archean).	1590
2		None	5.18 287.6 19	20/60/20 (Visual)	Wet to Moist	10YR 4/6 Dark Yellowish Brown to 2.5Y 3/3 Dark Olive Brown	SM			Silty sand with gravel, homogeneous, loose, sand is fine- to medium-grained, gravel is fine- to coarse-grained, subrounded to subangular. Occasional lenses with up to 40% clay (low plasticity). Matrix is magnetic, has mottles as above, also 30% strong brown (7.5YR 5/8) irregular to wavy mottles from 3-4'. Sand fraction is 70% quartz, 10% feldspar, and 20% lithic fragments. Cobbles are fine-grained black metasediment, fine-grained magnetic and non-magnetic cherty iron formation with rust coatings.	1588
4			5.63 262.4 16								1586
6			5.78 217.4 22				SM		Upper Till	Silty sand with gravel, homogeneous, dense, sand is fine- to coarse-grained, gravel is fine- to coarse-grained, angular to subangular. Matrix has a faint rotten egg odor after HCL, 1-2% yellowish red (5YR 4/6) mottles. Sand fraction is 75% quartz, 5% feldspar, and 20% lithic fragments. Cobbles are 40% magnetic black iron formation, 30% fine-grained black metasediment, 25% non-magnetic black iron formation, and 5% granitoid.	1584
8		None		30/50/20 (Visual)	Moist	10YR 4/2 Dark Grayish Brown					1582

(continued)

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Fax: 952-832-2601

Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 0.25-1', 1-5', 5-11'; Geotechnical samples: 1-5', 5-11'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation
Project Name Polymet Overburden Characterization
Number 23/69-B75 INV
Location NorthMet Mine Site

Drill Contractor Boart Longyear
Drill Method Rotasonic
Drilling Started 1/26/08 Ended 1/26/08
Logged By MMB/MJD

LOG OF Boring RS-08A

DRAFT

Elevation 1591.0
Total Depth 11.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH-ORP-Specific Cond.	%GR/SA/FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
			6.77 68.3 34				SM		Upper Till		
										Bedrock at 11'. Troctolite, no visible sulfides. End of Boring - 11 feet	1580
12											1578
14											1576
16											1574
18											1572

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 0.25-1', 1-5', 5-11'; Geotechnical samples: 1-5', 5-11'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/23/08 Ended 1/23/08

Location NorthMet Mine Site | Logged By REE/MJD

LOG OF Boring RS-09

DRAFT

SHEET 1 OF 1

Elevation 1610.5

Total Depth 8.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
		None		5/15/80 (Visual)	Frozen	7.5YR 2.5/3 Very Dark Brown	OL/OH		Soil	Silt with sand, homogeneous, sand is fine-grained. Organic content decreases from 75% to 50%. Some grayish mottles and black (7.5YR 2.5/1) lenses, matrix is magnetic. Sand faction is 50% quartz, 30% feldspar, and 20% lithic fragments. Cobbles are 80% fine-grained black metasediment and 20% granitoid. Abundant rust-colored staining on clasts.	1610
2										Sand with silt and gravel, homogeneous, sand is fine-grained, subangular to subrounded, gravel is fine- to coarse-grained, subangular to subrounded. Color change is gradational. Matrix is magnetic. Sand fraction is 50% quartz, 25% feldspars, and 25% lithic fragments. Cobbles are 60% fine-grained black metasediment, 20% magnetic black siltstone, 5-10% medium-grained bedded/foliated metasediment, 10% granitoid, and 5% biotite argillite. One cobble has orange precipitate or oxidation along microfractures. Increased granitoid cobbles from 5 to 7'. Occasional rust colored staining on clasts.	1608
		None	5.96 175.0 15								
4				20/70/10 (Visual)	Dry to Moist	10YR 4/4 Dark Yellowish Brown to 2.5Y 4/4 Olive Brown	SP-SM		Upper Till		1606
6		None	6.22 116.7 13								1604
			5.88 182.0 2	15/20/65 (Visual)	Wet	2.5Y 3/1 Very Dark Gray	CL		Lower Till	Sandy lean clay with gravel, homogeneous, soft, sand is fine-grained, gravel is fine-grained. Matrix is magnetic, has faint rotten egg odor after HCL. Sand fraction is 70% quartz, 10% feldspars, and 20% lithic material. Cobbles are 75% granitoid, 20% fine-grained black metasediment with rust-colored staining on some surfaces, and 5% banded red and black iron formation.	
8										Bedrock at 8'. Troctolite, no visible sulfides. End of Boring - 8 feet	1602

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 0-1', 2-5', 5-7', 7-8'; Geotechnical samples: 0-1', 1-7', 7-8'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/25/08 | Ended 1/25/08

Location NorthMet Mine Site | Logged By MMB/MJD/REE

LOG OF Boring RS-10

DRAFT

Elevation 1602.5

Total Depth 16.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH-ORP-Specific Cond.	%GR/SA/FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
		None			Frozen	7.5YR 2.5/2 Very Dark Brown	OL/OH		Soil	Organic soil with sand. 80% organic matter (grass, roots, branches). Mineral fraction is silty sand, laminated lenses [dark yellowish brown (10YR 3/6) and black (10YR 2/1)].	1602
		None		35/55/10 (Visual)	Moist	10YR 2/2 very Dark Grayish Brown	SP-SM			Sand with silt and gravel, homogeneous, fine- to medium-grained, gravel is fine- to coarse-grained, subrounded to subangular. Sand fraction is 40% quartz, 40% feldspar, and 20% lithic fragments. Cobbles are 70% granitoid, and 30% fine-grained black metasediment with rust-colored staining.	
2		None	6.07 193.0 30	25/60/15 (Visual)	Moist	10YR 3/6 Dark Yellowish Brown	SM			Silty sand with gravel, homogeneous, fine- to medium-grained, gravel is fine- to coarse-grained, angular to subangular. Matrix has mottles associated with break-down of pebbles [bluish black (gley2 2.5/5PB)]. Sand fraction is 20% quartz, 60% feldspar, and 20% lithic fragments. Cobbles are 30% granitoid and 70% black fine-grained metasediment.	1600
4		None	5.73 241.6 12	10/85/5 (Visual)	Moist	7.5YR 3/3 Dark Brown	SP		Upper Till	Sand, homogeneous, fine- to coarse-grained, trace angular to subangular pebbles and cobbles. Sand fraction is 40% quartz, 30% feldspar, and 30% lithic fragments. Cobbles are 95% fine-grained metasediment with possible trace pyrite or pyrrhotite, and 5% granitoid.	1598
6		None	7.08 60.2 20	20/75/5 (Visual)	Dry to Moist	10YR 4/3 Brown	SP			Sand with gravel, homogeneous, fine- to coarse-grained, with 20% fine- to medium-grained gravel, angular to subangular. Matrix is mottled with irregular yellowish red (5YR 4/6) and white (5YR 8/1) mottles. White mottles have no HCL reaction, but appear to be weakly cemented. Sand fraction is 85% quartz, 5% feldspar, and 10% lithic fragments. Cobbles are 95% black fine-grained metasediment and 5% magnetic cherty iron formation.	1596
8		None	6.81 152.3 30	40/40/20 (Visual)	Dry	5Y 3/1 Very Dark Gray	SM			Silty sand with gravel, homogeneous, fine- to medium-grained, gravel is fine- to coarse-grained, angular to subangular. Matrix has a faint odor after HCL. Sand fraction is 10% quartz, 20% feldspar, and 70% lithic fragments. Cobbles are 80% black fine-grained metasediment, 10% magnetic cherty iron formation, and 10% granitoid. Supernatant from 8.0' rinse test has metallic sheen/possible graphite from graphite-bearing Virginia formation rocks.	1594

(continued)

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 0-1', 1-2', 2-3', 3-5.5', 5.5-7.5', 7.5-10', 10-14'; Geotechnical samples: 2-3', 3.5-5', 5.5-7.5', 7.5-10', 10-14'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation
Project Name Polymet Overburden Characterization
Number 23/69-B75 INV
Location NorthMet Mine Site

Drill Contractor Boart Longyear
Drill Method Rotasonic
Drilling Started 1/25/08 Ended 1/25/08
Logged By MMB/MJD/REE

LOG OF Boring RS-10

DRAFT

Elevation 1602.5
Total Depth 16.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
										Sand with gravel, homogeneous, fine- to coarse-grained, gravel is fine- to coarse-grained, angular to subangular. Matrix has a few white lenses (precipitate?), no HCL reaction, no odor. Sand fraction is 10% quartz, 10% feldspar, and 80% lithic fragments. Cobbles are 65% black fine-grained metasediment, 20% augite troctolite with weathered brown minerals, 10% magnetic, black cherty iron formation with rust-colored staining, and 5% granitoid.	1592
12		None	6.50 145.3 26	40/45/5 (Visual)	Moist	5Y 4/3 Olive	SP		Upper Till		1590
14									Bed- rock	Bedrock at 14.0'. Crushed troctolite pieces.	1588
16										End of Boring - 16 feet	1586
18											1584

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 0-1', 1-2', 2-3', 3-5.5', 5.5-7.5', 7.5-10', 10-14'; Geotechnical samples: 2-3', 3.5-5', 5.5-7.5', 7.5-10', 10-14'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/25/08 Ended 1/25/08

Location NorthMet Mine Site | Logged By MMB/MJD

LOG OF Boring RS-11

DRAFT

Elevation 1594.0

Total Depth 33.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
0					Frozen					Fibrous peat (grass, roots, root material). Up to approximately 10% mineral soil below 5'.	1594
2											1592
4											1590
		None			Wet	5YR 2.5/1 Black	PT		Peat		
6											1588
			5.89 107.1 40								
8											1586
		None		60/30/10 (Visual)	Wet	10YR 3/2 Very Dark Grayish Brown	GP-GM		Upper Till	Gravel with silt and sand. Less than 5% organic matter, sand is fine- to coarse-grained, gravel is fine- to coarse grained. Sand fraction is 30% quartz, 10% feldspar, and 60% lithic fragments. *(continued)*	

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 0-9.5', 11.5-17', 17-25', 25-28', 28-31', 31-33'; Geotechnical samples: 9.5-10', 10-11.5', 17-25', 25-28', 28-31', 31-33'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/25/08 Ended 1/25/08

Location NorthMet Mine Site | Logged By MMB/MJD

LOG OF Boring RS-11

DRAFT

Elevation 1594.0

Total Depth 33.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
										Cobbles are 90% fine-grained black metasediment, and 10% biotite argillite.	
			6.31 -26.7 67	20/65/15 (Visual)		10YR 2/2 Black				Gradational change from silty sand with gravel to silty gravel with sand, sand is fine- to coarse-grained, gravel is fine- to medium-grained. Matrix has less than 5% organic material (black), and less than 5% reddish mottles (less than 1 mm in diameter) disseminated, and a faint rotten egg odor after HCL. Sand is 30% quartz, 5% feldspar, and 65% lithic fragments. Cobbles are 80-90% fine-grained black metasediment, 5-10% granitoid, and 5-10% biotite-containing anorthosite.	
12											1582
			6.47 -61.4 47	65/20/15 (Visual)	Wet	10YR 2/1 Black	SM to GM		Upper Till		
14		None									1580
			6.69 -44.1 12								
16											1578
										Gravel with sand, homogeneous, sand is fine- to coarse-grained, gravel is fine- to coarse-grained, subrounded to subangular. Matrix has a faint rotten egg odor after HCL. Sand fraction is 60% quartz, 5% feldspar, and 35% lithic fragments. Cobbles are 80-90% fine-grained black metasediment, 5-10% granitoid, and 5-10% chert (possible Archean rocks).	
18		None	6.56 -37.5 30	50/45/5 (Visual)	Moist to Wet	Gley1 2.5/N Black	GP		Out- wash		1576

(continued)

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 0-9.5', 11.5-17', 17-25', 25-28', 28-31', 31-33'; Geotechnical samples: 9.5-10', 10-11.5', 17-25', 25-28', 28-31', 31-33'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/25/08 | Ended 1/25/08

Location NorthMet Mine Site | Logged By MMB/MJD

LOG OF Boring RS-11

DRAFT

Elevation 1594.0

Total Depth 33.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH-ORP-Specific Cond.	%GR/SA/FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
22		None	6.51 17.0 9		Moist to Wet	Gley1 2.5/N Black	GP		Out-wash	20-25': Same as 17-20' interval. Note low recovery.	1572
24											1570
				0/90/10 (Visual)						Gradational change downward: sand with silt to sand with gravel. Sand is fine- to medium-grained, subrounded to subangular. Up to 2% organic matter in lower part of sample. Matrix has a faint rotten egg odor after HCL. Sand fraction is 50% quartz, 5% feldspar, and 45% lithic fragments. Cobbles are 85% fine-grained black metasediment, 10% magnetic cherty iron formation, and 5% granitoid.	
26		None	6.33 31.3 25	30/65/5 (Visual)	Wet	10YR 2/1 Black	SW-SM to SP				1568
28											1566
		None		30/60/10 (Visual)	Wet	Gley1 3/10Y Very Dark Greenish Gray	SP-SM		Lower Till	Sand with silt and gravel, homogeneous, sand is medium-grained, gravel is fine- to medium-grained. Matrix has a faint rotten egg odor after HCL. Sand fraction is 60% quartz, 10% feldspar, and 30% lithic fragments. Cobbles are 70% fine-grained black metasediment, 20% granitoid, and 10% other.	

(continued)

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 0-9.5', 11.5-17', 17-25', 25-28', 28-31', 31-33'; Geotechnical samples: 9.5-10', 10-11.5', 17-25', 25-28', 28-31', 31-33'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation
Project Name Polymet Overburden Characterization
Number 23/69-B75 INV
Location NorthMet Mine Site

Drill Contractor Boart Longyear
Drill Method Rotasonic
Drilling Started 1/25/08 Ended 1/25/08
Logged By MMB/MJD

LOG OF Boring RS-11

DRAFT

Elevation 1594.0
Total Depth 33.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
			6.50 -49.7 70				SP-SM			Sand with silt and gravel, homogeneous, sand is medium-grained, gravel is fine- to medium-grained. Matrix has a faint rotten egg odor after HCL. Sand fraction is 60% quartz, 10% feldspar, and 30% lithic fragments. Cobbles are 70% fine-grained black metasediment, 20% granitoid, and 10% other.*(continued)*	
32		None		15/80/5 (Visual)	Wet	Gley1 3/10Y Very Dark Greenish Gray	SP		Lower Till	Sand with gravel, homogeneous, fine- to coarse-grained, gravel is fine- to medium-grained. Cobbles are 65% fine-grained black metasediment, 30% granitoid, and 5% gabbroic (no visible sulfides).	1562
										Bedrock at 33.0'. End of Boring - 33 feet	
34											1560
36											1558
38											1556

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 0-9.5', 11.5-17', 17-25', 25-28', 28-31', 31-33'; Geotechnical samples: 9.5-10', 10-11.5', 17-25', 25-28', 28-31', 31-33'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/23/08 | Ended 1/23/08 | Elevation 1610.0

Location NorthMet Mine Site | Logged By MMB/MJD | Total Depth 22.0

LOG OF Boring RS-12

DRAFT

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH-ORP-Specific Cond.	%GR/SA/FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
		None		2/30/68 (Visual)	Frozen	7.5YR 2.5/2 Very Dark Brown to 7.5YR 2.5/3 Very Dark Brown	ML		Soil	Sandy silt, homogeneous, sand is fine-grained. Decreasing organic material from 0-2'. Approximately 2% medium-grained charcoal pieces in soil. Several clay coatings, very dark gray (7.5YR 3/1), approximately 2 mm thick at 1.2'. Sand fraction is 70% quartz, 20% feldspar, and 10% lithic fragments.	
2											1608
			6.77 114.8 8							Sand, homogeneous, fine-grained, angular to subround. Matrix has less than 5% carbonate-cemented nodules, weakly cemented, up to 2 cm in size. Several cobbles of black fine-grained metasediment, granitoid, and other lithologies.	
		Weak		2/95/3 (Visual)	Dry to Moist	10YR 5/4 Yellowish Brown	SP		Out-wash		
4											1606
6										Sand with gravel, homogeneous, fine- to medium-grained, gravel is fine- to coarse-grained, subrounded to subangular. Matrix has less than 5% dark reddish brown (5YR 3/4) mottles, irregular, up to 1 cm in diameter at 7'. Sand fraction is 80% quartz, 5% feldspar, and 15% lithic fragments. Cobbles are 50% granitoid, 20% black, fine-grained metasediment, 20% magnetic cherty iron formation, 5% troctolite containing approximately 5% disseminated phyrrotite and chalcopyrite, and 5% quartzite.	1604
		None	7.17 111.7 33	30/65/5 (Visual)	Moist	10YR 4/4 Dark Yellowish Brown	SP		Upper Till		
8										8-8.5': Zone of weakly cemented carbonate layers and nodules. Occurs as masses or bridges between grains; pink (7.5YR 7/4).	1602
					Moist to Wet						

(continued)

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 3-5', 7-9', 16-18', 17-20', 20-22'; Geotechnical samples: 0-2', 2-3', 3.5-5.5', 5.5-10', 10-15', 15-19.5', 19.5-20.5', 20.5-22'; Jar samples: 0-1', 4-5', 7-9', 20', 21'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 3/10/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/23/08 | Ended 1/23/08

Location NorthMet Mine Site | Logged By MMB/MJD

LOG OF Boring RS-12

DRAFT

Elevation 1610.0

Total Depth 22.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
12			7.19 116.6 15	3/65/5 (Visual)	Wet	2.5Y 4/3 Olive Brown	SP		Upper Till	Sand with gravel, homogeneous, fine- to medium-grained, gravel is fine- to coarse-grained, subrounded to subangular. Matrix has less than 5% dark reddish brown (5YR 3/4) mottles, irregular, up to 1 cm in diameter at 7'. Sand fraction is 80% quartz, 5% feldspar, and 15% lithic fragments. Cobbles are 50% granitoid, 20% black, fine-grained metasediment, 20% magnetic cherty iron formation, 5% troctolite containing approximately 5% disseminated phyrrotite and chalcopyrite, and 5% quartzite. *(continued)*	1598
		Weak									
14											1596
16			7.14 44 14	20/70/10 (Visual)	Wet	2.5Y 4/3 Olive Brown	SP-SM			Sand with silt and gravel, homogeneous, fine- to coarse-grained sand, gravel is fine- to coarse-grained, subangular to subrounded. Tiny fractures in soil matrix have approximately 2 mm thick discoloration to dark gray (2.5Y 4/1). Sand fraction is 85% quartz, 5% feldspar, and 10% lithic fragments. Cobbles are 40% fine-grained black metasediment with common red-brown staining, 40% black cherty iron formation with yellow precipitate in some fractures and rust-colored staining on surfaces, and 20% granitoid.	1594
18											1592
										19-19.5': Silt, abrupt contacts above and below, dark grayish brown (10YR 3/2).	
				0/100/0 (Visual)	Wet	10YR 4/3 Brown	SP		Out-wash	Sand, homogeneous, fine- to coarse-grained, subangular to subrounded. *(continued)*	

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 3-5', 7-9', 16-18', 17-20', 20-22'; Geotechnical samples: 0-2', 2-3', 3.5-5.5', 5.5-10', 10-15', 15-19.5', 19.5-20.5', 20.5-22'; Jar samples: 0-1', 4-5', 7-9', 20', 21'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 3/10/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/23/08 | Ended 1/23/08

Location NorthMet Mine Site | Logged By MMB/MJD

LOG OF Boring RS-12

DRAFT

Elevation 1610.0

Total Depth 22.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
		Weak		0/100/0 (Visual)	Wet	10YR 4/3 Brown	SP		Out-wash	Sand, homogeneous, fine- to coarse-grained, subangular to subrounded. *(continued)*	
		None	7.50 68.9 26	15/70/15 (Visual)	Wet	Gley1 3/N Very Dark Gray	SM		Lower Till	Silty sand with gravel, homogeneous, dense. Sand is fine-to medium-grained, gravel is fine- to coarse-grained, subangular to subrounded. Matrix has rotten-egg odor after HCL which may be associated with yellowish brown (10YR 5/6) mottles that are 1-3 mm in diameter and disseminated throughout 1-2% the matrix. Matrix also contains 20% very dark grayish brown (2.5Y 3/2) mottles from 20.5 to 21'. Sand fraction is 50% quartz, 10% feldspar, and 40% lithic fragments. Cobbles are 40% troctolite containing trace sulfides and patches of iron staining, 30% granitoid, 15% black, fine-grained metasediment, and 5% black chert or siltstone with 2% pyrrhotite veins.	
22										Bedrock at 22'. Dark gray-black troctolite containing 2% disseminated sulfides up to 2 mm in diameter. Chalcopyrite and pyrrhotite.	1588
										End of Boring - 22 feet	
24											1586
26											1584
28											1582

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 3-5', 7-9', 16-18', 17-20', 20-22'; Geotechnical samples: 0-2', 2-3', 3.5-5.5', 5.5-10', 10-15', 15-19.5', 19.5-20.5', 20.5-22'; Jar samples: 0-1', 4-5', 7-9', 20', 21'

Additional data may have been collected in the field which is not included on this log.

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/24/08 Ended 1/24/08

Location NorthMet Mine Site | Logged By MMB/MJD

LOG OF Boring RS-13

DRAFT

SHEET 1 OF 1

Elevation 1606.0

Total Depth 10.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
		None	6.15 62.7 42	5/85/10 (Visual)	Frozen Moist	7.5R 2.5/3 Very Dark Brown	SP-SM		Soil	Sand with silt, homogeneous, fine- to coarse-grained, subangular to subrounded. Organic matter decreases from 70% to 10% between 0-1.5'. Sand fraction is 70% quartz, 10% feldspar, and 20% lithic fragments. Several cobbles of fine-grained, black metasediment with rust-colored staining on surfaces.	
2		None		5/65/30 (Visual)	Wet	7.5R 2.5/3 Very Dark Brown and 7.5R 3/1 Very Dark Gray	SM			Silty sand, variegated, homogeneous, dense, fine- to medium-grained, subangular to subrounded, trace organic material. Several very dark gray (7.5YR 3/1) lenses. Sand fraction is same as 0-1.5' interval, cobbles are fine-grained black metasediment with rust-colored surfaces. Possible perched water at 1.5'.	1604
			6.07 106.6 27							Silty sand with gravel, homogeneous, dense, fine- to medium-grained. Gravel is fine- to coarse-grained, angular to well-rounded. Matrix has dark gray brown, dark red brown, and black mottles, and has a weak rotten egg odor after HCL. Sand fraction is 80% quartz and 20% lithic fragments. Cobbles are 65% black chert/siltstone iron formation containing some rust staining and yellow precipitate, 20% granitoid, 10% black, fine-grained metasediment, and 5% pink quartzite.	
4		None		30/55/15 (Visual)	Moist	Gley1 4/5GY Dark Greenish Gray	SM		Lower Till		1602
			6.47 72.3 22								
6										Interval is too destroyed by drilling to classify.	1600
8									Bed-rock	Bedrock at 8': Dark gray-black troctolite containing 5% visible sulfides (30% pyrrhotite, 50% chalcopyrite, 20% pyrite).	1598
										End of Boring - 10 feet	

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 0-1.5', 1.5-2.5', 2.5-6', 8-10'; Geotechnical samples: 0-1.5', 1.5-2.5', 2.5-6'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation | Drill Contractor Boart Longyear

Project Name Polymet Overburden Characterization | Drill Method Rotasonic

Number 23/69-B75 INV | Drilling Started 1/24/08 Ended 1/24/08

Location NorthMet Mine Site | Logged By REE/MJD

LOG OF Boring RS-14A

DRAFT

SHEET 1 OF 1

Elevation 1609.0

Total Depth 5.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
		None		0/40/60 (Visual)	Frozen	10YR 2/1 Black to 10YR 3/6 Dark Yellowish Brown	OL/OH		Soil	Organic soil with sand, decreasing organic matter from 90%-70%, sand is fine- to medium-grained. Gradational color change.	1608
2		None	5.41 239.0 19	10/70/20 (Visual)		7.5YR 3/4 Dark Brown	SM			Silty sand, homogeneous, sand is fine- to medium-grained, gravel is fine- to coarse-grained, subangular to angular. Matrix has approximately 10% rootlets with associated very dark brown (7.5YR 2.5/2) mottles. Cobbles are 100% black fine-grained metasediment.	1606
					Moist				Upper Till	Silty sand with gravel, homogeneous, dense, sand is fine- to medium-grained, gravel is fine- to coarse-grained, subangular to subrounded. Cobbles are 90% fine-grained black metasediment, 5% black coarse-grained gabbro (no sulfides), 5% granitoid.	
4		None		30/55/15 (Visual)		10YR 3/4 Dark Yellowish Brown	SM				1604
										Bedrock at 5.0'. Black biotite argillite. End of Boring - 5 feet	
6											1602
8											1600

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Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation
Project Name Polymet Overburden Characterization
Number 23/69-B75 INV
Location NorthMet Mine Site

Drill Contractor Boart Longyear
Drill Method Rotasonic
Drilling Started 1/24/08 Ended 1/24/08
Logged By REE/MJD

LOG OF Boring RS-14B

DRAFT

SHEET 1 OF 1

Elevation 1609.0
Total Depth 5.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH-ORP-Specific Cond.	%GR/SA/FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
							OL/OH		Soil	See RS-14A, 0-1.5' interval for description.	1608
2							SM			See RS-14A, 1.5-3' interval for description.	
							SM		Upper Till	Similar to RS-14A, 3.0-5.0' interval. Slightly fewer fines, mottled. Mottles are yellowish red (5YR 4/6) and very dark grayish brown (10YR 3/2). Rust coloring also seen on most cobbles. Cobbles are 85% black fine-grained magnetic cherty iron formation and 5% granitoid.	1606
4											1604
										Bedrock at 5.0'. Black biotite argillite. End of Boring - 5 feet	
6											1602
8											1600

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: 0-1.5', 1.5-3', 3-5'; Geotechnical samples: 0-1.5', 1.5-3', 3-5'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

LOG OF Boring RS-15A-E

DRAFT

SHEET 1 OF 1

Client PolyMet Mining Corporation
Project Name Polymet Overburden Characterization
Number 23/69-B75 INV
Location NorthMet Mine Site

Drill Contractor Boart Longyear
Drill Method Rotasonic
Drilling Started 1/27/08 Ended 1/27/08
Logged By MMB/REE

Elevation 1615.5
Total Depth 0.5

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
			5.59 275 104	2/75/23 (Visual)	Moist	Black 7.5YR 3/3 Dark Brown	OL/OH		Peat	Fibrous peat.	
							SM		Soil	Silty sand, homogeneous, no odor, no mottles, no visible sulfides.	
										Hand auger refusal on rocks. End of Boring - 0.5 feet	
2											1614
4											1612
6											1610
8											1608
											1606

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Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Client PolyMet Mining Corporation
Project Name Polymet Overburden Characterization
Number 23/69-B75 INV
Location NorthMet Mine Site

Drill Contractor Boart Longyear
Drill Method Rotasonic
Drilling Started 1/27/08 Ended 1/27/08
Logged By MMB/REE

LOG OF Boring RS-16A-C

DRAFT

SHEET 1 OF 1

Elevation 1605.0
Total Depth 2.0

DEPTH FEET	SAMP. LENGTH & RECOVERY	Matrix Effervescence	Soil pH- ORP- Specific Cond.	%GR/SA/ FINES	Moisture	Matrix Color	ASTM	LITHOLOGY	Stratigraphic Unit	DESCRIPTION	ELEV. FEET
0			5.29 290 8	0/80/20 (Visual)	Moist	10YR 3/6 Dark Yellowish Brown	SM		Soil	Silty sand, homogeneous, no odor, no mottles, no visible sulfides.	1604
2										Hand auger refusal on rocks. End of Boring - 2 feet	1602
4											1600
6											1598
8											1596

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Remarks: Soil matrix and clasts were examined for visible sulfides, HCl reaction, odor, and odor after HCl. No sulfides, reaction with HCl, or unusual odors were observed, unless otherwise noted. Geochemical samples: RS-16B 0-2'; Geotechnical samples: 0-2'

Additional data may have been collected in the field which is not included on this log.

POLYMET LOG OF BORING 2008 2369B75.GPJ BARR JAN06.GDT 2/25/08

Attachment B
Analytical Plan

Attachment B
Sample Selections

Borehole No.	Start	Finish	Middle	Length	pH	Specific Conductivity	ORP	HCl rxn	Material Type or %Gravel/Sand/Fines	Matrix Color	Other Color	NOTES	ABA	Metals	MWMP	Composite
	feet	feet	feet	feet		(µS/cm)	mV									
RS-01B	0	1	0.5	1	7.05	24	248.1	N	PEAT 90-100%	10YR 2/1 BLACK				X	X	
RS-01B	1	5	3	4	5.86	10	256.9	N	10/75/15	10YR 4/4 DK YELLOWISH BROWN			X	X		
RS-01B	5	10	7.5	5	6.55	10	268.1	N	15/75/10	10YR 3/2 V DK GRAYISH BROWN	2.5YR 3/6 DK RED (10% MTLS ON FRACTURES 9'-10')					
					5.97	17	258									
RS-01B	10	15	12.5	5	6.37	16	223.7	N	15/55/20	10YR 3/2 V DK GRAYISH BROWN (10'-12.5') & 2.5Y 4/3 OLIVE BROWN (12.5'-15')	7.5YR 3/4 DK BROWN ALONG TINY FRACTURES AND COBBLE INTERFACES, 5YR 3/4 DK REDDISH BROWN	12.5' WATER TABLE	X	X		10-15'
RS-01B	15	20	17.5	5	7.28	34	65.6	N	25/60/15	2.5Y 4/3 OLIVE BROWN	2.5YR 3/6 DK RED (MTLS ON FRACTURES 9'-10'), 10YR 6/6 WEATHERING ALONG PLANAR FRACTURES ON COBBLES		X	X	X	15 - 20.5
RS-01B	20	20.5	20.25	0.5	8.79	66	-40	N	10/50/40	GLEY 1 3/10Y V DK GREENISH GRAY	2.5YR 4/3 OLIVE BROWN AT BOTTOM OF HOLE		X	X	X	15 - 20.5
RS-03	0	5	2.5	5			#N/A	N	Fibrous PEAT	2.5YR 2.5/1 REDDISH BLACK		WATER TABLE AT 3'				
RS-03	5	10	7.5	5	5.17	116	65	N	PEAT	10YR 2/1 BLACK				X	X	
RS-03	10	12	11	2			#N/A	N	10/5/85	GLEY1 5/10Y GREENISH GRAY						
RS-03	12	15	13.5	3	5.46	36	3	N	15/10/75	GLEY1 5/10Y GREENISH GRAY						
RS-03	15	20	17.5	5	7.4	50	-208.7	N	15/40/35	GLEY1 5/10Y GREENISH GRAY			X	X		
					9.08	37	-27	N								
RS-03	20	22	21	2	9.42	63	-200	N	40/10/50	GLEY1 2.5/10Y GREENISH BLACK			X	X		
RS-04	0	1	0.5	1			#N/A	N				ORGANICS				
RS-04	1	5	3	4	5.77	22	124.3	N	30/30/40	2.5Y 3/3 DK OLIVE BROWN	2.5YR 3/4 DK REDDISH BROWN MTLS		X	X		
RS-04	5	10	7.5	5	5.91	19	82	N	30/50/20	10YR 4/3 BROWN						
RS-04	10	15	12.5	5	6.33	25	104.5	N	30/50/20	10YR 4/3 BROWN	10YR 3/1 V DK GRAY 13'-15' TRANSITIONAL		X	X	X	
RS-04	15	20	17.5	5	6.74	25	-90	N	25/55/20	10YR 3/1 V DK GRAY		18-25' TRACE SULFIDES (PYRITE?) COATINGS/FLAKES IN SOIL	X	X		
					6.85	25	25									
RS-04	20	25	22.5	5	7.83	17	-89.6	N	30/50/20	10YR 3/1 V DK GRAY		18-25' TRACE SULFIDES (PYRITE?) COATINGS/FLAKES IN SOIL	X	X	X	
RS-04	25	26	25.5	1	8.4	94	173	N	70/20/10	GLEY 1 2.5/N BLACK - GLEY 1 6/1 GREENISH GRAY	PATCHY URALTIZATION GLEY 1 4/1 DK GREENISH GRAY					
RS-05A					6.42	30	124.5									
RS-05A	5	8	6.5	3	6.55	22	88.7	N	20/60/20	10YR 3/4 DK YELLOWISH BROWN						
RS-05A	8	10	9	2	6.49	19	166.6	N	40/40/20	2.5Y 4/2 DK GRAYISH BROWN			X	X	X	
RS-05A	10	13	11.5	3	8.9	88	-70	N	70/20/10	2.5Y 4/2 DK GRAYISH BROWN (10-11.5') & 2.5Y 5/1 GRAY (11.5-13')			X	X		
RS-05B	0	3.5	1.75	3.5	6.13	21	179	N	30/50/20	10YR 4/4 DK YELLOWISH BROWN						
RS-05B	3.5	4	3.75	0.5	6.54	26	187	N	30/40/30	10YR 4/2 DK GRAY BROWN						
RS-05B	4	5	4.5	1	6.25	25	193	N	30/50/20	10YR 4/4 DK YELLOWISH BROWN						
RS-06A	0	0.25	0.125	0.25			#N/A	N	0/10/90	5YR 2.5/2 DK REDDISH BROWN		95% ORGANICS	X	X	X	0-3.5'
RS-06A	0.25	1	0.625	0.75	4.45	6	290.3	N	10/50/40	10YR 4/4 DK YELLOWISH BROWN		SOME RED-BROWN COATING/STAINING	X	X	X	0-3.5'
RS-06A	1	2	1.5	1	4.84	5	313	N	5/65/30	7.5YR 3/2 DK BROWN	10YR 2/1 BLACK ORGANIC MASSES OR LENSES		X	X	X	0-3.5'
RS-06A	2	3.5	2.75	1.5	4.99	11	279	N	20/65/15	7.5YR 3/4 DK BROWN	5YR 2.5/1 BLACK MTLS; 5YR 4/6 YELLOWISH RED FEW MTLS;	RUSTY COATING/STAINING ON SAND FRACTION; ALSO ASSOCIATED WITH BLACK COBBLES	X	X	X	0-3.5'
RS-06A	3.5	4.75	4.125	1.25	5.03	8	315.7	N	CRUSHED GRANITE BOULDER AND SOME SURROUNDING MATERIAL							
RS-06A	4.75	5	4.875	0.25			#N/A	N	20/65/15	7.5YR 3/4 DK BROWN	5YR 2.5/1 BLACK MTLS; 5YR 4/6 YELLOWISH RED FEW MTLS;					
RS-06A	5	7.5	6.25	2.5	5.82	12	264.4	N	5/20/75	10YR 4/3 BROWN	30-40% 10YR 4/6 DK YELLOWISH BROWN MTLS, CONTINUOUS LAYERS AND DISSEMMINATED; 10% 2.5YR 5/2 GRAYISH BROWN		X	X		5-10'
RS-06A	7.5	10	8.75	2.5	6.32	17	251	N	15/70/15	10YR 4/3 BROWN	10YR 3/1 V DK GREY FEW MTLS (<5%);		X	X		5-10'
RS-06A	10	15	12.5	5	6.81	18	235	N	15/65/20	10YR 4/3 BROWN	MTLS 10'-12': 7.5YR 3/4 DK BROWN MTLS (70%); 10YR 4/6 DK YELLOWISH BROWN MTLS (20%); 5YR 2.5/1 BLACK (10%); BROWN OXIDATION ON PLANAR SURFACES OF COBBLE	SAMPLE IS MOISTTO WET				
RS-06A	15	19	17	4	6.75	18	38	N	15/65/20	10YR 4/3 BROWN		WK ROTTEN EGG SMELL AFTER HCL; SAMPLE IS WET	X	X	X	15-21'
RS-06A	19	21	20	2	7.86	20	18	N	15/65/20	10YR 4/3 BROWN		ROTTEN EGG SMELL AFTER HCL	X	X	X	15-21'
RS-07	0	1	0.5	1			#N/A	N	ICE AND GRASS MAT			ICE & GRASS				
RS-07	1	2	1.5	1	5.61	45	97.8	N	5/65/30	10YR 2/2 V DK BROWN	FEW 10YR 3/3 DK BROWN MTLS		X	X		1-6'
RS-07	2	3	2.5	1			#N/A	N	5/85/10	2.5Y 3/3 DK OLIVE BROWN	FEW 7.5YR 3/3 DK BROWN MTLS AND LAYERS		X	X		1-6'
RS-07	3	6	4.5	3	6.1	52	27	N	20/70/10	7.5YR 3/3 DK BROWN	MTLS: 7.5YR 2.5/2 V DK BROWN (MOSTLY) & 7.5YR 5/8 STRONG BROWN (FEW)		X	X		1-6'
RS-07	6	10	8	4	6.4	17	60	N	30/65/5	5Y 2.5/1 BLACK			X	X		

Attachment B
Sample Selections

Borehole No.	Start	Finish	Middle	Length	pH	Specific Conductivity	ORP	HCl rxn	Material Type or %Gravel/Sand/Fines	Matrix Color	Other Color	NOTES	ABA	Metals	MWMP	Composite
	feet	feet	feet	feet		(µS/cm)	mV									
					6.61	24	38									
RS-07	10	11	10.5	1	7.15	19	-23	N	50/30/20	GLEY 1 2.5/10Y GREENISH BLACK		ROTTEN EGG SMELL AFTER HCL; TRACE PYRITE IN METASEDIMENT	X	X	X	6-11', 13.5-14
					6.63	51	-125									
RS-07R	13.5	14	13.75	0.5	7.48	82	#VALUE!									6-11', 13.5-14
RS-08A	0	0.25	0.125	0.25	4.35	196	347.5	N	0/15/85	10YR 2/1 BLACK TO 10YR 2/2 V DK BROWN						
RS-08A	0.25	1	0.625	0.75			#N/A	N	15/55/30	7.5YR 3/4 DK BROWN	2-5% OF MATRIX MOTTLED; 2.5YR 3/4 DK REDDISH BROWN WELL DISSEMINATED MTLS ASSOCIATED W/ ROOTLETS AND PBLS; 5YR 5/1 GRAY SOME MTLS AND IN LAYER 15cm BELOW SOIL LAYER; ORANGE-BROWN STAINING AND PRECIPITATE COATINGS ON COARSE FRACTION;		X	X		0.25-5'
RS-08A	1	5	3	4	5.18	19	287.6	N	20/60/20	10YR 4/6 DK YELLOWISH BROWN TO 2.5Y 3/3 DK OLIVE BROWN (GRADATIONAL)	7.5YR 5/8 STRONG BROWN, 2.5YR 3/4 DK REDDISH BROWN, AND 5YR 5/1 GRAY MTLS; 3-4' MTL CONCENTRATION	MATRIX IS MOIST-WET, POSSIBLE WATER TABLE OR PERCHED ZONE	X	X		0.25-5'
					5.63	16	262.4									
RS-08A	5	11	8	6	5.78	22	217.4	N	30/50/20	10YR 4/2 DK GRAYISH BROWN	5YR 4/6 YELLOWISH RD MTLS (1-2%);	WK ROTTEN EGG SMELL AFTER HCL; MOD RED-BROWN STAINING & COATINGS ON COARSE FRACTION;	X	X	X	5-11'
RS-09	0	1	0.5	1	6.77	34	68.3	N	5/15/80	7.5YR 2.5/3 V DK BROWN	7.5YR 2.5/3 V DK BROWN MTL LENSES					
RS-09	1	5	3	4	5.96	15	175	N	20/70/10	10YR 4/4 DK YELLOWISH BROWN - 2.5Y 4/4 OLIVE BROWN (GRADATIONAL)	2.5YR 4/8 RED-7.5YR 5/8 STRONG BROWN (SECONDARY PRECIPITATES IN MICRO FRACTURES)					1-7'
RS-09	5	7	6	2	6.22	13	116.7	N	20/70/11	10YR 4/4 DK YELLOWISH BROWN - 2.5Y 4/4 OLIVE BROWN (GRADATIONAL)						1-7'
RS-09	7	8	7.5	1	5.88	2	182	N	15/20/65	2.5Y 3/1 V DK GRAY		WK ROTTEN EGG SMELL AFTER HCL	X	X	X	
RS-10	0	1	0.5	1			#N/A	N	0/70/30	7.5YR 2.5/2 V DK BROWN	LENSES OF: 10YR 3/6 DK YELLOWISH BROWN AND 10YR 2/1 BLACK					
RS-10	1	2	1.5	1			#N/A	N	35/55/10	10YR 3/2 V DK GRAYISH BROWN			X	X		1-5.5'
RS-10	2	3	2.5	1	6.07	30	193	N	25/60/15	10YR 3/6 DK YELLOWISH BROWN	MTLS: GLEY 2 2.5/1 5PB BLUISH BLACK (ASSOCIATED W/ WEATHERED ROCK)		X	X		1-5.5'
RS-10	3	5.5	4.25	2.5	5.73	12	241.6	N	10/85/5	7.5YR 3/3 DK BROWN			X	X		1-5.5'
RS-10	5.5	7.5	6.5	2	7.08	20	60.2	N	20/75/5	10YR 4/3 BROWN	MTLS: 5YR 4/6 YELLOWISH RED; MTL/CEMENT 5YR 8/1 WHITE	CEMENT NOT CARBONATE	X	X		
RS-10	7.5	10	8.75	2.5	6.81	30	152.3	N	40/40/20	5Y 3/1 V DK GRAY			X	X		7.5-14'
RS-10	10	14	12	4	6.5	26	145.3	N	40/45/15	5Y 4/3 OLIVE	FEW WHITE PRECIPITATE LENSES		X	X		7.5-14'
RS-11	0	9.5	4.75	9.5	5.89	40	107.1	N	Peat	5YR 2.5/1 BLACK		90-100% ORGANICS; WATER TABLE AT SURFACE		X		
RS-11	9.5	10	9.75	0.5			#N/A	N	60/30/10	10YR 3/2 V DK GRAYISH BROWN						
RS-11	10	11.5	10.75	1.5	6.31	67	-26.7	N	5/50/45	10YR 2/2 V DK BROWN	2.5YR 2.5/1 REDDISH BLACK ORGANIC LENSES					
RS-11	11.5	17	14.25	5.5	6.47	47	-61.4	N	20/65/15 TO 65/20/15	10YR 2/2 V DK BROWN TO 10YR 2/1 BLACK	1 -1mm SIZED REDDISH MTL IN UPPER PORTION	GRADATIONAL CHANGES IN INTERVAL; POSSIBLE TRACE SULFIDES??; RED MTL HAS ROTTEN EGG SMELL AFTER HCL	X	X		
					6.69	12	-44.1									
RS-11	17	25	21	8	6.56	30	-37.5	N	50/45/5	GLEY 1 2.5/N BLACK		WK ROTTEN EGG SMELL AFTER HCL	X	X	X	
					6.51	9	17									
RS-11	25	28	26.5	3	6.33	25	31.3	N	0/90/10 TO 30/65/5	10YR 2/1 BLACK		WK ROTTEN EGG SMELL AFTER HCL				
RS-11	28	31	29.5	3	6.5	70	-49.7	N	30/60/10	GLEY 1 3/1 10Y V DK GREENISH GREY		ROTTEN EGG SMELL AFTER HCL	X	X		28-33'
RS-11	31	33	32	2			#N/A	N	15/80/5	GLEY 1 3/1 10Y V DK GREENISH GREY			X	X		28-33'
RS-12	0	2	1	2			#N/A		2/30/68	7.5YR 5/2 TO 7.5YR 5/3	7.5YR 3/1 CLAY COATINGS					0-5.5
RS-12	2	5.5	3.75	3.5	6.77	8	114.8		2/95/3	10YR 5/4						0-5.5
RS-12	5.5	10	7.75	4.5	7.17	33	111.7	no-wk	30/68/2	10YR 4/4 DK YELLOWISH BROWN	5YR 3/4 DK REDDISH BROWN CM SIZED MTLS @ 7',	sulfides in gabbroic troctolite (5%, pyrotite, chalco.), 8-8.5' Carbonate nodules(?),	X	X	X	
RS-12	10	15	12.5	5	7.19	15	116.6	very wk	30/65/5	2.5Y 4/3 OLIVE BROWN			X	X		10-19.5'
RS-12	15	19.5	17.25	4.5	7.14	14	44	very wk	20/70/20	2.5Y 4/3 OLIVE BROWN	2.5Y 4/1, 2mm GLEY ALTERATION ALONG FRACTURES	19-19.5' 100% SILT	X	X		10-19.5'
RS-12	19.5	20	19.75	0.5			#N/A	N	0/100/0	10YR 5/3 BROWN						
RS-12	20	20.5	20.25	0.5	7.5	26	68.9	N	0/100/0	10YR 5/3 BROWN						
RS-12	20.5	22	21.25	1.5			#N/A	very wk	15/70/15	GLEY 3/N V DK GRAY	10YR 5/6 YELLOWISH BROWN MTLS, 1-2% DISSEMINATED BLEBS; 2.5Y 3/2 V DK GREYISH BROWN, 20% IN MATRIX FROM 20.5'-21'	MTLS SMELL LIKE ROTTEN EGGS AFTER HCL; SULFIDES (pyrotite, chalcopyrite) IN METASED AND TROCTOLITE COBBLES AND BEDROCK	X	X		
RS-13	0	1.5	0.75	1.5	6.15	42	62.7	N	PEAT @ TOP T0 5/85/10	7.5R 2.5/3 V DK BROWN				X		0-6'
RS-13	1.5	2.5	2	1			#N/A	N	5/65/30	7.5R 2.5/3 V DK BROWN & 7.5R 3/1 V DK GRAY	VARIEGATED AND MTLS AND LENSES			X		0-6'

Attachment B
Sample Selections

Borehole No.	Start	Finish	Middle	Length	pH	Specific Conductivity	ORP	HCl rxn	Material Type or %Gravel/Sand/Fines	Matrix Color	Other Color	NOTES	ABA	Metals	MWMP	Composite
	feet	feet	feet	feet		(µS/cm)	mV									
RS-13	2.5	6	4.25	3.5	6.07	27	106.6	N	30/55/15	5GY 4/1 DK GREENISH GRAY	MTL GLOBS: DK GRY BROWN, DK RED BROWN, BLACK 2.5/N	WK ROTTEN EGG SMELL AFTER HCL		X		0-6'
					6.47	22	72.3									
RS-13	6	8	7	2			#N/A	N				DRILL SLUDGE				
RS-13	8	10	9	2	7.2	46	-68.7	N				BEDROCK: TROCTOLITE W/ SULFIDES (PO 30%, CHPY 50%, PY 20%)	X	X	X	
RS-14A	0	1.5	0.75	1.5			#N/A	N	0/40/60	10YR 2/1 BLACK TO 10YR 3/6 DK YELLOWISH BROWN		90-70% ORGANICS		X		
RS-14A	1.5	3	2.25	1.5	5.41	19	239	N	10/70/20	7.5YR 3/4 DK BROWN	7.5YR 2.5/2 V DK BROWN ASSOCIATED W/ 10% ORGANIC CLUSTERS		X	X		1.5-5
RS-14A	3	5	4	2			#N/A	N	30/55/15	10YR 3/4 DK YELLOWISH BROWN			X	X		1.5-5
RS-14B	0	1.5	0.75	1.5			#N/A					SAME AS RS-14A 0-1.5'		X		
RS-14B	1.5	2.5	2	1			#N/A					SAME AS RS-14A 1.5-3'	X	X		1.5-5
RS-14B	2.5	5	3.75	2.5			#N/A					SAME AS RS-14A 3-5'	X	X		1.5-5
RS-15A-E	0	0.5	0.25	0.5	5.59	104	275	#N/A	2/75/23	Black 7.5YR 3/3 Dark Brown						
RS-16A-C	0	2	1	2	5.29	8	290	#N/A	0/80/20	10YR 3/6 Dark Yellowish Brown			X	X		

Attachment 2-8

Kearney, C. PolyMet Overburden Sampling Plan. Email to Stuart Arkley and Jennifer Engstrom, MDNR. February 26, 2010.

From:	Christie Kearney
To:	David Blaha (David.Blaha@erm.com); Stuart Arkley (Stuart.Arkley@dnr.state.mn.us); Engstrom, Jennifer N (DNR)
Cc:	John Borovsky; Mehgan M. Blair; Day, Stephen; Tom Radue; Heather Arends (heather.arends@state.mn.us); Dennis Martin (dennis.martin@state.mn.us); Paul Eger (DNR) (a.paul.eger@state.mn.us); Jim Scott; Cheryl D. Feigum
Subject:	PolyMet Overburden Sampling Plan
Date:	Friday, February 26, 2010 8:51:17 AM
Attachments:	Field_Rinse_pH.PDF Pit_Wall Log.pdf Boring_Locations_January_2010.pdf

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Good morning! A conference call was held on Feb. 16, 2010 to discuss PolyMet's plan to sample unsaturated overburden. As described in Barr's *NorthMet Mine Site Construction Materials and Overburden Management* (February 10, 2010), PolyMet plans to do additional sampling of the unsaturated overburden to better characterize the material for use in construction. An exploratory drilling program at the Mine Site is currently underway to collect additional bedrock cores and obtain additional depth to bedrock data for the mine plan. Access to the drilling sites was granted by the U.S. Forest Service (USFS) for the specific purpose of exploratory drilling. With this drilling program underway, PolyMet determined that this would provide an opportunity to collect unsaturated overburden samples during the excavation of sumps associated with each drill site. The conference call on Feb. 16, 2010 was held to discuss the collection of unsaturated overburden samples during this effort.

The conference call attendees included the following:
MDNR – LAM: Jennifer Engstrom, Heather Arends, Paul Eger, Kim Lapakko
Barr: John Borovsky, Cheryl Feigum, Christie Kearney, and Steve Day with SRK

At this meeting, it was requested that PolyMet submit a sampling plan for this effort. The collection of unsaturated overburden at these sumps has already begun and a formal sampling plan has not previously been developed. According to the drilling plan accepted by the USFS, the sumps are to be "…generally 8' by 8' by 5' deep, or smaller," which typically has been the case. The sampling methods have been documented in the field and are generally as follows:

1) For sumps that have been excavated and filled with drill cuttings but not yet closed, as was the case for J020, J018, and the "central sump" located near MW-05-08, break up the frozen soil with a spud bar to collect the sample (as described in step 5). Describe any remaining exposed sidewall, and describe the soil sampled, once a split of the samples has been thawed.
2) For sumps not already opened, excavate the sump by segregating bulk soils of different compositions or colors into individual spoil piles. Individual layers are generally not segregated unless a significant thickness (minimum of 2') is observed, due to sample volume requirements and equipment inefficiencies at smaller scales. The depths each spoil pile came from (i.e., 0-3, 3-6, etc) is documented, and the sump profile is described (color; texture [ASTM D2488]; moisture; mottling, if present; reaction with dilute HCl; magnetic properties). An example of a completed sump profile is attached.
3) Photograph sump profiles and spoil piles;
4) GPS each sump location;
5) Collect samples from the spoil piles at each sump, collecting about 4-5 gallons of soil per

sample (lab may require up to 12 kg, depending on selected analyses);

6) Split the sample to keep about a half gallon of soil thawed for rinse testing and freeze the remaining sample for further analysis. The rinse testing procedure is attached.

In an email on Feb. 9, 2010, Jennifer Engstrom, MDNR-LAM, requested that this sampling effort be focused on "*areas that will be disturbed during construction and mining*" and "*areas of thicker unsaturated overburden since that represents a larger quantity of material*." Because this sampling coincides with an on-going drilling program, few changes can be made to the sump locations. At this time, available sumps have been described, and samples have been collected where unsaturated conditions have been observed. This includes the following drill sites: J020, J018, J107, J007, J012, J013, J008, J010/J011, J003/J004, J027, and J037. However, sump descriptions and sampling efforts going forward will be focused on sumps opened within or adjacent to the footprint of the RMSA pits: J039/J040, J036, J030, J024, J021/J022, J029, J019, J015/16, and possibly J025, J031, and J032. The scope and schedule of overburden sampling is constrained by the evolving scope and schedule of the exploratory drilling program and the accessibility of the sites (sites will become inaccessible as temperatures increase this spring).

The Feb. 9, 2010 email also requests the sampling of soil horizons separately.

Based on the results of the Feb. 16, 2010 conference call, we are proceeding with this sampling effort in accordance to the methods described above. If possible, depending on the depths of the soil horizons found at each sump, we will collect additional soil horizon samples for analyses, as requested.

The results of this additional sampling will be submitted to the MDNR with recommendations regarding subsequent laboratory analysis.

Please contact Mehgan Blair at 218-529-8237 or me at 218-262-8629 if you have any questions about or concerns about this sampling program.

Christie M. Kearney
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Attachment 2-9

Kearney, C. PolyMet Unsaturated Overburden Sampling Analysis Plan. Email to Stuart Arkley and Jennifer Engstrom, MDNR. March 3, 2010 (updated March 4, 2010).

From:	Christie Kearney
To:	Christie Kearney; David Blaha (David.Blaha@erm.com); Stuart Arkley (Stuart.Arkley@dnr.state.mn.us); "Engstrom, Jennifer N (DNR)"
Cc:	John Borovsky; Mehgan M. Blair; Day, Stephen; Tom Radue; Heather Arends (heather.arends@state.mn.us); Dennis Martin (dennis.martin@state.mn.us); Paul Eger (DNR) (a.paul.eger@state.mn.us); "Jim Scott"; Cheryl D. Feigum; Michael Olson (MDNR)
Subject:	PolyMet Unsaturated Overburden Sampling Analysis Plan
Date:	Wednesday, March 03, 2010 1:04:26 PM
Attachments:	Soil Genesis Grigal 2010.doc Preparation_Scheme-Analyses.ppt

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Proposed Unsaturated Overburden Sample Analysis Plan
March 3, 2010

PolyMet submitted an Overburden Sampling Plan on February 25, 2010 via email from Christie Kearney to Dave Blaha, Stuart Arkley, and Jennifer Engstrom, with copy to several others under the subject *PolyMet Overburden Sampling Plan*. In that email, it was stated that PolyMet would be submitting recommendations regarding the subsequent laboratory analysis. This email lays out the proposed plan for the laboratory analyses of the unsaturated overburden samples that are currently being collected.

In an email on Feb. 9, 2010, Jennifer Engstrom, MDNR-LAM, provided the following:

> Suggestions for sample analysis:
>
> 1) one sample of minus the particle size to be used for construction or other on-site use, crushed and analyzed (total mass) for similar suite of constituents as previous sampling and analysis plan,
> 2) separate sample of minus 2 mm, crushed and analyzed (total mass) for similar suite of constituents as previous sampling and analysis plan,
> 3) particle size distribution (gradations by weight %) to better understand the chemistry,
> 4) test soil horizons separately since previous sampling indicated elevated metal concentrations, and
> 5) remainder of tests that were conducted for the previous overburden sampling

Based on previous analyses and MDNR suggestions, the unsaturated overburden samples collected will be evaluated as shown on the attached figure:

- Rinse test for pH, oxidation/reduction potential, and specific conductivity of the bulk sample
- Particle size distribution of the bulk sample via sieve and hydrometer testing (ASTM D422 – Standard Test Method for Particle-Size Analysis of Soils)
- Meteoric Water Mobility Procedure (MWMP, Nevada DEP) of the bulk sample
- Metals analysis of the minus 2 mm fraction (aqua regia, as was previously done, and 1 *N* nitric acid)
- Moisture Content
- Acid-Base Accounting

The MWMP test is used to predict leaching by first contact of water when exposed by excavation. This test was used previously to assess the potential effects on water quality as a result of excavation of the overburden material on the site. Therefore, it will be the main tool used to

evaluate leaching. Acid-base accounting will indicate whether there is potential for long term water degradation.

The metal analysis of the minus 2 mm fraction will be used as backup information to confirm the metal concentrations in the sample population and to evaluate the results of the MWMP testing. Although the aqua regia analysis was previously used for metal analysis, the analysis with 1 *N* nitric acid (HNO_3) is a new extraction method for the overburden evaluation at PolyMet. This extraction is based on the recommendations of the Regional Technical Committee of the USDA (W-124, The Optimum Utilization of Sewage Sludge on Agricultural Land) (Pierce 1980). The extraction is presumed to include both labile (water soluble, exchangeable, associated with organic matter, and adsorbed on oxide surfaces) and other biogenetically available metals. The later term refers to metals that could reasonably be expected to become available to plants via weathering over relatively long time periods. Thus this extraction should provide a reasonable representation of metals available for movement with infiltrating water over the long term and the results from this extraction would provide a basis for a direct comparison to similar data for other Minnesota soils, including Rainy Lobe till (Pierce et al 1982). For further discussion of this extraction in regards to the Mine Site, see the attached paper from Dr. David Grigal that was submitted to the MDNR on March 1, 2010 from John Borovsky (Grigal 2010).

Based on Barr's *Preliminary Agency Preference Meeting Follow-Up: Action Item 1 – Stockpile Liner Construction* (January 27, 2010 Memorandum by Tom Radue to Dave Blaha, ERM, and Stuart Arkley, MDNR):

> *The selection of the maximum allowable particle size is typically based on engineering experience and judgment. It is Barr's experience that limiting the maximum particle size to approximately 1/3 the compacted soil lift thickness accommodates uniform compaction of the soil layer to the specified density.*

Therefore, maximum particle size used for construction is dependent on the application. The maximum particle size for a 9-inch thick soil lift, for example, would be 3 inches, and most construction uses for the unsaturated overburden would also be limited to something similar on the basis of the soil lift thickness as placed during a typical construction sequence. Collection of the samples in the field is done with a trenching shovel, capturing the samples into a bucket and discarding rocks in excess of approximately 5 inches or more. Therefore, the bulk sample encompasses material that is representative of our construction materials. Therefore, interpretation of the MWMP results of the bulk sample will be representative of the actual water quality we could expect from the construction areas where the unsaturated overburden is used.

The proposed analyses would not include metal analyses of the bulk sample or of minus 3 inches, because the MWMP is a better representation of what the actual water quality results will be than the metal assay analysis; therefore we are not proposing to include a sample to meet MDNR Item 1.

The proposed analysis plan does meet the suggestion of MDNR Items 2, 3, and 4. As described in our *Overburden Sampling Plan* email from February 25, 2010, depending on the depths of the soil horizons found at each sump, we will collect additional soil horizon samples for analyses if practical.

MDNR Item 5 suggests that we do the same analyses as was done previously. As described above, we are focusing our efforts on the MWMP evaluation with metal analyses on the less than 2mm

fraction to support the evaluation of MWMP results; therefore we are not proposing to do the full suite of analyses as the previous sampling effort and do not believe the additional analyses are beneficial for understanding the ultimate potential water quality effects of the use of this material in construction applications.

References

Grigal, D. 2010. Soil Materials at NorthMet. February 25.

Pierce, F.J. 1980. The content and distribution of Cd, Cr, Cu, Ni, Pb, and Zn in 16 selected Minnesota soil series. M.S. thesis, University of Minnesota, St. Paul, MN, USA. 140 p.

Pierce, F. J., R. H. Dowdy, and D. F. Grigal. 1982. Concentrations of six trace metals in some major Minnesota soil series. J. Environ. Qual. 11:416-422.

Please contact Mehgan Blair at 218-529-8237 or me at 218-262-8629 if you have any questions about or concerns about this sampling program.

Christie M. Kearney
Water Resources Environmental Engineer
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Barr Engineering Company
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From:	Christie Kearney
To:	Christie Kearney; David Blaha (David.Blaha@erm.com); Stuart Arkley (Stuart.Arkley@dnr.state.mn.us); "Engstrom, Jennifer N (DNR)"
Cc:	John Borovsky; Mehgan M. Blair; Day, Stephen; Tom Radue; Heather Arends (heather.arends@state.mn.us); Dennis Martin (dennis.martin@state.mn.us); Paul Eger (DNR) (a.paul.eger@state.mn.us); "Jim Scott"; Cheryl D. Feigum; Michael Olson (MDNR)
Subject:	RE: PolyMet Unsaturated Overburden Sampling Analysis Plan
Date:	Thursday, March 04, 2010 8:41:05 AM
Attachments:	PolyMet Preparation Scheme-Analyses 3-4-2010.pdf

Please replace the figure sent yesterday with the attached figure, which is a better representation of PolyMet's unsaturated overburden sample analysis plan. I apologize for the inconvenience!
Christie

From: Christie Kearney
Sent: Wednesday, March 03, 2010 1:04 PM
To: Christie Kearney; David Blaha (David.Blaha@erm.com); Stuart Arkley (Stuart.Arkley@dnr.state.mn.us); 'Engstrom, Jennifer N (DNR)'
Cc: John Borovsky; Mehgan M. Blair; 'Day, Stephen'; Tom Radue; Heather Arends (heather.arends@state.mn.us); Dennis Martin (dennis.martin@state.mn.us); Paul Eger (DNR) (a.paul.eger@state.mn.us); 'Jim Scott'; Cheryl D. Feigum; Michael Olson (MDNR)
Subject: PolyMet Unsaturated Overburden Sampling Analysis Plan

Proposed Unsaturated Overburden Sample Analysis Plan
March 3, 2010

PolyMet submitted an Overburden Sampling Plan on February 25, 2010 via email from Christie Kearney to Dave Blaha, Stuart Arkley, and Jennifer Engstrom, with copy to several others under the subject *PolyMet Overburden Sampling Plan*. In that email, it was stated that PolyMet would be submitting recommendations regarding the subsequent laboratory analysis. This email lays out the proposed plan for the laboratory analyses of the unsaturated overburden samples that are currently being collected.

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> Suggestions for sample analysis:
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> 2) separate sample of minus 2 mm, crushed and analyzed (total mass) for similar suite of constituents as previous sampling and analysis plan,
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D422 – Standard Test Method for Particle-Size Analysis of Soils)

- Meteoric Water Mobility Procedure (MWMP, Nevada DEP) of the bulk sample
- Metals analysis of the minus 2 mm fraction (aqua regia, as was previously done, and 1 *N* nitric acid)
- Moisture Content
- Acid-Base Accounting

The MWMP test is used to predict leaching by first contact of water when exposed by excavation. This test was used previously to assess the potential effects on water quality as a result of excavation of the overburden material on the site. Therefore, it will be the main tool used to evaluate leaching. Acid-base accounting will indicate whether there is potential for long term water degradation.

The metal analysis of the minus 2 mm fraction will be used as backup information to confirm the metal concentrations in the sample population and to evaluate the results of the MWMP testing. Although the aqua regia analysis was previously used for metal analysis, the analysis with 1 *N* nitric acid (HNO_3) is a new extraction method for the overburden evaluation at PolyMet. This extraction is based on the recommendations of the Regional Technical Committee of the USDA (W-124, The Optimum Utilization of Sewage Sludge on Agricultural Land) (Pierce 1980). The extraction is presumed to include both labile (water soluble, exchangeable, associated with organic matter, and adsorbed on oxide surfaces) and other biogenetically available metals. The later term refers to metals that could reasonably be expected to become available to plants via weathering over relatively long time periods. Thus this extraction should provide a reasonable representation of metals available for movement with infiltrating water over the long term and the results from this extraction would provide a basis for a direct comparison to similar data for other Minnesota soils, including Rainy Lobe till (Pierce et al 1982). For further discussion of this extraction in regards to the Mine Site, see the attached paper from Dr. David Grigal that was submitted to the MDNR on March 1, 2010 from John Borovsky (Grigal 2010).

Based on Barr's *Preliminary Agency Preference Meeting Follow-Up: Action Item 1 – Stockpile Liner Construction* (January 27, 2010 Memorandum by Tom Radue to Dave Blaha, ERM, and Stuart Arkley, MDNR):

> *The selection of the maximum allowable particle size is typically based on engineering experience and judgment. It is Barr's experience that limiting the maximum particle size to approximately 1/3 the compacted soil lift thickness accommodates uniform compaction of the soil layer to the specified density.*

Therefore, maximum particle size used for construction is dependent on the application. The maximum particle size for a 9-inch thick soil lift, for example, would be 3 inches, and most construction uses for the unsaturated overburden would also be limited to something similar on the basis of the soil lift thickness as placed during a typical construction sequence. Collection of the samples in the field is done with a trenching shovel, capturing the samples into a bucket and discarding rocks in excess of approximately 5 inches or more. Therefore, the bulk sample encompasses material that is representative of our construction materials. Therefore, interpretation of the MWMP results of the bulk sample will be representative of the actual water quality we could expect from the construction areas where the unsaturated overburden is used.

The proposed analyses would not include metal analyses of the bulk sample or of minus 3 inches, because the MWMP is a better representation of what the actual water quality results will be than the metal assay analysis; therefore we are not proposing to include a sample to meet MDNR Item 1.

The proposed analysis plan does meet the suggestion of MDNR Items 2, 3, and 4. As described in our *Overburden Sampling Plan* email from February 25, 2010, depending on the depths of the soil horizons found at each sump, we will collect additional soil horizon samples for analyses if practical.

MDNR Item 5 suggests that we do the same analyses as was done previously. As described above, we are focusing our efforts on the MWMP evaluation with metal analyses on the less than 2mm fraction to support the evaluation of MWMP results; therefore we are not proposing to do the full suite of analyses as the previous sampling effort and do not believe the additional analyses are beneficial for understanding the ultimate potential water quality effects of the use of this material in construction applications.

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Grigal, D. 2010. Soil Materials at NorthMet. February 25.

Pierce, F.J. 1980. The content and distribution of Cd, Cr, Cu, Ni, Pb, and Zn in 16 selected Minnesota soil series. M.S. thesis, University of Minnesota, St. Paul, MN, USA. 140 p.

Pierce, F. J., R. H. Dowdy, and D. F. Grigal. 1982. Concentrations of six trace metals in some major Minnesota soil series. J. Environ. Qual. 11:416-422.

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Attachment 2-10

SRK Consulting. RS53A – Waste Rock Characterization Progress Report – DRAFT, NorthMet Project. May 2006.

DRAFT
RS53A Waste Rock Characterization Progress Report

NorthMet Project

Report Prepared for

Polymet Mining Corp.

Report Prepared by

SRK Consulting
Engineers and Scientists

May 2006

DRAFT
RS53A Waste Rock Characterization Progress Report

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1 Introduction

1.1 Background

PolyMet Mining (PolyMet) is proposing to develop the NorthMet Project (Dunka Road Project of US Steel) near Babbitt, Minnesota. As a part of the Minnesota Department of Natural Resources (MDNR) "Permit to Mine" process a complete "mine waste characterization" will be required (Minnesota Rules Chapter 6132.1000).

The issues associated with waste rock at the NorthMet are expected to include acid rock drainage (ARD) and leaching of some heavy metals. The latter in particular are expected to include nickel and cobalt both of which do not require acidic conditions to be mobilized at elevated concentrations.

The specific objectives of this program include:

- Refinement of preliminary waste rock management criteria developed by PolyMet and MDNR.
- Development of mass-loading rates for input into water quality predictions for impact assessment and mitigation design.

In general, three "waste" categories have been defined. Non-reactive waste rock will ideally have leachate that does not require treatment prior to discharge to the receiving environment. Reactive waste rock has concentrations of commodity elements well below levels needed to justify processing in the mill and will have leachate chemistry that will require treatment prior to discharge. The third category is lean ore. This is rock that may be processed in the future to recover commodity metals. This rock will be disposed in a separate location and therefore requires separate estimation of water quality characteristics.

The characterization program was developed in consultation with the Minnesota Department of Natural Resources. The resulting plan (SRK 2006) is in Appendix A.

The sample selection methodology for waste rock was agreed upon prior to lean ore, and therefore 28 weeks of test data are now available. Components of the lean ore program were started at the same time, but testing of additional samples was requested by the DNR, and the majority of tests have 16 weeks of data, which is short of the 26 weeks defined as representing a minimum duration. Therefore, this report describes results for the waste rock tests. RS42 will provide the complete dataset for all tests.

The objective of this report is to provide a summary of the data obtained, initial conclusions on relationships between rock characteristics and leaching rates, and recommendations for modifications to the program. RS42 will apply the data to prediction of water chemistry for waste rock stockpiles.

2 Methods

2.1 Sample Selection, Collection and Compositing

Details of the sample selection, collection and compositing are provided in Appendix A. Characteristics of waste rock samples being tested in dissolution experiments are provided in Appendix B.

2.2 Mineralogical Characterization

All proposed mineralogical characterization has been completed. Each sample under test has been described using the optical properties of the minerals. Optical mineralogical descriptions were prepared by Steve Geerts (PolyMet). The descriptions included detailed characterization of sulfide minerals. Mineralogical results are provided in Appendix C.

2.3 Chemical Characterization

2.3.1 Bulk Chemistry

Bulk chemical analyses have been performed on the samples under test. These analyses included:

- Acid base accounting (total S, S as sulfate, carbonate).
- 27 elements by ICP scan following four-acid (nitric-hydrochloric-perchloric-hydrofluoric) digestion (near total) (0.5 g).
- 34 elements by ICP scan following aqua regia (nitiric-hydrochloric acid) digestion (0.5 g).
- Whole rock oxides (0.5 g).

2.3.2 Sub-Optical Analysis of Mineral Grains

Electron microprobe was used to estimate the concentrations of Fe, Cu, S, Ni, Co, Zn, Ti and As in 268 sulfide mineral grains and SiO_2, Al_2O_3, TiO_2, FeO, MnO, MgO, K_2O, CaO, Na_2O, NiO, CuO and CoO in 236 silicate mineral grains. Results are provided in Appendix D.

These analyses were used to confirm the identity of minerals determined optically. More than 90% of mineral grains were correctly determined optically. The most commonly mis-identified mineral was pentlandite. The microprobe results showed that a few grains of optically identified pentlandite was pyrite. A few grains of optically identified pyrrhotite were found to be pentlandite, cobaltite or bornite.

Optically determinations of silicates were 100% accurate.

2.3.3 Dissolution Tests

Details of the dissolution test procedures are provided in the Waste Rock and Lean Ore Characterization Plan (Appendix A).

Humidity Cells

Humidity cell tests are being conducted according to the ASTM procedure.

DNR Reactors and Size Fraction Tests

The DNR Reactor procedure was developed by Minnesota DNR. It involves leaching of a small sample (75 g) with 200 mL of water. The leachate analysis schedule is the same as for humidity cells.

The size fraction dissolution tests were performed on four fractions prepared from five samples. The two finest fractions (-35+100 mesh, and -100 mesh) are performed using the DNR Reactor procedures. The coarsest fractions (-1/4”+10 mesh, -10+35 mesh) are performed using a modification of the ASTM humidity cell procedure in which 1 kg of sample is leached with 300 mL of water. The analytical schedule is the same as that used for the humidity cells.

2.4 Comparison with Plan

The primary focus of the Waste Rock and Lean Ore Geochemical Characterization Plan was selection of samples to characterize the variables that could affect weathering behavior. A sample selection matrix was developed, and is reproduced in Table 1 to indicate sample obtained compared to the plan. The table indicates that all samples sought were obtained though the difference between target and obtained sulfur concentrations was greater than 50% for two non-reactive and two reactive samples. It was agreed with DNR that obtaining replacement samples was impractical and that the effect of these samples on the overall program would not be important.

Table 1: Comparison of Sampling Plan with Samples Obtained

Unit	Rock Type	Estimated Tonnages		Approximate Sulfur Contents											
		Non-Reactive	Reactive	Non-reactive[1]			Reactive[2]								P size
		M. tons	M. tons	NR1	NR2	NR3	P10	P25	P50	P75	P80	P85	P90	P95	P100	
1	Anorthositic	0.57	0.99				0.08	***0.1***	***0.15***	***0.29***				***1.09***	1.09	***4***
1	Sedimentary hornfels	0	1.6				***0.08***	***0.35***	***0.69***	***2.2***	2.32	***2.81***	3.38	3.5	3.78	
1	Troctolitic	17.2	40.1	***0.01***	***0.03***	***0.05***	0.06	0.07	***0.1***	0.18	***0.21***	0.26	***0.34***	***0.62***	***1.97***	***4***
1	Ultramafic	0.21	1.1				0.07	***0.08***	0.1	0.13	***0.2****	***0.3****	***0.5****	***0.8****	1.35	
2	Anorthositic	2.4	0.56	***0.01***	***0.03***	***0.05***	0.06	0.06	0.08	0.11				0.19		
2	Troctolitic	16.9	9.7	***0.01***	***0.03***	***0.05***	0.06	0.06	0.07	0.09	0.09	0.1	0.11	0.12	0.25	
2	Ultramafic	0.38	0.25	***0.03***	***0.04***	***0.05***	0	0	0	0				0		
3	Anorthositic	9.4	1.2	***0.01***	***0.03***	***0.05***	0.07	0.08	0.08	0.12				0.14		
3	Troctolitic (augite)	41.2	12.5	***0.01***	***0.03***	***0.05***	0.06	0.06	***0.08***	0.1	0.11	***0.12***	0.14	***0.19***	0.36	***4***
4	Troctolitic	7	2.2	***0.01***	***0.03***	***0.05***	0.06	***0.07***	0.09	***0.18***	0.19	0.19	***0.48***	***0.92***	1.53	***4***
5	Troctolitic	2.5	2	***0.01***		***0.05***	0.07	0.09	0.11	0.16				0.22	0.22	
6	Troctolitic	6.8	0.59	***0.02***	***0.04***		0	0	0	0				0		
20	Virginia	0	10.4				0.59	***1.25***	2.98	***4.15***	4.49	4.85	5.07	***6.06***	7.45	***4***

Notes:
1. Non-reactive rock categories (lower, medium and higher sulfur contents).
2. Sulfur percentiles for reactive rock types calculated by PolyMet are shown. "*" indicates approximate percentiles.
3. Grey – sampling plan.
4. Bold and italic – samples obtained.
5. Bold border – duplicate cell in operation.

3 Results and Discussion

3.1 Mineralogy

3.1.1 Non-Reactive Waste Rock

The dominant silicate phases in all samples were either plagioclase or olivine. One sample of ultramafic (00-368C-460-465) showed alteration to chlorite (20%) and serpentine (30%). One sample of troctolite (00-367C-290-310) also showed alteration to chlorite (40%). The only other significant mineral group were pyroxenes which occurred in concentrations varying from 0 to 25%. Clinopyroxene was dominant.

Sulfide mineral content was described as not detected (0% sulfide, rare and trace), which is consistent with the low sulfur content of the samples. The sulfide mineral content was further described in terms of minerals and proportion of the overall sulfur content. Ten sulfide minerals were identified, but five were present at proportions exceeding 15% in at least some samples with the approximate order of importance of these minerals represented by:

Chalcopyrite > Pyrrhotite > Pentlandite > Cubanite > Bornite

In a few samples, pyrrhotite exceeded chalcopyrite. Other sulfide minerals identified included digenite, covellite, violarite, mackinawite/valleriite and sphalerite. Pyrite was not recognized in any of the samples. The distribution of sulfide minerals in the non-reactive waste rock shows the importance of chalcopyrite rather than pyrrhotite as the main host for sulfur.

3.1.2 Reactive Waste Rock

The reactive waste rock group includes Duluth Complex igneous rocks, and xenoliths and footwall Virginia Formation rocks.

With the exception of one sample (00-357C-335-340) of the Duluth Complex rocks, the non-sulfide mineralogy of the samples was dominated by plagioclase, followed primarily by olivine. As in the non-reactive rocks, chlorite and serpentine alteration was present in some samples along with both orthopyroxene and clinopyroxene. Sample 00-357C-335-340, identified as ultramafic rock from Unit 1 was dominated by potassium feldspar, clinopyroxene, chloride and epidote, and was therefore not ultramacic.

Sulfide content in the Duluth Complex samples was dominated by similar minerals are the non-reactive rocks, but pyrrhotite became the dominant sulfide, and bornite was a small proportion so that the overall order of importance became:

Pyrrhotite > Chalcopyrite > Pentlandite = Cubanite

Other sulfide minerals identified were chalcocite, covellite, violarite, mackinawite/valleriite, sphalerite, galena, enargite and talnakhite.

The dominant non-sulfide minerals in the sedimentary hornfels xenoliths and Virginia Formation were potassium feldspar and cordierite. Other minerals included quartz, clinopyroxene, biotite, white mica, graphite, carbonate and idocrase. Carbonate and idocrase were identified in one sample of sedimentary hornfels (26030-1047-1052). The dominant sulfide mineral was pyrrhotite with mostly minor sphalerite, galena and chalcopyrite. One sample (26030-1047-1052) of sedimentary hornfels contained sphalerite as the dominant sulfide.

3.2 Analysis of Mineral Grains

Microprobe was used to estimate the concentrations of Fe, Cu, S, Ni, Co, Zn, Ti and As in 268 sulfide mineral grains and SiO_2, Al_2O_3, TiO_2, FeO, MnO, MgO, K_2O, CaO, Na_2O, NiO, CuO and CoO in 236 silicate mineral grains.

Table 2 provides simple statistics for nickel concentrations in the major sulfide minerals. The detection limit was approximately 0.02%. The main nickel-bearing mineral is pentlandite. In terms of overall significance as a source of nickel, pyrrhotite is the next most significant (median 0.1%). Other minerals containing nickel as a major component were cobaltite (CoAsS) and maucherite ($Ni_{11}As_8$) but both are rare minerals. Pyrite when detected contained more nickel than pyrrhotite but it is uncommon.

Table 3 shows nickel concentrations in silicates minerals. Concentrations expressed as nickel oxide were converted to nickel. The detection limit is 0.016% (0.02% NiO). These results show that olivine and biotite contained comparable levels of nickel followed by the pyroxenes. Plagioclase contained low levels of nickel.

Table 2: Nickel Concentrations (%) in Sulfide Minerals

	Bornite	Chalcopyrite	Cobaltite	Cubanite	Maucherite	Pentlandite	Pyrite	Pyrrhotite
n	3	75	3	41	3	70	8	64
Min.	0.021	0.021	11	0.02	46	26	0.02	0.02
P5	0.029	0.021	12	0.02	47	29	0.04	0.02
Median	0.106	0.022	18	0.02	52	32	0.44	0.10
Mean	0.085	0.057	16	0.94	50	32	0.51	0.22
P95	0.126	0.131	20	1.87	53	37	1.15	0.80
Max.	0.128	0.639	20	30.75	53	43	1.22	1.48

Table 3: Nickel Concentrations (%) in Silicate Minerals

	Olivine	Plagioclase	Clinopyroxene	Orthopyroxene	Biotite
n	13	71	57	20	71
Min.	0.016	0.016	0.016	0.016	0.016
P5	0.016	0.016	0.016	0.016	0.016
Median	0.082	0.016	0.016	0.016	0.058
Mean	0.077	0.017	0.023	0.022	0.057
P95	0.130	0.022	0.048	0.047	0.095
Max.	0.143	0.031	0.059	0.059	0.128

3.3 Humidity Cells

3.3.1 Data Plotting

The graphs in Appendices E and F show concentrations as a function of weekly leaching cycles for non-reactive and reactive rock samples, respectively. Lines on the graphs are color-coded by igneous unit with symbols for the three main rock types (Table 4).

Table 4: Colors and Symbols for Humidity Cell Graphs

Unit	Color	Rock Type	Symbol
1	Red	Troctolitic	Triangle
2	Green	Ultramafic	Circle
3	Grey	Anorthositic	Square
4	Dark Blue		
5	Yellow		
6	Light Blue		

The graphs for metal concentrations were prepared using results from low level ICP-MS and higher level ICP-OES results. For almost all parameters, ICP-MS has a lower reporting limits than ICP-OES and to avoid a “sawtooth” pattern to the trends, the following plotting rules were used:

- For parameters showing lower reporting limits by ICP-MS:
  - If the result is at or below the ICP-MS reporting limit, graph as the reporting limit.
  - If the result is above the ICP-MS reporting limit, show the result.
  - If the result is at or below the ICP-OES reporting limit, do not show the result on the graph.
  - If the result is above the ICP-OES reporting limit, show the result on the graph.

If the ICP-OES reporting limit is lower than ICP-MS, the rules are reversed.

3.3.2 Non-Reactive Waste Rock Results

Leachate Trends

Graphs illustrating concentrations in humidity cell leachates are provided in Appendix E.

Leachate chemistry indicated the following general features:

- pH initially decreased followed by stable pHs between 7.0 and 7.7, with some exceptions. The sample with chlorite and serpentine (00-368C-460-465) showed initially increasing pH (to more than 9) followed by decrease in pH near to values shown by other cells. Ultramafic sample 26064-44-54 also showed higher pHs. No mineralogical explanation was apparent.
- The main anions in leachates are sulfate, bicarbonate (alkalinity). Cation chemistry was initially dominated by sodium which steadily decreased for all cells and was replaced by calcium and magnesium as the dominant cations.
- Some parameters are showing stable concentrations including chloride, fluoride, aluminum, iron, manganese, and nickel.
- Some parameters are showing decreasing trends including antimony, arsenic, barium, copper, and potassium.
- Concentrations of some parameters were generally not detectable by ICP-MS including cadmium, chromium, cobalt, lead, mercury, molybdenum, selenium, silver, thallium and zinc.

Rate Relationships

Concentrations indicated for humidity cell leachates were converted to weekly release rates (mg/kg/week) using the volume of leachate recovered (approximately 500 mL/week) and the mass of sample tested (1 kg). Using release trends (i.e. release rate as a function of time), the average loading for each parameter in each cell was calculated for the time period over which release rates were generally stable.

The release rates obtained have been compared to the bulk characteristics in the sample design (rock type, geological unit, sulfur content and metal concentrations) and mineralogy to evaluate variables that control release rates. Based on a thorough analysis, only a few significant relationships have emerged for the main potential contaminants (Figures 1 and 2).

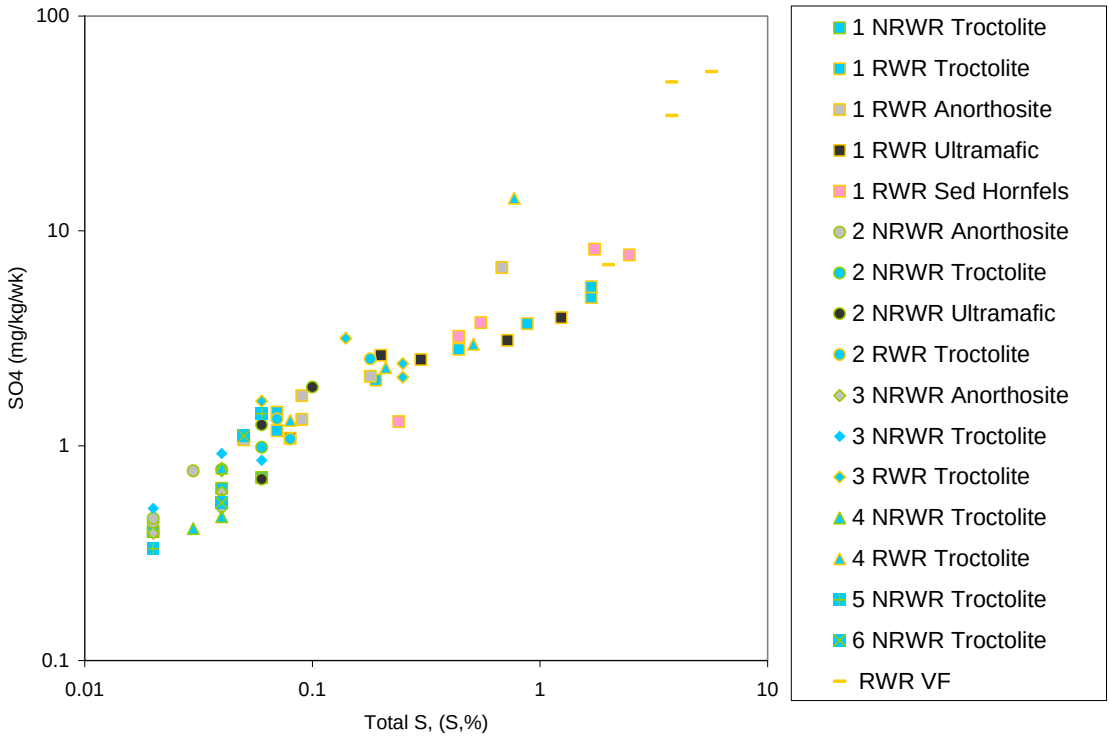

Figure 1: Average Sulfate Release Compared to Total Sulfur Content of Non-Reactive and Reactive Waste Rock. Green outline on symbols are non-reactive samples and orange outlines are reactive samples.

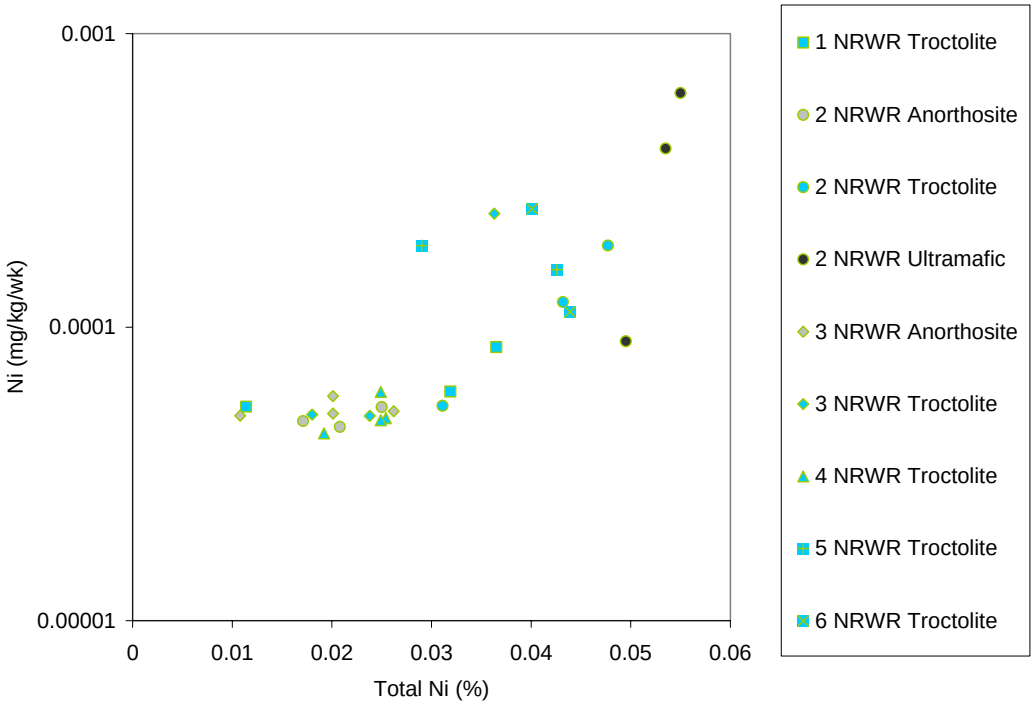

Figure 2: Average Nickel Release Compared to Total Nickel Content of Non-Reactive Waste Rock. See Figure 1 for explanation of symbols.

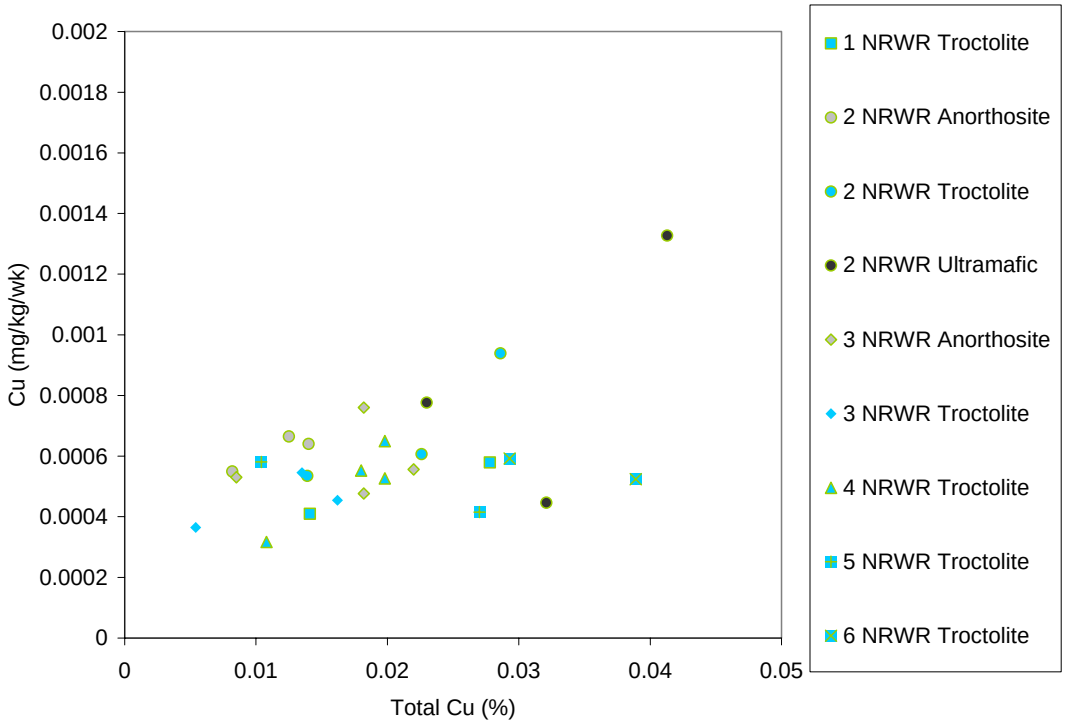

Figure 3: Average Copper Release Compared to Total Copper Content of Non-Reactive Waste Rock. See Figure 1 for explanation of symbols.

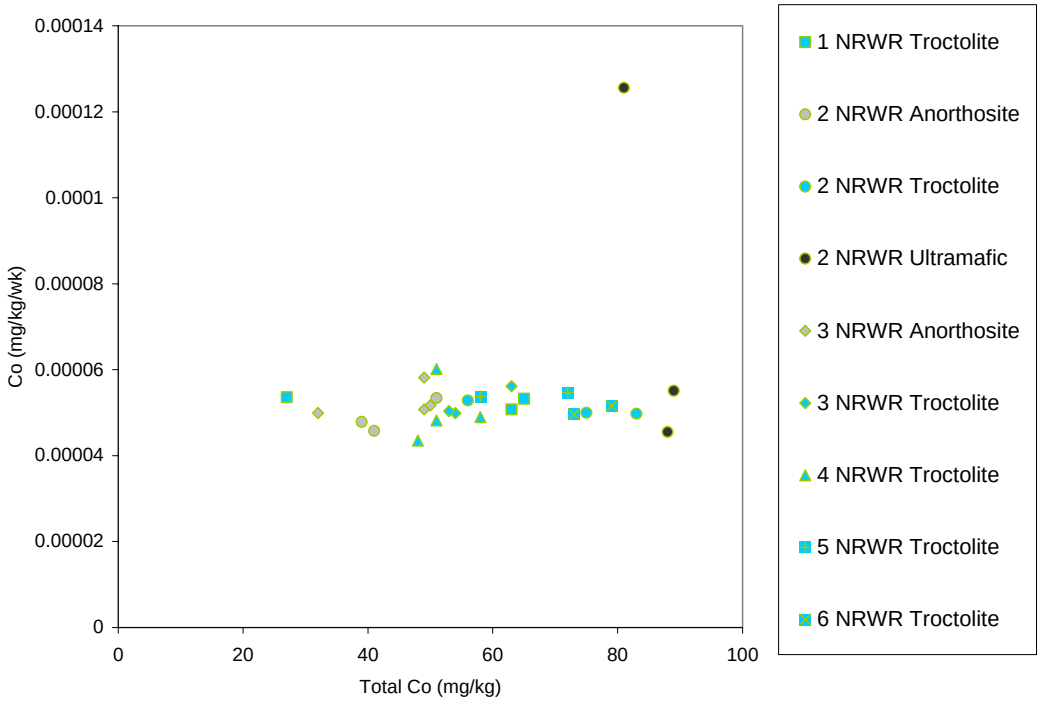

Figure 4: Average Cobalt Release Compared to Total Cobalt Content of Non-Reactive Waste Rock. See Figure 1 for explanation of symbols.

Figure 1 shows the very strong relationship between sulfur content and sulfate release for non-reactive waste rock samples (green outline symbols). The scatter in the datapoints is not related to any particular factor but is due to the analytical variability in sulfur analysis between 0.01 and 0.1%, and sulfate concentrations in leachates between 0.5 and 4 mg/L.

Figure 2 shows a weak relationship between nickel content of the samples and nickel release rates. The flat part of the relationship below a nickel content of about 0.035% (350 mg/kg) is due to non-detectable nickel concentrations (<0.0001 mg/L). The highest nickel release rates are for ultramafic rocks with elevated nickel content (dark symbols). The single ultramafic point below the main trend implies that nickel release shows a different relationship to total nickel than troctolitic or anorthositic rocks though only three points define a possibly steeper line for ultramafic rocks. It is also apparent that nickel release is lowest for anorthositic rocks. This suggests that the olivine content of the rocks is important because this rock type has lower olivine content by definition. Olivine leaching is implied as a source of leachable nickel.

Figure 3 shows that copper release rates and copper content are possibly correlated but there is considerable scatter. It appears that anorthositic rocks shower higher release rates than troctolitic rocks, but there several exceptions. The variation of mineralogical content in terms of chalcopyrite to pyrrhotite ratio was also considered as a factor, but it does not explain the observed scatter.

Figure 4 shows that cobalt was rarely detected and therefore this level of analysis did not detect a relationship between cobalt content and cobalt release rates. Low level analyses have been used to evaluate nickel and cobalt concentrations below the reporting limit. Results are discussed below in Section 3.2.3.

Relationships between metal release rates and sulfur content were not apparent.

3.3.3 Reactive Waste Rock

Reactive waste rock in the current definition includes Duluth Complex igneous rocks with sulfur content greater than 0.05%, Virginia Formation from the footwall and sedimentary rock xenoliths.

Graphs showing concentration results are provided in Appendix F.

Duluth Complex Rocks

Results obtained to date were very similar to the non-reactive waste rock samples in terms of overall leachate chemistry. The leachates were pH neutral to slightly alkaline and dominated initially by sodium and bicarbonate. Sodium concentrations have decreased and calcium has become dominant. Sulfate was an important cation. The pH of leachates decreased and then in many cases has flattened. The range of leachate pHs was wider than for non-reactive waste rock and the lowest values were between 6.5 and 7 compared to mostly above 7 for non-reactive waste rock. Trends in metal concentrations were similar to those seen for non-reactive waste rock though detections of some parameters were more frequent (for example, zinc).

Virginia Formation and Sedimentary Hornfels

Three samples of Virginia Formation (plus one as a duplicate) and five samples of sedimentary hornfels are being tested. The two samples containing 75th and 90th percentile sulfur concentrations (4.1 and 5.1%, respectively) produced acidic leachate (pH 4.3). The duplicate of the 75th percentile sample showed a similar result. The 25th percentile (1.9%) sample has not generated acidic leachate (pH 7.5). This sample contained detectable carbonate (0.09%) which may explain the lack of acidity. The decrease in pH after about 3 weeks of testing was accompanied by an increase in sulfate to over 100 mg/L, followed by initially rapid then slow decrease. The decrease in pH was accompanied by increase in Al, Be, Ca, Co, Cu, Fe, Pb, Mg, Mn, Ni, Se and Zn. In terms of absolute concentrations, Fe and sulfate were the main ions. Concentrations of Co, Ni and Zn exceeded 0.1 mg/L, and Ni approached 10 mg/L. The decrease in pH resulted in decreased release of Sb.

One sample of sedimentary hornfels (26027-740-745+26043-1501-1506) showed a decrease in leachate pH reaching a low value of 5.7 on the decreasing trend. This sample showed no detectable carbonate (<0.05% C) and had the 85th percentile sulfur concentration (2.47%). The 75th percentile (1.74%) also had no detectable carbonate and was showing a decreasing pH trend and a relatively low leachate pH (6.7) compared to other reactive rock samples. Other sedimentary hornfels samples contained detectable carbonate and lower sulfur concentrations. The slightly depressed pH for the 85th percentile sample was accompanied by slightly elevated sulfate and increasing Co (0.03 mg/L), Fe (0.2 mg/L), Mn (0.06 mg/L), Ni (0.2 mg/L) and Zn (0.06 mg/L). Other samples of sedimentary hornfels did not show evidence of accelerated leaching.

Rate Relationships

Figure 1 shows that the relationship between sulfur content and sulfate release is continuous from lower to higher levels of sulfur. For sulfur concentrations between 0.05 and 0.1%, the trend is linear, but between 0.2% and 0.5%, sulfate release shows a narrow range. Above 0.5%, sulfate release increases though the trend is flatter than at lower sulfur levels. The higher sulfate releases for the three Virginia Formation and one sedimentary hornfels samples are linked to the decrease in pH.

The sulfate release trend appears to reflect the shift in mineralogy between the chalcopyrite-dominated non-reactive waste rock to the mostly pyrrhotite-dominated reactive waste rock. The trend is counter-intuitive since it would be expected that trend would steepen for pyrrhotite-rich samples since pyrrhotite is generally considered to be more reactive than pyrrhotite. Other factors may be important such as the size and degree of crystal formation.

The relationship between nickel in the rock and nickel release is more scattered with the reactive and non-reactive waste rock samples are combined. Qualitatively, the ultramafic samples mostly show the highest nickel release but there is no relationship between nickel content and nickel release. There is some indication that the base level of nickel leaching increases as the nickel content increases as shown the upward trend in the base of the scatter for troctolitic rock samples. The increase in scatter for nickel release compared to the non-reactive waste rock may be related to the increase in absolute abundance of pentlandite and pyrrhotite. Different nickel leaching rates might be expected depending on which mineral is more abundant though the data do not indicate whether this is the case.

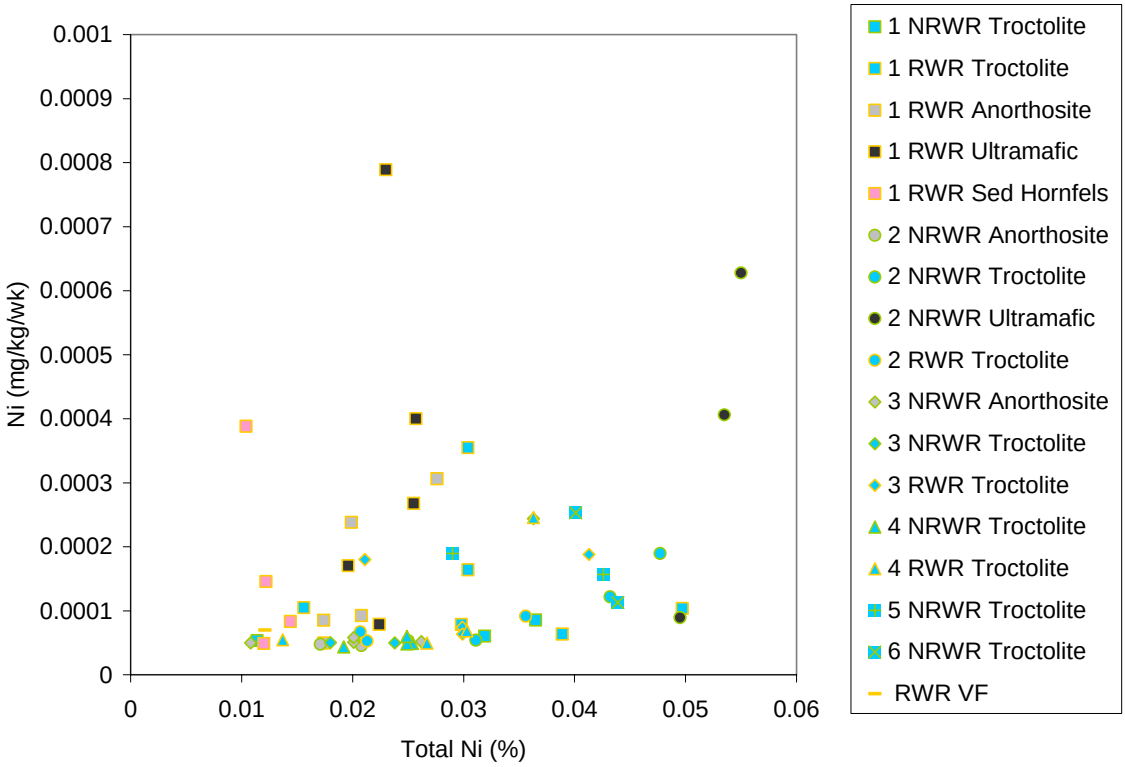

Figure 5: Average Nickel Release Compared to Total Nickel Content of Non-Reactive and Reactive Waste Rock. The y-axis is truncated at 0.001 mg/kg/week. Higher release rates are not shown for Virginia Formation samples. See Figure 1 for explanation of symbols.

Copper and cobalt do not show correlation between concentration in the rock and release rate.

3.3.4 Low Level Analysis

During planning of the program, it was expected based on DNR experience that cobalt and to a lesser extent nickel concentraions would be below the reporting limit of the analytical method (0.0001 mg/L or 0.1 µg/L). The results obtained so far have indicated that cobalt is rarely detected (Figure 4), and that detection of nickel results in the low release rates as shown in Figure 5. Several approaches were considered to address this issue, one of which was to use analytical methods with lower reporting limits. The laboratory indicated that it could report concentrations of 0.05 µg/L by concentrating composite leachate samples.

One round of ultra low level analyses has been completed for leachate samples collected during the first six cycles of the testwork on non-reactive and reactive waste rock. A second round is in progress. Leachates were analysed primarily because cobalt was below the reporting limit. Two samples were selected to quantify undetected nickel only. Each sample was analyzed for both cobalt and nickel using the lower reporting limit of 0.05 µg/L. A summary of detections with respect to the two reporting limits is shown in Table 5. The results how that 183 of the cobalt results were below

10 times the original reporting limit of 0.1 µg/L. Forty-eight results were between the two reporting limits. This is consistent with the expectation that analytical variability is greatest within 10 times the reporting limits. The majority of the remaining samples (72) were below the ultra low level reporting limit. Sixty-three results were between the reporting limits.

Table 5: Results of Low Level Analysis for Cobalt and Nickel

Concentration Range (µg/L)	Number of Co Results	Number of Ni Results
Total Determinations	186	187
Less Than 0.05	72	5
Detected at 0.05	9	0
Between 0.05 and 0.1	54	7
Between 0.5 and 0.1	48	85
Greater than 0.5	3	90

The results obtained have been considered in two ways:

- Do the short term release rates indicated by the results improve understanding of the correlation between the cobalt and nickel content of the rock and the release of these elements?
- Does the improved detection of nickel and cobalt allow a regression relationship to be developed between these parameters that will allow cobalt to be predicted from nickel?

The above comparison demonstrated that the majority of the cobalt concentrations below the routine reporting limits were also below the ultra low level reporting limits but Figure 6 shows that the low level reporting limit cobalt concentrations in the first did not result in improved understanding of the relationship between cobalt in the rock and cobalt release.

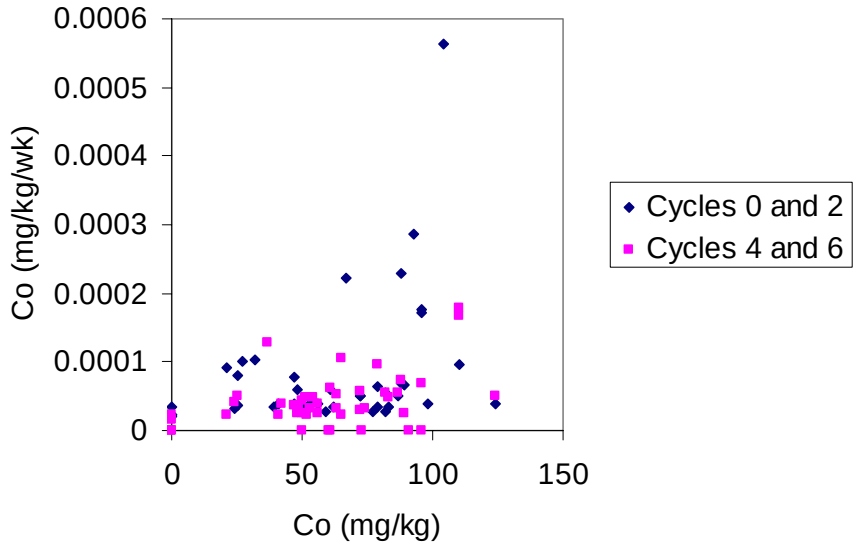

Figure 6: Short Term Cobalt Release Using Ultra Low Level Results Compared to Total Cobalt Content of Waste Rock.

Figure 7 shows a similar relationship for nickel. This graph shows a similar general effect to Figures 2 and 5 in that base nickel release increase as nickel content of the rock increases, but definitaton of the base level is not improved below 0.03%.

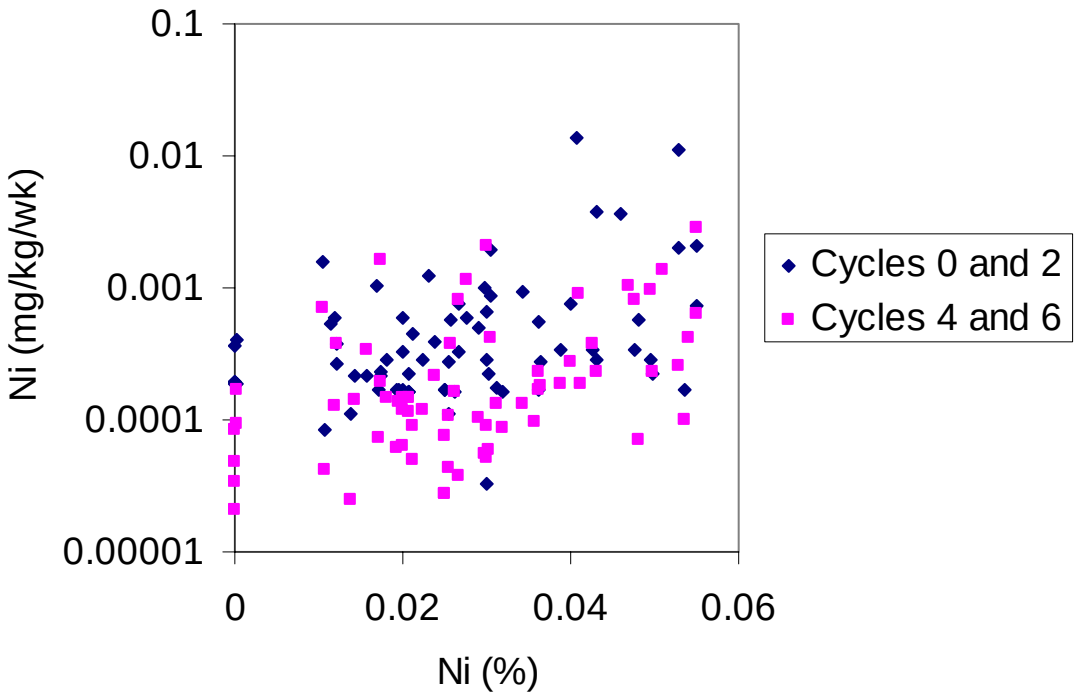

Figure 7: Short Term Nickel Release Using Low Level Results Compared to Total Nickel Content of Waste Rock.

Finally, detectable cobalt and nickel concentrations were compared to determine if cobalt could be predicted from nickel. Figure 8 shows that nickel and cobalt concentrations are correlated and the correlation coefficient is low (0.54) but statistically significant. However, this level of correlation is not reliable to predict cobalt release from nickel release.

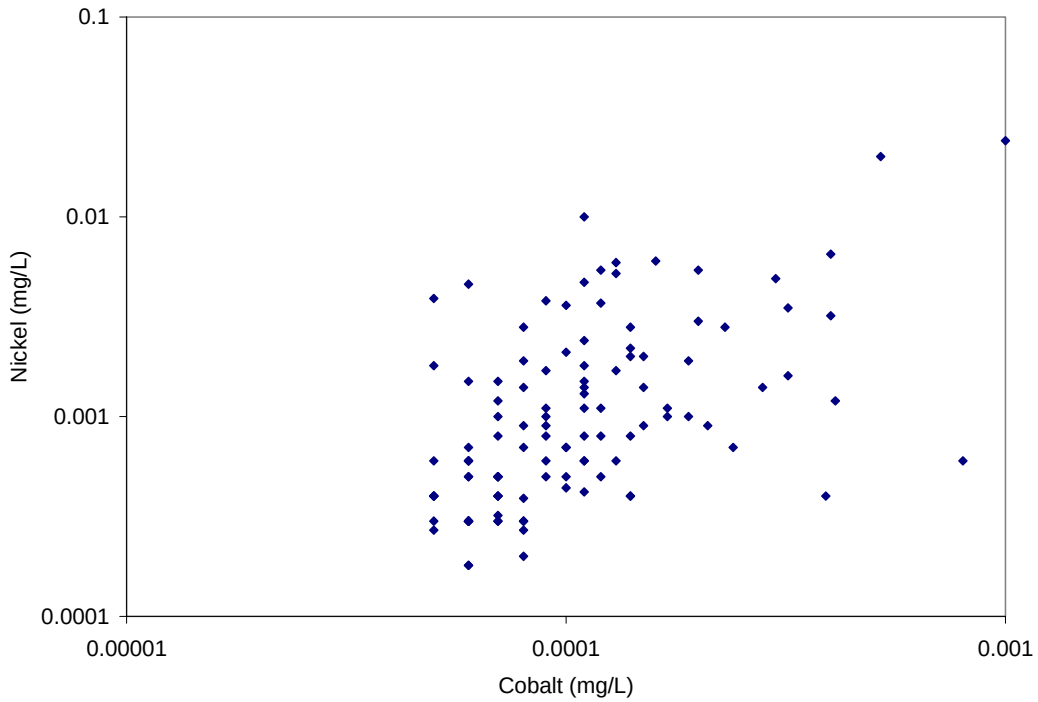

Figure 8: Comparison of Cobalt and Nickel Concentrations Obtained From Ultra-Low Level Analyses.

3.4 Size Fraction Dissolution Experiments

Size fraction experiments consisting of dissolution tests on four fractions (-1/4"+10 mesh, -10+35 mesh, -35+100 mesh and -100 mesh) are being performed on five samples. Four of these samples are classified as reactive. One of these samples is Virginia Formation. One non-reactive troctolitic rock sample is being tested. Analysis of the individual size fractions (Table 6) shows that sulfur concentrations tended to be greatest in the coarsest and finest size fractions.

Direct comparison of leachate chemistry for the four samples is not appropriate because the two finest samples are being leached using 2.7 L/kg of rock, whereas the two coarse fractions are leached at the rate of 0.3 L/kg. Therefore, release rates must be compared to see differences between fractions. Graphs showing selected results are provided in Appendix G. Average rates were calculated using the relatively stable portion of the leachate chemistry trend. Results for each size fraction are shown for sulfate, copper and nickel (Figure 9).

A general trend of increasing release rates as particle size decreases was shown for these parameters. The sulfate tend is the strongest with only a few samples showing decreases compared to the next coarsest fraction. The increase in sulfate was about an order of magnitude for all samples. Since the range of particle sizes in these experiments spanned about an order of magnitude, the results are consistent with the increase in surface area resulting from the decrease in particle size. For example, the approximate specific areas of particles in the coarsest fractions is about 3 m^2/kg compared to greater than 22 m^2/kg in the finest particle size.

Table 6: Characteristics of Size Fraction Samples

Sample ID (Drill Hole, Footage)	HC or DNR Cell	Fraction	Waste Type	Geological Unit	Rock Type	Target S Percentile or Level	S	Cu	Ni	Co
							%	%	%	mg/kg
00-361C(310-320)	HC 81	-1/4"+10#	Reactive	1	Anorthositic	P50	0.19	0.016	0.021	44
	HC 82	-10+35#					0.16	0.014	0.020	44
	L1	-35+100#					0.18	0.016	0.021	46
	L2	-100#					0.24	0.016	0.024	54
00-334C(640-660)	HC 83	-1/4"+10#	Reactive	1	Troctolitic	P25	0.10	0.032	0.051	84
	HC 84	-10+35#					0.07	0.028	0.051	82
	L3	-35+100#					0.06	0.026	0.046	77
	L4	-100#					0.07	0.025	0.054	92
00-369C(305-325)	HC 85	-1/4"+10#	Reactive	3	Troctolitic	P95	0.25	0.038	0.031	57
	HC 86	-10+35#					0.23	0.031	0.027	54
	L5	-35+100#					0.25	0.032	0.027	57
	L6	-100#					0.30	0.033	0.034	64
00-367C(290-310)	HC 87	-1/4"+10#	Non-reactive	4	Troctolitic	NR-0.05	0.04	0.019	0.024	46
	HC 88	-10+35#					0.04	0.019	0.023	43
	L7	-35+100#					0.04	0.017	0.028	55
	L8	-100#					0.08	0.019	0.031	56
00-364C(210-229)	HC 89	-1/4"+10#	Reactive	20	Virginia	P75	3.64	0.016	0.018	23
	HC 90	-10+35#					3.87	0.016	0.019	24
	L9	-35+100#					3.39	0.014	0.017	22
	L10	-100#					3.55	0.011	0.017	23

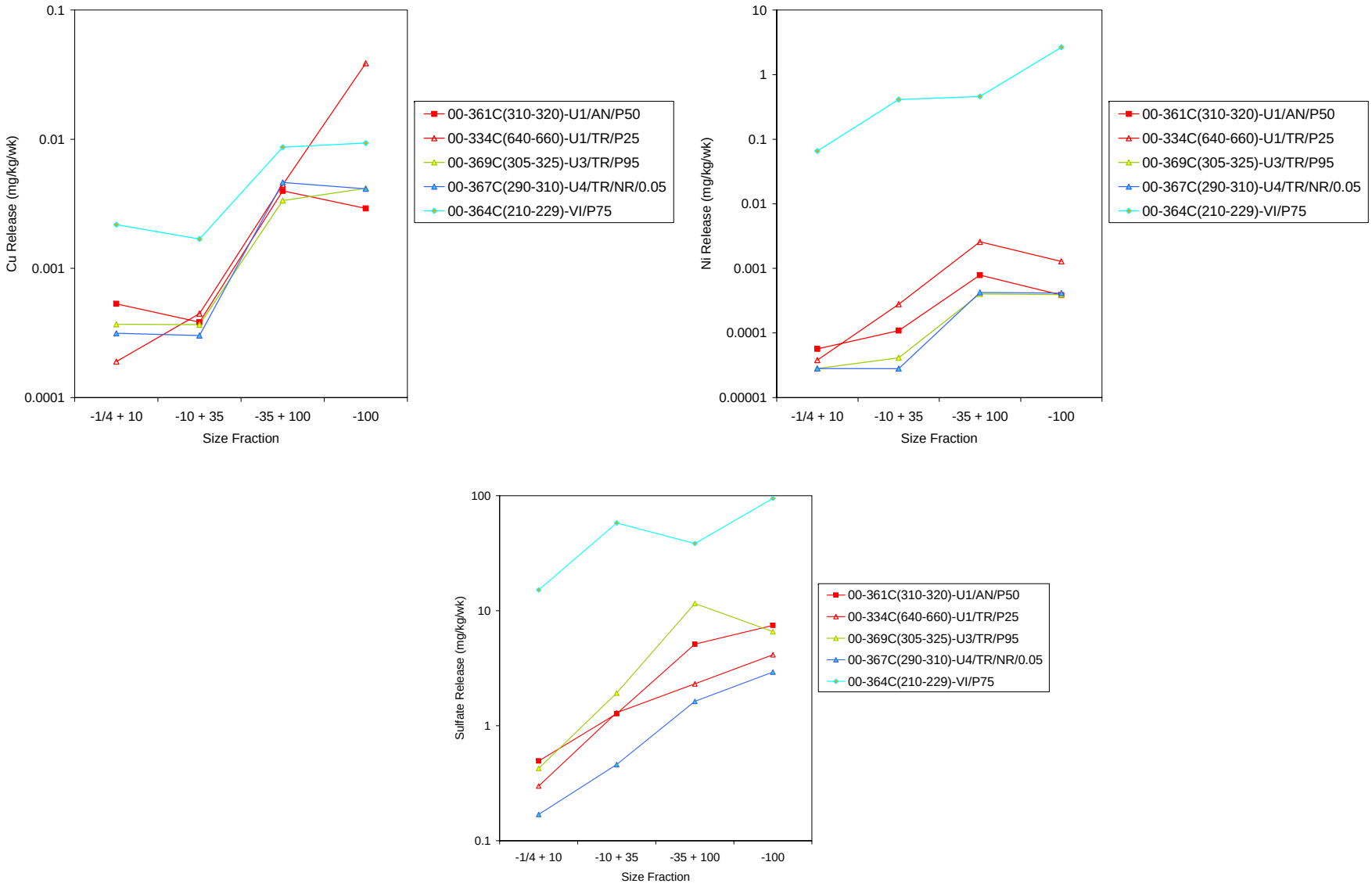

Figure 9: Size Fraction Average Release Rates.

4 Recommended Program Modifications

The current program was designed to form the basis for waste management planning water chemistry predictions. Samples were selected to evaluate all possible variables that could reasonably be used as a basis for waste categories, including:

- Igneous unit in the Duluth Complex at the NorthMet Project;
- Major rock types in each igneous unit;
- Sulfur concentrations on a rock types and igneous unit basis;
- Metal concentrations on a rock type and igneous unit basis; and
- Particle size.

These variables can be used as the basis for block modeling and ultimately to define operational waste rock management plans.

The results obtained from more than 26 weeks of testing have shown that:

- Sulfur concentration in the rock is the dominant variable strongly controlling oxidation rates. This is consistent with the DNR's research which has shown that sulfur concentration is the main variable controlling the degree of pH depression observed and the time frame for pH depression.
- Sulfide mineralogy may be an important controlling factor on overall oxidation rates. Low sulfur rocks show a different relationship to sulfur content of rock with higher sulfur. The difference be may be due to the distribution of sulfide mineral types (particularly pyrrhotite and chalcopyrite) and/or differences in the occurrence of the sulfide minerals. This factor is linked to sulfur content and is therefore not an independent variable.
- Igneous unit does not appear to be important variable, except for hornfels and Virginia Formation.
- Igneous rock type is not a controlling variable for sulfide oxidation but is possibly important in terms of metal leaching. The greater proportion of olivine in ultramafic and troctolitic rock types appear to be important as a source of leachable nickel particular in rocks containing low sulfur concentrations and low proportions of pentlandite.
- The Virginia Formation and Sedimentary Hornfels have been shown to be reactive.
- Particle size has been demonstrated to be an important factor in controlling oxidation and metal release rates, as expected.

Based on these results, the number of samples under test should be reduced to eliminate consideration of igneous unit as a variable, and primarily focus on the variation in sulfur content of various rock types. Maintaining rock type as a variable will also allow low level metal leaching effects to be evaluated. Furthermore, DNR's experience shows that changes in leachate chemistry will occur very slowly over many years. Early results for the testwork conducted on rock samples from the NorthMet project have shown that leachates from the testwork can be expected to evolve slowly in the same fashion and therefore that a smaller set of samples should be continued to confirm that results remain consistent with DNR's experience.

The following approach was used to select tests for continuation.

For non-reactive rock, three samples of each of the three categories (NR1, NR2 and NR3) were selected for each rock type from the Duluth Complex units containing the majority of expected tonnage.

- For ultramafic rock, this indicates that all three samples under test will be continued.
- For troctolitic rock, samples were selected from Units 1, 2 and 3. Nine candidate samples were available and three were selected using random numbers.
- For anorthositic rock, samples were selected from Units 2 and 3. Six candidate samples were available and three were selected using random numbers.

For reactive rock, three samples were selected from each of the rock types troctolitic, anorthositic, ultramafic, and sedimentary hornfels and two samples were selected from Virginia Formation primarily to capture low, mid-range and elevated sulfur concentrations. The selections were then adjusted to characterize a range of nickel and copper leaching rates.

The distribution of non-reactive and reactive rock samples selected in terms of sulfur, copper, nickel and cobalt content and their respective release rates are shown in Figure 10. The total number of samples proposed for continuation is 24. The characteristics of the samples are provided in Table 7.

It is proposed that the five DNR Reactor Tests on -100 mesh test material continue to provide a direct comparison with existing long term DNR tests. Since the relationship of leaching rates to particle size has been documented, the tests on coarser material should not continue.

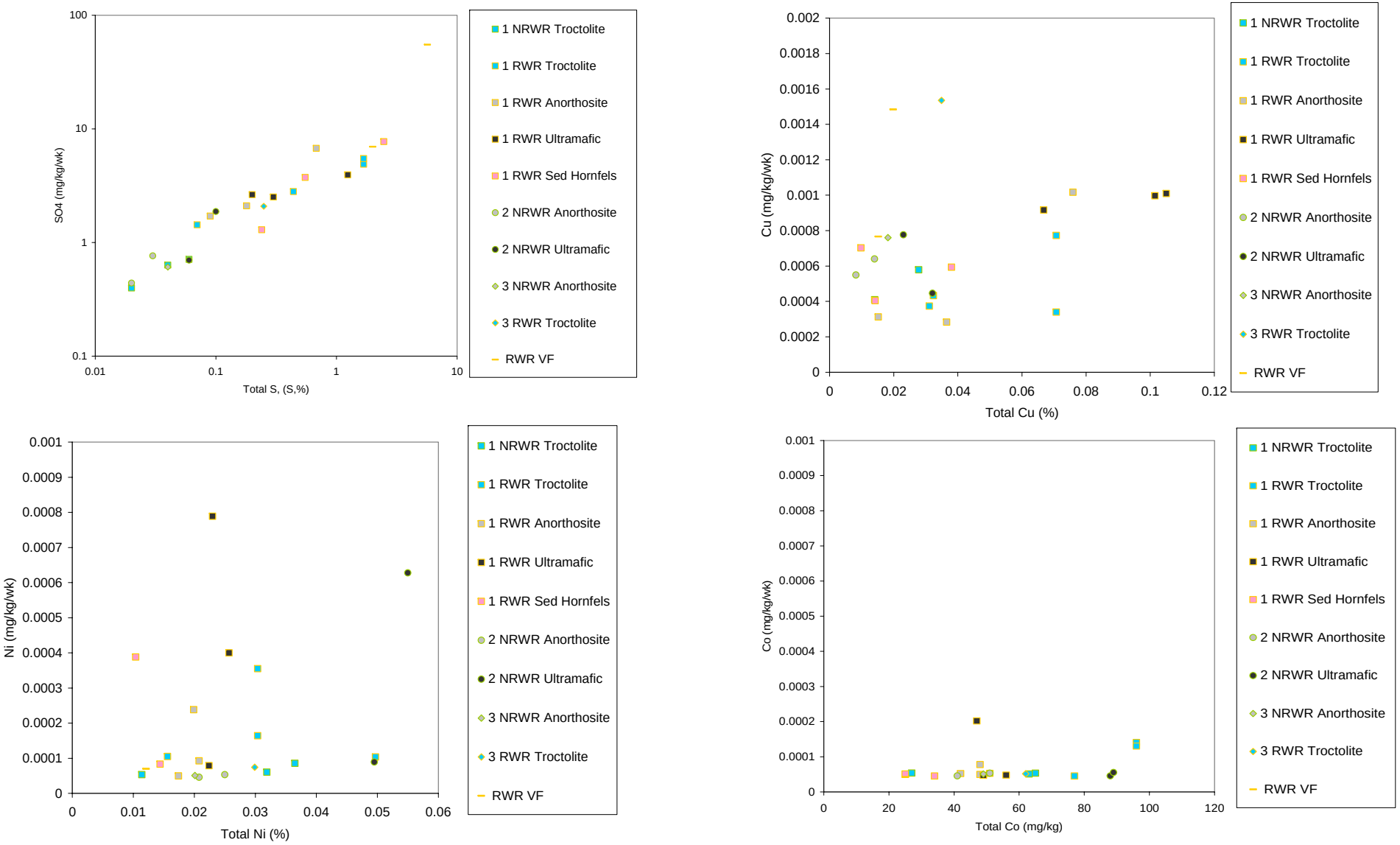

Figure 10: Distribution of Samples Proposed for Continuation.

Table 7: Characteristics of Samples Proposed for Continuation

HCT ID	Waste Type	Geological Unit	Rock Type	Target S Percentile or Level	Sample ID	Cu	Ni	S	Co
						%	%	%	mg/kg
2	Reactive	1	Anorthositic	P50	00-361C-310-320	0.0152	0.0208	0.18	42
4	Reactive	1	Anorthositic	P95	00-343C-240-250	0.0759	0.0199	0.68	48
9	Reactive	1	Anorthositic	P25	99-320C-830-850	0.0365	0.0174	0.09	48
5	Reactive	1	Sedimentary Hornfels	P10	26030-1047-1052	0.0098	0.0104	0.24	25
7	Reactive	1	Sedimentary Hornfels	P50	00-340C-990-995	0.0142	0.0144	0.55	34
11	Reactive	1	Sedimentary Hornfels	P85	26027-740-745+26043-1501-1506	0.04	0.02	2.47	48
13	Non-reactive	1	Troctolitic	NR-0.03	00-340C-595-615	0.0141	0.0365	0.04	65
14	Non-reactive	1	Troctolitic	NR-0.05	00-334C-580-600	0.0278	0.0319	0.06	63
74	Non-reactive	1	Troctolitic	NR-0.01	26029-815-825	0.0055	0.0114	0.02	27
10	Reactive	1	Troctolitic	P100	00-340C-765-780	0.0707	0.0304	1.68	96
15	Reactive	1	Troctolitic	P25	00-334C-640-660	0.0311	0.0497	0.07	77
18	Reactive	1	Troctolitic	P90	00-327C-225-245	0.0324	0.0156	0.44	51
20	Reactive	1	Troctolitic	P100	00-340C-765-780	0.0707	0.0304	1.68	96
22	Reactive	1	Ultramafic	P80	00-326C-680-685	0.105	0.023	0.30	47
23	Reactive	1	Ultramafic	P85	00-357C-535-540	0.1015	0.0257	0.20	56
25	Reactive	1	Ultramafic	P95	99-317C-460-470	0.0668	0.0224	1.24	49
26	Non-reactive	2	Anorthositic	NR-0.01	00-366C-185-205	0.0082	0.0208	0.02	41
28	Non-reactive	2	Anorthositic	NR-0.05	99-320C-165-175	0.014	0.025	0.03	51
36	Non-reactive	2	Ultramafic	NR-0.03	26055-940-945	0.0321	0.0495	0.06	88
39	Non-reactive	2	Ultramafic	NR-0.05	26098-145-148.5+00-337C-105-110	0.023	0.055	0.10	89
37	Non-reactive	3	Anorthositic	NR-0.03	00-368C-125-145	0.0182	0.0201	0.04	49
38	Reactive	3	Troctolitic	P95	00-369C-305-325	0.0349	0.0299	0.25	62
62	Reactive	20	Virginia	P25	00-361C-737-749	0.0152	0.0121	2.00	25
64	Reactive	20	Virginia	P90	00-337C-510-520	0.0198	0.0168	5.68	32

5 Conclusions

RS53A presents results from about 28 weeks of humidity cell testing on samples of waste rock from the NorthMet Project. Based on the dataset, it can be concluded:

- The initial results are comparable to the early stages of DNR's long term research work. Samples of Duluth Complex rock initially produce non-acidic leachate regardless of sulfur content.
- The Virginia Formation and Sedimentary Hornfels xenoliths are more reactive and in some cases have generated acidic leachates. Elevated leaching of metals such as nickel and cobalt occurs under acidic conditions as expected.
- Mineralogical assessment has shown that rock with very low sulfur content ("non-reactive" category) contains both chalcopyrite and pyrrhotite, with the former being dominant. As sulfur content increases, pyrrhotite dominates.
- The dominant variable controlling initial major element chemistry is sulfur content (sulfur primarily as sulfide). A strong correlation between average sulfur content of the rock and sulfate released by oxidation is apparent throughout the range of sulfur concentrations observed. The slope of the correlation change is not constant and may be related to mineralogical variations.
- Rock type is a secondary variable for the Duluth Complex rocks. It does not appear to control overall oxidation rates but affects nickel leaching according to the olivine content of the rock which decreases from ultramafic to troctolitic and anorthositic rock types.
- Oxidation rates and metal leaching does not vary systematically according to igneous unit in the Duluth Complex.
- Ultra low level analysis of cobalt and nickel has shown the cobalt leaching occurs at very low levels but leaching cannot be correlated with cobalt content of the rock.
- Dissolution testwork on samples of rock crushed to different size fractions has shown the oxidation rate increases as particle size decreases and the increase is consistent with the increase in estimated surface area of particles for smaller particle sizes.
- The findings obtained to date indicate that not all of the testwork needs to be continued. A reduction in the program has been recommended primarily to continue evaluation of sulfur content and rock type as significant variables.

6 References

SRK Consulting, 2006. Waste Rock and Lean Ore Geochemical Characterization Plan NorthMet Project, Minnesota. Report prepared for PolyMet Mining Corporation. May 2006.

Appendix A
Waste Rock and Lean Ore Geochemical Characterization Plan

Waste Rock and Lean Ore Geochemical Characterization Plan NorthMet Project, Minnesota

Prepared for

PolyMet Mining Corporation

Prepared by

SRK Consulting
Engineers and Scientists

May 2006

Waste Rock and Lean Ore Geochemical Characterization Plan NorthMet Project, Minnesota

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1 Introduction

1.1 Background

PolyMet Mining (PolyMet) is proposing to develop the NorthMet Project (Dunka Road Project of US Steel) near Babbitt, Minnesota. As a part of the Minnesota Department of Natural Resources (MDNR) "Permit to Mine" process a complete "mine waste characterization" will be required (Minnesota Rules Chapter 6132.1000). This document describes the plan developed for selection and testing of waste rock samples for the NorthMet Project, and the context for interpretation of the results.

The issues associated with waste rock at the NorthMet are expected to include acid rock drainage (ARD) and leaching of some heavy metals. The latter in particular are expected to include nickel and cobalt both of which do not require acidic conditions to be mobilized at elevated concentrations.

The specific objectives of this program include:

- Refinement of preliminary waste rock management criteria developed by PolyMet and MDNR.
- Development of mass-loading rates for input into water quality predictions for impact assessment and mitigation design.

1.2 Geological Setting

The NorthMet Deposit is located in the intrusive Duluth Complex of northern Minnesota. Disseminated copper-nickel-iron sulfides (chalcopyrite, cubanite, pentlandite and pyrrhotite) with associated platinum group element (PGE) mineralization will be extracted from several igneous stratigraphic horizons.

In the vicinity of the NorthMet deposit, the Duluth Complex intruded and assimilated the Virginia Formation, which consists of argillite and greywacke with minor interbeds of siltstone, graphitic argillite, chert, and carbonate. This formation is the stratigraphic footwall of the NorthMet deposit, but also occurs as xenoliths ("inclusions") within the deposit.

1.3 Agency Consultation and Design Process

This document was developed in consultation with staff from the Minnesota Department of Natural Resources (MDNR). The consultation included the following steps:

- December, 2004. PolyMet submitted a draft "Work Plan for Geochemical Characterization of Rock and Concentrator Flotation Tailings". The plan was presented to MDNR representatives.
- January 31 and February 1, 2005. Meetings were held by teleconference between SRK and MDNR representatives to further discuss the variables potentially affecting water chemistry from waste stockpiles.

- March 17, 2005. MDNR requested additional information on the tonnages of the major units and rock types, and the distribution of sulfur and minerals.
- March 28, 2005. PolyMet provided the requested information.
- April 12, 2005. MDNR provided a sample selection matrix. This matrix was accepted by PolyMet and is the basis for the selection of samples described in this document.
- May 15, 2005. MDNR provided a design for specific testwork.
- May 17, 2005. MDNR provided a design for specific testwork.
- June 6, 2005. A draft of this sampling plan was submitted to MDNR.
- June 15, 2005. MDNR provided comments on the draft plan.
- June 22, 2005. SRK provided responses and discussion of the MDNR comments in a letter to MDNR which were discussed during a teleconference on June 27, 2005.
- July 5, 2005. SRK provided results of candidate samples selected for kinetic testing to in a memorandum to MDNR.
- July 13, 2005. MDNR provided comments on the July 5, 2005 SRK memorandum.
- July 15, 2005. SRK provided clarification on sample selection in a memorandum to MDNR.
- July 20, 2005. MDNR notified SRK and PolyMet that kinetic testing on the majority of waste rock samples could be initiated. It was recognized that analysis of a few candidate samples was ongoing.
- August 4, 2005. MDNR Provided recommendations for lean ore characterization.
- August 29, 2005. As requested by SRK, MDNR provided additional rationale for the recommendations on lean ore sampling selection.
- September 14, 2005. Lean ore sample selection was further discussed during a conference call which were provided the basis for completion of this plan.

The plan was fully implemented in October 2005.

This document has been prepared to conclude the design process and seek MDNR approval of PolyMet's plans to respond to the waste rock characterization component of requirements under Minnesota Rules 6132.1000.

1.4 Organization of this Document

This document describes:

- Section 2. Design basis for the program.
- Section 3. Sample selection. This section describes the methods used to select samples from the NorthMet Project drill hole database.

- Section 4. Analytical methods. This section describes methods used to analyse solids and leachates.
- Section 5. Use of the results in the context of water chemistry predictions.

1.5 Acknowledgements

The following individuals cooperated in the preparation of this plan:

- John Borovsky, Barr Engineering Company.
- Stephen Day, SRK Consulting.
- Paul Eger, MDNR.
- Jennifer Engstrom, MDNR.
- Steve Geerts, PolyMet.
- Don Hunter, PolyMet.
- Kim Lapakko, MDNR.
- Richard Patelke, PolyMet.
- Jim Scott, PolyMet.

1.6 Analytical Laboratues

The following laboratories are performing the procedures described in this document (contact names for each laboratory are shown):

- ALS Chemex, North Vancouver, British Columbia – solids analysis listed in Section 4.1.1 (Bill Anslow).
- Optical – PolyMet (Richard Patelke).
- Sub-Optical Lab – McSwiggen and Associates (Pete McSwiggen).
- Canadian Environmental and Metallurgical Inc (CEMI), North Vancouver, British Columbia - kinetic testing (Rik Vos).
- Cantest Inc.. Vancouver, British Columbia - Kinetic test leachate analysis (Richard Jornitz).

2 Characterization Design

2.1 Background

Sample selection for this project was based on the December 2004 PolyMet geologic and assay database (assembled by PolyMet) and the February 2005 block model by Dr. Phil Hellman of Hellman & Schofield. Ultimately, the rock characterization data from these tests will be linked to the mine plan through this database and block model. Current and anticipated future geochemical data collection from drilling is described in documents submitted to MDNR by PolyMet on August 23, 2004 and September 15, 2004.

2.2 Design Basis

Based on discussions between SRK and MDNR on January 31 and February 1, 2005, the following critical variables were identified that potentially could affect drainage quality from waste rock stockpiles:

- Sulfur content.
- Sulfide mineral type.
- Rock type.
- Fragment particle size.

Other important variables include, mineral content, mineral grain size, mineral chemistry, and mode of mineral occurrence.

Sulfur content is considered the primary factor affecting the potential for acid generation and metal mobility based on existing research conducted by MDNR. At higher sulfur concentrations, it appears that acid generation starts earlier and results in lower pHs resulting in increased metal leaching. At lower sulfur concentrations, acid generation is not expected to occur, but sulfur content is correlated with metal content and is therefore expected to be related to metal release.

Sulfide mineral type can be important in terms of rate of reaction and metal release. For example, it is well know that pyrrhotite is more reactive than pyrite. Chalcopyrite and pentlandite are sources of copper and nickel, respectively in drainage. However, since the dominant sulfide mineral in the waste rock appears to be pyrrhotite (based on distribution of metal content) and the commodity sulfide minerals (chalcopyrite, pentlandite and cubanite) are expected to be present at low concentrations, sulfide mineral type was not considered as a primary variable for sample selection. As described below, concentrations of copper, nickel, cobalt and zinc were used as secondary factors for sample selection which is expected to capture variations in sulfide mineralogy. Lean ore characterization is considered separately. All samples are being characterized to evaluate assumptions about the mineralogical occurrence of the important metals.

All rock types in the Duluth Complex are variants of troctolite and to a lesser extent ultramafic rocks. Carbonate minerals are absent or occur at very low concentrations in this rock type. Therefore, the variation in silicate content of these rocks is considered to be an important variable controlling drainage pH.

MDNR also considered that igneous layer in the intrusive complex may be a significant variable. because the reactivity of the minerals may be different in each of the layers. SRK and PolyMet did not agree with this position based on evidence from MDNR's past testwork. Nonetheless, this variable has been carried through the sampling design.

Finally fragment particle size is an important factor because it controls exposure of the reactive minerals and the overall surface area available for reaction.

2.3 Sampling Matrix for Waste Rock

A sampling matrix for waste rock characterization (Table 1) was developed by MDNR, SRK and PolyMet through a series of discussions and exchange of relevant data. Table 1 shows how the main variables have been translated to a sampling design. The table also provides estimates of the tonnages of each major rock type within each unit. Reading from left to right, the columns in the table show the following:

- Unit. This refers to the stratigraphic igneous layers in the complex (number 1 to 7). Unit 20 refers to the footwall of the deposit composed of Virginia Formation and localized igneous intrusions
- Rock Type. This refers to a generalized rock description in the associated unit.
- Estimated Rock Tonnages. These tonnages indicate the estimated amounts of each rock type within each layer and therefore their relative importance. The categories were developed by MDNR and PolyMet to indicate rock with sulfur less than 0.05%[1] ("non-reactive"), sulfur greater than 0.05% but not likely ore grade ("reactive"), and rock with marginal ore grade (lean ore).

Selection of samples for each unit and rock type combination was based on sulfur concentrations in order to develop correlations between reactivity and bulk characteristics such as sulfur and metal content. This provides a basis for water chemistry predictions using bulk characteristics, prediction of waste management criteria (based on sulfur content and metal content) and ultimately for the selection of easily-measured parameters that can be used for waste management during mining (see Section 5.2 below, for additional discussion).

[1] Note that the non-reactive classification is a temporary criterion which will be refined by this testwork program.

Table 1: Matrix for Sample Selection in Waste Rock Types

Unit	Rock Type	Non-Reactive	Reactive	Approximate sulfur contents																		Reactive Rock
				Non-reactive[1]			Reactive[1]								Lean Ore[1]							P size
		M. tons	M. tons	NR1	NR2	NR3	P10	P25	P50	P75	P80	P85	P90	P95	P100	P10	P25	P50	P75	P80	P85	P90	P95	
1	Anorthositic	0.57	0.99				0.08	***0.1***	***0.15***	***0.29***				***1.09***	1.09	0.11	0.26	***0.36***	***0.93***				***1.95***	***4***
1	Gabbroic	0	0.68				0.06	0.07	0.07	0.08				0.19	0.5	0.09	0.1	0.18	0.37				0.5	
1	Sedimentary hornfels	0	1.6				***0.08***	***0.35***	***0.69***	***2.2***	2.32	***2.81***	3.38	3.5	3.78	0.34	1.37	***1.58***	1.76				***4.91***	
1	Troctolitic	17.2	40.1	***0.01***	***0.03***	***0.05***	0.06	**0.07**	***0.1***	0.18	***0.21***	0.26	***0.34***	***0.62***	***1.97***	0.08	***0.15***	***0.24***	0.42	0.49	***0.55***	0.68	***0.98***	***4***
1	Ultramafic	0.21	1.1				0.07	***0.08***	0.1	0.13	***0.2****	***0.3****	***0.5****	***0.8****	1.35	0.07	0.09	***0.14***	***0.33***	0.55	***0.56***	0.65	***0.81***	
1	Vein	0.055	0.022				0	0	0	0				0										
2	Anorthositic	2.4	0.56	***0.01***	***0.03***	***0.05***	0.06	0.06	0.08	0.11				0.19		0.09	0.12	0.17	0.21				0.25	
2	Basalt inclusions	0.28	0													0	0	0	0				0	
2	Gabbroic	0	0.082				0	0	0	0				0										
2	Troctolitic	16.9	9.7	***0.01***	***0.03***	***0.05***	0.06	0.06	0.07	0.09	0.09	0.1	0.11	0.12	0.25	0.05	0.07	0.12	0.18	0.2	***0.22***	0.26	***0.32***	
2	Ultramafic	0.38	0.25	***0.03***	***0.04***	***0.05***	0	0	0	0				0		0.04	***0.05***	0.07	0.12	***0.14***	0.16	0.19	***0.23***	
3	Anorthositic	9.4	1.2	***0.01***	***0.03***	***0.05***	0.07	0.08	0.08	0.12				0.14		0.06	0.08	0.12	0.27				0.38	
3	Fault-Breccia	0	0.055				0	0	0	0				0										
3	Gabbroic	0.2	0.72				0.09	0.15	0.18	0.27				0.29		0	0	0	0				0	
3	Noritic	0.11	0																					
3	Sedimentary hornfels	0.38	0.46				0.12	1.42	1.67	1.97				2.22		1.66	1.77	1.85	2.43				3.26	
3	Troctolitic (augite)	41.2	12.5	***0.01***	***0.03***	***0.05***	0.06	0.06	***0.08***	0.1	0.11	***0.12***	0.14	***0.19***	0.36	0.05	0.13	0.19	***0.32***	0.35	0.45	0.48	***0.52***	***4***
3	Ultramafic	0.24	0.071				0	0	0	0				0		0	0	0	0				0	
4	Anorthositic	0.16	0.055				0	0	0	0				0										
4	Sedimentary hornfels	0	0.055				0	0	0	0				0										
4	Troctolitic	7	2.2	***0.01***	***0.03***	***0.05***	0.06	***0.07***	0.09	***0.18***	0.19	0.19	***0.48***	***0.92***	1.53	0.09	0.15	0.22	0.47				***1.52***	***4***
4	Vein	0.055	0																					
5	Troctolitic	2.5	2	***0.01***		***0.05***	0.07	0.09	0.11	0.16				0.22	0.22	0.12	0.17	0.26	0.37				***0.45***	
6	Chlorite	0.16	0													0	0	0	0				0	
6	Fault-Breccia	0.055	0.055				0	0	0	0				0										
6	Troctolitic	6.8	0.59	***0.02***	***0.04***		0	0	0	0				0		0.01	0.01	0.03	0.07				***0.19***	
6	Ultramafic	0	0.11				0	0	0	0				0										
7	Ultramafic	0.082	0													0	0	0	0				0	
20	Troctolitic	0	0.27				3.18	3.45	3.76	4.3				4.31										
20	Virginia	0	10.4				0.59	***1.25***	2.98	***4.15***	4.49	4.85	5.07	***6.06***	7.45	0	0	0	0				0	***4***

Notes:
1. Non-reactive rock categories (lower, medium and higher sulfur contents).
2. Sulfur percentiles for reactive rock types calculated by PolyMet are shown. "*" indicates approximate percentiles.
3. Grey – sampling plan.
4. Bold and italic – samples obtained.
5. Bold border – duplicate cell in operation.

Separate sulfur ranges were defined for the “non-reactive” and “reactive” categories. Within the non-reactive category, three sulfur concentrations were selected to represent the lowest possible sulfur concentration in the rock type (typically 0.01%), the upper limit to this category (0.05%) and an intermediate (0.03%). The need for samples was identified for relatively abundant rock types contributing more than 1,000,000 tons (ie more than 1% of the rock mass). For the reactive category, sample selections were based on sulfur concentration percentiles calculated by Polymet. Again, rock types contributing more than 1,000,000 tons were identified for testing.

Sulfide mineral variability was considered by preferring samples with higher concentrations of Ni and Co, Cu and Zn.

The search for suitable samples included all candidate sulfur values indicated in grey shading Table 1.

2.4 Characterization of Lean Ore

Lean ore is defined as rock containing grades of commodity minerals below that at which processing can currently be justified, but may eventually be processed if project economics improve. In terms of sulfur content, lean ore mainly overlaps the “reactive” waste rock category and also some to degree the non-reactive catagory but contains higher nickel and copper concentrations than waste rock. Therefore, the main difference between lean ore and waste rock is expected to be in the mineralogical occurrence of sulfur. In waste rock, sulfur occurs mainly as iron sulfide but in lean ore the commodity minerals pentlandite, chalcopyrite and cubanite are expected to be more important. This has important implications for drainage chemistry. In particular, oxidation and leaching of pentlandite is expected to release more nickel than pyrrhotite due to the higher Ni/Fe ratio in pentlandite. This limits the co-precipitation of nickel with iron oxhydroxides during oxidation.

The overall approach to selection of samples was similar to that of waste rock.

3 Sampling

3.1 Methods

3.1.1 Sample Selection for Non-Reactive and Reactive Waste Types

Samples were selected from the NorthMet Project drill hole database. The following sequence was used to select samples:

1. The database was reduced to rock core. Only diamond drill holes were considered because this method of drilling results in the best characterised samples by eliminating mixing of intervals and allowing accurate logging.
2. Separate datasets were created for “waste rock” (non-reactive and reactive) and “lean ore”.
3. Due to ease of recovery from the core archives, core obtained by PolyMet drilling was preferred. In cases where suitable samples could not be found from this source, US Steel drillholes were considered.
4. To select samples, 20-foot moving-averages for consistent rock types were calculated for all parameters. This interval was selected to reduce the potential for characterizing small-scale local heterogeneity, and provide a sample width that relates to mining scale. The wider interval also ensured that sufficient sample is available for all the different tests. In practice, the need to obtain samples with specific unit, rock type and ulphur combinations resulted in sampling intervals of 5 to 20 feet. In three cases, sample intervals from two different locations had to be selected to yield sufficient material for testing.
5. Within each rock type and unit combination, samples were identified based on a ulphur range straddling the desired concentrations. The range was calculated as between the midpoint concentration to the two nearest ulphur concentrations. For example, a target concentration of 0.03% between 0.01% and 0.05% could be selected from intervals with ulphur concentrations between 0.02% and 0.04% (with a preference for 0.03%). This approach was necessary to provide candidate samples in all ranges and incorporate the need to target higher metal concentrations for tests.
6. If more than one candidate interval was identified, the interval with the highest nickel and cobalt concentrations was selected with a primary focus on nickel content. If several samples could still be chosen, copper and zinc concentrations were considered. This approach resulted in selection of samples with nickel concentrations approaching the highest values in the database. For example, the highest nickel concentration in a non-reactive sample is 0.043% compared to a maximum value of 0.043% in the database. Similarly, the highest reactive nickel concentration in the database is 0.042%, and the highest sample is 0.038%.

The sample lists (Table 2 and 3) were re-generated several times due to limitations of core availability. The first pass attempted to use PolyMet core only. Subsequent lists included US Steel core.

Lean ore sample selection was performed in two passes. The first pass targeted samples containing ulphur concentrations at the P95 level. The second pass selected samples for the remaining samples. To ensure that delays did not occur in the second pass, a large number of intervals were selected to ensure that at least one sample would be available for each target ulphur concentration.

3.1.2 Core Recovery

Samples were recovered from archived core boxes by PolyMet personnel. Since the core had previously halved for initial analysis, the remaining half core was quartered for this sampling.

All required intervals were sampled on a continuous basis and placed in plastic bags for shipment to the analytical laboratory.

3.2 Sample List

The resulting lists of waste rock samples and lean ore samples shipped to the laboratory are shown in Tables 2 and 3.

Table 2: List of Non-Reactive and Reactive Samples Selected

Unit	Rock Type	S Percentile – Non-Reactive and Reactive Waste Rock[1]	Selected Interval				Weighted Average Characteristics Calculated from Drill Hole Database				
			DDH	From	To	Length	S	Ni	Co	Cu	Zn
				Feet	Feet	Feet	%	%	mg/kg	%	mg/kg
1	Anorthositic	P25	99-320C	830	850	20	0.12	0.015	35.8	0.035	72
1	Anorthositic	P50	00-361C	310	320	10	0.16	0.018	36.0	0.013	51
1	Anorthositic	P75	00-361C	345	350	5	0.33	0.026	53.0	0.038	68
1	Anorthositic	P95	00-343C	240	250	10	0.67	0.020	38.0	0.069	69
1	Sedimentary Hornfels	P10	26030	1047	1052	5	0.17	0.006	10.0	0.006	34
1	Sedimentary Hornfels	P25	26061	1218	1233	15	0.43	0.011	19.3	0.012	193
1	Sedimentary Hornfels	P50	00-340C	990	995	10	0.62	0.012	24.0	0.031	148
1	Sedimentary Hornfels	P75	00-340C	965	974.5	15	1.49	0.015	23.7	0.031	249
1	Sedimentary Hornfels	P85	26043	1501	1506	5	2.76	0.024	36.0	0.037	344
1	Sedimentary Hornfels	P85(a)	26027	740	745	5	2.59	0.035	67.0	0.051	48
1	Troctolitic	NR1	26029	815	825	10					
1	Troctolitic	NR2	00-340C	595	615	20	0.03	0.025	49.8	0.013	71
1	Troctolitic	NR3	00-334C	580	600	20	0.05	0.024	52.0	0.024	83
1	Troctolitic	P25	00-334C	640	660	20	0.08	0.038	63.8	0.024	102
1	Troctolitic	P50	00-347C	795	815	20	0.09	0.034	70.8	0.035	94
1	Troctolitic	P80	00-350C	580	600	20	0.22	0.027	55.5	0.041	115
1	Troctolitic	P90	00-327C	225	245	20	0.44	0.015	50.5	0.032	73
1	Troctolitic	P95	00-371C	435	440	5	0.65	0.014	32.0	0.036	54
1	Troctolitic	P100	00-340C	765	780	15	1.72	0.022	78.0	0.064	69
1	Ultramafic	P25	00-357C	335	340	5	0.08	0.015	35.0	0.026	68
1	Ultramafic	P80	00-326C	680	685	5	0.21	0.016	33.0	0.085	82
1	Ultramafic	P85	00-357C	535	540	5	0.26	0.026	34.0	0.095	62
1	Ultramafic	P90	99-318C	725	735	10	0.44	0.011	23.0	0.032	45
1	Ultramafic	P95	99-317C	460	470	10	1.10	0.018	33.0	0.056	86
2	Anorthositic	NR1	00-366C	185	205	20	0.01	0.018	35.0	0.008	47
2	Anorthositic	NR2	00-366C	230	240	10	0.02	0.014	32.0	0.011	51
2	Anorthositic	NR3	99-320C	165	175	10	0.04	0.023	46.0	0.012	63
2	Troctolitic	NR1	99-318C	250	270	20	0.02	0.022	42.3	0.011	64
2	Troctolitic	NR2	00-373C	95	115	20	0.03	0.036	64.0	0.019	84
2	Troctolitic	NR3	00-373C	75	95	20	0.05	0.031	55.3	0.020	75
2	Troctolitic	P50	00-357C	110	130	20	0.07	0.024	53.5	0.029	88
2	Troctolitic	P80	99-320C	315	330	15	0.09	0.017	39.3	0.026	65
2	Troctolitic	P95	00-369C	335	345	10	0.16	0.021	43.0	0.046	68
2	Ultramafic	NR1	00-368C	460	465	5	0.03	0.042	70.0	0.033	98
2	Ultramafic	NR2	26055	940	945	5	0.04	0.037	69.0	0.025	88
2	Ultramafic	NR3	26098	145	148.5	3.5	0.05	0.037	76.0	0.014	112

Table 2: List of Non-Reactive and Reactive Samples Selected (Cont'd).

Unit	Rock Type	S Percentile – Non-Reactive and Reactive Waste Rock[1]	Selected Interval				Weighted Average Characteristics Calculated from Drill Hole Database				
			DDH	From	To	Length	S	Ni	Co	Cu	Zn
				Feet	Feet	Feet	%	%	mg/kg	%	mg/kg
2	Ultramafic	NR3(a)	00-337C	105	110	5	0.05	0.043	71.0	0.023	116
3	Anorthositic	NR1	00-334C	30	50	20	0.01	0.022	47.0	0.009	64
3	Anorthositic	NR2	00-368C	125	145	20	0.03	0.016	38.0	0.015	62
3	Anorthositic	NR3	00-368C	20	40	20	0.04	0.010	25.8	0.022	47
3	Troctolitic	NR1	00-366C	35	55	20	0.01	0.019	45.3	0.005	50
3	Troctolitic	NR2	00-334C	110	130	20	0.03	0.028	57.3	0.012	73
3	Troctolitic	NR3	00-347C	155	175	20	0.04	0.015	49.5	0.015	67
3	Troctolitic	P50	00-347C	280	300	20	0.08	0.018	45.3	0.035	61
3	Troctolitic	P85	00-326C	60	70	10	0.12	0.031	51.0	0.034	91
3	Troctolitic	P95	00-369C	305	325	20	0.27	0.030	51.5	0.037	57
4	Troctolitic	NR1	00-367C	50	65	15	0.02	0.015	38.7	0.010	59
4	Troctolitic	NR2	00-367C	260	280	20	0.04	0.024	53.8	0.018	78
4	Troctolitic	NR3	00-367C	290	310	20	0.04	0.021	41.3	0.018	64
4	Troctolitic	P25	00-370C	20	30	10	0.07	0.010	37.0	0.016	67
4	Troctolitic	P75	00-369C	20	30	10	0.14	0.021	39.0	0.043	60
4	Troctolitic	P90	00-367C	170	175	5	0.48	0.023	45.0	0.034	54
4	Troctolitic	P95	00-367C	395	400	5	0.92	0.028	76.0	0.080	96
5	Troctolitic	NR1	26064	44	54	10	0.01	0.035	62.0	0.009	58
5	Troctolitic	NR3	26064	264	269	5	0.04	0.017	42.0	0.005	56
5	Troctolitic	NR3(a)	26064	146	156	10	0.05	0.032	67.0	0.031	78
6	Troctolitic	NR1	26056	110	125	15	0.02	0.034	66.7	0.025	85
6	Troctolitic	NR2	26056	135	153	18	0.04	0.037	62.7	0.032	89
20	Virginia	P25	00-361C	737	749	15	1.91	0.015	22.3	0.039	225
20	Virginia	P75	00-364C	210	229	19	4.11	0.018	26.8	0.017	872
20	Virginia	P90	00-337C	510	520	10	5.12	0.016	29.5	0.019	492

Notes:

1. Designation (a) indicates that the sample will be mixed with the previous sample in the list to make a composite for testing.

Table 3: List of Lean Ore Samples Selected

Unit	Rock Type	S Percentile	Selected Interval				Weighted Average Characteristics Calculated from Drill Hole Database				
			DDH	From	To	Length	S	Ni	Co	Cu	Zn
				Feet	Feet	Feet	%	%	mg/kg	%	mg/kg
1	Anorthositic	P50	99-320C	400	405	5	0.32	0.06	63	0.07	68
1	Anorthositic	P75	00-331C	255	260	5	0.95	0.05	38	0.16	42
1	Anorthositic	P95	26027	616	626	10	2.31	0.054	77.0	0.144	75
1	Sedimentary Hornfels	P50	26058	704	715	11	1.59	0.04	40	0.08	175
1	Sedimentary Hornfels	P95	26062	993	998	5	5.49	0.049	81.0	0.153	166
1	Sedimentary Hornfels	P95	26026	565	568	3	3.83	0.033	70.0	0.074	88
1	Troctolitic	P25	00-326C	250	265	15	0.15	0.04	49	0.07	73
1	Troctolitic	P50	00-340C	910	925	15	0.33	0.04	50	0.15	77
1	Troctolitic	P85	00-331C	190	210	20	0.54	0.04	41	0.17	43
1	Troctolitic	P95	00-340C	725	745	20	1.06	0.045	116.0	0.114	95
1	Ultramafic	P50	00-326C	495	505	10	0.14	0.06	91	0.06	128
1	Ultramafic	P75	00-344C	630	635	5	0.33	0.05	54	0.13	82
1	Ultramafic	P85	00-330C	275	280	5	0.60	0.06	123	0.10	86
1	Ultramafic	P95	00-344C	515	520	5	1.10	0.057	72.0	0.090	130
2	Troctolitic	P85	99-318C	325	330	5	0.21	0.05	63	0.14	90
2	Troctolitic	P95	00-340C	380	385	5	0.30	0.04	56	0.13	84
2	Ultramafic	P80	00-326C	225	235	10	0.13	0.06	87	0.10	122
2	Ultramafic	P95	00-361C	240	245	5	0.20	0.085	103.0	0.017	130
3	Troctolitic	P75	00-367C	495	500	5	0.28	0.05	52	0.16	58
3	Troctolitic	P95	26049	358	362	4	0.55	0.058	73.0	0.151	60
3	Troctolitic	P95	26030	291	296	5	0.50	0.055	78.0	0.149	104
4	Troctolitic	P95	00-367C	400	405	5	1.52	0.031	79.0	0.116	86
5	Troctolitic	P95	26056	302	312	10	0.45	0.04	50	0.11	67
6	Troctolitic	P95	26142	360	365	5	0.15	0.043	50.0	0.109	80

4 Sample Preparation and Analysis

4.1 Solids Characterization

4.1.1 Sample Preparation

Samples were shipped to Canadian Environmental and Metallurgical Inc (CEMI) as whole core pieces. The following procedures were used for sample preparation:

- Upon arrival, each sample was weighed and its weight recorded. Specific gravity was determined.
- Each sample was crushed to pass a 0.25 inch screen.
- A 3 kg split of crushed sample was split and saved for humidity cell testing. This provides sufficient sample for duplicate kinetic testing if needed
- A 200 g split was used for solids characterization
- A 50 g split was saved for additional archive and petrographic analysis

4.1.2 Chemical Analysis

A split of each sample was submitted for an extensive suite of analysis, as follows:

- Acid base accounting (total S, carbonate, paste pH). Sulfur as sulfate is not needed because previous work shows that sulfur occurs exclusively as sulfide. Carbonate rather than neutralization potential is being determined because neutralization potential determinations on rocks containing reactive silicates are ambiguous and do not reflect field capacity to neutralize acid. Carbonate indicates the field reactive component of acid neutralization potential.
- 27 elements by ICP scan following four-acid (nitric-hydrochloric-perchloric-hydrofluoric) digestion (near total) (0.5 g).
- 34 elements by ICP scan following aqua regia (nitiric-hydrochloric acid) digestion (0.5 g).
- Whole rock oxides (0.5 g).

These methods were selected to provide continuity with the earlier work and will therefore allow the samples selected to be compared with the existing project database.

Method detection limits are provided in Appendix A.

In addition, 200 g of all samples were split into four size fractions (-100+270 mesh, -35+100 mesh, -10+35 mesh and -0.25”+10 mesh) for analysis of total S and 27 elements by four acid digestion.

Chemical analysis of all samples was completed prior to implementation of kinetic testing.

4.1.3 Optical Analysis

Two pieces of typical core from each interval sampled were taken for preparation of polished thin sections to confirm the rock type and quantify reactive minerals. Optical mineralogy reports will indicate mineral types, mineral abundance, grain sizes and mineral occurrence.

4.1.4 Sub-Optical Analysis

Sub-optical analysis included determination of the trace element content of major minerals on selected samples using microprobe analyses.

4.2 Kinetic Test Methods

4.2.1 Humidity Cell

Humidity cell testing is being performed using ASTM Procedure D 5744 – 96 (Reapproved 2001). This procedure was selected for the following reasons:

- Similar procedures have been in use under different names since the late 1980s (e.g. MEND 1991). The results can therefore be evaluated in the context of more than a decade of experience using the procedure.
- It is a standard procedure approved by the ASTM and is therefore defensible as a method.

The ASTM procedure provides some options for varying the test procedure. Appendix B provides a detailed listing of the requirement of the ASTM procedure, options chosen and any variances from the ASTM procedure

4.2.2 MDNR Reactor

To evaluate size fraction effects, four size fractions (-100 mesh, -35+100 mesh, -10+35 mesh and -0.25”+10 mesh) from five samples are being tested using a procedure referred to as the “MDNR Reactor” experiment. The two smallest size fractions are being tested in a specifically designed apparatus designed by MDNR (Appendix C) to contain 75 g. The two coarser fractions are being tested in cells with the same configuration as ASTM Procedure D 5744–96. Details of the construction of the smaller MDNR reactors as provided by MDNR are attached in Appendix C.

For the small reactors, a weekly volume of 200 mL is being used. For the larger samples, the leachate volume is 300 mL.

4.2.3 Leachate Analysis

Leachates from kinetic tests are being analyzed for the parameters indicated in Table 4 every four weeks beginning on the first rinsing cycle (week 0). Every four weeks on weeks 2, 6, 10 etc. the leachates are analysed for a higher level scan to evaluate trends in major elements. Based on experience, testing of non-reactive rock samples with very low sulfur concentrations is expected to

result in very dilute leachates containing low concentrations of the metals of interest. Back-calculation of metal concentrations from other testwork performed by DNR indicates that nickel and cobalt concentrations could be as low as 0.0002 mg/L (200 ng/L) and 0.00001 (10 ng/L), respectively. Quantification of these low metal concentrations is needed to provide reasonably constrained estimates of metals concentrations in waste rock seepage.

A number of different approaches are available to quantify low levels of metals:

- The routine leachate analysis will achieve a detection level of 0.0001 mg/L (100 ng/L). Should concentrations be undetected, detection limits of 50 ng/L can be obtained with additional processing effort using the same routine method.
- Specialist methods can achieve lower detection limits. These are non-routine (for example, evaporation to increase concentrations) and will need to be developed as the need arises. In order to generate a 10 times decrease in detection limit, the samples would need to be concentrated at 1east 10 times. A composite leachate sample would be prepared from several cycles.
- Existing testwork demonstrates that good correlations exist between cobalt and nickel concentrations in leachates. Detectable nickel concentrations can be used to estimate cobalt concentrations if this relationship can be demonstrated.
- The particle size experiments provide a larger surface area and provide greater likelihood that lower concentrations will be detected.
- In the event of undetectable low levels, detection limit values would be used in subsequent calculations. A scale-up methodology will be agreed upon with MDNR to translate non-detectable concentrations to waste rock seepage concentrations. Section 5.2 provides discussion of possible scale-up approaches.

Table 4: List of Parameters for Humidity Cell Leachate Analyses. Concentrations in mg/L except where indicated

Parameter	Limit	Parameter	Limit
pH (standard units)	-	Acidity	1
Conductivity (µS/cm)	1	Alkalinity	1
Chloride	0.2	Sulfate	0.5
Fluoride	0.05	Total Inorganic Carbon	1
ORP (mV)	-		
Dissolved Elements			
Aluminum	0.001	Molybdenum	0.00005
Antimony	0.0001	Nickel	0.0001 (0.00005)[1]
Arsenic	0.0001	Potassium	0.02
Barium	0.0001	Selenium	0.0002
Beryllium	0.0002	Silicon	0.05
Bismuth	0.0002	Silver	0.00005
Boron	0.005	Sodium	0.01
Cadmium	0.00004	Strontium	0.0001
Calcium	0.01	Tellurium	0.0002
Chromium	0.0002	Thallium	0.00002
Cobalt	0.0001 (0.00005)[1]	Thorium	0.0001
Copper	0.0001	Tin	0.0001
Iron	0.01	Titanium	0.0002
Lead	0.00005	Uranium	0.00005
Lithium	0.0002	Vanadium	0.0002
Magnesium	0.005	Zinc	0.001
Manganese	0.00005		

Notes:

1. Low detection limits are available for cobalt and nickel as shown.

4.3 Quality Assurance/Quality Control

To summarize, QA/QC includes the following components:

- Roughly 10% of all solids analyses are performed in duplicate.
- Roughly 10% of all cell and reactor tests are run as duplicates.
- A blank cell and reactor containing no sample is being operated to check for contamination of leachates by construction materials.
- Individual leachate results are reviewed.
- Ion balances on leachate results are reviewed. In general, imbalances of ±10% are considered acceptable. Re-analysis if requested depending on the nature of the imbalance.
- Data trends in kinetic test leachates are analysed to check for anomalies.

5 Analytical Results for Samples Selected

Table 5 compares analytical results for all samples selected with concentrations calculated from individual intervals in the database. This table reflects the final sample selection following review of data to locate potential replacements, and replacement of one sample. The final three columns indicate the difference between calculated and analytical results using:

$$\% \text{ Difference} = \frac{\text{Target- Actual}}{\text{Target}} \times 100\%$$

Sulfur results showed the greatest percentage differences for samples in the non-reactive category. However, these differences reflect small absolute differences (0.01 to 0.02%) resulting from analytical variability near the detection limit. One sample was replaced in this category because it had 0.05% sulphur compared to target of 0.01%.

A few samples in the reactive class had differences between 50% and 100%. Review of the database indicated no suitable replacements.

One sample in the lean ore class showed a difference of 50%.

Based on these results, MDNR, SRK and PolyMet agreed that the sample selections mostly provided good coverage with respect to the targeted sample ranges indicated in Table 2 and that set up of kinetic tests could proceed.

Table 5: Comparison of Weighted Sample Characteristics Used for Sample Selection with Analytical Results for Interval Composites

Unit	Rock Type	Parameter		Percentiles for Sulfur Contents																			
				Non-reactive			Reactive									Lean Ore							
				NR1	NR2	NR3	P10	P25	P50	P75	P80	P85	P90	P95	P100	P10	P25	P50	P75	P80	P85	P90	P95
1	Anorthositic	Target	Total S, %				0.08	0.1	0.15	0.29				1.09	1.09	0.11	0.26	0.36	0.93				1.95
		Test Material	Total S, %					0.09	0.18	0.05				0.68				0.18	0.86				1.83
	Sedimentary hornfels	Target	Total S, %				0.08	0.35	0.69	2.2	2.32	2.81	3.38	3.5	3.78	0.34	1.37	1.58	1.76				4.91
		Test Material	Total S, %				0.24	0.44	0.55	1.74		2.47						1.46					4.46
	Troctolitic	Target	Total S, %	0.01	0.03	0.05	0.06	0.07	0.1	0.18	0.21	0.26	0.34	0.62	1.97	0.08	0.15	0.24	0.42	0.49	0.55	0.68	0.98
		Test Material	Total S, %	0.02	0.04	0.06			0.07		0.19		0.44	0.88	1.68		0.08	0.36			0.42		0.91
	Ultramafic	Target	Total S, %				0.07	0.08	0.1	0.13	0.2	0.3	0.5	0.8	1.35	0.07	0.09	0.14	0.33	0.55	0.56	0.65	0.81
		Test Material	Total S, %					0.08			0.3	0.2	0.72	1.24				0.16	0.34		0.75		1.2
2	Anorthositic	Target	Total S, %	0.01	0.03	0.05	0.06	0.06	0.08	0.11				0.19		0.09	0.12	0.17	0.21				0.25
		Test Material	Total S, %	0.02	0.02	0.03																	
	Troctolitic	Target	Total S, %	0.01	0.03	0.05	0.06	0.06	0.07	0.09	0.09	0.1	0.11	0.12	0.25	0.05	0.07	0.12	0.18	0.2	0.22	0.26	0.32
		Test Material	Total S, %	0.04	0.04	0.06			0.08		0.07			0.18							0.17		0.15
	Ultramafic	Target	Total S, %	0.03	0.04	0.05	0	0	0	0				0		0.04	0.05	0.07	0.12	0.14	0.16	0.19	0.23
		Test Material	Total S, %	0.06	0.06	0.1														0.12			0.06
3	Anorthositic	Target	Total S, %	0.01	0.03	0.05	0.07	0.08	0.08	0.12				0.14		0.06	0.08	0.12	0.27				0.38
		Test Material	Total S, %	0.02	0.04	0.04																	
	Troctolitic (augite)	Target	Total S, %	0.01	0.03	0.05	0.06	0.06	0.08	0.1	0.11	0.12	0.14	0.19	0.36	0.05	0.13	0.19	0.32	0.35	0.45	0.48	0.52
		Test Material	Total S, %	0.02	0.04	0.06			0.06			0.14		0.25					0.28				0.59
4	Troctolitic	Target	Total S, %	0.01	0.03	0.05	0.06	0.07	0.09	0.18	0.19	0.19	0.48	0.92	1.53	0.09	0.15	0.22	0.47				1.52
		Test Material	Total S, %	0.03	0.04	0.04		0.08		0.21			0.51	0.77									1.37
5	Troctolitic	Target	Total S, %	0.01		0.05	0.07	0.09	0.11	0.16				0.22	0.22	0.12	0.17	0.26	0.37				0.45
		Test Material	Total S, %	0.02		0.06												0.23					0.32
6	Troctolitic	Target	Total S, %	0.02	0.04		0	0	0	0				0		0.01	0.01	0.03	0.07				0.19
		Test Material	Total S, %	0.04	0.05																		
20	Virginia	Target	Total S, %				0.59	1.25	2.98	4.15	4.49	4.85	5.07	6.06	7.45	0	0	0	0				0
		Test Material	Total S, %					2		3.79			5.68										

Unit	Rock Type	Parameter		Deviations																			
				Non-reactive			Reactive									Lean Ore							
				NR1	NR2	NR3	P10	P25	P50	P75	P80	P85	P90	P95	P100	P10	P25	P50	P75	P80	P85	P90	P95
1	Anorthositic	Target	Total S, %																				
		Test Material	Total S, %					-10%	20%	-83%				-38%				-50%	-8%				-6%
	Sedimentary hornfels	Target	Total S, %																				
		Test Material	Total S, %				200%	26%	-20%	-21%		-12%						-8%					-9%
	Troctolitic	Target	Total S, %																				
		Test Material	Total S, %	100%	33%	20%			-30%		-10%		29%	42%	-15%		-47%	50%			-24%		-7%
	Ultramafic	Target	Total S, %																				
		Test Material	Total S, %					0%			50%	-33%	44%	55%				14%	3%		34%		48%
2	Anorthositic	Target	Total S, %																				
		Test Material	Total S, %	100%	-33%	-40%																	
	Troctolitic	Target	Total S, %																				
		Test Material	Total S, %	300%	33%	20%			14%		-22%			50%							-23%		-53%
	Ultramafic	Target	Total S, %																				
		Test Material	Total S, %	100%	50%															-14%			-74%
3	Anorthositic	Target	Total S, %																				
		Test Material	Total S, %	100%	33%	-20%																	
	Troctolitic (augite)	Target	Total S, %																				
		Test Material	Total S, %	100%	33%	20%			-25%			17%		32%					-13%				13%
4	Troctolitic	Target	Total S, %																				
		Test Material	Total S, %	200%	33%	-20%		14%		17%			6%	-16%									-10%
5	Troctolitic	Target	Total S, %																				
		Test Material	Total S, %	100%		20%													-12%				-29%
6	Troctolitic	Target	Total S, %																				
		Test Material	Total S, %	100%	25%																		
20	Virginia	Target	Total S, %																				
		Test Material	Total S, %					60%		-9%			12%										

Notes
Target sulfur levels for samples tested in humidity cells (selected by DNR).

Notes:
Greater than 100% deviation for non-reactive samples and between 50% and 100% deviation for reactive and lean ore samples.
Greater than 100% deviation for reactive and lean ore samples.

6 Implementation Schedule

The majority of waste rock testwork and some lean ore test work was implemented beginning August 8, 2005 following approval by MDNR on July 20, 2005. The balance of lean ore testwork was started on October 28, 2005. Based on the agreement between PolyMet and MDNR that 26 weeks will provide sufficient data for initial analysis, and allowing 8 weeks for reporting and quality assurance evaluations of metals results, the timing of 26 weeks of available data is as follows:

- Waste Rock –Early April, 2006.
- Lean Ore – Late June, 2006.

Based on these time frames, waste rock data will be reported in mid-May and lean ore in early August. These reports will contain recommendations for modifications to the test program. Termination of testwork will consider the following factors:

- Observation of stable trends for all monitored parameters. Stable is defined as a either flat or steady decrease in metal release.
- Demonstration that results for tests are similar and results can be grouped. Selected tests represent the groups of tests will be continued to demonstrate stability of trends.
- Similarity of results with previous DNR testwork.

It is recognized based on MDNR long term experience with kinetic testing of waste rock from other locations in the Duluth Complex that the pH of initial leachates may be elevated compared to long term pH, and that this may result in under-estimation of metal release. This factor will be considered when selecting samples for continuation.

7 Use of Data for Water Quality Predictions

7.1 Purpose of this Section

This section of the plan describes how the data obtained from kinetic tests are used as inputs into prediction of water chemistry for the NorthMet Project. Section 5.2 describes how water quality predictions fit into the overall mine planning process. Section 5.3 provides discussion on scaling up data obtained from small lab experiments to full scale site stockpiles and waste dumps.

7.2 General Context to Water Quality Predictions in Mine Planning

The ultimate objectives of geochemical characterization are to obtain data that can be used as inputs to:

1. Waste management planning (for example, is the rock/tailings acid generating and/or metal leaching?); and

2. Impact assessment (what concentrations of metals and other components might leach from rock/tailings?).

Figure 1 illustrates the general flow of data collection to achieve the above objectives. The bulk geological and geochemical characteristics (indicated by the geological and lithogeochemical models) are interpreted in the context of release rates and geochemical waste classification criteria, and are input into waste scheduling. The resulting waste composition allows release rate information to be used in scale-up calculations, which in turn are used to develop water chemistry predictions.

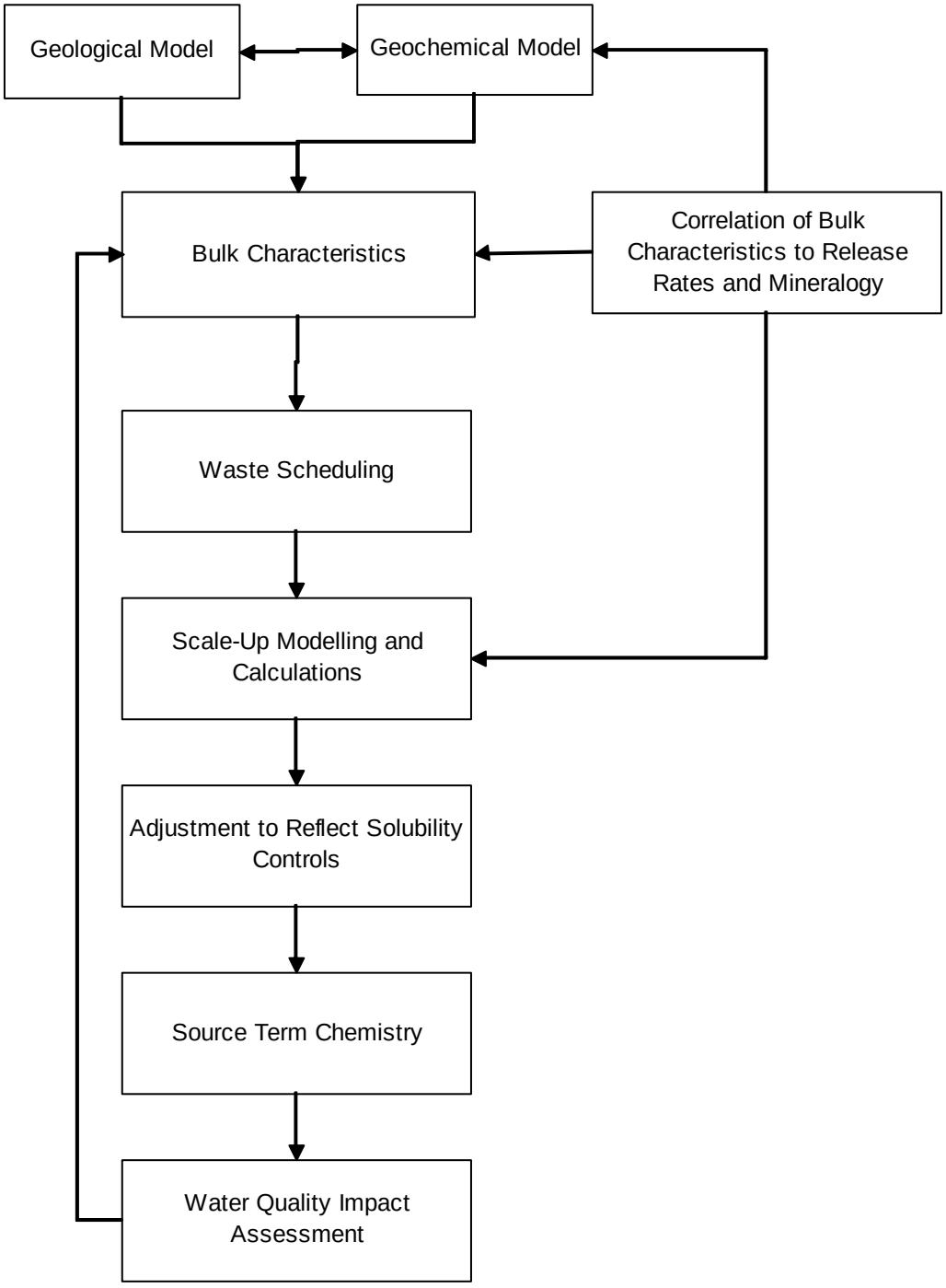

Figure 1: Flow of Information for Water Quality Predictions During Mine Planning

The overall components of geochemical characterization therefore include:

1. Bulk characterization of the rock mass using geological and/or geochemical variables that can be used to predict the waste characteristics for the purpose of waste management planning.
2. Correlation of the characteristics used for bulk characterization with relevant ML/ARD (metal leaching/acid rock drainage) variables and development of criteria based on that correlation (e.g. correlate sulfur content with acid generation and correlate metal leaching rates with bulk metal content).
3. Prediction of contaminant release rates on a mass basis from rock and tailings under various disposal scenarios.
4. Determination of water quality controls (e.g. solubility limits, attenuation effects etc.) for prediction of source term concentrations for individual facilities. Data obtained for this component will be used to adjust water quality predictions obtained from scale-up of laboratory kinetic tests.

All four components are relevant to both objectives and the process is iterative. For example, the last component may indicate parameters that should be used for classification of waste leading to requirements for waste modeling in the first component (Figure 1).

7.3 Approach to Developing Water Quality Predictions

A number of general approaches are available to obtain water quality predictions. These include:

- Theoretical (“First Principals”);
- Site comparisons; and
- Empirical

Discussion of each of these approaches is provided in the following sections.

7.3.1 Theoretical Approach

The theoretical approach involves working from first principals with reaction kinetics and thermodynamics for the processes involved in sulfide oxidation, acid neutralization and metal leaching and attenuation (MEND 2000). There are a number of limitations to this approach which include the difficulty of modeling processes for site-specific conditions. For example, sulfide mineral reactivity can vary widely due to differences in mineral type, occurrence, crystallinity and trace element content. To address these limitations, practitioners typically introduce site-specific calibrations for some processes resulting in predictions that contain empirical aspects. These calibrations may involve actual measurements of oxygen consumption rates, heat generation and seepage chemistry, most of which require that a waste rock dump exists. The ability to make predictions for completely new facilities is therefore limited using purely theoretical approaches (MEND 1995) and should not be pursued further for the NorthMet Project

7.3.2 Site Comparisons

Site comparisons are based on the assumption that mineralogy has a strong influence on water quality, and therefore that comparison of mineral deposits with similar mineralogy is a legitimate approach to making water quality predictions for facilities in the same geological setting.

For example, Caruccio and Ferm (1974) first proposed that paleo-environment is an important factor in determining water quality for coal mines because coal seams formed in salt water environments have higher initial sulfur content and are therefore more prone to generation of acid due to the formation of pyrite during lithification.

Recently, Red Chris Development Co. (2005) compiled data for six porphyry copper mine sites in western Canada and found strong similarities between geographically scattered sites despite variations in host rock geology and climate (for example Figure 2). Porphyry deposits form by interaction of hot water with volcanic or plutonic rocks typically in a sub-volcanic environment. The similarities in drainage chemistry reflected the relatively simple sulfide mineralogy of these deposits and the formation of common alumino-silicate alteration minerals.

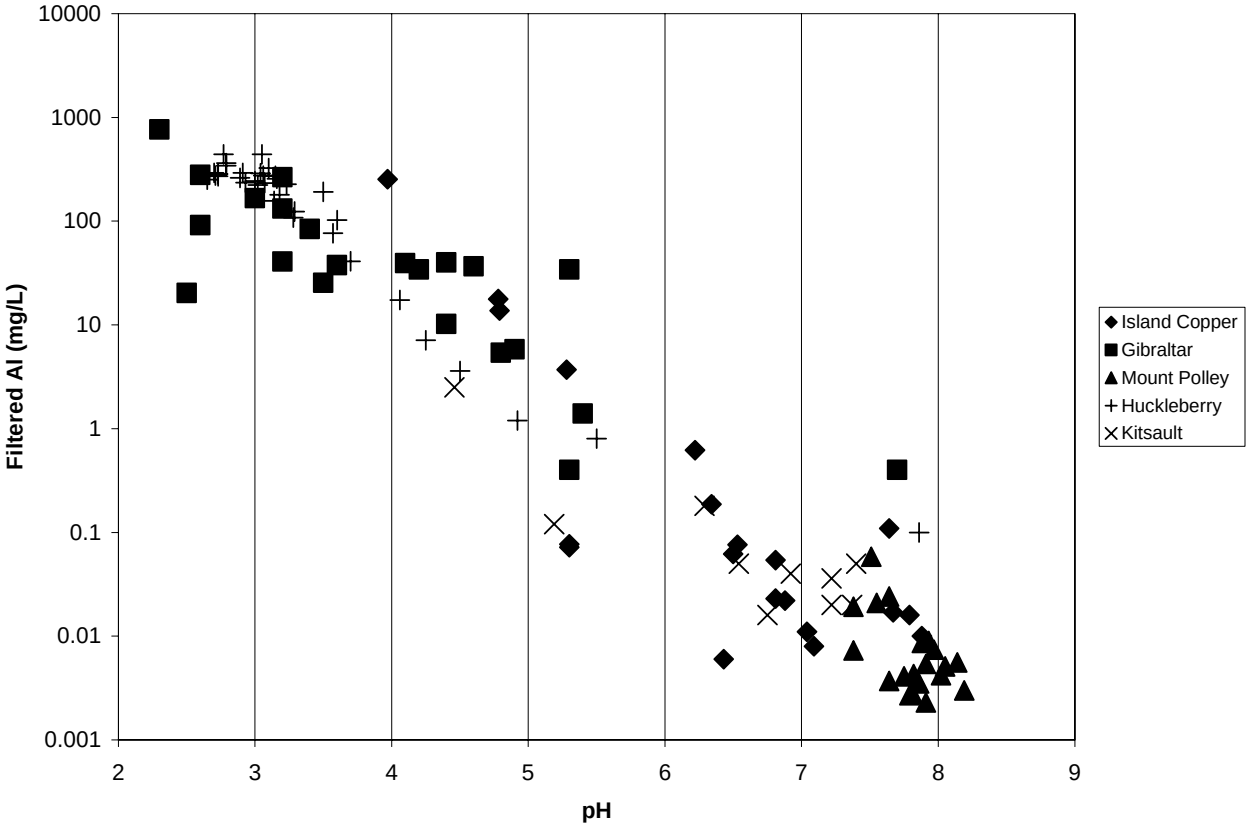

Figure 2: Strong Correlation between Aluminum Concentrations and pH for Porphyry Copper Deposits in Western Canada (Red Chris Development Co. 2004)

A limitation of these comparisons is that geographical proximity does not always guarantee geological similarity because ore forming processes can vary over short lateral and vertical distances, especially as a result of interaction with different rock types. An example of this limitation is seen at the Mount Washington mine site on Vancouver Island where two nearby pits have strongly acidic copper-bearing and non-acidic arsenic-bearing drainages (SRK 2000). The host rock geology is clearly different though for the two pits.

Based on the experience with site comparisons, there are a number of reasons to indicate that mineral deposits within the Duluth Complex can be compared including:

- Uniform, troctolitic to ultramafic composition of the mineralization and metamorphosed siliclastic footwall rocks (Virginia Formation);
- Relatively simple iron sulfide mineralogy; and
- Magmatic rather than hydrothermal mineral deposit formation.

The latter is particularly important because the ore-forming process in the NorthMet Deposit and nearby occurrences have not altered the associated primary silicate minerals. It is concluded therefore that the water quality data collected from nearby full-scale facilities (such as the Dunka Pit Duluth Complex waste rock dumps) and testwork should be factored into the water quality predictions for the NorthMet Project.

7.3.3 The Empirical Method

Introduction

The Empirical Method is also sometimes referred to as “scale-up calculations” because it involves translation of results from small laboratory or field tests to full-scale facilities. The attraction of this approach is that it involves the use of site-specific laboratory and field data, and does not rely on theoretical calculations. The results are transparent and easily explained. However, a significant issue is that the resulting concentrations are typically excessively conservative. This may be attractive for environmental assessment purposes but the resulting predictions may unreasonably over-predict the need for mitigation measures to address potential water quality impacts. A necessary component of the Empirical Method is the adjustment of resulting predictions to reflect basic geochemical controls and experience from other sites.

There are three main steps in the method:

1. Design of a laboratory program to collect rate information;
2. Calculation of concentrations based on rock mixtures, scale-up factors, and hydrological considerations; and
3. Adjustment of calculated concentrations to reflect geochemical constraints indicated by testwork, thermodynamic constraints and experience.

Additional description of these steps is provided in the following sections.

Laboratory Program

The laboratory program is designed to obtain weathering rates, typically expressed as mass of component released per mass of rock per week. Rates are obtained for all rock types and a range of the characteristics for each rock type. Generally, the objective is to obtain rates that can be correlated with bulk characteristics of the rock so that overall rates can be calculated for mixtures. Examples of strong correlations of sulfur content with sulfate release are common and include MDNR's humidity cell data (Figure 3). When good correlations are established, the data can be interpolated between points. For example, in Figure 3, although no rate was specifically measured at a sulfur concentration of 0.9%, it is reasonable to use the overall trend to interpolate a rate. Figure 3 also shows that since the correlation indicates no sulfate release if no sulfur is present, it is also reasonable to extrapolate between 0 and 0.2% sulfur.

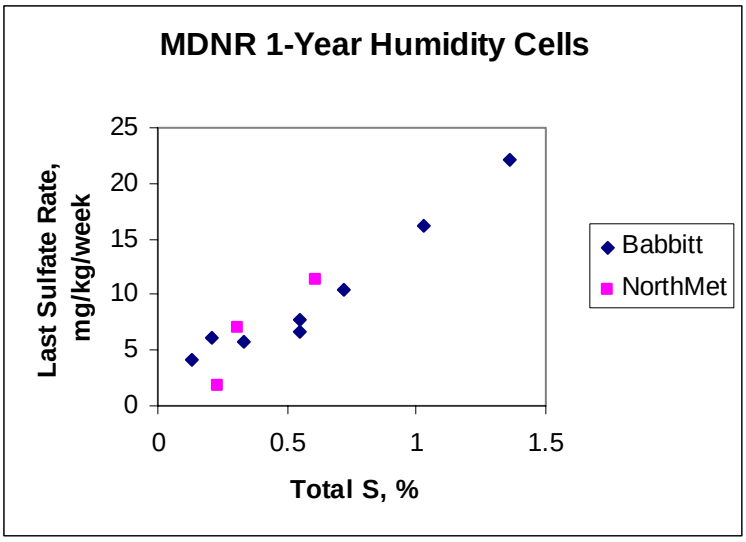

Figure 3: Correlation of Sulfur Content and Sulfate Release for MDNR ASTM Procedure Humidity Cells.

Calculation of Concentrations Using Scale-Up Factors

The purpose of the scale-up calculation is to convert laboratory measured generation rates (R) (for example in mg/kg/week) to seepage concentrations (C) (in mg/L). The scale-up calculations need to consider the rock type mixture, temperature effects, grain size, rock mass, flow path development, and water volume. Pore water concentrations are calculated for each rock type, then mixed according to the proportion of rock types indicated by the mine planners.

Temperature should be considered because oxidation rates decrease as temperatures decreases and vice versa. This correction is typically applied based on the average annual site temperature, and can be calculated using the Arrhenius equation. This equation provides a good approximation of actual

rate decrease observed in laboratory experiments. The laboratory rate (R) is therefore adjusted using a constant factor (k_T) to obtain the adjusted rate (R_a):

$R_a = R.k_T$

This correction should be applied cautiously for reactive materials because the sulfide oxidation reaction is exothermic and will offset cooler site conditions.

The next step is to consider particle size effects. There are two issues to consider:

- Oxidation is a surface area phenomenon. A larger surface area provides a greater reactive surface area, and
- Reactive minerals encapsulated in large rock types do not oxidize at the same rate as exposed reactive particles because oxygen must diffuse through a solid rather than a gas.

Figure 4 illustrates the relationship between particle size and surface area for particles occurring as cubes. The graph shows that below a particle size of 0.1 cm, the available surface area increases exponentially. For larger particles, the area contribution is insignificant. Therefore, a standard humidity cell containing -¼" (0.6 cm) material provides a good representation of the surface area of a rock mixture containing much larger particles. For example, in a typical rock mixture containing 5% by weight finer than this size, the particles finer than 0.6 cm can account for 95% of the surface area.

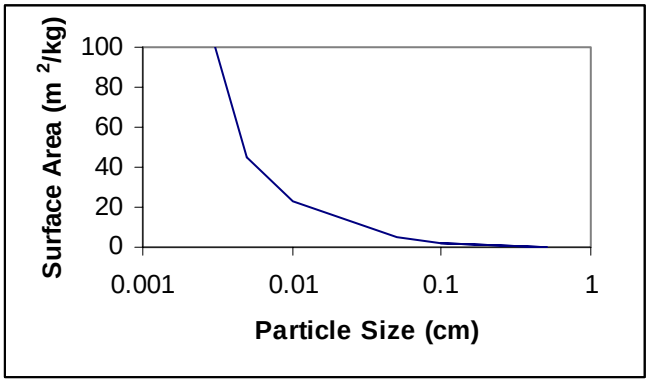

Figure 4: Particle Surface Area as a Function of Particle Size for Cubic Particles

The correction for particle size then becomes a ratio of the fine-grained reactive mass (M_r) to the total mass (M):

$R_a = R.k_T.(M_r/M)$.

The scale up of rate to full scale is then obtained by multiplying by M to obtain:

$R_a' = R.k_T.M_r$ (in mg/week).

R_a' is the scale-up of laboratory rate to field rate for total mass; however, it represents production rather than release because humidity cells are designed to be fully flushed. Under field conditions, the entire rock mass is not flushed due to flow path development. For thin waste rock dumps, flushing is likely to be relatively thorough but as the flow path length increases the degree of flushing decreases (eg Morin 1991; Morin and Hutt 1997). Simple calculations for long flow paths indicate that the proportion flushed may be as low as 20% (Day and Harpley 1992). R_a' can therefore be converted to leached mass (L) by multiplying by a flushed proportion (k_f):

$L = R.k_T.M_r. k_f$ (mg/week).

This leached loading can then converted to a concentration (C) by dividing by the volume of infiltrating water (Q):

$C = R.k_T.M_r.k_f/Q$ (mg/L).

The application of this method to calculation of actual concentrations is illustrated by the following example.

Adjustment of Calculated Concentrations – Example Calculation

Example Dataset

As noted previously, calculation of concentrations using the empirical method often results in unusually high concentrations and it is therefore necessary to evaluate the individual concentrations with consideration of chemical principals and experience from other sites. In order to explain the approach, a recently released dataset from the environmental assessment of a porphyry copper project is used (Red Chris Development Company 2004). For that project, fourteen ASTM-style humidity cells were operated on several rock types which included mineralized quartz diorite and andesitic volcanics and unmineralized siltstones. The resulting calculated average rates from two tests representing a range of conditions are provided in Table 6. The calculation applies to non-acidic drainage.

As an example calculation, concentrations were calculated using an input infiltration rate of 240 mm/year acting on a waste rock dump 50 m high. No temperature correction was applied (k_T), but it was assumed that the bulk of the surface area is contained in 10% of the waste rock and that 20% of the rock is actually flushed.

Evaluation and Adjustment of Major Parameters

The first step in evaluation of the calculated scale-up concentrations is to examine the major parameters (sulfate, calcium, magnesium, etc). An appropriate approach is to enter the data into a thermodynamic equilibrium model (such as MINTEQ, PHREEQE). These models can assist with identifying concentrations that are not supportable thermodynamically. For example, when dissolving common salt in a container of water, only a finite amount can be dissolved after which

any additional salt remains as solid in the bottom of the container. The water is said to be saturated with respect to salt, and the resulting sodium and chloride concentrations in solution can be no greater than when the salt stops dissolving. The reverse is not always true though. It is possible for a solution to be over-saturated with respect to a solid during evaporation. In this case, the energy required to start forming (or nucleating) the first crystals is not available. The thermodynamic models must therefore be used cautiously.

Table 6: Example of Empirical (Scale-up) Calculation of Waste Rock Seepage Chemistry for pH Neutral Drainage Using Humidity Cell Data (from Red Chris Development Co. 2004)

Parameter	Unit	Typical Release for Specific Cells Rates		Calculated Concentrations		P_{95} Concentrations at Other Porphyry Mine Sites (pH>6)
		Major Parameters Low	Major Parameters High	Low	High	
		mg/kg/wk	mg/kg/wk	mg/L	mg/L	mg/L
SO_4		4	73	1633	26978	1526
Mo		0.0004	0.017	0.2	6	0.3
Cu		0.0003	0.0015	0.1	1	2
Pb		0.00005	0.0002	0.018	0.08	0.0002
Zn		0.0004	0.0051	0.2	1.9	0.7
Ni		0.00005	0.0013	0.017	0.50	0.07
Co		0.0001	0.0005	0.020	0.19	0.167
Mn		0.007	0.067	3	25	4
Fe		0.01	0.06	5	23	0.1
As		0.0002	0.0005	0.08	0.20	0.01
Cd		0.0000	0.0001	0.004	0.03	0.002
Ca		3.4	8.8	1254	3252	727
Mg		1.9	1.3	689	477	101
Al		0.040	0.094	15	35	0.1
Na		0.9	32	322	11772	53
K		1.1	0.7	397	275	37
Se		0.0002	0.0089	0.08	3.3	0.2
Si		0.28	0.62	104	229	34
Infiltration	mm/a			240	240	
Density	t/m^3			1.7	1.7	
k_T	-			1	1	
k_f	-			0.2	0.2	
M_r/M	-			0.1	0.1	
Dump Height	m			50	50	

In the case of this dataset, the initial evaluation would allow the following example incompatibilities to be assessed:

- Sulfate and calcium concentrations are much higher in the "high" case than would be expected based on the solubility of gypsum. Since gypsum is well known to form readily, the calcium and sulfate concentrations could be adjusted to reflect precipitation of gypsum.
- Aluminum is not soluble at these levels at neutral pH and could be adjusted to reflect the solubility of basic aluminum sulfates.
- Iron may be present at these concentrations but not under well-oxygenated conditions as would be present in a coarse waste rock pile. Iron hydroxides would be expected to form significantly lowering dissolved iron concentrations.
- Alkalinity was not calculated but can be estimated by assuming a carbonate mineral is present.

Downward adjustment of sulfate obviously impacts the charge balance of the water probably leaving a positive imbalance due to the high sodium and potassium concentrations. While this can be rectified by adding another anion (like chloride), the source of that anion needs to be justified (ie. chloride may not be present in the rock).

In summary, the result of the empirical calculation is a set of concentrations for major ions that typically exceed expected values. This indicates that some products of the weathering reactions remain stored in the rock and are not leached by infiltrating processes. This fact can be applied to adjustment of minor and trace parameters.

Evaluation and Adjustment of Minor and Trace Parameters

Evaluation of these parameters is treated separately because they occur at concentrations that are not a major component of the ion balance and with some exceptions occur at concentrations below limits implied by saturation controls. Copper and manganese concentrations in Table 5 are expected to be close to saturation limits for their carbonates and oxides at neutral pH and can therefore be adjusted to reflect the solubility of these minerals.

Other elements, including for example, lead, zinc, nickel cobalt, cadmium and selenium are predicted to be released at concentrations which seem to be "high" based on experience. In some cases, the concentrations are result of scaling up of detection limit values (for example, arsenic and lead). To refine the predictions for the project in the example, a database of seepage chemistry for other similar porphyry copper mine sites was evaluated. The 95^{th} percentile concentrations from the database are shown in Table 7. Comparison of these concentrations to the calculated concentrations indicated that the calculated low-end lead, arsenic, cadmium and selenium concentrations are higher than the database concentrations, and all calculated high end concentrations are greater than the database concentrations. In other words, the empirical calculation is most likely over-estimating the concentrations of the main trace parameters.

The explanation for this effect is probably that these elements are being released as part of weathering processes but remain stored in the rock. The expected retention of iron represents a sink for these elements through sorption processes. Pyrite is the dominant source of iron, and is likely also the source for many trace elements. The ratio of iron to other metals is very high and represents a significant source of sorptive capacity. Since this process is pH dependent, it is expected that metal concentrations would be negatively correlated with pH. An example of this type of relationship used as part of this example is shown in Figure 5. Data for two sites show that nickel concentrations are strongly related to pH. For example, between pH 4 and 7.5, the data from Huckleberry Mine shows a negative correlation with pH. Likewise, the Island Copper Mine dataset shows a good correlation throughout the pH range.

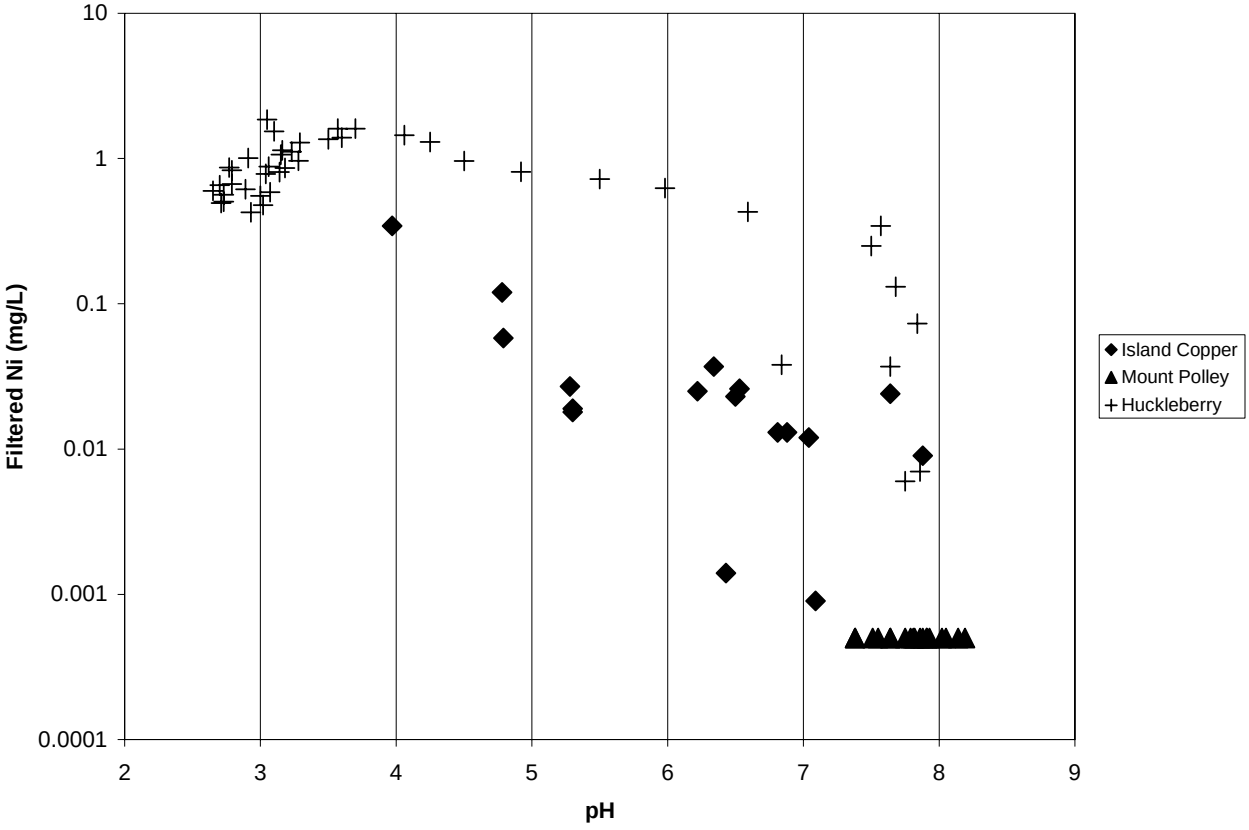

Figure 5: Example of Relationship Between Nickel Concentrations and pH (Red Chris Development Company 2004).

Example of Combined Empirical and Site Comparison Approach in Alaska

Water quality predictions waste rock and dry stack tailings for the recently permitted Pogo Project in Alaska were obtained using a combination of empirically-calculated concentrations scaled-up from humidity cells adjusted to reflect concentrations observed in groundwater, surface water, leach columns, meteoric water extraction procedures and seepage from a pile of waste rock from underground development.

Documentation from the project can be obtained from:

http://www.dnr.state.ak.us/mlw/mining/largemine/pogo/

7.3.4 Implementation for the NorthMet Project

The proposed overall approach is comparable to the combined empirical and site comparisons approach described in the example above. The initial empirical calculation will be based on interpolation and extrapolation of humidity cell results with adjustments for major parameters based on thermodynamic equilibrium calculations and reference to concentrations measured in test pile and waste rock pile drainage.

Scale-up of low concentrations of nickel and cobalt in the non-reactive rock category (sulfur less than 0.05%) is expected to require additional data since the majority of testwork to date has been focused on "reactive" rock containing higher concentrations of sulfur. Since drainage from non-reactive materials is expected to be non-acidic, reliable relationships that indicate correlations between metal concentrations and pH (for example, as shown in Figure 3 and see also Norecol Dames & Moore 1996) may be used to predict metal concentrations. The limitation of the current dataset is that nickel can be expected to be very mobile under non-acidic conditions when the metal to iron ratio is high (for example, if pentlandite is present). A distinctive water quality dataset is needed for low sulfur rock piles. The following approaches may be considered:

- Sampling of seepage from existing Duluth Complex waste rock piles or rock exposures known to contain low concentrations of sulfur.
- Evaluation of oxide coatings to understand the attenuation of metals and comparison with loads leached from humidity cells. Generally speaking, sulfate is conservative in slowly reactive humidity cells, therefore if the nickel to sulfur ratio is lower in the leachate than the sulfide minerals, then nickel is being attenuated (assuming that nickel originates from oxidation of sulfides). Likewise, comparison of iron and nickel release with iron and nickel ratios in sulfides and oxides coatings will indicate how nickel is attenuated relative to iron.

7.4 Conclusions

An empirical scale-up approach is proposed to translate weathering rates observed in humidity cells to full-scale concentrations. The resulting predicted concentrations will be evaluated and adjusted based on solubility constraints and data from existing monitoring. The incorporation data from past or existing waste rock facilities (AMAX test piles, Dunka Pit waste rock) ensures that the predictions are consistent with large scale operational experience. Additional testing may be designed to evaluate mobility and attenuation of metals such as nickel and cobalt.

8 References

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Appendix A

Parameter Lists and Detection Limits for Analysis of Solids

ALS/CHEMEX METHOD CODE	ITEM	UINTS	CHEMEX DETECTION LIMIT
ME-ICP61 (four acid)	CU%	%	0.001
ME-ICP61 (four acid)	NI%	%	0.001
S-IR08 (LECO SULFUR)	S%TOT	%	0.01
ME-ICP61 (four acid)	S%ICP	%	0.01
PGM-ICP23 (30 GRAM)	PT_PPB	PPB	5
PGM-ICP23 (30 GRAM)	PD_PPB	PPB	1
PGM-ICP23 (30 GRAM)	AU_PPB	PPB	1
ME-ICP61 (four acid)	CO_PPM	PPM	1
ME-ICP61 (four acid)	AG_PPM	PPM	0.5
ME-ICP61 (four acid)	ZN_PPM	PPM	2
ME-ICP61 (four acid)	CD_PPM	PPM	0.5
ME-ICP61 (four acid)	MO_PPM	PPM	1
ME-ICP61 (four acid)	PB_PPM	PPM	2
ME-ICP61 (four acid)	AS_PPM	PPM	5
ME-ICP61 (four acid)	CR_PPM	PPM	1
ME-ICP61 (four acid)	V_PPM	PPM	1
ME-ICP61 (four acid)	TI%	%	0.01
ME-ICP61 (four acid)	AL%	%	0.01
ME-ICP61 (four acid)	CA%	%	0.01
ME-ICP61 (four acid)	FE%	%	0.01
ME-ICP61 (four acid)	K%	%	0.01
ME-ICP61 (four acid)	NA%	%	0.01
ME-ICP61 (four acid)	MG%	%	0.01
ME-ICP61 (four acid)	MN_PPM	PPM	5
ME-ICP61 (four acid)	P_PPM	PPM	10
ME-ICP61 (four acid)	BA_PPM	PPM	10
ME-ICP61 (four acid)	BE_PPM	PPM	0.5
ME-ICP61 (four acid)	BI_PPM	PPM	2
ME-ICP61 (four acid)	SB_PPM	PPM	5
ME-ICP61 (four acid)	SR_PPM	PPM	1
ME-ICP61 (four acid)	W_PPM	PPM	10

ALS/CHEMEX METHOD CODE	ITEM	UINTS	CHEMEX DETECTION LIMIT
ME-ICP41 (Aqua regia digestion)	CU%	%	0.001
ME-ICP41 (Aqua regia digestion)	NI%	%	0.001
ME-ICP41 (Aqua regia digestion)	S%ICP	%	0.01
ME-ICP41 (Aqua regia digestion)	CO_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	AG_PPM	PPM	0.2
ME-ICP41 (Aqua regia digestion)	ZN_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	CD_PPM	PPM	0.5
ME-ICP41 (Aqua regia digestion)	MO_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	PB_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	AS_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	CR_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	V_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	TI%	%	0.01
ME-ICP41 (Aqua regia digestion)	AL%	%	0.01
ME-ICP41 (Aqua regia digestion)	CA%	%	0.01
ME-ICP41 (Aqua regia digestion)	FE%	%	0.01
ME-ICP41 (Aqua regia digestion)	K%	%	0.01
ME-ICP41 (Aqua regia digestion)	NA%	%	0.01
ME-ICP41 (Aqua regia digestion)	MG%	%	0.01
ME-ICP41 (Aqua regia digestion)	MN_PPM	PPM	5
ME-ICP41 (Aqua regia digestion)	P_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	B_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	BA_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	BE_PPM	PPM	0.5
ME-ICP41 (Aqua regia digestion)	BI_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	GA_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	HG_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	LA_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	SB_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	SC_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	SR_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	W_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	TL_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	U_PPM	PPM	10

ALS/CHEMEX METHOD CODE	ITEM	UINTS	CHEMEX DETECTION LIMIT
ME-ICP06--whole rock geochemisrty by ICP-AES	SIO2	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	AL203	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	TIO2	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	FE2O3	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	CAO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	MGO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	MNO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	NA2O	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	K2O	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	P2O5	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	BAO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	SRO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	LOI	%	0.01

Appendix B

Options and Variance in ASTM Humidity Cell Procedure

9. Sample Preparation

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
9.1	Air dry as-received bulk samples of solid material to prevent the additional oxidation of reactive minerals or compounds. If air drying is not practicable, oven dry the solid material at a maximum temperature of 50 ± 2°C for 24 h, or until a constant weight is reached.	Samples were air-dried at room temperature (~ 20 °C).	
9.1.1	If exploration-generated or run-of-mine solid material samples are not readily available, archived dried and crushed samples from geological exploratory or development drilling programs may be used for preliminary evaluations of ore and waste rock from new operations; this is provided that the available solid material samples are not significantly finer than 95 % passing a No. 12 (1.7-mm) sieve. Document the sample drying and preparation procedures used during the drill sampling program in order to interpret the results properly. Evaluate the effects of drying temperature on metals volatilization (for example, mercury in cinnabar vaporizes at temperatures exceeding 80 to 90°C) and mineral morphology and chemistry modifications (for example, on heating at temperatures exceeding 100°C, chalcocite changes crystal form and is oxidized subsequently from Cu_2S to CuO, $CuSO_4$, and SO_2). Especially ensure that the effects of particle size distribution changes resulting from the more finely crushed sample are considered in the interpretation (this is, the potential for increased liberation of acid-producing and acid-consuming minerals with an attendant increase in mineral surface area).	NA	
9.1.2	In mining waste evaluations, the particle size for mill tailings will be significantly finer (commonly less than 150 µm/100 mesh) than the particle size distributions from ore and waste rock. Pilot plant tailings should be used if mill tailings are not available.	NA	
9.2	Screen the air-dried bulk samples through a 6.3-mm (¼-in.) screen in accordance with Test Method E 276. Crush any oversize material so that 100 % passes the screen.	ASTM	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
Note 7	Caution: Recent accelerated weathering studies of run-of mine waste rock from metal mines demonstrate that crushing a bulk sample so it passes a 6.3-mm (¼-in.) screen may change the character of the sample by artificially increasing liberation and consequent surface areas of acid-producing and acid-consuming minerals contained in the + 6.3-mm (¼-in.) material. A suggestion for avoiding this problem is to segregate the - 6.3-mm (¼-in.) fraction by screening rather than crushing, and to test that fraction according to the protocol and equipment described in this text. The + 6.3-mm (¼-in.) material can be tested separately (for example, Brodie, et al (10) describe a large-scale humidity cell test that would accommodate – 75-mm material). Samples from the drill core and cuttings also present material sizing problems, which must be considered when interpreting drill core and cuttings accelerated data. The drill core must be crushed to -6.3-mm (¼-in.) to fit the cell described in this test method. The resulting size distribution from crushing will differ from that of run-of-mine due to differences in fracture patterns inherent to blasting practices that produce run-of-mine material. By contrast, drill cuttings size fractions are commonly less than 6.3-mm (¼-in.) due to the rotary-percussive nature of obtaining the sample.	NA	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
9.3	Mix and divide the bulk sample to obtain a representative test unit with a weight in the range of 8 to 10 kg, using a riffle splitter with 1-in. (2.54-cm) chutes. Divide the test unit into eight nominal 1-kg specimens. Seal each test specimen in a moisture-barrier bag.	Samples mixed by riffle splitter, but different sample weights were available (see column at the right).	All material available for each sample was mixed by the riffle splitter (0.552 kg – 10.54 kg). The test unit was divided into the following amounts: Samples received June 06, 2005 (lean ore): 150 – 200 g for Chemex Assay; Reject for archive, screen assay, etc. Samples received May 20, 2005 (waste rock): 200 g for Chemex Assay, 50 g of crushed archive, 100 g for screen assay, store rejects for HC
Note 8	The dried sample should be mixed through the riffle splitter at least once before making any splits; recombine the splits resulting from the sample mixing exercise by pouring individual splits either over each other or through the splitter again. Once the actual split is made, it is wise to re-mix it (according to the above procedure) prior to making the next split.		Samples were mixed through the riffle splitter once.
9.4	Select one test specimen at random, and determine the moisture content by weighing and drying to constant weight at 80 ±5°C.		Determined at 20 °C
9.4.1	Crush the dried test specimen so that at least 95 % passes a 1.7-mm (10-mesh) screen, in accordance with Test Method E 276.		See 9.3
9.4.2	Divide the crushed test specimen in half twice, using a riffle splitter with 6.35-mm (¼-in.) chutes, and select a ¼ subsample at random.		See 9.3
9.4.3	Transfer the selected subsample to a ring and puck grinding mill and grind to a nominal of 95 % passing a 150-µm (100-mesh) screen, in accordance with Test Method E 276. Use the subsample for chemical and mineralogical characterization of the test unit.		See 9.3

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
9.5	Select one test specimen at random, and determine the particle size distribution in accordance with Test Method E 276.		100 g were removed after 9.3 for screen assays: -1/4" + 10 mesh, -10 mesh + 35 mesh, -35 mesh + 100 mesh, -270 mesh; each size was weighed and submitted to Chemex for Total S (S-IR08) and four acid digestion (ME-ICP61).
9.6	Select one test specimen at random for use in the accelerated test method. Divide the test specimen into four nominal 250-g subsamples using the riffle splitter with 25.4-mm (1-in.) chutes, and label and store in vapor-barrier bags until it is time to load the humidity cells.	See 9.3 - variance column	
9.7	Reserve the remaining test specimens for replicated testing or to resolve disputed results.	See 9.3 - variance column	

10. Apparatus Assembly

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
10.1	The humidity cells are table-mounted at a height sufficient to accommodate the placement of both the humidifier and one Erlenmeyer flask for effluent collection from the bottom of each cell. During the water-saturated and dry-air portions of each weekly cycle, feed air is metered to the bottom of each cell at the selected rate (1 to 10L/min). Feed air for the three-day dry–air portion is routed first through a desiccant column and then to each of the cells through a dry-air manifold. Feed air for the water-saturated air portion is routed through a water-filled humidifier by means of aeration stones or gas dispersion fritted cylinders/disks, and then to each humidity cell lid air exit port to prevent the short circuiting of air through cells containing more permeable solid material samples. A separatory funnel rack is mounted on the table that holds the cells if the weekly water leach is applied dropwise (drip trickle). Multiple separatory funnels (one for each cell) are held in the rack during the drip trickle leach that is performed on the seventh day of each weekly cycle. The separatory funnel can be used to meter the required water volume slowly down the sides of the cell wall until the sample is flooded if the weekly leach is to be a flooded leach.	Humidity cells are constructed of acrylic tubing with an inside diameter of four inches and an overall height of twelve inches, with an acrylic base plate. The base plate is glued to the tube and threaded with a nylon hose adapter to which a length of tubing is attached to allow for leachate drainage into a collection container. A perforated PVC support plate is positioned inside the cell, one inch above the base plate and covered with six layers of nylon mesh. A nylon adapter is threaded into the side of the cell between the support plate and the base plate and a length of tubing was connected from the side adapter to the humidifier to facilitate the inflow of humid air to the cell. A dry air line is also connected to each cell. Each cell is covered with a removable acrylic lid.	Approximately 16 cells per humidifier Flood leaching: peristaltic pump using a peristaltic pump Temperature: 20 ± 2°C. Feed air rate to be determined.

11. Procedure

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.1	Cell Loading:		
11.1.1	If more than one humidity cell is used at one time, label each with a sequential number, and use the same number for the matching collection vessel (Erlenmeyer flask).	ASTM	
11.1.2	Weigh each humidity cell (without its lid) and each collection vessel; record the tare weights of each to the nearest 0.1 g.	ASTM	
11.1.3	Cut the filter media (such as 12-oz/yd^2 polypropylene described in 6.11) to the humidity cell's inside diameter dimensions so that it fits snugly yet lies flat on the perforated support.		PVC perforated disk & nylon mesh
11.1.4	Re-weigh the humidity cell, and record the resulting tare to the nearest 0.1 g; the original cell tare (11.1.2) minus the new cell tare is the weight of the filter media.	ASTM	
11.1.5	Transfer the contents from each of the four bags containing the 250-g samples (9.6) into the humidity cell. Prior to the transfer, mix the contents of each bag by gentle rolling to eliminate possible stratification that may have occurred during sample storage.	ASTM	
11.1.6	Re-weigh the loaded cell, and record the weight to the nearest 0.1 g; the loaded cell weight minus the combined cell and filter-media tare weight is the weight of the sample charge.	ASTM	
11.2	First Leach:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.2.1	The first leach (whether drip trickle or flooded), designated as the Week 0 leach, initiates the 20-week long humidity cell test and establishes the starting or initial characteristics of the leachate. Either a 500-mL or 1-L volume of water may be used for the weekly leachates, depending on the weekly pore volume desired or the quantity of solution required for analytical purposes; however, once a weekly volume has been selected, that weekly volume must remain constant throughout the 20-week testing period. A centrifuged cell culture of *Thiobacillus ferrooxidans* may be used in the first leach in order to ensure that optimum conditions for accelerates weathering are present at the beginning of the test.	500 mL Flood Leach	
Note 9	In the testing of mining wastes, cation (including metals and trace metals) and anion loadings are commonly high in the Week 0 leachate due to the dissolution of pre-existing soluble oxidation salts present in the sample prior to sample collection. The average number of weekly accelerated weathering cycles required to flush these pre-existing salts ranges from 3 to 5 weeks. Oxidation products observed during these 3 to 5 weeks are principally from the pre-existing salts, while those products observed after this period are considered to be solely a function of the accelerated weathering procedure. A method for estimating the amount of pre-existing oxidation salts present in a solid material sample is described by Sobek, et al (6). A comparison of estimated salt storage data obtained using this method with the first thee weeks of humidity cell effluent loadings from three different samples is describes by White and Jeffers (7).	NA	
11.2.2	Fill a separatory funnel with for each cell with de-ionized water using a volumetric flask. If the leach is to be performed using the drip trickle method, set each separatory funnel above its corresponding cell, and adjust the drip rate (approximately 3 to 4 L/min) so that the solid material sample is wetted thoroughly but not flooded.	NA	
11.2.3	A minimum of 2 to 3 h is commonly required to complete the drip trickle leach.	NA	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.2.4	If the leach is to be performed by flooding, the separatory funnel can be used to meter the selected water volume slowly down the sides of the cell wall until the sample is flooded. This application method reduces hydraulic agitation of the sample surface commonly caused by pouring liquid from an open-mouthed vessel. Alternatively, flooding may be accomplished by any application apparatus (for example, a peristaltic pump) that supplies the selected volume of leachant at a reasonable rate without causing agitation and suspension of the finer fractions contained in the sample charge.	ASTM	
11.2.4.1	Allow the flooded cell to sit for a period of 1 h before draining the leachate into the Erlenmeyer collection flask. The 1-h leach time commences after all of the leachant has been placed in the cell. The solid material sample should be saturated and covered with leachant to a depth sufficient to maintain sample saturation. In testing mining wastes, the observed depth of leachant cover from a 500-mL flooded leach performed in 10.2-cm (4.0-in.) ID cells is approximately 2.5 cm (1.0 in.).	ASTM	
11.2.5	The following is performed once the leaching process has been completed: to reduce the effects of evaporation, and to prevent the contamination of each cell by airborne contaminants, place the lids on their corresponding cells and let the cells complete the leachate draining process for the remainder of the leaching day and overnight.	ASTM	
11.2.6	Disconnect the cells on the day following the leach, and weigh and record the weight of each cell and Erlenmeyer collection flask. Set each filled collection flask aside for leachate analyses. (Measurements of pH and Eh and sample preservation procedures must be performed as soon as possible after leachate collection.) Return each cell, replace the filled collection flasks with clean, tared Erlenmeyer flasks, hook up all connections, and begin the dry-air cycle.	ASTM	
11.3	Dry-Air Cycle:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.3.1	The commencement of the three-day dry-air period marks the beginning of each new weekly cycle of the accelerated weathering humidity cell test; the first full-week cycle after the first leaching is designated Week 1; subsequent weeks (commencing with the second dry-air period) are designated as Week 2, Week 3 … . Week n, etc.	ASTM	
11.3.2	To perform the dry-air cycle, feed air is metered to the humidity cell array with a flowmeter (see 6.3) set at a target rate in the range of 1 to 10 L/min per cell, depending on the objectives of the testing. The air flow rate must be checked daily and adjusted to the target value ± 0.5 L/min.	ASTM	
11.3.3	Feed air from the flowmeter is routed first through a desiccant column and then to each of the sells through a dry-air manifold. Air exiting the desiccant column should have a relative humidity of less than 10 % as measured with a hygrometer (see 6.23).	ASTM	
11.3.4	To maintain similar positive air pressure through the cells, attach a water-bubbling vessel to each humidity cell air exit port coming out of the humidity cell lid; a 50-mL Erlenmeyer flask with a rubber stopper containing a vent and air inlet tube serves as a simple and efficient bubbler.	ASTM	
11.3.5	The dry air is passed through each humidity cell for three days. Air flow rates from each of the cells should be checked each day, recorded, and adjusted, if necessary. See also Note 10.	ASTM	
11.4	Wet-Air Cycle:		
11.4.1	The three-day wet-air period commences on the fourth day of each weekly cycle.	ASTM	
11.4.2	To perform the wet-air cycle of the method, feed air is routed through a water-filled humidifier via aeration stones or gas dispersion fritted cylinders/disks and then to each humidity cell.	ASTM	
11.4.3	The water temperature in the humidifier is maintained at 30 ± 2°C to ensure that the sparged air maintains a relative humidity of approximately 95 % as measured with a hygrometer (see 6.23) from one of the humidifier exit lines. Air flow rates to each of the cells should be checked each day, recorded, and adjusted, if necessary.	ASTM	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
Note 10	It is good practice to measure the air flow rates and relative humidity of the air exiting each humidity cell during each day of the three-day dry- and wet-air periods; the measurements should be taken at the same time each day from the humidity cell air exit port; these measurements can be accomplished by installing a quick-disconnect fitting in the tubing that connects the air exit port to the bubbler.	NA	
Note 11	Coals spoils in eastern states are commonly saturated; Caruccio (10) has suggested the following geographic control alternative to the dry-air versus saturated-air scheduling: (1) Eastern States Samples – Six days of saturated air (versus three days dry/three days wet); and (2) Western States Samples – Three days dry/three days wet.	NA	
11.5	Subsequent Weekly Leaches:		
11.5.1	A second leach with water is performed on the day following the end of the three-day wet-air period (that is, day seven of the first weekly cycle). This leach marks the end of the first weekly cycle and is designated as the Week 1 leach.	ASTM	
11.5.2	Subsequent leaches are designates as Week2, Week 3 … Week n, and they mark the end of the weekly cycle for that numbered week. Perform each weekly leach as described in 11.2.2 – 11.2.5. Weekly weighing of the test cells is optional.	ASTM	No weekly weighing of the cells.
11.6	It is recommended that the weekly accelerated weathering cycles described in 11.2, 11.3, 11.4 and 11.5 be performed for a minimum of 20 weeks.	ASTM	
Note 12	Additional weeks of accelerated weathering may be required to demonstrate the nature of the material, depending on the chemical composition of the solid material. For some metal mining wastes, researchers have shown that as much as 60 to 120 weeks of accelerated weathering data may be required to demonstrate the complete weathering characteristics of a particular sample (7, 12). The criteria for ending the testing may be site specific and should be agreed before initiating the testing.	ASTM	
11.7	Leachate Analyses:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.7.1	Analyze the leachates for specific constituents or properties, or use them for biological testing procedures as desired, using (1) appropriate ASTM test methods or (2) methods accepted for the site where disposal will occur. Where no appropriate ASTM test method exists, other test methods may be used and recorded in the report, provided that they are sufficiently sensitive to assess potential water quality impacts at the proposed disposal site. Suggested minimum weekly analyses should include pH, Eh, conductivity, and selected metals could be analyzed less frequently (for example, at Weeks 0, 1, 2, 4, 8, 12, 16, and 20), especially if changes in leachate chemistry are slow. Whether visible phase separation during storage of the leachates occurs or not, appropriate mixing should be used to ensure the homogeneity of the leachates prior to their use in such analyses.	At the end of weekly cycle the volume of leachate collected is recorded. The leachate is filtered through a Gelman magnetic filter funnel fitted with a membrane filter with pore size of 0.45 microns and analyzed for the parameters listed in Table 2 of the RFP. Filtered leachate samples will be submitted to ALS Environmental/Cantest Ltd. for dissolved metals analysis as requested in Table 4 of the Waste Rock and Lean Ore Geochemical Characterization Plan. Conductivity, Eh, and pH are measured in the CEMI laboratory using standard procedures. An aliquot of filtered leachate is titrated with standardized sulphuric acid to pH 4.5 to calculate total alkalinity. Standardized sodium hydroxide is used to titrate an aliquot of leachate to pH 4.5 and to pH 8.3 to calculate total acidity. Analysis frequency: pH, cond, Eh every cycle; SO4, Cl, F, alkalinity, TIC, acidity cycle 0, 2, 4, 6 etc.; ICP-MS including Hg and Si cycle 0, 4, 8, 12, etc., ICP-ES including Si cycle 2, 6, 10, 14, etc.	
11.7.2	Table 1 is an example of a spreadsheet format used for recording 20 weeks of leachate analytical data.	ASTM	
11.7.3	Fig. 5 is an example of a method used to plot the temporal variation (by week) of leachate pH, sulfate load, and cumulative sulfate load from 21 weeks of accelerated load and release rates).	ASTM	
11.8	Weathered Solid Material Analyses:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.8.1	Weigh the humidity cell after collection of the final effluent and completion of a three-day dry-air period.	ASTM	
11.8.2	Transfer the weathered residue and filter media to a clean drying pan, and dry to constant weight at 50 ± 5°C. Record the final weight.	ASTM	
Note 13	Perform any gross sample examination (for example, sample texture and weathering product mineralogic characterization) desired for the weathered residues prior to pulverization. To facilitate such an examination, empty the humidity cell contains into a clean drying pan carefully by pushing gently on the bottom of the perforated plate with a wooden dowel until the sample exits the cell mouth. The perforate plate is accessed through the humidity cell drain port.	NA	
11.8.3	Identify and mark the top versus bottom portions of the sample for gross sampling purposes. Formations of cemented lumps of sample termed “ferricrete” that result from the accelerated weathering process arte common in iron-sulfide-mineral rich samples. Depending on the sample mineralogy, the degree of “ferricrete” cementation may vary vertically within the sample, and the investigator may wish to segregate the sample into upper, middle, and lower thirds to document and characterize such changes.	Procedure to be determined	
11.8.4	After drying to constant weight and prior to splitting, use an instrument such as a rolling pin to break up cemented lumps in the sample (if the cemented lumps cannot be sufficiently reduced to pass through the chutes of a riffle splitter, remove, record, and weigh separately):	ASTM	
11.8.4.1	Split the sample into halves using a riffle splitter with 2.54-cm (1-in.) chutes, and reserve one half to determine the particle size distribution in accordance with Test Method E 276.		Repeat same screen assay method as for pre-test characterization (s.9.5)
11.8.4.2	Split the remaining half sample into two quarters using a riffle splitter with 2.54-cm (1-in.) chutes, and submit one quarter for mineralogical characterization; pulverize the other quarter in either a ring-and-puck or disk-pulverizing machine to 95 % passing a 150-µm (100-mesh) screen in accordance with Test Method E 276.	Procedure to be determined	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.8.5	Mix the pulverized residue in a blender or on a rolling cloth. Use the prepared residue for chemical characterization and for comparison with the pre-weathered solid material sample.	Procedure to be determined	

Appendix C

Design of MDNR Reactor

Day, Stephen

From: Kim Lapakko [kim.lapakko@dnr.state.mn.us]
Sent: Tuesday, May 17, 2005 9:55 AM
To: Stephen Day
Cc: Dave Antonson; Jennifer Engstrom; Paul Eger
Subject: RE: Small reactor

Attachments: MN DNR psize methods 050517.doc

MN DNR psize methods 050517.do...

Steve,

Attached is a description of the reactors, masses, and rinse volumes used for various size fractions of Duluth Complex rock in our particle size experiment. As indicated in the attachment, I won't have access to the trace metal data from that experiment until tomorrow. I will need to examine this to help evaluate the expected metal concentrations in drainage relative to detection limits. I'm not sure it will give us as much as hoped because the sulfur contents of the samples typically were on the order of 0.9% to 1.3%. This may make extrapolation by more than an order of magnitude tenuous. It will be another pertinent piece of information.

Kim

>>> "Stephen Day" <sday@srk.com> 5/17/2005 11:18:50 AM >>>
Dave

A design drawing should be fine along with description of the procedure.

The main question is what do you do to scale-up the sample mass as the particle size increases? I want to copy your procedure exactly.

Thanks
Steve.

-----Original Message-----
From: Kim Lapakko [mailto:kim.lapakko@dnr.state.mn.us]
Sent: Tuesday, May 17, 2005 8:38 AM
To: Stephen Day
Cc: Dave Antonson
Subject: Small reactor

Steve,

Dave Antonson will email a figure depicting our small reactor, along with some design details (perforated plate, adehesive, filter). He could also send a reactor. Please contact him directly, with an address to send it, if you think that would be helpful.

Kim

17 May 2005

Steve,

In our particle size tests we used a small reactor and 75-g mass for particle sizes of –270, +270/-100, and +100/-35 mesh. We used the ASTM cell and 1000-g mass for +35/-10, +10/-0.25 inch, and +0.25/-0.75 inch particle sizes. For rinse volumes, we used 200 mL for the 75-g samples and 300 mL for the 1000-g samples. The 300-mL rinse volume was determined as the quantity of water, rounded up to the nearest 100 mL, required to submerge the solids.

I won't have access to the metal release data for the particle size experiment until tomorrow. As mentioned on the phone, sulfate release rates appear to vary linearly with surface area. It seems likely that nickel release rates will vary similarly, and I'll look into this further tomorrow. Hopefully this information will shed some light on the maximum particle size question.

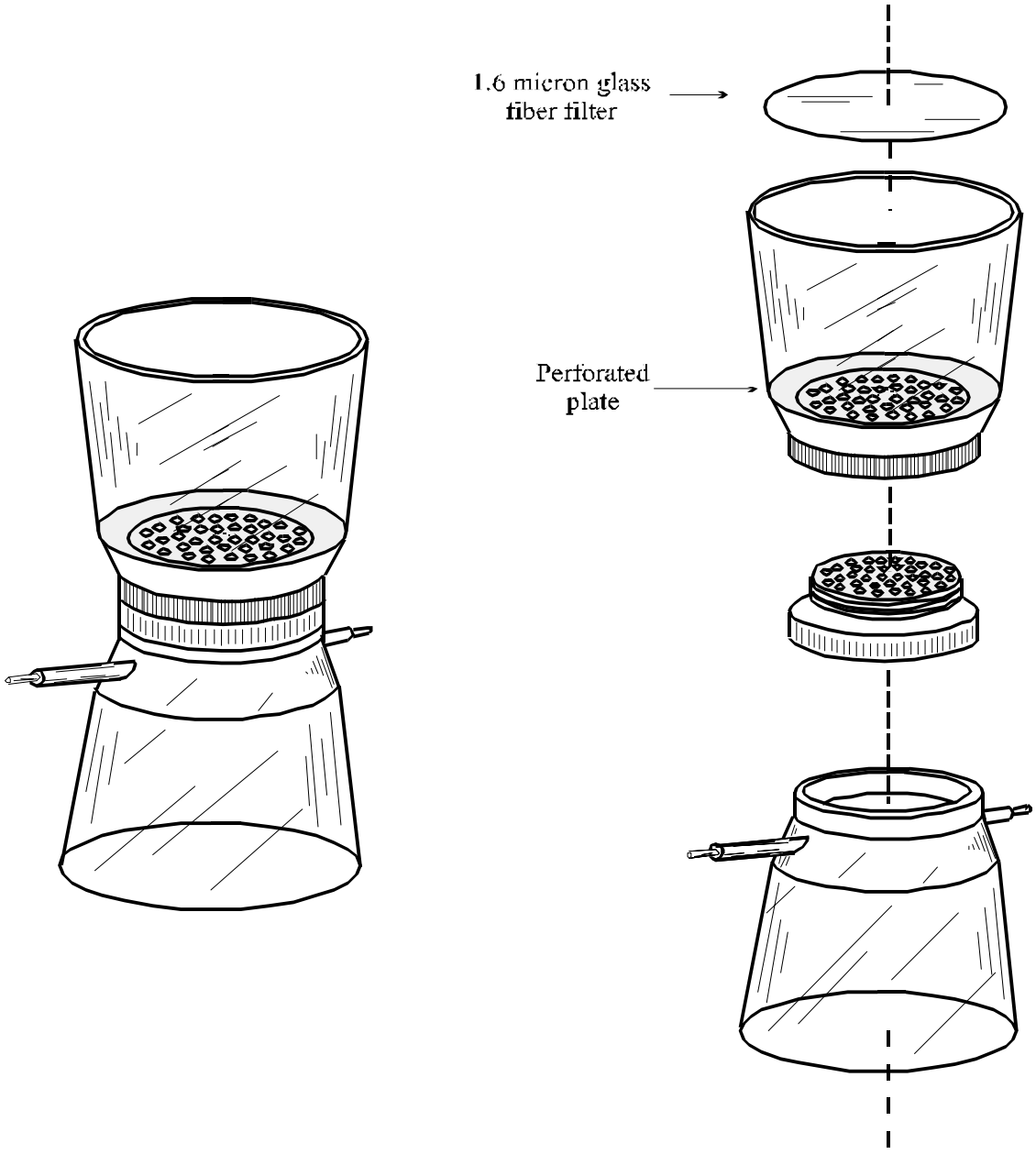
Day, Stephen

From: Dave Antonson [dave.antonson@dnr.state.mn.us]
Sent: Tuesday, May 17, 2005 11:53 AM
To: Kim Lapakko
Subject: reactor

Attachments: small reactor.doc

small reactor.doc
(271 KB)

see if this makes any sense. you can edit it if you want. if it seems adequate you can forward it to steve. maybe he doesn't need a sample of the base.


1.6 micron glass
fiber filter
Perforated
plate

The reactors were purchased from Millipore Corporation (1-800-645-5476). They are 47 mm Sterifil aseptic systems. You will need the 250 ml receiver flask, 250 ml funnel (top), silicone o-rings, and the filter holder base and support screen.

The perforated acrylic plastic base was purchased as flat stock and fabricated to fit the top funnel. The plates are 1/8" thick, 2 1/4" in diameter and tapered to fit into the reactor top. Approximately sixteen 1/16" holes were drilled in the plate. The plate was glued into the reactor using acrylic solvent cement purchased from United States Plastics (1-800-537-9724). Catalog # 44629 for 5 oz. tube. The acrylic flat stock was also purchased from United States Plastics.

After the plate is glued into the top of the reactor there should be approximatly a 3/8" gap between the bottom of the perforated plate and the top of the support screen of the filter unit.

The filter that rests on the perforated plate is a 55 mm Whatman GF/A glass microfibre filter (catalog # 1820 055).

Note: Before adding the solids to the filter you should wet the filter slightly with distilled water so no solids escape around the filter.

Appendix B
Characteristics of Samples in Dissolution Experiments

										WEIGHTED INTERVAL FROM DATABASE						THIN SECTION INFO					
HCT ID	Comment	HCT Full ID (Drill Hole, Footage, HCT)	Waste Type	Geological Unit	Rock Type From Logging	Rock Type From Mineralogy	Target S Percentile or Level	DDH	From	To	Interval	S (%)	NI (%)	Co (ppm)	Cu (%)	Zn (ppm)	Individual petrographic section footages		Type -Waste Rock or Lean Ore	CEMI Sample Number	Classification	CHEMEX Version of CEMI Sample Number
1		DDH-99-320C(830-850)-1	Reactive	1	Anorthositic	Anorthositic Troctolite	P25	99-320C	830	850	20	0.12	0.015	35.8	0.035	72	833	841	waste rock	24913	REACTIVE	24913
2		DDH-00-361C(310-320)-2	Reactive	1	Anorthositic	Anorthositic Troctolite	P50	00-361C	310	320	10	0.16	0.018	36.0	0.013	51	311	316	waste rock	24914	REACTIVE	24914
3		DDH-00-361C(345-350)-3	Reactive	1	Anorthositic	Troctolitic Anorthosite	P75	00-361C	345	350	5	0.33	0.026	53.0	0.038	68	347	349	waste rock	24915	REACTIVE	24915
4		DDH-00-343C(240-250)-4	Reactive	1	Anorthositic	Norite	P95	00-343C	240	250	10	0.67	0.020	38.0	0.069	69	241	249	waste rock	24916	REACTIVE	24916
5		DDH-26030(1047-1052)-5	Reactive	1	Sedimentary Hornfels	Recrystallized/Granoblastic Calc-Silicate	P10	26030	1047	1052	5	0.17	0.006	10.0	0.006	34	1048	1050	waste rock	24917	REACTIVE	
6		DDH-26061(1218-1233)-6	Reactive	1	Sedimentary Hornfels	Granoblastic cordierite biotite Graywacke	P25	26061	1218	1233	15	0.43	0.011	19.3	0.012	193	1220	1231	waste rock	24918	REACTIVE	24918
7		DDH-00-340C(990-995)-7	Reactive	1	Sedimentary Hornfels	Granoblastic Graywacke	P50	00-340C	990	995	10	0.62	0.012	24.0	0.031	148	991	994	waste rock	24919	REACTIVE	24919
8		DDH-00-340C(965-974.5)-8	Reactive	1	Sedimentary Hornfels	Granoblastic cordierite biotite pyrrhotite Graywacke	P75	00-340C	965	974.5	15	1.49	0.015	23.7	0.031	249	966	972	waste rock	24920	REACTIVE	24920
9	1 Dup	DDH-99-320C(830-850)-9D	Reactive	1	Anorthositic	Anorthositic Troctolite	P25	99-320C	830	850	20	0.12	0.015	35.8	0.035	72	833	841	waste rock	24913	REACTIVE	24913
10		DDH-00-340C(765-780)-10D	Reactive	1	Troctolitic	Augite Troctolite	P100	00-340C	765	780	15	1.72	0.022	78.0	0.064	69	771	778	waste rock	24930	REACTIVE	24930
11		DDH-26043&26027(1501&740-1506&745)-11	Reactive	1	Sedimentary Hornfels		P85	Composite														
12	Blank	Method Blank - 12																				
13		DDH-00-340C(595-615)-13	Non-reactive	1	Troctolitic	Olivine Gabbro	NR-0.03	00-340C	595	615	20	0.03	0.025	49.8	0.013	71	596	613	waste rock	24923	NON-REACTIVE (</= 0.5%S)	24923
14		DDH-00-334C(580-600)-14	Non-reactive	1	Troctolitic	Augite Troctolite	NR-0.05	00-334C	580	600	20	0.05	0.024	52.0	0.024	83	589	598	waste rock	24924	NON-REACTIVE (</= 0.5%S)	24924
15		DDH-00-334C(640-660)-15	Reactive	1	Troctolitic	Augite Troctolite	P25	00-334C	640	660	20	0.08	0.038	63.8	0.024	102	642	657	waste rock	24925	REACTIVE	24925
16		DDH-00-347C(795-815)-16	Reactive	1	Troctolitic	Augite Troctolite	P50	00-347C	795	815	20	0.09	0.034	70.8	0.035	94	798	812	waste rock	24926	REACTIVE	24926
17		DDH-00-350C(580-600)-17	Reactive	1	Troctolitic	Augite Troctolite	P80	00-350C	580	600	20	0.22	0.027	55.5	0.041	115	582	591	waste rock	24927	REACTIVE	24927
18		DDH-00-327C(225-245)-18	Reactive	1	Troctolitic	MelaTroctolite (Picrite)/Troctolite	P90	00-327C	225	245	20	0.44	0.015	50.5	0.032	73	226	242	waste rock	24928	REACTIVE	24928
19		DDH-00-371C(435-440)-19	Reactive	1	Troctolitic	Norite	P95	00-371C	435	440	5	0.65	0.014	32.0	0.036	54	436	438	waste rock	24929	REACTIVE	24929
20	10 Dup	DDH-00-340C(765-780)-20	Reactive	1	Troctolitic	Augite Troctolite	P100	00-340C	765	780	15	1.72	0.022	78.0	0.064	69	771	778	waste rock	24930	REACTIVE	24930
21		DDH-00357C(335-340)-21	Reactive	1	Ultramafic	MelaTroctolite (Picrite)	P25	00-357C	335	340	5	0.08	0.015	35.0	0.026	68	337	338	waste rock	24931	REACTIVE	
22		DDH-00326C(680-685)-22	Reactive	1	Ultramafic	MelaTroctolite (Picrite)	P80	00-326C	680	685	5	0.21	0.016	33.0	0.085	82	680	684	waste rock	24932	REACTIVE	
23		DDH-00-357C(535-540)-23	Reactive	1	Ultramafic	MelaGabbro	P85	00-357C	535	540	5	0.26	0.026	34.0	0.095	62	536	539	waste rock	24933	REACTIVE	
24		DDH-99-318C(725-735)-24	Reactive	1	Ultramafic	Basalt Inclusion	P90	99-318C	725	735	10	0.44	0.011	23.0	0.032	45	728	735	waste rock	24934	REACTIVE	24934
25		DDH-99-317C(460-470)-25	Reactive	1	Ultramafic	MelaGabbro	P95	99-317C	460	470	10	1.10	0.018	33.0	0.056	86	462	465	waste rock	24935	REACTIVE	24935
26		DDH-00-366C(185-205)-26	Non-reactive	2	Anorthositic	Gabbroic Anorthosite	NR-0.01	00-366C	185	205	20	0.01	0.018	35.0	0.008	47	187	204	waste rock	24936	NON-REACTIVE (</= 0.5%S)	24936
27		DDH-00-366C(230-240)-27	Non-reactive	2	Anorthositic	Troctolitic Anorthosite	NR-0.03	00-366C	230	240	10	0.02	0.014	32.0	0.011	51	231	239	waste rock	24937	NON-REACTIVE (</= 0.5%S)	24937
28		DDH-99-320C(165-175)-28	Non-reactive	2	Anorthositic	Gabbroic Anorthosite	NR-0.05	99-320C	165	175	10	0.04	0.023	46.0	0.012	63	167	172	waste rock	24938	NON-REACTIVE (</= 0.5%S)	24938
29		DDH-99-318C(250-370)-29	Non-reactive	2	Troctolitic	Troctolite	NR-0.01	99-318C	250	270	20	0.02	0.022	42.3	0.011	64	257	267	waste rock	24939	NON-REACTIVE (</= 0.5%S)	24939
30		DDH-00-373C(95-115)-30	Non-reactive	2	Troctolitic	Troctolite	NR-0.03	00-373C	95	115	20	0.03	0.036	64.0	0.019	84	99	114	waste rock	24940	NON-REACTIVE (</= 0.5%S)	24940
31		DDH-00-373C(75-95)-31	Non-reactive	2	Troctolitic	Troctolite	NR-0.05	00-373C	75	95	20	0.05	0.031	55.3	0.020	75	78	93	waste rock	24941	NON-REACTIVE (</= 0.5%S)	24941
32		DDH-00-357C(110-130)-32	Reactive	2	Troctolitic	Troctolite	P50	00-357C	110	130	20	0.07	0.024	53.5	0.029	88	112	126	waste rock	24942	REACTIVE	24942
33		DDH-99-320C(315-330)-33	Reactive	2	Troctolitic	Troctolite	P80	99-320C	315	330	15	0.09	0.017	39.3	0.026	65	319	328	waste rock	24943	REACTIVE	24943
34		DDH-00-369C(335-345)-34	Reactive	2	Troctolitic	Troctolite	P95	00-369C	335	345	10	0.16	0.021	43.0	0.046	68	336	341	waste rock	24944	REACTIVE	24944
35		DDH-00-368C(460-465)-35	Non-reactive	2	Ultramafic	Serpentinized MelaTroctolite (Picrite)	NR-0.01	00-368C	460	465	5	0.03	0.042	70.0	0.033	98	462	463	waste rock	24945	NON-REACTIVE (</= 0.5%S)	24945
36		DDH-26055(940-945)-36	Non-reactive	2	Ultramafic	MelaTroctolite (Picrite)	NR-0.03	26055	940	945	5	0.04	0.037	69.0	0.025	88	941	944	waste rock	24946	NON-REACTIVE (</= 0.5%S)	24946
37		DDH-00-368C(125-145)-37D	Non-reactive	3	Anorthositic	Anorthositic Troctolite	NR-0.03	00-368C	125	145	20	0.03	0.016	38.0	0.015	62	132	143	waste rock	24949	NON-REACTIVE (</= 0.5%S)	24949
38		DDH-00-369C(305-325)-38D	Reactive	3	Troctolitic	Troctolite	P95	00-369C	305	325	20	0.27	0.030	51.5	0.037	57	306	318	waste rock	24956	REACTIVE	24956
39		DDH-26098&00-337C(145&105-148.5&110)-39	Non-reactive	2	Ultramafic		NR-0.05	Composite											x			
40		DDH-00-334C(30-50)-40	Non-reactive	3	Anorthositic	Troctolitic Anorthosite	NR-0.01	00-334C	30	50	20	0.01	0.022	47.0	0.009	64	31	50	waste rock	24948	NON-REACTIVE (</= 0.5%S)	24948
41	37 Dup	DDH-00-368C(125-145)-41	Non-reactive	3	Anorthositic	Anorthositic Troctolite	NR-0.03	00-368C	125	145	20	0.03	0.016	38.0	0.015	62	132	143	waste rock	24949	NON-REACTIVE (</= 0.5%S)	24949
42		DDH-00-368C(20-40)-42	Non-reactive	3	Anorthositic	Troctolite	NR-0.05	00-368C	20	40	20	0.04	0.010	25.8	0.022	47	28	39	waste rock	24950	NON-REACTIVE (</= 0.5%S)	24950
43		DDH-00-366C(35-55)-43	Non-reactive	3	Troctolitic	Troctolite	NR-0.01	00-366C	35	55	20	0.01	0.019	45.3	0.005	50	37	47	waste rock	24951	NON-REACTIVE (</= 0.5%S)	24951
44		DDH-00-334C(110-130)-44	Non-reactive	3	Troctolitic	Troctolite	NR-0.03	00-334C	110	130	20	0.03	0.028	57.3	0.012	73	112	128	waste rock	24952	NON-REACTIVE (</= 0.5%S)	24952
45		DDH-00-347C(155-175)-45	Non-reactive	3	Troctolitic	Troctolite/Anorthositic Troctolite	NR-0.05	00-347C	155	175	20	0.04	0.015	49.5	0.015	67	157	174	waste rock	24953	NON-REACTIVE (</= 0.5%S)	24953
46		DDH-00-347C(280-300)-46	Reactive	3	Troctolitic	Troctolite	P50	00-347C	280	300	20	0.08	0.018	45.3	0.035	61	282	299	waste rock	24954	REACTIVE	24954
47		DDH-00-326C(60-70)-47	Reactive	3	Troctolitic	Troctolite	P85	00-326C	60	70	10	0.12	0.031	51.0	0.034	91	61	66	waste rock	24955	REACTIVE	24955
48	38 Dup	DDH-00-369C(305-325)-48	Reactive	3	Troctolitic	Troctolite	P95	00-369C	305	325	20	0.27	0.030	51.5	0.037	57	306	318	waste rock	24956	REACTIVE	24956
49		DDH-00-367C(50-65)-49	Non-reactive	4	Troctolitic	Serpentinized Troctolite	NR-0.01	00-367C	50	65	15	0.02	0.015	38.7	0.010	59	55	62	waste rock	24957	NON-REACTIVE (</= 0.5%S)	24957
50		DDH-00-367C(260-280)-50	Non-reactive	4	Troctolitic	Augite Troctolite	NR-0.03	00-367C	260	280	20	0.04	0.024	53.8	0.018	78	269	279	waste rock	24958	NON-REACTIVE (</= 0.5%S)	24958
51		DDH-00-367C(290-310)-51	Non-reactive	4	Troctolitic	Troctolite	NR-0.05	00-367C	290	310	20	0.04	0.021	41.3	0.018	64	297	308	waste rock	24959	NON-REACTIVE (</= 0.5%S)	24959
52		DDH-00-370C(20-30)-52	Reactive	4	Troctolitic	Augite Troctolite	P25	00-370C	20	30	10	0.07	0.010	37.0	0.016	67	22	27	waste rock	24960	REACTIVE	24960
53		DDH-00-369C(20-30)-53	Reactive	4	Troctolitic	Troctolite	P75	00-369C	20	30	10	0.14	0.021	39.0	0.043	60	22	29	waste rock	24961	REACTIVE	24961
54		DDH-00-367C(170-175)-54	Reactive	4	Troctolitic	Augite Troctolite	P90	00-367C	170	175	5	0.48	0.023	45.0	0.034	54	171	173	waste rock	24962	REACTIVE	24962
55		DDH-00-367C(395-400)-55	Reactive	4	Troctolitic	Chloritic Troctolite	P95	00-367C	395	400	5	0.92	0.028	76.0	0.080	96	395	398	waste rock	24963	REACTIVE	24963
56		DDH-26064(44-54)-56	Non-reactive	5	Troctolitic	Anorthositic Troctolite	NR-0.01	26064	44	54	10	0.01	0.035	62.0	0.009	58	45	53	waste rock	24964	NON-REACTIVE (</= 0.5%S)	24964
57	51 Dup	DDH-00-367C(290-310)-57D	Non-reactive	4	Troctolitic	Troctolite	NR-0.05	00-367C	290	310	20	0.04	0.021	41.3	0.018	64	297	308	waste rock	24959	NON-REACTIVE (</= 0.5%S)	24959
58		DDH-00-364C(210-229)-58D	Reactive	20	Virginia	Granoblastic pyrrhotite Graywacke	P75	00-364C	210	229	19	4.11	0.018	26.8	0.017	872	211	228	waste rock	24969	REACTIVE	24969
59		DDH-26064(264&146-269&156)-59	Non-reactive	5	Troctolitic		NR-0.05	Composite											x			
60		DDH-26056(110-125)-60	Non-reactive	6	Troctolitic	Troctolite	NR-0.01	26056	110	125	15	0.02	0.034	66.7	0.025	85	113	123	waste rock	24966	NON-REACTIVE (</= 0.5%S)	24966
62		DDH-00-361C(737-749)-62	Reactive	20	Virginia	Granoblastic graphitic cordierite Argillite	P25	00-361C	737	749	15	1.91	0.015	22.3	0.039	225	739	748	waste rock	24968	REACTIVE	24968
63	58 Dup	DDH-00-364C(210-229)-63	Reactive	20	Virginia	Granoblastic pyrrhotite Graywacke	P75	00-364C	210	229	19	4.11	0.018	26.8	0.017	872	211	228	waste rock	24969	REACTIVE	24969
64		DDH-00-337C(510-520)-64	Reactive	20	Virginia	Bedded Granoblastic pyrrhotite cordierite Graywacke	P90	00-337C	510	520	10	5.12	0.016	29.5	0.019	492	511	519	waste rock	24970	REACTIVE	24970
74		26029(interval 815-825)-74	Non-reactive	1	Troctolitic		NR-0.01	26029	815	825	10								waste rock			
78		26056(135-153)-78	Non-reactive	6	Troctolitic	Troctolite	NR-0.03	26056	135	153	18	0.04	0.037	62.7	0.032	89	138	152	waste rock	24967		

					4 ACID DIGESTIONS W/LECO AND PGE																																	
					ME-ICP61	ME-ICP61	S-IR08	ME-ICP61	PGM-ICP23	PGM-ICP23	PGM-ICP23		ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61
			Analytical Method		4-Acid	4-Acid	Leco	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid
HCT ID	Comment	HCT Full ID (Drill Hole, Footage, HCT)	Sample Weight (kg)	Specific Gravity	Cu	Ni	S	S ICP	Pt	Pd	Au	Pt+Pd+Au	Co	Ag	Zn	Cd	Mo	Pb	As	Cr	V	Ti	Al	Ca	Fe	K	Na	Mg	Mn	P	Ba	Be	Bi	Sb	Sr	W
					%	%	%	%	ppb	ppb	ppb	ppb	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	%	%	%	%	%	%	%	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
1		DDH-99-320C(830-850)-1	6	3.00	0.0365	0.0174	0.09	0.09	160	210	40	410	48	<0.5	118	<0.5	<1	2	<5	140	190	1.33	8.90	6.28	8.58	0.90	2.15	3.26	1150	1280	260	1.0	<2	<5	374	<10
2		DDH-00-361C(310-320)-2	5.95	2.96	0.0152	0.0208	0.18	0.16	<5	40	20	60	42	<0.5	72	<0.5	<1	<2	<5	260	102	0.46	8.85	6.76	5.53	0.20	2.06	2.96	703	430	110	0.5	<2	<5	322	<10
3		DDH-00-361C(345-350)-3	2.05	3.01	0.0236	0.0276	0.05	0.05	110	160	50	320	63	<0.5	91	<0.5	<1	<2	<5	152	83	0.45	8.24	5.68	7.77	0.30	1.91	4.55	945	420	110	<0.5	<2	<5	267	<10
4		DDH-00-343C(240-250)-4	2.75	3.09	0.0759	0.0199	0.68	0.68	<5	190	50	240	48	<0.5	192	<0.5	8	10	10	185	377	2.43	8.03	4.26	11.05	1.11	2.18	2.90	1255	2230	270	1.5	<2	<5	218	<10
5		DDH-26030(1047-1052)-5	1.191	3.13	0.0098	0.0104	0.24	0.26	<5	1	3	4	25	<0.5	168	0.8	2	6	14	151	153	0.47	8.10	13.60	5.47	0.53	0.89	2.20	2730	670	240	1.4	<2	<5	326	10
6		DDH-26061(1218-1233)-6	3.27	2.87	0.0146	0.0122	0.44	0.44	<5	30	20	50	24	<0.5	225	0.8	10	37	27	196	216	0.52	8.19	2.78	4.51	3.09	1.84	2.20	1355	730	770	2.1	<2	<5	222	<10
7		DDH-00-340C(990-995)-7	1.7	2.83	0.0142	0.0144	0.55	0.5	80	30	60	170	34	<0.5	399	1.1	8	12	11	260	285	0.71	8.64	3.53	7.01	2.14	1.54	3.03	1070	610	880	2.4	<2	<5	269	<10
8		DDH-00-340C(965-974.5)-8	2.08	2.89	0.0198	0.012	1.74	1.49	60	30	30	120	21	0.5	348	1.7	12	30	17	164	204	0.45	7.56	0.69	5.27	2.90	1.48	1.71	327	670	600	2.0	<2	<5	173	<10
9	1 Dup	DDH-99-320C(830-850)-9D	6	3.00	0.0365	0.0174	0.09	0.09	160	210	40	410	48	<0.5	118	<0.5	<1	2	<5	140	190	1.33	8.90	6.28	8.58	0.90	2.15	3.26	1150	1280	260	1.0	<2	<5	374	<10
10		DDH-00-340C(765-780)-10D	4.255	3.04	0.0707	0.0304	1.68	1.41	100	210	260	570	96	<0.5	116	<0.5	6	13	11	489	358	1.11	8.76	5.10	12.05	0.58	2.02	4.46	1075	820	170	0.7	<2	<5	220	<10
11		DDH-26043&26027(1501&740-1506&745)-11			0.04	0.02	2.47	2.46	<5	7.00	8.00	15	48.00	<0.5	413.00	1.80	15.00	10.00	42.00	288.00	343	0.85	8.67	2.13	9.69	1.07	1.16	3.30	747	380	360	2.1	<2	<5	252	<10
12	Blank	Method Blank - 12																																			
13		DDH-00-340C(595-615)-13	5.55	3.07	0.0141	0.0365	0.04	0.04	70	140	10	220	65	<0.5	92	<0.5	<1	<2	<5	174	97	0.53	8.40	6.10	7.78	0.36	2.05	4.96	994	500	120	0.5	<2	<5	279	<10
14		DDH-00-334C(580-600)-14	4.95	3.26	0.0278	0.0319	0.06	0.05	150	170	30	350	63	<0.5	111	<0.5	<1	2	<5	164	137	0.88	8.84	6.36	9.41	0.45	2.19	4.99	1205	800	150	0.7	<2	<5	270	<10
15		DDH-00-334C(640-660)-15	5.75	3.18	0.0311	0.0497	0.07	0.08	170	240	30	440	77	<0.5	139	<0.5	<1	7	<5	98	173	1.30	7.46	5.03	11.35	0.59	1.78	6.95	1415	1160	170	0.9	<2	<5	207	<10
16		DDH-00-347C(795-815)-16	5.167	3.18	0.0446	0.0389	0.07	0.07	150	240	150	540	79	0.5	125	<0.5	<1	4	<5	444	170	0.91	7.43	4.90	10.50	0.48	1.92	5.83	1280	840	140	0.7	<2	<5	237	<10
17		DDH-00-350C(580-600)-17	5.35	2.90	0.0423	0.0298	0.19	0.18	110	90	30	230	59	<0.5	146	<0.5	<1	8	<5	140	237	1.57	7.70	3.92	10.70	1.32	1.92	4.95	1465	1300	420	1.3	<2	<5	212	<10
18		DDH-00-327C(225-245)-18	4.98	2.96	0.0324	0.0156	0.44	0.39	110	80	40	230	51	<0.5	132	<0.5	2	4	<5	124	148	0.65	9.49	4.39	7.76	0.94	1.81	4.37	813	660	220	1.3	<2	<5	241	<10
19		DDH-00-371C(435-440)-19	2.49	3.11	0.0563	0.0267	0.88	0.8	160	280	50	490	61	<0.5	182	<0.5	3	10	19	118	515	3.27	7.44	4.94	11.85	0.88	1.98	3.14	1425	2990	260	1.1	<2	<5	221	<10
20	10 Dup	DDH-00-340C(765-780)-20	4.255	3.04	0.0707	0.0304	1.68	1.41	100	210	260	570	96	<0.5	116	<0.5	6	13	11	489	358	1.11	8.76	5.10	12.05	0.58	2.02	4.46	1075	820	170	0.7	<2	<5	220	<10
21		DDH-00357C(335-340)-21	1.2	3.06	0.0269	0.0255	0.08	0.08	<5	7	1	8	57	<0.5	126	<0.5	<1	7	<5	116	176	1.31	9.26	6.10	9.05	0.63	1.94	4.36	1210	1360	200	0.9	<2	<5	246	<10
22		DDH-00326C(680-685)-22	1.45	3.27	0.105	0.023	0.30	0.29	18	99	11	128	47	<0.5	157	0.9	3	8	<5	93	293	2.15	7.49	6.03	11.55	0.69	1.88	3.13	1540	2020	250	1.2	<2	<5	217	<10
23		DDH-00-357C(535-540)-23	1.24	3.21	0.1015	0.0257	0.2	0.21	16	47	5	68	56	<0.5	148	<0.5	1	7	<5	108	240	1.80	8.81	6.11	11.00	0.85	2.00	3.61	1445	1760	260	1.4	<2	<5	236	<10
24		DDH-99-318C(725-735)-24	2.81	3.17	0.0534	0.0196	0.72	0.63	150	280	50	480	52	<0.5	174	<0.5	3	7	<5	115	279	1.96	7.90	4.69	11.10	0.85	1.95	3.02	1360	2110	280	1.5	<2	<5	209	<10
25		DDH-99-317C(460-470)-25	2.47	3.12	0.0668	0.0224	1.24	1.02	90	220	40	350	49	0.5	257	<0.5	6	8	10	221	301	1.38	9.52	2.93	10.20	0.87	1.41	3.22	842	930	380	2.5	<2	<5	181	<10
26		DDH-00-366C(185-205)-26	8.75	2.96	0.0082	0.0208	0.02	0.02	50	150	<1	200	41	<0.5	61	<0.5	<1	<2	<5	85	64	0.40	8.75	7.19	4.94	0.15	2.24	2.70	665	290	100	<0.5	<2	<5	309	<10
27		DDH-00-366C(230-240)-27	4.48	3.01	0.0125	0.0171	0.02	0.03	<5	100	10	110	39	<0.5	66	<0.5	<1	<2	<5	167	109	0.61	9.13	7.25	5.41	0.21	2.34	2.65	740	360	110	<0.5	<2	<5	307	<10
28		DDH-99-320C(165-175)-28	2.893	2.99	0.014	0.025	0.03	0.03	60	130	10	200	51	<0.5	83	<0.5	<1	<2	<5	207	123	0.68	9.35	6.36	7.01	0.29	2.11	3.83	892	300	110	0.5	<2	<5	271	<10
29		DDH-99-318C(250-370)-29	5.15	3.03	0.0139	0.0311	0.04	0.03	<5	50	<1	50	56	<0.5	90	<0.5	<1	<2	<5	124	110	0.71	8.57	6.68	7.52	0.29	1.96	4.49	982	700	120	0.5	<2	<5	276	<10
30		DDH-00-373C(95-115)-30	10.54	3.18	0.0226	0.0477	0.04	0.04	110	260	50	420	83	<0.5	114	<0.5	<1	<2	<5	164	104	0.60	7.67	5.07	10.30	0.34	1.80	6.91	1295	570	110	0.5	<2	<5	219	<10
31		DDH-00-373C(75-95)-31	9.94	3.11	0.0286	0.0432	0.06	0.06	290	1010	120	1420	75	<0.5	113	<0.5	<1	<2	<5	162	120	0.79	8.46	5.16	10.15	0.43	1.91	6.57	1265	770	130	0.6	<2	<5	225	<10
32		DDH-00-357C(110-130)-32	5.305	3.08	0.0427	0.0356	0.08	0.08	290	920	90	1300	67	<0.5	112	<0.5	<1	2	<5	152	145	0.95	8.38	5.53	9.45	0.48	2.05	5.29	1195	940	150	0.7	<2	<5	245	<10
33		DDH-99-320C(315-330)-33	3.75	3.11	0.0355	0.0213	0.07	0.06	210	190	50	450	48	<0.5	99	<0.5	<1	4	<5	171	148	1.03	9.05	6.16	7.77	0.47	2.27	3.82	1020	1110	160	0.7	<2	<5	276	<10
34		DDH-00-369C(335-345)-34	4.75	2.96	0.0433	0.0207	0.18	0.17	140	120	40	300	49	0.8	95	<0.5	<1	5	<5	62	104	0.53	9.77	6.15	7.61	0.48	2.36	3.72	924	770	170	0.6	<2	<5	283	<10
35		DDH-00-368C(460-465)-35	2.09	3.08	0.0413	0.0535	0.06	0.05	70	70	30	170	81	<0.5	114	<0.5	<1	4	5	170	106	0.57	6.75	4.03	10.55	0.28	1.12	8.24	1385	540	90	0.5	<2	<5	147	<10
36		DDH-26055(940-945)-36	1.545	3.24	0.0321	0.0495	0.06	0.05	180	90	40	310	88	<0.5	116	<0.5	<1	5	<5	224	136	0.68	6.87	4.76	11.30	0.32	1.60	7.30	1405	550	120	0.5	<2	<5	197	<10
37		DDH-00-368C(125-145)-37D	9.063	2.95	0.0182	0.0201	0.04	0.04	90	70	20	180	49	<0.5	86	<0.5	<1	<2	<5	110	116	0.71	9.38	6.82	6.99	0.31	2.34	3.34	917	620	130	0.6	<2	<5	298	<10
38		DDH-00-369C(305-325)-38D	8	2.98	0.0349	0.0299	0.25	0.23	150	120	40	310	62	<0.5	73	<0.5	<1	5	<5	118	89	0.36	9.15	6.41	6.81	0.29	2.17	3.61	838	320	120	<0.5	<2	<5	281	<10
39		DDH-26098&00-337C(145&105-148.5&110)-39			0.023	0.055	0.1	0.09	<5	7	4	11	89	<0.5	137	<0.5	<1	4	8	151	154	1.08	6.13	4.19	12.50	0.46	1.28	8.29	1570	1070	150	0.7	<2	<5	152	<10
40		DDH-00-334C(30-50)-40	4.864	2.91	0.0085	0.0262	0.02	0.02	<5	<1	<1		50	<0.5	69	<0.5	<1	2	<5	255	64	0.27	8.31	6.38	5.50	0.16	2.10	3.73	754	160	110	<0.5	<2	<5	293	<10
41	37 Dup	DDH-00-368C(125-145)-41	9.063	2.95	0.0182	0.0201	0.04	0.04	90	70	20	180	49	<0.5	86	<0.5	<1	<2	<5	110	116	0.71	9.38	6.82	6.99	0.31	2.34	3.34	917	620	130	0.6	<2	<5	298	<10
42		DDH-00-368C(20-40)-42	9.5	2.79	0.022	0.0108	0.04	0.04	50	20	20	90	32	<0.5	72	<0.5	<1	<2	7	128	147	0.95	9.86	7.23	5.59	0.35	2.52	2.18	754	930	150	0.6	<2	<5	307	<10
43		DDH-00-366C(35-55)-43	10.189	3.04	0.0054	0.0238	0.02	0.01	100	<1	<1	100	54	<0.5	67	<0.5	<1	<2	<5	133	56	0.30	8.79	6.40	6.16	0.17	2.09	4.34	810	290	90	<0.5	<2	<5	286	<10
44		DDH-00-334C(110-130)-44	5.05	3.08	0.0135	0.0363	0.04	0.03	100	80	10	190	63	<0.5	81	<0.5	<1	<2	5	195	79	0.29	8.77	5.88	7.53	0.23	1.98	5.00	950	180	80	<0.5	<2	<5	264	<10
45		DDH-00-347C(155-175)-45	5.894	3.01	0.0162	0.018	0.06	0.03	110	60	70	240	53	<0.5	89	<0.5	<1	<2	<5	179	142	0.72	8.99	6.93	7.32	0.28	2.28	3.77	998	480	120	0.5	<2	<5	291	<10
46		DDH-00-347C(280-300)-46	5.21	3.05	0.0355	0.0211	0.06	0.05	180	180	40	400	47	<0.5	72	<0.5	<1	<2	<5	115	91	0.53	8.67	6.54	6.15	0.24	2.14	3.18	830	500	110	<0.5	<2	<5	271	<10
47		DDH-00-326C(60-70)-47	2.661	3.15	0.043	0.0413	0.14	0.13	220	150	40	410	74	<0.5	100	<0.5	<1	4	<5	180	118	0.70	7.52	5.45	8.97	0.33	1.85	5.26	1140	620	110	0.5	<2	<5	230	<10
48	38 Dup	DDH-00-369C(305-325)-48	8	2.98	0.0349	0.0299	0.25	0.23	150	120	40	310	62	<0.5	73	<0.5	<1	5	<5	118	89	0.36	9.15	6.41	6.81	0.29	2.17	3.61	838	320	120	<0.5	<2	<5	281	<10
49		DDH-00-367C(50-65)-49	7	2.98	0.0108	0.0192	0.03	0.02	430	520	10	960	48	<0.5	71	<0.5	<1	<2	<5	178	91	0.43	8.30	6.91	5.69	0.24	2.07	3.19	799	310	90	<0.5	<2	<5	302	<10
50		DDH-00-367C(260-280)-50	9.281	3.08	0.018	0.0254	0.04	0.03	80	40	20	140	58	<0.5	90	<0.5	<1	<2	<5	233	114	0.55	7.95	5.97	7.75	0.31	1.97	4.30	1040	460	110	<0.5	<2	<5	255	<10
51		DDH-00-367C(290-310)-51	9.343	3.03	0.0198	0.0249	0.04	0.04	70	20	30	120	51	<0.5	85	<0.5	<1	5	5	192	110	0.62	8.17	6.47	6.56	0.31	2.04	3.65	914	490	120	<0.5	<2	<5	275	<10
52		DDH-00-370C(20-30)-52	4.13	2.98	0.0177	0.0137	0.08	0.08	130	50	<1	180	52	<0.5	95	<0.5	<1	3	<5	178	230	0.91	8.70	6.95	8.32	0.52	1.96	4.48	1120	2220	150	0.7	<2	<5	246	<10
53		DDH-00-369C(20-30)-53	4.18	2.98	0.0577	0.0303	0.21	0.17	130	120	60	310	55	<0.5	86	<0.5	<1	<2	<5	176	122	0.70	8.58	6.34	7.07	0.30	2.01	3.56	908	310	120	0.5	<2	<5	269	<10
54		DDH-00-367C(170-175)-54	2.159	2.99	0.0331	0.0267	0.51	0.4	80	120	70	270	62	<0.5	105	<0.5	1	6	<5	238	203	0.91	7.83	5.32	7.85	0.55	1.72	3.55	1015	570	150	0.6	<2	<5	246	<10
55		DDH-00-367C(395-400)-55	2.45	3.53	0.112	0.0363	0.77	0.63	160	360	70	590	110	<0.5	211	<0.5	3	14	<5	1670	730	1.95	4.18	2.47	14.05	0.53	1.08	6.16	1695	510	110	0.9	<2	<5	130	<10
56		DDH-26064(44-54)-56	1.727	3.05	0.0104	0.0426	0.02	0.01	<5	10	10	20	72	<0.5	74	<0.5	<1	<2	<5	165	35	0.13	7.28	5.70	6.87	0.19	1.79	5.40	918	80	60	<0.5	<2	<5	264	<10
57	51 Dup	DDH-00-367C(290-310)-57D	9.343	3.03	0.0198	0.0249	0.04	0.04	70	20	30	120	51	<0.5	85	<0.5	<1	5	5	192	110	0.62	8.17	6.47	6.56	0.31	2.04	3.65	914	490	120	<0.5	<2	<5	275	<10
58		DDH-00-364C(210-229)-58D	8.878	2.77	0.0155	0.0186	3.79	3.6	<5	30	30	60	26	1.1	899	6.5	29	46	31	128	222	0.08	7.80	0.37	6.90	3.70	1.20	1.40	197	530	410	1.7	<2	<5	153	<10
59		DDH-26064(264&146-269&156)-59			0.027	0.029	0.06	0.06	<5	10	12	22	58	<0.5	82	<0.5	<1	<2	8	162	69	0.30	9.28	5.04	7.55	0.28	1.66	5.11	978	180	90	<0.5	<2	<5	258	<10
60		DDH-26056(110-125)-60	2.5	3.06	0.0293	0.0401	0.04	0.02	110	240	70	420	79	<0.5	100	<0.5	<1	<2	<5	102	60	0.31	7.48	5.23	9.69	0.24	1.89	5.74	1260	260	80	<0.5	<2	<5	251	<10
62		DDH-00-361C(737-749)-62	5.335	2.92	0.0152	0.0121	2	1.77	<5	20	30	50	25	<0.5	291	0.9	12	24	39	170	188	0.40	8.20	1.98	6.62	1.99	1.30	2.56	809	600	500	2.2	<2	<5	191	<10
63	58 Dup	DDH-00-364C(210-229)-63	8.878	2.77	0.0155	0.0186	3.79	3.6	<5	30	30	60	26	1.1	899	6.5	29	46	31	128	222	0.08	7.80	0.37	6.90	3.70	1.20	1.40	197	530	410	1.7	<2	<5	153	<10
64		DDH-00-337C(510-520)-64	2.55	2.84	0.0198	0.0168	5.68	3.78	<5	30	30	60	32	1	522	2.9	26	41	27	122	178	0.13	7.66	0.45	8.45	3.01	1.49	1.34	180	450	490	1.8	<2	<5	148	<10
74		26029(interval 815-825)-74			0.0055	0.0114	0.02	0.01	<5	1	18	19	27	<0.5	44	<0.5	<1	<2	<5	55	51	0.34	9.31	7.94	3.63	0.15	2.37	1.58	469	280	100	<0.5	<2	<5	358	<10
78		26056(135-153)-78			0.0389	0.0439	0.05	0.04	320	820	250	1390	73	<0.5	104	<0.5	<1	<2	<5	133	66	0.35	7.94	5.44	8.39	0.30	1.86	5.49	1130	290	90	<0.5	<2	<5	265	<10

					AQUA REGIA DIGESTIONS WITHOUT LECO AND PGE								AQUA REGIA DIGESTIONS WITHOUT LECO AND PGE																	
					ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41
					Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia
HCT ID	Comment	HCT Full ID (Drill Hole, Footage, HCT)	Certificate Number	Lab Date	Cu	Ni	S ICP	Co	Ag	Zn	Cd	Mo	Pb	As	Cr	V	Ti	Al	Ca	Fe	K	Na	Mg	Mn	P	B	Ba	Be	Bi	Ga	Hg
					%	%	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	%	%	%	%	%	%	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
1		DDH-99-320C(830-850)-1	VA05044293 - Finalized	20050616	0.032	0.014	0.100	34	<0.2	72	<0.5	<1	<2	3	122	84	0.22	4.20	2.61	5.58	0.32	0.50	1.98	607	1230	10	80	<0.5	<2	10	<1
2		DDH-00-361C(310-320)-2	VA05044293 - Finalized	20050616	0.014	0.019	0.170	36	<0.2	46	<0.5	<1	<2	2	36	16	0.07	5.94	3.59	4.56	0.12	0.81	2.68	492	420	<10	40	<0.5	<2	10	1
3		DDH-00-361C(345-350)-3	VA05044293 - Finalized	20050616	0.022	0.025	0.050	55	<0.2	71	<0.5	<1	<2	<2	36	18	0.09	4.22	2.53	6.91	0.14	0.60	4.54	778	430	<10	40	<0.5	<2	<10	<1
4		DDH-00-343C(240-250)-4	VA05044293 - Finalized	20050616	0.069	0.018	0.720	34	<0.2	67	<0.5	5	2	8	76	107	0.22	1.73	1.43	3.80	0.24	0.30	0.84	255	2150	<10	40	<0.5	<2	<10	<1
5		DDH-26030(1047-1052)-5	VA05058859 - Finalized	20050726	0.010	0.007	0.260	11	<0.2	33	0.7	2	4	17	39	26	0.13	2.95	8.15	1.24	0.12	0.16	0.37	799	630	10	50	0.6	<2	10	<1
6		DDH-26061(1218-1233)-6	VA05044293 - Finalized	20050616	0.013	0.011	0.460	19	0.2	181	0.5	9	16	27	154	160	0.26	2.81	1.27	3.51	0.78	0.07	1.58	381	580	20	160	0.9	<2	10	<1
7		DDH-00-340C(990-995)-7	VA05044293 - Finalized	20050616	0.013	0.014	0.580	26	<0.2	200	<0.5	8	3	12	235	232	0.46	3.47	1.74	4.23	1.22	0.08	1.78	331	490	50	370	0.9	2	10	1
8		DDH-00-340C(965-974.5)-8	VA05044293 - Finalized	20050616	0.020	0.013	1.900	21	0.4	324	1.7	14	2	26	177	205	0.28	2.85	0.25	5.37	0.97	0.07	1.37	206	540	<10	90	1	<2	10	<1
9	1 Dup	DDH-99-320C(830-850)-9D	VA05044293 - Finalized	20050616	0.032	0.014	0.100	34	<0.2	72	<0.5	<1	<2	3	122	84	0.22	4.20	2.61	5.58	0.32	0.50	1.98	607	1230	10	80	<0.5	<2	10	<1
10		DDH-00-340C(765-780)-10D	VA05044293 - Finalized	20050616	0.064	0.029	1.660	92	<0.2	83	<0.5	6	4	17	143	92	0.17	2.65	1.66	9.56	0.24	0.36	3.30	760	780	<10	50	<0.5	<2	<10	<1
11		DDH-26043&26027(1501&740-1506&745)-11	VA05099352	20051120	0.039	0.025	2.590	45	0.3	223	1.4	16	8	30	126	152	0.22	3.31	0.64	6.09	0.51	0.14	1.42	191	330	<10	110	0.9	2	10	<1
12	Blank	Method Blank - 12																													
13		DDH-00-340C(595-615)-13	VA05044293 - Finalized	20050616	0.012	0.034	0.040	58	<0.2	75	<0.5	<1	<2	<2	80	28	0.12	4.28	2.59	6.91	0.21	0.63	5.01	815	500	<10	60	<0.5	2	<10	<1
14		DDH-00-334C(580-600)-14	VA05044293 - Finalized	20050616	0.024	0.028	0.050	53	<0.2	83	<0.5	<1	2	<2	128	56	0.16	2.67	1.68	7.31	0.20	0.39	4.20	885	770	<10	60	<0.5	<2	<10	1
15		DDH-00-334C(640-660)-15	VA05044293 - Finalized	20050616	0.028	0.047	0.090	71	<0.2	106	<0.5	<1	<2	2	65	67	0.23	2.18	1.37	8.90	0.28	0.30	5.63	1065	1110	<10	70	<0.5	<2	<10	<1
16		DDH-00-347C(795-815)-16	VA05044293 - Finalized	20050616	0.040	0.037	0.070	70	<0.2	98	<0.5	<1	2	<2	289	73	0.2	2.49	1.53	8.95	0.24	0.40	5.53	1060	810	<10	70	<0.5	<2	<10	<1
17		DDH-00-350C(580-600)-17	VA05044293 - Finalized	20050616	0.039	0.028	0.200	54	<0.2	116	<0.5	<1	3	7	110	115	0.31	2.95	1.08	8.10	0.48	0.26	3.92	1100	1280	<10	120	0.5	<2	10	<1
18		DDH-00-327C(225-245)-18	VA05044293 - Finalized	20050616	0.031	0.015	0.460	49	<0.2	74	<0.5	3	<2	6	49	38	0.11	4.41	2.41	5.71	0.18	0.50	3.19	552	600	<10	50	0.5	<2	10	<1
19		DDH-00-371C(435-440)-19	VA05044293 - Finalized	20050616	0.051	0.026	0.920	40	<0.2	53	<0.5	3	2	12	32	81	0.25	1.62	1.57	3.53	0.18	0.29	0.56	207	2970	<10	40	<0.5	<2	<10	<1
20	10 Dup	DDH-00-340C(765-780)-20	VA05044293 - Finalized	20050616	0.064	0.029	1.660	92	<0.2	83	<0.5	6	4	17	143	92	0.17	2.65	1.66	9.56	0.24	0.36	3.30	760	780	<10	50	<0.5	<2	<10	<1
21		DDH-00357C(335-340)-21	VA05058859	20050726	0.027	0.000	0.080	40	<0.2	72	<0.5	<1	<2	2	94	76	0.24	3.55	2.22	5.75	0.27	0.54	2.61	635	1370	<10	70	<0.5	<2	10	1
22		DDH-00326C(680-685)-22	VA05099352	20051120	0.104	0.022	0.290	31	0.5	78	<0.5	2	4	<2	71	109	0.36	1.87	1.36	5.06	0.42	0.30	0.97	485	2020	<10	150	<0.5	4	10	<1
23		DDH-00-357C(535-540)-23	VA05058859	20050726	0.098	0.000	0.200	32	0.4	70	<0.5	1	5	<2	88	98	0.32	2.39	1.36	5.06	0.61	0.35	1.38	473	1630	<10	160	<0.5	<2	<10	1
24		DDH-99-318C(725-735)-24	VA05044293 - Finalized	20050616	0.049	0.017	0.750	29	<0.2	47	<0.5	2	5	9	42	67	0.22	2.16	1.73	3.02	0.22	0.34	0.50	203	2130	<10	50	0.5	<2	10	<1
25		DDH-99-317C(460-470)-25	VA05044293 - Finalized	20050616	0.061	0.021	1.280	37	0.3	79	<0.5	6	2	13	91	101	0.21	2.71	0.65	4.89	0.36	0.15	1.03	207	900	<10	110	0.7	<2	<10	<1
26		DDH-00-366C(185-205)-26	VA05044293 - Finalized	20050616	0.008	0.019	0.020	35	<0.2	49	<0.5	<1	<2	<2	72	23	0.09	6.14	3.77	4.68	0.15	0.92	3.03	548	290	<10	50	<0.5	<2	10	<1
27		DDH-00-366C(230-240)-27	VA05044293 - Finalized	20050616	0.012	0.015	0.030	32	<0.2	48	<0.5	<1	<2	<2	140	44	0.11	4.48	2.78	4.57	0.13	0.69	2.50	525	360	<10	40	<0.5	<2	10	<1
28		DDH-99-320C(165-175)-28	VA05044293 - Finalized	20050616	0.013	0.023	0.030	45	<0.2	67	<0.5	<1	2	<2	185	56	0.15	3.90	2.34	6.14	0.13	0.58	3.70	718	290	<10	40	<0.5	<2	10	<1
29		DDH-99-318C(250-370)-29	VA05044293 - Finalized	20050616	0.013	0.028	0.040	50	<0.2	71	<0.5	<1	<2	<2	91	39	0.12	4.86	3.03	6.37	0.16	0.69	4.22	763	700	<10	50	<0.5	<2	10	1
30		DDH-00-373C(95-115)-30	VA05044293 - Finalized	20050616	0.022	0.046	0.040	78	<0.2	102	<0.5	<1	<2	<2	130	39	0.13	2.82	1.67	9.40	0.17	0.43	6.96	1175	560	<10	50	<0.5	<2	<10	<1
31		DDH-00-373C(75-95)-31	VA05044293 - Finalized	20050616	0.027	0.041	0.060	71	<0.2	97	<0.5	<1	<2	<2	118	50	0.15	2.44	1.51	8.72	0.21	0.37	5.98	1065	770	<10	60	<0.5	<2	<10	<1
32		DDH-00-357C(110-130)-32	VA05044293 - Finalized	20050616	0.040	0.033	0.090	59	<0.2	89	<0.5	<1	3	<2	120	61	0.19	2.57	1.61	7.81	0.23	0.37	4.59	935	950	<10	70	<0.5	<2	<10	<1
33		DDH-99-320C(315-330)-33	VA05044293 - Finalized	20050616	0.033	0.019	0.070	39	<0.2	67	<0.5	<1	2	2	143	67	0.19	2.87	1.83	5.93	0.19	0.44	2.80	661	1140	<10	60	<0.5	<2	10	<1
34		DDH-00-369C(335-345)-34	VA05044293 - Finalized	20050616	0.039	0.019	0.180	46	0.3	74	<0.5	<1	<2	4	22	18	0.09	2.89	1.85	6.50	0.12	0.44	3.12	705	740	<10	40	<0.5	<2	<10	<1
35		DDH-00-368C(460-465)-35	VA05044293 - Finalized	20050616	0.039	0.049	0.060	74	<0.2	99	<0.5	<1	2	2	145	52	0.12	3.95	1.92	9.66	0.15	0.29	8.20	1265	540	20	40	<0.5	<2	<10	<1
36		DDH-26055(940-945)-36	VA05044293 - Finalized	20050616	0.030	0.046	0.060	82	<0.2	101	<0.5	<1	<2	<2	126	44	0.14	2.19	1.37	9.99	0.17	0.30	6.96	1225	530	<10	50	<0.5	<2	<10	<1
37		DDH-00-368C(125-145)-37D	VA05044293 - Finalized	20050616	0.016	0.018	0.040	40	<0.2	64	<0.5	<1	<2	2	95	49	0.15	4.14	2.53	5.85	0.16	0.62	3.00	682	600	<10	50	<0.5	<2	10	<1
38		DDH-00-369C(305-325)-38D	VA05044293 - Finalized	20050616	0.033	0.030	0.260	58	<0.2	61	<0.5	<1	<2	2	29	16	0.07	3.93	2.39	6.54	0.12	0.58	3.68	699	310	<10	30	<0.5	<2	<10	1
39		DDH-26098&00-337C(145&105-148.5&110)-39	VA05099352	20051120	0.024	0.055	0.090	85	0.2	118	<0.5	<1	7	<2	95	69	0.25	1.66	0.96	10.95	0.27	0.23	7.49	1290	1040	<10	70	<0.5	<2	<10	<1
40		DDH-00-334C(30-50)-40	VA05044293 - Finalized	20050616	0.008	0.024	0.030	47	<0.2	58	<0.5	<1	<2	<2	51	12	0.05	5.31	3.22	5.55	0.10	0.79	4.31	681	140	<10	40	<0.5	<2	10	<1
41	37 Dup	DDH-00-368C(125-145)-41	VA05044293 - Finalized	20050616	0.016	0.018	0.040	40	<0.2	64	<0.5	<1	<2	2	95	49	0.15	4.14	2.53	5.85	0.16	0.62	3.00	682	600	<10	50	<0.5	<2	10	<1
42		DDH-00-368C(20-40)-42	VA05044293 - Finalized	20050616	0.020	0.009	0.050	23	<0.2	44	<0.5	<1	2	<2	112	68	0.16	3.48	2.29	3.86	0.14	0.52	1.44	417	930	<10	50	<0.5	<2	<10	<1
43		DDH-00-366C(35-55)-43	VA05044293 - Finalized	20050616	0.005	0.022	0.020	51	<0.2	57	<0.5	<1	<2	<2	87	18	0.07	4.55	2.78	5.90	0.11	0.67	4.57	703	280	<10	40	<0.5	<2	10	<1
44		DDH-00-334C(110-130)-44	VA05044293 - Finalized	20050616	0.013	0.035	0.040	62	<0.2	74	<0.5	<1	<2	<2	32	12	0.05	3.70	2.16	7.37	0.08	0.52	5.44	903	160	<10	30	<0.5	<2	<10	<1
45		DDH-00-347C(155-175)-45	VA05044293 - Finalized	20050616	0.015	0.016	0.040	44	<0.2	65	<0.5	<1	<2	<2	102	41	0.14	3.47	2.10	6.09	0.14	0.54	3.38	718	460	<10	40	<0.5	<2	<10	1
46		DDH-00-347C(280-300)-46	VA05044293 - Finalized	20050616	0.034	0.020	0.060	41	<0.2	59	<0.5	<1	<2	<2	88	28	0.12	3.99	2.42	5.61	0.13	0.63	3.24	662	500	<10	50	<0.5	<2	<10	<1
47		DDH-00-326C(60-70)-47	VA05044293 - Finalized	20050616	0.041	0.041	0.150	68	<0.2	88	<0.5	<1	<2	<2	128	40	0.13	2.62	1.58	7.98	0.15	0.40	5.38	970	620	<10	40	<0.5	<2	<10	<1
48	38 Dup	DDH-00-369C(305-325)-48	VA05044293 - Finalized	20050616	0.033	0.030	0.260	58	<0.2	61	<0.5	<1	<2	2	29	16	0.07	3.93	2.39	6.54	0.12	0.58	3.68	699	310	<10	30	<0.5	<2	<10	1
49		DDH-00-367C(50-65)-49	VA05044293 - Finalized	20050616	0.010	0.018	0.040	41	<0.2	58	<0.5	<1	<2	<2	147	40	0.1	5.26	3.33	5.42	0.13	0.72	3.42	652	330	<10	40	<0.5	<2	10	<1
50		DDH-00-367C(260-280)-50	VA05044293 - Finalized	20050616	0.018	0.025	0.050	53	<0.2	76	<0.5	<1	<2	<2	168	50	0.14	3.59	2.15	7.01	0.19	0.51	4.41	859	510	<10	50	<0.5	<2	<10	<1
51		DDH-00-367C(290-310)-51	VA05044293 - Finalized	20050616	0.019	0.023	0.050	42	<0.2	64	<0.5	<1	3	2	72	25	0.12	4.68	3.11	5.58	0.16	0.64	3.35	674	520	<10	50	<0.5	<2	10	<1
52		DDH-00-370C(20-30)-52	VA05044293 - Finalized	20050616	0.016	0.011	0.090	40	<0.2	67	<0.5	<1	2	2	49	45	0.12	3.26	2.47	5.97	0.23	0.39	2.85	652	2140	<10	50	<0.5	<2	<10	<1
53		DDH-00-369C(20-30)-53	VA05044293 - Finalized	20050616	0.057	0.030	0.210	47	<0.2	64	<0.5	<1	<2	<2	86	33	0.15	3.90	2.31	5.99	0.19	0.59	3.04	650	320	<10	50	<0.5	<2	<10	<1
54		DDH-00-367C(170-175)-54	VA05044293 - Finalized	20050616	0.033	0.025	0.510	50	<0.2	52	<0.5	<1	4	<2	119	89	0.21	4.73	2.82	4.69	0.45	0.63	2.04	400	610	<10	100	<0.5	<2	10	<1
55		DDH-00-367C(395-400)-55	VA05044293 - Finalized	20050616	0.106	0.036	0.730	95	<0.2	113	<0.5	2	4	6	229	175	0.31	3.71	1.37	10.75	0.36	0.28	5.72	1130	540	10	70	<0.5	<2	10	<1
56		DDH-26064(44-54)-56	VA05044293 - Finalized	20050616	0.009	0.040	0.020	67	<0.2	66	<0.5	<1	<2	<2	36	7	0.03	5.27	3.14	7.32	0.07	0.76	6.74	888	80	<10	20	<0.5	<2	10	<1
57	51 Dup	DDH-00-367C(290-310)-57D	VA05044293 - Finalized	20050616	0.019	0.023	0.050	42	<0.2	64	<0.5	<1	3	2	72	25	0.12	4.68	3.11	5.58	0.16	0.64	3.35	674	520	<10	50	<0.5	<2	10	<1
58		DDH-00-364C(210-229)-58D	VA05044293 - Finalized	20050616	0.016	0.020	4.100	27	0.9	918	6.5	27	17	42	76	107	0.01	2.67	0.09	7.09	0.40	0.04	1.30	179	230	20	30	1.4	<2	10	<1
59		DDH-26064(264&146-269&156)-59	VA05099352	20051120	0.026	0.026	0.060	55	<0.2	70	<0.5	<1	6	3	58	20	0.07	5.56	2.70	6.89	0.11	0.72	4.72	821	170	10	30	<0.5	3	10	1
60		DDH-26056(110-125)-60	VA05044293 - Finalized	20050616	0.027	0.039	0.030	72	<0.2	90	<0.5	<1	<2	<2	94	28	0.06	2.66	1.60	9.45	0.07	0.43	6.49	1185	270	<10	20	<0.5	<2	<10	<1
62		DDH-00-361C(737-749)-62	VA05044293 - Finalized	20050616	0.015	0.013	2.080	24	0.3	252	1.1	12	5	41	149	157	0.19	3.12	0.79	5.80	0.66	0.09	1.94	377	470	10	100	1.2	<2	10	<1
63	58 Dup	DDH-00-364C(210-229)-63	VA05044293 - Finalized	20050616	0.016	0.020	4.100	27	0.9	918	6.5	27	17	42	76	107	0.01	2.67	0.09	7.09	0.40	0.04	1.30	179	230	20	30	1.4	<2	10	<1
64		DDH-00-337C(510-520)-64	VA05044293 - Finalized	20050616	0.021	0.019	5.330	36	1.0	554	3.3	28	8	45	78	99	0.02	2.14	0.06	9.00	0.32	0.06	0.94	125	120	<10	30	1.2	<2	10	<1
74		26029(interval 815-825)-74	VA05068941	20050901	0.005	0.011	<0.01	21	<0.2	33	<0.5	<1	<2	<2	43	25	0.09	6.12	3.56	3.22	0.14	0.94	1.71	351	270	<10	40	<0.5	<2	10	<1
78		26056(135-153)-78	VA05044293 - Finalized	20050616	0.038	0.043	0.050	72	<0.2	94	<0.5	<1	<2	3	95	27	0.08	3.64	2.08	8.15	0.12	0.51	5.97	1035	290	<10	30	<0.5	<2	<10	<1

			AQUA REGIA DIGESTIONS WITHOUT LECO AND PGE							WHOLE ROCK DATA															ABA RESULTS			
			ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP41	ME-ICP06	ME-ICP06	ME-ICP06	ME-ICP06	Fe-VOL05	ME-ICP06	ME-ICP06	ME-ICP06	ME-ICP06	ME-ICP06	ME-ICP06	ME-ICP06	ME-ICP06	ME-ICP06	OA-GRA05	TOT-ICP06				
			Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Whole Rock	Whole Rock	Whole Rock	Whole Rock	Whole Rock	Whole Rock	Whole Rock	Whole Rock	Whole Rock	Whole Rock	Whole Rock	Whole Rock	Whole Rock	Whole Rock	Whole Rock	Whole Rock					
HCT ID	Comment	HCT Full ID (Drill Hole, Footage, HCT)	La	Sb	Sc	Sr	W	Tl	U	SiO_2	Al_2O_3	TiO_2	Fe_2O_3	FeO	CaO	MgO	MnO	Na_2O	K_2O	P_2O_5	Cr_2O_3	BaO	SrO	LOI	Total	C (%)	CO_2	Cu/S (wt/wt)	Ni/S
			ppm	ppm	ppm	ppm	ppm	ppm	ppm	%	%	%	%	%	%	%	%	%	%	%	%	%	%		%				
1		DDH-99-320C(830-850)-1	10	2	3	93	<10	<10	<10	46.5	18.5	2.25	13.1	10.8	9.60	5.94	0.16	2.62	1.10	0.21	0.03	0.03	0.04	0.49	101	<0.05	0.2	0.41	0.19
2		DDH-00-361C(310-320)-2	<10	<2	2	143	<10	<10	<10	46.5	23.1	0.88	9.0	7.8	10.60	6.31	0.10	2.66	0.38	0.01	0.06	0.01	0.04	0.41	100	<0.05	<0.2	0.08	0.12
3		DDH-00-361C(345-350)-3	<10	<2	3	99	<10	<10	<10	46.0	19.3	0.84	12.7	10.8	9.09	9.52	0.14	2.44	0.37	0.23	0.04	0.01	0.03	0.06	101	<0.05	<0.2	0.47	0.55
4		DDH-00-343C(240-250)-4	20	<2	6	40	<10	<10	<10	49.0	14.1	4.16	16.5	14.1	6.38	4.89	0.18	2.61	1.22	0.43	0.03	0.03	0.03	1.13	101	0.09	0.3	0.11	0.03
5		DDH-26030(1047-1052)-5	10	<2	2	94	10	<10	<10	48.0	14.3	0.74	8.3	6.9	20.10	3.92	0.35	1.20	0.63	0.13	0.03	0.09	0.04	2.19	100	0.26	1	0.04	0.04
6		DDH-26061(1218-1233)-6	20	2	12	29	<10	<10	<10	59.1	16.0	0.92	6.8	5.8	4.23	3.77	0.19	2.23	3.44	0.15	0.03	0.09	0.03	3.74	101	0.09	0.3	0.03	0.03
7		DDH-00-340C(990-995)-7	20	2	15	38	<10	<10	<10	51.8	18.3	1.19	10.6	9.7	5.34	5.64	0.15	1.98	2.70	0.02	0.04	0.11	0.03	2.08	100	0.05	0.2	0.03	0.03
8		DDH-00-340C(965-974.5)-8	10	2	20	14	<10	<10	<10	59.9	16.6	0.83	8.2	7.6	1.18	3.23	0.05	2.09	3.71	0.18	0.03	0.07	0.02	2.94	99	<0.05	<0.2	0.01	0.01
9	1 Dup	DDH-99-320C(830-850)-9D	10	2	3	93	<10	<10	<10	46.5	18.5	2.25	13.1	10.8	9.60	5.94	0.16	2.62	1.10	0.21	0.03	0.03	0.04	0.49	101	<0.05	0.2	0.41	0.19
10		DDH-00-340C(765-780)-10D	10	2	3	52	<10	<10	<10	43.0	15.8	1.79	18.1	16.0	7.29	7.72	0.13	2.33	0.59	0.12	0.09	0.02	0.03	0.55	98	<0.05	<0.2	0.04	0.02
11		DDH-26043&26027(1501&740-1506&745)-11	10	<2	11	39	<10	<10	<10	48.9	18.9	1.55	16.0	13.8	3.24	6.41	0.11	1.58	1.49	0.09	0.05	0.05	0.03	2.44	101	<0.05	<0.2	0.02	0.01
12	Blank	Method Blank - 12																											
13		DDH-00-340C(595-615)-13	<10	<2	3	104	<10	<10	<10	46.4	18.8	0.97	11.9	10.4	9.14	9.79	0.14	2.48	0.39	<0.01	0.03	0.01	0.03	-0.25	100	<0.05	<0.2	0.35	0.91
14		DDH-00-334C(580-600)-14	10	<2	2	60	<10	<10	<10	45.2	18.0	1.49	13.7	11.5	9.37	9.05	0.16	2.46	0.53	0.15	0.03	0.02	0.03	-0.59	100	<0.05	<0.2	0.46	0.53
15		DDH-00-334C(640-660)-15	10	<2	3	46	<10	<10	<10	44.4	13.5	2.18	16.8	14.2	7.56	11.95	0.20	2.16	0.61	0.08	0.03	0.02	0.02	0.23	100	<0.05	<0.2	0.44	0.71
16		DDH-00-347C(795-815)-16	10	2	3	60	<10	<10	<10	45.3	16.1	1.56	16.0	13.7	7.85	11.20	0.18	2.47	0.53	0.08	0.09	0.02	0.03	-0.78	101	<0.05	<0.2	0.64	0.56
17		DDH-00-350C(580-600)-17	10	<2	10	41	<10	<10	<10	45.0	15.0	2.77	16.4	13.9	6.14	8.99	0.21	2.35	1.51	0.20	0.03	0.05	0.03	1.53	100	<0.05	<0.2	0.22	0.16
18		DDH-00-327C(225-245)-18	10	<2	4	94	<10	<10	<10	49.2	18.9	1.18	11.9	10.5	6.66	7.87	0.11	2.33	1.16	0.08	0.02	0.03	0.03	0.93	101	<0.05	<0.2	0.07	0.04
19		DDH-00-371C(435-440)-19	20	<2	4	38	<10	<10	<10	45.7	13.5	5.62	18.1	15.8	7.72	5.48	0.21	2.43	0.99	0.68	0.02	0.03	0.03	0.23	101	<0.05	<0.2	0.06	0.03
20	10 Dup	DDH-00-340C(765-780)-20	10	2	3	52	<10	<10	<10	43.0	15.8	1.79	18.1	16.0	7.29	7.72	0.13	2.33	0.59	0.12	0.09	0.02	0.03	0.55	98	<0.05	<0.2	0.04	0.02
21		DDH-00357C(335-340)-21	10	<2	2	85	<10	<10	<10	46.1	17.3	2.13	14.0	11.5	9.31	7.85	0.16	2.65	0.77	0.28	0.02	0.02	0.03	0.19	101	<0.05	<0.2	0.34	0.32
22		DDH-00326C(680-685)-22	10	<2	3	41	<10	<10	<10	46.0	13.9	3.75	18.3	15.4	8.73	5.49	0.22	2.44	0.85	0.44	0.02	0.03	0.02	-0.63	100	<0.05	<0.2	0.35	0.08
23		DDH-00-357C(535-540)-23	10	<2	3	52	<10	<10	<10	47.0	15.5	2.91	16.3	13.9	8.99	6.32	0.19	2.70	1.02	0.38	0.02	0.04	0.03	-0.31	101	<0.05	<0.2	0.51	0.13
24		DDH-99-318C(725-735)-24	20	<2	3	43	<10	<10	<10	49.5	14.5	3.41	16.9	15.1	7.30	5.28	0.20	2.44	0.94	0.28	0.02	0.03	0.03	0.11	101	<0.05	<0.2	0.07	0.03
25		DDH-99-317C(460-470)-25	10	<2	4	22	<10	<10	<10	47.7	19.9	2.40	16.2	14.3	4.58	6.07	0.13	1.78	1.09	0.06	0.04	0.05	0.02	0.80	101	<0.05	<0.2	0.05	0.02
26		DDH-00-366C(185-205)-26	<10	<2	2	149	<10	<10	<10	47.6	22.9	0.73	8.0	6.6	11.05	6.01	0.09	2.83	0.32	<0.01	0.02	0.01	0.04	0.02	100	<0.05	<0.2	0.41	1.04
27		DDH-00-366C(230-240)-27	<10	<2	1	108	<10	<10	10	48.1	22.3	1.10	8.5	7.1	11.15	5.62	0.10	2.93	0.37	0.02	0.04	0.01	0.04	-0.13	100	<0.05	<0.2	0.63	0.86
28		DDH-99-320C(165-175)-28	<10	<2	2	89	<10	<10	<10	46.4	20.0	1.22	11.1	9.1	10.15	7.61	0.13	2.63	0.38	<0.01	0.05	0.01	0.03	0.02	100	<0.05	<0.2	0.47	0.83
29		DDH-99-318C(250-370)-29	10	<2	2	116	<10	<10	<10	45.8	19.4	1.28	11.6	9.7	9.99	8.68	0.14	2.38	0.38	0.08	0.03	0.01	0.03	-0.08	100	<0.05	<0.2	0.35	0.78
30		DDH-00-373C(95-115)-30	<10	<2	3	63	<10	<10	<10	45.4	15.9	1.05	15.7	13.3	7.83	12.65	0.18	2.27	0.36	0.05	0.04	0.01	0.03	-0.53	101	<0.05	<0.2	0.57	1.19
31		DDH-00-373C(75-95)-31	10	<2	3	54	<10	<10	<10	45.3	16.1	1.36	15.3	13.2	7.98	11.55	0.18	2.39	0.48	0.22	0.04	0.01	0.03	-0.46	101	<0.05	<0.2	0.48	0.72
32		DDH-00-357C(110-130)-32	10	2	3	55	<10	<10	<10	45.4	16.9	1.66	14.7	12.4	8.69	9.83	0.17	2.54	0.56	0.23	0.04	0.02	0.03	-0.32	101	<0.05	<0.2	0.53	0.45
33		DDH-99-320C(315-330)-33	10	<2	2	66	<10	<10	<10	46.4	18.5	1.81	12.0	9.9	9.43	7.00	0.14	2.81	0.59	0.08	0.04	0.02	0.03	-0.05	99	<0.05	<0.2	0.51	0.30
34		DDH-00-369C(335-345)-34	10	<2	2	65	<10	<10	<10	47.4	20.0	0.93	11.9	10.1	9.05	7.07	0.13	3.01	0.60	0.11	0.01	0.02	0.04	0.04	101	<0.05	<0.2	0.24	0.12
35		DDH-00-368C(460-465)-35	10	2	3	53	<10	<10	<10	42.7	13.2	0.96	15.8	13.3	6.34	14.60	0.19	1.39	0.30	<0.01	0.04	0.01	0.02	4.01	100	<0.05	<0.2	0.69	0.89
36		DDH-26055(940-945)-36	<10	3	3	46	<10	<10	<10	44.1	14.4	1.22	17.2	14.4	7.52	13.60	0.20	2.10	0.36	0.06	0.05	0.01	0.02	0.27	101	<0.05	<0.2	0.54	0.83
37		DDH-00-368C(125-145)-37D	<10	<2	2	94	<10	<10	<10	47.8	20.9	1.22	10.8	9.0	10.40	6.58	0.13	2.87	0.44	0.15	0.03	0.01	0.04	-0.23	101	<0.05	<0.2	0.46	0.50
38		DDH-00-369C(305-325)-38D	<10	<2	2	85	<10	<10	<10	47.2	20.5	0.67	11.1	9.4	9.72	7.50	0.12	2.86	0.40	0.01	0.03	0.01	0.03	0.33	101	<0.05	0.2	0.14	0.12
39		DDH-26098&00-337C(145&105-148.5&110)-39	10	<2	4	33	<10	<10	<10	43.3	11.3	1.87	19.7	15.9	6.34	15.10	0.22	1.78	0.58	0.23	0.03	0.02	0.02	-0.11	101	<0.05	0.2	0.23	0.55
40		DDH-00-334C(30-50)-40	<10	3	2	123	<10	<10	<10	47.2	21.7	0.50	9.1	7.4	10.10	8.14	0.11	2.74	0.29	<0.01	0.06	0.01	0.04	1.14	101	0.05	0.2	0.43	1.31
41	37 Dup	DDH-00-368C(125-145)-41	<10	<2	2	94	<10	<10	<10	47.8	20.9	1.22	10.8	9.0	10.40	6.58	0.13	2.87	0.44	0.15	0.03	0.01	0.04	-0.23	101	<0.05	<0.2	0.46	0.50
42		DDH-00-368C(20-40)-42	<10	<2	1	80	<10	<10	<10	47.6	22.3	1.67	8.9	7.1	11.40	4.30	0.11	3.23	0.55	0.23	0.03	0.02	0.04	-0.09	100	<0.05	<0.2	0.55	0.27
43		DDH-00-366C(35-55)-43	<10	2	2	110	<10	<10	<10	46.8	20.8	0.54	9.7	8.0	9.95	8.57	0.11	2.66	0.27	0.02	0.03	0.01	0.04	0.01	100	<0.05	<0.2	0.27	1.19
44		DDH-00-334C(110-130)-44	<10	<2	3	79	<10	<10	<10	45.7	19.1	0.52	11.9	10.0	8.73	9.81	0.13	2.43	0.27	<0.01	0.05	0.01	0.03	0.42	99	<0.05	<0.2	0.34	0.91
45		DDH-00-347C(155-175)-45	<10	<2	2	81	<10	<10	<10	47.9	19.9	1.25	11.2	9.4	10.30	7.32	0.14	2.79	0.38	<0.01	0.04	0.01	0.03	-0.43	101	<0.05	<0.2	0.27	0.30
46		DDH-00-347C(280-300)-46	<10	<2	2	95	<10	<10	<10	47.5	20.9	1.01	10.1	8.5	10.40	6.74	0.12	2.86	0.36	0.01	0.03	0.01	0.04	-0.22	100	<0.05	<0.2	0.59	0.35
47		DDH-00-326C(60-70)-47	<10	<2	3	56	<10	<10	<10	45.3	16.9	1.29	14.5	12.2	8.58	10.85	0.16	2.45	0.40	0.11	0.04	0.01	0.03	-0.10	101	<0.05	<0.2	0.31	0.30
48	38 Dup	DDH-00-369C(305-325)-48	<10	<2	2	85	<10	<10	<10	47.2	20.5	0.67	11.1	9.4	9.72	7.50	0.12	2.86	0.40	0.01	0.03	0.01	0.03	0.33	101	<0.05	0.2	0.14	0.12
49		DDH-00-367C(50-65)-49	<10	<2	2	117	<10	<10	<10	46.4	21.0	0.76	9.2	7.6	10.25	6.92	0.11	2.61	0.38	0.06	0.04	0.01	0.04	0.59	98	<0.05	<0.2	0.36	0.64
50		DDH-00-367C(260-280)-50	<10	<2	2	80	<10	<10	<10	45.9	18.8	1.03	12.8	10.4	9.11	9.01	0.15	2.57	0.45	0.09	0.06	0.01	0.03	0.19	100	<0.05	<0.2	0.45	0.64
51		DDH-00-367C(290-310)-51	<10	<2	2	100	<10	<10	<10	46.9	19.3	1.14	10.9	8.6	9.93	7.65	0.13	2.65	0.49	0.06	0.05	0.01	0.03	1.21	101	<0.05	0.2	0.50	0.62
52		DDH-00-370C(20-30)-52	10	<2	2	58	<10	<10	<10	45.1	16.9	1.55	12.7	10.3	10.25	8.25	0.15	2.41	0.61	0.43	0.04	0.02	0.03	1.78	100	0.1	0.4	0.22	0.17
53		DDH-00-369C(20-30)-53	<10	<2	2	92	<10	<10	<10	47.2	19.5	1.32	11.7	9.6	9.68	7.24	0.13	2.70	0.40	0.04	0.04	0.01	0.03	0.13	100	<0.05	<0.2	0.27	0.14
54		DDH-00-367C(170-175)-54	10	<2	3	106	<10	<10	<10	47.8	17.7	1.64	12.5	11.0	7.90	7.22	0.14	2.32	0.68	0.14	0.05	0.02	0.03	0.67	99	0.05	0.2	0.06	0.05
55		DDH-00-367C(395-400)-55	<10	2	11	41	<10	<10	<10	38.1	11.8	3.59	22.6	17.6	3.67	12.65	0.24	1.40	0.57	0.08	0.33	0.01	0.02	5.21	101	0.07	0.2	0.15	0.05
56		DDH-26064(44-54)-56	<10	<2	3	121	<10	<10	<10	45.6	19.0	0.26	11.6	9.4	8.54	11.70	0.12	2.25	0.26	0.01	0.04	0.01	0.03	0.93	101	0.06	0.2	0.52	2.13
57	51 Dup	DDH-00-367C(290-310)-57D	<10	<2	2	100	<10	<10	<10	46.9	19.3	1.14	10.9	8.6	9.93	7.65	0.13	2.65	0.49	0.06	0.05	0.01	0.03	1.21	101	<0.05	0.2	0.50	0.62
58		DDH-00-364C(210-229)-58D	10	2	10	9	<10	<10	10	56.1	16.5	0.74	10.3	7.9	0.51	2.69	0.03	1.66	4.70	0.09	0.03	0.07	0.02	7.10	101	<0.05	<0.2	0.00	0.00
59		DDH-26064(264&146-269&156)-59	<10	2	3	119	<10	<10	<10	45.1	18.8	0.53	11.9	8.5	7.87	9.61	0.14	2.37	0.38	0.04	0.03	0.01	0.03	3.43	100	0.07	0.2	0.45	0.48
60		DDH-26056(110-125)-60	<10	<2	3	63	<10	<10	<10	44.8	16.9	0.56	15.1	13.0	7.66	11.65	0.16	2.35	0.26	<0.01	0.02	0.01	0.03	-0.51	99	<0.05	<0.2	0.73	1.00
62		DDH-00-361C(737-749)-62	10	<2	16	26	<10	<10	<10	57.1	16.4	0.86	10.1	8.6	2.81	4.62	0.11	1.73	2.39	0.12	0.03	0.06	0.02	4.41	101	0.09	0.3	0.01	0.01
63	58 Dup	DDH-00-364C(210-229)-63	10	2	10	9	<10	<10	10	56.1	16.5	0.74	10.3	7.9	0.51	2.69	0.03	1.66	4.70	0.09	0.03	0.07	0.02	7.10	101	<0.05	<0.2	0.00	0.00
64		DDH-00-337C(510-520)-64	10	4	12	11	<10	<10	10	54.8	16.4	0.75	13.3	11.4	0.66	2.64	0.03	2.19	3.90	0.13	0.03	0.07	0.02	5.66	101	<0.05	<0.2	0.00	0.00
74		26029(interval 815-825)-74	<10	<2	1	167	<10	<10	10	48.9	24.4	0.54	5.9	4.9	11.40	3.56	0.05	3.20	0.36	0.10	0.01	0.01	0.04	-0.10	98			0.28	0.57
78		26056(135-153)-78	<10	2	3	81	<10	<10	<10	46.1	17.8	0.61	13.3	10.8	8.08	10.90	0.15	2.36	0.35	0.01	0.03	0.01	0.03	1.07	101	0.05	0.2	0.78	0.88

			KINETIC TEST RESULTS (RATES RELEASE)																																
HCT ID	Comment	HCT Full ID (Drill Hole, Footage, HCT)	Initial Date	Final Date	Duration of the Test	Acidity (pH=4.5)	Acidity (pH=8.3)	Alkalinity	Inorganic Carbon	Hardness	F	Cl	SO4	Al	Sb	As	Ba	Be	Bi	B	Cd	Ca	Cr	Co	Cu	Fe	Pb	Li	Mg	Mn	Hg	Mo	Ni	PO4
					weeks	(mg/kg/wk)	(mg/kg/wk)	(mg/kg/wk)

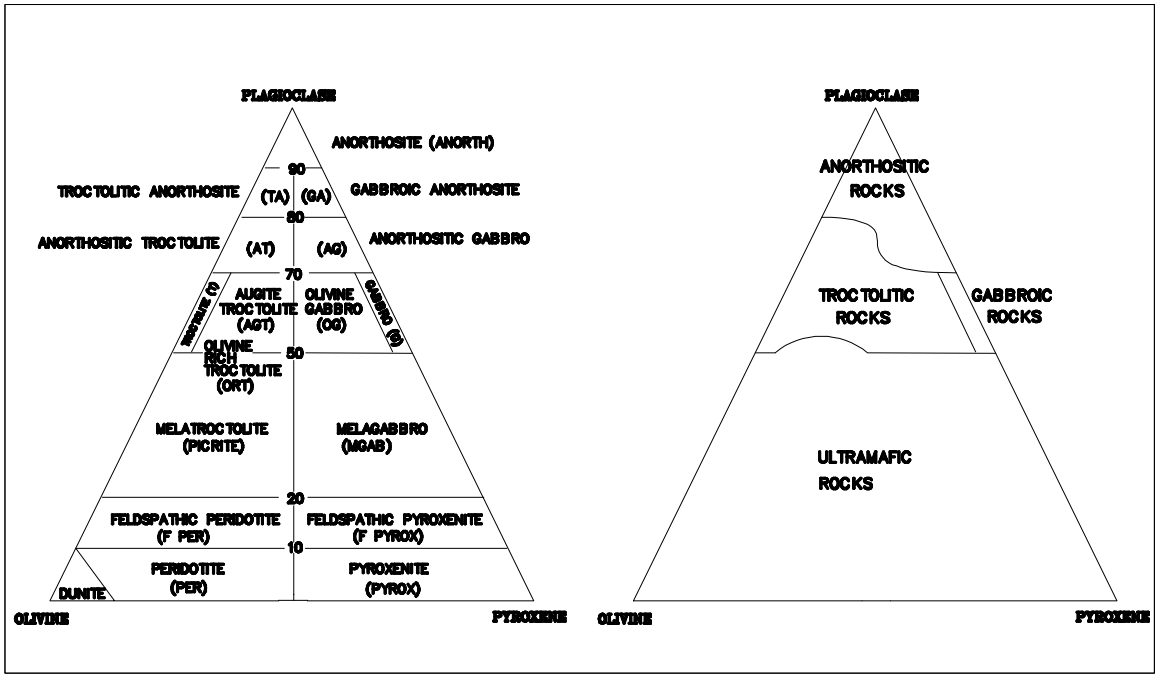
			KINETIC TEST RESULTS (RATES RELEASE)															PETROGRAPHIC ANALISIS																			
																		Modal Percentages of Minerals																			
HCT ID	Comment	HCT Full ID (Drill Hole, Footage, HCT)	K	Se	SiO2	Ag	Na	Sr	Te	Tl	Th	Sn	Ti	U	V	Zn	Zr	Plagioclase	Olivine	K-spar	Quartz	Cordierite	OPX	CPX	Amphibole	Biotite	White Mica	Chlorite	Clay minerals	Serpentine	Uralite	Epidote	Apatite	Zircon	Rutile	Graphite	Carbonates	Vesuvianite/Idocrase
			(mg/kg/wk)	(mg/kg/wk)	(mg/kg/wk)	(mg/kg/wk)	(mg/kg/wk)	(mg/kg/wk)	(mg/kg/wk)	(mg/kg/wk)	(mg/kg/wk)	(mg/kg/wk)	(mg/kg/wk)	(mg/kg/wk)	(mg/kg/wk)	(mg/kg/wk)	(mg/kg/wk)																						
1		DDH-99-320C(830-850)-1	0.39	1.07E-04	0.90	2.68E-05	0.61	7.98E-03	1.07E-04	1.07E-05	5.36E-05	5.36E-05	1.68E-04	2.68E-05	9.05E-04	6.41E-04	1.07E-03	78.0	15.0				1.0	4.0		0.5		trace					trace					
2		DDH-00-361C(310-320)-2	0.26	1.04E-04	0.60	2.61E-05	0.53	4.79E-03	1.04E-04	1.04E-05	5.22E-05	1.78E-03	1.57E-04	2.61E-05	3.13E-04	4.72E-04	1.04E-03	75.0	20.0					3.0		1.0							rare					
3		DDH-00-361C(345-350)-3	0.36	9.84E-05	0.63	2.46E-05	0.59	3.80E-03	9.84E-05	9.84E-06	4.92E-05	9.61E-05	1.31E-04	2.46E-05	1.55E-04	6.87E-04	9.84E-04	82.0	16.0				trace	2.0		trace	trace	trace					trace					
4		DDH-00-343C(240-250)-4	1.72	1.64E-04	1.95	2.37E-05	9.41	1.55E-02	9.21E-05	9.21E-06	4.61E-05	4.61E-05	9.35E-04	2.30E-04	7.10E-03	4.61E-04	9.21E-04	65.0	5.0				8.0	6.0		trace	trace	10.0	trace				1.5				trace	
5		DDH-26030(1047-1052)-5	0.43	1.28E-04	4.09	2.55E-05	0.98	8.44E-03	1.02E-04	1.02E-05	5.09E-05	5.75E-05	2.24E-04	4.67E-05	9.85E-04	1.20E-03	1.02E-03			10.0				48.0			10.0							1.0	25.0	5.0	
6		DDH-26061(1218-1233)-6	0.64	1.66E-04	0.60	2.76E-05	0.62	2.21E-02	1.10E-04	1.10E-05	5.52E-05	2.78E-03	1.78E-04	1.64E-03	1.79E-03	5.52E-04	1.10E-03			55.0		5.0				5.0	30.0							2.0				
7		DDH-00-340C(990-995)-7	0.42	9.07E-05	0.80	2.27E-05	2.10	7.62E-03	9.07E-05	9.07E-06	4.53E-05	1.71E-03	5.89E-04	1.25E-04	1.74E-03	6.95E-04	9.07E-04			25.0	30.0			30.0		0.5		10.0		trace				1.0				
8		DDH-00-340C(965-974.5)-8	0.34	9.45E-05	0.57	2.36E-05	1.16	8.48E-03	9.45E-05	9.45E-06	4.73E-05	4.73E-05	1.31E-04	2.03E-04	3.11E-04	4.73E-04	9.45E-04			10.0		65.0				10.0	trace		trace					5.0				
9	1 Dup	DDH-99-320C(830-850)-9D	0.39	9.93E-05	0.78	2.48E-05	0.58	7.82E-03	9.93E-05	9.93E-06	4.96E-05	4.96E-05	2.30E-04	2.48E-05	7.25E-04	4.96E-04	9.93E-04	78.0	15.0				1.0	4.0		0.5		trace					trace					
10		DDH-00-340C(765-780)-10D	0.85	1.05E-04	0.46	2.62E-05	0.19	3.37E-03	1.05E-04	1.05E-05	5.25E-05	5.25E-05	1.73E-04	2.62E-05	2.79E-04	6.96E-04	1.05E-03	60.0	25.0					5.0		2.0	trace	trace	trace					trace					
11		DDH-26043&26027(1501&740-1506&745)-11	0.70	1.05E-04	0.70	2.64E-05	0.38	1.30E-02	1.05E-04	1.05E-05	5.27E-05	5.27E-05	1.05E-04	2.64E-05	1.05E-04	1.69E-02	1.05E-03																					
12	Blank	Method Blank - 12	0.01	1.01E-04	0.03	2.51E-05	0.01	5.04E-05	1.01E-04	1.01E-05	5.04E-05	5.04E-05	1.01E-04	2.52E-05	1.01E-04	1.23E-03	1.01E-03																					
13		DDH-00-340C(595-615)-13	0.58	1.06E-04	0.46	2.66E-05	0.16	2.31E-03	1.06E-04	1.06E-05	5.32E-05	5.32E-05	1.55E-04	2.66E-05	1.18E-04	5.32E-04	1.06E-03	60.0	10.0				1.0	25.0		1.0	trace	trace						1.0				
14		DDH-00-334C(580-600)-14	0.24	1.02E-04	0.43	2.54E-05	0.17	2.43E-03	1.02E-04	1.02E-05	5.08E-05	5.08E-05	1.62E-04	2.54E-05	1.52E-04	5.08E-04	1.02E-03	58.0	32.0				trace	6.0		2.0								trace				
15		DDH-00-334C(640-660)-15	0.15	9.08E-05	0.78	2.27E-05	3.25	3.77E-03	9.08E-05	9.08E-06	4.54E-05	6.40E-05	4.14E-04	2.27E-05	3.64E-04	1.01E-03	9.08E-04	55.0	30.0					6.0		2.0	trace	1.0		2.0				trace				
16		DDH-00-347C(795-815)-16	0.18	9.56E-05	0.41	2.39E-05	0.31	2.85E-03	9.56E-05	9.56E-06	4.78E-05	4.78E-05	2.14E-04	2.39E-05	1.75E-04	1.75E-03	9.56E-04	55.0	35.0					6.0		2.0								trace				
17		DDH-00-350C(580-600)-17	0.16	1.26E-04	1.27	2.47E-05	4.53	6.89E-04	9.87E-05	9.87E-06	4.93E-05	1.16E-04	1.91E-03	3.25E-05	8.37E-03	4.93E-04	9.87E-04	55.0	30.0				trace	8.0		2.0	trace	trace	trace	trace	trace			trace				
18		DDH-00-327C(225-245)-18	0.40	1.05E-04	0.51	2.63E-05	1.07	3.38E-03	1.05E-04	1.05E-05	5.27E-05	5.27E-05	1.37E-04	1.95E-04	3.79E-04	8.43E-04	1.05E-03	50.0	40.0				3.0	4.0		1.0	trace	trace	trace		trace			0.5				trace	
19		DDH-00-371C(435-440)-19	1.03	1.06E-04	1.28	2.64E-05	0.55	5.33E-03	1.06E-04	1.06E-05	5.29E-05	5.29E-05	2.23E-04	9.70E-05	1.66E-03	4.75E-04	1.06E-03	60.0			trace		32.0	1.0		trace	trace	trace					2.0	trace	trace			
20	10 Dup	DDH-00-340C(765-780)-20	0.91	1.05E-04	0.49	2.62E-05	0.19	3.70E-03	1.05E-04	1.05E-05	5.24E-05	5.24E-05	1.44E-04	2.62E-05	2.37E-04	6.51E-04	1.05E-03	60.0	25.0					5.0		2.0	trace	trace	trace					trace				
21		DDH-00357C(335-340)-21	0.59	9.48E-05	0.55	2.37E-05	0.27	5.30E-03	9.48E-05	9.48E-06	4.74E-05	7.13E-05	1.56E-04	2.37E-05	5.08E-04	5.84E-04	9.48E-04		3.0	25.0				12.0			3.0	25.0	3.0	5.0		12.0	2.0	trace			2.0	
22		DDH-00326C(680-685)-22	0.44	9.10E-05	0.50	2.28E-05	0.16	2.93E-03	9.10E-05	9.10E-06	4.55E-05	5.68E-05	1.41E-04	2.28E-05	3.95E-04	4.55E-04	9.10E-04	45.0	30.0					15.0		1.0	trace	trace		trace				1.0				
23		DDH-00-357C(535-540)-23	0.72	1.20E-04	0.54	2.39E-05	0.23	3.73E-03	9.58E-05	9.58E-06	4.79E-05	1.84E-03	1.07E-04	2.39E-05	3.46E-04	4.79E-04	9.58E-04	45.0	20.0				3.0	25.0		2.0	trace	1.0						1.0				
24		DDH-99-318C(725-735)-24	0.89	8.90E-05	0.71	2.23E-05	0.57	3.98E-03	8.90E-05	8.90E-06	4.45E-05	4.45E-05	1.68E-04	9.38E-05	1.45E-03	7.27E-04	8.90E-04	55.0	20.0	8.0			15.0	trace		1.0	trace		trace					trace	rare			
25		DDH-99-317C(460-470)-25	0.59	9.40E-05	0.70	2.35E-05	0.57	4.97E-03	9.40E-05	9.40E-06	4.70E-05	1.84E-03	9.40E-05	3.91E-05	6.94E-04	4.70E-04	9.40E-04	45.0	15.0				5.0	30.0		3.0	trace	trace						trace				
26		DDH-00-366C(185-205)-26	0.30	9.15E-05	0.51	2.29E-05	0.14	1.41E-03	9.15E-05	9.15E-06	4.58E-05	4.58E-05	9.15E-05	2.29E-05	9.15E-05	5.76E-04	9.15E-04	88.0	3.0				1.0	5.0		trace								trace	trace			
27		DDH-00-366C(230-240)-27	0.25	9.57E-05	0.51	2.39E-05	0.20	1.34E-03	9.57E-05	9.57E-06	4.78E-05	4.78E-05	9.57E-05	2.39E-05	9.57E-05	4.78E-04	9.57E-04	88.0	8.0				1.0	2.0		trace								trace				
28		DDH-99-320C(165-175)-28	0.34	1.07E-04	0.51	2.67E-05	0.32	3.10E-03	1.07E-04	1.07E-05	5.34E-05	2.76E-03	1.07E-04	2.80E-05	1.20E-04	5.34E-04	1.07E-03	85.0	trace					12.0		trace			trace									
29		DDH-99-318C(250-370)-29	0.26	1.06E-04	0.45	2.64E-05	0.15	1.72E-03	1.06E-04	1.06E-05	5.29E-05	5.29E-05	1.06E-04	2.64E-05	1.31E-04	5.29E-04	1.06E-03	60.0	34.0				trace	4.0		trace		trace		trace				trace				
30		DDH-00-373C(95-115)-30	0.22	9.95E-05	0.49	2.49E-05	0.25	2.08E-03	9.95E-05	9.95E-06	4.98E-05	4.98E-05	9.95E-05	2.49E-05	9.95E-05	4.98E-04	9.95E-04	65.0	30.0				1.0	2.0		1.0	trace	trace		trace				trace				
31		DDH-00-373C(75-95)-31	0.26	1.00E-04	0.50	2.50E-05	0.36	2.34E-03	1.00E-04	1.00E-05	5.00E-05	5.00E-05	1.29E-04	2.50E-05	2.00E-04	6.07E-04	1.00E-03	60.0	35.0				trace	3.0		0.5	trace	trace		trace				trace				
32		DDH-00-357C(110-130)-32	0.38	9.44E-05	0.61	3.08E-05	0.39	4.65E-03	9.44E-05	9.44E-06	4.72E-05	6.52E-05	1.41E-04	2.36E-05	3.31E-04	1.04E-03	9.44E-04	54.0	40.0				trace	4.0		0.5								rare				
33		DDH-99-320C(315-330)-33	0.27	1.05E-04	0.61	2.64E-05	0.51	6.09E-03	1.05E-04	1.05E-05	5.27E-05	5.27E-05	2.20E-04	2.64E-05	3.53E-04	8.88E-04	1.05E-03	65.0	26.0				trace	3.0		1.0	trace							trace				
34		DDH-00-369C(335-345)-34	0.45	1.70E-04	0.78	2.64E-05	0.38	3.25E-03	1.05E-04	1.05E-05	5.27E-05	5.27E-05	1.56E-04	2.64E-05	1.70E-04	5.27E-04	1.05E-03	60.0	35.0				trace	4.0		trace								trace				
35		DDH-00-368C(460-465)-35	0.09	9.51E-05	1.72	2.38E-05	8.59	1.09E-03	9.51E-05	9.51E-06	4.76E-05	4.76E-05	1.44E-03	2.38E-05	1.73E-03	5.81E-04	9.51E-04	45.0	trace					3.0		1.0		20.0		30.0				trace				
36		DDH-26055(940-945)-36	0.32	9.10E-05	0.45	2.28E-05	0.21	2.72E-03	9.10E-05	9.10E-06	4.55E-05	4.55E-05	1.21E-04	2.28E-05	1.06E-04	4.55E-04	9.10E-04	40.0	50.0				1.5	5.0		0.5	trace	trace		trace								
37		DDH-00-368C(125-145)-37D	0.20	1.02E-04	0.59	2.54E-05	0.52	2.64E-03	1.02E-04	1.02E-05	5.08E-05	5.08E-05	1.08E-04	2.54E-05	1.78E-04	6.87E-04	1.02E-03	72.0	20.0				1.0	4.0		1.0			trace									
38		DDH-00-369C(305-325)-38D	1.55	1.03E-04	0.55	2.57E-05	0.73	2.51E-03	1.03E-04	1.03E-05	5.14E-05	2.05E-03	2.30E-04	2.57E-05	1.23E-04	6.10E-04	1.03E-03	68.0	20.0				trace	2.0		trace	trace	8.0	trace	2.0				trace				
39		DDH-26098&00-337C(145&105-148.5&110)-39	0.25	1.10E-04	0.96	2.76E-05	0.35	7.16E-03	1.10E-04	1.10E-05	5.51E-05	5.51E-05	1.20E-04	2.76E-05	4.83E-04	2.15E-03	1.10E-03																					
40		DDH-00-334C(30-50)-40	0.56	1.03E-04	0.64	2.59E-05	0.73	3.83E-03	1.03E-04	1.03E-05	5.17E-05	5.17E-05	1.03E-04	2.59E-05	1.16E-04	5.17E-04	1.03E-03	82.0	15.0				trace	2.5		trace	trace	trace		trace								trace
41	37 Dup	DDH-00-368C(125-145)-41	0.17	1.16E-04	0.59	2.91E-05	0.27	2.27E-03	1.16E-04	1.16E-05	5.82E-05	5.82E-05	1.16E-04	2.91E-05	1.45E-04	8.52E-04	1.16E-03	72.0	20.0				1.0	4.0		1.0			trace									
42		DDH-00-368C(20-40)-42	0.20	9.98E-05	0.60	2.50E-05	0.44	3.54E-03	9.98E-05	9.98E-06	4.99E-05	1.33E-03	2.48E-03	3.83E-05	2.57E-04	7.15E-04	9.98E-04	60.0	35.0				1.0	1.0		1.0		trace									
43		DDH-00-366C(35-55)-43	0.20	9.98E-05	0.35	2.49E-05	0.15	1.45E-03	9.98E-05	9.98E-06	4.99E-05	1.82E-03	9.98E-05	2.49E-05	9.98E-05	4.99E-04	9.98E-04	65.0	35.0				trace	trace		trace								trace				
44		DDH-00-334C(110-130)-44	0.38	1.12E-04	0.62	2.81E-05	0.67	5.08E-03	1.12E-04	1.12E-05	5.61E-05	5.61E-05	1.83E-04	2.81E-05	1.12E-04	7.09E-04	1.12E-03	55.0	44.0					0.5		trace	trace	trace		trace								
45		DDH-00-347C(155-175)-45	0.51	1.01E-04	0.53	2.52E-05	0.31	2.65E-03	1.01E-04	1.01E-05	5.04E-05	5.04E-05	1.14E-04	2.52E-05	1.77E-04	5.04E-04	1.01E-03	70.0	25.0					2.5		0.5	trace	trace		trace								
46		DDH-00-347C(280-300)-46	0.31	1.12E-04	0.97	2.79E-05	0.24	2.96E-03	1.12E-04	1.12E-05	5.58E-05	5.58E-05	2.35E-04	2.79E-05	1.67E-04	5.58E-04	1.12E-03	66.0	30.0				1.0	2.0		0.5								trace				
47		DDH-00-326C(60-70)-47	0.63	1.05E-04	0.62	2.62E-05	0.77	4.96E-03	1.05E-04	1.05E-05	5.23E-05	5.23E-05	1.27E-04	2.62E-05	1.18E-04	1.28E-03	1.05E-03	60.0	38.0				trace	2.0		trace	trace	trace		trace								trace
48	38 Dup	DDH-00-369C(305-325)-48	1.54	1.02E-04	0.56	2.55E-05	0.67	3.10E-03	1.02E-04	1.02E-05	5.11E-05	5.11E-05	1.27E-04	2.55E-05	1.02E-04	5.11E-04	1.02E-03	68.0	20.0				trace	2.0		trace	trace	8.0	trace	2.0				trace				
49		DDH-00-367C(50-65)-49	0.36	8.70E-05	0.65	2.18E-05	0.64	3.99E-03	8.70E-05	8.70E-06	4.35E-05	4.35E-05	2.72E-04	2.18E-05	1.41E-04	6.63E-04	8.70E-04	65.0	trace				trace	3.0		0.5	5.0	10.0	trace	15.0								
50		DDH-00-367C(260-280)-50	0.42	9.80E-05	1.01	2.45E-05	0.53	6.27E-03	9.80E-05	9.80E-06	4.90E-05	2.54E-03	9.80E-05	2.45E-05	1.96E-04	4.90E-04	9.80E-04	45.0	45.0				trace	8.0		0.5	trace	trace		trace				trace				
51		DDH-00-367C(290-310)-51	0.55	9.64E-05	0.82	2.41E-05	2.71	3.19E-03	9.64E-05	9.64E-06	4.82E-05	2.16E-03	3.43E-04	2.41E-05	4.59E-04	5.19E-04	9.64E-04	52.0					2.0	3.0		1.0	trace	40.0				1.0						
52		DDH-00-370C(20-30)-52	2.69	1.10E-04	1.02	2.74E-05	4.74	2.46E-03	1.10E-04	1.10E-05	5.48E-05	5.48E-05	2.97E-04	2.74E-05	8.33E-04	9.48E-04	1.10E-03	52.0	26.0					8.0		0.5	trace	6.0	trace	6.0								
53		DDH-00-369C(20-30)-53	0.80	1.15E-04	0.70	2.42E-05	0.51	4.62E-03	9.68E-05	9.68E-06	4.84E-05	4.84E-05	2.39E-04	2.42E-05	1.26E-04	2.09E-03	9.68E-04	65.0	20.0				1.0	3.0		1.0	trace	8.0	trace	1.0					trace			
54		DDH-00-367C(170-175)-54	1.41	1.77E-04	1.02	2.49E-05	2.75	6.80E-03	9.94E-05	9.94E-06	4.97E-05	4.97E-05	3.68E-04	1.33E-04	7.67E-04	4.97E-04	9.94E-04	52.0	3.0	3.0				12.0	5.0	4.0	3.0	10.0										
55		DDH-00-367C(395-400)-55	0.85	8.11E-04	2.40	1.25E-04	23.97	1.98E-03	9.76E-05	9.76E-06	4.88E-05	9.08E-05	2.63E-03	3.57E-05	3.53E-02	4.88E-04	9.76E-04	55.0	2.0				1.0	2.0		2.0	2.0	28.0	trace	2.0				trace				
56		DDH-26064(44-54)-56	0.35	1.09E-04	0.61	3.04E-05	0.52	4.34E-03	1.09E-04	1.09E-05	5.47E-05	2.12E-03	1.32E-04	2.73E-05	1.09E-04	5.47E-04	1.09E-03	74.0	24.0					2.0			trace	trace		trace				rare				
57	51 Dup	DDH-00-367C(290-310)-57D	0.58	1.20E-04	0.88	3.01E-05	3.07	4.15E-03	1.20E-04	1.20E-05	6.02E-05	6.02E-05	2.42E-04	3.01E-05	4.65E-04	6.02E-04	1.20E-03	52.0					2.0	3.0		1.0	trace	40.0				1.0						
58		DDH-00-364C(210-229)-58D	2.07	5.87E-04	1.87	6.19E-05	0.50	2.96E-02	1.12E-04	1.12E-05	5.61E-05	5.61E-05	1.40E-04	1.99E-03	1.12E-04	3.90E-01	1.12E-03			65.0		25.0		trace		0.5	1.5							trace		2.0		
59		DDH-26064(264&146-269&156)-59	0.32	1.08E-04	1.70	2.69E-05	10.73	1.61E-03	1.08E-04	1.08E-05	5.38E-05	5.38E-05	2.20E-04	6.88E-05	1.51E-03	1.53E-03	1.08E-03																				
60		DDH-26056(110-125)-60	0.27	1.03E-04	0.48	2.58E-05	0.31	2.56E-03	1.03E-04	1.03E-05	5.16E-05	5.16E-05	1.03E-04	2.58E-05	1.03E-04	5.58E-04	1.03E-03	55.0	42.0					3.0		trace	trace	trace		trace				trace				
62		DDH-00-361C(737-749)-62	0.35	8.43E-05	0.73	5.20E-05	1.97	1.64E-02	8.43E-05	8.43E-06	4.22E-05	4.22E-05	2.12E-04	7.39E-04	1.07E-03	4.22E-04	8.43E-04			10.0		70.0				5.0		2.0								10.0		
63	58 Dup	DDH-00-364C(210-229)-63	1.72	6.22E-04	2.12	3.79E-05	0.34	1.89E-02	1.26E-04	1.26E-05	6.32E-05	2.98E-03	1.26E-04	7.84E-04	1.26E-04	1.25E-01	1.26E-03			65.0		25.0		trace		0.5	1.5								trace	2.0		
64		DDH-00-337C(510-520)-64	1.07	5.11E-04	1.37	3.86E-05	1.42	1.53E-02	1.12E-04	1.12E-05	5.59E-05	5.59E-05	1.12E-04	9.22E-04	1.12E-04	3.75E-02	1.12E-03					85.0	trace			2.0	trace									3.0	
74		26029(interval 815-825)-74	0.36	1.07E-04	0.49	2.68E-05	0.27	2.71E-03	1.07E-04	1.07E-05	5.36E-05	3.95E-03	1.07E-04	2.68E-05	1.07E-04	5.36E-04	1.07E-03																				
78		26056(135-153)-78	0.74	9.93E-05	1.28	2.48E-05	2.70	6.06E-03	9.93E-05	9.93E-06	4.96E-05	7.44E-05	9.93E-05	2.48E-05	2.98E-04	4.96E-04	9.93E-04	55.0	45.0							trace		trace	trace									

			PETROGRAPHIC ANALISIS																								
			Modal Percentages of Minerals			Oxides			Breakdown of Sulfide Mineralogy																		
HCT ID	Comment	HCT Full ID (Drill Hole, Footage, HCT)	Oxides	Sulfides	Percent Check	Magnetite	Ilmenite	Percent Check	Pyrite	Pyrrhotite	Chalcopyrite	Bornite	Digenite	Chalcocite	Covellite	Cubanite	Pentlandite	Violarite	PGMs	Silver	Mackinawite/Valleriite	Sphalerite	Galena	Millerite	Enargite	Talnakhite
1		DDH-99-320C(830-850)-1	1.5	trace	101	5.0	95.0	100		35.0	50.0						15.0	trace	rare			trace	rare			
2		DDH-00-361C(310-320)-2	1.0	trace	102	5.0	95.0	100		40.0	40.0	trace				10.0	10.0									
3		DDH-00-361C(345-350)-3	trace	trace	103	5.0	95.0	100		45.0	45.0	trace					5.0	rare			5.0	trace				
4		DDH-00-343C(240-250)-4	2.0	2.5	104		100.0	100	trace	80.0	18.0						2.0					trace				
5		DDH-26030(1047-1052)-5		1.0	105					trace	trace											95.0	5.0			
6		DDH-26061(1218-1233)-6	1.0	2.0	106	100.0		100		90.0	5.0						trace					5.0				
7		DDH-00-340C(990-995)-7	2.0	1.5	107		100.0	100		95.0	5.0											trace	trace			
8		DDH-00-340C(965-974.5)-8	trace	10.0	108	trace	100.0	100		100.0	trace															
9	1 Dup	DDH-99-320C(830-850)-9D	1.5	trace	109	5.0	95.0	100		35.0	50.0						15.0	trace	rare			trace	rare			
10		DDH-00-340C(765-780)-10D	5.0	3.0	110	10.0	90.0	100		80.0	10.0			trace		4.0	6.0								trace	
11		DDH-26043&26027(1501&740-1506&745)-11																								
12	Blank	Method Blank - 12																								
13		DDH-00-340C(595-615)-13	2.0	trace	113	85.0	15.0	100		45.0	35.0					10.0	10.0	trace			trace					
14		DDH-00-334C(580-600)-14	2.0	trace	114	5.0	95.0	100		30.0	25.0	trace				35.0	10.0					trace				
15		DDH-00-334C(640-660)-15	4.0	trace	115	50.0	50.0	100		55.0	20.0					5.0	20.0				trace	trace				
16		DDH-00-347C(795-815)-16	2.0	trace	116	trace	100.0	100		30.0	50.0					20.0	trace									
17		DDH-00-350C(580-600)-17	5.0	trace	117	5.0	95.0	100		60.0	30.0	trace					10.0					trace				trace
18		DDH-00-327C(225-245)-18	1.0	0.5	118	trace	100.0	100		65.0	35.0						trace	trace			trace					
19		DDH-00-371C(435-440)-19	5.0	trace	119	30.0	70.0	100		95.0	5.0					trace						trace	trace			
20	10 Dup	DDH-00-340C(765-780)-20	5.0	3.0	120	10.0	90.0	100		80.0	10.0			trace		4.0	6.0								trace	
21		DDH-00357C(335-340)-21	8.0	trace	121	10.0	90.0	100			60.0						40.0					trace	trace			
22		DDH-00326C(680-685)-22	3.0	5.0	122	50.0	50.0	100		20.0	20.0	trace				35.0	25.0		rare		trace	trace				
23		DDH-00-357C(535-540)-23	3.0	trace	123		100.0	100		50.0	50.0						trace				trace					
24		DDH-99-318C(725-735)-24	1.0	trace	124	10.0	90.0	100		80.0	10.0					5.0	5.0				trace	trace				
25		DDH-99-317C(460-470)-25	2.0	trace	125	trace	100.0	100		10.0	85.0	5.0					trace		rare			trace				
26		DDH-00-366C(185-205)-26	3.0	trace	126	1.0	99.0	100		45.0	40.0	trace			trace	10.0	5.0					rare				
27		DDH-00-366C(230-240)-27	1.0	trace	127	40.0	60.0	100		5.0	90.0	trace					5.0				trace					
28		DDH-99-320C(165-175)-28	2.0	trace	128	65.0	35.0	100		40.0	30.0					15.0	15.0									
29		DDH-99-318C(250-370)-29	2.0	trace	129	30.0	70.0	100		10.0	50.0	trace				25.0	15.0	trace				trace				
30		DDH-00-373C(95-115)-30	1.0	trace	130	30.0	70.0	100		30.0	60.0	trace			trace	5.0	5.0			rare						
31		DDH-00-373C(75-95)-31	1.5	trace	131	40.0	60.0	100		30.0	40.0	trace			trace	20.0	10.0	trace				trace				
32		DDH-00-357C(110-130)-32	1.5	trace	132	trace	100.0	100		40.0	45.0					10.0	5.0					trace				trace
33		DDH-99-320C(315-330)-33	1.0	4.0	133	10.0	90.0	100		20.0	35.0	trace			trace	30.0	15.0				trace	trace				trace
34		DDH-00-369C(335-345)-34	1.0	trace	134	trace	100.0	100		60.0	20.0						15.0				5.0					
35		DDH-00-368C(460-465)-35	1.0	trace	135	50.0	50.0	100			100.0					trace	trace									
36		DDH-26055(940-945)-36	3.0	trace	136	5.0	95.0	100		30.0	40.0	trace	trace		trace	15.0	15.0									
37		DDH-00-368C(125-145)-37D	2.0	trace	137	5.0	95.0	100		35.0	55.0						5.0				5.0	trace				
38		DDH-00-369C(305-325)-38D	trace	trace	138	1.0	99.0	100		85.0	10.0										5.0					
39		DDH-26098&00-337C(145&105-148.5&110)-39																								
40		DDH-00-334C(30-50)-40	0.5	trace	140	35.0	65.0	100		45.0	45.0	trace					10.0									
41	37 Dup	DDH-00-368C(125-145)-41	2.0	trace	141	5.0	95.0	100		35.0	55.0						5.0				5.0	trace				
42		DDH-00-368C(20-40)-42	2.0	trace	142	trace	100.0	100		55.0	40.0	2.0					trace				3.0	rare				
43		DDH-00-366C(35-55)-43	trace	rare	143	10.0	90.0	100		trace	75.0	15.0			10.0	trace	trace									
44		DDH-00-334C(110-130)-44	trace	trace	144	30.0	70.0	100		45.0	40.0	trace				5.0	10.0									
45		DDH-00-347C(155-175)-45	2.0	trace	145	50.0	50.0	100		45.0	45.0	trace				5.0	5.0									
46		DDH-00-347C(280-300)-46	0.5	trace	146	trace	100.0	100		35.0	60.0					trace	trace				5.0					
47		DDH-00-326C(60-70)-47	trace	trace	147	50.0	50.0	100		10.0	75.0	trace				10.0	5.0					trace				
48	38 Dup	DDH-00-369C(305-325)-48	trace	trace	148	1.0	99.0	100		85.0	10.0										5.0					
49		DDH-00-367C(50-65)-49	1.5	rare	149	80.0	20.0	100			95.0	5.0					trace									
50		DDH-00-367C(260-280)-50	1.5	trace	150	30.0	70.0	100		15.0	48.0	trace	trace		trace	30.0	5.0				trace	trace				2.0
51		DDH-00-367C(290-310)-51	1.0	trace	151	50.0	50.0	100		5.0	80.0						15.0		rare							
52		DDH-00-370C(20-30)-52	1.5	trace	152	trace	100.0	100		70.0	20.0	trace				2.0	3.0				5.0	trace				
53		DDH-00-369C(20-30)-53	trace	1.0	153	trace	100.0	100		35.0	25.0										40.0					
54		DDH-00-367C(170-175)-54	trace	8.0	154		100.0	100		70.0	25.0						5.0					trace				
55		DDH-00-367C(395-400)-55	3.0	3.0	155	trace	100.0	100		trace	3.0			trace			95.0	2.0				trace				
56		DDH-26064(44-54)-56	trace	rare	156	50.0	50.0	100			40.0	30.0			30.0		trace									
57	51 Dup	DDH-00-367C(290-310)-57D	1.0	trace	157	50.0	50.0	100		5.0	80.0						15.0		rare							
58		DDH-00-364C(210-229)-58D		6.0	158					98.0	trace											1.5	0.5			
59		DDH-26064(264&146-269&156)-59																								
60		DDH-26056(110-125)-60	trace	trace	160	45.0	55.0	100		trace	70.0	5.0	5.0		5.0	10.0	4.0					1.0				
62		DDH-00-361C(737-749)-62	trace	3.0	162	100.0		100		100.0	trace											trace				
63	58 Dup	DDH-00-364C(210-229)-63		6.0	163					98.0	trace											1.5	0.5			
64		DDH-00-337C(510-520)-64		10.0	164					98.0	trace											2.0				
74		26029(interval 815-825)-74																								
78		26056(135-153)-78	trace	trace	178	5.0	95.0	100		5.0	75.0	10.0	3.0		2.0		5.0		trace							

Appendix C
Mineralogical Report for Samples in Dissolution Experiments

SUMMARY OF PETROGRAPHIC OBSERVATIONS
FOR WASTE ROCK CHARACTERIZATION

As part of Polymet's waste rock characterization, total of 91 intervals ranging from 5 to 20 feet in length were selected to represent a variety of rock types, sulfur/metals ranges, and stratigraphic units within the NorthMet Deposit. Petrographic observations were recorded from a representative polished thin-section from each of the intervals, including; mineral identification, estimated modal percents, crystal shape/grain size/textural characteristics, and the subsequent assigning of a rock name. It is important to note, however, that the rock name assigned to the polished thin-section (based on estimated modal percents of plagioclase, olivine and pyroxene) may not be the same as that assigned to the entire interval, due to a difference in just a few modal percent of any one of the three minerals and the fact that this was derived from a one-square inch representative of rock (see Phinney's diagram below). Approximately 90% of the petrographic-derived rock names do nevertheless match that assigned to the core interval. All data is included in an Excel spreadsheet.



Rock classification scheme (after Phinney, 1972)

In addition to the raw data presented in the accompanying Excel spreadsheet, a number of broad generalizations are included below:

- In general, approximately 90% of the rocks are classified as adcumulates, with plagioclase and olivine occurring as the two main cumulate phases. Pyroxenes often occur interstitially as ophitic or reactionary textures. Minor accessory minerals include; biotite, apatite and trace zircon, as interstitial or inclusions

within the major silicate phases. Secondary silicates associated with alteration occur rarely and include; amphibole, carbonates (calcite), epidote, and potassium feldspar.

- Overall, the vast majority of rock (>90%) is unaltered. Where alteration is present, it ranges from weak to moderate, and is usually limited to microfractures (0.01-0.1mm wide) that cross-cut mineralogy. This pattern suggests that both fracturing and accompanying alteration of the rock occur as a result of the migration of late-stage deuteric fluids during the cooling phase. Alterations commonly observed include; chloritization/serpentinization of olivine, sericitization/saussuritization of plagioclase, and uralitization of pyroxenes. Less than 2% of the rock displays wide spread alteration, ranging from moderate to strongly altered, where most of the primary minerals have been replaced by secondary mineralization.

- Also observed in some of the more wide spread alteration, is anhedral aggregates and microveinlet in-fill, of pyrrhotite and chalcopyrite within microfractures that cross-cut altered silicate mineralogy. Inclusions of secondary sulfide mineralization, mostly occurring as tiny (<0.005mm), euhedral/subhedral grains, of chalcopyrite, pentlandite, and pyrrhotite, were observed in some altered mafic silicates (olivine and pyroxene). Minute amounts of copper-enriched sulfides including, bornite, covellite, digenite, and chalcocite, were also present in some of these zones. The locally remobilized and enriched sulfides associated with alteration, represent only a few percent of the total sulfide mineralization present.

- Approximately 95-98% of the sulfide mineralization (usually larger grain size, >0.1mm) are interstitial between cumulate silicate mineralogy (plagioclase and olivine). This also includes sulfides included in interstitial silicates such as pyroxenes and micas. The remaining 2-5% of sulfides occur as inclusions (generally <0.1mm) in silicates, microveinlets in microfractures, and as secondary remobilized scraps in altered silicates.

- The ratio of pyrrhotite to chalcopyrite varies depending on the stratigraphic unit. Within the basal unit 1, the pyrrhotite to chalcopyrite ratio increases noticeably toward the basal contact with the Virginia Formation and the contact with sedimentary inclusion. However, above unit 3 the amount of pyrrhotite present drops substantially.

- Oxide mineralogy is similar to that of sulfide mineralogy, where nearly 99% is interstitial to cumulate silicate mineralogy. The remaining percent or two is observed as inclusions in silicates, and as secondary magnetite as a result of serpentinization of olivine. The latter, occur as anhedral collections or in-fill in microfractures in or adjacent to altered olivine.

Unit	Rock Type	S Percentile	#	Sulfur Range			Selected Interval				Weighted Average Characteristics					Individual petrographic section footages	Individual petrographic section footages		Petrographic Section Observed	Petrographic Observations: Modal Percentages of Minerals																		Percent Check
				Low	From Table 1	High	DDH	From	To	Length	S	Ni	Co	Cu	Zn					Plagioclase	Olivine	K-spar	Quartz	Cordierite	OPX	CPX	Amphibole	Biotite	White Mica	Chlorite	Clay minerals	Serpentine	Uralite	Epidote	Apatite	Zircon	Rutile	Graphite	Carbonates	Vesuvianite/ Idocrase	Oxides	Sulfides	
				%	%	%		Feet	Feet	Feet	%	%	mg/kg	%	mg/kg																										
1	Anorthositic	P25	1	0.09	0.1	0.125	99-320C	830	850	20	0.12	0.015	35.8	0.035	72	833	841		833	78.0	15.0				1.0	4.0		0.5		tr					tr						1.5	tr	100
1	Anorthositic	P50	2	0.125	0.15	0.22	00-361C	310	320	10	0.16	0.018	36.0	0.013	51	311	316		311	75.0	20.0					3.0		1.0							rare						1.0	tr	100
1	Anorthositic	P75	3	0.22	0.29	0.36	00-361C	345	350	5	0.33	0.026	53.0	0.038	68	347	349		347	82.0	16.0				tr	2.0		tr	tr	tr					tr						tr	tr	100
1	Anorthositic	P95	4		1.09	1.09	00-343C	240	250	10	0.67	0.020	38.0	0.069	69	241	249		241	65.0	5.0				8.0	6.0		tr	tr	10.0	tr				1.5				tr		2.0	2.5	100
1	Sedimentary Hornfels	P10	5	0.08	0.08	0.215	26030	1047	1052	5	0.17	0.006	10.0	0.006	34	1048	1050		1050			10.0				48.0			10.0									1.0	25.0	5.0		1.0	100
1	Sedimentary Hornfels	P25	6	0.215	0.35	0.52	26061	1218	1233	15	0.43	0.011	19.3	0.012	193	1220	1231		1231			55.0		5.0				5.0	30.0									2.0			1.0	2.0	100
1	Sedimentary Hornfels	P50	7	0.52	0.69	1.445	00-340C	990	995	10	0.62	0.012	24.0	0.031	148	991	994		994			25.0	30.0			30.0		0.5		10.0		tr							1.0			2.0	1.5	100
1	Sedimentary Hornfels	P75	8	1.445	2.2	2.26	00-340C	965	974.5	15	1.49	0.015	23.7	0.031	249	966	972		966			10.0		65.0				10.0	tr		tr				tr				5.0			tr	10.0	100
1	Sedimentary Hornfels	P85	9	2.565	2.81	3.095	26043	1501	1506	5	2.76	0.024	36.0	0.037	344	1502	1505		1505	tr		25.0		45.0	20.0			2.0	tr		tr								3.0			1.0	4.0	100
1	Sedimentary Hornfels	P85(a)	9(a)	2.565	2.81	3.095	26027	740	745	5	2.59	0.035	67.0	0.051	48	741	744		741			30.0		40.0	20.0			2.0											1.0			3.0	4.0	100
1	Troctolitic	P1	10	0.01	0.01	0.02	00-368C	565	570	5	0.01	0.013	25.0	0.021	52	566	569		569	30.0						8.0		1.0	25.0	25.0	tr	5.0		5.0	tr						1.0	tr	100
1	Troctolitic	P1	10(a)	0.01	0.01	0.02	26098	247	255	8	0.01	0.015	24.0	0.006	37	248	254		248	52.0	38.0				1.0	5.0		1.0		tr		tr			tr						3.0	tr	100
1	Troctolitic	P1	10(b)	0.01	0.01	0.02	26029	815	825	10	0.01	0.014	30.0	0.007	43	816	824		824	68.0	24.0					6.0		0.5		tr	tr									1.5	tr	100
1	Troctolitic	P1	10(c)	0.01	0.01	0.02	26017	433	443	10	0.01	0.016	32.0	0.011	46	435	442		435	68.0	28.0				1.0	2.5		0.5	tr	tr											tr	tr	100
1	Troctolitic	P2	11	0.02	0.03	0.04	00-340C	595	615	20	0.03	0.025	49.8	0.013	71	596	613		596	60.0	10.0				1.0	25.0		1.0	tr	tr					1.0						2.0	tr	100
1	Troctolitic	P3	12	0.04	0.05	0.05	00-334C	580	600	20	0.05	0.024	52.0	0.024	83	589	598		598	58.0	32.0				tr	6.0		2.0							tr						2.0	tr	100
1	Troctolitic	P25	13	0.065	0.07	0.085	00-334C	640	660	20	0.08	0.038	63.8	0.024	102	642	657		642	55.0	30.0					6.0		2.0	tr	1.0		2.0			tr						4.0	tr	100
1	Troctolitic	P50	14	0.085	0.1	0.14	00-347C	795	815	20	0.09	0.034	70.8	0.035	94	798	812		798	55.0	35.0					6.0		2.0							tr						2.0	tr	100
1	Troctolitic	P80	15	0.195	0.21	0.235	00-350C	580	600	20	0.22	0.027	55.5	0.041	115	582	591		582	55.0	30.0				tr	8.0		2.0	tr	tr	tr	tr			tr						5.0	tr	100
1	Troctolitic	P90	16	0.3	0.34	0.48	00-327C	225	245	20	0.44	0.015	50.5	0.032	73	226	242		242	50.0	40.0				3.0	4.0		1.0	tr	tr	tr		tr		0.5				tr		1.0	0.5	100
1	Troctolitic	P95	17	0.48	0.62	1.295	00-371C	435	440	5	0.65	0.014	32.0	0.036	54	436	438		436	60.0			tr		32.0	1.0		tr	tr	tr					2.0	tr	tr				5.0	tr	100
1	Troctolitic	P100	18	1.295	1.97	1.97	00-340C	765	780	15	1.72	0.022	78.0	0.064	69	771	778		778	60.0	25.0					5.0		2.0	tr	tr	tr				tr						5.0	3.0	100
1	Ultramafic	P25	19	0.075	0.08	0.09	00-357C	335	340	5	0.08	0.015	35.0	0.026	68	337	338		338		3.0	25.0				12.0			3.0	25.0	3.0	5.0		12.0	2.0	tr			2.0		8.0	tr	100
1	Ultramafic	P80	20	0.165	0.2	0.25	00-326C	680	685	5	0.21	0.016	33.0	0.085	82	680	684		684	45.0	30.0					15.0		1.0	tr	tr		tr			1.0						3.0	5.0	100
1	Ultramafic	P85	21	0.25	0.3	0.4	00-357C	535	540	5	0.26	0.026	34.0	0.095	62	536	539		536	45.0	20.0				3.0	25.0		2.0	tr	1.0					1.0						3.0	tr	100
1	Ultramafic	P90	22	0.4	0.5	0.925	99-318C	725	735	10	0.44	0.011	23.0	0.032	45	728	735		735	55.0	20.0	8.0			15.0	tr		1.0	tr		tr				tr	rare					1.0	tr	100
1	Ultramafic	P95	23	0.925	1.35	1.35	99-317C	460	470	10	1.10	0.018	33.0	0.056	86	462	465		462	45.0	15.0				5.0	30.0		3.0	tr	tr					tr						2.0	tr	100
2	Anorthositic	P1	24	0.01	0.01	0.02	00-366C	185	205	20	0.01	0.018	35.0	0.008	47	187	204		187	88.0	3.0				1.0	5.0		tr							tr	tr					3.0	tr	100
2	Anorthositic	P2	25	0.02	0.03	0.04	00-366C	230	240	10	0.02	0.014	32.0	0.011	51	231	239		239	88.0	8.0				1.0	2.0		tr							tr						1.0	tr	100
2	Anorthositic	P3	26	0.04	0.05	0.05	99-320C	165	175	10	0.04	0.023	46.0	0.012	63	167	172		172	85.0	tr					12.0		tr			tr				1.0						2.0	tr	100
2	Troctolitic	P1	27	0.01	0.01	0.02	99-318C	250	270	20	0.02	0.022	42.3	0.011	64	257	267		267	60.0	34.0				tr	4.0		tr		tr		tr			tr						2.0	tr	100
2	Troctolitic	P2	28	0.02	0.03	0.04	00-373C	95	115	20	0.03	0.036	64.0	0.019	84	99	114		99	65.0	30.0				1.0	2.0		1.0	tr	tr		tr			tr						1.0	tr	100
2	Troctolitic	P3	29	0.04	0.05	0.05	00-373C	75	95	20	0.05	0.031	55.3	0.020	75	78	93		93	60.0	35.0				tr	3.0		0.5	tr	tr		tr			tr						1.5	tr	100
2	Troctolitic	P50	30	0.065	0.07	0.08	00-357C	110	130	20	0.07	0.024	53.5	0.029	88	112	126		112	54.0	40.0				tr	4.0		0.5							rare						1.5	tr	100
2	Troctolitic	P80	31	0.09	0.09	0.095	99-320C	315	330	15	0.09	0.017	39.3	0.026	65	319	328		319	65.0	26.0				tr	3.0		1.0		tr					tr						1.0	4.0	100
2	Troctolitic	P95	32	0.115	0.12	0.185	00-369C	335	345	10	0.16	0.021	43.0	0.046	68	336	341		341	60.0	35.0				tr	4.0		tr							tr						1.0	tr	100
2	Ultramafic	P1	33	0.03	0.03	0.035	00-368C	460	465	5	0.03	0.042	70.0	0.033	98	462	463		462	45.0	tr					3.0		1.0		20.0		30.0			tr						1.0	tr	100
2	Ultramafic	P2	34	0.035	0.04	0.045	26055	940	945	5	0.04	0.037	69.0	0.025	88	941	944		944	40.0	50.0				1.5	5.0		0.5	tr	tr		tr									3.0	tr	100
2	Ultramafic	P3	35	0.045	0.05	0.05	26098	145	148.5	3.5	0.05	0.037	76.0	0.014	112	146	148		148	43.0	55.0				tr	2.0		tr							tr						tr	tr	100
2	Ultramafic	P3(a)	35(a)	0.045	0.05	0.05	00-337C	105	110	5	0.05	0.043	71.0	0.023	116	106	108		106	40.0	50.0					5.0		1.0				2.0									2.0	tr	100
3	Anorthositic	P1	36	0.01	0.01	0.02	00-334C	30	50	20	0.01	0.022	47.0	0.009	64	31	50		31	82.0	15.0				tr	2.5		tr	tr	tr		tr							tr		0.5	tr	100
3	Anorthositic	P2	37	0.02	0.03	0.04	00-368C	125	145	20	0.03	0.016	38.0	0.015	62	132	143		132	72.0	20.0				1.0	4.0		1.0			tr									2.0	tr	100
3	Anorthositic	P3	38	0.04	0.05	0.05	00-368C	20	40	20	0.04	0.010	25.8	0.022	47	28	39		39	60.0	35.0				1.0	1.0		1.0		tr										2.0	tr	100
3	Troctolitic	P1	39	0.01	0.01	0.02	00-366C	35	55	20	0.01	0.019	45.3	0.005	50	37	47		37	65.0	35.0				tr	tr		tr							tr						tr	rare	100
3	Troctolitic	P2	40	0.02	0.03	0.04	00-334C	110	130	20	0.03	0.028	57.3	0.012	73	112	128		128	55.0	44.0					0.5		tr	tr	tr		tr									tr	tr	100
3	Troctolitic	P3	41	0.04	0.05	0.05	00-347C	155	175	20	0.04	0.015	49.5	0.015	67	157	174		157	70.0	25.0					2.5		0.5	tr	tr		tr									2.0	tr	100
3	Troctolitic	P50	42	0.07	0.08	0.09	00-347C	280	300	20	0.08	0.018	45.3	0.035	61	282	299		299	66.0	30.0				1.0	2.0		0.5							tr						0.5	tr	100
3	Troctolitic	P85	43	0.115	0.12	0.13	00-326C	60	70	10	0.12	0.031	51.0	0.034	91	61	66		66	60.0	38.0				tr	2.0		tr	tr	tr		tr							tr		tr	tr	100
3	Troctolitic	P95	44	0.165	0.19	0.275	00-369C	305	325	20	0.27	0.030	51.5	0.037	57	306	318		318	68.0	20.0				tr	2.0		tr	tr	8.0	tr	2.0			tr						tr	tr	100
4	Troctolitic	P1	45	0.01	0.01	0.02	00-367C	50	65	15	0.02	0.015	38.7	0.010	59	55	62		55	65.0	tr				tr	3.0		0.5	5.0	10.0	tr	15.0									1.5	rare	100
4	Troctolitic	P2	46	0.02	0.03	0.04	00-367C	260	280	20	0.04	0.024	53.8	0.018	78	269	279		269	45.0	45.0				tr	8.0		0.5	tr	tr		tr			tr						1.5	tr	100
4	Troctolitic	P3	47	0.04	0.05	0.05	00-367C	290	310	20	0.04	0.021	41.3	0.018	64	297	308		308	52.0					2.0	3.0		1.0	tr	40.0			1.0								1.0	tr	100
4	Troctolitic	P25	48	0.065	0.07	0.08	00-370C	20	30	10	0.07	0.010	37.0	0.016	67	22	27		22	52.0	26.0					8.0		0.5	tr	6.0	tr	6.0									1.5	tr	100
4	Troctolitic	P75	49	0.135	0.18	0.185	00-369C	20	30	10	0.14	0.021	39.0	0.043	60	22	29		29	65.0	20.0				1.0	3.0		1.0	tr	8.0	tr	1.0				tr					tr	1.0	100
4	Troctolitic	P90	50	0.335	0.48	0.7	00-367C	170	175	5	0.48	0.023	45.0	0.034	54	171	173		171	52.0	3.0	3.0				12.0	5.0	4.0	3.0	10.0											tr	8.0	100
4	Troctolitic	P95	51	0.7	0.92	1.225	00-367C	395	400	5	0.92	0.028	76.0	0.080	96	395	398		398	55.0	2.0				1.0	2.0		2.0	2.0	28.0	tr	2.0			tr						3.0	3.0	100
5	Troctolitic	P1	52	0.01	0.01	0.02	26064	44	54	10	0.01	0.035	62.0	0.009	58	45	53		45	74.0	24.0					2.0			tr	tr		tr			rare						tr	rare	100
5	Troctolitic	P3	53	0.04	0.05	0.05	26064	264	269	5	0.04	0.017	42.0	0.005	56	264	268		264	68.0	5.0				0.5	0.5		tr	1.0	24.0	tr	tr									0.5	0.5	100
5	Troctolitic	P3(a)	53(a)	0.04	0.05	0.05	26064	146	156	10	0.05	0.032	67.0	0.031	78	148	151		148	55.0	40.0				tr	2.0		tr	tr	2.0		1.0									tr	tr	100
6	Troctolitic	P1	54	0.02	0.02	0.03	26056	110	125	15	0.02	0.034	66.7	0.025	85	113	123		123	55.0	42.0					3.0		tr	tr	tr		tr			tr						tr	tr	100
6	Troctolitic	P2	55	0.03	0.04	0.04	26056	135	153	18	0.04	0.037	62.7	0.032	89	138	152		138	55.0	45.0					tr		tr		tr	tr										tr	tr	100
20	Virginia	P25	56	0.92	1.25	2.115	00-361C	737	749	15	1.91	0.015	22.3	0.039	225	739	748		739			10.0		70.0				5.0		2.0								10.0			tr	3.0	100
20	Virginia	P75	57	3.565	4.15	4.32	00-364C	210	229	19	4.11	0.018	26.8	0.017	872	211	228		211			65.0		25.0		tr		0.5	1.5							tr		2.0				6.0	100
20	Virginia	P90	58	4.96	5.07	5.565	00-337C	510	520	10	5.12	0.016	29.5	0.019	492	511	519		519					85.0	tr			2.0	tr										3.0				10.0	100

#	Rock Name		Oxides		Percent		Break-Down of Sulfide Mineralogy																	Percent		Silicates/Alteration Minerals/Phosphates/Carbonates								
																										Plagioclase			Olivine			K-spar		
			Magnetite	Ilmenite	Check		Pyrite	Pyrrhotite	Chalcopyrite	Bornite	Digenite	Chalcocite	Covellite	Cubanite	Pentlandite	Violarite	PGMs	Silver	Mackinawite/ Valleriite	Sphalerite	Galena	Millerite	Enargite	Talnakhite	Check		crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture
1	Anorthositic Troctolite		5.0	95.0	100			35.0	50.0						15.0	tr	rare			tr	rare				100		subhedral	1-10mm	adcum./sympl. Opx	anhedral	0.5-3mm	intercumulate			
2	Anorthositic Troctolite		5.0	95.0	100			40.0	40.0	tr				10.0	10.0									100		subhedral	1-5mm	adcumulate	anhed/subhed	1-5mm oikos	poikilitic			
3	Troctolitic Anorthosite		5.0	95.0	100			45.0	45.0	tr					5.0	rare			5.0	tr					100		subhedral	2-4mm	adcum./sympl. Opx	anhedral	1-4mm	adcumulate			
4	Norite			100.0	100		tr	80.0	18.0						2.0					tr					100		subhedral	2-8mm	adcumulate	anhedral	1-2mm	adcumulate			
5	Recrystallized/Granoblastic Calc-Silicate							tr	tr											95.0	5.0				100								anhedral	0.5-1mm	porphyroblastic
6	Granoblastic cordierite biotite Graywacke		100.0		100			90.0	5.0						tr					5.0					100								anhedral	0.1-0.15mm	granoblastic
7	Granoblastic Graywacke			100.0	100			95.0	5.0											tr	tr				100								anhedral	0.01-0.1mm	granoblastic
8	Granoblastic cordierite biotite pyrrhotite Graywacke		tr	100.0	100			100.0	tr																100								subhed/anhed	1-2mm	granoblastic
9	Granoblastic cordierite Graywacke		100.0		100			100.0	tr																100		subhedral	0.2-0.5mm	random grains				subhedral	2-5mm	poikilitic/wk foliated
9(a)	Granoblastic cordierite Graywacke			100.0	100			90.0	5.0					1.0	1.0					3.0					100								subhedral	0.1-0.5mm	granoblastic/porphy.
10	Augite Troctolite		2.0	98.0	100			5.0	80.0			rare			15.0	tr				tr					100		subhedral	1-2mm	adcumulate						
10(a)	Augite Troctolite		60.0	40.0	100			55.0	20.0	tr				20.0	5.0					tr					100		subhedral	0.5-1.5mm	adcum./sympl. Opx	anhedral	1-2mm	adcumulate			
10(b)	Augite Troctolite		20.0	80.0	100			60.0	40.0						tr				tr	tr					100		subhedral	1-10mm	adcum./sympl. Opx	anhedral	2-8mm	adcumulate			
10(c)	Troctolite			100.0	100			60.0	35.0						5.0					tr					100		subhedral	1-5mm	adcum./sympl. Opx	anhedral	1-4mm	adcumulate			
11	Olivine Gabbro		85.0	15.0	100			45.0	35.0					10.0	10.0	tr			tr						100		subhedral	1cm	orthocum./sympl. Opx	anhedral	1-3mm	orthocumulate			
12	Augite Troctolite		5.0	95.0	100			30.0	25.0	tr				35.0	10.0					tr					100		subhedral	1-5mm	adcumulate	anhedral	1-2mm	adcum/intercum/sympl			
13	Augite Troctolite		50.0	50.0	100			55.0	20.0					5.0	20.0				tr	tr					100		subhedral	1-2mm	mesocumulate/wk zone	anhedral	0.1-1mm	adcumulate/subequant			
14	Augite Troctolite		tr	100.0	100			30.0	50.0					20.0	tr										100		subhedral	1mm	adcumulate	equant/subhed	0.1-0.5mm	adcumulate			
15	Augite Troctolite		5.0	95.0	100			60.0	30.0	tr					10.0					tr			tr		100		subhedral	1-2mm	adcum./sympl. Opx	equant	0.2-0.4mm	adcumulate			
16	MelaTroctolite (Picrite)/Troctolite		tr	100.0	100			65.0	35.0						tr	tr			tr						100		subhedral	1-4mm	adcum./sympl. Opx	anhedral	1-10mm oikos	poikilitic			
17	Norite		30.0	70.0	100			95.0	5.0					tr						tr	tr				100		subhedral	2-12mm	adcum./sympl. Opx						
18	Augite Troctolite		10.0	90.0	100			80.0	10.0			tr		4.0	6.0								tr		100		subhedral	1-2mm	adcum./sympl. Cpx	anhedral	1-3mm oikos	poikilitic/adcumulate			
19	MelaTroctolite (Picrite)		10.0	90.0	100				60.0						40.0					tr	tr				100					anhedral	3mm	poikilitic	anhedral	1-8mm	sauss. alt. of Plag.
20	MelaTroctolite (Picrite)		50.0	50.0	100			20.0	20.0	tr				35.0	25.0		rare		tr	tr					100		subhedral	1-4mm	adcumulate	anhedral	5-8mm	poikilitic/sympl. Cp.			
21	MelaGabbro			100.0	100			50.0	50.0						tr				tr						100		subhedral	1-3mm	adcumulate	anhedral	0.5-1mm	adcumulate			
22	Basalt Inclusion		10.0	90.0	100			80.0	10.0					5.0	5.0				tr	tr					100		subhedral	0.5-1mm	zoned	anhedral	5mm oikos	poikilitic	anhedral	3-5mm oikos	poikilitic
23	MelaGabbro		tr	100.0	100			10.0	85.0	5.0					tr		rare			tr					100		subhedral	0.5-6mm	adcum./sympl. Opx	anhedral	1cm oikos	poikilitic/adcumulate			
24	Gabbroic Anorthosite		1.0	99.0	100			45.0	40.0	tr			tr	10.0	5.0					rare					100		subhedral	2-6mm	adcum./sympl. Opx	anhedral	5mm oikos	poikilitic			
25	Troctolitic Anorthosite		40.0	60.0	100			5.0	90.0	tr					5.0				tr						100		subhedral	2-6mm	adcum./sympl. Opx	anhedral	2-6mm	intercum./interst.			
26	Gabbroic Anorthosite		65.0	35.0	100			40.0	30.0					15.0	15.0										100		subhedral	1-5mm	adcumulate	anhedral	0.1-0.2mm	random/trace			
27	Troctolite		30.0	70.0	100			10.0	50.0	tr				25.0	15.0	tr				tr					100		subhedral	1-8mm	adcum./sympl. Opx	anhedral	1-5mm	adcumulate			
28	Troctolite		30.0	70.0	100			30.0	60.0	tr			tr	5.0	5.0		rare								100		subhedral	1-6mm	adcum./sympl. Opx	anhedral	4-6mm oikos	adcumulate/poikilitic			
29	Troctolite		40.0	60.0	100			30.0	40.0	tr			tr	20.0	10.0	tr				tr					100		subhedral	1-3mm	adcum./sympl. Opx	anhedral	4-5mm oikos	subpoik./adcum./sympl.			
30	Troctolite		tr	100.0	100			40.0	45.0					10.0	5.0					tr			tr		100		subhedral	1-3mm	adcumulate	anhed/subhed	0.5-1mm	poikilitic/adcumulate			
31	Troctolite		10.0	90.0	100			20.0	35.0	tr			tr	30.0	15.0				tr	tr			tr		100		subhedral	1-3mm	adcum./sympl. Opx	anhedral	1-3mm	adcumulate			
32	Troctolite		tr	100.0	100			60.0	20.0						15.0				5.0						100		subhedral	1-3mm	adcum./sympl. Opx	anhedral	2-3cm oikos	poikilitic/adcumulate			
33	Serpentinized MelaTroctolite (Picrite)		50.0	50.0	100				100.0					tr	tr										100		subhedral	0.5-1mm	adcumulate	equant	0.3-0.5mm	adcumulate			
34	MelaTroctolite (Picrite)		5.0	95.0	100			30.0	40.0	tr	tr		tr	15.0	15.0										100		subhedral	1-4mm	adcumulate	anhedral	2-5mm oikos	subpoik./adcum./sympl.			
35	MelaTroctolite (Picrite)		2.0	98.0	100			30.0	45.0	tr				25.0	tr										100		subhedral	1-2mm	adcumulate	anhedral	2-5mm	poikilitic/adcumulate			
35(a)	MelaTroctolite (Picrite)		5.0	95.0	100			35.0	40.0					15.0	10.0					tr					100		subhedral	0.5-1mm	adcumulate	equant	0.1-0.5mm	adcumulate			
36	Troctolitic Anorthosite		35.0	65.0	100			45.0	45.0	tr					10.0										100		subhedral	2-8mm	adcumulate	anhedral	6-8mm oikos	poikilitic			
37	Anorthositic Troctolite		5.0	95.0	100			35.0	55.0						5.0				5.0	tr					100		subhedral	1-3mm	orthocumulate/zoned	anhedral	0.5-1cm oikos	poikilitic			
38	Troctolite		tr	100.0	100			55.0	40.0	2.0					tr				3.0	rare					100		subhed/anhed	0.5-5mm	adcum./sympl. Opx	anhedral	3-4cm oikos	poikilitic/adcumulate			
39	Troctolite		10.0	90.0	100			tr	75.0	15.0			10.0	tr	tr										100		subhedral	0.5-5mm	adcum./sympl. Opx	anhedral	3mm	poikilitic/adcumulate			
40	Troctolite		30.0	70.0	100			45.0	40.0	tr				5.0	10.0										100		subhedral	1-5mm	adcumulate	anhedral	1-2cm oikos	poikilitic			
41	Troctolite/Anorthositic Troctolite		50.0	50.0	100			45.0	45.0	tr				5.0	5.0										100		subhedral	1-3mm	adcumulate	anhedral	1-2cm oikos	poikilitic			
42	Troctolite		tr	100.0	100			35.0	60.0					tr	tr				5.0						100		subhedral	1-5mm	adcumulate	anhedral	3-4cm oikos	poikilitic			
43	Troctolite		50.0	50.0	100			10.0	75.0	tr				10.0	5.0					tr					100		subhedral	1-4mm	adcum./sympl. Opx	anhedral	3-6cm oikos	adcumulate			
44	Troctolite		1.0	99.0	100			85.0	10.0										5.0						100		subhedral	1-4mm	adcum./sympl. Opx	anhedral	4-5mm oikos	poikilitic/adcumulate			
45	Serpentinized Troctolite		80.0	20.0	100				95.0	5.0					tr										100		subhedral	1-5mm	adcumulate	anhedral	1-2mm	adcumulate/subpoikilitic			
46	Augite Troctolite		30.0	70.0	100			15.0	48.0	tr	tr		tr	30.0	5.0				tr	tr				2.0	100		subhedral	2-8mm	adcumulate	anhedral	1-4mm	adcumulate			
47	Troctolite		50.0	50.0	100			5.0	80.0						15.0		rare								100		subhedral	1-5mm	adcum./sympl. Opx						
48	Augite Troctolite		tr	100.0	100			70.0	20.0	tr				2.0	3.0				5.0	tr					100		subhedral	0.5-2mm	adcumulate	anhedral	4cm oikos	adcumulate/subpoikilitic			
49	Troctolite		tr	100.0	100			35.0	25.0										40.0						100		subhedral	1-4mm	adcumulate	anhedral	1-3mm	adcumulate			
50	Augite Troctolite			100.0	100			70.0	25.0						5.0					tr					100		subhedral	1-3mm	orthocumulate/zoned	anhedral	0.5-1mm	cumulate	anhedral	2-6mm	massive/cavity in-fill
51	Chloritic Troctolite		tr	100.0	100			tr	3.0			tr			95.0	2.0				tr					100		anhedral	0.5-2.5mm	adcumulate	anhedral	0.05-1mm	adcumulate/altered			
52	Anorthositic Troctolite		50.0	50.0	100				40.0	30.0			30.0		tr										100		subhedral	1-10mm	adcumulate	anhedral	1-4mm	adcumulate			
53	Chloritic Troctolite		tr	100.0	100			25.0	50.0					tr	5.0	tr			20.0	tr					100		subhedral	2-5mm	adcumulate	anhedral	1-4mm	adcumulate/altered			
53(a)	Troctolite		5.0	95.0	100			5.0	65.0	10.0			10.0	5.0	5.0										100		subhedral	2-6mm	adcumulate	anhedral	0.1-10mm	adcumulate			
54	Troctolite		45.0	55.0	100			tr	70.0	5.0	5.0		5.0	10.0	4.0					1.0					100		subhedral	1-4mm	adcumulate	anhedral	1-3mm	adcumulate/sympl. Cp			
55	Troctolite		5.0	95.0	100			5.0	75.0	10.0	3.0		2.0		5.0		tr								100		subhedral	0.5-2mm	adcumulate	subhed/equant	0.5-2mm	adcumulate			
56	Granoblastic graphitic cordierite Argillite		100.0		100			100.0	tr											tr					100								anhedral	0.1-0.5mm	granoblastic
57	Granoblastic pyrrhotite Graywacke							98.0	tr											1.5	0.5				100								anhedral	0.1-0.4mm	granoblastic/porphy.
58	Bedded Granoblastic pyrrhotite cordierite Graywacke							98.0	tr											2.0					100										

#	Silicates/Alteration Minerals/Phosphates/Carbonates																														
	Quartz			Cordierite			Opx			Cpx			Amphibole			Biotite			White Mica			Chlorite			Clay Minerals			Serpentine		
	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture
1							anhedral	0.1-0.3mm	sympl./rims Ol	subhedral	2-5mm ophites	ophitic/rims Olivine				subhedral	0.1-0.5mm	interst./rims Ox&Sulf				anhedral	<0.01mm	alt. of Olivine						
2										subhedral	1-2cm ophites	ophitic/rims Olivine				subhedral	0.1-0.2mm	interst./rims Ox&Sulf												
3							anhedral	0.1-0.5mm	sympl./rims Ol	subhedral	5mm ophites	ophitic/rims Olivine				subhedral	0.1-0.5mm	interst./rims Ox&Sulf	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.1-0.4mm	alt. of Olivine						
4							subhedral	1-3mm	invert. Pigeonite	subhedral	3-4mm ophites	ophitic				subhedral	0.1-0.5mm	interst./rims Ox&Sulf	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.2mm	alt. of Olivine	anhedral	<0.01mm	brn clouding of Plag.			
5										anhedral	1-3mm ophites	poikiloblastic							anhedral	0.5-2mm	muscovite/poik.									
6				anhedral	0.1mm	granoblastic										subhedral	0.1-0.5mm	wk foliated	anhedral	0.01-0.05mm	sericitic alt. of Kspar									
7				anhedral	0.01-0.1mm	granoblastic				anhedral	0.1-0.3mm	poikiloblastic				subhedral	0.05-0.1mm	interstitial				anhedral	0.01-0.05mm	granoblastic				anhedral	<0.01mm	in-fill fractures
8				subhed/anhed	0.5-1mm	granoblastic										subhedral	0.1-2mm	granoblastic	subhedral	0.1-0.2mm	fibrous/radiating				anhedral	<0.01mm	brn clouding of Plag.			
9				anhedral	0.05-0.5mm	granoblastic	subhedral	0.1-0.2mm	granoblastic							subhedral	0.1-3mm	wk foliated	anhedral	<0.01mm	sericitic alt. of Plag.				anhedral	<0.01mm	brn clouding of Plag.			
9(a)				anhedral	0.1-0.3mm	granoblastic	anhedral	0.1-0.3mm	granoblastic							anhed/subhed	0.1-1mm	poikioloblast/rims sulf.												
10										subhedral	3-4mm ophites	ophitic				subhedral	0.1-1mm	interst./rims Ox&Sulf	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.1mm	alt. of Olivine	anhedral	<0.01mm	brn clouding of Plag.	anhedral	0.01-0.05mm	alt. of Olivine
10(a)							anhedral	0.1-0.5mm	rim Ol/sympl. Plag/Mag	subhedral	1cm ophites	ophitic/sympl. Mag.				subhedral	0.5-2mm	interst./rims Ox&Sulf				anhedral	0.01-0.5mm	alt. of Olivine				anhedral	<0.01mm	alt. of Olivine
10(b)										subhedral	1.5-2cm ophite	ophitic/rims Ol/sympl.				subhedral	0.1-2mm	interst./rims Ox&Sulf				anhedral	<0.01mm	alt. of Olivine	anhedral	<0.01mm	brn clouding of Plag.			
10(c)							anhedral	0.1-1mm	rim Ol/sympl. Plag	subhedral	3-5mm ophites	ophitic				subhedral	0.1-2mm	interst./rims Oxides	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.5mm	alt. of Olivine						
11							anhedral	0.1-0.5mm	sympl./rims Ol	subhedral	1.5cm	pegmatitic				subhedral	0.1-4mm	interst./rims Ox&Sulf	anhedral	0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.1mm	alt. of Olivine						
12							anhedral	0.05-0.1mm	sympl./rims Ol	subhedral	2-3cm ophites	ophitic/rims Olivine				subhedral	0.1-2.75mm	interst./rims Ox&Sulf												
13										subhedral	1cm ophites	ophitic				subhedral	0.1-2mm	poikilitic/rims Ox	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.2mm	alt. of Olivine				anhedral	0.01-0.1mm	alt. of Olivine
14										subhedral	3-5mm ophites	ophitic				subhedral	0.5-1mm	interst./rims Ox&Sulf												
15							anhedral	0.1-0.3mm	rim Ol/sympl. Plag	subhedral	3-5mm ophites	ophitic/sympl. Plag.				subhedral	0.1-0.8mm	interst./rims Oxides	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.1mm	alt. of Olivine	anhedral	<0.01mm	brn clouding of Plag.	anhedral	<0.01mm	alt. of Olivine
16							anhedral	0.1-0.5mm	sympl./rims Ol	subhedral	1cm ophites	ophitic				subhedral	0.1-3mm	interst./rims Ox&Sulf	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.3mm	alt. of Olivine	anhedral	<0.01mm	brn clouding of Plag.			
17	anhedral	3mm	cavity in-fill/zr & rutile				anhedral	1-6mm	rim Ol/sympl. Plag	subhedral	2-4mm ophites	ophitic/rims rem. Ol				subhedral	0.1-1.2mm	interst./incl.	anhedral	<0.01-0.8mm	ser. rad. alt. of Plag.	anhedral	0.01-0.05mm	alt. of Olivine						
18										subhedral	1cm ophites	ophitic				subhedral	0.1-3mm	interst./rims Sulf&Ox	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	<0.01mm	alt. of Olivine	anhedral	<0.01mm	brn clouding of Plag.			
19										anhedral	1-6mm	interstitial							anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.1mm	alt. of Olivine	anhedral	<0.01mm	brn clouding of Plag.	anhedral	<0.01mm	alt. of Olivine
20										subhedral	3-4mm ophites	subophitic				subhedral	0.1-2mm	interst./rims Sulf&Ox	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.1mm	alt. of Olivine				anhedral	<0.01mm	alt. of Olivine
21							anhedral	1-2mm	rims Olivine	subhedral	2-5mm ophites	ophitic				subhedral	0.1-1mm	interst./rims Ox&Sulf	subhedral	0.01-0.1mm	sericitic alt. of Plag.	anhedral	0.1-1mm	alt. of Olivine						
22							anhedral	4-5mm ophites	ophitic/invert. Pigeonite	subhedral	1-2mm ophites	ophitic				subhedral	0.1-1.5mm	poikilitic	anhedral	<0.01mm	sericitic alt. of Plag.				anhedral	<0.01mm	brn clouding of Plag.			
23							anhedral	0.1-1.5mm	rim Ol/sympl/invert. Pig.	subhedral	2-5mm ophites	ophitic				subhedral	0.1-1.2mm	interst./rims Sulf&Ox	subhedral	0.01-0.05mm	sericitic alt. of Plag.	anhedral	<0.01mm	alt. of Olivine						
24							anhedral	0.05-0.1mm	rim Ol/sympl. Plag/Mag	subhedral	1cm ophites	ophitic				subhedral	0.1-0.5mm	interst./rims Ox&Sulf												
25							anhedral	0.1-1mm	rim Ol/sympl. Plag/Mag	subhedral	1-2cm ophites	ophitic				subhedral	0.1-0.6mm	interst./rims Ox&Sulf												
26										subhedral	>3cm ophites	ophitic				subhedral	2mm oikos	poikilitic/rims Ox							anhedral	<0.01mm	brn clouding of Plag.			
27							anhedral	0.1-0.5mm	rim Ol/sympl. Plag/Ilm	subhedral	1-1.5cm ophite	ophitic/rims Olivine				subhedral	0.1-1mm	interst./incl. in Cpx				anhedral	<0.01mm	alt. of Ol./microfract.				anhedral	<0.01mm	alt. of Olivine
28							anhedral	0.1-1mm	rim Ol/sympl. Plag	subhedral	3-4mm ophites	ophitic/rims Olivine				subhedral	0.1-0.8mm	interst./rims Oxides	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.1mm	alt. of Olivine				anhedral	<0.01mm	alt. of Olivine
29							anhedral	0.05-0.1mm	rim Ol/sympl. Plag	subhedral	2cm ophites	ophitic/rims Olivine				subhedral	0.1-2mm	interst./rims Ox&Sulf	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	<0.01mm	alt. of Olivine				anhedral	<0.01mm	alt. of Olivine
30							subhedral	1mm	rim Ol/sympl. Ilm	subhedral	5mm ophites	ophitic				subhedral	0.2-1mm	interst./rims Ox&Sulf												
31							anhedral	0.1-0.2mm	rim Ol/sympl. Plag/Mag	subhedral	1-2cm ophites	ophitic/rims Olivine				subhedral	0.1-2mm	subpoik/rims Ox&Sulf				anhedral	0.01-0.05mm	alt. of Olivine						
32							anhedral	0.1mm	rim Ol/sympl. Plag	subhedral	1-2cm ophites	ophitic				subhedral	0.1-1mm	interst./incl. in Cpx												
33										subhedral	2-5mm ophites	ophitic				subhedral	0.1-1mm	interst./rims oxides				anhedral	0.01-0.3mm	alt. of Olivine				anhedral	<0.01mm	alt. of Olivine
34							anhedral	2-3mm ophites	rims Ol/ophitic	subhedral	5-8mm ophites	ophitic/rims Olivine				subhedral	0.1-1.5mm	interst./rims Ox&Sulf	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.1mm	alt. of Olivine				anhedral	0.01-0.02mm	alt. of Olivine
35							anhedral	0.5-1mm	rims Olivine	subhedral	2-3mm ophites	ophitic				subhedral	0.5-1mm	interst./rims Ox&Sulf												
35(a)										subhedral	5-10mm ophite	ophitic				subhedral	1-3mm oikos	poikilitic/rims Ox										anhedral	0.01-0.1mm	alt. of Olivine
36							anhedral	0.1-0.5mm	rim Ol/sympl. Mag	subhedral	1cm ophites	ophitic/rims Olivine				subhedral	0.1-0.8mm	interst./incl. in Cpx	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.3mm	alt. of Olivine				anhedral	<0.01mm	alt. of Olivine
37							anhedral	0.1-0.5mm	rim Ol/sympl. Plag/Mag	subhedral	1cm ophites	ophitic				subhedral	0.1-0.5mm	interst./rims Ox&Sulf							anhedral	<0.01mm	brn clouding of Plag.			
38							anhedral	0.1-1mm	rim Ol/sympl. Plag	subhedral	1cm ophites	ophitic				subhedral	0.1-1mm	interst./rims Ox&Sulf				anhedral	0.05-0.1mm	alt. of Olivine						
39							anhedral	0.1-0.5mm	rim Ol/sympl. Plag/Mag	subhedral	4-5mm ophites	ophitic				subhedral	0.1-0.5mm	interst./rims Ox&Sulf												
40										subhedral	3-5mm ophites	ophitic/rims Olivine				subhedral	0.1-0.4mm	interst./rims Ox&Sulf	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.2mm	alt. of Olivine				anhedral	<0.01mm	alt. of Olivine
41										subhedral	1cm ophites	ophitic				subhedral	0.1-1mm	interst./rims Oxides	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.05mm	alt. of Olivine				anhedral	<0.01mm	alt. of Olivine
42							anhedral	1-2mm	rim Ol/sympl. Plag/Mag	subhedral	1-2cm ophites	ophitic				subhedral	0.5-1mm	interst./rims Ox&Sulf												
43							anhedral	0.1-0.8mm	rim Ol/sympl. Plag/Mag	subhedral	5mm ophites	ophitic/rims Olivine				subhedral	0.1-1mm	interst./rims Oxides	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.1mm	alt. of Olivine				anhedral	<0.01mm	alt. of Olivine
44							anhedral	0.05-0.2mm	rim Ol/sympl. Plag	subhedral	3-5mm ophites	ophitic/rims Olivine				subhedral	0.1-2mm	interst./rims Ox&Sulf	subhedral	0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.1mm	alt. of Olivine	anhedral	<0.01mm	brn clouding of Plag.	anhedral	0.01-0.05mm	alt. of Olivine
45							anhedral	0.1-1mm	rim Ol/sympl. Mag	subhedral	1-2cm ophites	ophitic/rims Olivine				subhedral	0.1-0.5mm	interst./rims Ox	anhedral	<0.01-0.1mm	sericitic alt. of Plag.	anhedral	0.01-1mm	alt. of Olivine	anhedral	<0.01mm	brn clouding of Plag.	anhedral	<0.01mm	alt. of Olivine
46							anhedral	0.1-0.2mm	rims Olivine	subhedral	1cm ophites	ophitic/rims Olivine				subhedral	0.1-1.5mm	interst./rims Ox	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.1mm	alt. of Olivine				anhedral	<0.01mm	alt. of Olivine
47							anhedral	0.05-0.1mm	rim Ol/sympl. Plag/Mag	subhedral	3mm ophites	ophitic				subhedral	0.1-1mm	interst./rims Oxides	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-2mm	alt. of Olivine						
48										subhedral	1-1.5cm ophite	ophitic/rims Olivine				subhedral	0.1-0.8mm	interst./rims Ox	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.3mm	alt. of Olivine	anhedral	<0.01mm	brn clouding of Plag.	anhedral	<0.01mm	alt. of Olivine
49							anhedral	0.1-0.5mm	rim Ol/sympl. Plag	subhedral	0.5-1cm ophite	ophitic/rims Olivine				subhedral	0.1-5mm	interst./rims Ox&Sulf	subhedral	0.5mm	sericitic alt. of Plag.	anhedral	0.01mm	alt. of Olivine	anhedral	<0.01mm	brn clouding of Plag.	anhedral	0.01mm	alt. of Olivine
50										subhedral	1-4mm	cumulate	subhedral	0.1-1mm	ural. Alt. pyroxene	subhedral	0.1-5mm	interst./incl. rims Sulf	anhed/subhed	0.01-0.5mm	ser/sauss alt of Plag	anhedral	0.01-0.1mm	radiat. alt. of Olivine						
51							anhedral	0.5-1mm	rims Olivine	anhedral	1-2mm	interstitial				subhedral	0.1-1mm	interst./rims Ox&Sulf	subhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.05mm	alt. of Olivine	anhedral	<0.01mm	brn clouding of Plag.	anhedral	0.01mm	alt. of Olivine
52										subhedral	0.5-1cm ophite	ophitic/rims Olivine							anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	<0.01mm	alt. of Olivine				anhedral	<0.01mm	alt. of Olivine
53							anhed/subhed	0.1-0.5mm	rims Ol/ophitic	subhedral	1cm ophites	ophitic				subhedral	0.1-0.5mm	interst./rims Ox&Sulf	subhedral	0.01mm	sericitic alt. of Plag.	subhed/anhed	0.01-0.1mm	alt. of Olivine	anhedral	<0.01mm	brn clouding of Plag.	anhedral	0.01mm	alt. of Olivine
53(a)							anhedral	0.05-0.2mm	rim Ol/sympl. Mag	subhedral	5mm ophites	ophitic/rims Olivine				subhedral	0.1-1mm	incl. Cpx/rims Ol	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.05mm	alt. of Olivine				anhedral	<0.01mm	alt. of Olivine
54										subhedral	5mm ophites	ophitic/rims Olivine				subhedral	4mm oikos	poik./rims Ox&Sulf	anhedral	<0.01mm	sericitic alt. of Plag.	anhedral	0.01-0.1mm	alt. of Olivine				anhedral	<0.01mm	alt. of Olivine
55										subhedral	1-2mm ophites	subophitic/rims Ol				subhedral	0.1-0.5mm	rims Ox&Sulf				anhedral	0.01-0.05mm	alt. of Olivine	anhedral	<0.01mm	brn clouding of Plag.			
56				anhedral	0.2-0.8mm	poikilobl./granoblastic										subhedral	0.1-0.3mm	random				anhedral	0.01-0.1mm	random						
57				anhedral	0.05-0.2mm	granoblastic				subhedral	0.1-2mm	porphyroblastic				subhedral	0.1-0.8mm	porphyroblastic	subhedral	0.1-0.4mm	muscovite									
58				anhedral	0.1-0.2mm	granoblastic/layered	anhedral	0.5mm	granoblastic							subhedral	1-3mm oikos	poikiloblastic/layered	subhedral	0.5mm oikos	poikiloblastic									

#	Silicates/Alteration Minerals/Phosphates/Carbonates																								Oxides/Sulfides					
	Uralite			Epidote			Apatite			Zircon			Rutile			Graphite			Carbonates			Vesuvianite/Idocrase			Magnetite			Ilmenite		
	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture
1							euhedral	0.1-1.2mm	interst./incl																anhedral	0.01-0.5mm	interst./incl./sympl.	anhed/euhed	0.01-4mm	interst./incl. in Plag
2							euhedral	0.1mm	interst./incl																subhedral	0.1-0.5mm	interst.	subhedral	0.1-1mm	interst./incl.
3							euhedral	0.1-0.7mm	interst./incl																anhedral	0.01-0.8mm	interst.	subhed/anhed	0.01-1.5mm	interst./incl./sympl.
4							euhedral	0.1-3mm	interst./incl										anhedral	0.1mm	sauss. alt. of Plag.							subhed/anhed	0.01-4mm	interst./incl.
5																anhedral	0.1-0.6mm	interstitial/random	anhedral	0.1-0.3mm	poikiloblastic	anhedral	1-2mm	porphyroblastic						
6																subhedral	0.1-0.2mm	weakly lineated							anhedral	0.01-0.05mm	interst./random			
7																anhedral	0.05-0.1mm	granoblastic										anhedral	0.05-0.1mm	subpoikiloblastic
8							euhedral	0.5mm	incl. within Biot							subhedral	0.1-0.2mm	interstitial							subhedral	0.01-0.1mm	incl. in Po	anhed/subhed	0.01-0.1mm	interst./incl.
9																subhedral	0.1-0.4mm	interstitial							anhedral	0.01-0.1mm	equant/random grains			
9(a)																subhedral	0.1-0.25mm	granobl./wk. lineated										anhedral	0.01-0.1mm	granoblastic
10				anhedral	0.01-0.05mm	sauss. alt. of Plag.	euhedral	0.1-0.35mm	interst./incl																anhedral	0.01-0.3mm	incl in Ilm/second. Alt	anhedral	0.01-3mm	interst./incl.
10(a)							euhedral	0.1-0.5mm	interst./incl. Cpx/Biot.																anhedral	0.01-3.5mm	interst./sympl. Cpx	anhedral	0.01-2.5mm	interst./incl.
10(b)							euhedral	0.1-1mm	interst./incl																anhedral	0.01-2mm	interst./sympl. Cpx	subhedral	0.01-4mm	interst./exsol. in Mag
10(c)							euhedral	0.1-0.5mm	incl. within Cpx/Biot																			anhed/subhed	0.01-1.3mm	interst./incl.
11							euhedral	0.1-1mm	interst./incl																anhedral	0.01-4mm	interst./incl.	anhed/subhed	0.01-1mm	interst./exsol. in Mag
12							euhedral	0.1-0.5mm	incl. within Cpx/Biot																anhedral	0.01-1.2mm	incl. in Ilm.	subhedral	0.01-4mm	interst./incl./sympl.
13							euhedral	0.1-0.5mm	interst./incl																anhedral	0.01-3mm	poikilitic/incl.	anhedral	0.01-3mm	poikilitic/incl.
14							euhedral	0.2mm	interstitial																subhedral	0.1mm	interst./incl. in Ilm.	subhed/anhed	0.1-1mm	interst./incl.
15							euhedral	0.1-0.6mm	interst./incl																anhedral	0.01-0.3mm	incl. in Ilm.	anhed/subhed	0.01-2.5mm	interst./incl. in Cpx
16	anhedral	<0.01mm	alt. of Cpx				euhedral	0.2-2mm	interst./incl										anhedral	0.01-0.1mm	sauss. alt. of Plag				subhedral	0.01-0.1mm	exsol. lamell. In Ilm.	anhed/subhed	0.01-2mm	interst./incl.
17							euhedral	0.1-2.2mm	interst./incl	subhedral	1.2mm	interstitial	subhedral	0.05-0.4mm	interstitial										anhedral	0.01-3.5mm	interst./subpoikilitic	anhed/subhed	0.01-2.5mm	interst./subpoikilitic
18							euhedral	0.1-0.5mm	interstitial																subhed/anhed	0.01-1mm	exsol. lamell. In Ilm.	anhed/subhed	0.01-5mm	interst./incl.
19				anhedral	0.1-1mm	sauss. alt. of Plag.	euhedral	0.1-2.5mm	interst./incl	euhedral	0.5mm	zoned, inclusions							anhedral	2mm	sauss. alt. Plag.				subhedral	0.5-2mm	incl. in Ilm.	anhed/subhed	0.5-13mm	interst./skeletal
20							euhedral	0.1-1mm	interst./incl																anhedral	0.1-3.5mm	poikilitic/incl.	anhedral	0.01-2.2mm	interst./incl.
21							euhedral	0.1-0.5mm	incl. within Cpx/Biot																			subhedral	<0.01-5mm	interst./incl. in Cpx
22							euhedral	0.1-1mm	interst./incl	subhedral	0.8mm	zoned, interstitial													anhed/subhed	0.01-0.8mm	interst./incl.	subhed/anhed	0.01-1mm	prism./weak lineation
23							euhedral	0.1-0.5mm	interst./incl																anhedral	0.01-0.05mm	incl. in Ilm.	subhed/anhed	0.01-1.5mm	interst./incl.
24							subhedral	0.01-0.5mm	incl. within Cpx/Biot	subhedral	0.01-0.06mm	incl. in Plag.													anhedral	0.01-0.2mm	interst./incl./sympl.	subhed/anhed	0.01-2.5mm	interst./incl. Plag/Cpx
25							euhedral	0.05-1mm	interstitial																anhedral	0.01-1mm	interst./sympl. Opx	anhed/subhed	0.01-2.5mm	interst./exsol. in Mag
26							euhedral	0.1-1.5mm	interst./incl																anhedral	0.01-5mm	poikilitic/incl. in Ilm	anhedral	0.01-1.5mm	interstitial
27							euhedral	0.05-0.3mm	interst./incl																anhedral	0.1-1mm	incl in Ilm/second. Alt	anhedral	0.01-2.5mm	interst./incl./sympl.
28							euhedral	0.1-0.8mm	interst./incl Cpx																anhedral	0.01-0.9mm	incl in Ilm/second. Alt	anhedral	0.01-2mm	interst./incl.
29							euhedral	0.1-0.2mm	interst./incl Cpx/Biot																anhedral	0.01-0.8mm	interst./incl. in Ilm.	subhedral	0.01-2mm	interst./incl. in Cpx
30							euhedral	0.1mm	incl. in Olivine																anhedral	0.01-1mm	exsol. lamell. In Ilm.	anhedral	0.01-2mm	interst./incl./sympl.
31							euhedral	0.1-0.3mm	interstitial																anhedral	0.01-1mm	interst./sympl. Opx	anhedral	0.01-2mm oiko	subpoikilitic/incl.
32							euhedral	0.1-0.3mm	interstitial																subhedral	0.01mm	exsol. Lamell. In Ilm.	subhed/anhed	0.01-3mm	interst./incl.
33							euhedral	0.1-0.2mm	interst./incl																anhedral	1-2mm oikos	poikilitic	anhedral	0.01-2mm	interst./incl.
34																									anhedral	0.01-0.2mm	incl in Ilm/second. Alt	anhedral	0.01-3mm oiko	subpoikilitic/incl.
35							euhedral	0.5-2mm	incl. within Cpx/Biot																anhedral	0.1mm	incl in Ilm/second. Alt	anhedral	0.5-1mm	interst./incl.
35(a)																									anhedral	0.1-1mm	second./alt. Olivine	anhedral	1-4mm oikos	subpoik./interst./incl.
36																			anhedral	0.3mm	sauss. alt. Plag.				anhedral	0.01-0.8mm	incl in Ilm/second. Alt	anhed/subhed	0.01-0.8mm	interst./incl.
37																									subhed/anhed	0.01-0.05mm	exsol. Ilm/sympl. Opx	anhedral	0.1-5mm	interst./incl.
38																									anhedral	0.01mm	exsol. lamell. In Ilm.	anhedral	1cm oikos	poikilitic/interst.
39							subhedral	0.1-0.3mm	interst./incl																subhedral	0.01-0.5mm	interst./incl.	anhed/subhed	0.01-1mm	interst./incl./sympl.
40																									anhedral	0.1-2mm	incl. in Ilm.	anhedral	0.01-2.2mm	interst./incl.
41																									anhedral	0.01-1mm	incl in Ilm/second. Alt	anhed/subhed	0.01-1.5mm	interst./incl.
42							euhedral	0.5-1mm	interst./incl Ol.																anhed/subhed	0.01-0.05mm	exsol. Ilm/sympl. Opx	anhedral	0.5-1mm	interst./incl.
43																			anhedral	0.01-0.1mm	sauss. alt. Plag.				anhedral	0.01-1.8mm	incl in Ilm/second. Alt	anhedral	0.01-1mm	interst./incl.
44							euhedral	0.1-0.5mm	interstitial																anhedral	0.01-0.04mm	interst./incl. in Ilm.	subhed/anhed	0.01-1.5mm	interstitial
45																									anhedral	0.01-3mm	interst./sympl. Opx	subhedral	0.01-0.5mm	interst./incl.
46							euhedral	0.1-0.5mm	interst./incl Cpx/Biot																anhedral	0.01-1mm	interst./incl. in Ilm.	anhedral	0.01-2.5mm	interst./incl. in Cpx
47	anhedral	<0.01mm	alt. of Cpx																						anhedral	0.01-1.5mm	interst./sympl. Opx	anhedral	0.01-1.2mm	interst./incl.
48																									anhedral	0.15mm	incl in Ilm/second. Alt	anhed/subhed	2-3mm oikos	poikilitic/interst.
49										euhedral	0.05-0.1mm	clusters incl. in Plag.													anhedral	0.05-0.1mm	interst./incl.	anhedral	0.1-1mm	interst./incl.
50																												anhed/subhed	0.01-0.4mm	interst./incl.
51							euhedral	0.1mm	interstitial																anhedral	0.02mm	interst./incl. in Ilm.	anhed/subhed	0.01-2.5mm	interst./incl.
52							euhedral	0.1mm	interst./incl																anhed/subhed	0.01-1mm	incl in Ilm/second. Alt	anhedral	0.01-1mm	interstitial
53																									subhedral	0.01-0.05mm	exsol. Lamell. In Ilm.	anhedral	0.1-1mm	interst./incl.
53(a)																									anhedral	0.01-0.35mm	interst./incl./sympl.	anhedral	0.01-1.5mm	interst./incl.
54							euhedral	0.05-0.15mm	interst./incl																anhedral	0.01-1.8mm	incl in Ilm/second. Alt	anhedral	0.01-1.5mm	interst./incl. in Cpx
55																									anhedral	0.01-1mm	incl in Ilm/second. Alt	subhed/anhed	0.01-2mm	interst./exsol. in Mag
56																anhedral	0.01-0.1mm	granoblastic							subhedral	0.01-0.2mm	random			
57										euhedral	0.01-0.1mm	random				subhedral	0.05-0.1mm	granoblastic												
58																subhedral	0.1-0.2mm	granoblastic												

#	Oxides/Sulfides																														
	Pyrite			Pyrrhotite			Chalcopyrite			Bornite			Digenite			Chalcocite			Covellite			Cubanite			Pentlandite						
	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture				
1				anhedral	0.01-0.2mm	incl. in Cp/alt. of Ol.	anhedral	0.01-0.1mm	interst./incl./microvein.																subhedral	0.01-0.1mm	incl. in Cp				
2				anhedral	0.01-0.15mm	interst./incl.	anhedral	0.01-0.1mm	interst./incl.	anhedral	0.01-0.02mm	interst./incl./exsol. Cp										subhedral	0.01-0.025mm	laths within CP	subhed/anhed	0.01-0.05mm	interst./incl.				
3				anhedral	0.04-0.2mm	interst./incl.	anhedral	0.01-0.2mm	interst./incl.	anhedral	0.01-0.15mm	exsol. Lamell. In Cp													subhedral	0.01-0.1mm	interst./incl. in Cp				
4	euhedral	<0.01mm	second. alt.	anhedral	0.01-5mm	interst./incl.	anhedral	0.01-0.7mm	interst./incl. in Po																subhedral	0.8mm	interst./incl. in Cp				
5				anhedral	0.01-0.02mm	interstitial/random	anhedral	0.01-0.02mm	interstitial/random																						
6				anhedral	0.01-0.2mm	interstitial	anhedral	0.01-0.1mm	interst./incl.																anhedral	<0.01mm	flame struct. In Po				
7				anhedral	0.3-0.4mm	poikiloblastic	anhedral	0.01-0.05mm	incl. in Po																						
8				anhedral	0.01-0.5mm	incl./interst.	anhedral	0.01-0.1mm	incl. in Po																						
9				anhedral	0.01-0.8mm	incl./interst./granobl.	anhedral	0.01-0.1mm	incl. in Po																						
9(a)				anhedral	0.01-0.6mm	granoblastic	anhedral	0.01-0.1mm	incl. in Po																subhedral	0.01-0.12mm	laths within Cp	anhedral	0.01-0.08mm	flame struct. In Po	
10				anhedral	0.01-0.02mm	incl. in alt. of Ol.	anhedral	0.01-0.6mm	interst./incl./microvein.							subhedral	<0.01mm	incl. in alt. of Ol.								subhedral	0.01-0.15mm	incl. in Cp			
10(a)				anhedral	0.01-0.8mm	interst./incl.	anhedral	0.01-0.2mm	interst./incl./microvein.	anhedral	0.01-0.05mm	exsol. Lamell. In Cp										subhedral	0.01-0.4mm	laths within Cp	subhedral	0.01-0.2mm	incl. in Po/Cp				
10(b)				anhedral	0.01-0.6mm	incl./interst.	anhed/subhed	0.01-0.7mm	interst./incl./microvein.																subhedral	0.1mm	interst./incl. in Cp				
10(c)				anhedral	0.01-0.4mm	incl. in alt. of Ol.	anhedral	0.01-0.25mm	interst./incl. in Po																anhedral	0.01-0.1mm	incl. in Po				
11				anhedral	0.01-1.5mm	interstitial	anhedral	0.01-0.3mm	interst./incl./microvein.														subhedral	0.01-0.1mm	laths within Cp	subhedral	0.01-0.2mm	interst./incl. in Cp			
12				anhedral	0.01-0.4mm	interst./incl.	anhedral	0.01-0.15mm	interst/microvein/sympl	anhedral	<0.01mm	exsol. Lamell. In Cp										subhedral	0.01-0.3mm	laths within Cp/Po	subhedral	0.01-0.1mm	interst./incl. in Cp/Po				
13				anhedral	0.01-0.15mm	interst./incl.	anhedral	0.01-0.1mm	interst./incl.														subhedral	0.01-0.05mm	laths within Cp	anhedral	0.01-0.1mm	incl. in Po/Cp			
14				anhedral	0.01-0.2mm	interst./incl.	anhedral	0.01-0.1mm	interst./incl.														subhedral	0.01-0.1mm	laths within Cp	subhedral	0.01-0.04mm	interst./incl. in Cp/Cb			
15				anhedral	0.01-0.1mm	interst./incl.	anhedral	0.01-0.05mm	interst./incl. in Po	anhedral	<0.01mm	exsol. in Cp/Tal													subhedral	0.01-0.05mm	incl. in Po				
16				anhedral	0.01-0.3mm	interst./incl.	anhedral	0.01-0.2mm	interst./incl.																subhedral	0.01-0.05mm	interst./incl. in Cp/Po				
17				anhedral	0.01-0.3mm	interst./incl. in alt. Ol.	anhedral	0.01-0.15mm	interst./incl. in Po														subhedral	0.01-0.1mm	laths within Cp						
18				anhedral	0.01-4mm	interst./incl. in alt. Ol.	subhedral	0.01-0.5mm	interst./incl. in Po							subhedral	0.1-0.2mm	incl. in Po, Cp & Cb					subhedral	0.1-0.3mm	laths within Cp	subhedral	0.1-1mm	interst./incl. in Cp			
19							anhedral	<0.01-0.6mm	interst./incl.																anhedral	0.01-0.5mm	interst./incl. in Cp				
20				anhedral	0.01-2.2mm	interst./incl.	anhedral	0.01-1mm	interst/microvein/sympl	anhedral	0.01-0.05mm	exsol. Lamell. In Cp										anhed/subhed	0.01-3mm	laths in Cp/microvein.	subhedral	0.01-3mm	incl. in Po/Cp				
21				anhedral	0.01-0.2mm	interst./incl.	subhed/anhed	0.01-0.2mm	interst./incl./microvein.																anhedral	0.01mm	incl. in Cp				
22				anhedral	0.01-0.35mm	interst./incl.	anhedral	0.01-0.05mm	incl. in Po														anhedral	0.01-0.1mm	laths within Cp	anhedral	0.01-0.1mm	incl. in Po			
23				anhedral	0.01-0.1mm	interst./incl. in Cp	anhedral	0.01-0.3mm	interst./incl./microvein.	anhedral	0.01-0.1mm	exsol. Lamell. In Cp													anhedral	0.01-0.06mm	incl. in Cp				
24				anhedral	0.01-0.25mm	interstitial	anhedral	0.01-0.15mm	interst./incl. in Po	anhedral	0.01-0.05mm	exsol. Lamell. In Cp							anhedral	0.02mm	Lamell. In Cp	subhedral	0.01-0.16mm	laths within Cp	subhedral	0.01-0.1mm	incl. in Cp/Po				
25				anhedral	0.3mm	interstitial	anhedral	0.01-0.5mm	interst./incl.	subhedral	0.01-0.05mm	exsol. Lamell. In Cp													subhedral	0.1-0.2mm	interst./incl. in Cp				
26				anhedral	0.1-0.6mm	interstitial	anhedral	0.01-0.6mm	interst./incl. in Po														subhedral	0.1-0.6mm	laths within Cp	subhedral	0.1-0.3mm	interst./incl. in Po/Cp			
27				anhedral	0.01-0.2mm	interst./incl.	anhedral	0.01-0.2mm	interst./incl./microvein.	anhedral	0.02mm	exsol. Lamell. In Cp													subhedral	0.01-0.1mm	laths within Cp	subhedral	0.01-0.1mm	interst./incl. in Cp/Po		
28				anhedral	0.01-0.15mm	interst./incl. in Cp	anhedral	0.01-0.2mm	interst./incl./microvein.	anhedral	0.01-0.03mm	exsol. Lamell. In Cp							anhedral	0.01-0.02mm	incl. in Bn/Cp/silic.	subhedral	0.01-0.2mm	laths within Cp	subhedral	0.01-0.5mm	interst./incl. in Po/Cp				
29				anhedral	0.01-0.2mm	interst./incl.	anhedral	0.01-0.18mm	interst./incl./microvein.	subhedral	<0.01mm	exsol. Lamell. In Cp							anhedral	<0.01mm	incl. in Cp/silicates	subhed/anhed	0.01-0.2mm	laths within Cp	anhedral	0.01-0.1mm	incl. in Po/Cp				
30				anhedral	0.01-0.3mm	interst./incl.	subhed/anhed	0.01-0.2mm	interst./incl.														subhedral	0.01-0.2mm	laths within Cp	subhedral	0.01-0.05mm	interst./incl. in Cp/Po			
31				anhedral	0.01-1mm	interst./incl. in alt. Ol.	anhedral	0.01-0.5mm	interst./incl./microvein.	anhedral	0.01-0.02mm	exsol. Lamell. In Cp							anhedral	0.01-0.02mm	incl. in Cp/silicates	subhedral	0.01-1mm	laths within Cp	anhed/subhed	0.01-1.8mm	incl. in Po/Cp				
32				anhedral	0.01-1mm	interstitial	anhedral	0.01-0.1mm	interst./incl./microvein.																subhedral	0.1-0.3mm	interst./incl. Po/Cp				
33							anhedral	<0.01-0.3mm	interst./incl./microvein.														subhedral	0.01mm	laths within Cp	subhedral	0.01mm	incl. in Cp				
34				anhedral	0.01-0.4mm	interstitial	anhedral	0.01-0.2mm	interst./incl./sympl. Ol.	anhedral	0.01-0.03mm	exsol. Lamell. In Cp	anhedral	0.01-0.05mm	Incl. in Plag./alt. Cov.				anhedral	0.01-0.02mm	incl. in Plag.	subhedral	0.01-0.1mm	laths within Cp	anhedral	0.01-0.2mm	incl. in Po				
35				anhedral	0.01-0.2mm	interst./incl.	subhed/anhed	0.01-0.05mm	interst./incl.	anhedral	0.01mm	exsol. Lamell. In Cp										subhedral	0.1-0.05mm	laths within Cp	subhedral	0.1mm	interst./incl. in Cp/Po				
35(a)				anhedral	0.01-0.15mm	interst./incl.	anhedral	0.01-0.2mm	interst./incl./microvein.														anhedral	0.01-0.1mm	laths within Cp	anhedral	0.01-0.1mm	incl. in Po/Cp			
36				anhedral	0.01-0.25mm	interst./incl. in alt. Ol.	anhedral	0.01-0.2mm	interst./incl.	anhedral	0.01-0.02mm	exsol. Lamell. In Cp													subhedral	0.01-0.25mm	incl. in Po/Cp				
37				anhedral	0.01-0.3mm	interst./incl.	anhedral	0.01-0.3mm	interst./incl.																subhedral	0.01-0.04mm	incl. in Cp				
38				anhedral	0.01-0.2mm	interstitial	anhedral	0.01-0.2mm	interst./incl./microvein.	subhedral	0.01-0.02mm	exsol. Lamell. In Cp													subhedral	0.01mm	interst./incl. in Cp				
39				anhedral	0.01mm	interst./incl.	anhedral	0.01-0.15mm	interst./incl.	subhedral	0.01-0.1mm	exsol. Lamell. In Cp							anhedral	0.01-0.02mm	interst./incl.	subhedral	0.01-0.02mm	laths within Cp	anhedral	0.01mm	incl. in Cp				
40				anhedral	0.01-0.25mm	interst./incl. in alt. Ol.	anhedral	0.01-0.2mm	interst./incl./microvein.	anhedral	<0.01mm	exsol. Lamell. In Cp										anhedral	0.01-0.05mm	laths within Po/Cp	anhedral	0.01-0.1mm	incl. in Cp/Cb				
41				anhedral	0.01-0.2mm	interst./incl. in alt. Ol.	anhedral	0.01-0.1mm	interst./incl.	anhedral	0.01-0.05mm	exsol. Lamell. In Cp										anhedral	0.01-0.1mm	laths within Cp	anhedral	0.01-0.03mm	incl. in Po/Cp				
42				anhedral	0.1-0.5mm	interst./incl.	anhedral	0.01-0.2mm	interst./incl./microvein.														subhedral	0.1-0.2mm	laths within Cp	subhedral	0.01-0.02mm	incl. in Cp			
43				anhedral	0.01-0.15mm	interst./incl. in alt. Ol.	anhedral	0.01-0.4mm	interst./incl./microvein.	anhedral	0.01-0.02mm	exsol. Lamell. In Cp										subhedral	0.05-0.2mm	laths within Cp	anhedral	0.01-0.04mm	incl. in Cp				
44				anhedral	0.01-0.2mm	interst./incl. in alt. Ol.	anhedral	0.01-0.1mm	interst./incl.																						
45							anhedral	0.01-0.1mm	interst./incl.	anhedral	0.01-0.1mm	exsol. Lamell. In Cp													subhedral	0.01mm	incl. in Cp				
46				anhedral	0.01-0.25mm	interst./incl. in Cp	anhedral	0.01-0.3mm	interst./incl./microvein.	anhedral	<0.01mm	exsol. Lamell. In Cp	anhedral	0.01mm	incl./in-fill fract				anhedral	<0.01mm	incl./in-fill fract	subhedral	0.01-0.5mm	laths within Cp	subhedral	0.01-0.15mm	incl. in Cp/Po				
47				anhedral	<0.01mm	incl. in alt. Ol.	anhedral	0.01-0.2mm	interst./incl.																anhedral	0.01-0.05mm	incl. in Cp				
48				anhedral	0.01-0.3mm	interst./incl./microvein.	anhedral	0.01-0.25mm	interst./incl. in Po	anhedral	<0.01mm	exsol. Lamell. In Cp										subhedral	0.01-0.1mm	laths within Cp	subhedral	0.01-0.1mm	interst./incl. in Cp				
49				anhedral	0.1-0.2mm	interst./incl.	anhedral	0.1-0.2mm	interst./incl.																						
50				anhedral	0.01-0.3mm	interstitial	anhedral	0.01-0.3mm	interst./incl. in Po																anhedral	0.01-0.2mm	flame struct. In Po				
51				anhedral	0.03mm	incl. in Ilm.	anhedral	0.01-0.1mm	incl. in Pn							anhedral	0.05mm	incl. in Cp							subhedral	0.01-3.5mm	interstitial				
52							anhedral	0.01-0.05mm	interst./incl.	anhedral	0.01-0.05mm	exsol. Lamell. In Cp							anhedral	0.01-0.05mm	interst./incl.				subhedral	0.02mm	incl. in Cp				
53				anhedral	0.01-0.1mm	interst./incl.	anhedral	<0.01-0.1mm	interst./incl.														subhedral	0.05mm	laths within Cp	subhed/anhed	0.01-0.1mm	Interst/flame struct Po			
53(a)				anhedral	0.01-0.075mm	incl. in Cp	anhedral	0.01-0.2mm	interst./incl./microvein.	anhedral	0.01-0.1mm	exsol. Lamell. In Cp							anhedral	0.01-0.02mm	incl. in Plag.	subhedral	0.05-0.1mm	laths within Cp	anhedral	0.01-0.05mm	incl. in Cp				
54				anhedral	0.01-0.05mm	interst./incl. in Cp	anhedral	0.01-0.2mm	interst./incl./microvein.	anhedral	0.01-0.15mm	exsol. Lamell. In Cp	anhedral	0.01-0.03mm	in-fill in fract.				anhedral	0.01-0.2mm	in-fill in fract.	subhedral	0.01-0.25mm	laths within Cp	anhed/subhed	0.01-0.1mm	incl. in Cp				
55				anhedral	0.01-0.05mm	interst./incl.	anhedral	0.01-0.1mm	interst./incl./microvein.	subhed/anhed	0.01-0.05mm	exsol. Lamell. In Cp	anhedral	0.01-0.05mm	incl./in-fill fract				anhedral	0.01-0.05mm	incl./in-fill fract				anhed/subhed	0.01-0.02mm	interst./incl. in Cp				
56				anhedral	0.01-0.2mm	granoblastic	anhedral	0.01-0.1mm	incl. in Po																						
57				anhedral	0.01-0.6mm	granoblastic	anhedral	0.01-0.05mm	incl. in Po/Sp																						
58				anhedral	0.01-0.5mm	layered/bedded	anhedral	0.01-0.05mm	incl. in Po/Sp																						

#	Oxides/Sulfides																														
	Violarite			PGMs			Silver			Mackinawite/Valleriite			Sphalerite			Galena			Millerite			Enargite			Talnakhite						
	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture	crystal shape	grain size	texture			
1	anhedral	<0.01mm	oxidation of Pn	euhed./anhed.	0.02mm	incl. in Plag							anhedral	<0.01mm	exsol. lamell. in Cp	euhedral	0.025mm	incl. in Plag												
2																														
3	anhedral	<0.01mm	oxidation of Pn							anhedral	<0.01mm	incl. in Pn	subhedral	<0.01mm	exsol. lamell. in Cp															
4													anhedral	0.01-0.02mm	exsol. lamell. in Cp															
5													anhedral	0.01-0.4mm	porphyroblastic	subhedral	0.01-0.03mm	interstitial/random											
6													anhedral	0.01-0.04mm	exsol. lamell. in Cp															
7													anhedral	0.01-0.02mm	interstitial	subhedral	<0.01mm	interstitial/incl in Po											
8																														
9																														
9(a)													anhedral	0.01-0.1mm	granoblastic															
10	anhedral	<0.01mm	oxidation of Pn										anhedral	<0.01mm	exsol. lamell. in Cp															
10(a)													anhedral	0.01-0.1mm	exsol. lamell. in Cp															
10(b)										subhedral	0.01mm	incl. in Po/alt. of Ol.	subhedral	<0.01mm	exsol. lamell. in Cp															
10(c)													anhedral	<0.01mm	exsol. lamell. in Cp															
11	anhedral	<0.01mm	oxidation of Pn							anhedral	0.01-0.02mm	incl. in Pn,Cp & Cb																		
12													anhedral	<0.01mm	exsol. lamell. in Cp															
13										anhedral	<0.01mm	blades in Po/Pn/Cp	anhed/subhed	0.01-0.15mm	exsol. lamell. in Cp															
14																														
15													anhedral	0.01mm	exsol. lamell. in Cp													anhedral	0.01-0.02mm	interst./incl.
16	anhedral	<0.01mm	oxidation of Pn							anhedral	0.01-0.02mm	incl. in alt. of Ol.																		
17													anhedral	0.01-0.04mm	exsol in Cp/incl in alt	subhedral	0.01mm	incl. in alt.											
18																						anhedral	0.1mm	incl. in Po, Cp & Cb						
19													anhedral	<0.01mm	exsol. lamell. in Cp	euhedral	<0.01-0.04mm	incl. in alt. Plag.												
20				euhedral	<0.01mm	incl. in olivine				anhedral	<0.01mm	blades in Pn	anhedral	0.01-0.05mm	exsol. in Cp/Cb															
21										subhedral	<0.01mm	incl. in Po/alt. of Ol.																		
22										anhedral	0.01-0.02mm	aggreg. in Po	anhedral	<0.01mm	exsol. lamell. in Cp															
23				anhedral	<0.01mm	incl. in Cp							anhed/subhed	<0.01mm	exsol. lamell. in Cp															
24													anhedral	<0.01mm	exsol. lamell. In Cp															
25										anhedral	0.01mm	incl. in Po																		
26																														
27	anhedral	<0.01mm	oxidation of Pn										subhedral	<0.01mm	exsol. lamell. In Cp															
28							anhedral	0.02mm	vein in-fill																						
29	anhedral	<0.01mm	oxidation of Pn										anhedral	<0.01mm	exsol. lamell. in Cp															
30													subhedral	<0.01mm	exsol. lamell. in Cp													subhedral	0.1mm	incl. in silicates
31										anhedral	0.01-0.04mm	aggreg. in Po/Pn	anhedral	<0.01mm	exsol. lamell. in Cp													anhedral	0.01-0.2mm	incl. in silicates
32										subhedral	0.01-0.05mm	incl. in Po/alt. fract.																		
33																														
34																														
35																														
35(a)													anhedral	<0.01mm	exsol. lamell. in Cp															
36																														
37										subhedral	0.01mm	radial blades	subhed/anhed	<0.01mm	exsol. lamell. in Cp															
38										subhedral	0.01-0.05mm	incl. in alt. fract. Ol.	anhedral	0.01mm	exsol. lamell. in Cp															
39																														
40																														
41																														
42										subhedral	0.01-0.02mm	blade-like fringe on Po																		
43													anhedral	<0.01mm	exsol. lamell. in Cp															
44										anhedral	0.01mm	incl. in Po/alt. fract.																		
45																														
46										anhedral	<0.01mm	incl. in Pn	anhedral	<0.01mm	exsol. lamell. in Cp													anhedral	0.02-0.2mm	interst./incl.
47				subhedral	0.02mm	incl. in alt. Ol.																								
48										anhedral	<0.01mm	incl. in alt. fract. Ol.	anhedral	<0.01mm	exsol. lamell. in Cp															
49										subhedral	0.01-0.1mm	second./incl. alt. Ol.																		
50													anhedral	0.01-0.2mm	incl. in Cp/Po															
51	subhedral	0.01-0.02mm	oxidation of Pn										anhedral	0.01mm	exsol. lamell. in Cp															
52																														
53	anhedral	<0.01mm	oxidation of Pn							subhedral	<0.01-0.05mm	incl. in alt. fract. Ol.	anhedral	<0.01mm	exsol. lamell. in Cp															
53(a)																														
54													anhedral	0.01-0.15mm	exsol. Lamell. In Cp															
55				anhedral	<0.01mm	incl. in Cp																								
56													anhedral	0.01-0.1mm	random															
57													anhedral	0.01-0.1mm	random	anhedral	0.05-0.4mm	random											
58													anhedral	0.01-0.5mm	layered/bedded															

Abbreviations used in Petrographic Database spreadsheet

Abbreviation	
adcum	adcumulate
aggreg	aggregates
alt	alteration
anhed	anhedral crystal shape
Biot	biotite
Bn	bornite
brn	brown
Cb	cubanite
cm	centimeter
Cp	chalcopyrite
Cpx	clinopyroxene
Cv	covellite
euhed	euhedral crystal shape
exsol	exsolution
f-gr	fine-grained
fract	fractures
granobl	granoblastic
Ilm	ilmenite
incl	inclusion
intercum	intercumulate
interst	interstitial
invert	inverted
lamell	lamellae
Mag	magnetite
mesocum	mesocumulate
microfract	microfractures
microvein	microveining
mm	millimeter
oiko	oikocryst
Ol	olivine
Opx	orthopyroxene
orthocum	orthocumulate
Ox	oxide
Pig	pigeonite
Plag	plagioclase
Po	pyrrhotite
poik	poikilitic
porphyrobl	porphyroblastic
prism	prismatic
rem	remnant
sauss	saussuritization
second	secondary
ser	sericitic alteration
silic	silicates
Sp	sphalerite
struct	structure
subhed	subhedral crystal shape
subpoik	subpoikilitic
Sulf	sulfide
sympl	symplectic intergrowth texture
wk	weak

Drill Hole Number	From	To	Polymet Petrographic Footage	Vancouver Petrographic Number
99-320C	830	850	833	Z-131
99-320C	830	850	841	Z-132
00-361C	310	320	311	Z-75
00-361C	310	320	316	Z-76
00-361C	345	350	347	Z-77
00-361C	345	350	349	Z-78
00-343C	240	250	241	Z-59
00-343C	240	250	249	Z-60
26030	1047	1052	1048	Z-7
26030	1047	1052	1050	Z-8
26061	1218	1233	1220	Z-17
26061	1218	1233	1231	Z-18
00-340C	990	995	991	Z-57
00-340C	990	995	994	Z-58
00-340C	965	974.5	966	Z-55
00-340C	965	974.5	972	Z-56
26043	1501	1506	1502	Z-9
26043	1501	1506	1505	Z-10
26027	740	745	741	Z-3
26027	740	745	744	Z-4
00-368C	565	570	566	Z-105
00-368C	565	570	569	Z-106
26098	247	255	248	Z-27
26098	247	255	254	Z-28
26029	815	825	816	Z-5
26029	815	825	824	Z-6
26017	433	443	435	Z-1
26017	433	443	442	Z-2
00-340C	595	615	596	Z-51
00-340C	595	615	613	Z-52
00-334C	580	600	589	Z-43
00-334C	580	600	598	Z-44
00-334C	640	660	642	Z-45
00-334C	640	660	657	Z-46
00-347C	795	815	798	Z-65
00-347C	795	815	812	Z-66
00-350C	580	600	582	Z-67
00-350C	580	600	591	Z-68
00-327C	225	245	226	Z-37
00-327C	225	245	242	Z-38
00-371C	435	440	436	Z-115
00-371C	435	440	438	Z-116
00-340C	765	780	771	Z-53
00-340C	765	780	778	Z-54
00-357C	335	340	337	Z-71
00-357C	335	340	338	Z-72
00-326C	680	685	680	Z-35
00-326C	680	685	684	Z-36
00-357C	535	540	536	Z-73
00-357C	535	540	539	Z-74
99-318C	725	735	728	Z-125
99-318C	725	735	735	Z-126
99-317C	460	470	462	Z-121
99-317C	460	470	465	Z-122
00-366C	185	205	187	Z-85
00-366C	185	205	204	Z-86
00-366C	230	240	231	Z-87
00-366C	230	240	239	Z-88
99-320C	165	175	167	Z-127
99-320C	165	175	172	Z-128
99-318C	250	270	257	Z-123
99-318C	250	270	267	Z-124
00-373C	95	115	99	Z-119
00-373C	95	115	114	Z-120
00-373C	75	95	78	Z-117
00-373C	75	95	93	Z-118
00-357C	110	130	112	Z-69
00-357C	110	130	126	Z-70
99-320C	315	330	319	Z-129
99-320C	315	330	328	Z-130
00-369C	335	345	336	Z-111
00-369C	335	345	341	Z-112
00-368C	460	465	462	Z-103
00-368C	460	465	463	Z-184
26055	940	945	941	Z-11
26055	940	945	944	Z-12
26098	145	148.5	146	Z-25
26098	145	148.5	148	Z-26
00-337C	105	110	106	Z-47
00-337C	105	110	108	Z-48
00-334C	30	50	31	Z-39
00-334C	30	50	50	Z-40
00-368C	125	145	132	Z-101
00-368C	125	145	143	Z-102
00-368C	20	40	28	Z-99
00-368C	20	40	39	Z-100
00-366C	35	55	37	Z-83
00-366C	35	55	47	Z-84
00-334C	110	130	112	Z-41
00-334C	110	130	128	Z-42
00-347C	155	175	157	Z-61
00-347C	155	175	174	Z-62
00-347C	280	300	282	Z-63
00-347C	280	300	299	Z-64
00-326C	60	70	61	Z-33
00-326C	60	70	66	Z-34
00-369C	305	325	306	Z-109
00-369C	305	325	318	Z-110
00-367C	50	65	55	Z-89
00-367C	50	65	62	Z-90
00-367C	260	280	269	Z-93
00-367C	260	280	279	Z-94
00-367C	290	310	297	Z-95
00-367C	290	310	308	Z-96
00-370C	20	30	22	Z-113
00-370C	20	30	27	Z-114
00-369C	20	30	22	Z-107
00-369C	20	30	29	Z-108
00-367C	170	175	171	Z-91
00-367C	170	175	173	Z-92
00-367C	395	400	395	Z-97
00-367C	395	400	398	Z-98
26064	44	54	45	Z-19
26064	44	54	53	Z-20
26064	264	269	264	Z-23
26064	264	269	268	Z-24
26064	146	156	148	Z-21
26064	146	156	151	Z-22
26056	110	125	113	Z-13
26056	110	125	123	Z-14
26056	135	153	138	Z-15
26056	135	153	152	Z-16
00-361C	737	749	739	Z-79
00-361C	737	749	748	Z-80
00-364C	210	229	211	Z-81
00-364C	210	229	228	Z-82
00-337C	510	520	511	Z-49
00-337C	510	520	519	Z-50

Two slides

Drill Hole Number	From	To	Polymet Petrographic Footage	Vancouver Petrographic Number	
26027	616	626	620	ON-3	
26027	616	626	624	ON-4	
26062	993	998	994	ON-11	
26062	993	998	996	ON-12	
26026	565	568	566	ON-1	
26026	565	568	567	ON-2	
00-340C	725	745	726	ON-19	
00-340C	725	745	744	ON-20	
00-344C	515	520	516	ON-21	
00-344C	515	520	518	ON-22	
00-340C	380	390	381	ON-17	
00-340C	380	390	389	ON-18	
00-361C	240	245	241	ON-23	
00-361C	240	245	244	ON-24	
26049	358	362	359	ON-7	
26049	358	362	361	ON-8	
26030	291	296	292	ON-5	
26030	291	296	295	ON-6	
00-367C	400	405	401	ON-25	
00-367C	400	405	404	ON-26	
26056	302	312	306	ON-9	
26056	302	312	311	ON-10	
26142	360	365	361	ON-15	
26142	360	365	364	ON-16	
26142	345	350	346	ON-13	Two slides
26142	345	350	349	ON-14	

99-320C	400	405	401	PM-1	
99-320C	400	405	404	PM-2	
00-331C	255	260	256	PM-3	
00-331C	255	260	259	PM-4	
26058	704	715	706	PM-5	
26058	704	715	712	PM-6	
00-326C	250	265	251	PM-7	
00-326C	250	265	261	PM-8	
00-340C	910	925	912	PM-9	
00-340C	910	925	922	PM-10	
00-331C	190	210	194	PM-11	
00-331C	190	210	206	PM-12	
00-326C	495	505	496	PM-13	
00-326C	495	505	504	PM-14	
00-344C	630	635	631	PM-15	
00-344C	630	635	634	PM-16	
00-330C	275	280	276	PM-17	
00-330C	275	280	279	PM-18	
99-318C	325	330	326	PM-19	
99-318C	325	330	329	PM-20	
00-326C	225	235	226	PM-21	
00-326C	225	235	234	PM-22	
26039	310	315	311	PM-23	
26039	310	315	314	PM-24	
00-367C	495	500	496	PM-25	
00-367C	495	500	499	PM-26	
26056	282	292	283	PM-27	
26056	282	292	291	PM-28	

Appendix D
Microprobe Results

Silicates: Biotite

Thin Section:	SiO2	Al2O3	TiO2	FeO	MnO	MgO	K2O	CaO	Na2O	NiO	CuO	CoO
00-347C-798	36.973	13.880	7.750	12.870	0.047	14.350	9.410	0.011	0.143	0.067	0.031	0.031
00-347C-798	37.072	13.840	7.630	11.530	0.053	15.280	9.640	0.048	0.225	0.072	0.031	0.027
00-347C-798	36.410	13.810	7.600	12.790	0.072	14.310	9.350	0.011	0.165	0.082	0.031	0.038
00-340C-778	36.904	13.850	6.720	13.050	0.050	14.430	9.300	0.017	0.278	0.031	0.031	0.028
00-340C-778	37.427	14.250	6.050	12.720	0.025	14.980	8.650	0.011	0.703	0.020	0.031	0.029
00-340C-778	36.647	14.320	6.460	13.830	0.047	13.740	9.220	0.023	0.261	0.020	0.031	0.027
00-340C-596	36.327	14.520	5.480	15.162	0.067	14.720	9.190	0.056	0.124	0.105	0.031	0.027
00-340C-596	38.267	15.570	2.914	8.892	0.073	19.690	9.180	0.011	0.417	0.053	0.031	0.027
00-340C-596	36.445	13.540	7.320	17.034	0.036	12.820	9.410	0.012	0.216	0.072	0.031	0.041
26026-566	37.274	14.790	3.930	15.970	0.025	13.580	8.780	0.011	0.164	0.020	0.031	0.030
26026-566	37.749	13.980	4.840	15.150	0.054	13.610	9.250	0.011	0.224	0.036	0.031	0.027
26026-566	37.430	14.510	3.350	16.290	0.036	13.730	8.710	0.070	0.192	0.041	0.031	0.027
00-344C-518	36.480	14.884	5.340	11.360	0.078	17.910	9.040	0.059	0.273	0.066	0.031	0.027
00-344C-518	37.800	14.657	6.150	9.700	0.025	16.800	9.150	0.011	0.379	0.088	0.031	0.027
00-344C-518	38.050	13.630	5.480	10.260	0.025	17.890	9.260	0.015	0.165	0.113	0.046	0.027
99-317C-462	36.985	14.820	4.180	15.200	0.029	14.780	8.980	0.015	0.492	0.069	0.031	0.027
99-317C-462	36.113	14.520	4.570	18.780	0.063	11.750	8.850	0.011	0.301	0.116	0.031	0.027
99-317C-462	36.436	14.950	4.300	17.410	0.061	12.810	8.890	0.017	0.346	0.075	0.031	0.032
00-367C-401	36.420	13.843	6.910	17.730	0.057	11.030	9.410	0.037	0.192	0.020	0.031	0.027
00-367C-401	36.840	14.163	6.240	16.420	0.027	12.100	9.280	0.011	0.133	0.020	0.031	0.027
00-367C-401	36.540	14.173	5.990	17.650	0.025	11.890	9.680	0.011	0.136	0.020	0.031	0.029
00-367C-398	37.170	14.730	6.140	11.910	0.045	15.970	9.120	0.011	0.759	0.020	0.031	0.027
00-367C-398	37.880	14.920	5.100	11.340	0.048	16.870	8.270	0.027	0.787	0.029	0.031	0.027
00-367C-398	37.840	15.120	5.770	11.930	0.089	15.820	8.490	0.011	0.698	0.020	0.031	0.027
00-340C-389	37.084	13.515	6.500	15.000	0.032	12.670	9.750	0.011	0.151	0.077	0.038	0.027
00-340C-389	37.405	13.872	5.160	13.290	0.054	14.500	9.680	0.011	0.221	0.130	0.031	0.035
00-340C-389	37.872	13.576	5.640	12.270	0.059	15.490	9.090	0.011	0.217	0.086	0.031	0.027
26142-364	38.493	14.137	4.730	11.240	0.045	17.360	7.720	0.044	0.804	0.096	0.031	0.040
26142-364	37.975	13.872	4.760	10.860	0.054	17.150	9.060	0.044	0.256	0.079	0.031	0.027
26142-364	37.865	13.719	5.620	12.100	0.070	16.560	7.970	0.069	0.729	0.058	0.031	0.041
26049-361	37.413	12.730	6.720	17.190	0.025	11.100	9.160	0.013	0.089	0.080	0.031	0.037
26049-361	37.675	12.862	7.030	15.810	0.074	11.990	9.200	0.071	0.084	0.086	0.031	0.030
26049-361	37.099	13.250	6.440	17.390	0.094	11.200	9.220	0.028	0.090	0.067	0.031	0.027
26056-311	38.140	14.710	4.960	10.100	0.073	18.020	9.560	0.057	0.090	0.109	0.031	0.029
26056-311	38.588	14.657	3.900	8.650	0.084	18.850	9.390	0.075	0.186	0.067	0.031	0.027
26056-311	37.864	14.500	5.230	9.740	0.051	17.800	9.240	0.025	0.240	0.115	0.031	0.027
26056-311	38.354	14.188	4.710	8.860	0.025	17.570	9.750	0.023	0.077	0.109	0.031	0.027
26056-311	38.243	14.430	5.480	9.060	0.025	17.420	9.410	0.021	0.284	0.133	0.031	0.027
26056-311	39.067	14.840	4.200	8.060	0.025	18.950	9.140	0.055	0.121	0.075	0.031	0.027
26056-311	37.430	14.470	5.680	10.550	0.025	16.260	8.950	0.011	0.179	0.125	0.031	0.028
26056-311	38.223	14.260	4.670	9.560	0.025	17.490	9.120	0.062	0.154	0.043	0.031	0.027
26056-311	38.229	14.880	4.490	9.890	0.050	18.200	9.930	0.022	0.040	0.095	0.031	0.027
00-347C-299	37.565	14.160	5.990	11.290	0.036	16.380	9.580	0.011	0.135	0.066	0.031	0.027
00-347C-299	37.674	13.640	5.140	13.190	0.050	15.660	9.910	0.011	0.027	0.066	0.031	0.039
00-347C-299	37.249	14.300	5.510	10.790	0.025	16.900	9.470	0.011	0.075	0.055	0.031	0.027
99-318C-267	37.957	13.790	4.620	10.730	0.025	18.520	8.740	0.033	0.153	0.075	0.031	0.027
99-318C-267	37.798	14.350	5.050	10.190	0.025	17.610	9.630	0.011	0.102	0.108	0.031	0.027
26064-264	37.323	14.690	4.010	16.401	0.036	14.140	9.120	0.050	0.075	0.086	0.031	0.027
26064-264	38.254	15.520	2.780	7.382	0.053	20.490	8.980	0.060	0.157	0.072	0.031	0.028
26064-264	37.303	15.000	4.110	13.356	0.025	15.790	10.040	0.032	0.013	0.074	0.031	0.027
00-370C-22	37.481	13.780	6.280	12.640	0.048	15.370	9.060	0.062	0.246	0.073	0.031	0.049
00-370C-22	38.471	13.620	5.240	12.000	0.025	16.700	9.000	0.011	0.278	0.076	0.031	0.034
00-370C-22	38.521	13.240	5.750	11.670	0.061	16.660	9.570	0.016	0.201	0.027	0.031	0.033
00-331C-194	36.410	14.320	4.950	14.930	0.025	14.150	9.030	0.017	0.411	0.020	0.031	0.029
00-331C-194	37.712	14.580	4.450	15.120	0.061	13.350	8.660	0.041	0.278	0.023	0.031	0.037
00-331C-194	36.868	13.560	5.180	17.530	0.029	12.160	9.100	0.016	0.278	0.067	0.031	0.027
99-320C-172	36.209	13.870	6.650	17.150	0.072	12.080	9.810	0.032	0.034	0.086	0.031	0.027
99-320C-172	36.563	13.540	6.490	15.640	0.081	13.150	9.270	0.040	0.088	0.077	0.031	0.027
99-320C-172	36.593	13.550	6.240	16.060	0.025	12.850	9.110	0.032	0.121	0.117	0.031	0.027
26098-148	37.481	14.090	7.350	12.023	0.064	15.450	9.340	0.017	0.292	0.096	0.031	0.052
26098-148	37.333	13.260	6.460	12.663	0.092	16.650	8.920	0.055	0.234	0.073	0.031	0.027
26098-148	37.224	14.400	7.010	10.605	0.025	16.530	9.480	0.011	0.164	0.069	0.031	0.027
26056-138	37.519	14.940	4.540	10.347	0.025	17.710	9.180	0.011	0.300	0.082	0.031	0.027
26056-138	38.060	14.720	5.000	10.454	0.025	18.040	9.360	0.012	0.210	0.162	0.031	0.027
26056-138	37.794	14.680	4.470	10.465	0.025	18.380	9.420	0.011	0.242	0.109	0.031	0.027
00-368C-132	37.840	14.340	6.090	13.030	0.036	15.450	9.590	0.011	0.103	0.090	0.031	0.027
00-368C-132	37.600	13.820	5.530	14.450	0.047	14.680	9.450	0.011	0.065	0.065	0.031	0.027
00-368C-132	37.060	13.370	6.740	16.870	0.042	12.540	9.460	0.011	0.039	0.093	0.031	0.027
00-357C-112	36.572	14.220	6.820	11.340	0.025	16.280	9.680	0.028	0.167	0.054	0.031	0.027
00-357C-112	35.976	13.510	6.720	12.516	0.025	15.790	9.630	0.011	0.132	0.085	0.039	0.043
00-357C-112	36.825	14.210	6.030	11.214	0.025	16.710	9.590	0.011	0.057	0.090	0.031	0.044
Min.	35.976	12.730	2.780	7.382	0.025	11.030	7.720	0.011	0.013	0.020	0.031	0.027
P5	36.369	13.255	3.915	8.876	0.025	11.820	8.570	0.011	0.039	0.020	0.031	0.027
Median	37.430	14.210	5.510	12.516	0.045	15.660	9.240	0.016	0.186	0.074	0.031	0.027
Mean	37.405	14.180	5.540	12.910	0.045	15.431	9.226	0.026	0.232	0.072	0.031	0.030
P95	38.482	14.975	7.335	17.470	0.083	18.685	9.780	0.065	0.716	0.121	0.031	0.042
Max.	39.067	15.570	7.750	18.780	0.094	20.490	10.040	0.075	0.804	0.162	0.046	0.052
D.L.	0.015	0.013	0.020	0.050	0.025	0.012	0.015	0.011	0.013	0.020	0.031	0.027

Silicates: Plagioclase

Thin Section:	SiO2	Al2O3	TiO2	FeO	MnO	MgO	K2O	CaO	Na2O	NiO	CuO	CoO
00347C-798	52.143	30.330	0.065	0.200	0.028	0.010	0.358	12.230	4.150	0.039	0.000	0.000
00347C-798	51.334	30.560	0.064	0.190	0.000	0.016	0.340	13.990	3.990	0.000	0.000	0.023
00347C-798	51.818	30.570	0.115	0.227	0.036	0.039	0.321	12.400	4.150	0.008	0.002	0.023
00-340C-778	56.654	27.150	0.051	0.054	0.017	0.042	0.434	8.660	6.340	0.031	0.000	0.019
00-340C-778	52.824	29.970	0.180	0.102	0.042	0.009	0.230	11.670	4.760	0.000	0.000	0.000
00-340C-778	52.982	29.770	0.054	0.102	0.006	0.021	0.270	12.660	4.710	0.000	0.017	0.000
00-340C-596	55.367	27.850	0.073	0.233	0.028	0.025	0.542	10.430	5.730	0.000	0.000	0.000
00-340C-596	51.693	29.420	0.165	0.267	0.014	0.017	0.400	12.680	4.520	0.024	0.022	0.000
00-340C-596	57.780	25.930	0.019	0.136	0.050	0.013	0.675	8.390	6.810	0.000	0.007	0.001
00-344C-518	50.240	32.270	0.112	0.240	0.006	0.052	0.219	14.560	3.350	0.000	0.000	0.000
00-344C-518	51.660	31.920	0.142	0.123	0.004	0.030	0.230	12.850	3.800	0.000	0.000	0.000
00-344C-518	52.250	30.530	0.211	0.100	0.000	0.030	0.286	13.420	4.090	0.000	0.006	0.000
00-344C-518	51.540	30.890	0.168	0.132	0.000	0.021	0.267	12.570	4.060	0.000	0.007	0.000
99-317C-462	56.722	27.270	0.038	0.201	0.002	0.013	0.364	8.790	5.870	0.000	0.011	0.019
99-317C-462	57.379	26.860	0.090	0.165	0.027	0.004	0.350	8.340	6.420	0.016	0.000	0.010
99-317C-462	55.742	28.050	0.061	0.101	0.016	0.006	0.314	10.160	5.670	0.011	0.007	0.019
00-367C-401	54.550	29.561	0.087	0.136	0.013	0.018	0.389	11.410	5.060	0.019	0.000	0.004
00-367C-401	55.020	29.355	0.093	0.039	0.011	0.005	0.528	10.090	5.020	0.000	0.000	0.004
00-367C-401	53.170	29.685	0.099	0.386	0.000	0.039	0.439	10.650	4.780	0.007	0.000	0.000
00-367C-398	56.910	27.410	0.073	0.254	0.000	0.058	0.257	9.760	6.390	0.004	0.000	0.019
00-367C-398	56.560	27.840	0.040	0.025	0.000	0.002	0.300	9.270	6.170	0.000	0.049	0.008
00-367C-398	55.630	28.380	0.069	0.032	0.023	0.000	0.193	10.890	5.820	0.006	0.000	0.000
00-340C-389	52.560	30.233	0.077	0.140	0.000	0.027	0.385	12.910	4.450	0.000	0.000	0.000
00-340C-389	53.706	29.335	0.132	0.092	0.016	0.028	0.404	10.840	4.830	0.028	0.019	0.000
00-340C-389	52.575	29.600	0.120	0.137	0.030	0.034	0.332	12.390	4.630	0.002	0.000	0.000
26142-364	51.085	30.824	0.187	0.195	0.000	0.028	0.344	13.820	3.880	0.014	0.004	0.031
26142-364	52.458	29.917	0.177	0.224	0.000	0.037	0.377	12.630	4.440	0.000	0.025	0.000
26142-364	50.954	30.692	0.192	0.213	0.005	0.014	0.316	13.510	3.930	0.000	0.034	0.010
26049-361	52.224	30.600	0.134	0.140	0.025	0.026	0.421	13.200	4.160	0.004	0.000	0.000
26049-361	49.560	32.630	0.144	0.082	0.000	0.033	0.243	14.330	2.991	0.000	0.013	0.000
26049-361	51.020	31.273	0.135	0.082	0.000	0.025	0.328	14.280	3.720	0.000	0.015	0.000
00-361C-311	51.056	30.784	0.222	0.218	0.000	0.027	0.361	12.610	3.750	0.000	0.015	0.000
00-361C-311	49.990	32.050	0.023	0.086	0.011	0.024	0.234	15.720	3.210	0.000	0.015	0.010
00-361C-311	50.334	31.691	0.153	0.252	0.016	0.019	0.292	13.780	3.220	0.000	0.002	0.002
00-361C-311	54.633	29.170	0.169	0.140	0.000	0.157	0.326	10.900	4.670	0.015	0.000	0.014
00-361C-311	55.094	29.180	0.160	0.132	0.000	0.032	0.397	11.990	4.770	0.000	0.005	0.000
00-361C-311	50.330	32.040	0.127	0.143	0.000	0.025	0.229	15.330	3.310	0.033	0.026	0.017
00-361C-311	51.356	31.273	0.258	0.325	0.005	0.030	0.337	14.100	3.660	0.000	0.005	0.002
00-361C-311	54.417	29.460	0.157	0.166	0.011	0.024	0.437	11.320	4.510	0.025	0.013	0.000
00-361C-311	49.308	32.120	0.018	0.155	0.000	0.019	0.196	14.880	3.060	0.000	0.000	0.000
00-347C-299	50.159	31.260	0.094	0.254	0.022	0.063	0.313	14.900	3.460	0.000	0.000	0.005
00-347C-299	53.693	28.850	0.046	0.261	0.000	0.039	0.538	10.600	4.920	0.000	0.009	0.008
00-347C-299	53.022	29.520	0.042	0.251	0.000	0.013	0.487	12.630	4.630	0.000	0.000	0.005
99-318C-267	51.856	30.800	0.123	0.288	0.030	0.048	0.393	13.850	3.800	0.022	0.001	0.021
99-318C-267	52.609	29.860	0.060	0.274	0.000	0.036	0.371	12.890	4.350	0.000	0.000	0.000
99-318C-267	49.540	32.630	0.174	0.370	0.005	0.028	0.241	15.090	2.777	0.000	0.004	0.000
26064-264	51.777	30.440	0.246	0.390	0.034	0.018	0.375	13.790	4.020	0.000	0.029	0.016
26064-264	51.728	30.280	0.158	0.269	0.042	0.023	0.370	13.590	4.070	0.000	0.007	0.010
26064-264	52.054	30.320	0.094	0.264	0.031	0.064	0.426	13.490	4.210	0.026	0.000	0.000
00-370C-22	55.409	28.820	0.047	0.116	0.045	0.007	0.516	10.870	5.290	0.011	0.016	0.000
00-370C-22	53.570	28.910	0.050	0.034	0.000	0.014	0.309	12.090	5.260	0.000	0.000	0.000
00-370C-22	55.237	28.750	0.073	0.043	0.003	0.029	0.575	11.640	5.230	0.000	0.017	0.018
00-331C-194	54.506	30.280	0.071	0.163	0.000	0.019	0.118	12.240	4.190	0.000	0.001	0.008
00-331C-194	55.193	27.480	0.107	0.129	0.017	0.000	0.246	9.360	5.640	0.028	0.000	0.000
00-331C-194	52.095	30.150	0.085	0.138	0.007	0.011	0.172	12.500	3.970	0.000	0.000	0.014
99-320C-172	54.855	28.150	0.081	0.245	0.000	0.000	0.534	10.230	5.150	0.002	0.002	0.000
99-320C-172	53.348	29.070	0.092	0.323	0.000	0.031	0.356	12.140	4.710	0.000	0.015	0.000
99-320C-172	53.781	28.980	0.053	0.279	0.007	0.019	0.367	10.380	5.110	0.013	0.013	0.000
99-320C-172	53.244	29.290	0.128	0.319	0.037	0.039	0.463	11.480	4.260	0.000	0.024	0.000
26098-148	56.064	27.760	0.043	0.212	0.022	0.030	0.732	9.480	5.770	0.000	0.001	0.008
26098-148	51.985	30.220	0.081	0.202	0.003	0.025	0.334	13.520	4.240	0.000	0.029	0.013
26098-148	56.222	27.700	0.065	0.216	0.000	0.040	0.800	9.350	5.790	0.000	0.000	0.010
26056-138	53.032	29.370	0.091	0.375	0.000	0.124	0.442	12.360	4.790	0.000	0.000	0.023
26056-138	50.747	31.010	0.121	0.262	0.006	0.021	0.280	14.290	3.670	0.000	0.003	0.000
26056-138	53.259	29.590	0.090	0.276	0.020	0.050	0.450	12.170	4.900	0.000	0.001	0.008
00-368C-132	53.370	29.870	0.047	0.279	0.000	0.011	0.380	13.080	4.480	0.012	0.021	0.005
00-368C-132	54.530	28.930	0.036	0.225	0.003	0.021	0.497	11.370	4.840	0.000	0.009	0.017
00-368C-132	54.580	29.220	0.032	0.222	0.014	0.006	0.478	12.060	5.030	0.018	0.000	0.010
00-357C-112	52.783	30.390	0.112	0.326	0.000	0.030	0.414	13.240	4.130	0.020	0.003	0.000
00-357C-112	54.318	27.430	0.084	0.276	0.000	0.024	0.664	10.070	5.300	0.009	0.000	0.012
00-357C-112	54.217	28.100	0.056	0.273	0.000	0.019	0.446	12.730	4.460	0.006	0.000	0.004
Min.	49.308	25.930	0.018	0.025	0.000	0.000	0.118	8.340	2.777	0.000	0.000	0.000
P5	50.075	27.340	0.034	0.041	0.000	0.003	0.208	9.030	3.215	0.000	0.000	0.000
Median	53.022	29.685	0.091	0.202	0.005	0.025	0.364	12.390	4.510	0.000	0.002	0.002
Mean	53.175	29.688	0.103	0.196	0.011	0.028	0.377	12.139	4.582	0.006	0.007	0.006
P95	56.688	32.085	0.202	0.348	0.039	0.061	0.619	14.890	6.255	0.028	0.027	0.022
Max.	57.780	32.630	0.258	0.390	0.050	0.157	0.800	15.720	6.810	0.039	0.049	0.031
D.L.	0.015	0.013	0.020	0.050	0.025	0.012	0.015	0.011	0.013	0.020	0.031	0.027

Silicates: Olivine

Thin Section:	SiO2	Al2O3	TiO2	FeO	MnO	MgO	K2O	CaO	Na2O	NiO	CuO	CoO
00-347C-798	36.572	14.220	6.820	35.660	0.444	27.230	9.680	0.037	0.167	0.101	0.031	0.077
00-347C-798	36.572	14.220	6.820	36.550	0.495	27.260	9.680	0.050	0.167	0.054	0.031	0.102
00-347C-798	36.572	14.220	6.820	35.150	0.530	27.610	9.680	0.064	0.167	0.100	0.031	0.053
00-340C-778	36.572	14.220	6.820	39.680	0.374	24.350	9.680	0.060	0.167	0.054	0.031	0.048
00-340C-778	36.572	14.220	6.820	39.260	0.415	23.860	9.680	0.036	0.167	0.054	0.031	0.068
00-340C-778	36.572	14.220	6.820	39.880	0.455	24.430	9.680	0.049	0.167	0.054	0.031	0.070
00-340C-778	36.572	14.220	6.820	39.120	0.495	24.850	9.680	0.057	0.167	0.054	0.031	0.066
00-340C-596	36.572	14.220	6.820	40.050	0.561	25.650	9.680	0.029	0.167	0.116	0.031	0.073
00-340C-596	36.572	14.220	6.820	40.050	0.543	26.200	9.680	0.049	0.167	0.168	0.031	0.067
00-340C-596	36.572	14.220	6.820	40.072	0.516	25.730	9.680	0.056	0.167	0.138	0.031	0.060
00-344C-518	37.030	14.220	6.820	30.080	0.325	32.400	9.680	0.028	0.167	0.118	0.031	0.067
00-344C-518	36.572	14.220	6.820	33.120	0.036	30.060	9.680	0.046	0.167	0.112	0.031	0.027
00-344C-518	36.970	14.220	6.820	30.110	0.317	31.460	9.680	0.032	0.167	0.142	0.031	0.055
00-344C-518	36.830	14.220	6.820	29.290	0.356	31.880	9.680	0.082	0.167	0.102	0.031	0.054
99-317C-462	36.572	14.220	6.820	36.210	0.436	27.180	9.680	0.034	0.167	0.135	0.035	0.056
99-317C-462	36.572	14.220	6.820	40.470	0.414	24.690	9.680	0.036	0.167	0.106	0.031	0.066
99-317C-462	36.572	14.220	6.820	48.820	0.580	16.780	9.680	0.069	0.167	0.091	0.031	0.096
00-367C-401	36.572	14.220	6.820	43.730	0.510	20.980	9.680	0.088	0.167	0.054	0.031	0.068
00-367C-401	36.572	14.220	6.820	43.920	0.503	20.050	9.680	0.130	0.167	0.054	0.031	0.059
00-367C-401	36.572	14.220	6.820	45.520	0.519	19.430	9.680	0.065	0.167	0.054	0.031	0.075
00-340C-389	36.572	14.220	6.820	37.440	0.510	25.170	9.680	0.040	0.167	0.102	0.031	0.067
00-340C-389	36.572	14.220	6.820	38.030	0.390	24.840	9.680	0.036	0.167	0.082	0.031	0.068
00-340C-389	36.572	14.220	6.820	39.310	0.445	24.590	9.680	0.051	0.167	0.110	0.031	0.093
26142-364	36.639	14.220	6.820	30.670	0.373	28.700	9.680	0.069	0.167	0.160	0.031	0.038
26142-364	36.675	14.220	6.820	30.880	0.407	30.000	9.680	0.039	0.167	0.115	0.031	0.065
26142-364	36.741	14.220	6.820	31.510	0.358	29.970	9.680	0.056	0.167	0.143	0.031	0.048
26049-361	36.572	14.220	6.820	33.140	0.341	29.380	9.680	0.058	0.167	0.117	0.031	0.057
26049-361	36.572	14.220	6.820	31.380	0.293	29.710	9.680	0.066	0.167	0.080	0.031	0.080
26049-361	36.572	14.220	6.820	30.710	0.262	30.400	9.680	0.031	0.167	0.054	0.031	0.048
00-361C-311	36.572	14.220	6.820	33.370	0.483	29.030	9.680	0.043	0.167	0.150	0.031	0.070
00-361C-311	36.572	14.220	6.820	34.640	0.415	28.120	9.680	0.085	0.167	0.129	0.031	0.066
00-361C-311	37.259	14.220	6.820	29.900	0.324	31.810	9.680	0.028	0.167	0.117	0.031	0.057
00-361C-311	36.572	14.220	6.820	34.330	0.408	28.100	9.680	0.061	0.167	0.089	0.031	0.060
00-361C-311	37.698	14.220	6.820	29.710	0.329	32.030	9.680	0.053	0.167	0.114	0.031	0.043
00-361C-311	36.572	14.220	6.820	33.820	0.500	29.000	9.680	0.053	0.167	0.126	0.031	0.040
00-361C-311	37.464	14.220	6.820	29.620	0.349	32.140	9.680	0.056	0.167	0.105	0.031	0.027
00-361C-311	36.650	14.220	6.820	33.150	0.502	30.210	9.680	0.088	0.167	0.072	0.031	0.070
00-347C-299	36.572	14.220	6.820	35.170	0.505	27.660	9.680	0.041	0.167	0.091	0.031	0.056
00-347C-299	36.572	14.220	6.820	36.280	0.570	27.300	9.680	0.039	0.167	0.098	0.031	0.044
00-347C-299	36.572	14.220	6.820	35.470	0.459	27.240	9.680	0.058	0.167	0.130	0.031	0.086
99-318C-267	36.699	14.220	6.820	33.960	0.419	29.890	9.680	0.028	0.167	0.167	0.031	0.061
99-318C-267	36.590	14.220	6.820	33.060	0.369	30.100	9.680	0.038	0.167	0.133	0.031	0.055
99-318C-267	36.670	14.220	6.820	33.540	0.377	30.120	9.680	0.053	0.167	0.112	0.031	0.093
00-370C-22	36.572	14.220	6.820	37.280	0.496	26.630	9.680	0.079	0.167	0.054	0.031	0.063
00-370C-22	36.572	14.220	6.820	37.100	0.619	26.900	9.680	0.080	0.167	0.096	0.031	0.086
00-370C-22	36.572	14.220	6.820	37.120	0.558	26.080	9.680	0.030	0.167	0.074	0.031	0.063
00-331C-194	36.572	14.220	6.820	46.480	0.506	18.230	9.680	0.070	0.167	0.054	0.031	0.063
00-331C-194	36.572	14.220	6.820	49.130	0.560	16.700	9.680	0.038	0.167	0.054	0.031	0.067
00-331C-194	36.572	14.220	6.820	47.740	0.536	17.500	9.680	0.075	0.167	0.054	0.031	0.062
26098-148	36.572	14.220	6.820	32.834	0.459	30.680	9.680	0.056	0.167	0.144	0.031	0.065
26098-148	36.572	14.220	6.820	33.558	0.412	30.710	9.680	0.050	0.167	0.141	0.031	0.048
26098-148	36.689	14.220	6.820	33.422	0.459	30.250	9.680	0.028	0.167	0.156	0.031	0.070
26056-138	36.572	14.220	6.820	33.095	0.453	31.890	9.680	0.075	0.167	0.182	0.031	0.066
26056-138	36.573	14.220	6.820	32.448	0.459	31.470	9.680	0.033	0.167	0.147	0.031	0.072
26056-138	36.572	14.220	6.820	32.603	0.405	31.260	9.680	0.034	0.167	0.166	0.031	0.054
00-368C-132	36.572	14.220	6.820	39.600	0.612	24.350	9.680	0.053	0.167	0.121	0.031	0.059
00-368C-132	36.572	14.220	6.820	39.870	0.587	24.630	9.680	0.049	0.167	0.063	0.031	0.082
00-368C-132	36.572	14.220	6.820	39.890	0.577	24.460	9.680	0.082	0.167	0.094	0.031	0.070
00-357C-112	36.572	14.220	6.820	35.721	0.528	28.510	9.680	0.043	0.167	0.097	0.031	0.079
00-357C-112	36.572	14.220	6.820	35.921	0.497	27.810	9.680	0.033	0.167	0.090	0.031	0.050
00-357C-112	36.815	14.220	6.820	35.973	0.532	28.220	9.680	0.044	0.167	0.137	0.031	0.078
Min.	36.572	14.220	6.820	29.290	0.036	16.700	9.680	0.028	0.167	0.054	0.031	0.027
P5	36.572	14.220	6.820	29.900	0.317	18.230	9.680	0.028	0.167	0.054	0.031	0.040
Median	36.572	14.220	6.820	35.660	0.459	27.660	9.680	0.050	0.167	0.105	0.031	0.065
Mean	36.651	14.220	6.820	36.322	0.450	27.113	9.680	0.053	0.167	0.104	0.031	0.064
P95	37.030	14.220	6.820	46.480	0.580	31.890	9.680	0.085	0.167	0.166	0.031	0.093
Max.	37.698	14.220	6.820	49.130	0.619	32.400	9.680	0.130	0.167	0.182	0.035	0.102
D.L.	0.015	0.013	0.020	0.050	0.025	0.012	0.015	0.011	0.013	0.020	0.031	0.027

Silicates: Altered Olivine

Thin Section:	SiO2	Al2O3	TiO2	FeO	MnO	MgO	K2O	CaO	Na2O	NiO	CuO	CoO
26064-264	41.867	14.510	3.350	23.289	0.543	13.730	8.710	2.372	0.192	0.041	0.031	0.027
26064-264	42.659	14.510	3.350	24.696	0.618	13.730	8.710	2.457	0.192	0.041	0.031	0.030
26064-264	42.521	14.510	3.350	23.762	0.538	13.730	8.710	2.036	0.583	0.041	0.031	0.027
00-347C-299	43.181	14.510	3.350	25.020	0.258	13.730	8.710	2.360	0.192	0.069	0.031	0.043
Min.	41.867	14.510	3.350	23.289	0.258	13.730	8.710	2.036	0.192	0.041	0.031	0.027
P5	41.965	14.510	3.350	23.360	0.300	13.730	8.710	2.085	0.192	0.041	0.031	0.027
Median	42.590	14.510	3.350	24.229	0.541	13.730	8.710	2.366	0.192	0.041	0.031	0.028
Mean	42.557	14.510	3.350	24.192	0.489	13.730	8.710	2.306	0.289	0.048	0.031	0.032
P95	43.103	14.510	3.350	24.971	0.606	13.730	8.710	2.444	0.524	0.064	0.031	0.041
Max.	43.181	14.510	3.350	25.020	0.618	13.730	8.710	2.457	0.583	0.069	0.031	0.043
D.L.	0.015	0.013	0.020	0.050	0.025	0.012	0.015	0.011	0.013	0.020	0.031	0.027

Silicates: Clinopyroxene

Thin Section:	SiO2	Al2O3	TiO2	FeO	MnO	MgO	K2O	CaO	Na2O	NiO	CuO	CoO
00-331C-194	49.587	14.680	4.470	12.350	0.242	18.380	9.420	21.630	0.242	0.109	0.031	0.027
00-331C-194	51.823	14.680	4.470	12.210	0.208	18.380	9.420	20.250	0.242	0.109	0.031	0.027
00-331C-194	51.615	14.680	4.470	12.950	0.240	18.380	9.420	19.140	0.242	0.109	0.031	0.027
00-340C-389	51.253	14.680	4.470	10.465	0.248	18.380	9.420	21.390	0.252	0.109	0.031	0.027
00-340C-389	51.027	14.680	4.470	10.465	0.212	18.380	9.420	20.440	0.263	0.109	0.031	0.027
00-340C-389	51.662	14.680	4.470	10.465	0.141	18.380	9.420	20.090	0.338	0.109	0.031	0.029
00-340C-596	51.328	14.680	4.470	10.850	0.283	18.380	9.420	21.020	0.246	0.109	0.031	0.027
00-340C-596	51.072	14.680	4.470	11.396	0.190	18.380	9.420	18.460	0.322	0.109	0.031	0.027
00-344C-518	51.730	14.680	4.470	10.465	0.227	18.380	9.420	20.400	0.321	0.109	0.031	0.027
00-347C-299	50.367	14.680	4.470	10.820	0.288	18.380	9.420	19.300	0.300	0.109	0.031	0.035
00-347C-798	52.123	14.680	4.470	10.465	0.330	18.380	9.420	20.360	0.321	0.109	0.031	0.027
00-347C-798	52.153	14.680	4.470	10.760	0.301	18.380	9.420	20.360	0.357	0.109	0.031	0.027
00-347C-798	50.900	14.680	4.470	10.465	0.325	18.380	9.420	19.040	0.365	0.109	0.047	0.027
00-357C-112	50.157	14.680	4.470	10.465	0.291	18.380	9.420	21.950	0.377	0.109	0.031	0.027
00-357C-112	50.995	14.680	4.470	10.868	0.295	18.380	9.420	20.500	0.347	0.109	0.031	0.034
00-357C-112	49.719	14.680	4.470	11.645	0.266	18.380	9.420	18.090	0.347	0.109	0.031	0.027
00-361C-311	51.875	14.680	4.470	10.465	0.181	18.380	9.420	20.410	0.305	0.109	0.031	0.027
00-361C-311	50.998	14.680	4.470	10.465	0.187	18.380	9.420	19.630	0.283	0.109	0.031	0.027
00-361C-311	51.370	14.680	4.470	10.465	0.239	18.380	9.420	19.630	0.247	0.109	0.031	0.027
00-361C-311	50.820	14.680	4.470	11.260	0.344	18.380	9.420	19.130	0.242	0.109	0.031	0.027
00-361C-311	51.130	14.680	4.470	11.470	0.265	18.380	9.420	18.120	0.242	0.109	0.031	0.030
00-367C-401	51.930	14.680	4.470	12.400	0.262	18.380	9.420	21.160	0.242	0.109	0.031	0.027
00-367C-401	52.330	14.680	4.470	11.520	0.217	18.380	9.420	20.100	0.245	0.109	0.031	0.027
00-368C-132	51.070	14.680	4.470	11.710	0.324	18.380	9.420	21.180	0.296	0.109	0.031	0.027
00-368C-132	51.370	14.680	4.470	11.630	0.357	18.380	9.420	19.610	0.313	0.109	0.031	0.027
00-368C-132	52.030	14.680	4.470	11.970	0.276	18.380	9.420	19.140	0.289	0.109	0.031	0.029
00-370C-22	51.460	14.680	4.470	10.465	0.372	18.380	9.420	21.380	0.242	0.109	0.031	0.027
00-370C-22	51.167	14.680	4.470	10.465	0.294	18.380	9.420	19.630	0.305	0.109	0.031	0.027
00-370C-22	51.439	14.680	4.470	10.560	0.297	18.380	9.420	19.580	0.242	0.109	0.031	0.027
26039-311	50.727	14.680	4.470	10.465	0.178	18.380	9.420	21.960	0.296	0.109	0.031	0.027
26039-311	52.121	14.680	4.470	10.465	0.161	18.380	9.420	20.080	0.320	0.109	0.031	0.027
26049-361	51.450	14.680	4.470	10.465	0.164	18.380	9.420	21.910	0.242	0.109	0.033	0.027
26049-361	50.056	14.680	4.470	10.465	0.244	18.380	9.420	21.210	0.251	0.109	0.031	0.032
26049-361	50.932	14.680	4.470	10.465	0.209	18.380	9.420	20.320	0.242	0.109	0.031	0.027
26056-138	51.644	14.680	4.470	10.465	0.288	18.380	9.420	22.100	0.303	0.109	0.031	0.027
26056-138	51.092	14.680	4.470	10.465	0.232	18.380	9.420	22.010	0.275	0.109	0.031	0.027
26056-138	50.993	14.680	4.470	10.465	0.240	18.380	9.420	20.150	0.314	0.109	0.031	0.027
26056-311	49.280	14.680	4.470	10.465	0.216	18.380	9.420	21.860	0.343	0.109	0.031	0.027
26056-311	52.305	14.680	4.470	10.465	0.196	18.380	9.420	20.830	0.242	0.109	0.031	0.027
26056-311	52.257	14.680	4.470	10.870	0.213	18.380	9.420	20.730	0.242	0.109	0.031	0.027
26064-264	51.965	14.680	4.470	10.465	0.243	18.380	9.420	20.910	0.242	0.109	0.031	0.027
26064-264	51.490	14.680	4.470	10.465	0.193	18.380	9.420	20.720	0.296	0.109	0.031	0.027
26098-148	51.371	14.680	4.470	10.465	0.249	18.380	9.420	21.440	0.325	0.109	0.031	0.051
26098-148	51.134	14.680	4.470	10.465	0.285	18.380	9.420	21.330	0.382	0.109	0.044	0.027
26098-148	51.688	14.680	4.470	10.465	0.199	18.380	9.420	19.970	0.361	0.109	0.031	0.027
26142-364	51.173	14.680	4.470	10.465	0.201	18.380	9.420	22.190	0.329	0.109	0.031	0.027
26142-364	51.319	14.680	4.470	10.465	0.173	18.380	9.420	20.580	0.279	0.109	0.031	0.027
26142-364	51.830	14.680	4.470	10.465	0.217	18.380	9.420	20.150	0.310	0.109	0.031	0.027
99-317C-462	50.940	14.680	4.470	13.920	0.279	18.380	9.420	22.130	0.242	0.109	0.031	0.027
99-317C-462	49.941	14.680	4.470	15.240	0.246	18.380	9.420	19.840	0.367	0.109	0.041	0.027
99-317C-462	51.568	14.680	4.470	14.890	0.319	18.380	9.420	19.710	0.242	0.109	0.031	0.027
99-318C-267	51.856	14.680	4.470	10.465	0.218	18.380	9.420	20.670	0.281	0.109	0.031	0.027
99-318C-267	49.955	14.680	4.470	10.465	0.222	18.380	9.420	20.570	0.304	0.109	0.031	0.027
99-318C-267	50.262	14.680	4.470	11.150	0.231	18.380	9.420	19.360	0.242	0.109	0.031	0.027
99-320-172	49.735	14.680	4.470	14.080	0.217	18.380	9.420	18.270	0.270	0.109	0.031	0.027
99-320C-172	52.481	14.680	4.470	11.160	0.248	18.380	9.420	20.650	0.242	0.109	0.031	0.027
99-320C-172	48.462	14.680	4.470	12.880	0.236	18.380	9.420	18.710	0.271	0.109	0.031	0.027
Min.	48.46	14.68	4.47	10.46	0.14	18.38	9.42	18.09	0.24	0.11	0.03	0.03
P5	49.69	14.68	4.47	10.46	0.17	18.38	9.42	18.42	0.24	0.11	0.03	0.03
Median	51.32	14.68	4.47	10.46	0.24	18.38	9.42	20.40	0.28	0.11	0.03	0.03
Mean	51.17	14.68	4.47	11.15	0.25	18.38	9.42	20.38	0.29	0.11	0.03	0.03
P95	52.27	14.68	4.47	13.95	0.33	18.38	9.42	22.03	0.37	0.11	0.03	0.03
Max.	52.48	14.68	4.47	15.24	0.37	18.38	9.42	22.19	0.38	0.11	0.05	0.05
D.L.	0.015	0.013	0.020	0.050	0.025	0.012	0.015	0.011	0.013	0.020	0.031	0.027

Silicates: Orthopyroxene

Thin Section:	SiO2	Al2O3	TiO2	FeO	MnO	MgO	K2O	CaO	Na2O	NiO	CuO	CoO
00-340C-778	52.242	14.137	4.730	22.200	0.399	20.890	7.720	1.190	0.804	0.096	0.031	0.043
00-340C-778	52.331	14.137	4.730	23.010	0.393	21.200	7.720	0.995	0.804	0.096	0.031	0.040
00-340C-596	52.727	14.137	4.730	23.690	0.546	22.250	7.720	1.553	0.804	0.096	0.031	0.050
26026-566	48.312	14.137	4.730	28.700	0.178	17.360	7.720	0.196	0.804	0.096	0.031	0.040
26026-566	49.807	14.137	4.730	28.430	0.222	17.360	7.720	0.182	0.804	0.096	0.031	0.040
26026-566	50.144	14.137	4.730	28.280	0.180	17.360	7.720	0.160	0.804	0.096	0.031	0.040
00-344C-518	51.750	14.137	4.730	17.620	0.317	25.370	7.720	0.929	0.804	0.096	0.031	0.040
00-367C-401	51.470	14.137	4.730	26.610	0.449	18.030	7.720	1.588	0.804	0.096	0.031	0.040
00-367C-398	52.880	14.137	4.730	22.290	0.544	21.210	7.720	0.543	0.804	0.096	0.031	0.061
00-367C-398	52.400	14.137	4.730	22.140	0.528	21.410	7.720	0.691	0.804	0.096	0.031	0.055
00-367C-398	52.580	14.137	4.730	22.100	0.492	21.860	7.720	0.602	0.804	0.096	0.031	0.040
00-367C-398	52.200	14.137	4.730	22.080	0.416	21.360	7.720	0.612	0.804	0.096	0.031	0.040
00-367C-398	52.240	14.137	4.730	22.320	0.539	21.890	7.720	0.603	0.804	0.096	0.046	0.040
00-367C-398	51.830	14.137	4.730	22.420	0.465	21.860	7.720	0.673	0.804	0.096	0.031	0.040
00-347C-299	53.614	14.137	4.730	19.820	0.487	23.380	7.720	1.056	0.804	0.096	0.031	0.042
99-318C-267	53.866	14.137	4.730	19.670	0.383	24.150	7.720	0.738	0.804	0.096	0.031	0.040
26064-264	53.292	14.137	4.730	20.780	0.349	22.730	7.720	2.329	0.804	0.096	0.031	0.040
99-320C-172	52.412	14.137	4.730	22.700	0.411	21.010	7.720	1.981	0.804	0.096	0.031	0.040
99-320C-172	51.456	14.137	4.730	24.600	0.446	19.950	7.720	1.536	0.804	0.096	0.031	0.040
99-320C-172	52.185	14.137	4.730	24.120	0.480	20.400	7.720	1.474	0.804	0.096	0.031	0.040
Min.	48.312	14.137	4.730	17.620	0.178	17.360	7.720	0.160	0.804	0.096	0.031	0.040
P5	49.732	14.137	4.730	19.568	0.180	17.360	7.720	0.181	0.804	0.096	0.031	0.040
Median	52.241	14.137	4.730	22.370	0.431	21.285	7.720	0.833	0.804	0.096	0.031	0.040
Mean	51.987	14.137	4.730	23.179	0.411	21.052	7.720	0.981	0.804	0.096	0.032	0.043
P95	53.626	14.137	4.730	28.444	0.545	24.211	7.720	1.998	0.804	0.096	0.032	0.056
Max.	53.866	14.137	4.730	28.700	0.546	25.370	7.720	2.329	0.804	0.096	0.046	0.061
	0.015	0.013	0.020	0.050	0.025	0.012	0.015	0.011	0.013	0.020	0.031	0.027

Silicates: Cordierite

Thin Section	SiO2	Al2O3	TiO2	FeO	MnO	MgO	K2O	CaO	Na2O	NiO	CuO	CoO
26026-566	48.946	33.940	3.350	16.290	0.048	13.730	8.710	0.070	0.192	0.041	0.031	0.027
26026-566	48.866	33.820	3.350	16.290	0.084	13.730	8.710	0.070	0.192	0.041	0.031	0.027
26026-566	48.609	33.820	3.350	16.290	0.041	13.730	8.710	0.070	0.192	0.041	0.031	0.027
26026-566	48.659	33.620	3.350	16.290	0.057	13.730	8.710	0.070	0.192	0.041	0.031	0.027
Min.	48.609	33.620	3.350	16.290	0.041	13.730	8.710	0.070	0.192	0.041	0.031	0.027
P5	48.616	33.650	3.350	16.290	0.042	13.730	8.710	0.070	0.192	0.041	0.031	0.027
Median	48.762	33.820	3.350	16.290	0.052	13.730	8.710	0.070	0.192	0.041	0.031	0.027
Mean	48.770	33.800	3.350	16.290	0.057	13.730	8.710	0.070	0.192	0.041	0.031	0.027
P95	48.934	33.922	3.350	16.290	0.080	13.730	8.710	0.070	0.192	0.041	0.031	0.027
Max.	48.946	33.940	3.350	16.290	0.084	13.730	8.710	0.070	0.192	0.041	0.031	0.027
	0.015	0.013	0.020	0.050	0.025	0.012	0.015	0.011	0.013	0.020	0.031	0.027

Oxides: Ilmenite

Thin Section	SiO2	Al2O3	MgO	FeO (t)	MnO	TiO2	Cr2O3	CuO	CoO	NiO	ex O2 *
00-344C-518	0.012	0.014	2.660	41.943	0.572	52.868	0.124	0.023	0.049	0.026	1.71
00-367C-401	0.012	0.013	0.597	45.447	0.600	52.064	0.022	0.023	0.061	0.026	1.14
00-367C-401	0.012	0.021	0.555	45.618	0.566	52.406	0.022	0.023	0.053	0.026	0.70
00-367C-401	0.012	0.011	0.681	45.407	0.561	51.974	0.022	0.023	0.039	0.026	1.24
26039-311	0.012	0.055	2.704	42.729	0.535	52.155	0.370	0.023	0.061	0.055	1.30
26039-311	0.012	0.047	2.790	42.547	0.506	51.793	0.312	0.023	0.056	0.038	1.88
26039-311	0.012	0.054	2.750	42.547	0.548	52.175	0.290	0.023	0.057	0.045	1.50
00-331C-194	0.012	0.050	0.971	45.175	0.495	51.351	0.107	0.023	0.064	0.031	1.72
00-331C-194	0.012	0.035	0.953	45.487	0.439	51.441	0.099	0.023	0.050	0.026	1.43
00-331C-194	0.012	0.035	0.882	45.387	0.481	51.089	0.108	0.023	0.050	0.027	1.91
00-331C-194	0.012	0.030	0.279	45.779	0.383	52.104	0.022	0.023	0.037	0.026	1.31
26039-311	0.012	0.032	2.387	42.708	0.545	52.808	0.080	0.023	0.042	0.038	1.32
26142-364	0.012	0.028	1.080	44.883	0.628	50.938	0.110	0.023	0.062	0.026	2.21
26142-364	0.012	0.025	0.966	45.376	0.642	51.521	0.081	0.023	0.082	0.028	1.24
00-340C-389	0.012	0.087	2.165	42.960	0.520	51.150	0.085	0.023	0.064	0.026	2.91
00-340C-389	0.012	0.051	2.377	43.202	0.517	51.582	0.050	0.023	0.057	0.026	2.10
00-340C-389	0.012	0.052	2.238	43.605	0.487	51.994	0.113	0.023	0.056	0.044	1.38
26049-361	0.012	0.018	2.127	42.900	0.563	52.155	0.122	0.023	0.062	0.026	1.99
99-318C-267	0.012	0.080	2.666	45.014	0.444	49.089	0.083	0.023	0.065	0.026	2.50
99-318C-267	0.012	0.084	2.304	45.638	0.434	48.767	0.051	0.023	0.089	0.031	2.57
99-318C-267	0.012	0.078	2.261	46.031	0.466	49.290	0.060	0.023	0.062	0.030	1.69
26026-566	0.012	0.024	0.201	45.779	0.456	51.119	0.022	0.023	0.028	0.026	2.31
26026-566	0.012	0.031	0.333	45.779	0.430	51.833	0.022	0.023	0.047	0.026	1.46
26026-566	0.012	0.033	0.280	45.719	0.356	51.139	0.022	0.023	0.035	0.026	2.36
99-320C-172	0.012	0.046	2.389	44.312	0.600	49.980	0.141	0.023	0.051	0.026	2.42
99-317C-462	0.012	0.038	0.940	45.748	0.428	50.840	0.022	0.023	0.039	0.026	1.88
99-317C-462	0.012	0.028	0.525	46.405	0.522	52.400	0.022	0.023	0.047	0.026	-0.01
99-317C-462	0.012	0.026	0.418	46.395	0.478	50.650	0.022	0.023	0.049	0.026	1.90
00-367C-398	0.012	0.018	1.328	44.150	0.522	53.250	0.077	0.023	0.041	0.026	0.55
00-367C-398	0.012	0.014	1.439	43.928	0.525	51.780	0.091	0.023	0.054	0.026	2.11
00-367C-398	0.012	0.027	1.360	44.262	0.581	53.510	0.117	0.023	0.042	0.026	0.04
00-368C-132	0.012	0.084	1.809	46.081	0.455	49.170	0.022	0.023	0.055	0.026	2.26
00-368C-132	0.012	0.065	1.826	46.021	0.445	50.690	0.022	0.023	0.047	0.026	0.82
00-368C-132	0.012	0.080	1.871	45.657	0.462	49.650	0.022	0.023	0.059	0.030	2.13
00-370C-22	0.012	0.033	1.055	44.433	0.659	53.550	0.022	0.023	0.059	0.026	0.13
00-370C-22	0.012	0.021	0.746	44.808	0.754	51.630	0.022	0.023	0.060	0.026	1.90
00-370C-22	0.012	0.018	0.211	45.455	0.933	53.170	0.022	0.023	0.053	0.026	0.08
00-361C-311	0.012	0.039	2.492	42.361	0.525	52.320	0.079	0.023	0.050	0.026	2.07
00-361C-311	0.012	0.038	2.516	42.007	0.524	54.370	0.098	0.023	0.059	0.026	0.33
00-361C-311	0.012	0.028	2.750	42.068	0.439	52.410	0.163	0.023	0.043	0.034	2.03
00-347C-299	0.012	0.031	2.161	43.645	0.551	52.740	0.087	0.023	0.049	0.026	0.68
00-347C-299	0.012	0.044	2.263	43.786	0.560	50.790	0.143	0.023	0.031	0.034	2.31
00-357C-112	0.012	0.082	1.812	45.141	0.443	50.030	0.101	0.023	0.050	0.037	2.27
00-357C-112	0.012	0.060	1.883	45.293	0.450	51.280	0.098	0.023	0.066	0.026	0.81
00-357C-112	0.012	0.056	2.447	44.211	0.448	50.060	0.212	0.023	0.070	0.045	2.42
00-340C-778	0.012	0.012	1.648	43.412	0.424	53.280	0.223	0.023	0.033	0.026	0.91
00-340C-778	0.012	0.024	1.935	42.877	0.383	51.740	0.189	0.023	0.053	0.026	2.74
00-340C-778	0.012	0.026	1.812	43.554	0.367	53.340	0.176	0.023	0.049	0.026	0.62
00-347C-798	0.012	0.062	1.901	44.373	0.494	50.490	0.148	0.023	0.053	0.026	2.42
00-347C-798	0.012	0.085	1.821	45.353	0.476	51.220	0.180	0.023	0.054	0.026	0.75
00-347C-798	0.189	0.210	2.123	44.363	0.474	51.150	0.108	0.023	0.057	0.026	1.28
26064-264	0.012	0.025	2.344	42.775	0.514	52.120	0.229	0.037	0.065	0.026	1.85
26064-264	0.085	0.064	2.205	42.836	0.496	51.770	0.172	0.023	0.054	0.026	2.27
26064-264	0.012	0.044	2.645	42.523	0.498	51.930	0.260	0.023	0.063	0.041	1.96
26098-148	0.012	0.079	2.556	43.716	0.465	50.590	0.252	0.023	0.063	0.045	2.20
26098-148	0.012	0.048	2.553	43.230	0.482	51.300	0.210	0.023	0.056	0.028	2.06
26098-148	0.012	0.052	2.372	43.857	0.444	50.700	0.109	0.023	0.056	0.033	2.34
00-340C-596	0.012	0.045	2.170	44.797	0.420	50.470	0.097	0.023	0.040	0.032	1.89
00-340C-596	0.012	0.047	2.477	43.453	0.484	51.390	0.121	0.023	0.034	0.028	1.93
26056-138	0.012	0.035	0.952	45.030	0.808	50.590	0.353	0.023	0.050	0.057	2.09
26056-138	0.012	0.031	1.877	43.240	0.644	51.810	0.250	0.023	0.056	0.033	2.02
26056-138	0.012	0.044	2.538	42.947	0.548	51.910	0.143	0.023	0.045	0.042	1.75
Min.	0.012	0.011	0.201	41.943	0.356	48.767	0.022	0.023	0.028	0.026	-0.010
P5	0.012	0.014	0.283	42.369	0.385	49.308	0.022	0.023	0.034	0.026	0.138
Median	0.012	0.038	1.892	44.337	0.496	51.606	0.100	0.023	0.053	0.026	1.890
Mean	0.016	0.045	1.732	44.293	0.514	51.530	0.117	0.023	0.053	0.030	1.647
P95	0.012	0.084	2.702	46.030	0.658	53.337	0.289	0.023	0.066	0.045	2.494
Max.	0.189	0.210	2.790	46.405	0.933	54.370	0.370	0.037	0.089	0.057	2.908
D.L.	0.012	0.011	0.009	0.021	0.019	0.020	0.022	0.023	0.020	0.026	

Oxides: Magnetite

Thin Section	SiO2	Al2O3	MgO	FeO (t)	MnO	TiO2	Cr2O3	CuO	CoO	NiO	ex O2 *
00-344C-518	0.133	0.024	0.234	93.481	0.456	51.119	0.022	0.023	0.100	0.026	5.962
00-344C-518	0.489	0.024	0.431	92.646	0.561	51.119	0.022	0.023	0.124	0.026	5.741
26039-311	0.012	7.000	1.795	55.525	0.456	51.119	19.760	0.023	0.086	0.108	4.822
26039-311	0.012	11.350	1.542	49.434	0.456	51.119	27.030	0.023	0.074	0.066	4.624
26039-311	0.012	8.000	1.606	53.793	0.456	51.119	22.040	0.023	0.085	0.104	5.523
26142-364	0.012	6.150	1.244	63.600	0.456	51.119	16.700	0.023	0.108	0.143	6.355
26049-361	0.164	11.990	1.789	45.779	0.456	51.119	36.120	0.023	0.085	0.060	5.117
26049-361	0.012	12.770	1.962	45.779	0.456	51.119	35.260	0.023	0.084	0.041	4.630
99-320C-172	0.012	2.673	0.798	80.951	0.456	51.119	2.584	0.023	0.099	0.108	5.338
99-320C-172	0.012	2.752	0.808	79.687	0.456	51.119	2.577	0.023	0.093	0.090	5.076
00-347C-299	0.012	4.450	0.824	74.096	0.456	51.119	9.020	0.023	0.097	0.107	5.691
00-347C-299	0.012	4.760	0.959	72.024	0.456	51.119	10.020	0.023	0.119	0.123	5.234
00-340C-596	0.012	3.800	1.043	73.843	0.456	51.119	5.400	0.023	0.096	0.143	5.857
26056-138	0.012	4.710	1.047	69.941	0.456	51.119	8.350	0.023	0.096	0.171	5.370
Min.	0.012	0.024	0.234	45.779	0.456	51.119	0.022	0.023	0.074	0.026	4.624
P5	0.012	0.024	0.362	45.779	0.456	51.119	0.022	0.023	0.080	0.026	4.627
Median	0.012	4.735	1.045	70.982	0.456	51.119	9.520	0.023	0.096	0.105	5.354
Mean	0.066	5.747	1.149	67.899	0.463	51.119	13.922	0.023	0.096	0.094	5.381
P95	0.278	12.263	1.853	92.938	0.493	51.119	35.561	0.023	0.121	0.153	6.100
Max.	0.489	12.770	1.962	93.481	0.561	51.119	36.120	0.023	0.124	0.171	6.355
D.L.	0.012	0.011	0.009	0.021	0.019	0.020	0.022	0.023	0.020	0.026	

Sulfides: Chalcopyrite

Thin Section	Mineral Rpt	Mineral Actual	Fe	Cu	S	Ni	Co	Zn	Ti	As
26064-264	Chalcopyrite	Chalcopyrite	30.730	33.950	34.630	0.106	0.042	0.030	0.020	0.030
26064-264	Chalcopyrite	Chalcopyrite	30.720	34.410	33.710	0.027	0.037	0.024	0.020	0.030
26064-264	Chalcopyrite	Chalcopyrite	30.640	34.390	33.750	0.022	0.039	0.026	0.020	0.030
26064-264	Chalcopyrite	Chalcopyrite	30.660	34.020	34.630	0.045	0.022	0.024	0.020	0.030
26098-148	Chalcopyrite	Chalcopyrite	30.550	33.580	34.970	0.089	0.054	0.024	0.020	0.030
26098-148	Chalcopyrite	Chalcopyrite	30.510	33.670	34.490	0.045	0.040	0.033	0.020	0.030
26056-138	Chalcopyrite	Chalcopyrite	30.610	34.220	34.020	0.025	0.029	0.024	0.020	0.030
26056-138	Chalcopyrite	Chalcopyrite	31.030	34.440	34.980	0.021	0.028	0.024	0.020	0.030
26056-138	Chalcopyrite	Chalcopyrite	30.660	33.720	34.840	0.117	0.031	0.024	0.020	0.030
00-340C-596	Chalcopyrite	Chalcopyrite	30.980	34.230	34.060	0.021	0.042	0.024	0.020	0.030
00-340C-596	Chalcopyrite	Chalcopyrite	30.840	34.060	35.110	0.021	0.033	0.024	0.020	0.030
00-340C-596	Chalcopyrite	Chalcopyrite	30.760	34.440	34.210	0.021	0.044	0.042	0.020	0.030
00-368C-132	Chalcopyrite	Chalcopyrite	30.960	33.710	33.760	0.110	0.045	0.024	0.020	0.030
00-368C-132	Chalcopyrite	Chalcopyrite	30.800	34.090	34.850	0.021	0.023	0.024	0.020	0.030
00-368C-132	Chalcopyrite	Chalcopyrite	30.770	33.970	34.860	0.021	0.036	0.024	0.020	0.030
00-367C-398	Chalcopyrite	Chalcopyrite	29.940	33.480	34.670	0.094	0.050	0.024	0.020	0.030
00-367C-398	Chalcopyrite	Chalcopyrite	30.790	33.370	34.540	0.021	0.049	0.024	0.020	0.030
00-367C-398	Chalcopyrite	Chalcopyrite	31.140	33.300	35.250	0.073	0.075	0.024	0.020	0.030
00-370C-22	Chalcopyrite	Chalcopyrite	30.750	34.140	34.950	0.021	0.023	0.024	0.020	0.030
00-370C-22	Chalcopyrite	Chalcopyrite	30.780	33.640	33.810	0.021	0.023	0.024	0.020	0.030
00-370C-22	Chalcopyrite	Chalcopyrite	31.140	33.420	35.000	0.021	0.037	0.024	0.020	0.030
00-361C-311	Chalcopyrite	Chalcopyrite	30.680	33.620	34.900	0.141	0.048	0.024	0.020	0.030
00-361C-311	Chalcopyrite	Chalcopyrite	30.380	33.510	34.680	0.021	0.034	0.024	0.020	0.030
00-361C-311	Chalcopyrite	Chalcopyrite	30.810	33.940	34.660	0.024	0.025	0.033	0.020	0.030
00-347C-299	Chalcopyrite	Chalcopyrite	30.350	34.000	34.760	0.041	0.038	0.024	0.020	0.030
00-347C-299	Chalcopyrite	Chalcopyrite	30.740	34.270	33.600	0.034	0.030	0.024	0.020	0.030
00-347C-299	Chalcopyrite	Chalcopyrite	30.630	33.880	34.970	0.097	0.038	0.026	0.020	0.030
00-357C-112	Chalcopyrite	Chalcopyrite	30.440	33.900	33.710	0.031	0.034	0.027	0.020	0.030
00-357C-112	Chalcopyrite	Chalcopyrite	31.100	33.720	34.460	0.136	0.038	0.024	0.020	0.030
00-357C-112	Chalcopyrite	Chalcopyrite	30.790	33.560	34.650	0.639	0.116	0.024	0.020	0.030
00-347C-798	Chalcopyrite	Chalcopyrite	30.800	33.890	34.590	0.021	0.028	0.024	0.020	0.030
00-347C-798	Chalcopyrite	Chalcopyrite	30.550	33.420	34.990	0.120	0.059	0.024	0.020	0.030
00-347C-798	Chalcopyrite	Chalcopyrite	30.140	33.470	34.350	0.076	0.049	0.035	0.020	0.030
00-340C-778	Chalcopyrite	Chalcopyrite	30.840	34.350	33.800	0.021	0.022	0.024	0.020	0.030
00-340C-778	Chalcopyrite	Chalcopyrite	31.010	34.100	33.720	0.021	0.026	0.036	0.020	0.030
00-340C-778	Chalcopyrite	Chalcopyrite	30.660	34.030	34.510	0.096	0.116	0.024	0.020	0.030
00-340C-778	Chalcopyrite	Chalcopyrite	30.640	34.060	34.520	0.099	0.107	0.024	0.020	0.030
99-318C-267	Chalcopyrite	Chalcopyrite	31.290	33.630	34.780	0.021	0.047	0.024	0.020	0.030
99-318C-267	Chalcopyrite	Chalcopyrite	30.500	33.770	33.580	0.021	0.028	0.024	0.020	0.030
99-318C-267	Chalcopyrite	Chalcopyrite	30.390	34.180	35.330	0.021	0.046	0.024	0.020	0.030
99-318C-267	Chalcopyrite	Chalcopyrite	31.570	32.880	35.110	0.246	0.048	0.024	0.020	0.030
26026-566	Chalcopyrite	Chalcopyrite	30.690	34.300	34.720	0.021	0.032	0.024	0.020	0.030
26026-566	Chalcopyrite	Chalcopyrite	30.610	34.580	34.020	0.021	0.041	0.024	0.020	0.030
26026-566	Chalcopyrite	Chalcopyrite	30.810	34.100	34.880	0.021	0.048	0.024	0.020	0.030
99-320C-172	Chalcopyrite	Chalcopyrite	30.600	34.050	34.800	0.094	0.036	0.024	0.020	0.030
99-320C-172	Chalcopyrite	Chalcopyrite	30.660	34.680	33.840	0.021	0.028	0.025	0.020	0.030
99-320C-172	Chalcopyrite	Chalcopyrite	30.790	34.340	33.950	0.021	0.032	0.024	0.020	0.030
99-317C-462	Chalcopyrite	Chalcopyrite	30.450	34.060	35.150	0.021	0.030	0.024	0.020	0.030
99-317C-462	Chalcopyrite	Chalcopyrite	30.310	33.930	34.050	0.071	0.036	0.024	0.020	0.030
99-317C-462	Chalcopyrite	Chalcopyrite	30.720	34.220	33.920	0.047	0.025	0.027	0.020	0.030
26056-311	Chalcopyrite	Chalcopyrite	30.680	34.030	34.570	0.021	0.028	0.024	0.020	0.030
26056-311	Chalcopyrite	Chalcopyrite	30.640	34.280	33.940	0.021	0.030	0.024	0.020	0.030
26056-311	Chalcopyrite	Chalcopyrite	30.900	33.370	34.820	0.021	0.038	0.028	0.026	0.030
26142-364	Chalcopyrite	Chalcopyrite	30.410	33.390	34.990	0.021	0.031	0.024	0.020	0.030
26142-364	Chalcopyrite	Chalcopyrite	30.250	34.040	34.820	0.037	0.028	0.024	0.020	0.030
26142-364	Chalcopyrite	Chalcopyrite	30.570	34.200	33.720	0.021	0.025	0.024	0.020	0.030
26142-364	Chalcopyrite	Chalcopyrite	30.390	33.800	35.220	0.083	0.028	0.024	0.020	0.030
26049-361	Chalcopyrite	Chalcopyrite	29.760	34.290	35.250	0.021	0.020	0.024	0.020	0.030
26049-361	Chalcopyrite	Chalcopyrite	30.330	34.420	33.900	0.021	0.029	0.024	0.020	0.030
26049-361	Chalcopyrite	Chalcopyrite	30.350	34.490	34.810	0.021	0.025	0.024	0.020	0.030
26049-361	Chalcopyrite	Chalcopyrite	29.970	33.530	34.720	0.066	0.030	0.035	0.043	0.030
00-340C-389	Chalcopyrite	Chalcopyrite	30.770	34.200	34.210	0.033	0.023	0.024	0.020	0.030
00-340C-389	Chalcopyrite	Chalcopyrite	30.460	34.610	34.490	0.021	0.031	0.024	0.020	0.030
00-344C-518	Chalcopyrite	Chalcopyrite	30.800	34.190	35.250	0.125	0.042	0.059	0.020	0.030
00-344C-518	Chalcopyrite	Chalcopyrite	30.060	34.210	34.640	0.024	0.048	0.024	0.020	0.030
00-344C-518	Chalcopyrite	Chalcopyrite	30.600	34.120	34.690	0.043	0.028	0.024	0.020	0.030
00-367C-401	Chalcopyrite	Chalcopyrite	30.570	34.380	34.590	0.021	0.030	0.030	0.020	0.030
00-367C-401	Chalcopyrite	Chalcopyrite	30.430	34.130	34.840	0.021	0.020	0.024	0.020	0.030
00-367C-401	Chalcopyrite	Chalcopyrite	30.680	34.390	34.040	0.021	0.019	0.024	0.020	0.030
00-331C-194	Chalcopyrite	Chalcopyrite	30.580	34.360	34.240	0.061	0.048	0.024	0.020	0.030
00-331C-194	Chalcopyrite	Chalcopyrite	30.560	34.300	33.870	0.021	0.037	0.024	0.020	0.030
00-331C-194	Chalcopyrite	Chalcopyrite	30.610	34.610	34.070	0.021	0.047	0.024	0.020	0.030
26039-311	Chalcopyrite	Chalcopyrite	30.420	33.930	33.870	0.106	0.035	0.031	0.020	0.030
26039-311	Chalcopyrite	Chalcopyrite	30.670	34.350	33.840	0.129	0.029	0.024	0.020	0.030
26039-311	Chalcopyrite	Chalcopyrite	30.450	33.950	33.750	0.050	0.019	0.024	0.020	0.030
n			75	75	75	75	75	75	75	75
Min.			29.760	32.880	33.580	0.021	0.019	0.024	0.020	0.030
P5			30.116	33.384	33.717	0.021	0.021	0.024	0.020	0.030
Median			30.660	34.060	34.590	0.022	0.034	0.024	0.020	0.030
Mean			30.635	34.003	34.470	0.057	0.038	0.026	0.020	0.030
P95			31.112	34.517	35.229	0.131	0.064	0.035	0.020	0.030
Max.			31.570	34.680	35.330	0.639	0.116	0.059	0.043	0.030
D.L.			0.016	0.018	0.009	0.021	0.015	0.024	0.020	0.030

Sulfides: Pentlandite

Thin Section	Mineral Rpt	Mineral Actual	Fe	Cu	S	Ni	Co	Zn	Ti	As
26064-264	Pyrrhotite	Pentlandite	30.860	0.193	33.630	32.540	3.630	0.024	0.020	0.030
26064-264	Pyrrhotite	Pentlandite	32.430	0.034	33.160	32.740	2.051	0.024	0.020	0.030
26064-264	Pyrrhotite	Pentlandite	32.030	0.040	33.640	32.070	2.952	0.024	0.020	0.030
26064-264	Pyrrhotite	Pentlandite	33.460	0.077	32.590	31.490	1.885	0.024	0.020	0.030
26064-264	Pentlandite	Pentlandite	33.520	0.165	32.460	31.460	1.897	0.024	0.020	0.030
26098-148	Pentlandite	Pentlandite	34.510	0.126	32.380	31.050	1.358	0.024	0.020	0.030
26098-148	Pentlandite	Pentlandite	27.300	0.369	33.380	35.730	2.365	0.024	0.020	0.030
26098-148	Pentlandite	Pentlandite	29.960	0.114	33.370	35.910	1.389	0.024	0.020	0.030
26098-148	Pentlandite	Pentlandite	29.570	0.222	32.310	35.690	1.465	0.024	0.020	0.030
26056-138	Pentlandite	Pentlandite	34.920	4.380	34.320	26.110	1.530	0.024	0.020	0.030
00-340C-596	Pentlandite	Pentlandite	30.690	0.064	32.260	35.050	1.493	0.024	0.020	0.030
00-340C-596	Pentlandite	Pentlandite	29.820	0.077	33.720	35.910	1.650	0.024	0.020	0.030
00-340C-596	Pentlandite	Pentlandite	28.780	0.154	33.380	36.600	1.796	0.024	0.020	0.030
00-368C-132	Pentlandite	Pentlandite	32.130	0.152	33.530	32.680	2.076	0.043	0.020	0.030
00-368C-132	Pentlandite	Pentlandite	31.840	0.035	32.770	32.660	2.419	0.024	0.020	0.030
00-368C-132	Pentlandite	Pentlandite	32.210	0.055	33.470	32.740	1.999	0.026	0.020	0.030
00-370C-22	Pentlandite	Pentlandite	35.130	0.061	32.570	29.070	2.864	0.024	0.020	0.030
00-370C-22	Pentlandite	Pentlandite	32.960	0.106	32.230	31.130	2.724	0.024	0.020	0.030
00-361C-311	Cubanite	Pentlandite	33.220	0.068	32.240	32.150	1.661	0.024	0.020	0.043
00-361C-311	Pentlandite	Pentlandite	33.570	0.035	33.250	32.100	1.105	0.024	0.020	0.030
00-361C-311	Pentlandite	Pentlandite	32.080	0.101	32.280	33.600	1.287	0.024	0.020	0.030
00-361C-311	Pentlandite	Pentlandite	31.930	0.208	32.850	32.860	1.448	0.024	0.020	0.030
00-361C-311	Cubanite	Pentlandite	32.850	0.051	32.930	31.930	1.476	0.024	0.020	0.030
00-361C-311	Pentlandite	Pentlandite	35.600	0.303	32.520	30.170	0.903	0.024	0.020	0.030
00-347C-299	Pentlandite	Pentlandite	28.110	0.118	32.300	36.260	1.876	0.024	0.020	0.030
00-347C-299	Pentlandite	Pentlandite	28.020	1.241	33.420	35.260	2.323	0.024	0.020	0.030
00-347C-299	Pentlandite	Pentlandite	28.460	0.303	32.300	35.720	2.035	0.024	0.020	0.030
00-347C-299	Pyrrhotite	Pentlandite	25.790	0.347	37.750	30.970	2.218	0.024	0.020	0.030
00-357C-112	Pentlandite	Pentlandite	35.450	0.260	32.940	29.220	1.636	0.024	0.020	0.030
00-357C-112	Pentlandite	Pentlandite	37.660	0.097	32.310	27.400	1.576	0.024	0.020	0.030
00-357C-112	Chalcopyrite	Pentlandite	38.080	0.206	32.420	27.250	1.479	0.024	0.020	0.030
00-357C-112	Pentlandite	Pentlandite	35.050	0.071	32.360	30.120	1.699	0.024	0.020	0.030
00-347C-798	Pentlandite	Pentlandite	32.100	0.108	33.160	32.620	1.653	0.024	0.020	0.030
00-347C-798	Pentlandite	Pentlandite	32.410	0.049	32.980	32.490	1.814	0.024	0.020	0.030
00-347C-798	Pentlandite	Pentlandite	34.290	0.133	32.330	31.020	2.038	0.024	0.020	0.030
00-340C-778	Pentlandite	Pentlandite	30.850	0.038	33.160	28.810	7.090	0.024	0.020	0.030
00-340C-778	Pentlandite	Pentlandite	30.140	0.019	33.020	28.940	7.910	0.024	0.020	0.030
00-340C-778	Pentlandite	Pentlandite	30.550	0.082	31.710	29.560	6.700	0.024	0.020	0.030
99-318C-267	Pentlandite	Pentlandite	33.740	0.181	32.730	30.880	1.825	0.024	0.020	0.030
99-318C-267	Pentlandite	Pentlandite	34.290	0.058	31.970	30.840	1.540	0.024	0.020	0.030
99-318C-267	Pentlandite	Pentlandite	35.570	0.060	32.630	29.380	1.847	0.024	0.020	0.030
26026-566	Pentlandite	Pentlandite	31.350	0.022	32.350	30.090	5.040	0.024	0.020	0.030
26026-566	Pentlandite	Pentlandite	31.980	0.021	32.810	29.810	5.060	0.024	0.020	0.030
26026-566	Pentlandite	Pentlandite	31.910	0.033	32.470	29.950	5.130	0.024	0.020	0.030
99-320C-172	Pentlandite	Pentlandite	33.130	0.189	32.520	31.970	1.710	0.024	0.020	0.030
99-320C-172	Pentlandite	Pentlandite	30.710	0.294	33.640	33.740	1.585	0.024	0.020	0.030
99-320C-172	Pentlandite	Pentlandite	33.480	0.045	33.410	31.140	1.863	0.024	0.020	0.030
99-317C-462	Pentlandite	Pentlandite	31.850	0.207	32.670	32.580	2.590	0.024	0.020	0.030
99-317C-462	Pentlandite	Pentlandite	31.130	0.132	33.460	32.810	2.388	0.024	0.020	0.030
99-317C-462	Pentlandite	Pentlandite	30.010	0.797	33.530	33.750	2.294	0.024	0.020	0.030
26056-311	Pentlandite	Pentlandite	25.070	0.345	32.050	39.730	1.330	0.025	0.020	0.030
26056-311	Pentlandite	Pentlandite	28.080	0.239	32.510	37.140	1.402	0.024	0.020	0.030
26142-364	Pentlandite	Pentlandite	31.730	0.234	34.480	31.060	2.486	0.024	0.020	0.030
26142-364	Cubanite	Pentlandite	35.130	0.082	33.050	29.010	1.711	0.024	0.020	0.030
26142-364	Pentlandite	Pentlandite	35.130	0.311	33.360	29.080	1.205	0.024	0.020	0.030
26049-361	Pentlandite	Pentlandite	28.130	0.132	33.380	28.550	9.370	0.024	0.020	0.098
26049-361	Pentlandite	Pentlandite	32.670	0.162	33.800	30.000	3.670	0.028	0.020	0.052
26049-361	Pentlandite	Pentlandite	33.250	0.268	33.530	30.780	1.974	0.024	0.020	0.030
00-340C-389	Pentlandite	Pentlandite	34.010	0.144	34.030	30.010	2.517	0.024	0.020	0.030
00-340C-389	Pentlandite	Pentlandite	34.140	0.195	34.150	31.690	0.740	0.024	0.020	0.030
00-340C-389	Pentlandite	Pentlandite	33.480	0.056	33.420	31.770	1.654	0.024	0.020	0.030
00-344C-518	Pentlandite	Pentlandite	28.740	1.008	33.410	35.540	1.508	0.024	0.020	0.030
00-344C-518	Pentlandite	Pentlandite	20.090	0.313	33.220	43.130	2.992	0.024	0.020	0.030
00-344C-518	Pentlandite	Pentlandite	30.940	1.856	34.460	29.200	1.454	0.024	0.020	0.030
00-331C-194	Pentlandite	Pentlandite	32.790	0.046	32.650	32.270	1.900	0.024	0.020	0.030
00-331C-194	Pentlandite	Pentlandite	33.350	0.072	32.500	32.010	1.699	0.024	0.020	0.030
00-331C-194	Pentlandite	Pentlandite	32.520	0.060	32.570	31.750	2.059	0.024	0.020	0.030
26039-311	Pentlandite	Pentlandite	28.360	0.760	32.550	36.870	0.734	0.024	0.020	0.030
26039-311	Pentlandite	Pentlandite	28.480	2.954	32.790	32.700	0.552	0.043	0.036	0.030
26039-311	Pentlandite	Pentlandite	28.960	2.611	32.860	33.030	0.724	0.024	0.020	0.030
n			70	70	70	70	70	70	70	70
Min.			20.090	0.019	31.710	26.110	0.552	0.024	0.020	0.030
P5			27.624	0.034	32.235	28.667	0.813	0.024	0.020	0.030
Median			32.090	0.132	32.895	31.950	1.836	0.024	0.020	0.030
Mean			31.750	0.345	33.034	32.162	2.315	0.025	0.020	0.031
P95			35.516	1.579	34.244	36.749	5.994	0.025	0.020	0.030
Max.			38.080	4.380	37.750	43.130	9.370	0.043	0.036	0.098
D.L.			0.016	0.018	0.009	0.021	0.015	0.024	0.020	0.030

Sulfides: Pyrrhotite

Thin Section	Mineral Rpt	Mineral Actual	Fe	Cu	S	Ni	Co	Zn	Ti	As
26098-148	Pyrrhotite	Pyrrhotite	63.610	0.018	36.890	0.021	0.059	0.024	0.020	0.030
26098-148	Pyrrhotite	Pyrrhotite	60.430	0.061	38.600	0.240	0.056	0.024	0.020	0.030
26098-148	Pyrrhotite	Pyrrhotite	60.020	0.102	38.730	0.411	0.061	0.024	0.020	0.030
26056-138	Pyrrhotite	Pyrrhotite	63.830	0.039	36.990	0.021	0.042	0.024	0.020	0.030
26056-138	Pyrrhotite	Pyrrhotite	64.110	0.148	36.060	0.021	0.050	0.024	0.020	0.030
26056-138	Pyrrhotite	Pyrrhotite	63.730	0.045	35.760	0.021	0.063	0.024	0.020	0.030
00-340C-596	Pyrrhotite	Pyrrhotite	63.960	0.018	36.350	0.021	0.070	0.024	0.020	0.030
00-340C-596	Pyrrhotite	Pyrrhotite	63.630	0.023	35.980	0.021	0.075	0.024	0.020	0.030
00-340C-596	Pyrrhotite	Pyrrhotite	61.280	0.030	37.930	0.028	0.054	0.024	0.020	0.030
00-368C-132	Pyrrhotite	Pyrrhotite	61.190	0.046	38.700	0.255	0.052	0.024	0.020	0.030
00-368C-132	Pyrrhotite	Pyrrhotite	61.450	0.033	38.020	0.283	0.049	0.024	0.020	0.030
00-368C-132	Pyrrhotite	Pyrrhotite	61.950	0.046	37.910	0.269	0.063	0.024	0.020	0.030
00-370C-22	Pyrrhotite	Pyrrhotite	63.880	0.037	35.820	0.021	0.063	0.024	0.020	0.030
00-370C-22	Pyrrhotite	Pyrrhotite	63.430	0.023	36.950	0.021	0.056	0.024	0.020	0.030
00-370C-22	Pyrrhotite	Pyrrhotite	64.130	0.036	35.780	0.021	0.072	0.024	0.020	0.030
00-370C-22	Pentlandite	Pyrrhotite	60.910	0.062	35.290	1.251	0.203	0.024	0.020	0.030
00-361C-311	Pyrrhotite	Pyrrhotite	61.010	0.041	37.840	0.116	0.057	0.024	0.020	0.030
00-361C-311	Cubanite	Pyrrhotite	60.020	0.365	39.870	0.148	0.061	0.024	0.020	0.030
00-361C-311	Pyrrhotite	Pyrrhotite	60.110	0.103	39.860	0.183	0.053	0.024	0.020	0.030
00-361C-311	Pyrrhotite	Pyrrhotite	60.720	0.032	38.130	0.165	0.052	0.024	0.020	0.030
00-361C-311	Pyrrhotite	Pyrrhotite	63.740	0.018	35.630	0.039	0.030	0.024	0.020	0.030
00-347C-299	Pyrrhotite	Pyrrhotite	59.640	0.055	39.640	0.446	0.074	0.024	0.020	0.030
00-347C-299	Pyrrhotite	Pyrrhotite	59.610	0.404	38.670	0.577	0.057	0.024	0.020	0.030
00-357C-112	Pyrrhotite	Pyrrhotite	63.560	0.061	35.570	0.035	0.066	0.024	0.020	0.030
00-357C-112	Pyrrhotite	Pyrrhotite	63.770	0.024	35.780	0.026	0.056	0.024	0.020	0.030
00-357C-112	Pyrrhotite	Pyrrhotite	63.510	0.030	36.840	0.021	0.072	0.024	0.020	0.030
00-347C-798	Pyrrhotite	Pyrrhotite	60.410	0.109	38.500	0.056	0.060	0.024	0.028	0.030
00-347C-798	Pyrrhotite	Pyrrhotite	61.310	0.038	37.790	0.120	0.061	0.024	0.020	0.030
00-347C-798	Pyrrhotite	Pyrrhotite	63.300	0.027	36.480	0.021	0.061	0.024	0.020	0.030
00-340C-778	Pyrrhotite	Pyrrhotite	61.800	0.026	37.450	0.105	0.075	0.024	0.020	0.030
00-340C-778	Pyrrhotite	Pyrrhotite	61.630	0.020	38.670	0.085	0.071	0.024	0.020	0.030
00-340C-778	Pyrrhotite	Pyrrhotite	61.420	0.036	37.300	0.185	0.076	0.024	0.020	0.030
99-318C-267	Pyrrhotite	Pyrrhotite	61.060	0.098	38.000	0.027	0.041	0.033	0.020	0.030
99-318C-267	Pyrrhotite	Pyrrhotite	60.920	0.090	38.630	0.084	0.064	0.024	0.020	0.030
99-318C-267	Pyrrhotite	Pyrrhotite	63.690	0.046	35.750	0.046	0.058	0.024	0.020	0.030
26026-566	Pyrrhotite	Pyrrhotite	60.820	0.021	39.100	0.128	0.057	0.024	0.020	0.030
26026-566	Pyrrhotite	Pyrrhotite	60.440	0.029	39.270	0.111	0.063	0.024	0.020	0.030
26026-566	Pyrrhotite	Pyrrhotite	60.900	0.028	38.300	0.109	0.052	0.024	0.020	0.030
99-320C-172	Pyrrhotite	Pyrrhotite	60.950	0.043	38.960	0.061	0.073	0.024	0.020	0.030
99-320C-172	Pyrrhotite	Pyrrhotite	60.570	0.027	39.140	0.081	0.057	0.024	0.022	0.030
99-320C-172	Pyrrhotite	Pyrrhotite	61.210	0.018	37.740	0.087	0.053	0.024	0.020	0.030
99-317C-462	Pyrrhotite	Pyrrhotite	60.190	0.069	39.610	0.203	0.048	0.024	0.020	0.030
99-317C-462	Pyrrhotite	Pyrrhotite	60.620	0.038	38.260	0.255	0.056	0.024	0.020	0.030
99-317C-462	Pyrrhotite	Pyrrhotite	59.700	0.090	39.620	0.230	0.057	0.024	0.020	0.030
26056-311	Pyrrhotite	Pyrrhotite	59.530	0.151	39.730	0.622	0.054	0.024	0.020	0.030
26056-311	Pyrrhotite	Pyrrhotite	59.380	0.094	38.720	1.015	0.061	0.024	0.020	0.030
26142-364	Pyrrhotite	Pyrrhotite	62.710	0.040	36.470	0.021	0.063	0.024	0.020	0.030
26142-364	Pyrrhotite	Pyrrhotite	63.530	0.027	35.580	0.021	0.052	0.024	0.020	0.030
26049-361	Pyrrhotite	Pyrrhotite	60.070	0.127	38.700	0.044	0.062	0.024	0.020	0.030
26049-361	Pyrrhotite	Pyrrhotite	59.730	0.225	38.680	0.167	0.091	0.024	0.020	0.087
26049-361	Pyrrhotite	Pyrrhotite	61.240	0.064	37.810	0.055	0.057	0.024	0.020	0.030
00-340C-389	Pyrrhotite	Pyrrhotite	63.430	0.019	35.660	0.021	0.051	0.024	0.020	0.030
00-340C-389	Pyrrhotite	Pyrrhotite	63.600	0.088	36.170	0.021	0.058	0.024	0.020	0.030
00-340C-389	Pyrrhotite	Pyrrhotite	63.320	0.022	36.440	0.021	0.062	0.024	0.020	0.030
00-344C-518	Pyrrhotite	Pyrrhotite	58.370	0.398	39.170	1.480	0.065	0.024	0.020	0.030
00-367C-401	Pyrrhotite-Mono	Pyrrhotite-Mono	58.070	0.018	41.120	0.482	0.099	0.024	0.020	0.030
00-367C-401	Pyrrhotite-Hex	Pyrrhotite-Hex	59.640	0.034	39.570	0.201	0.124	0.024	0.020	0.030
00-367C-401	Pyrrhotite	Pyrrhotite	58.090	0.190	41.610	0.485	0.095	0.024	0.023	0.030
00-331C-194	Pyrrhotite	Pyrrhotite	60.880	0.026	38.130	0.108	0.066	0.024	0.020	0.030
00-331C-194	Pyrrhotite	Pyrrhotite	60.890	0.021	37.860	0.103	0.053	0.024	0.020	0.030
00-331C-194	Pyrrhotite	Pyrrhotite	60.840	0.037	37.960	0.103	0.064	0.024	0.020	0.030
26039-311	Pyrrhotite	Pyrrhotite	59.630	0.229	38.950	0.675	0.052	0.024	0.020	0.030
26039-311	Pyrrhotite	Pyrrhotite	59.060	0.419	39.070	0.779	0.077	0.024	0.020	0.030
26039-311	Pyrrhotite	Pyrrhotite	59.670	0.280	38.660	0.798	0.066	0.024	0.020	0.030
n			64	64	64	64	64	64	64	64
Min.			58.070	0.018	35.290	0.021	0.030	0.024	0.020	0.030
P5			59.108	0.018	35.635	0.021	0.048	0.024	0.020	0.030
Median			60.980	0.040	38.010	0.104	0.061	0.024	0.020	0.030
Mean			61.404	0.082	37.878	0.217	0.064	0.024	0.020	0.031
P95			63.873	0.352	39.841	0.795	0.095	0.024	0.020	0.030
Max.			64.130	0.419	41.610	1.480	0.203	0.033	0.028	0.087
D.L.			0.016	0.018	0.009	0.021	0.015	0.024	0.020	0.030

Sulfides: Cubanite

Thin Section	Mineral Rpt	Mineral Actual	Fe	Cu	S	Ni	Co	Zn	Ti	As
26098-148	Cubanite	Cubanite	41.490	22.550	35.470	0.021	0.028	0.024	0.020	0.030
26098-148	Chalcopyrite	Cubanite	40.760	22.950	34.430	1.874	0.085	0.024	0.020	0.030
26098-148	Cubanite	Cubanite	41.270	22.500	34.600	0.028	0.039	0.024	0.020	0.030
26098-148	Cubanite	Cubanite	41.620	22.590	35.410	0.021	0.024	0.024	0.020	0.030
00-340C-596	Cubanite	Cubanite	41.590	23.010	34.840	0.021	0.027	0.024	0.020	0.030
00-340C-596	Cubanite	Cubanite	41.470	23.110	34.430	0.021	0.028	0.024	0.020	0.030
00-340C-596	Cubanite	Cubanite	41.460	22.850	35.590	0.021	0.043	0.024	0.020	0.030
00-370C-22	Cubanite	Cubanite	41.250	23.040	35.200	0.031	0.046	0.024	0.020	0.030
00-347C-299	Cubanite	Cubanite	41.510	22.710	34.250	0.119	0.041	0.024	0.020	0.030
00-347C-299	Cubanite	Cubanite	40.900	23.090	35.200	0.021	0.039	0.024	0.020	0.030
00-347C-299	Cubanite	Cubanite	41.430	22.770	34.890	0.036	0.057	0.028	0.020	0.030
00-357C-112	Cubanite	Cubanite	41.140	23.150	35.160	0.021	0.043	0.024	0.020	0.030
00-357C-112	Cubanite	Cubanite	41.080	23.160	35.340	0.194	0.069	0.024	0.020	0.030
00-357C-112	Cubanite	Cubanite	41.570	22.740	34.300	0.193	0.054	0.024	0.020	0.030
00-347C-798	Cubanite	Cubanite	41.630	22.620	35.090	0.021	0.034	0.024	0.028	0.030
00-347C-798	Cubanite	Cubanite	41.390	22.710	34.120	0.021	0.048	0.024	0.020	0.030
00-347C-798	Cubanite	Cubanite	41.700	22.430	34.040	0.021	0.045	0.024	0.020	0.030
00-340C-778	Cubanite	Cubanite	41.550	23.160	35.410	0.021	0.028	0.024	0.020	0.030
00-340C-778	Cubanite	Cubanite	41.600	23.000	34.970	0.040	0.081	0.024	0.020	0.030
99-318C-267	Cubanite	Cubanite	41.140	22.910	35.000	0.021	0.037	0.024	0.020	0.030
99-318C-267	Cubanite	Cubanite	40.930	23.260	35.690	0.021	0.029	0.024	0.020	0.030
99-318C-267	Cubanite	Cubanite	41.170	22.760	35.540	0.152	0.064	0.025	0.020	0.030
26026-566	Cubanite	Cubanite	41.140	23.370	34.330	0.021	0.026	0.024	0.020	0.030
26026-566	Cubanite	Cubanite	40.900	23.170	35.530	0.021	0.033	0.027	0.020	0.030
26026-566	Cubanite	Cubanite	41.100	23.040	35.590	0.046	0.043	0.024	0.020	0.030
99-320C-172	Cubanite	Cubanite	34.290	0.053	32.420	30.750	1.770	0.024	0.020	0.030
99-320C-172	Cubanite	Cubanite	40.920	23.060	34.480	0.021	0.031	0.024	0.020	0.030
26142-364	Cubanite	Cubanite	55.410	2.152	36.100	4.270	0.950	0.024	0.020	0.030
26142-364	Pyrrhotite	Cubanite	40.950	22.630	35.370	0.139	0.044	0.024	0.020	0.030
26049-361	Cubanite	Cubanite	41.040	22.930	35.640	0.021	0.052	0.025	0.020	0.030
26049-361	Cubanite	Cubanite	40.650	23.120	35.240	0.066	0.082	0.024	0.020	0.030
26049-361	Cubanite	Cubanite	41.010	23.000	35.140	0.025	0.040	0.024	0.020	0.030
00-340C-389	Cubanite	Cubanite	41.160	23.190	35.740	0.038	0.034	0.024	0.020	0.030
00-340C-389	Cubanite	Cubanite	30.490	34.470	34.880	0.021	0.036	0.024	0.020	0.030
00-340C-389	Cubanite	Cubanite	40.790	23.400	35.470	0.021	0.043	0.024	0.020	0.030
00-340C-389	Cubanite	Cubanite	41.100	23.250	35.180	0.021	0.033	0.024	0.020	0.030
00-367C-401	Cubanite	Cubanite	41.350	23.200	35.250	0.021	0.032	0.024	0.020	0.030
00-367C-401	Cubanite	Cubanite	40.840	23.170	34.560	0.021	0.042	0.024	0.020	0.030
00-331C-194	Cubanite	Cubanite	40.940	23.270	34.830	0.021	0.034	0.024	0.020	0.030
00-331C-194	Cubanite	Cubanite	41.160	23.140	34.490	0.021	0.030	0.024	0.020	0.030
00-331C-194	Cubanite	Cubanite	40.970	23.160	34.340	0.021	0.043	0.039	0.035	0.030
			41	41	41	41	41	41	41	41
Min.			30.490	0.053	32.420	0.021	0.024	0.024	0.020	0.030
P5			40.650	22.430	34.120	0.021	0.027	0.024	0.020	0.030
Median			41.140	23.040	35.140	0.021	0.041	0.024	0.020	0.030
Mean			41.119	22.191	34.965	0.940	0.107	0.025	0.021	0.030
P95			41.630	23.370	35.690	1.874	0.085	0.027	0.020	0.030
Max.			55.410	34.470	36.100	30.750	1.770	0.039	0.035	0.030
D.L.			0.016	0.018	0.009	0.021	0.015	0.024	0.020	0.030

Sulfides: Bornite

Thin Section	Mineral Rpt	Mineral Actual	Fe	Cu	S	Ni	Co	Zn	Ti	As
26056-138	Pentlandite	Bornite	8.500	67.720	24.540	0.128	0.032	0.024	0.020	0.030
26056-138	Pentlandite	Bornite	13.740	60.000	25.790	0.106	0.023	0.024	0.020	0.030
26142-364	Pentlandite	Bornite	14.280	59.340	25.930	0.021	0.015	0.024	0.020	0.030
n			3	3	3	3	3	3	3	3
Min.			8.500	59.340	24.540	0.021	0.015	0.024	0.020	0.030
P5			9.024	59.406	24.665	0.029	0.016	0.024	0.020	0.030
Median			13.740	60.000	25.790	0.106	0.023	0.024	0.020	0.030
Mean			12.173	62.353	25.420	0.085	0.023	0.024	0.020	0.030
P95			14.226	66.948	25.916	0.126	0.031	0.024	0.020	0.030
Max.			14.280	67.720	25.930	0.128	0.032	0.024	0.020	0.030
D.L.			0.016	0.018	0.009	0.021	0.015	0.024	0.020	0.030

Sulfides: Cobalite

Thin Section	Mineral Rpt	Mineral Actual	Fe	Cu	S	Ni	Co	Zn	Ti	As
00-367C-398	Pyrrhotite	Cobaltite	9.750	7.660	39.730	18.350	22.940	0.024	0.020	0.030
00-367C-398	Pyrrhotite	Cobaltite	10.520	5.460	41.160	19.880	22.010	0.024	0.020	0.030
00-340C-778	Unknown	Cobaltite	3.880	0.299	18.770	11.260	20.770	0.024	0.020	45.020
			3	3	3	3	3	3	3	3
Min.			3.880	0.299	18.770	11.260	20.770	0.024	0.020	0.030
P5			4.467	0.815	20.866	11.969	20.894	0.024	0.020	0.030
Median			9.750	5.460	39.730	18.350	22.010	0.024	0.020	0.030
Mean			8.050	4.473	33.220	16.497	21.907	0.024	0.020	15.027
P95			10.443	7.440	41.017	19.727	22.847	0.024	0.020	40.521
Max.			10.520	7.660	41.160	19.880	22.940	0.024	0.020	45.020
D.L.			0.016	0.018	0.009	0.021	0.015	0.024	0.020	0.030

Sulfides: Maucherite

Thin Section	Mineral Rpt	Mineral Actual	Fe	Cu	S	Ni	Co	Zn	Ti	As
00-340C-778	Unknown	Maucherite	0.562	0.077	0.026	45.880	0.396	0.024	0.020	49.550
26026-566	Unknown	Maucherite-like	0.164	0.039	0.030	52.880	1.392	0.024	0.020	38.360
26026-566	Unknown	Maucherite-like	0.349	0.051	0.046	52.080	1.676	0.024	0.020	38.280
n			3	3	3	3	3	3	3	3
Min.			0.164	0.039	0.026	45.880	0.396	0.024	0.020	38.280
P5			0.182	0.040	0.026	46.500	0.496	0.024	0.020	38.288
Median			0.349	0.051	0.030	52.080	1.392	0.024	0.020	38.360
Mean			0.358	0.055	0.034	50.280	1.155	0.024	0.020	42.063
P95			0.540	0.074	0.044	52.800	1.648	0.024	0.020	48.431
Max.			0.562	0.077	0.046	52.880	1.676	0.024	0.020	49.550
D.L.			0.016	0.018	0.009	0.021	0.015	0.024	0.020	0.030

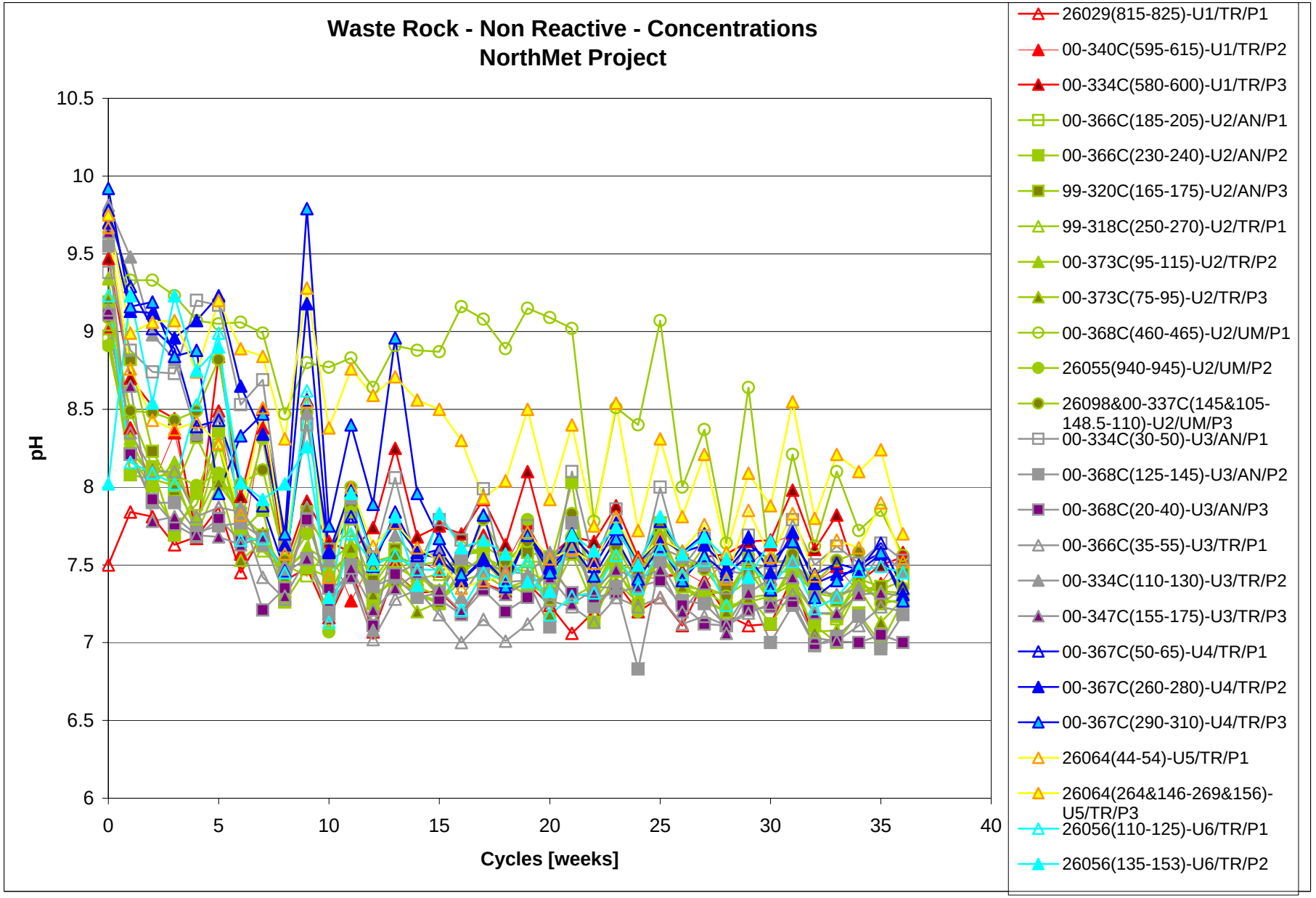
Sulfides: Pyrite

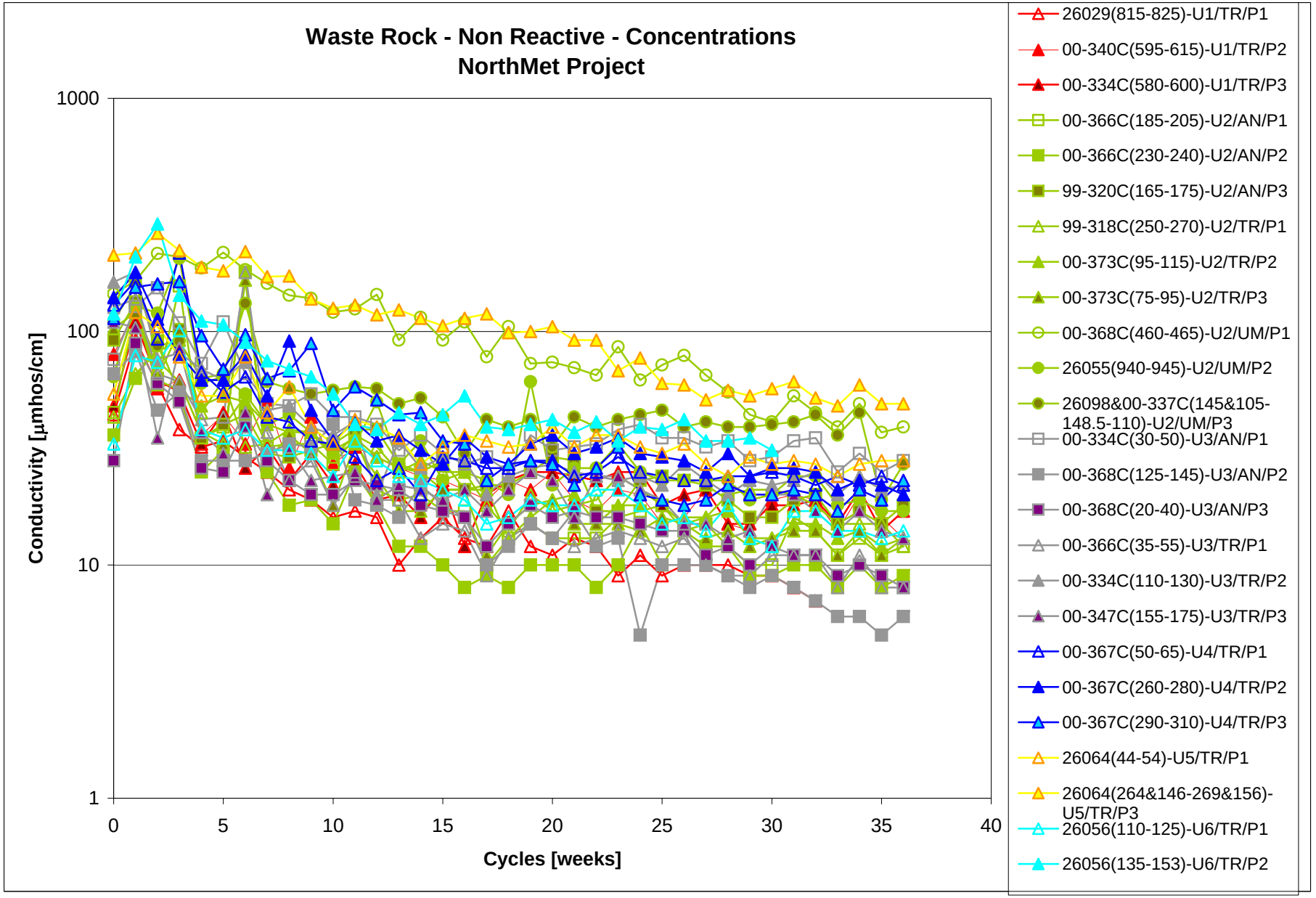
Thin Section	Mineral Rpt	Mineral Actual	Fe	Cu	S	Ni	Co	Zn	Ti	As
00-367C-398	Violarite	Pyrite	46.200	0.091	53.560	1.011	0.089	0.014	0.000	0.001
00-367C-398	Pentlandite	Pyrite	46.700	0.312	52.070	0.177	0.054	0.009	0.003	0.000
00-367C-398	Chalcopyrite	Pyrite	47.320	0.217	53.350	0.062	0.060	0.000	0.000	0.000
00-367C-398	Pentlandite	Pyrite	47.390	0.044	52.570	0.020	0.051	0.000	0.000	0.003
00-367C-398	Pentlandite	Pyrite	47.010	0.043	52.960	0.260	0.108	0.000	0.000	0.000
26056-311	Pentlandite	Pyrite	45.690	0.043	50.460	1.219	0.052	0.013	0.000	0.042
26056-311	Pentlandite	Pyrite	46.330	0.034	53.910	0.675	0.043	0.004	0.000	0.000
00-367C-401	Pentlandite	Pyrite	46.590	0.532	53.590	0.623	0.041	0.011	0.004	0.003
n			8	8	8	8	8	8	8	8
Min.			45.690	0.034	50.460	0.020	0.041	0.000	0.000	0.000
P5			45.869	0.037	51.024	0.035	0.042	0.000	0.000	0.000
Median			46.645	0.068	53.155	0.442	0.053	0.006	0.000	0.001
Mean			46.654	0.164	52.809	0.506	0.062	0.006	0.001	0.006
P95			47.366	0.455	53.798	1.147	0.101	0.014	0.004	0.028
Max.			47.390	0.532	53.910	1.219	0.108	0.014	0.004	0.042
D.L.			0.016	0.018	0.009	0.021	0.015	0.024	0.020	0.030

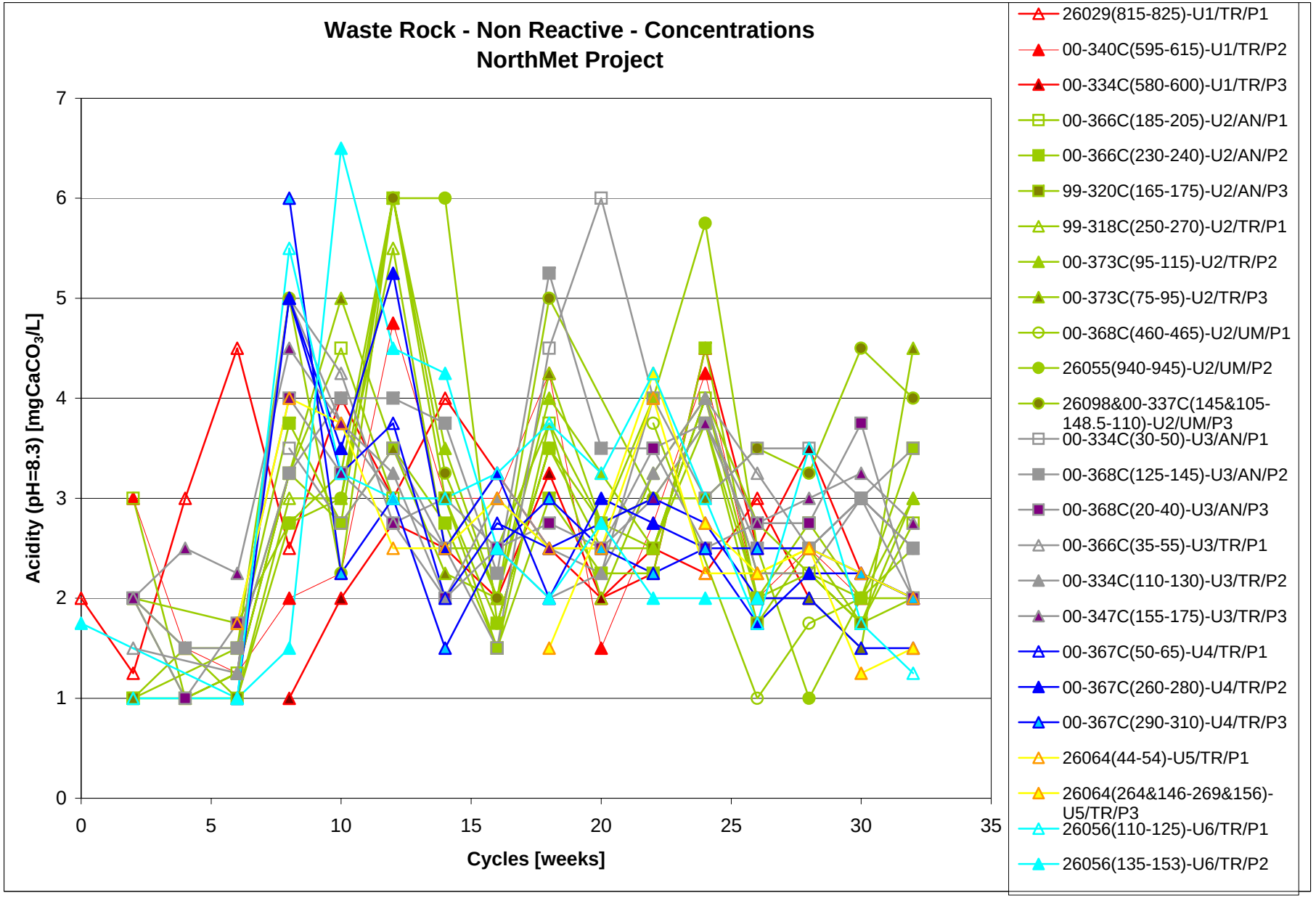
Sulfides: Sphalerite

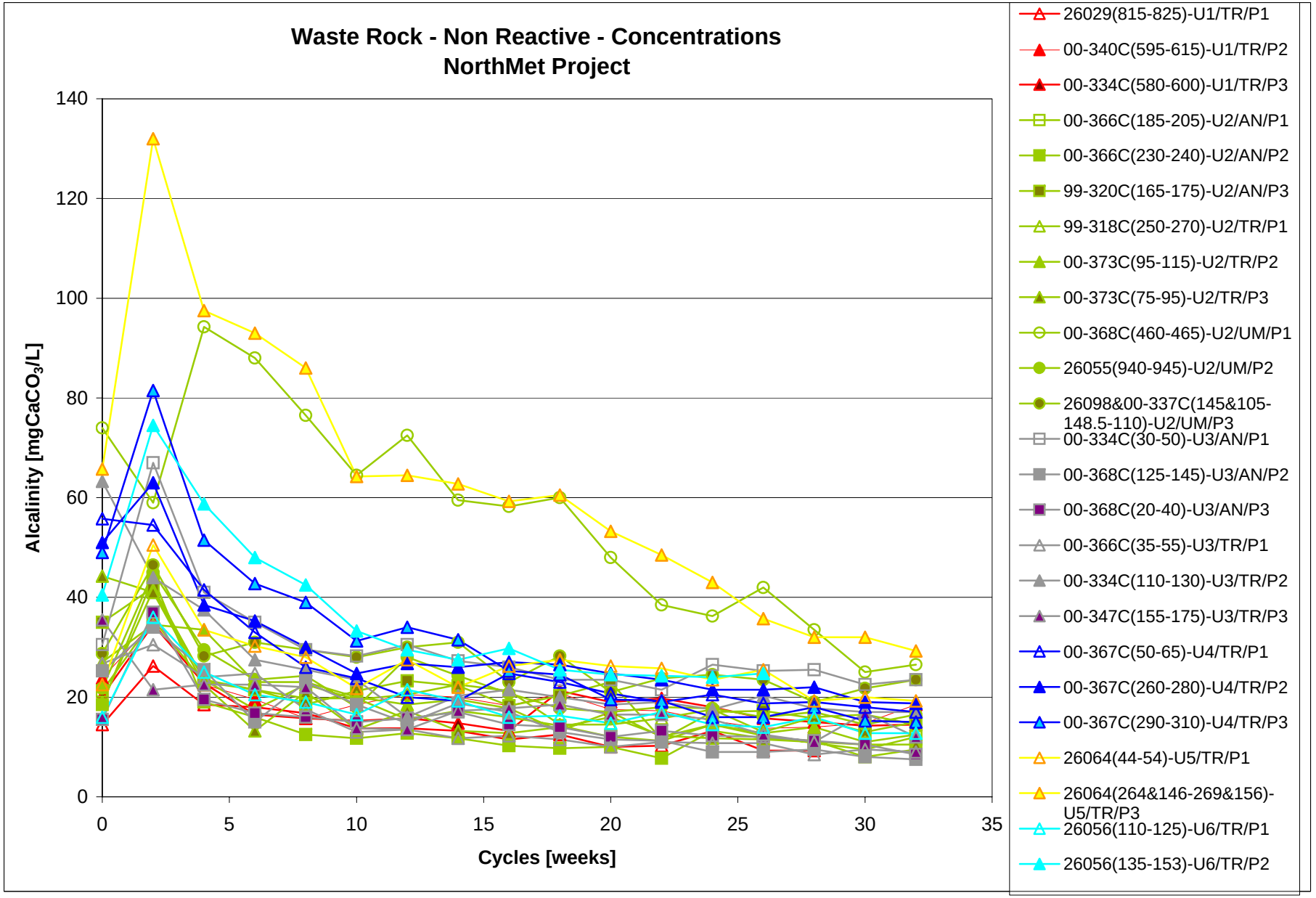
Thin Section	Mineral Rpt	Mineral Actual	Fe	Cu	S	Ni	Co	Zn	Ti	As
00-367C-398	Sphalerite	Sphalerite	9.630	0.103	32.130	0.065	0.111	57.170	0.020	0.030

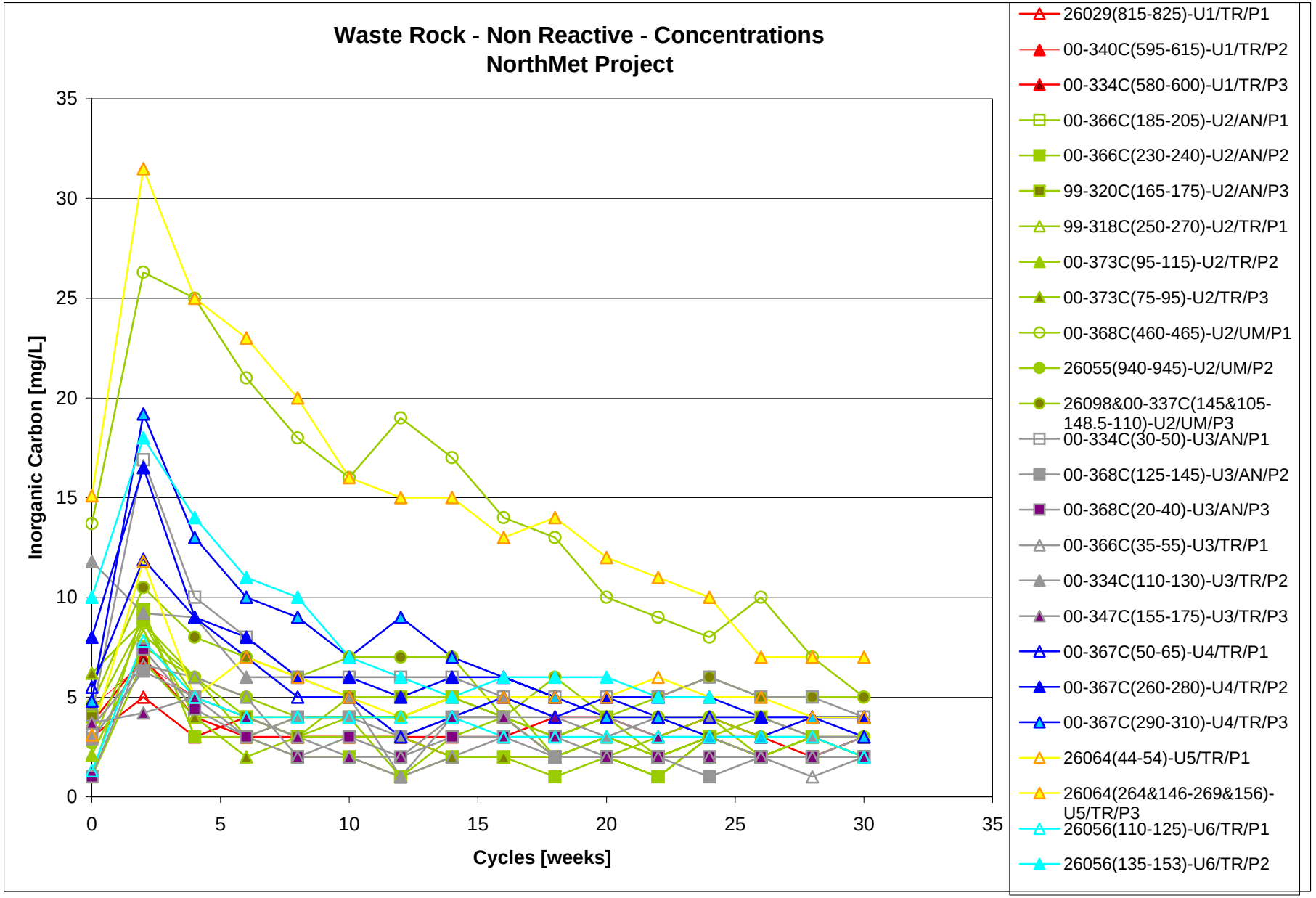
Appendix E
Leachate Chemistry Graphs – Non-Reactive Waste Rock

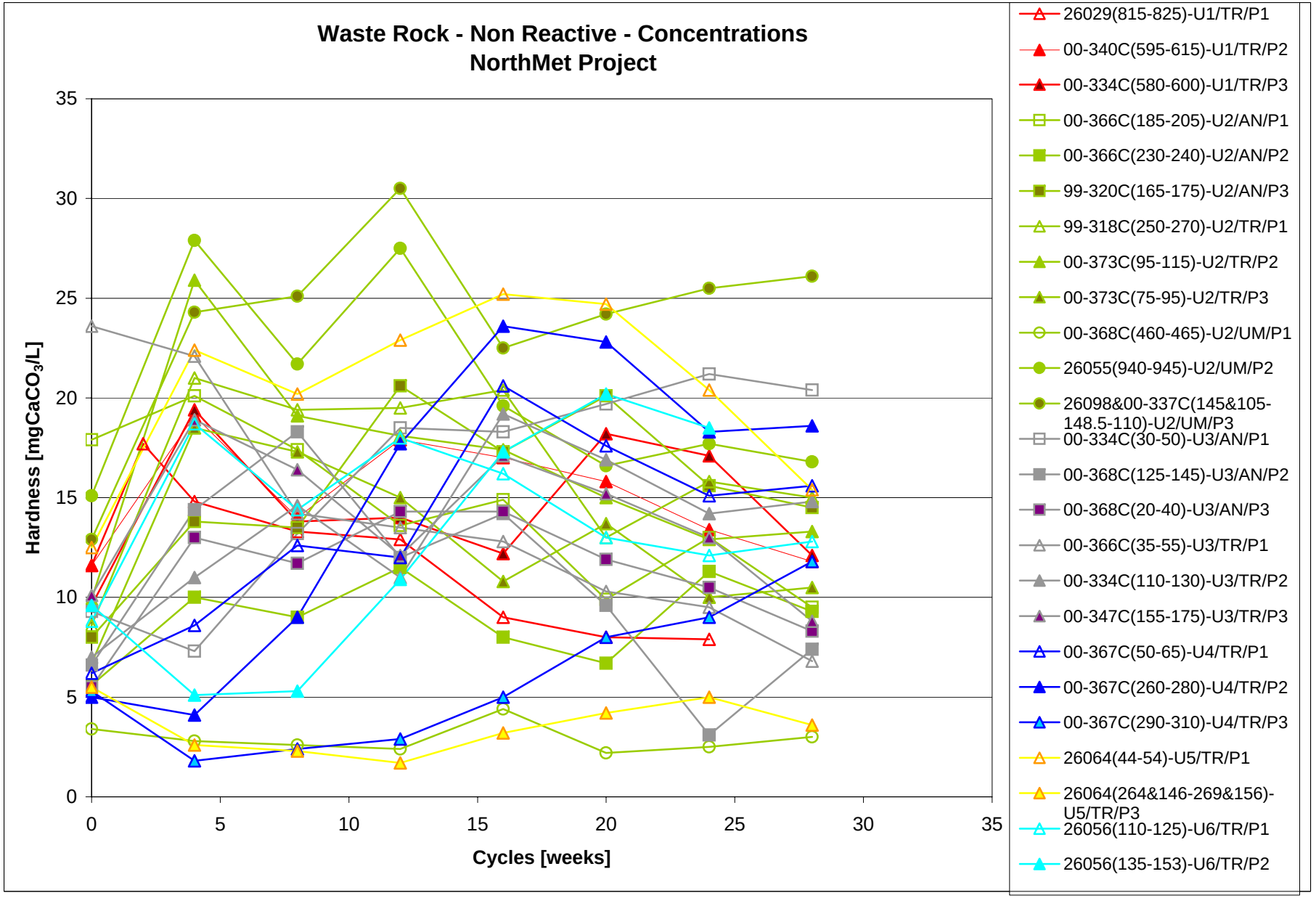

Waste Rock - Non Reactive - Concentrations
NorthMet Project
pH
Cycles [weeks]
10.5
10
9.5
9
8.5
8
7.5
7
6.5
6
0
5
10
15
20
25
30
35
40
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

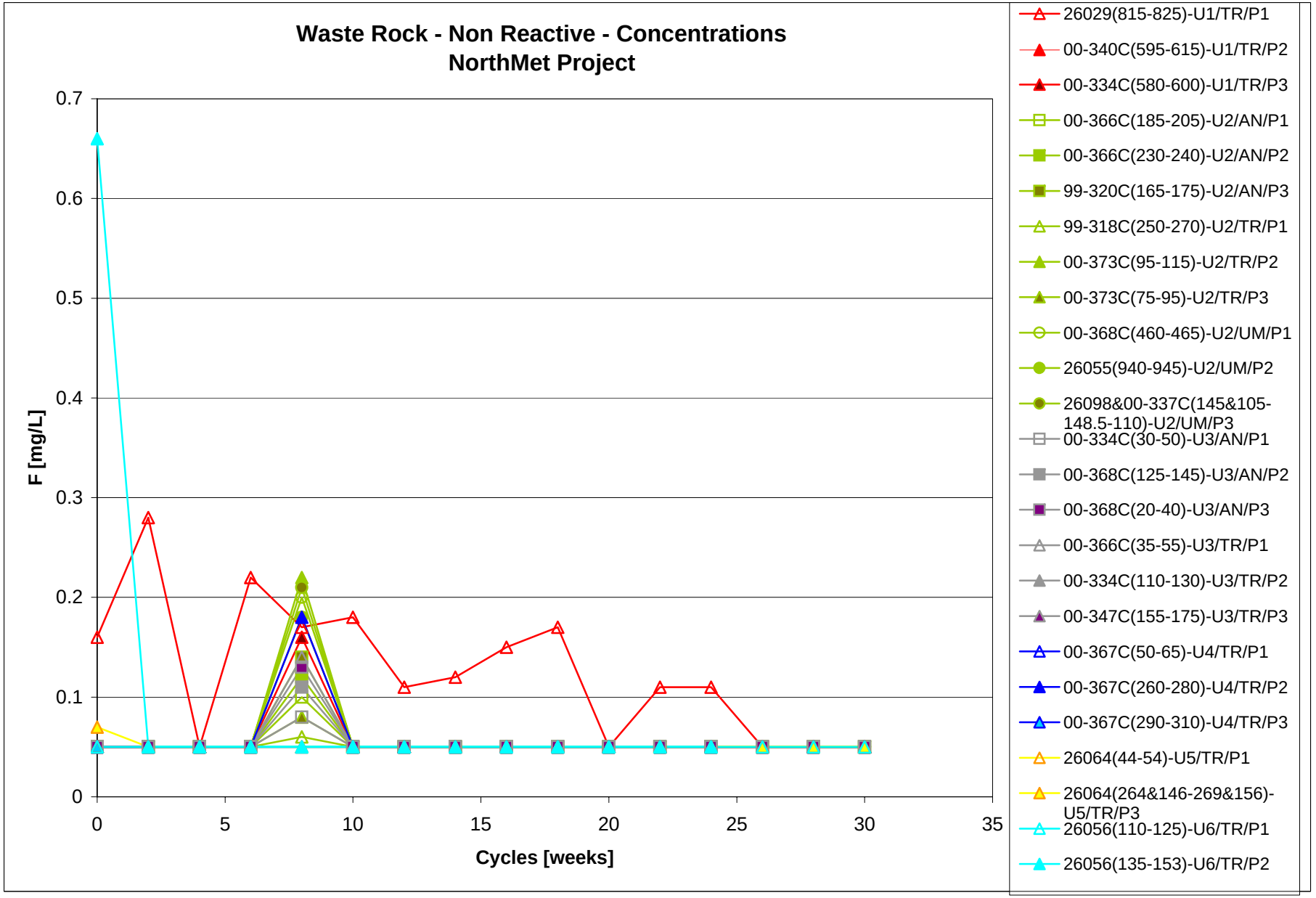

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Conductivity [µmhos/cm]
1000
100
10
1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

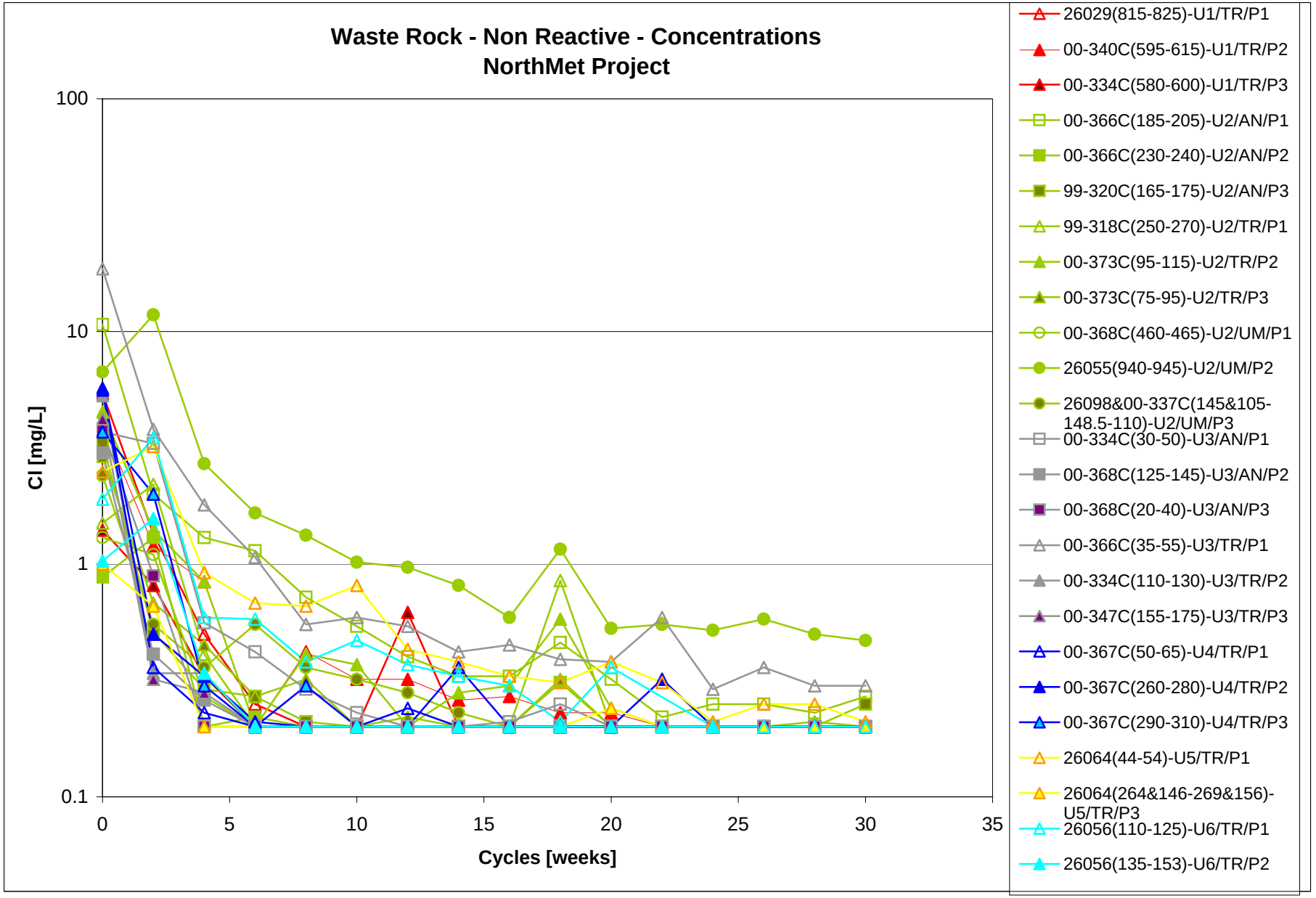

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Acidity (pH=8.3) [mgCaCO3/L]
Cycles [weeks]
0
1
2
3
4
5
6
7
0
5
10
15
20
25
30
35
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

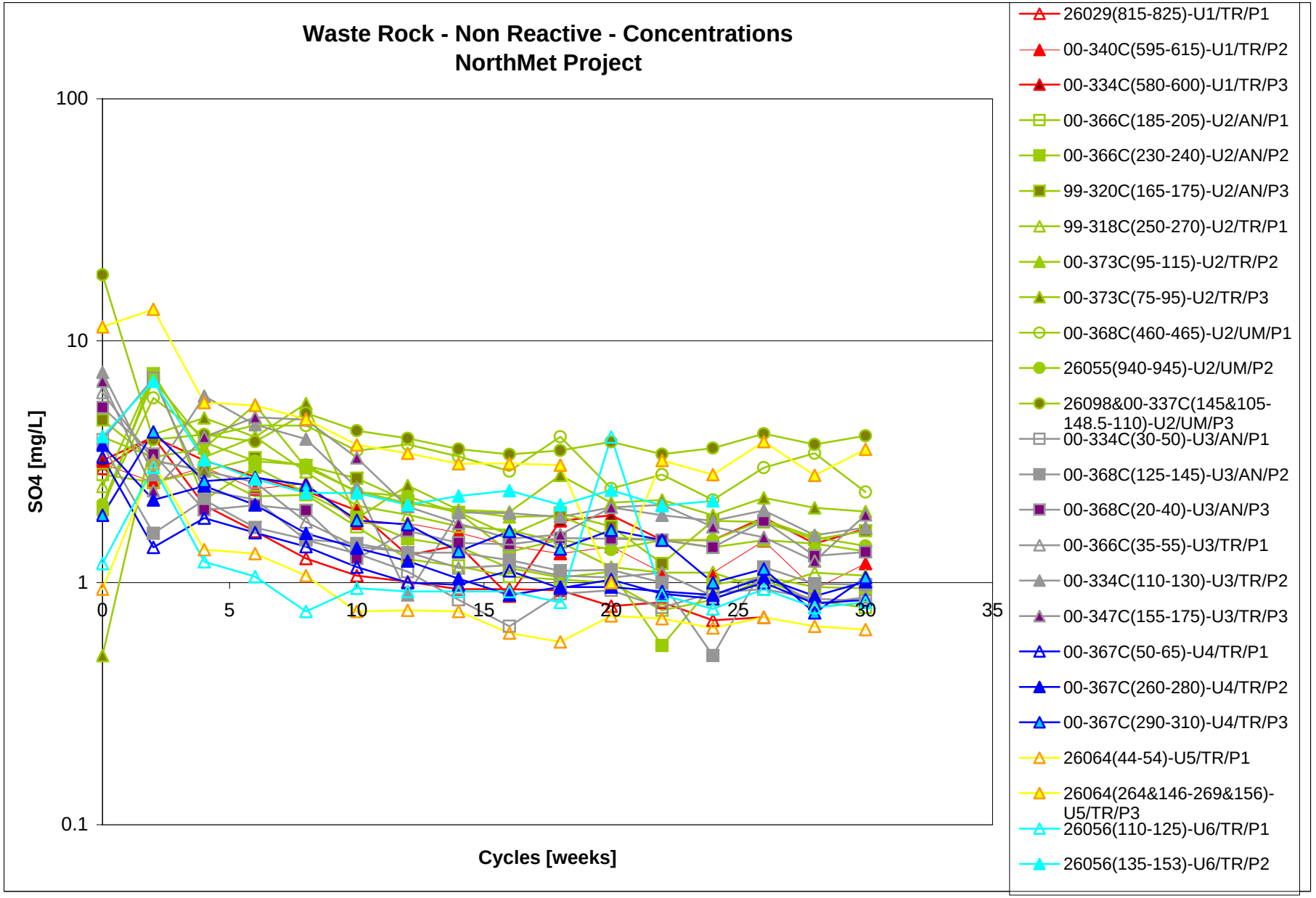

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Alcalinity [mgCaCO3/L]
Cycles [weeks]
0
20
40
60
80
100
120
140
0
5
10
15
20
25
30
35
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

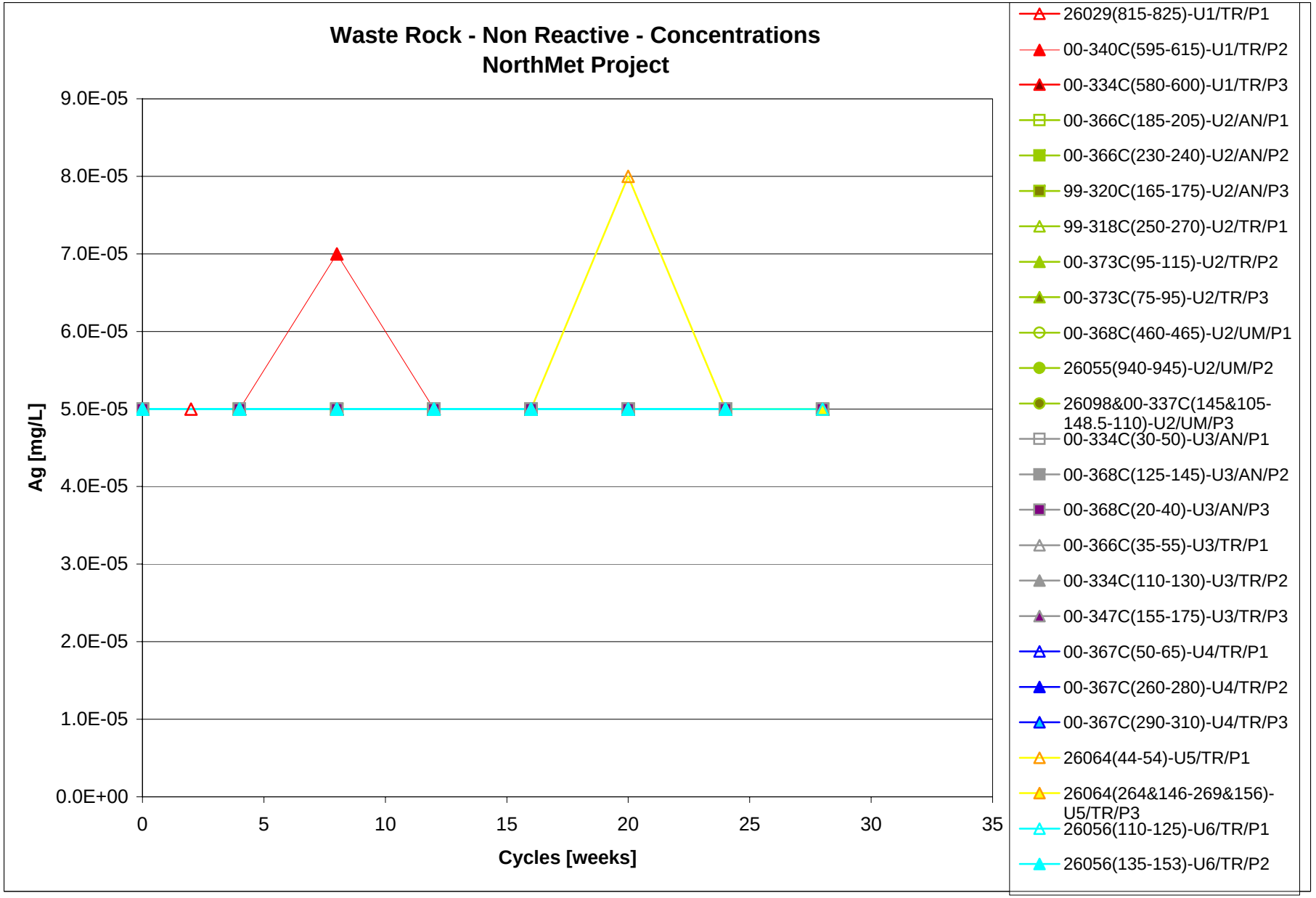

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Inorganic Carbon [mg/L]
Cycles [weeks]
0
5
10
15
20
25
30
35
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

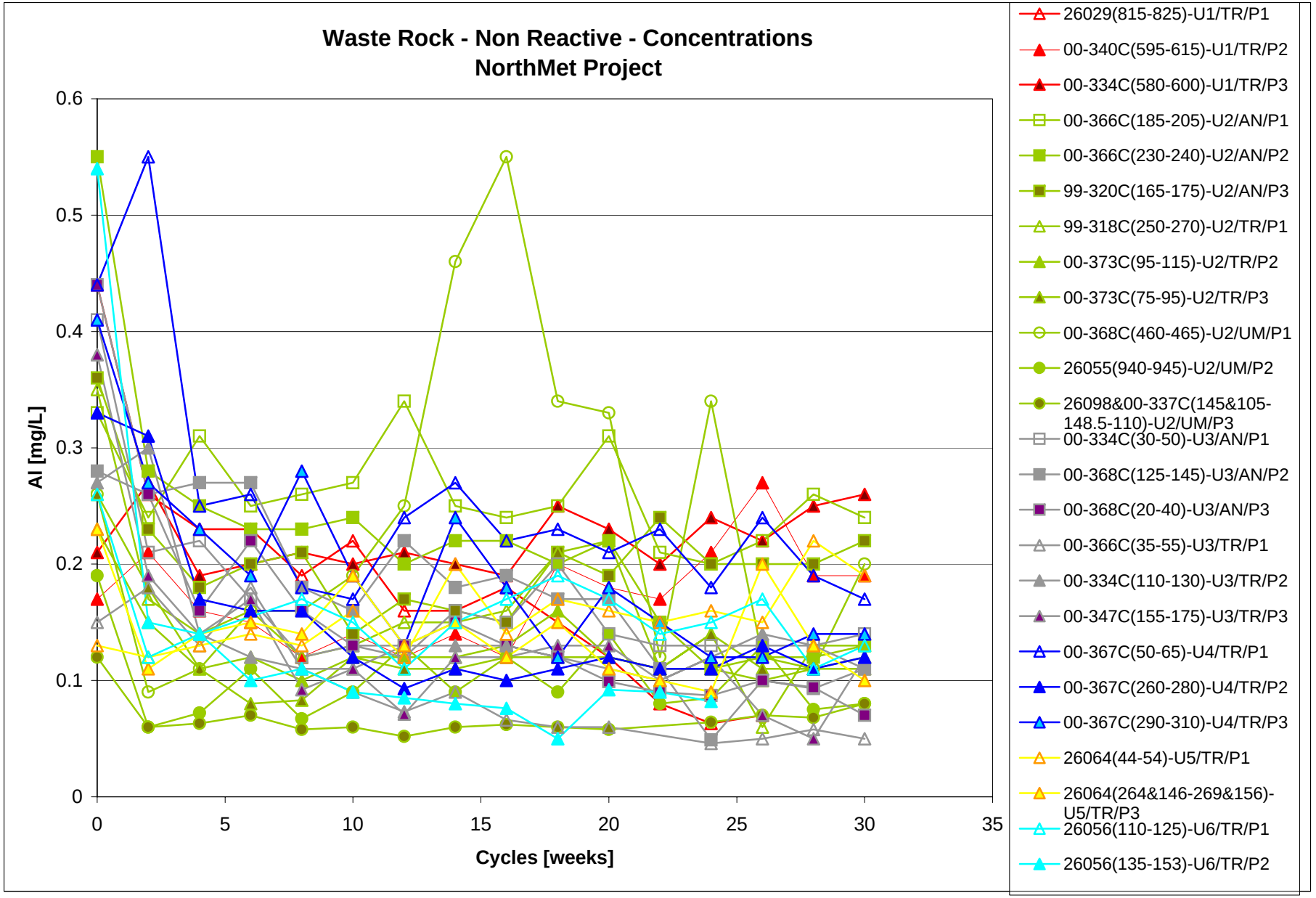

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Hardness [mgCaCO3/L]
Cycles [weeks]
0
5
10
15
20
25
30
35
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

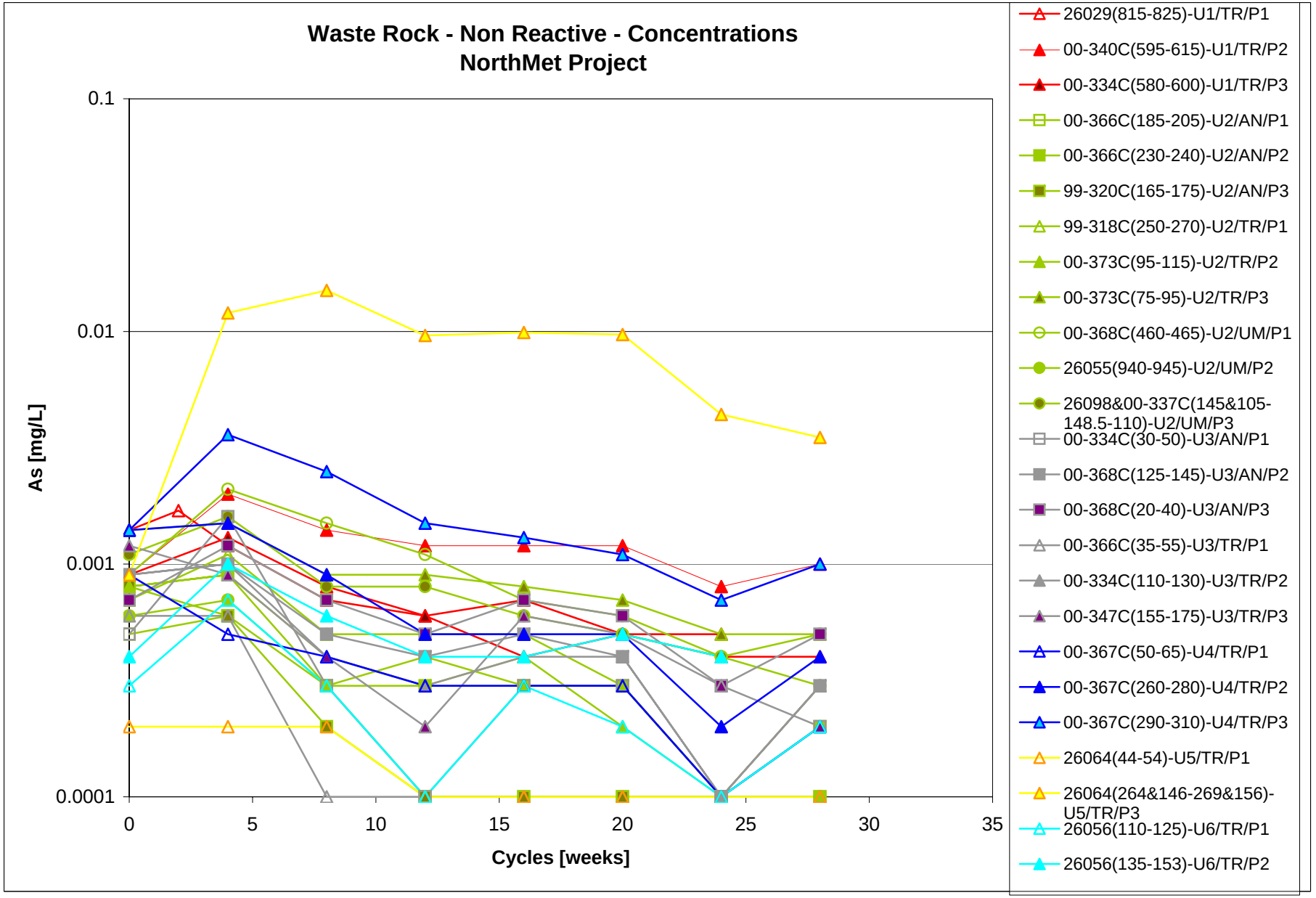

Waste Rock - Non Reactive - Concentrations
NorthMet Project
F [mg/L]
0.7
0.6
0.5
0.4
0.3
0.2
0.1
0
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

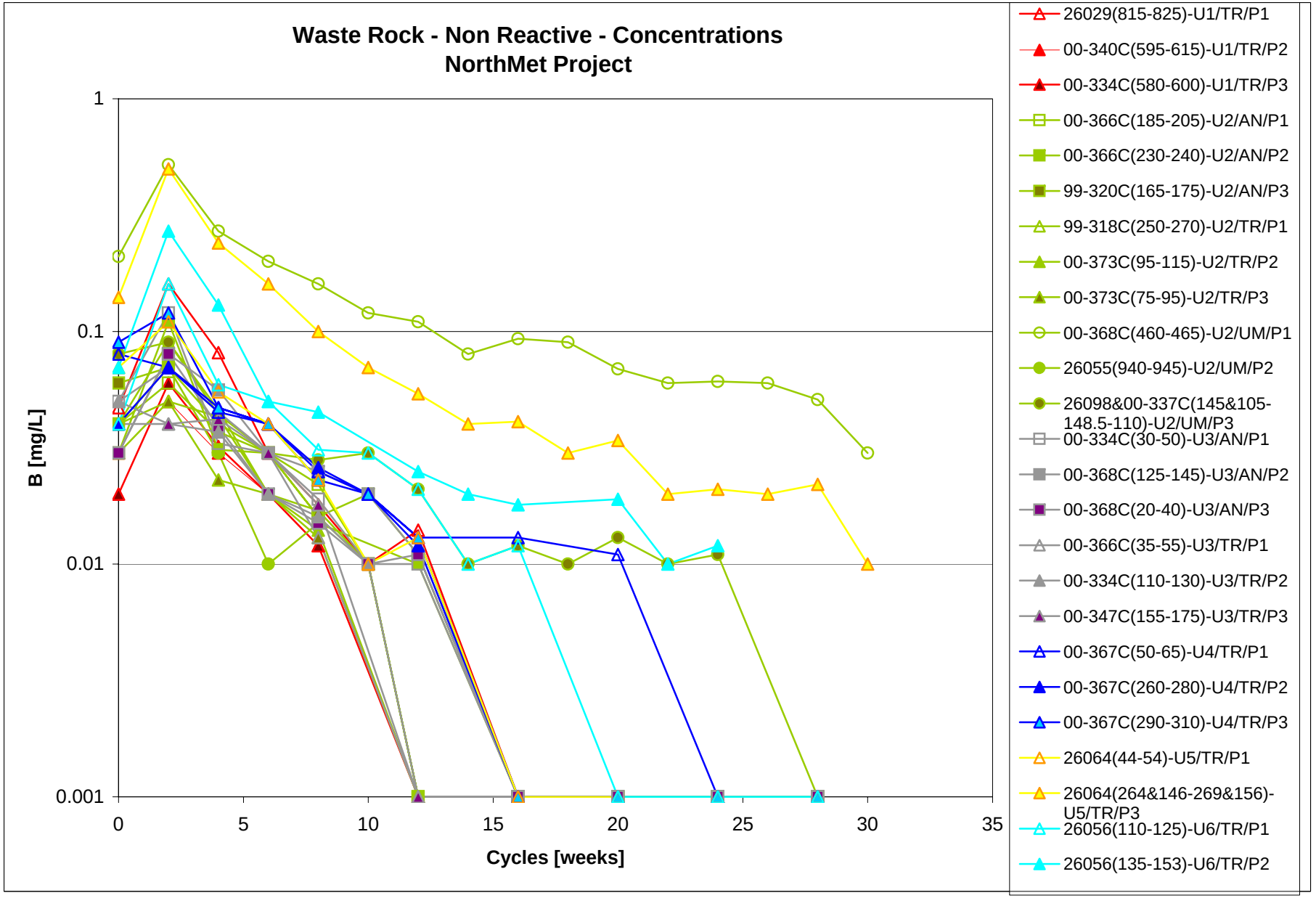

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Cl [mg/L]
100
10
1
0.1
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

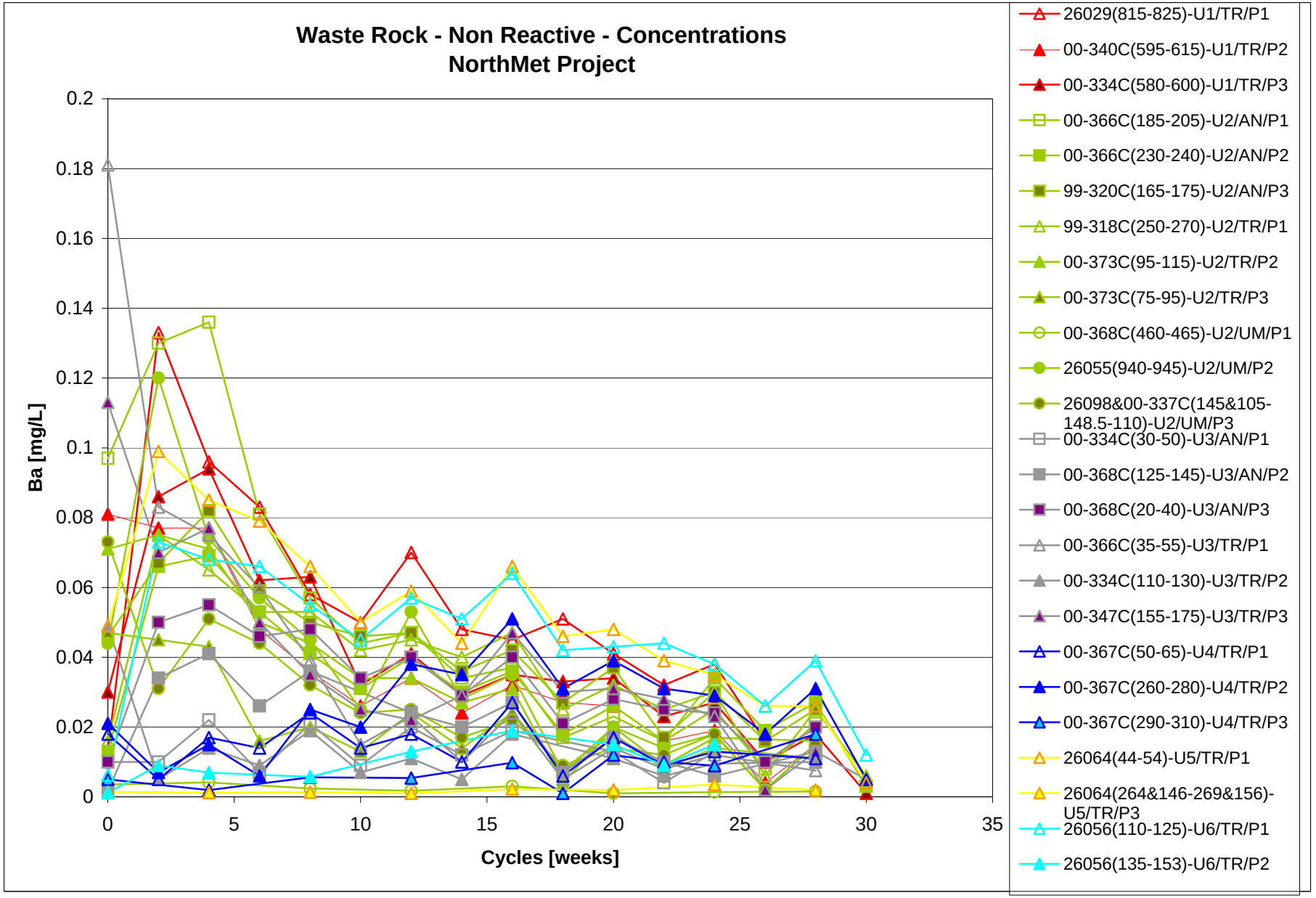

Waste Rock - Non Reactive - Concentrations
NorthMet Project
SO4 [mg/L]
100
10
1
0.1
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

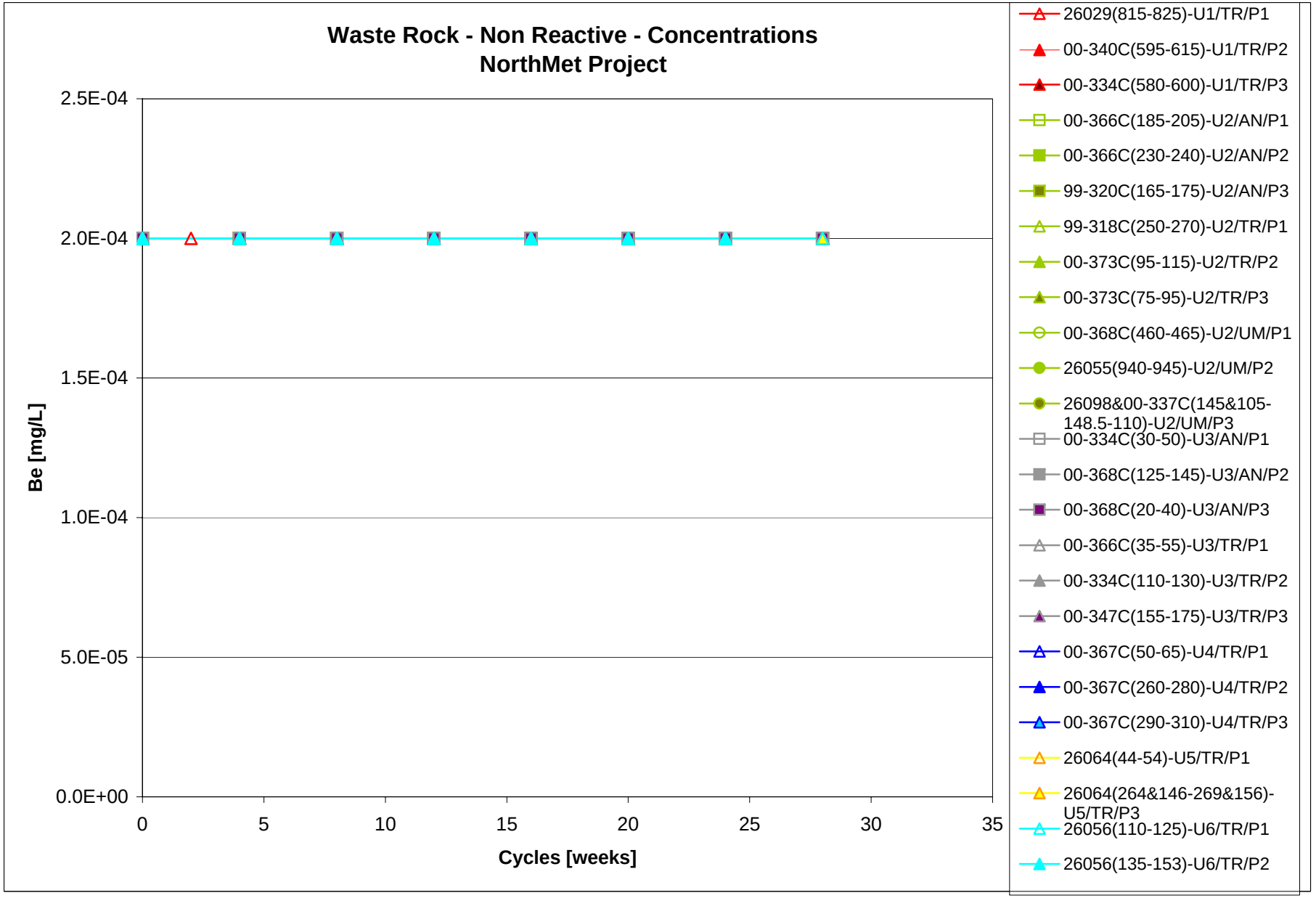

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Ag [mg/L]
9.0E-05
8.0E-05
7.0E-05
6.0E-05
5.0E-05
4.0E-05
3.0E-05
2.0E-05
1.0E-05
0.0E+00
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

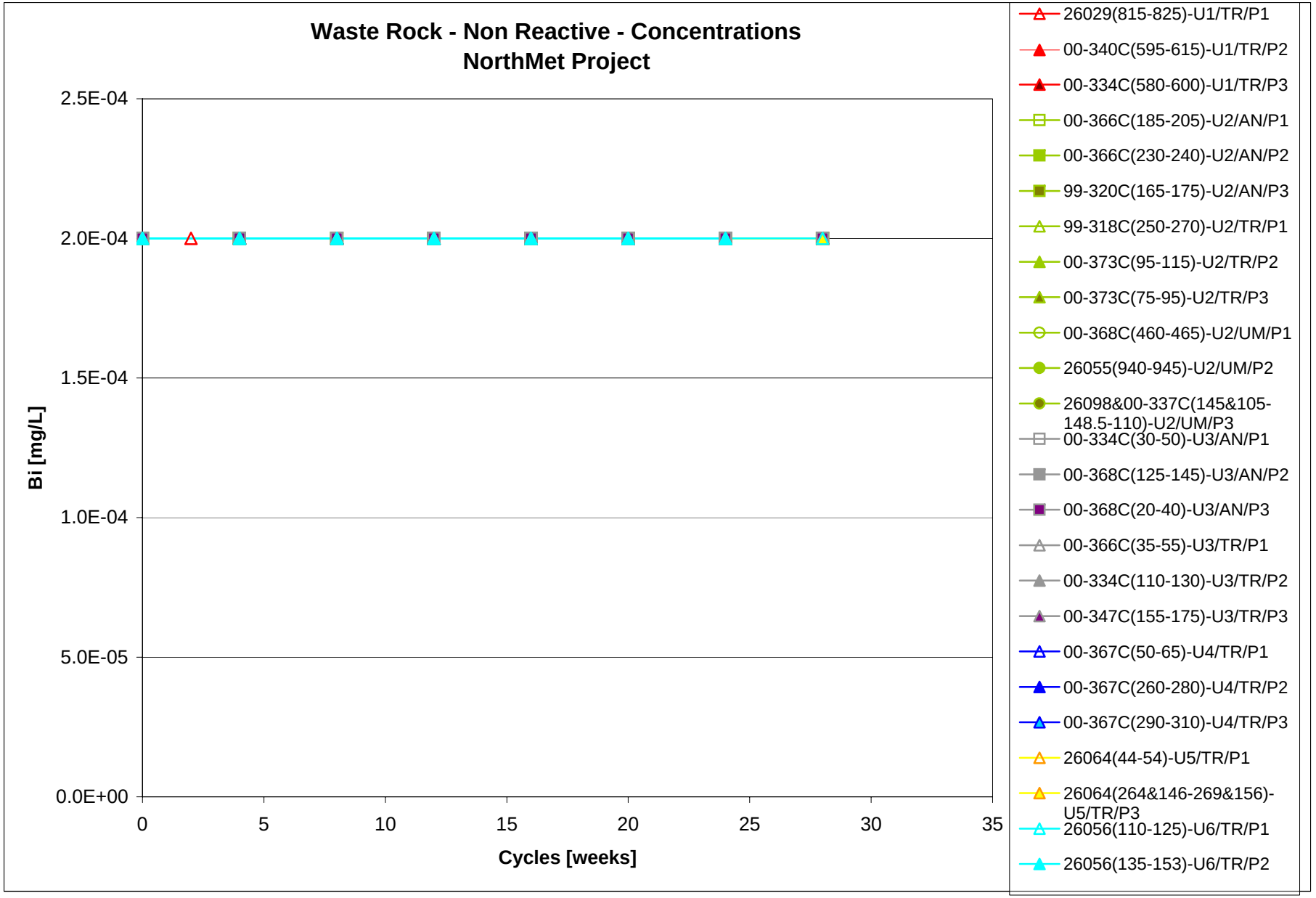

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Al [mg/L]
Cycles [weeks]
0
0.1
0.2
0.3
0.4
0.5
0.6
0
5
10
15
20
25
30
35
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

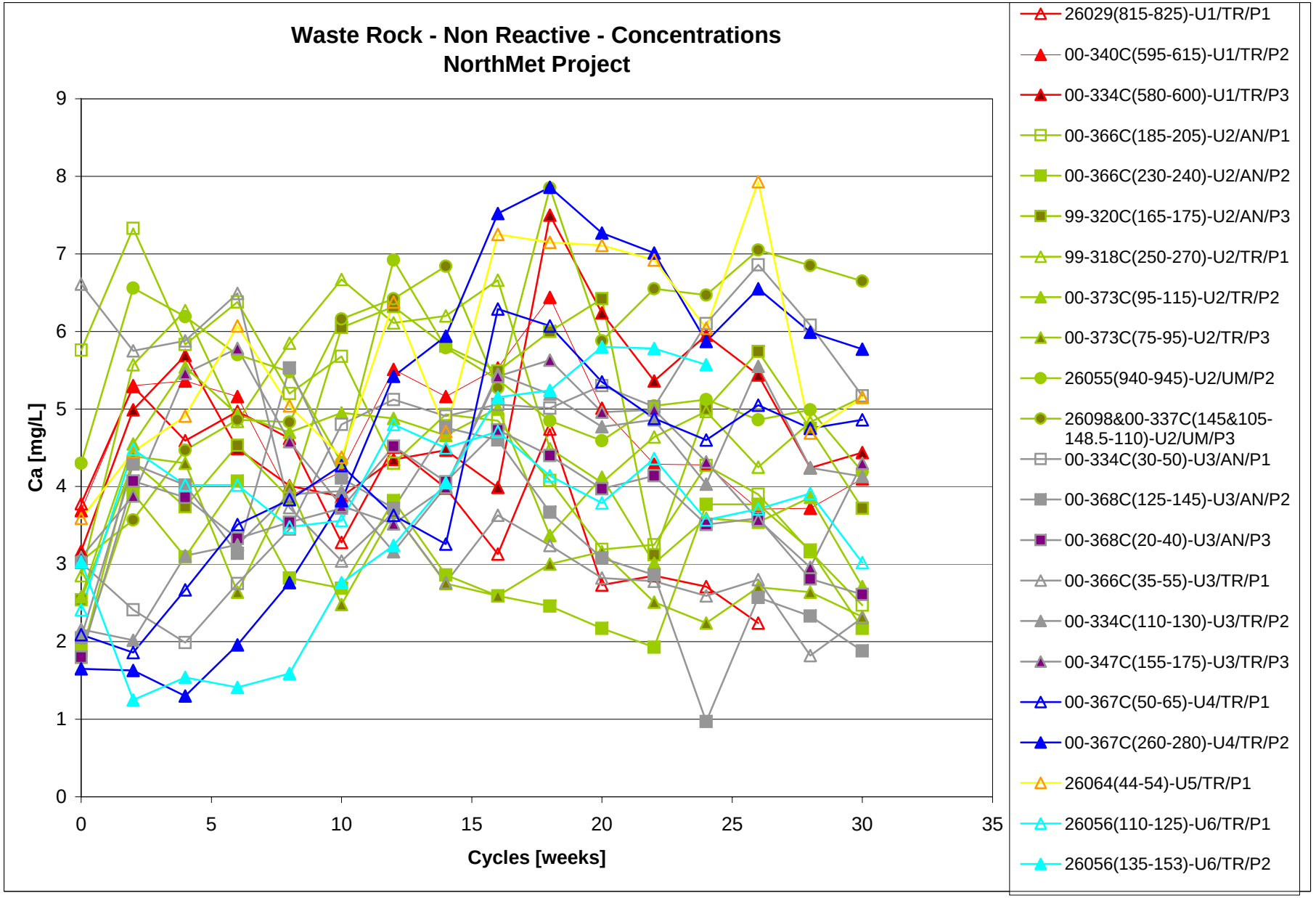

Waste Rock - Non Reactive - Concentrations
NorthMet Project
As [mg/L]
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

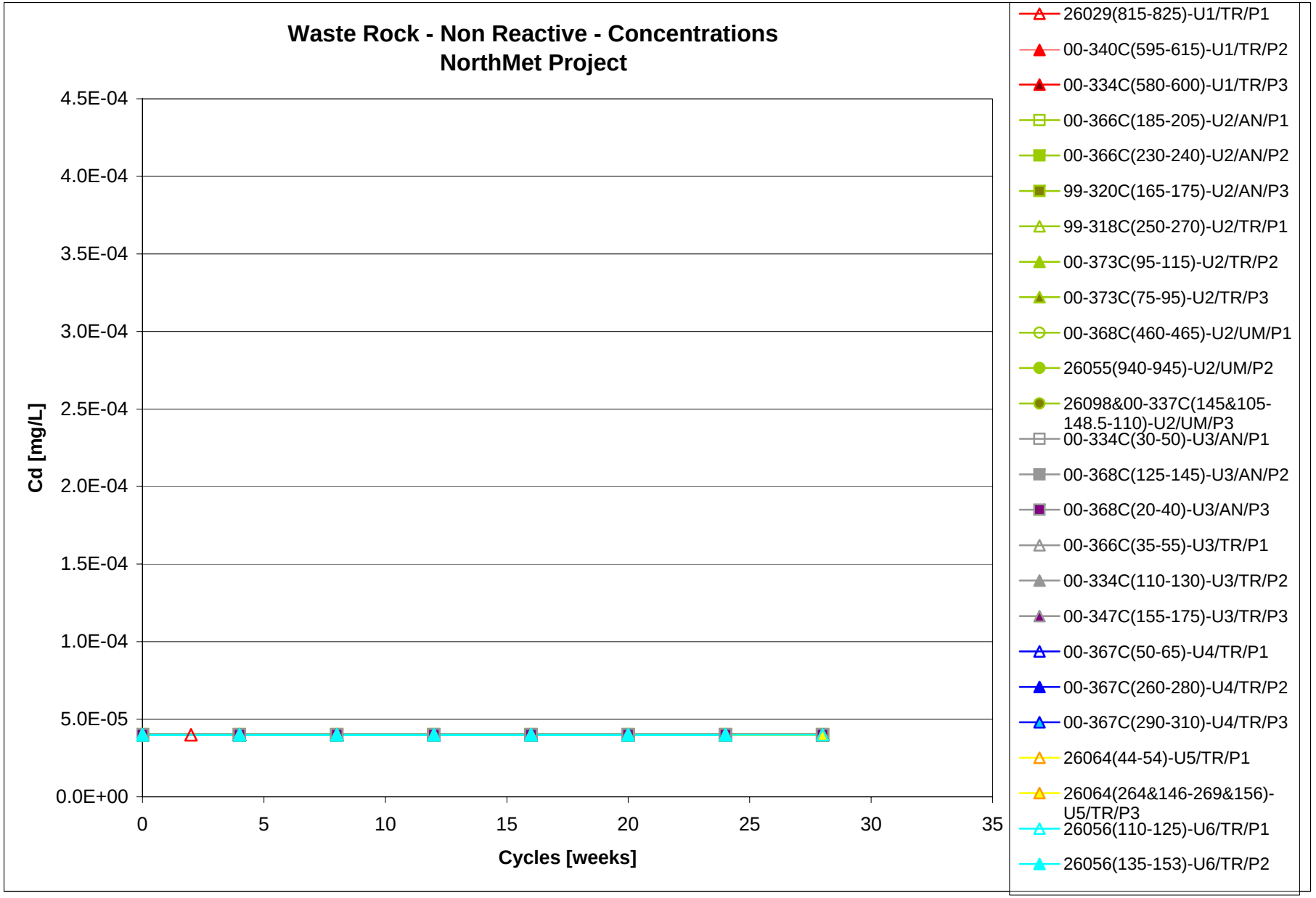

Waste Rock - Non Reactive - Concentrations
NorthMet Project
B [mg/L]
1
0.1
0.01
0.001
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

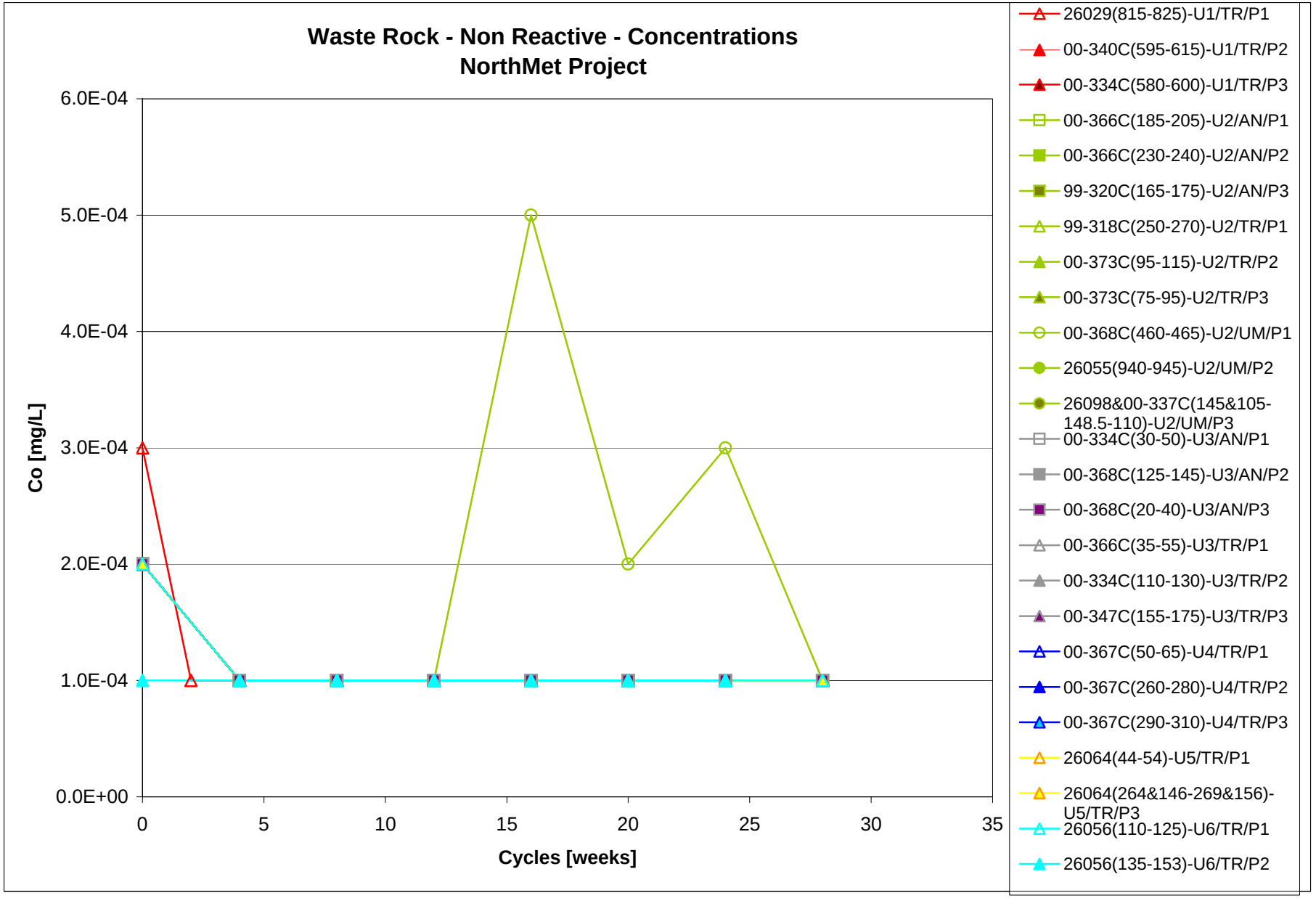

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Ba [mg/L]
0.2
0.18
0.16
0.14
0.12
0.1
0.08
0.06
0.04
0.02
0
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

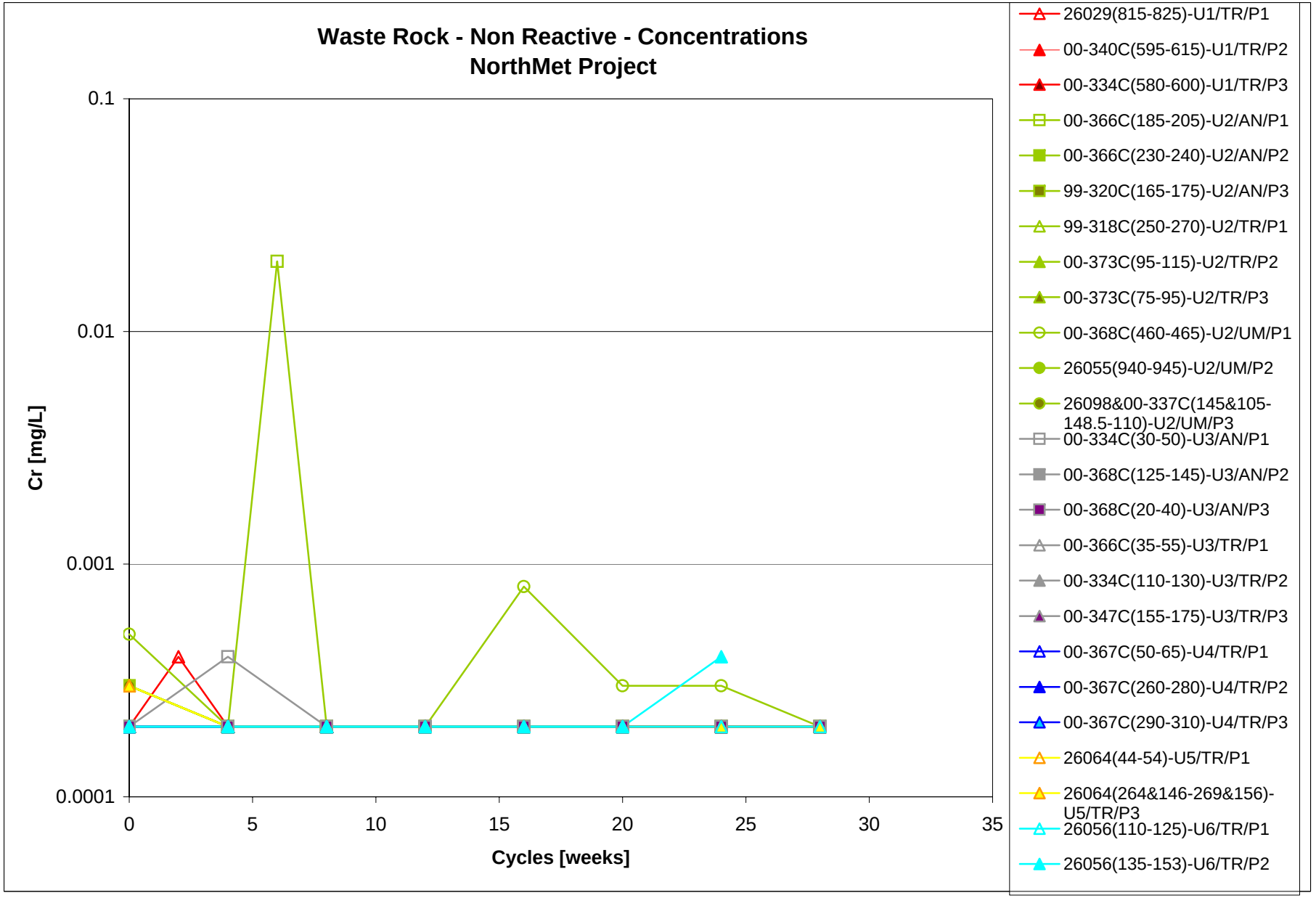

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Be [mg/L]
2.5E-04
2.0E-04
1.5E-04
1.0E-04
5.0E-05
0.0E+00
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

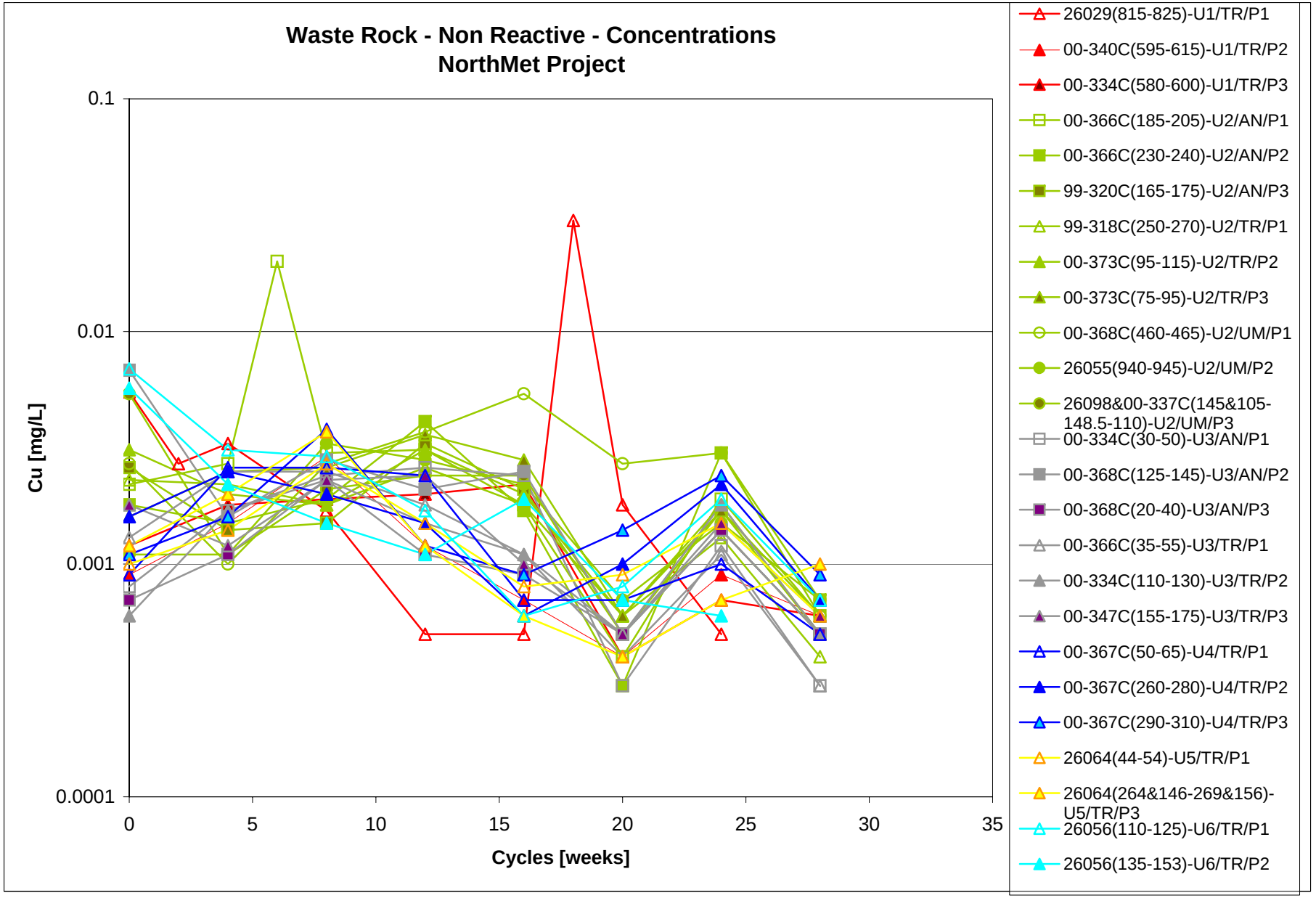

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Bi [mg/L]
2.5E-04
2.0E-04
1.5E-04
1.0E-04
5.0E-05
0.0E+00
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

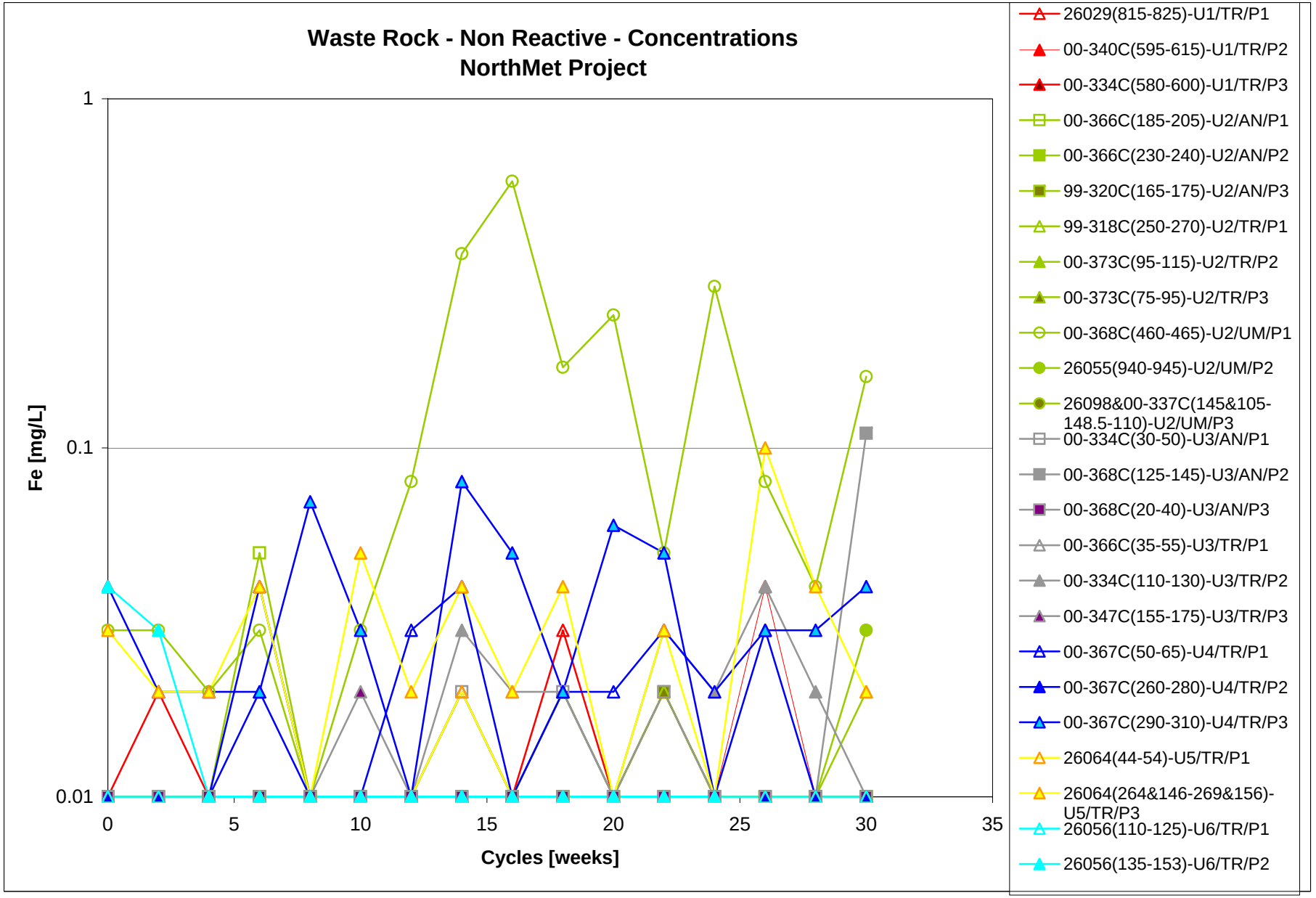

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Ca [mg/L]
Cycles [weeks]
0
1
2
3
4
5
6
7
8
9
0
5
10
15
20
25
30
35
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
26064(44-54)-U5/TR/P1
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

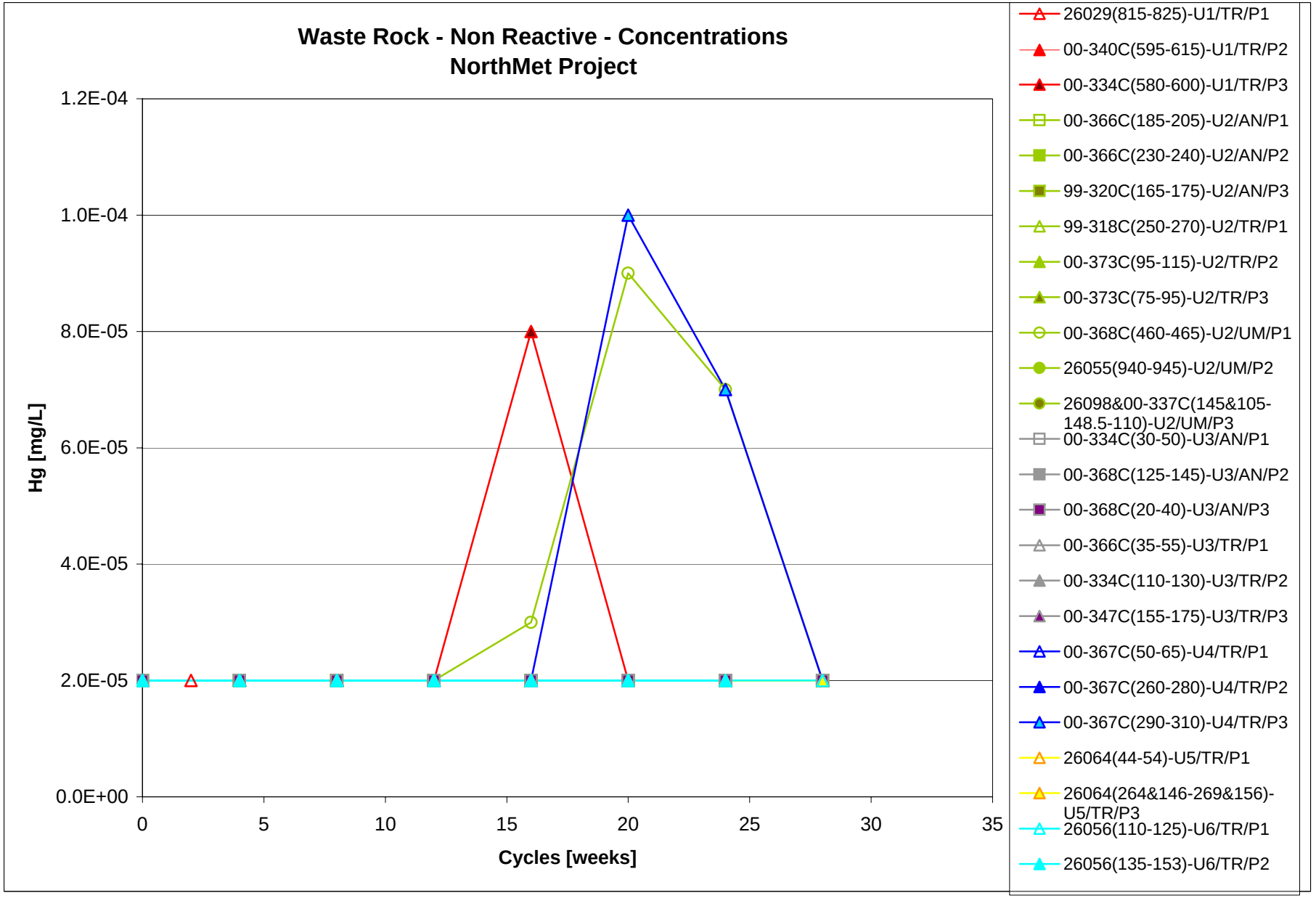

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Cd [mg/L]
4.5E-04
4.0E-04
3.5E-04
3.0E-04
2.5E-04
2.0E-04
1.5E-04
1.0E-04
5.0E-05
0.0E+00
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

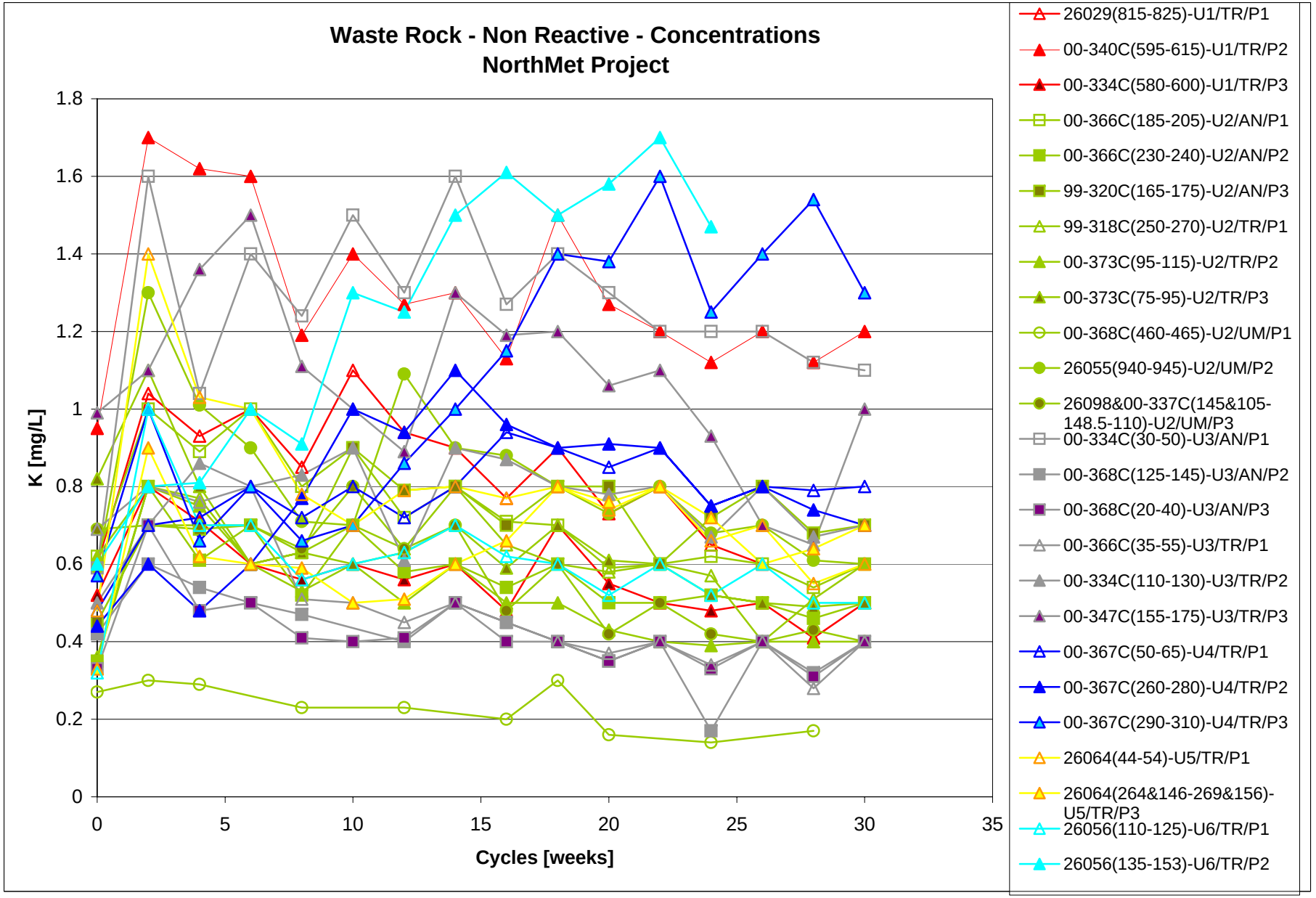

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Co [mg/L]
6.0E-04
5.0E-04
4.0E-04
3.0E-04
2.0E-04
1.0E-04
0.0E+00
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

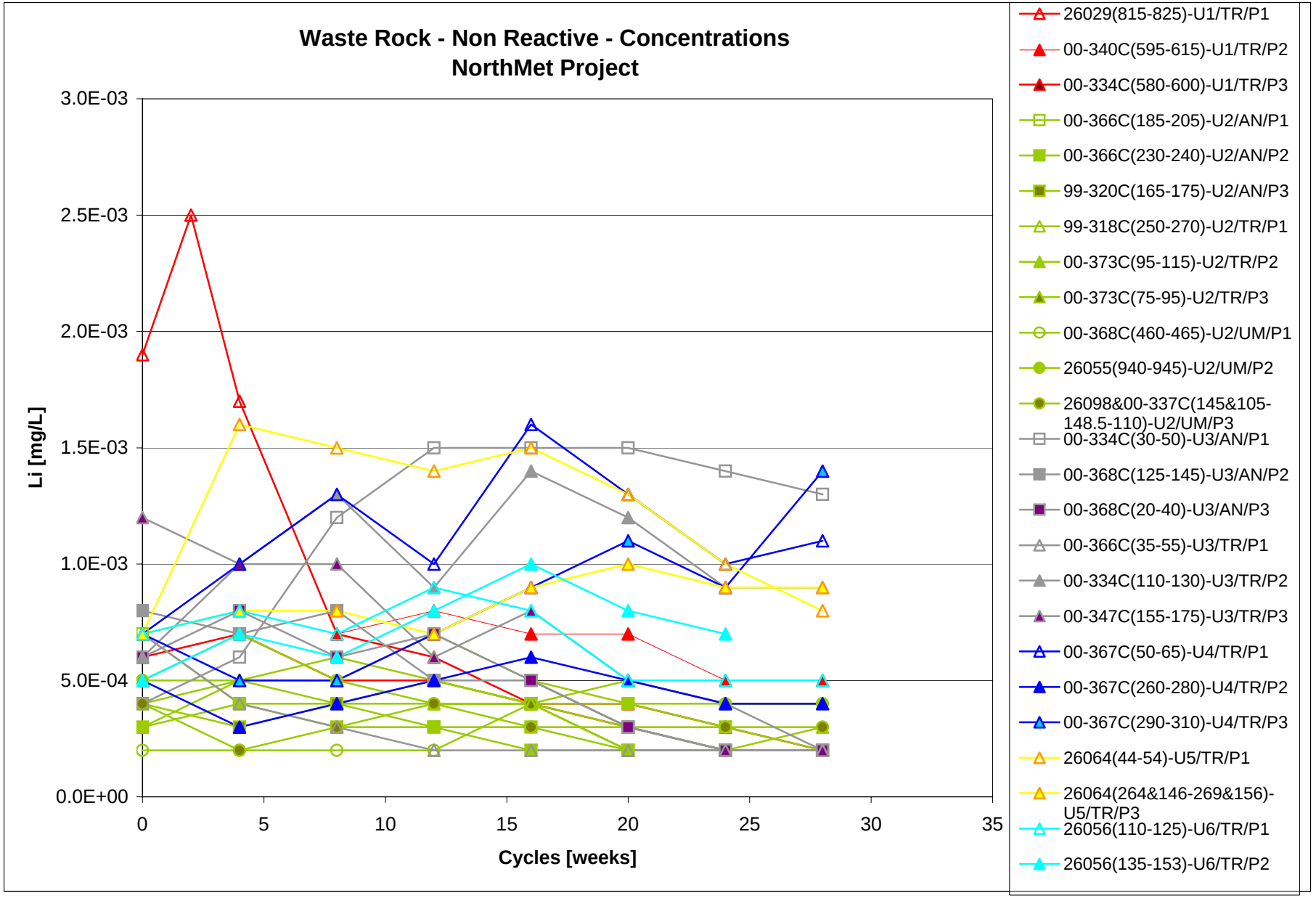

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Cr [mg/L]
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

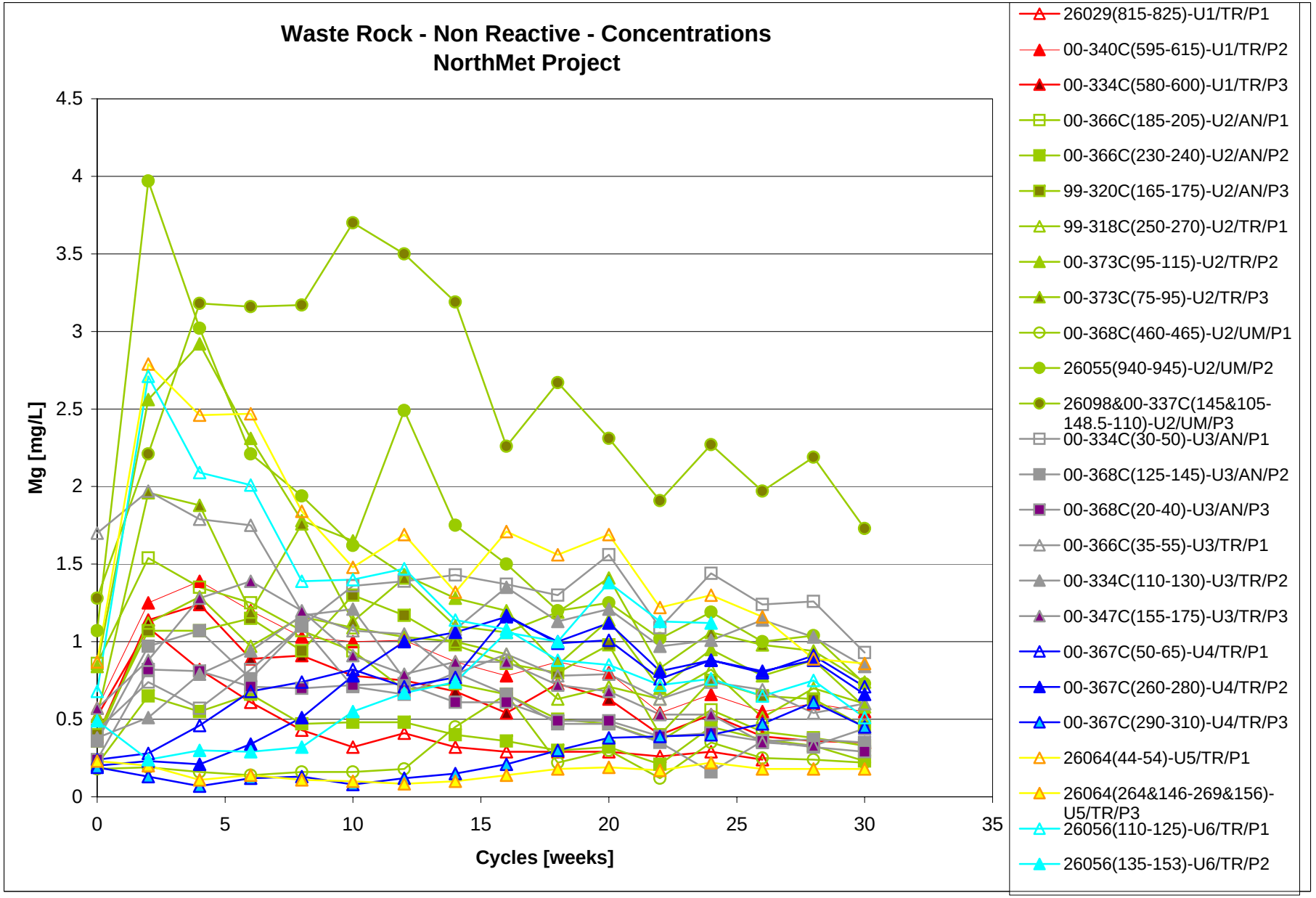

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Cu [mg/L]
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

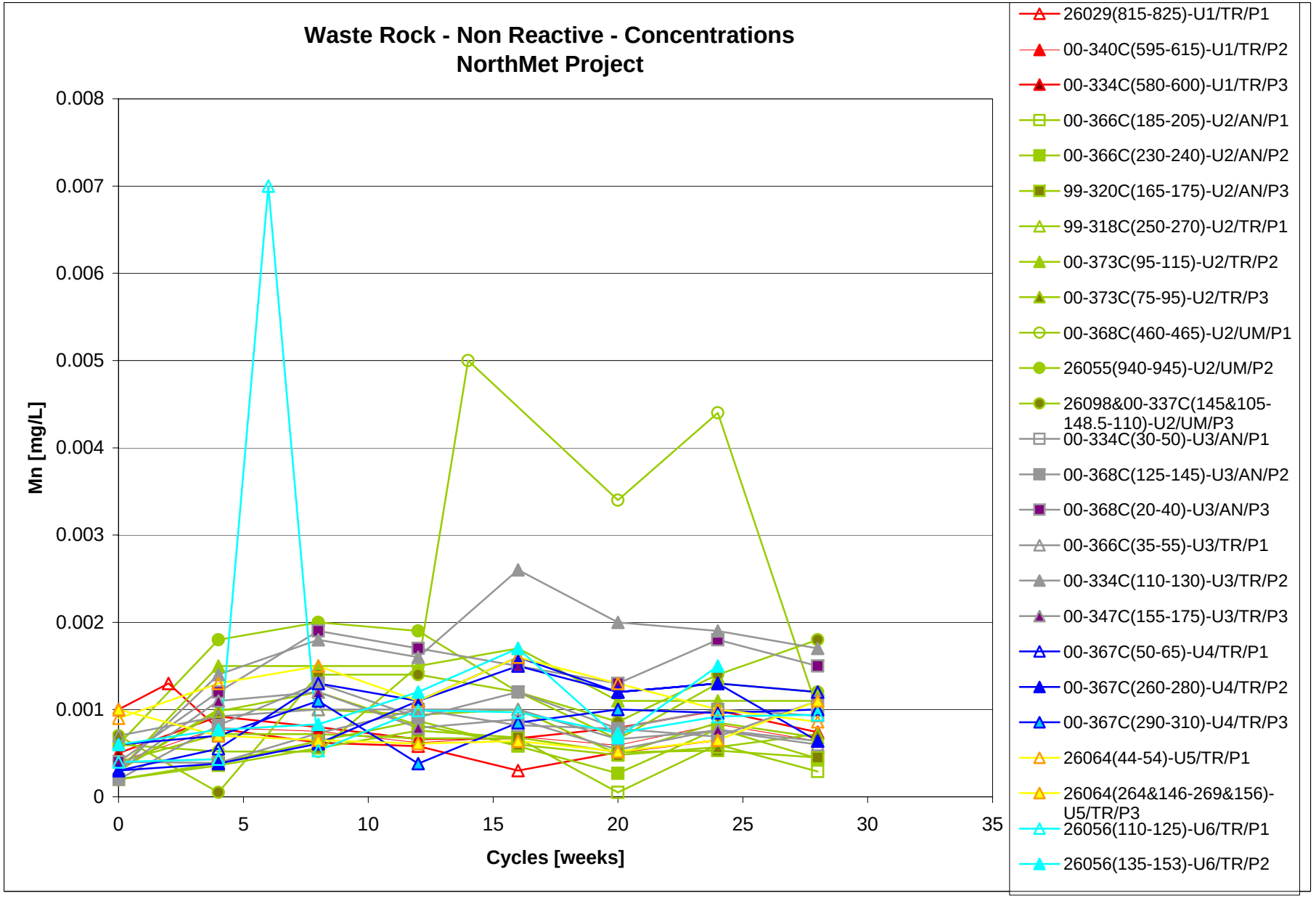

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Fe [mg/L]
1
0.1
0.01
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

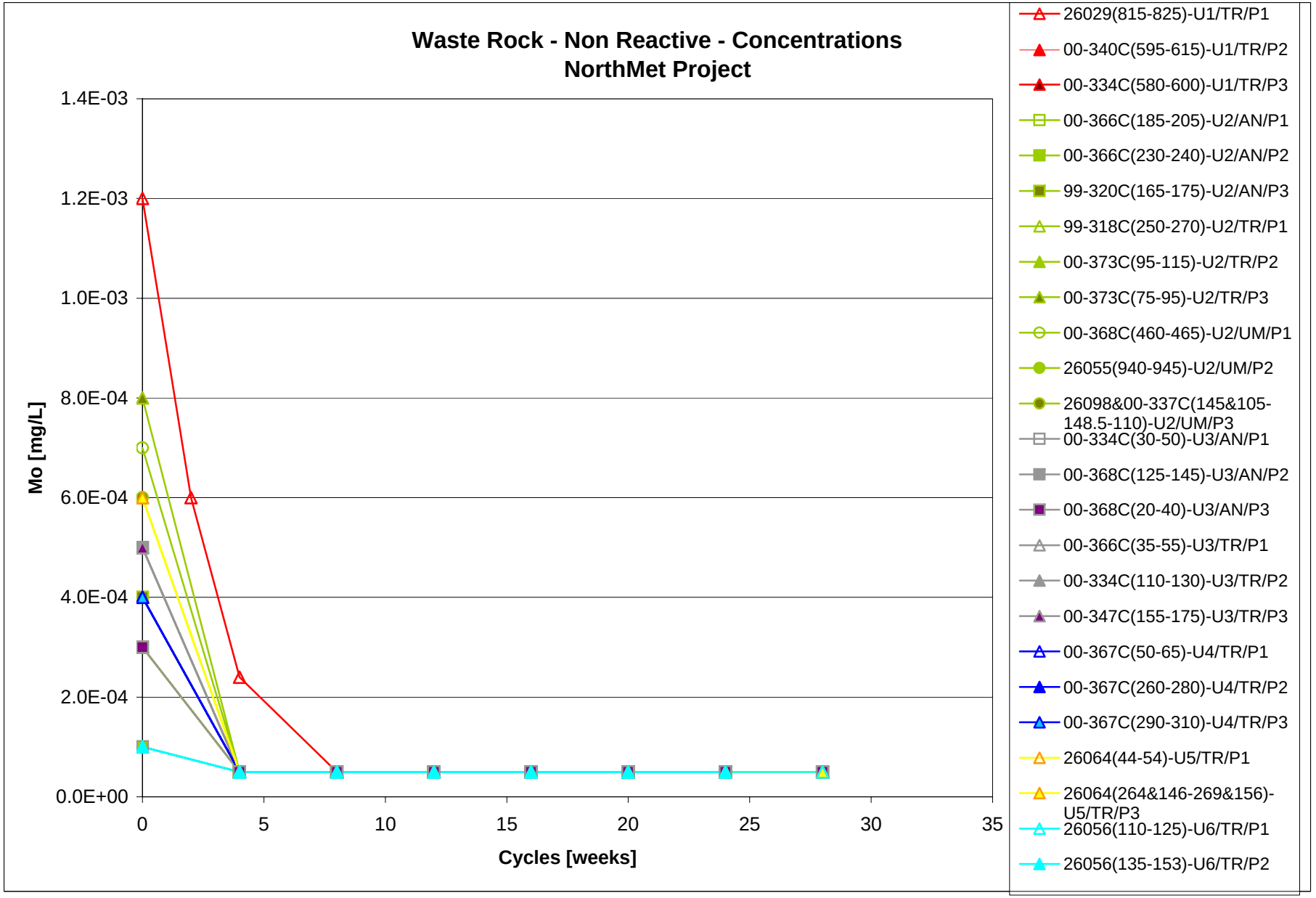

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Hg [mg/L]
1.2E-04
1.0E-04
8.0E-05
6.0E-05
4.0E-05
2.0E-05
0.0E+00
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

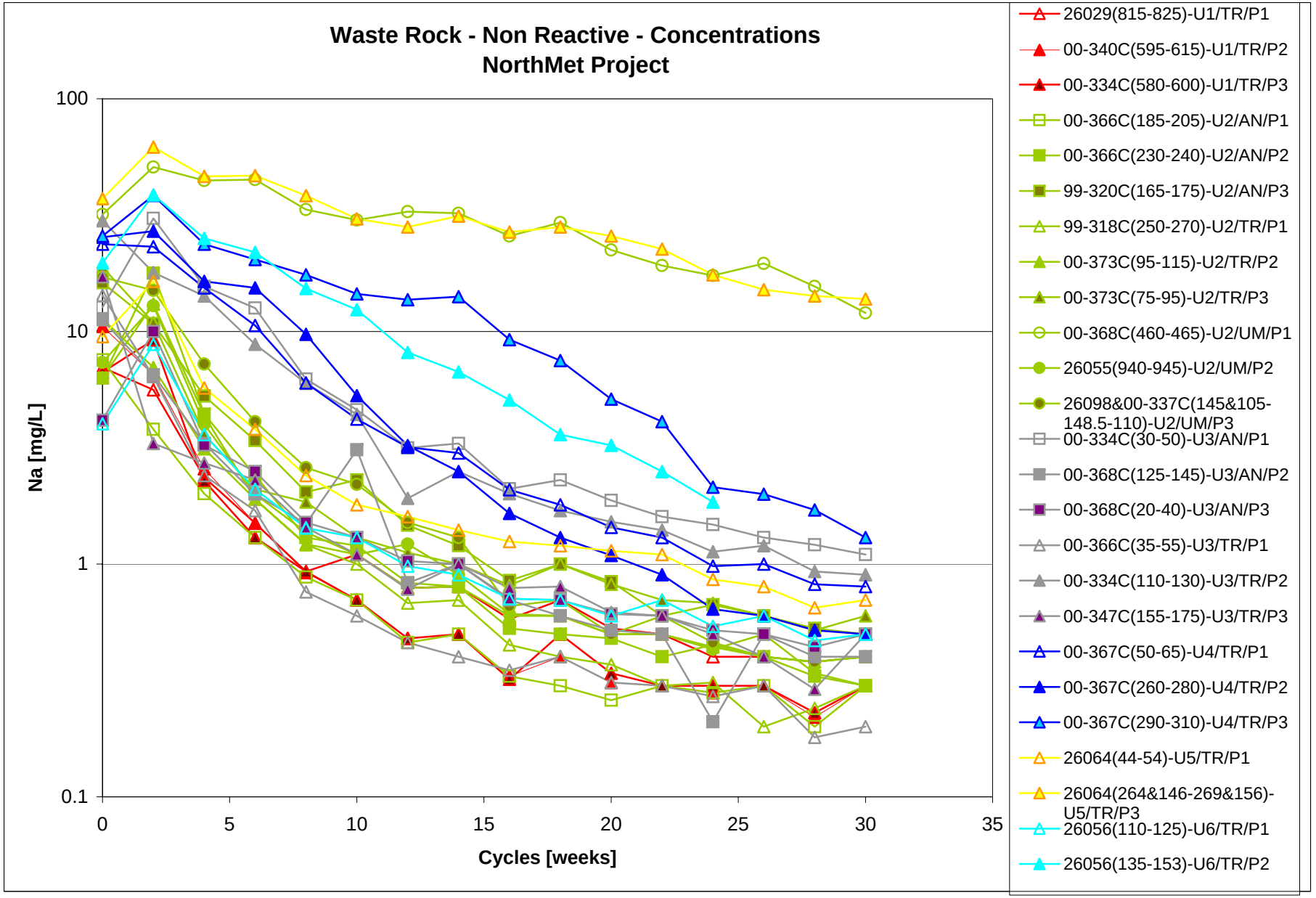

Waste Rock - Non Reactive - Concentrations
NorthMet Project
K [mg/L]
Cycles [weeks]
0
0.2
0.4
0.6
0.8
1
1.2
1.4
1.6
1.8
0
5
10
15
20
25
30
35
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

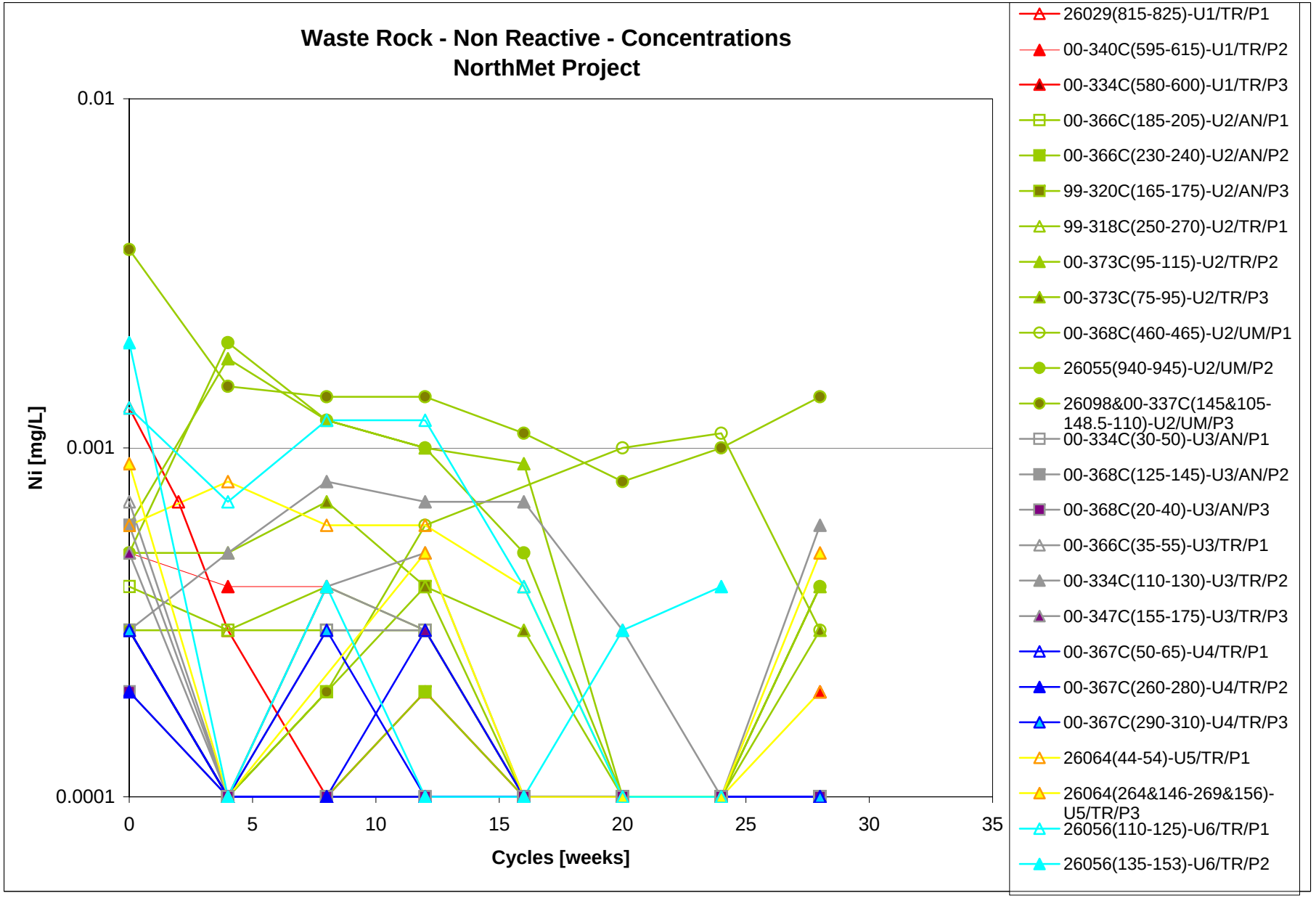

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Li [mg/L]
3.0E-03
2.5E-03
2.0E-03
1.5E-03
1.0E-03
5.0E-04
0.0E+00
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

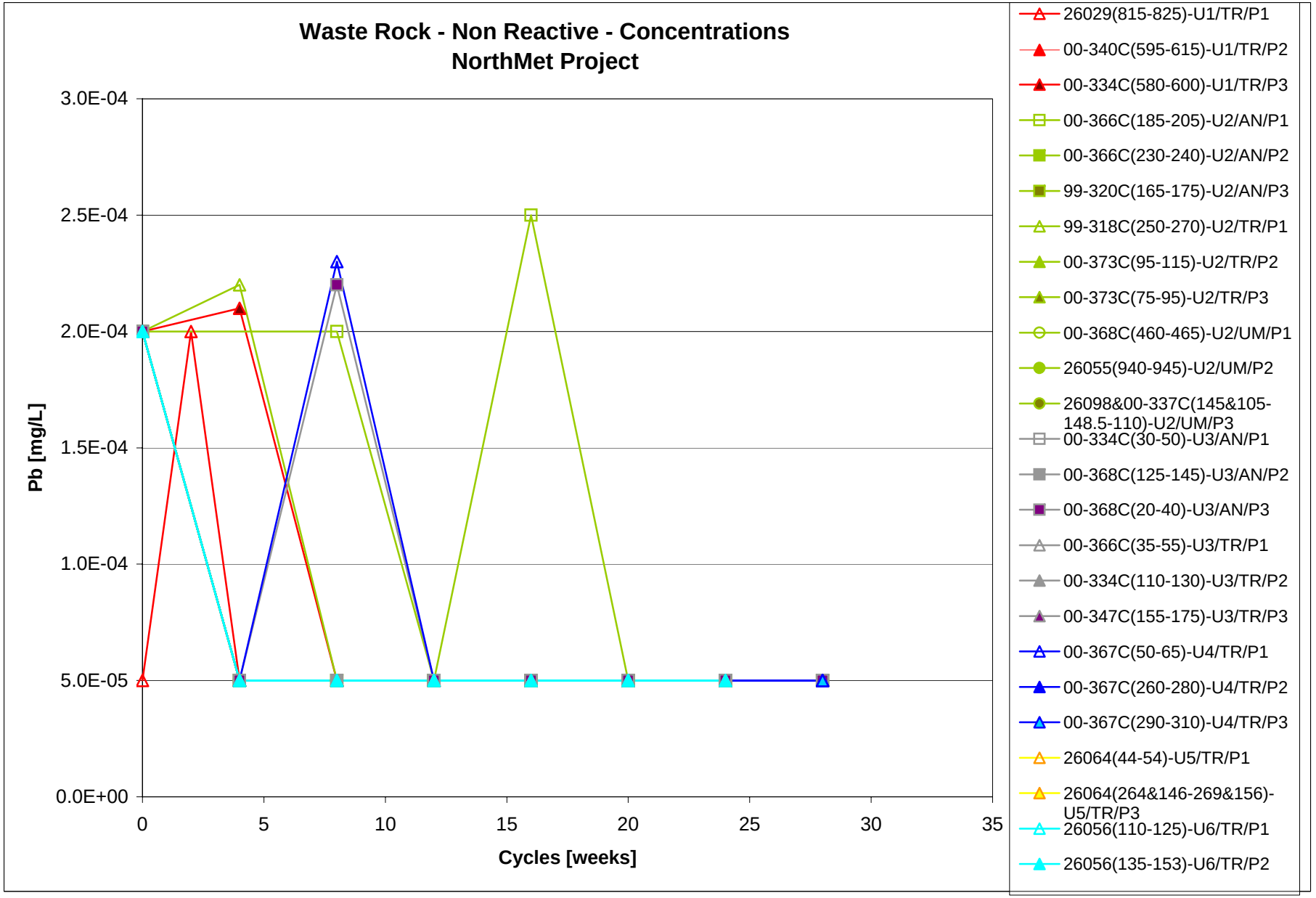

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Mg [mg/L]
Cycles [weeks]
0
0.5
1
1.5
2
2.5
3
3.5
4
4.5
0
5
10
15
20
25
30
35
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

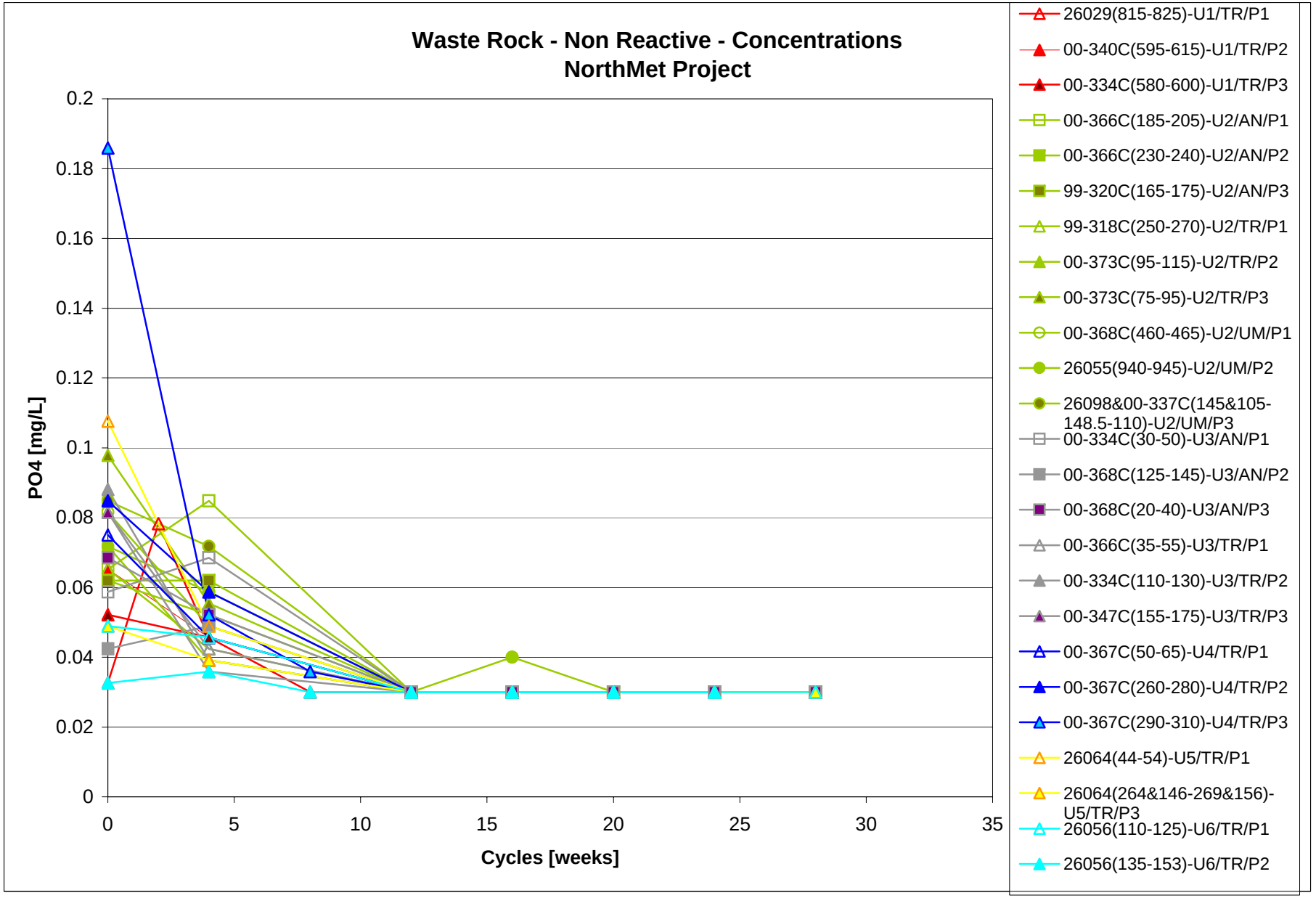

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Mn [mg/L]
0.008
0.007
0.006
0.005
0.004
0.003
0.002
0.001
0
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

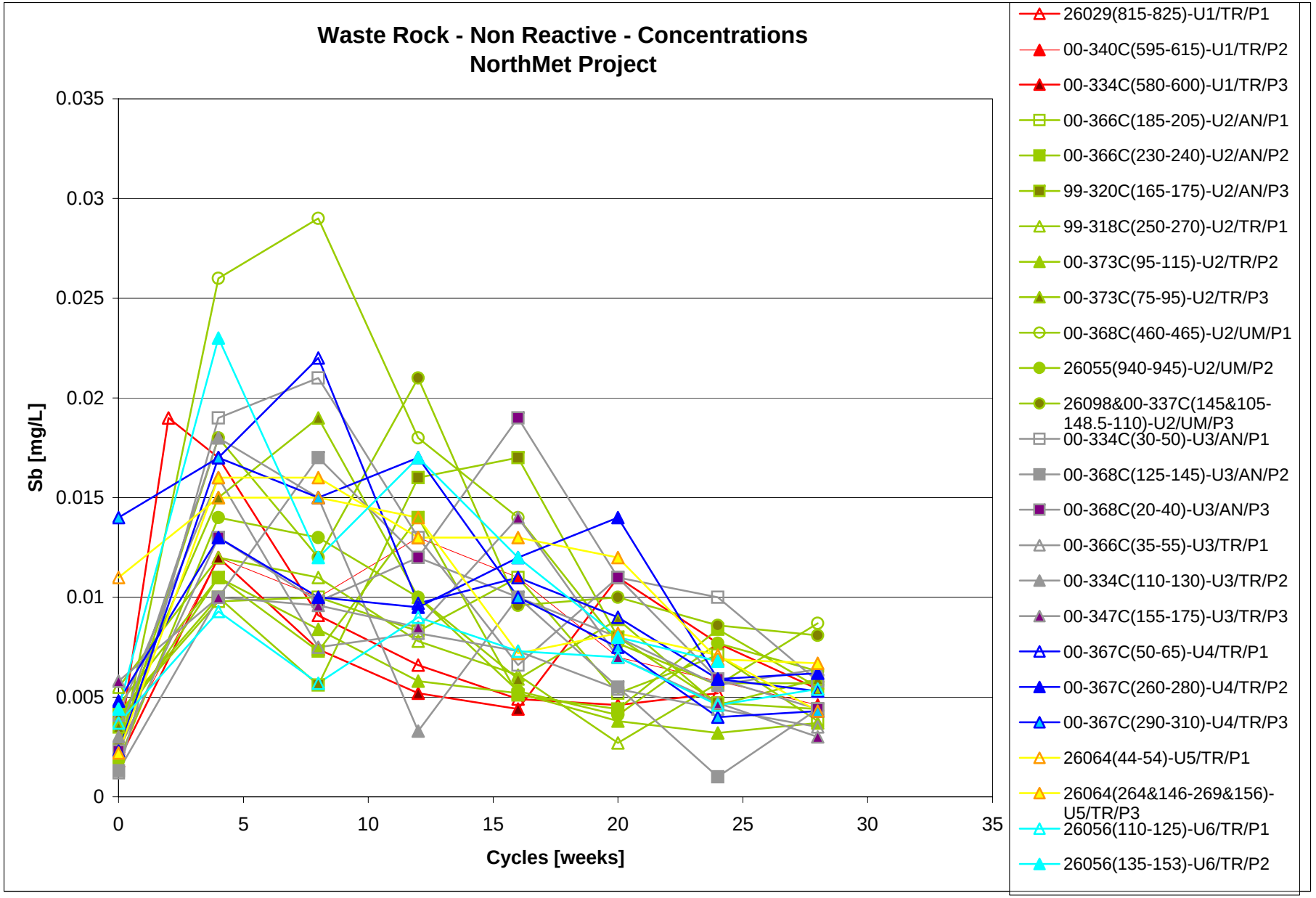

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Mo [mg/L]
1.4E-03
1.2E-03
1.0E-03
8.0E-04
6.0E-04
4.0E-04
2.0E-04
0.0E+00
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

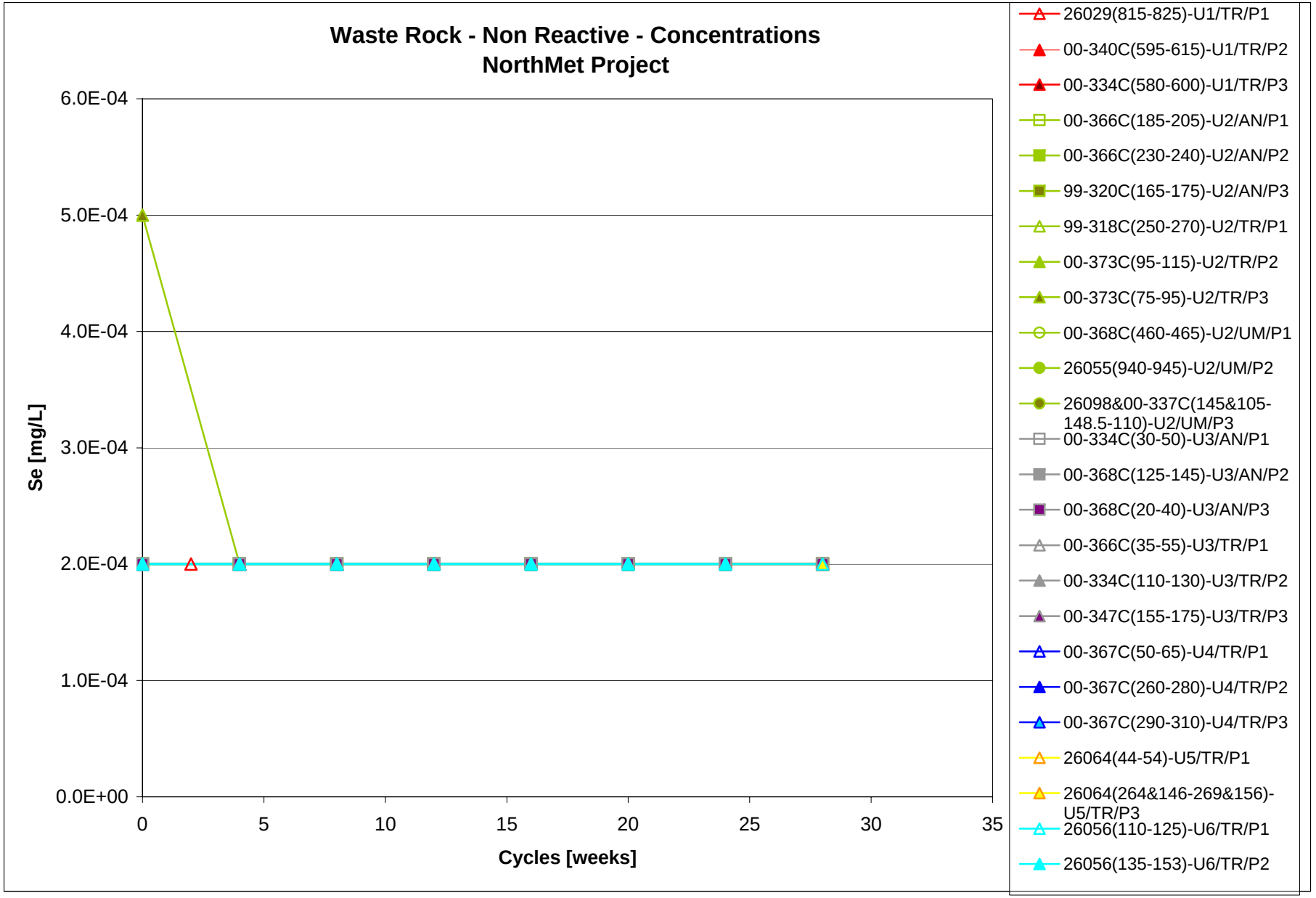

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Na [mg/L]
100
10
1
0.1
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

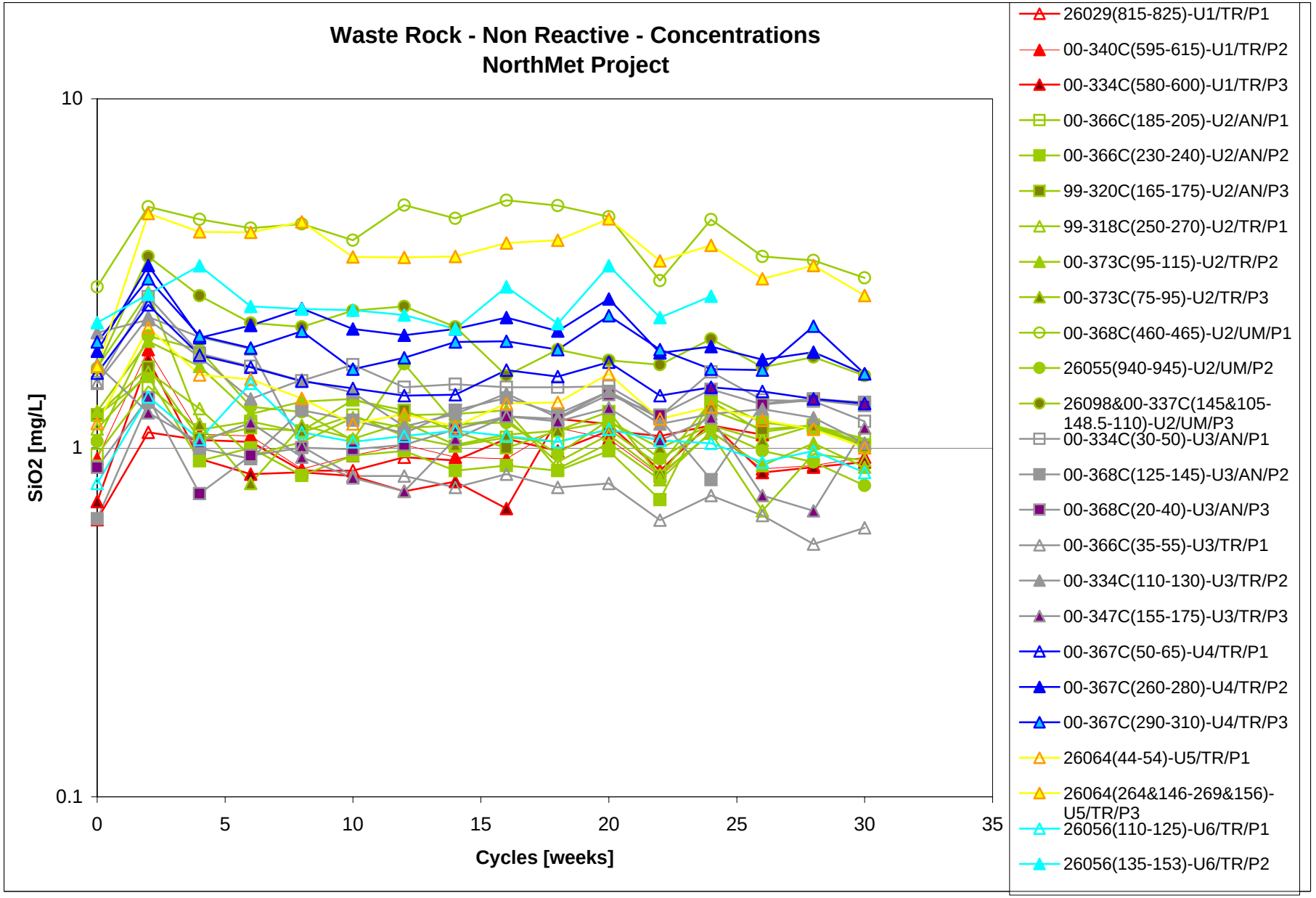

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Ni [mg/L]
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

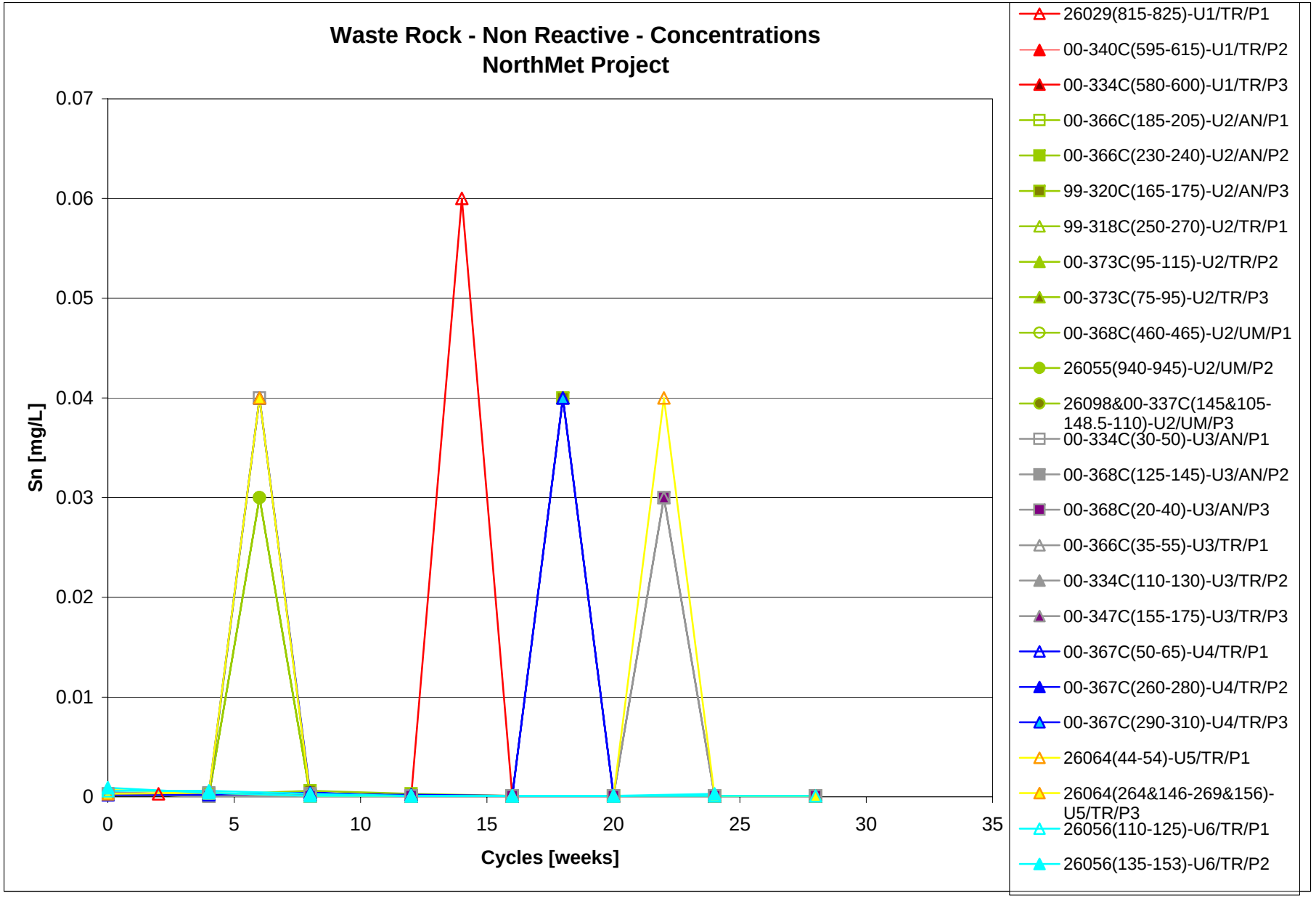

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Pb [mg/L]
3.0E-04
2.5E-04
2.0E-04
1.5E-04
1.0E-04
5.0E-05
0.0E+00
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

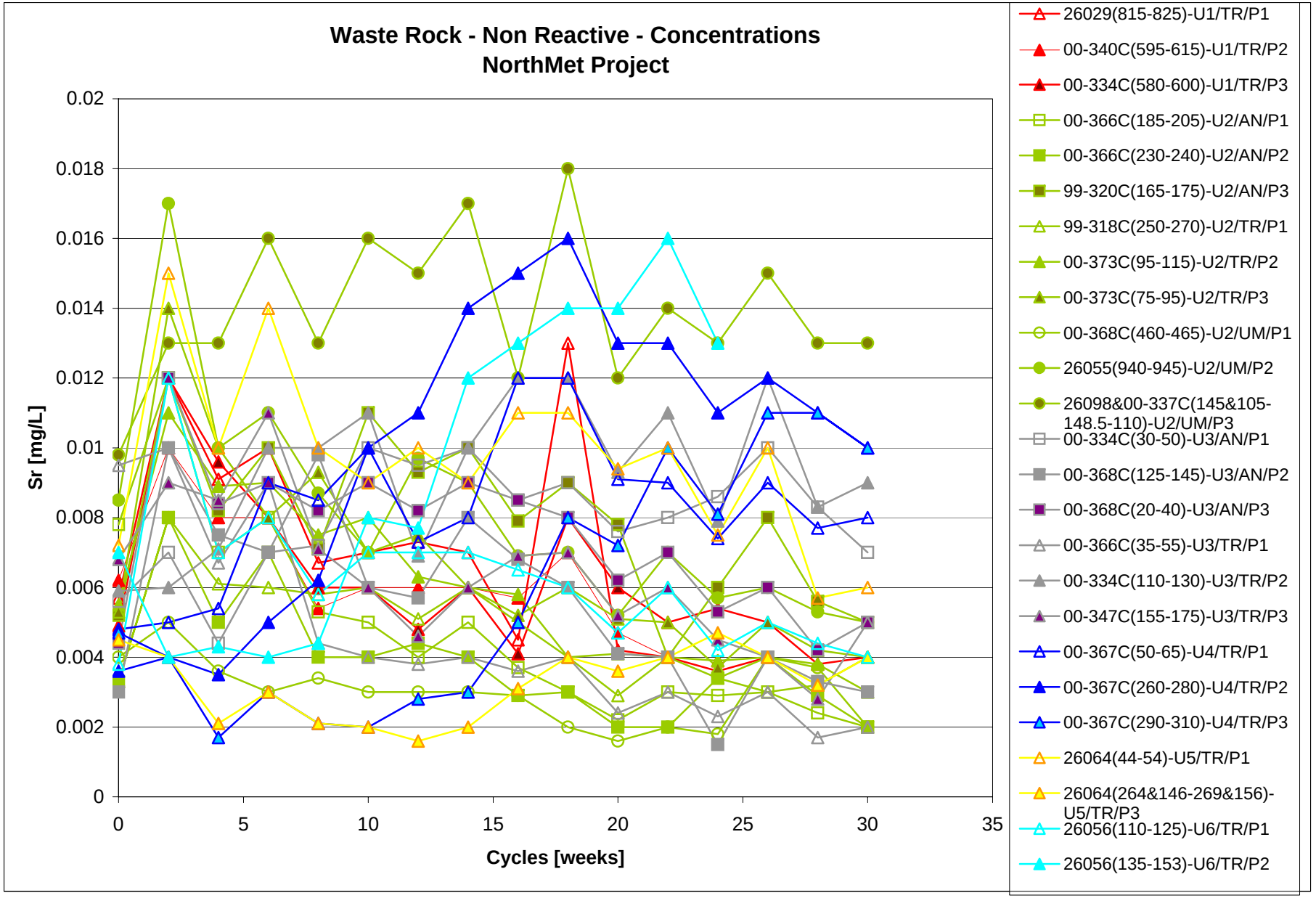

Waste Rock - Non Reactive - Concentrations
NorthMet Project
PO4 [mg/L]
0.2
0.18
0.16
0.14
0.12
0.1
0.08
0.06
0.04
0.02
0
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

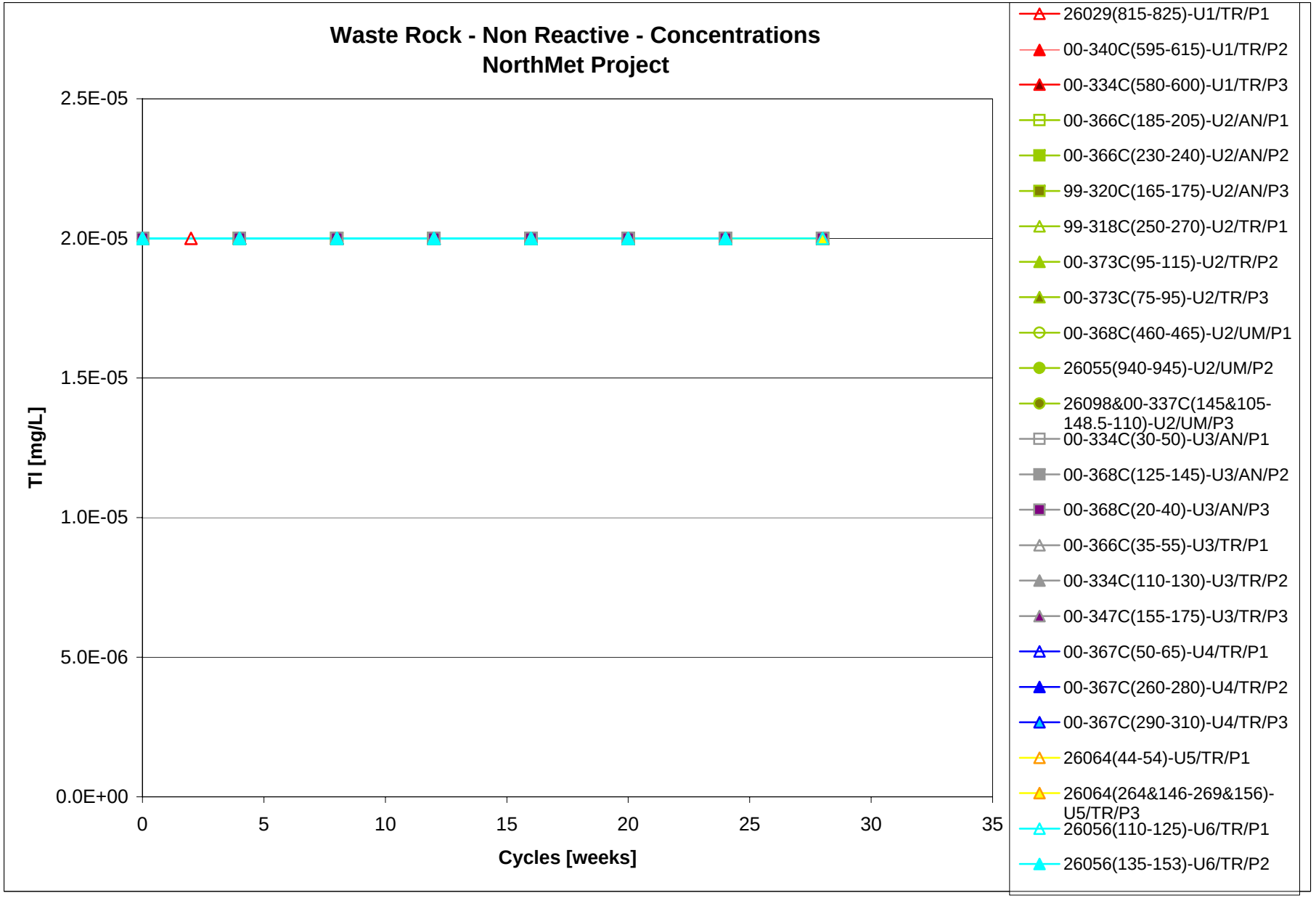

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Sb [mg/L]
0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

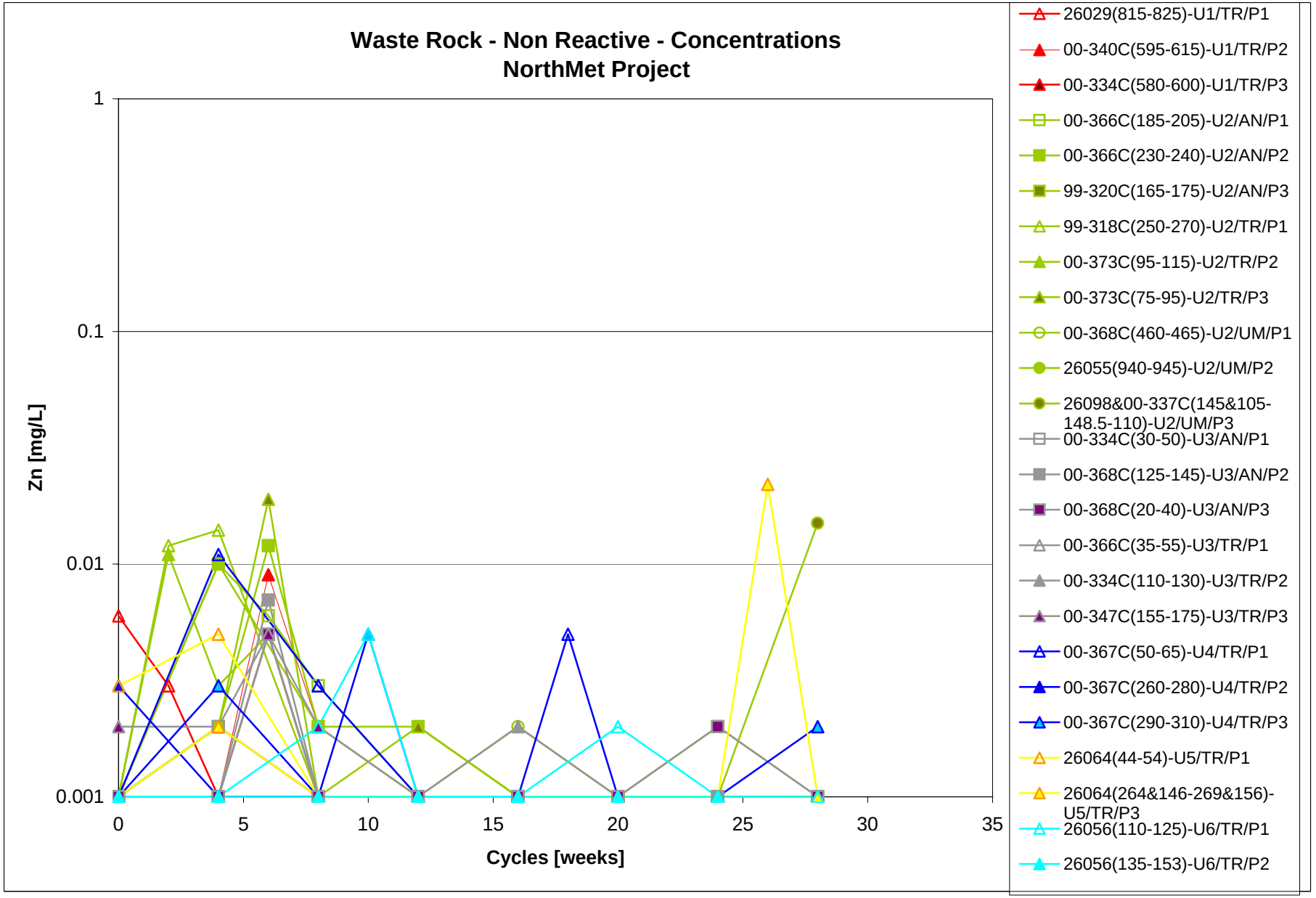

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Se [mg/L]
6.0E-04
5.0E-04
4.0E-04
3.0E-04
2.0E-04
1.0E-04
0.0E+00
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2


Waste Rock - Non Reactive - Concentrations
NorthMet Project
SiO2 [mg/L]
10
1
0.1
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

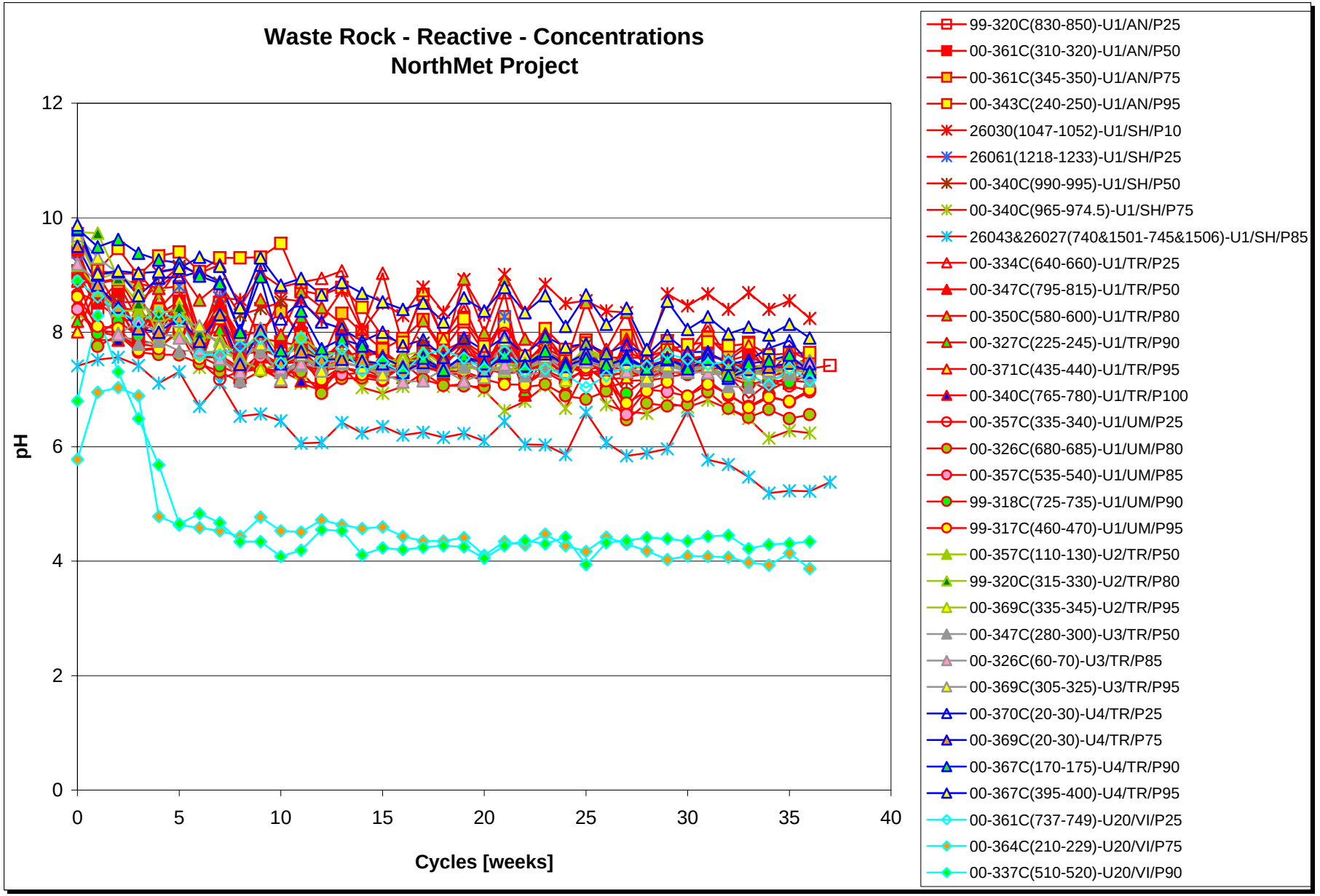

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Sn [mg/L]
0.07
0.06
0.05
0.04
0.03
0.02
0.01
0
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

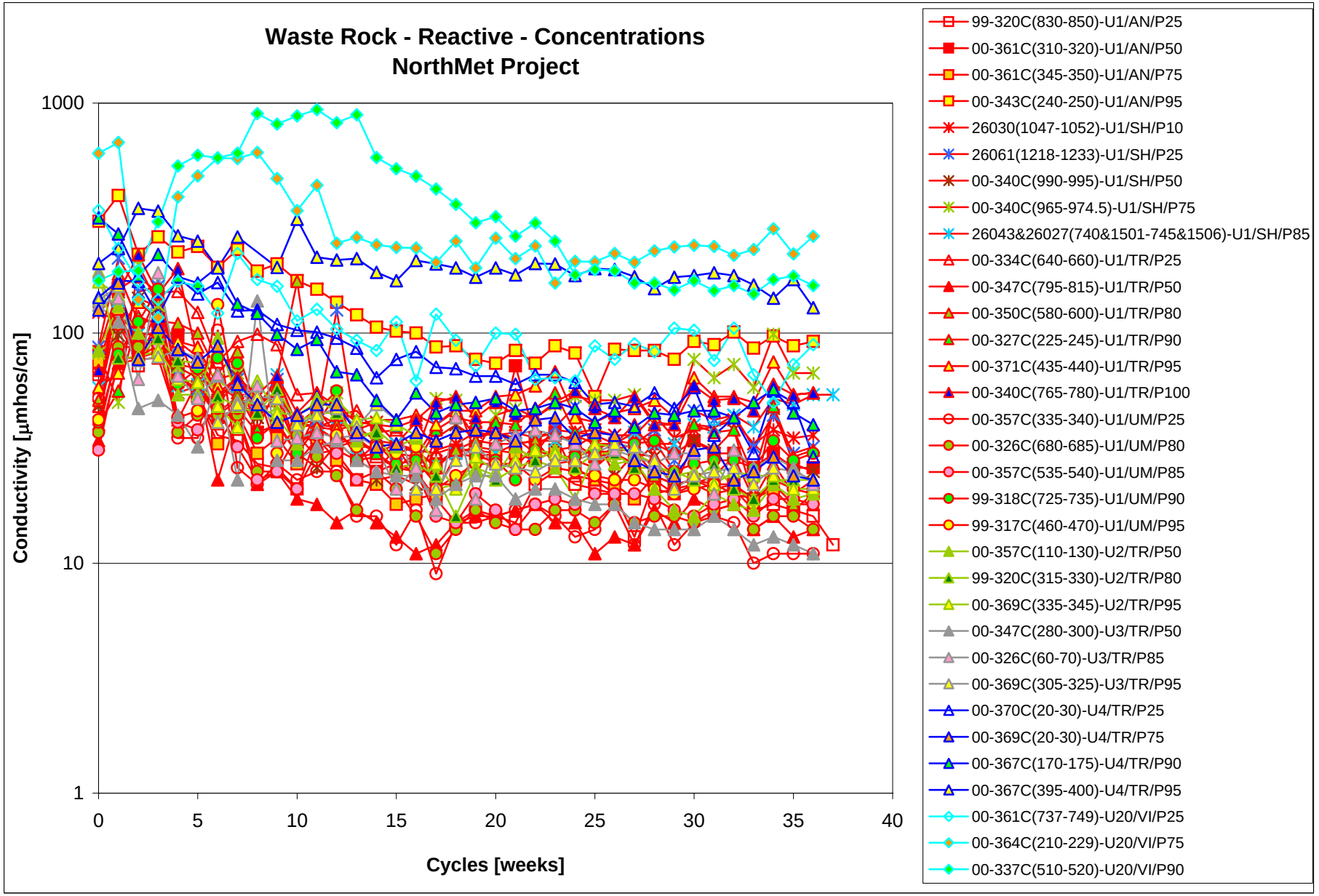

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Sr [mg/L]
0.02
0.018
0.016
0.014
0.012
0.01
0.008
0.006
0.004
0.002
0
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

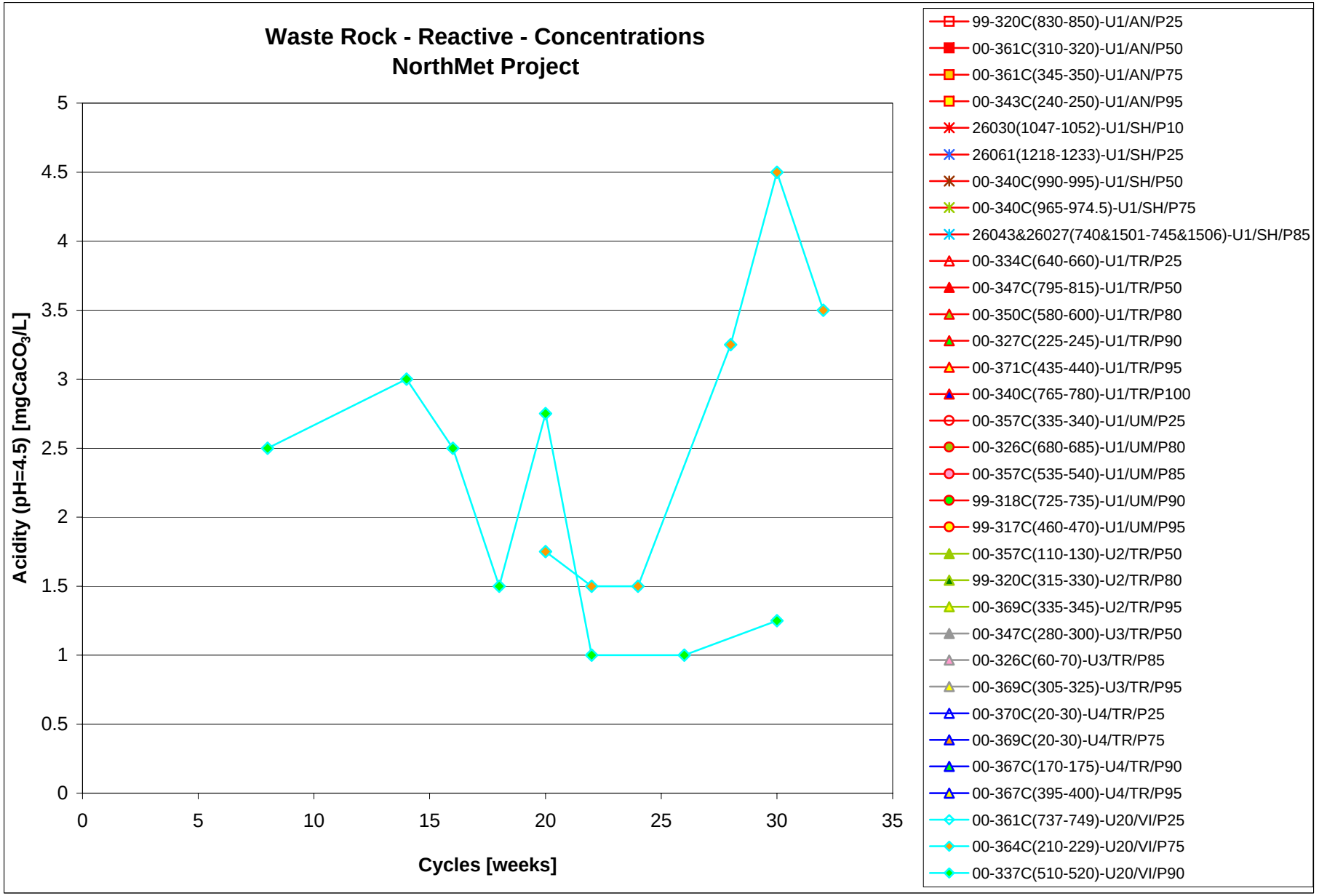

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Tl [mg/L]
2.5E-05
2.0E-05
1.5E-05
1.0E-05
5.0E-06
0.0E+00
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

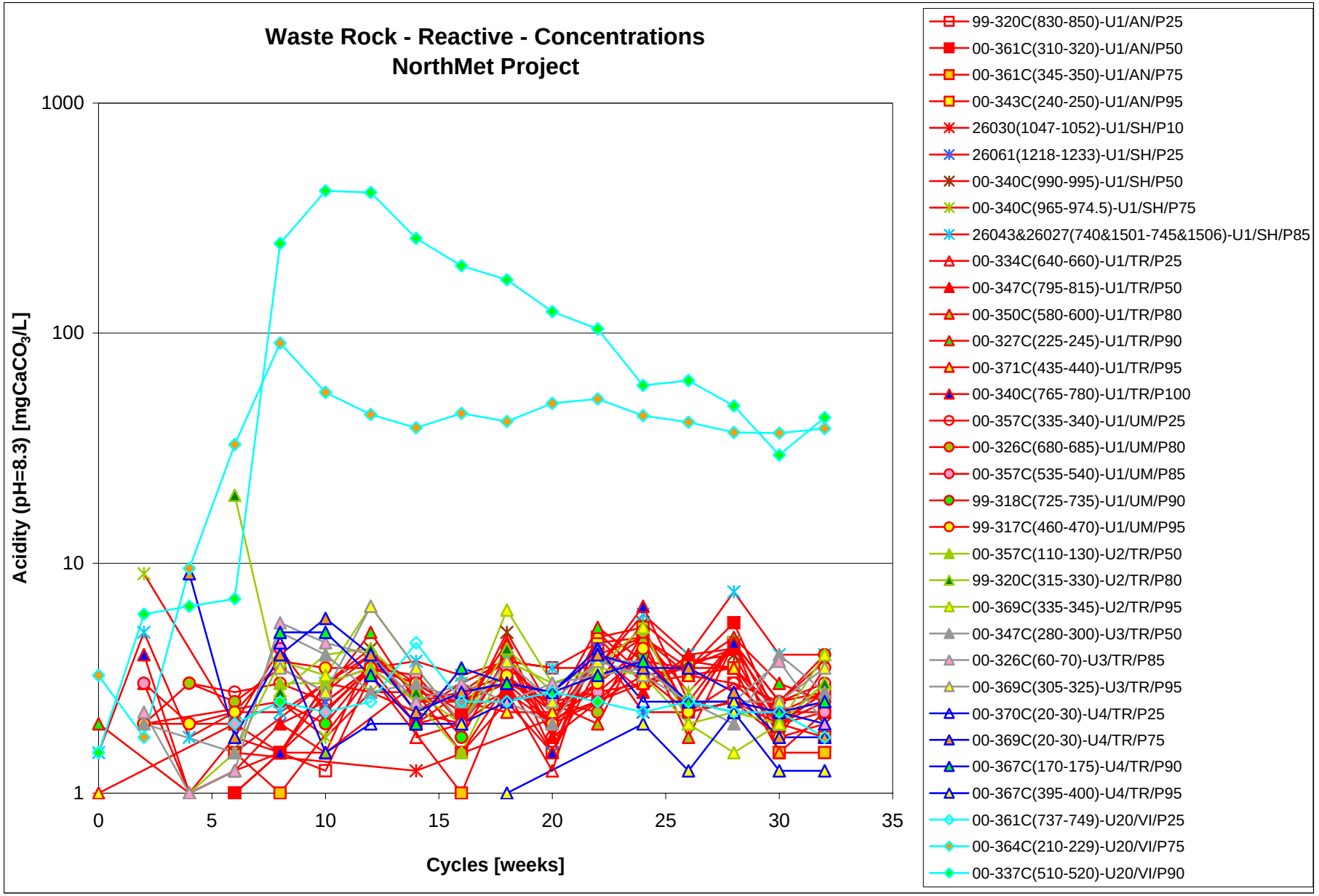

Waste Rock - Non Reactive - Concentrations
NorthMet Project
Zn [mg/L]
1
0.1
0.01
0.001
0
5
10
15
20
25
30
35
Cycles [weeks]
26029(815-825)-U1/TR/P1
00-340C(595-615)-U1/TR/P2
00-334C(580-600)-U1/TR/P3
00-366C(185-205)-U2/AN/P1
00-366C(230-240)-U2/AN/P2
99-320C(165-175)-U2/AN/P3
99-318C(250-270)-U2/TR/P1
00-373C(95-115)-U2/TR/P2
00-373C(75-95)-U2/TR/P3
00-368C(460-465)-U2/UM/P1
26055(940-945)-U2/UM/P2
26098&00-337C(145&105-148.5-110)-U2/UM/P3
00-334C(30-50)-U3/AN/P1
00-368C(125-145)-U3/AN/P2
00-368C(20-40)-U3/AN/P3
00-366C(35-55)-U3/TR/P1
00-334C(110-130)-U3/TR/P2
00-347C(155-175)-U3/TR/P3
00-367C(50-65)-U4/TR/P1
00-367C(260-280)-U4/TR/P2
00-367C(290-310)-U4/TR/P3
26064(44-54)-U5/TR/P1
26064(264&146-269&156)-U5/TR/P3
26056(110-125)-U6/TR/P1
26056(135-153)-U6/TR/P2

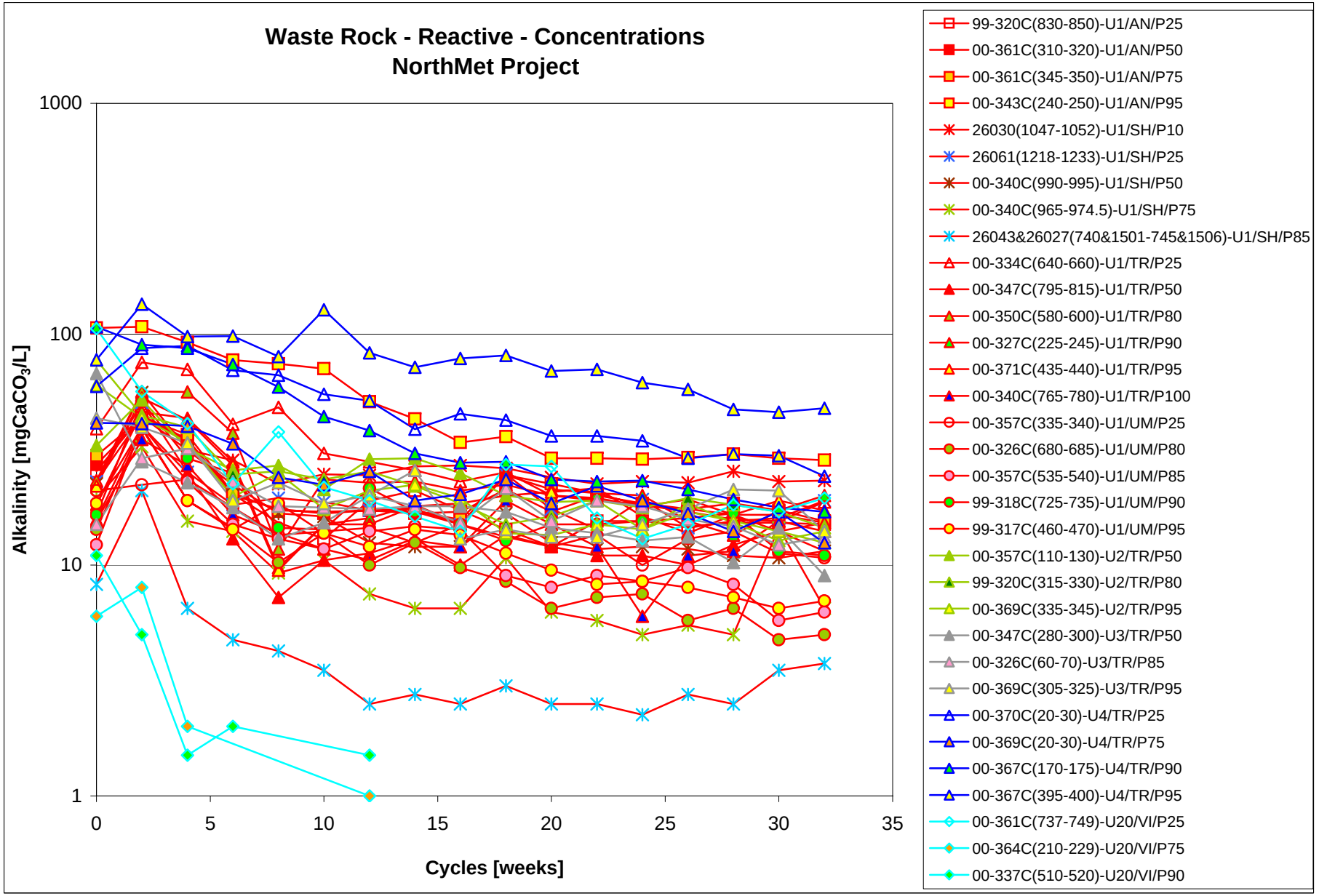
Appendix F
Leachate Chemistry Graphs – Reactive Waste Rock

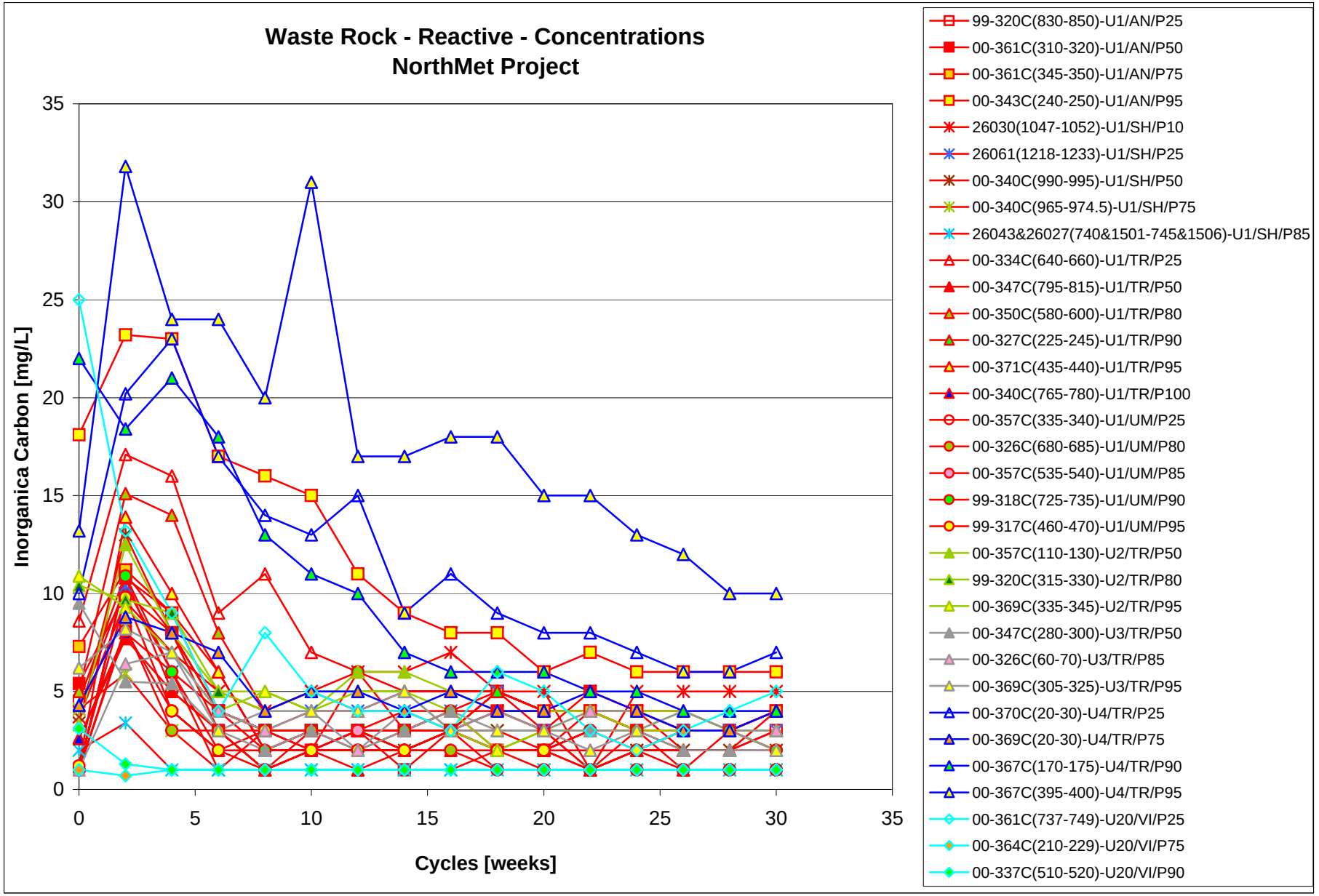

Waste Rock - Reactive - Concentrations
NorthMet Project
pH
Cycles [weeks]
0
2
4
6
8
10
12
0
5
10
15
20
25
30
35
40
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

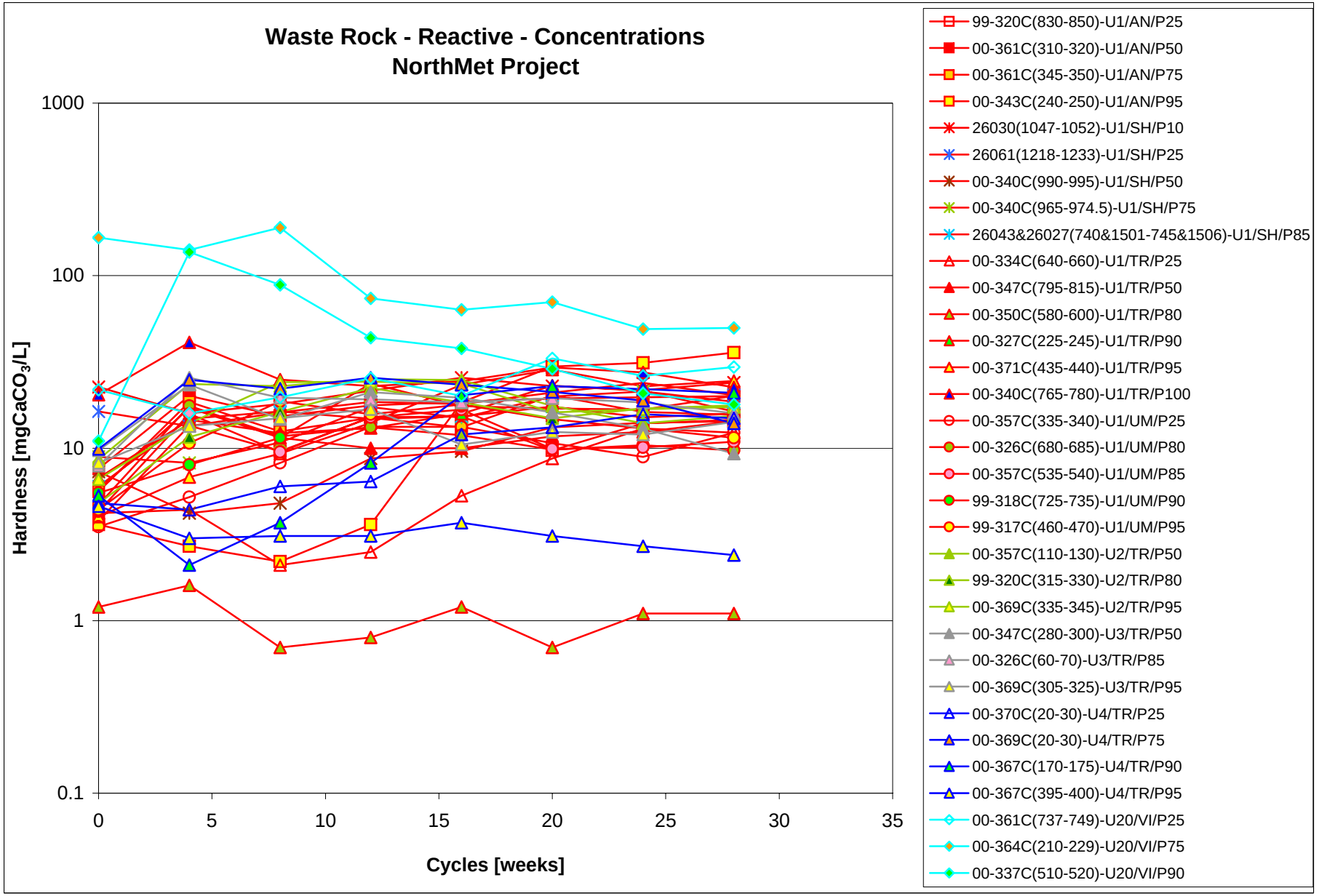

Waste Rock - Reactive - Concentrations
NorthMet Project
Conductivity [µmhos/cm]
1000
100
10
1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

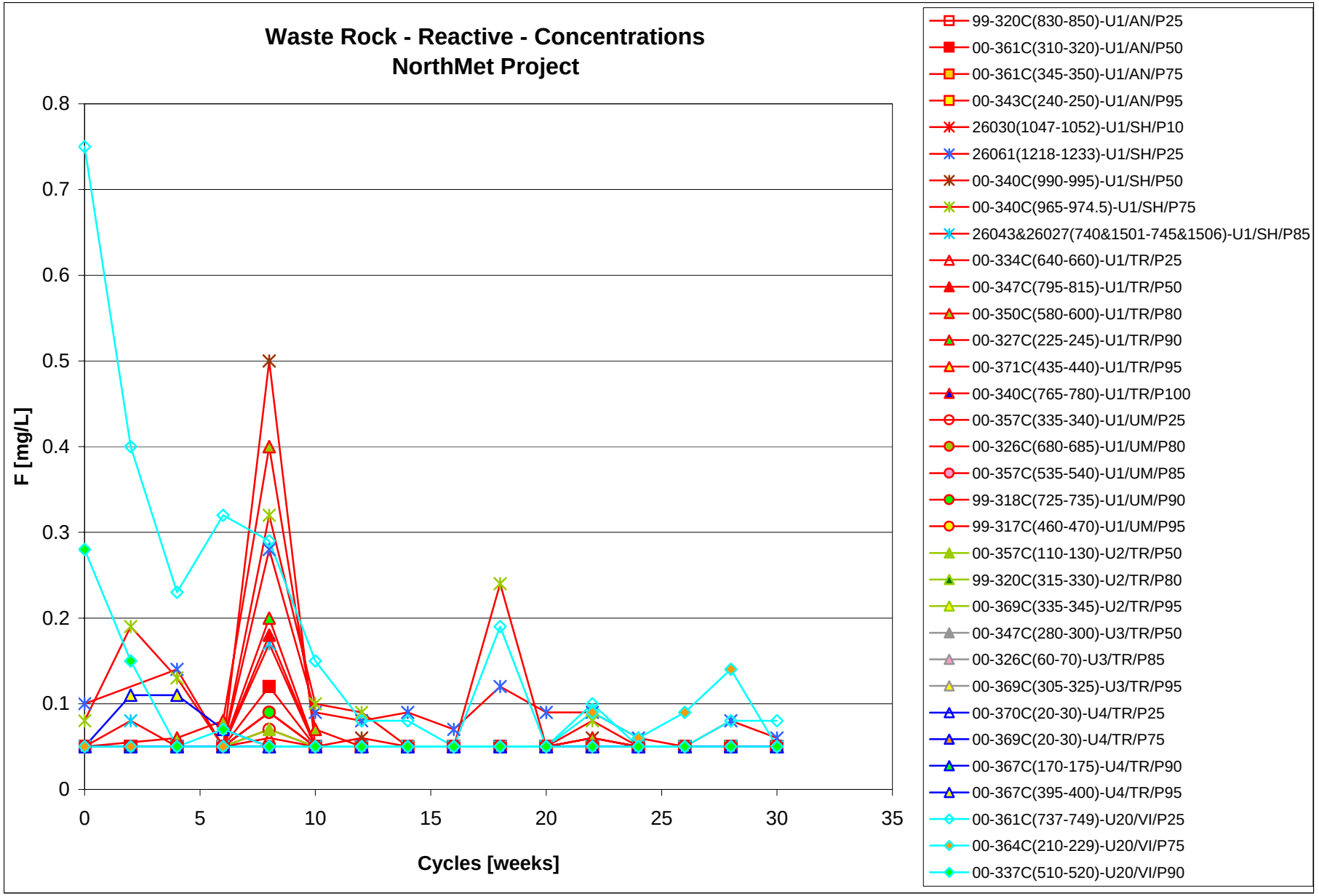

Waste Rock - Reactive - Concentrations
NorthMet Project
Acidity (pH=4.5) [mgCaCO3/L]
Cycles [weeks]
0
0.5
1
1.5
2
2.5
3
3.5
4
4.5
5
0
5
10
15
20
25
30
35
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

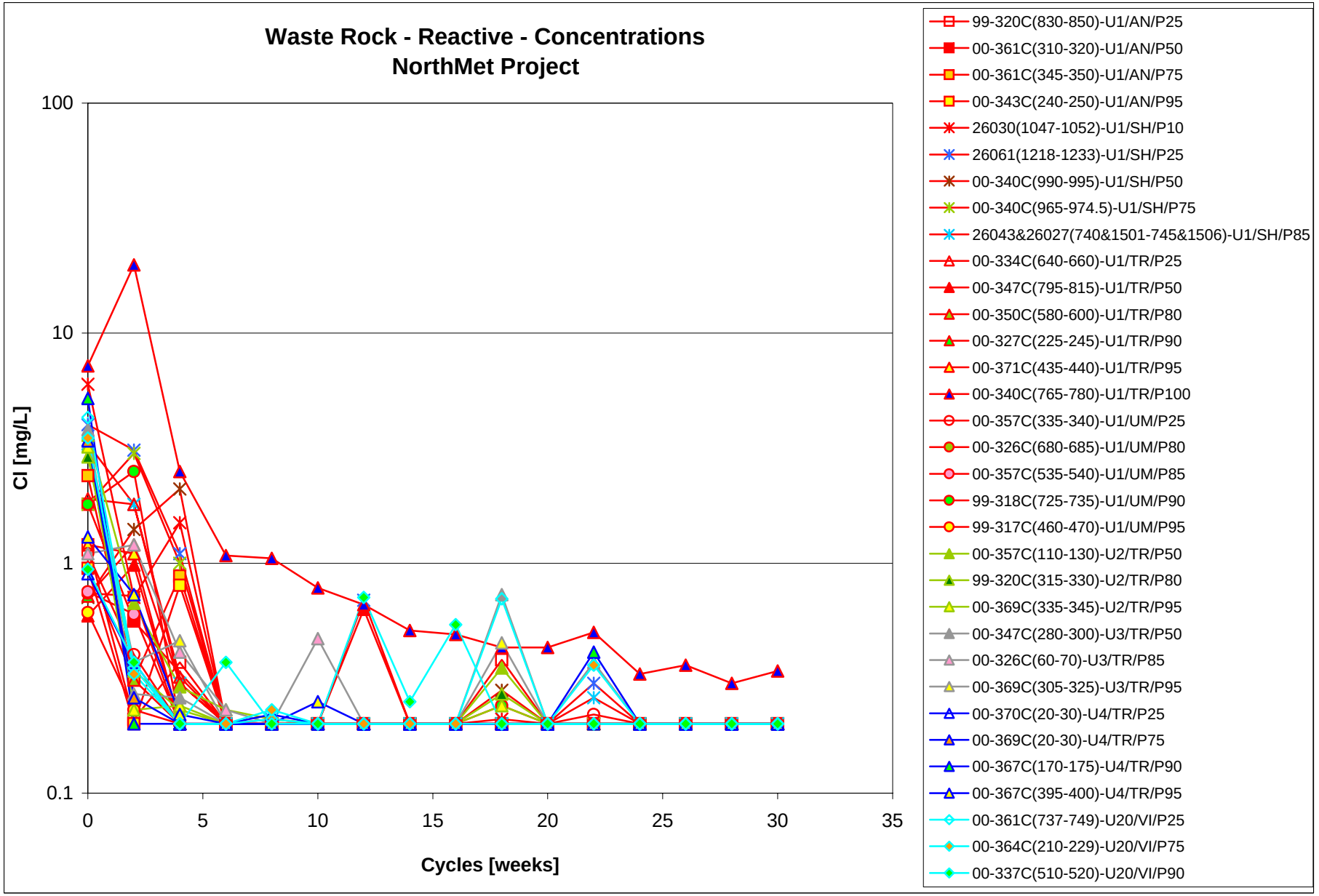

Waste Rock - Reactive - Concentrations
NorthMet Project
Acidity (pH=8.3) [mgCaCO3/L]
1000
100
10
1
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

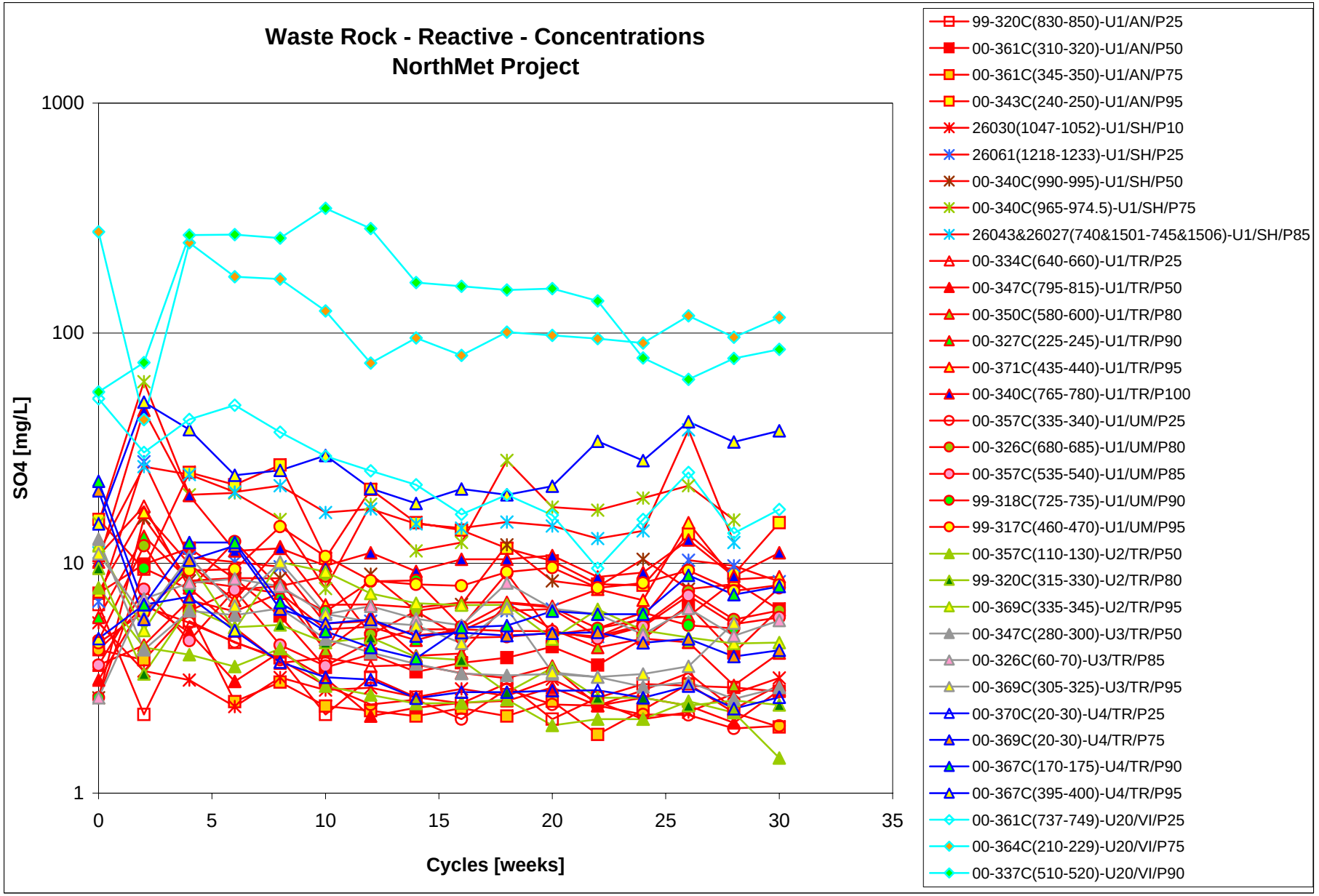

Waste Rock - Reactive - Concentrations
NorthMet Project
Alkalinity [mgCaCO3/L]
1000
100
10
1
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

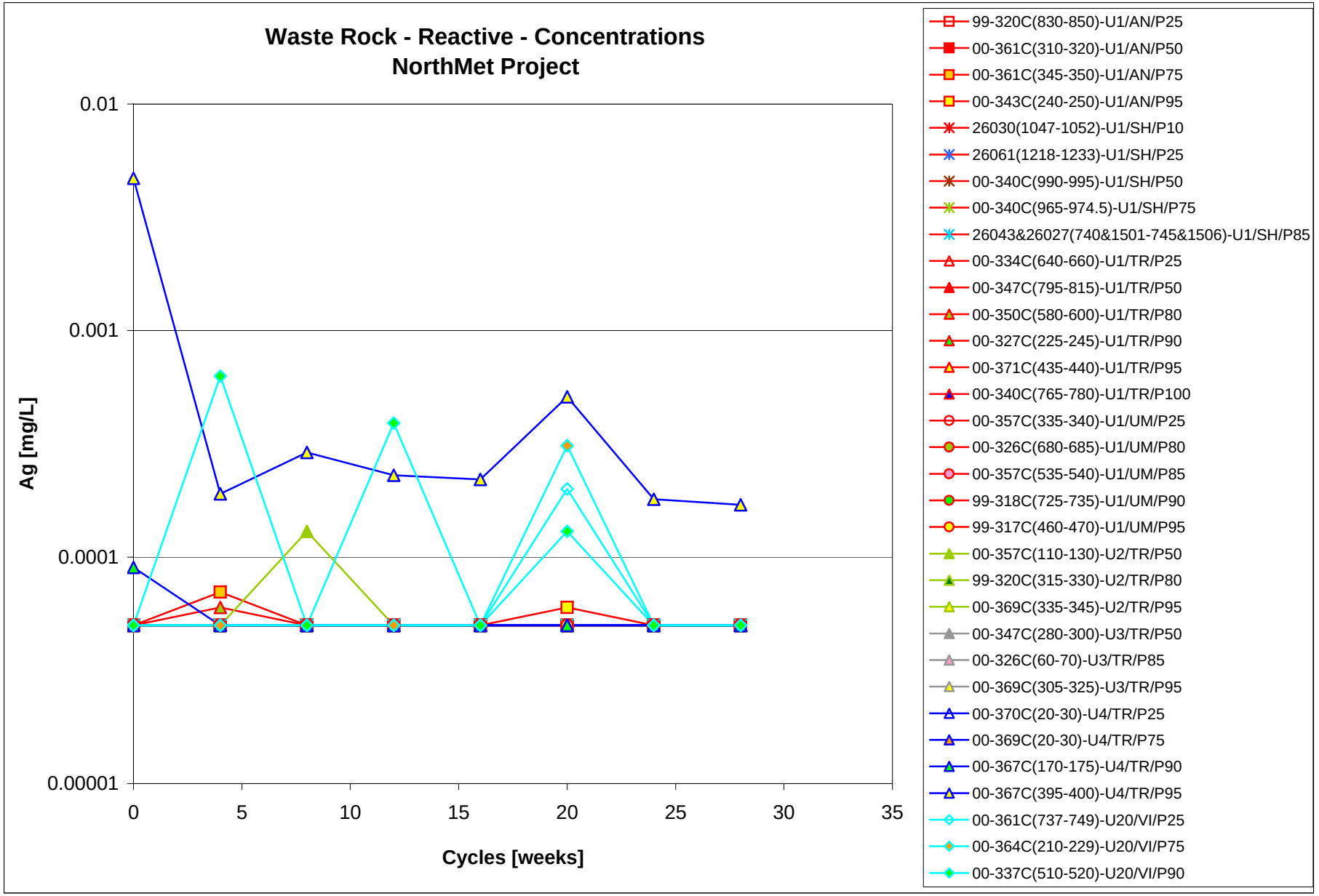

Waste Rock - Reactive - Concentrations
NorthMet Project
Inorganica Carbon [mg/L]
Cycles [weeks]
0
5
10
15
20
25
30
35
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

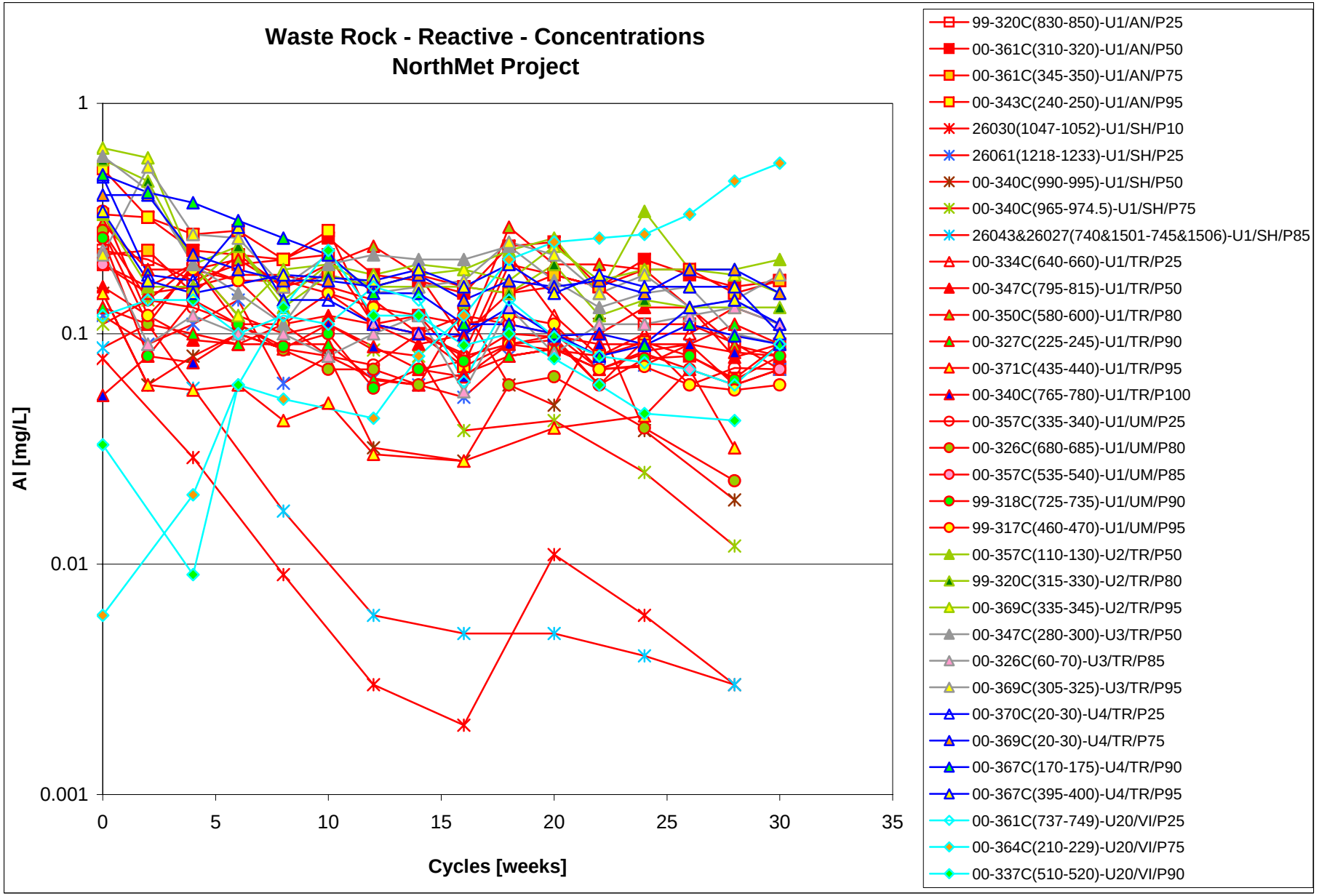

Waste Rock - Reactive - Concentrations
NorthMet Project
Hardness [mgCaCO3/L]
1000
100
10
1
0.1
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

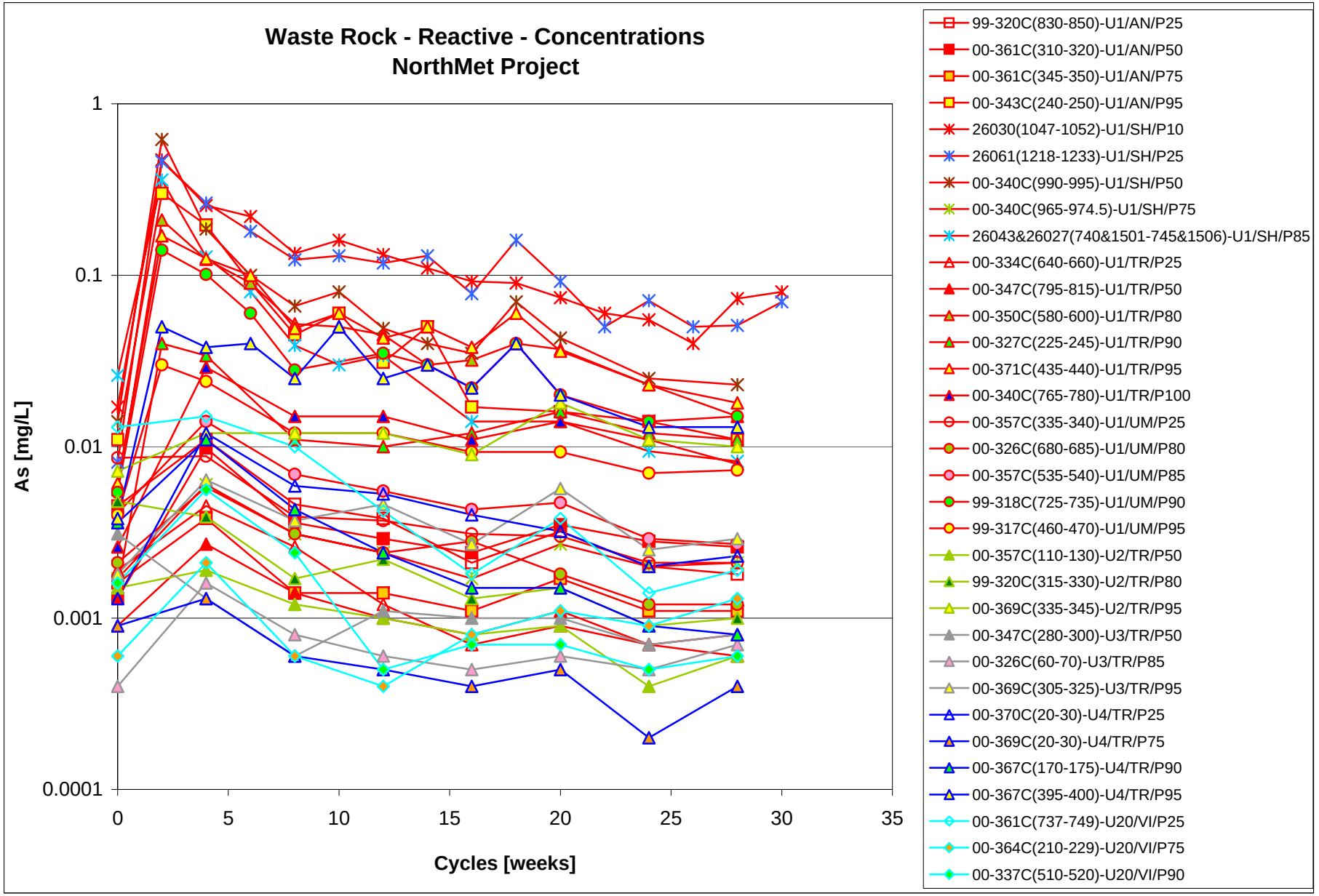

Waste Rock - Reactive - Concentrations
NorthMet Project
F [mg/L]
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
0
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

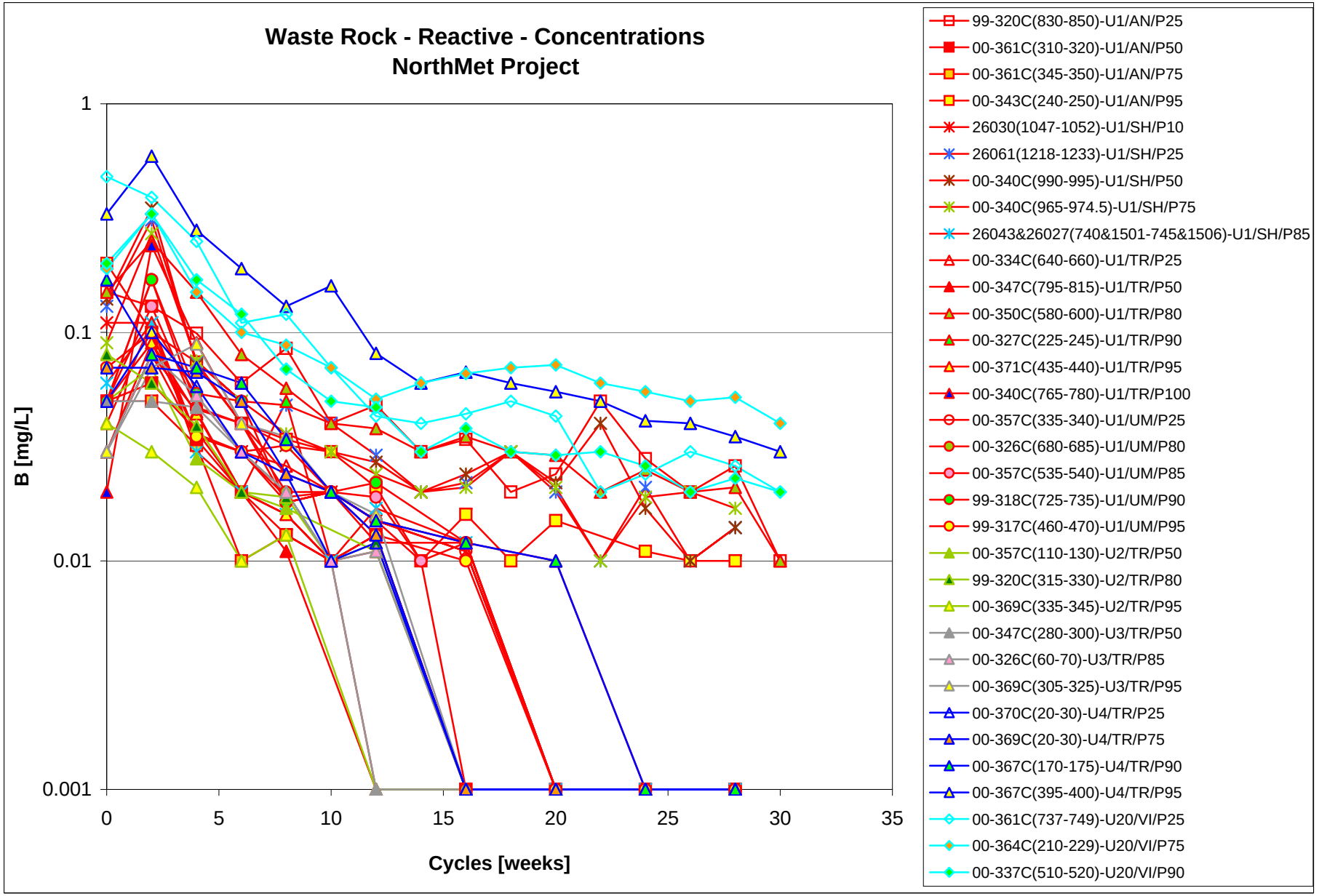

Waste Rock - Reactive - Concentrations
NorthMet Project
Cl [mg/L]
100
10
1
0.1
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

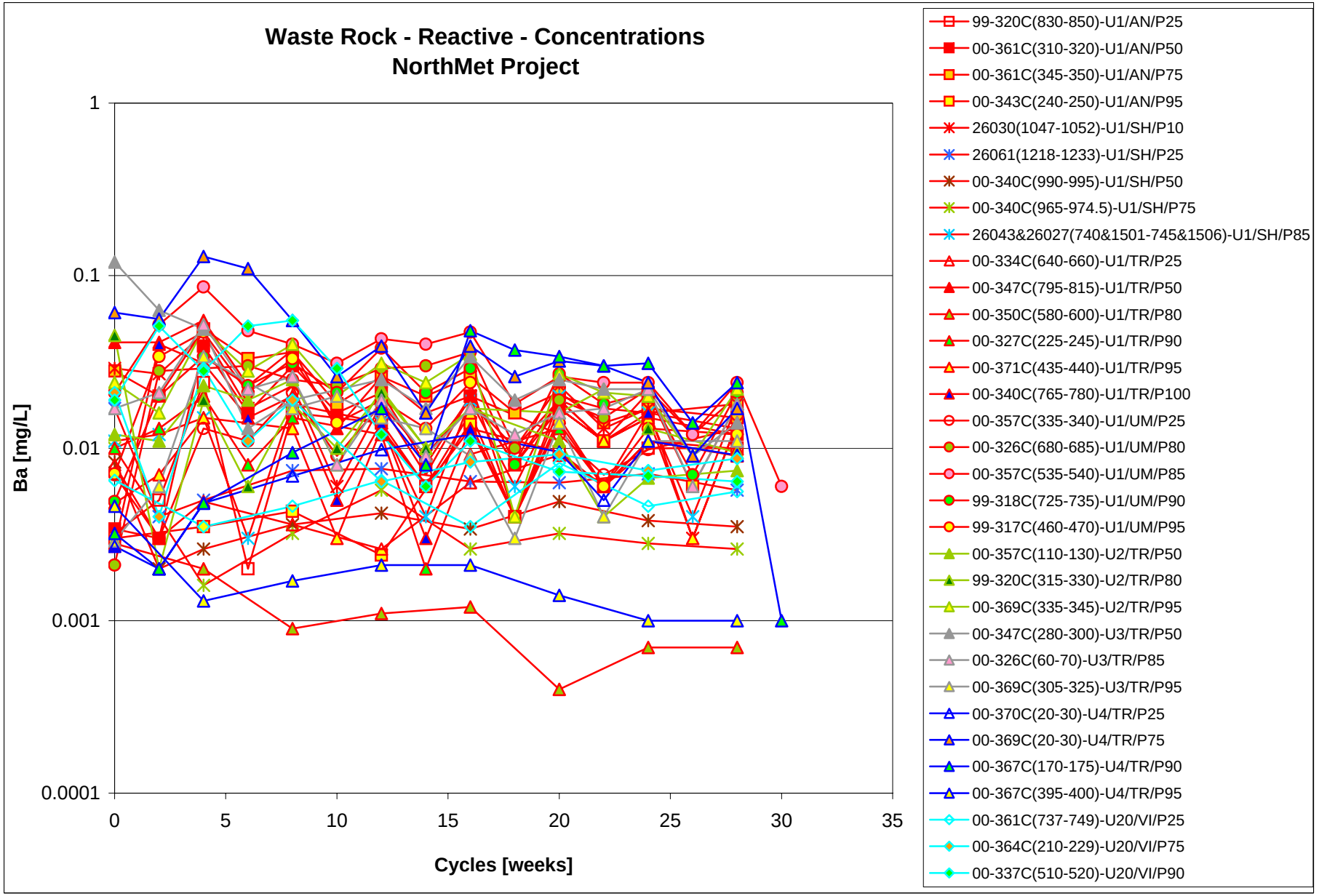

Waste Rock - Reactive - Concentrations
NorthMet Project
SO4 [mg/L]
1000
100
10
1
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

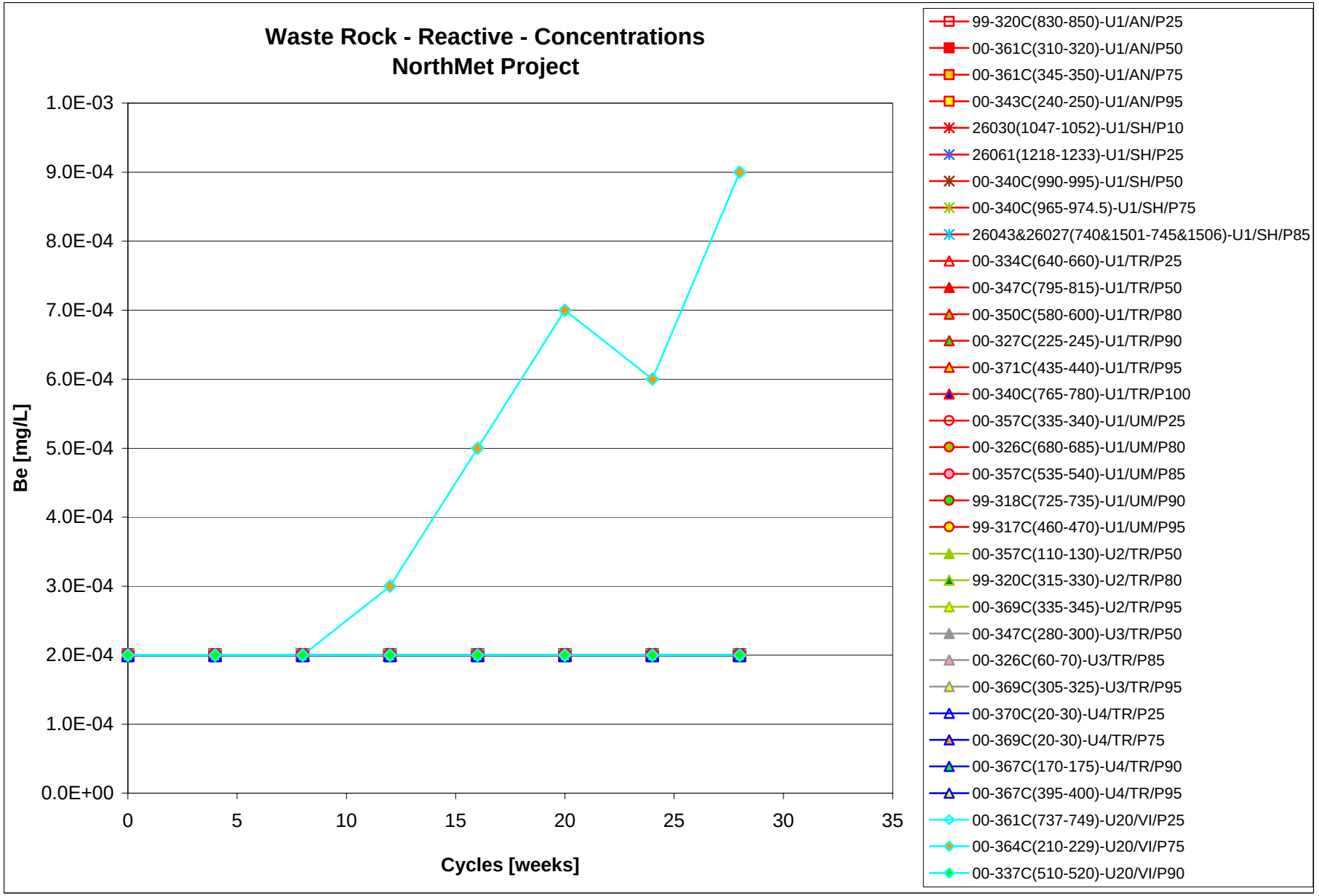

Waste Rock - Reactive - Concentrations
NorthMet Project
Ag [mg/L]
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

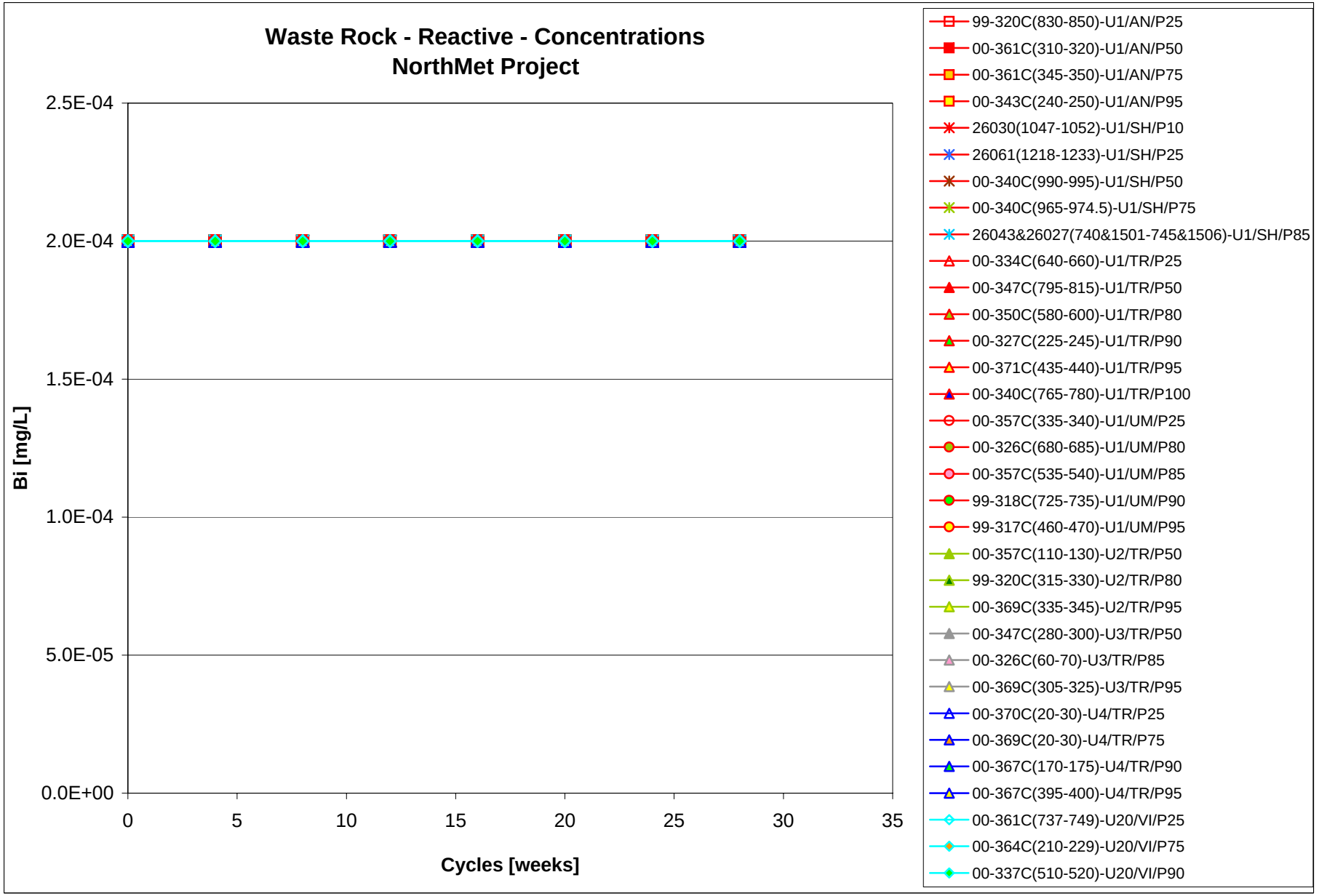

Waste Rock - Reactive - Concentrations
NorthMet Project
Al [mg/L]
1
0.1
0.01
0.001
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

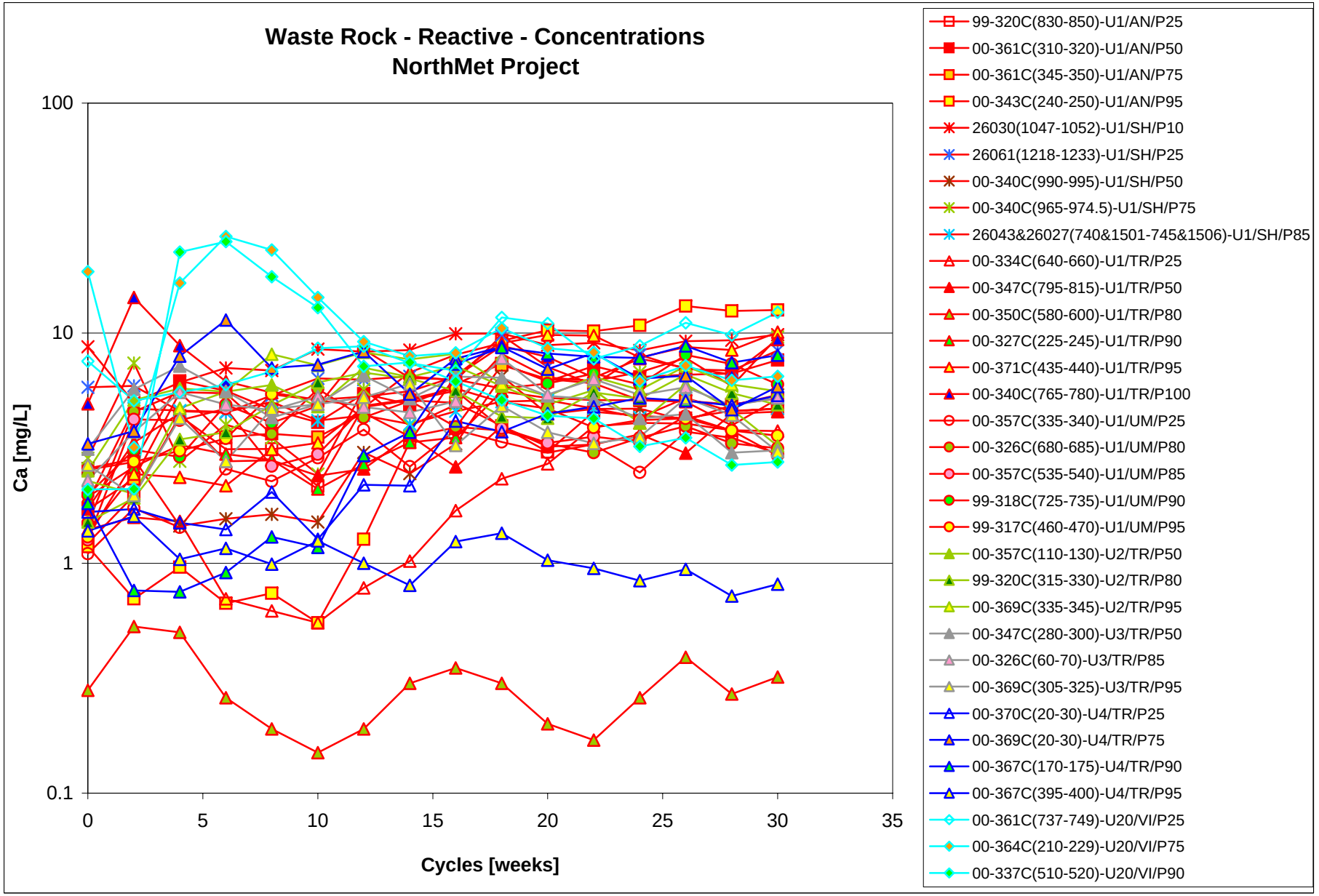

Waste Rock - Reactive - Concentrations
NorthMet Project
As [mg/L]
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

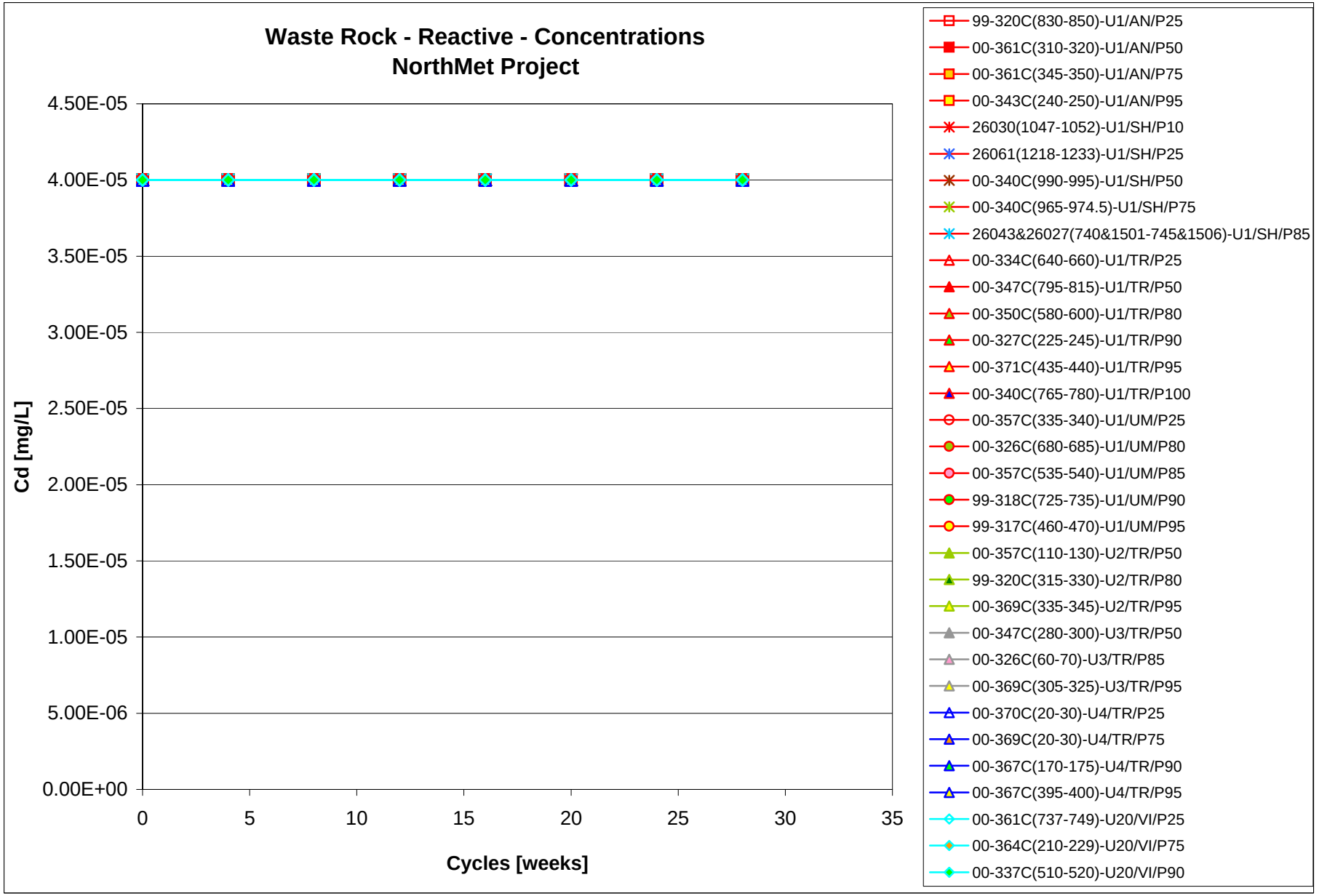

Waste Rock - Reactive - Concentrations
NorthMet Project
B [mg/L]
1
0.1
0.01
0.001
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

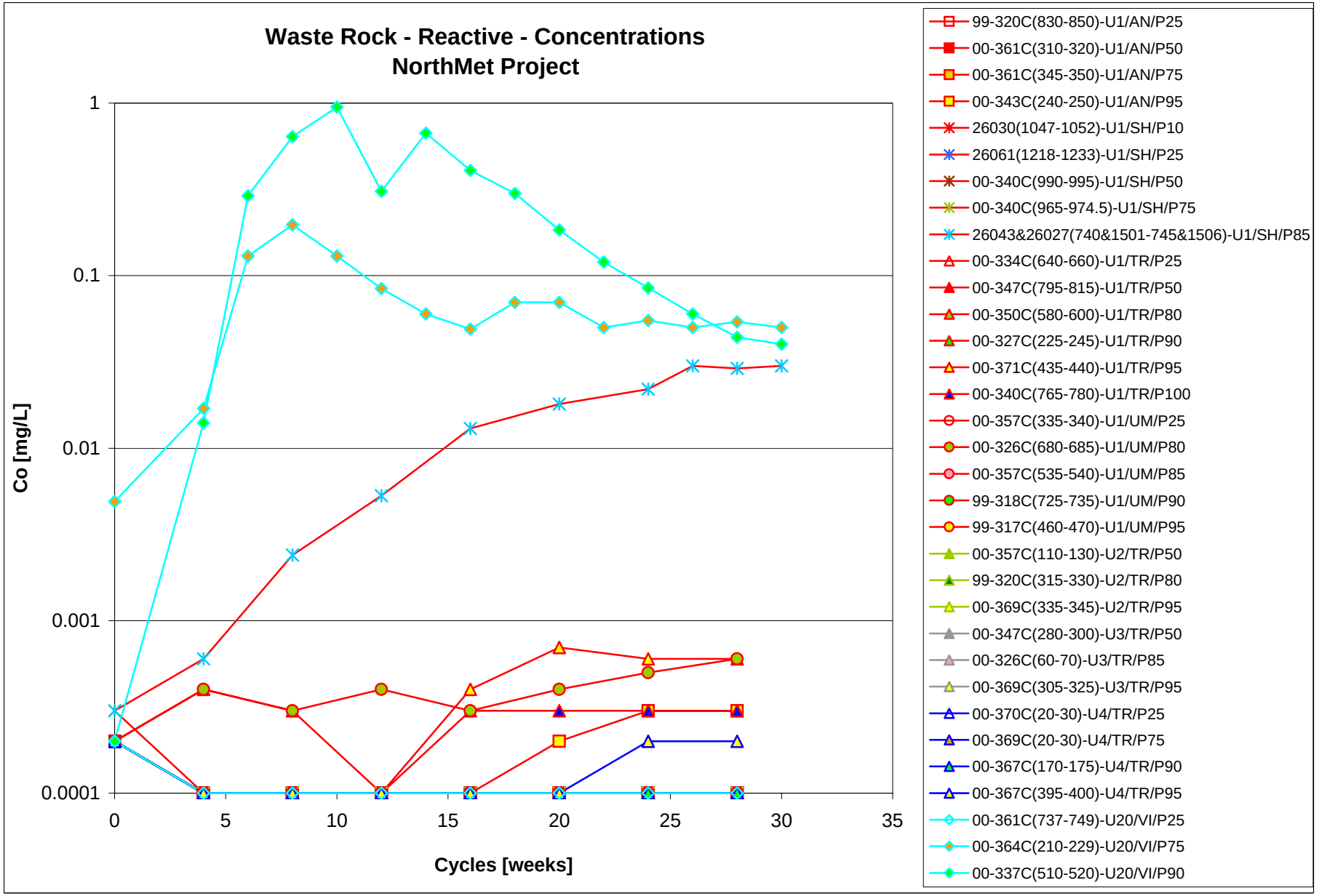

Waste Rock - Reactive - Concentrations
NorthMet Project
Ba [mg/L]
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

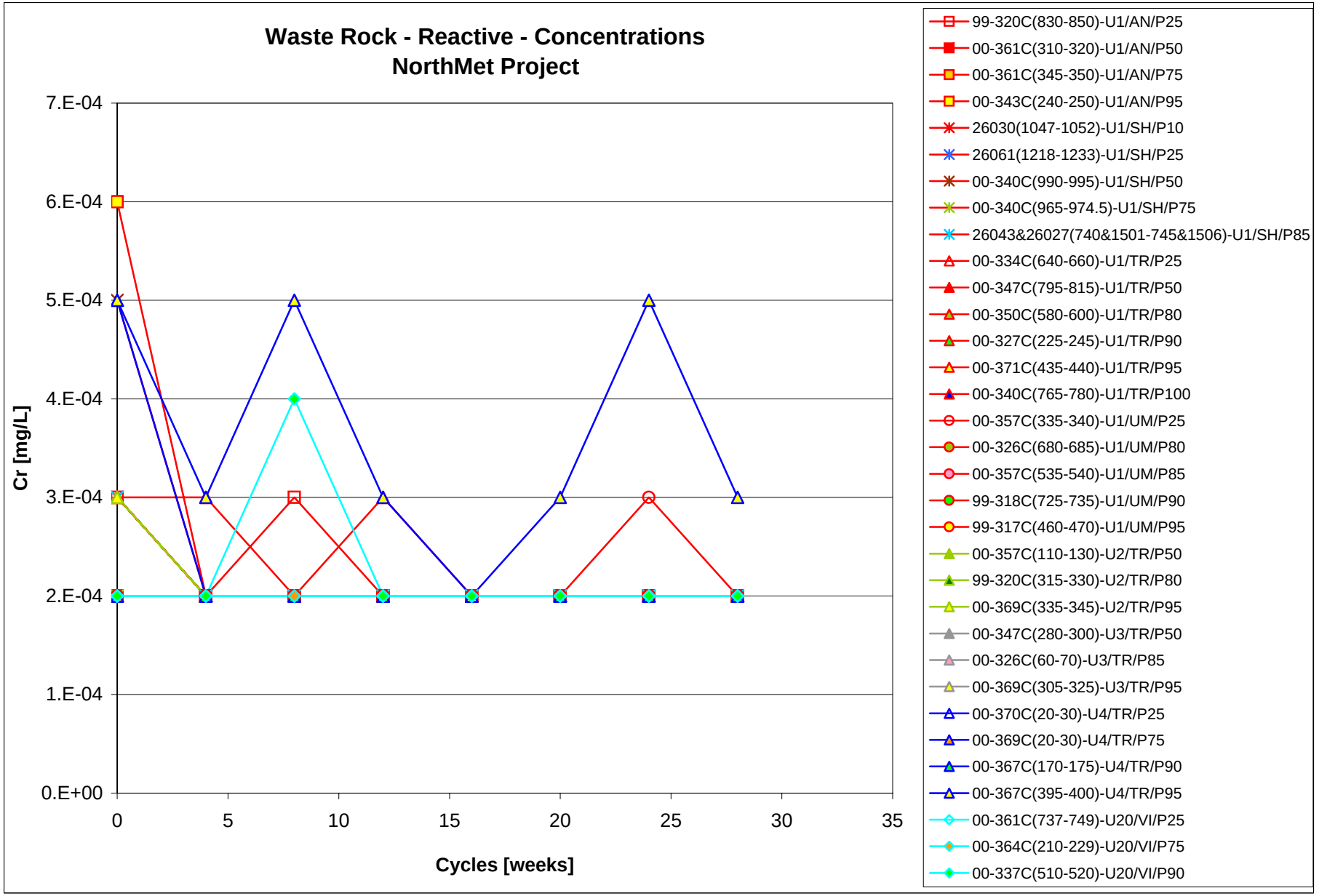

Waste Rock - Reactive - Concentrations
NorthMet Project
Be [mg/L]
1.0E-03
9.0E-04
8.0E-04
7.0E-04
6.0E-04
5.0E-04
4.0E-04
3.0E-04
2.0E-04
1.0E-04
0.0E+00
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

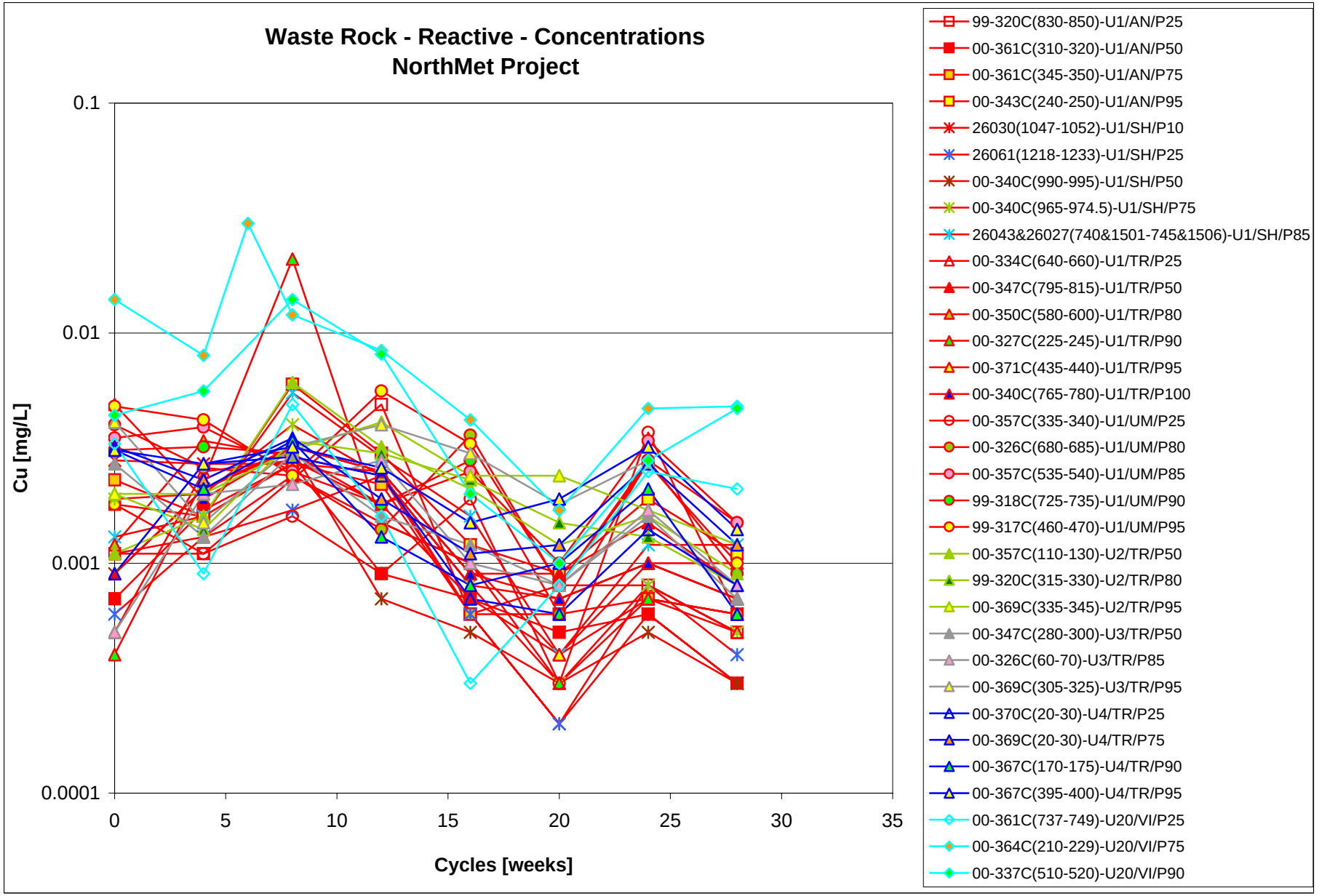

Waste Rock - Reactive - Concentrations
NorthMet Project
Bi [mg/L]
2.5E-04
2.0E-04
1.5E-04
1.0E-04
5.0E-05
0.0E+00
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

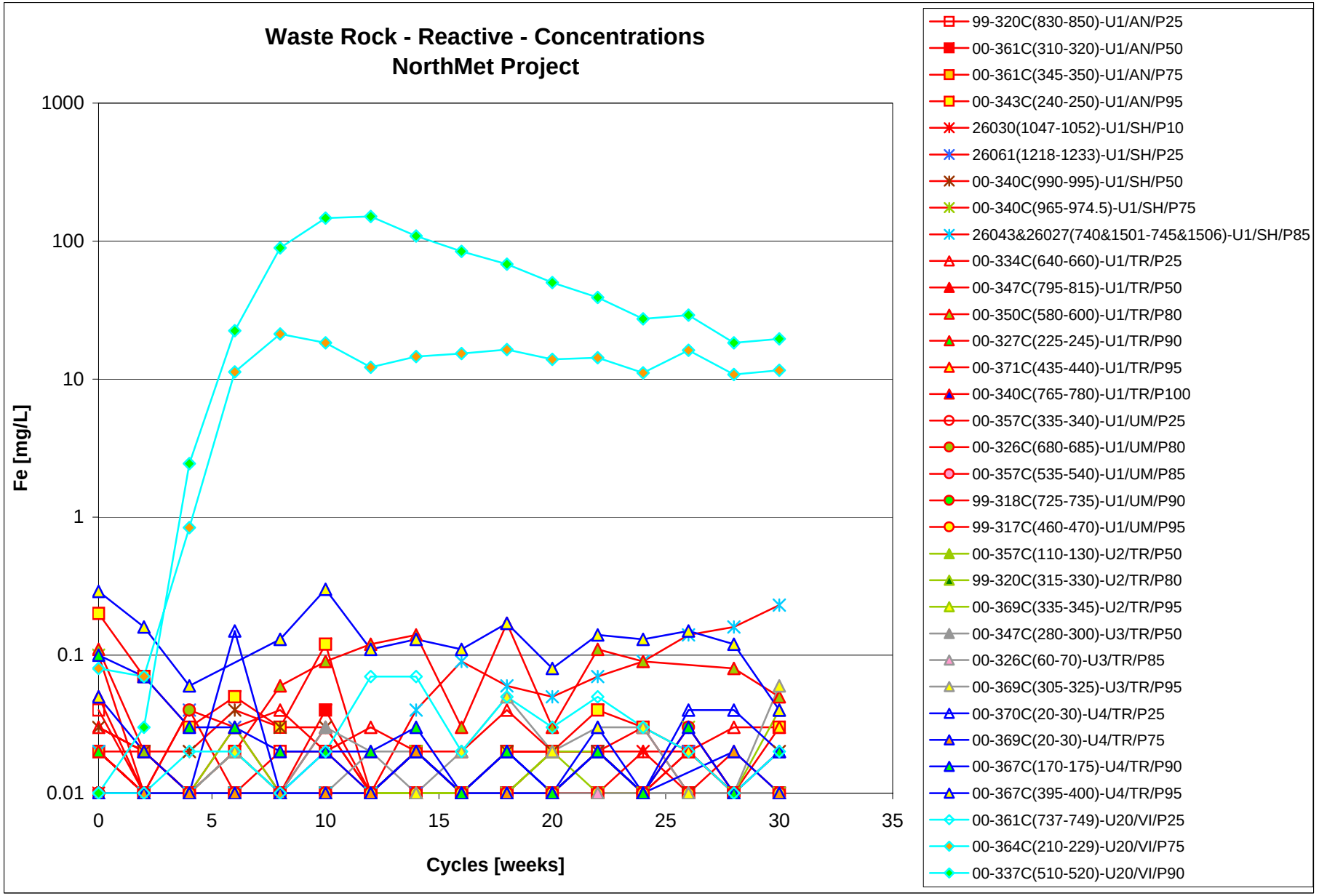

Waste Rock - Reactive - Concentrations
NorthMet Project
Ca [mg/L]
100
10
1
0.1
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

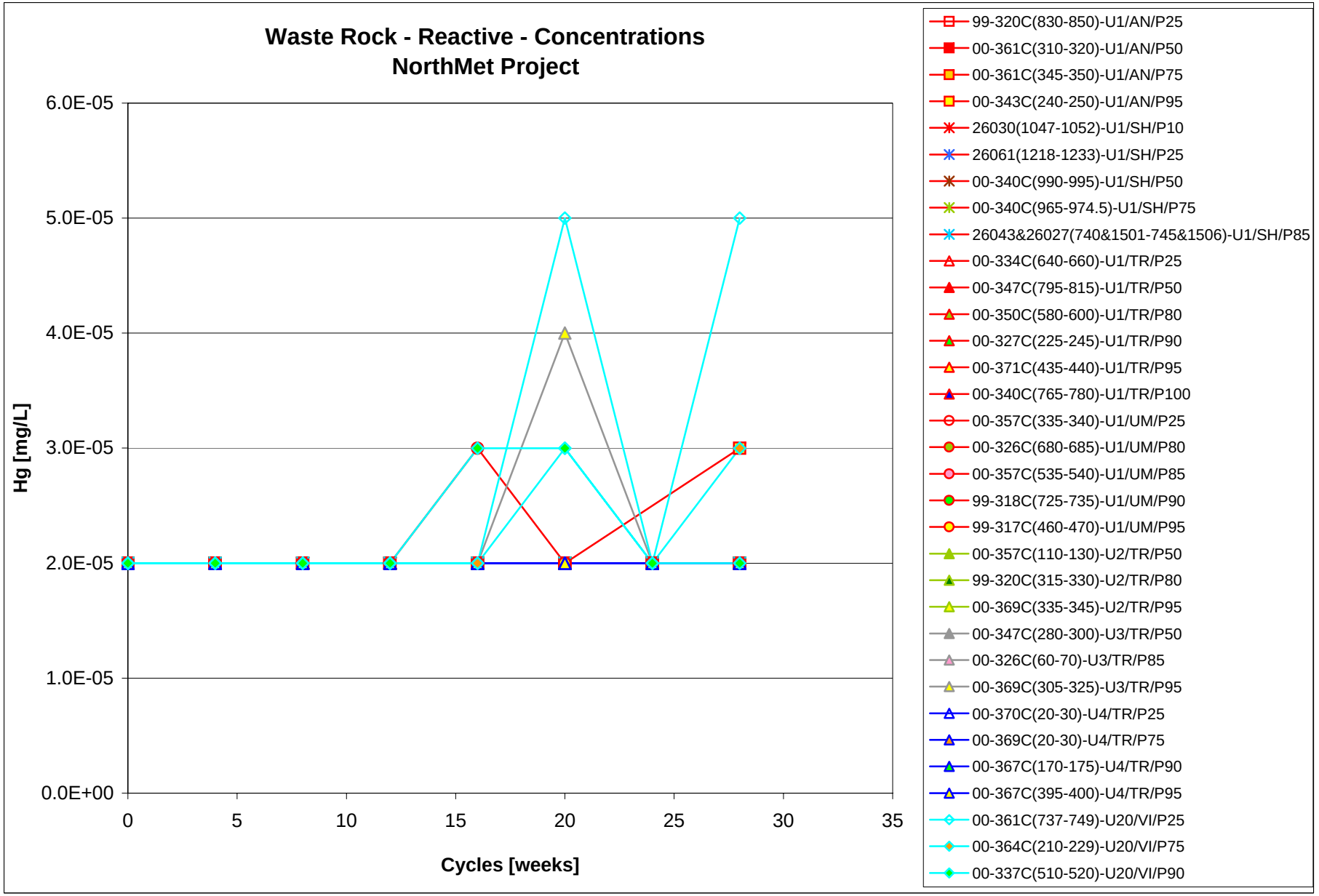

Waste Rock - Reactive - Concentrations
NorthMet Project
Cd [mg/L]
4.50E-05
4.00E-05
3.50E-05
3.00E-05
2.50E-05
2.00E-05
1.50E-05
1.00E-05
5.00E-06
0.00E+00
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

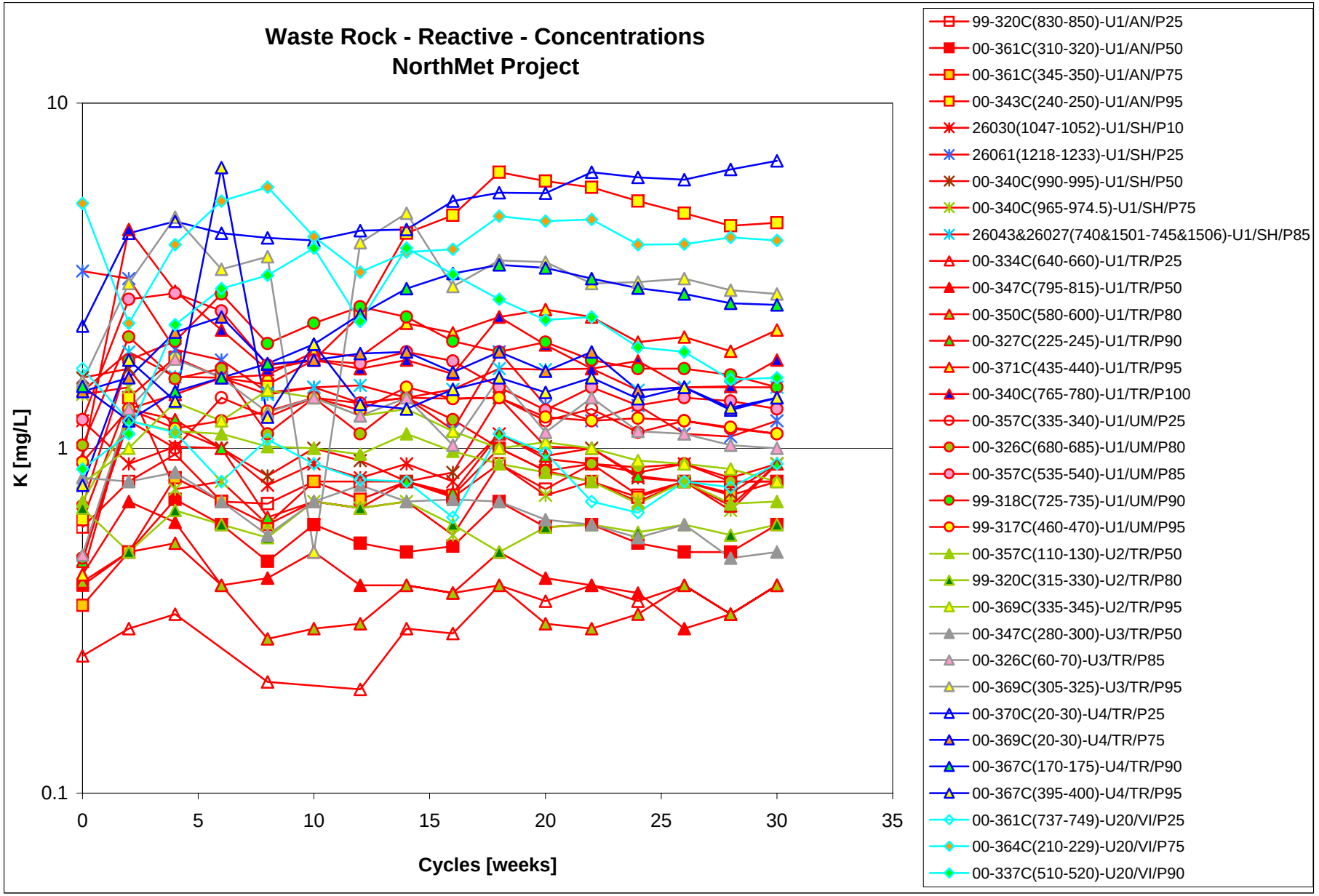

Waste Rock - Reactive - Concentrations
NorthMet Project
Co [mg/L]
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

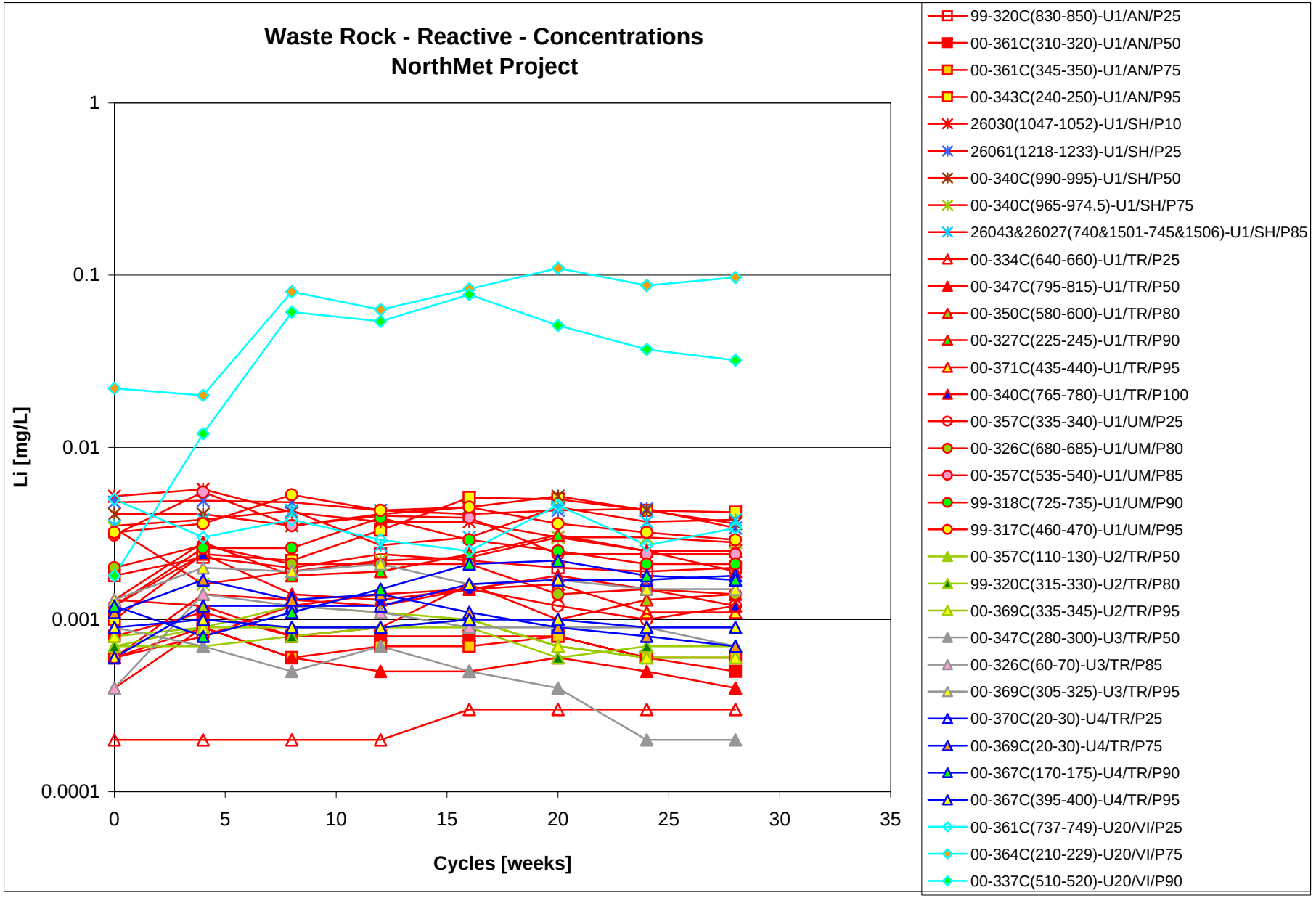

Waste Rock - Reactive - Concentrations
NorthMet Project
7.E-04
6.E-04
5.E-04
4.E-04
3.E-04
2.E-04
1.E-04
0.E+00
Cr [mg/L]
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

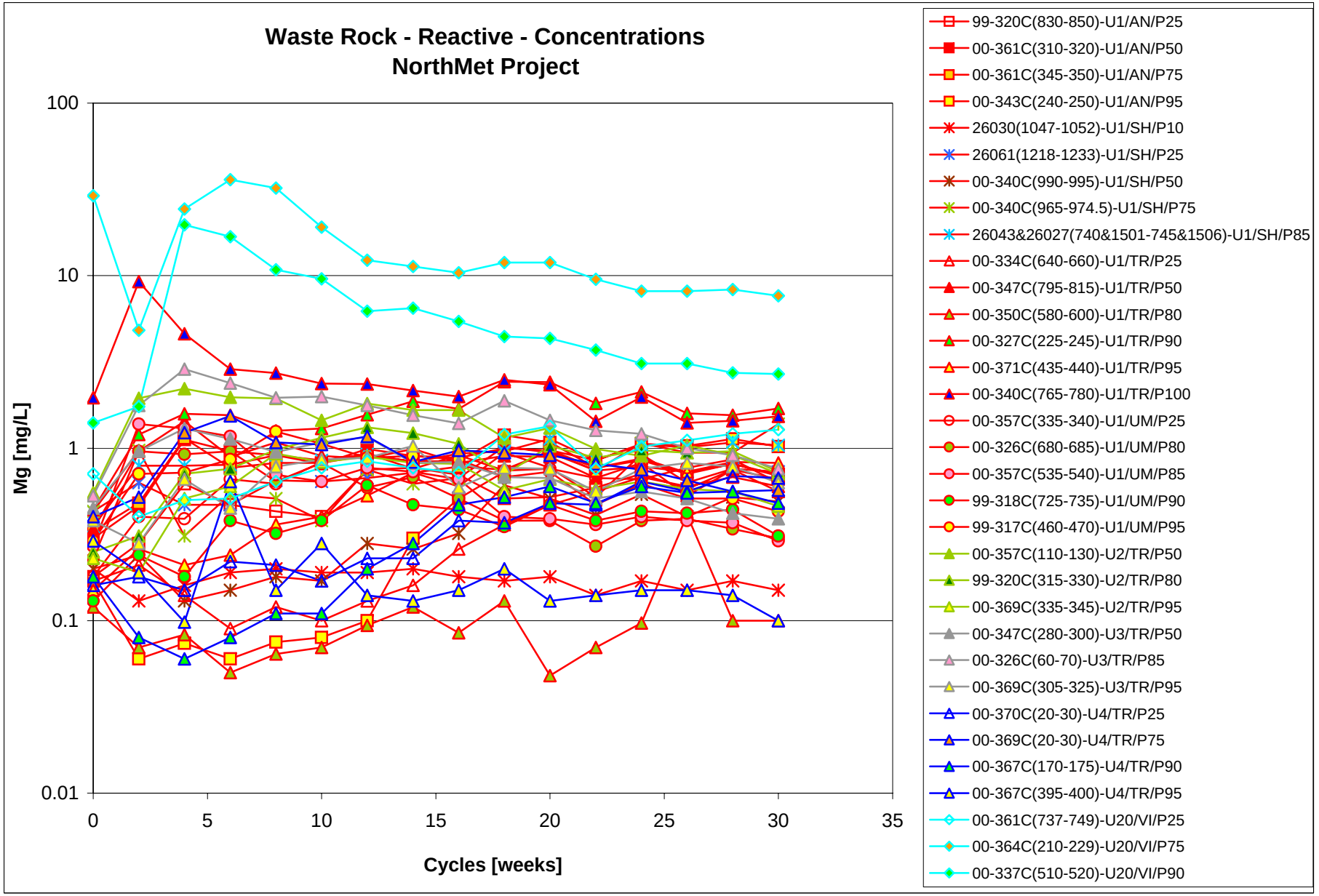

Waste Rock - Reactive - Concentrations
NorthMet Project
Cu [mg/L]
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

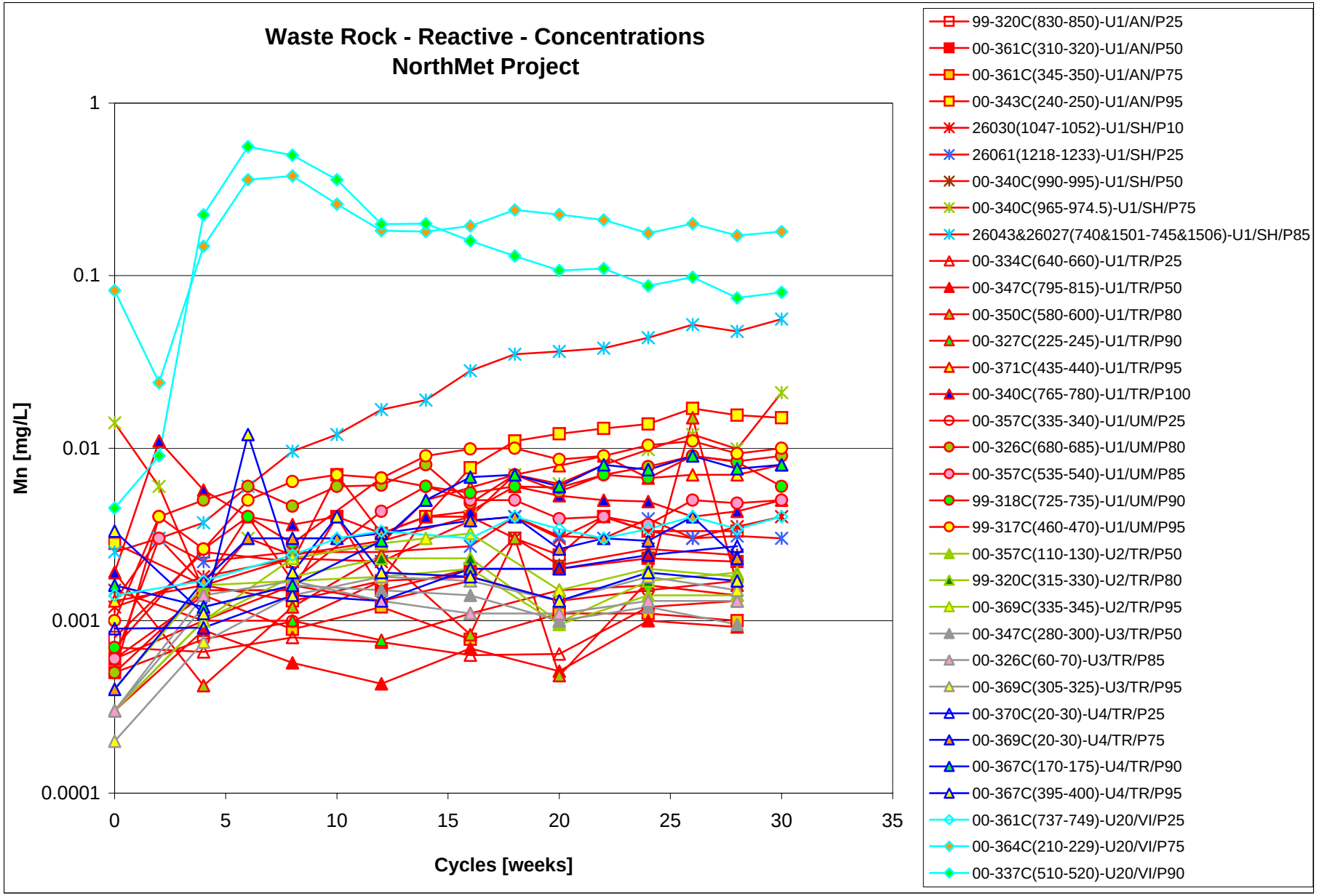

Waste Rock - Reactive - Concentrations
NorthMet Project
Fe [mg/L]
1000
100
10
1
0.1
0.01
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

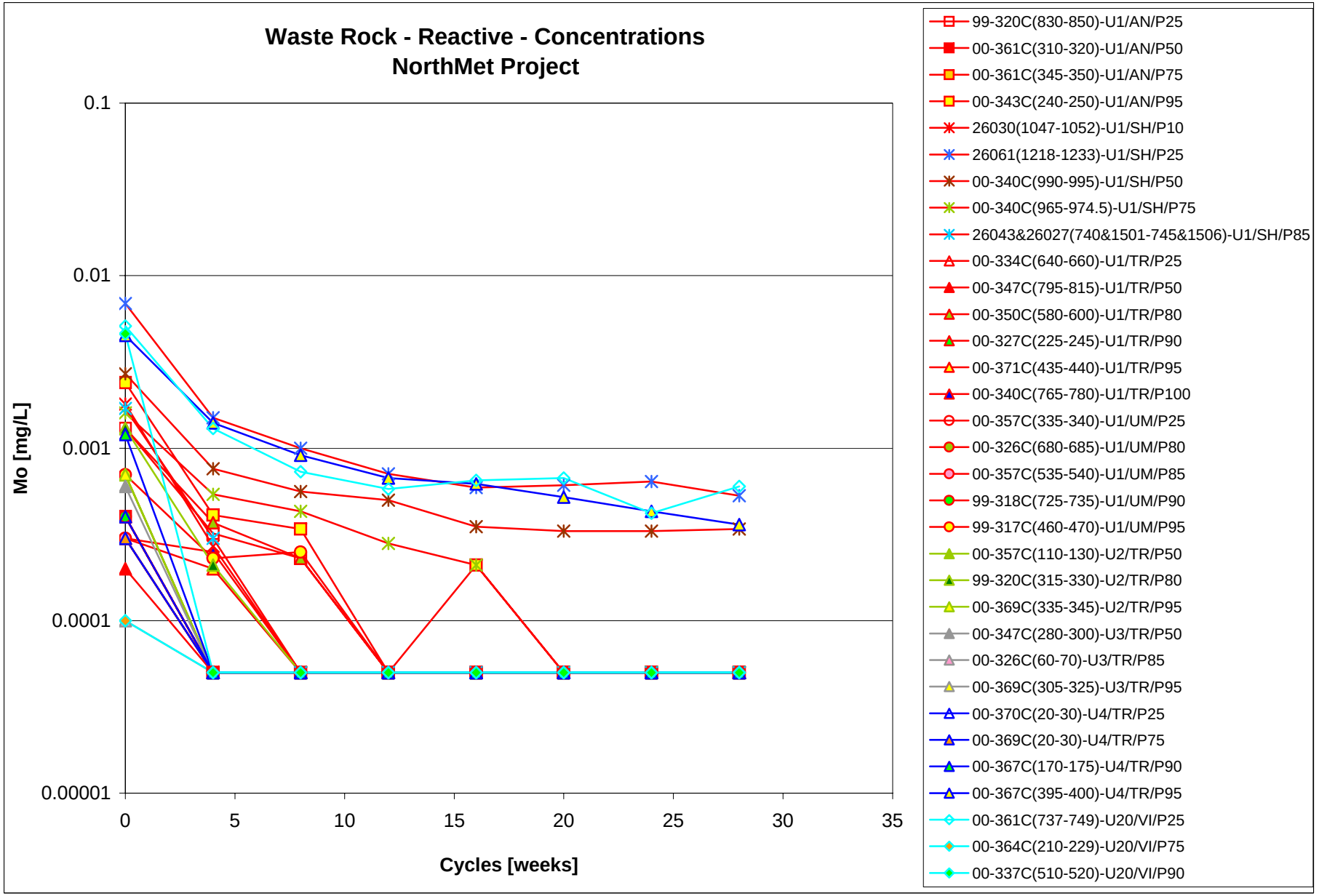

Waste Rock - Reactive - Concentrations
NorthMet Project
Hg [mg/L]
6.0E-05
5.0E-05
4.0E-05
3.0E-05
2.0E-05
1.0E-05
0.0E+00
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

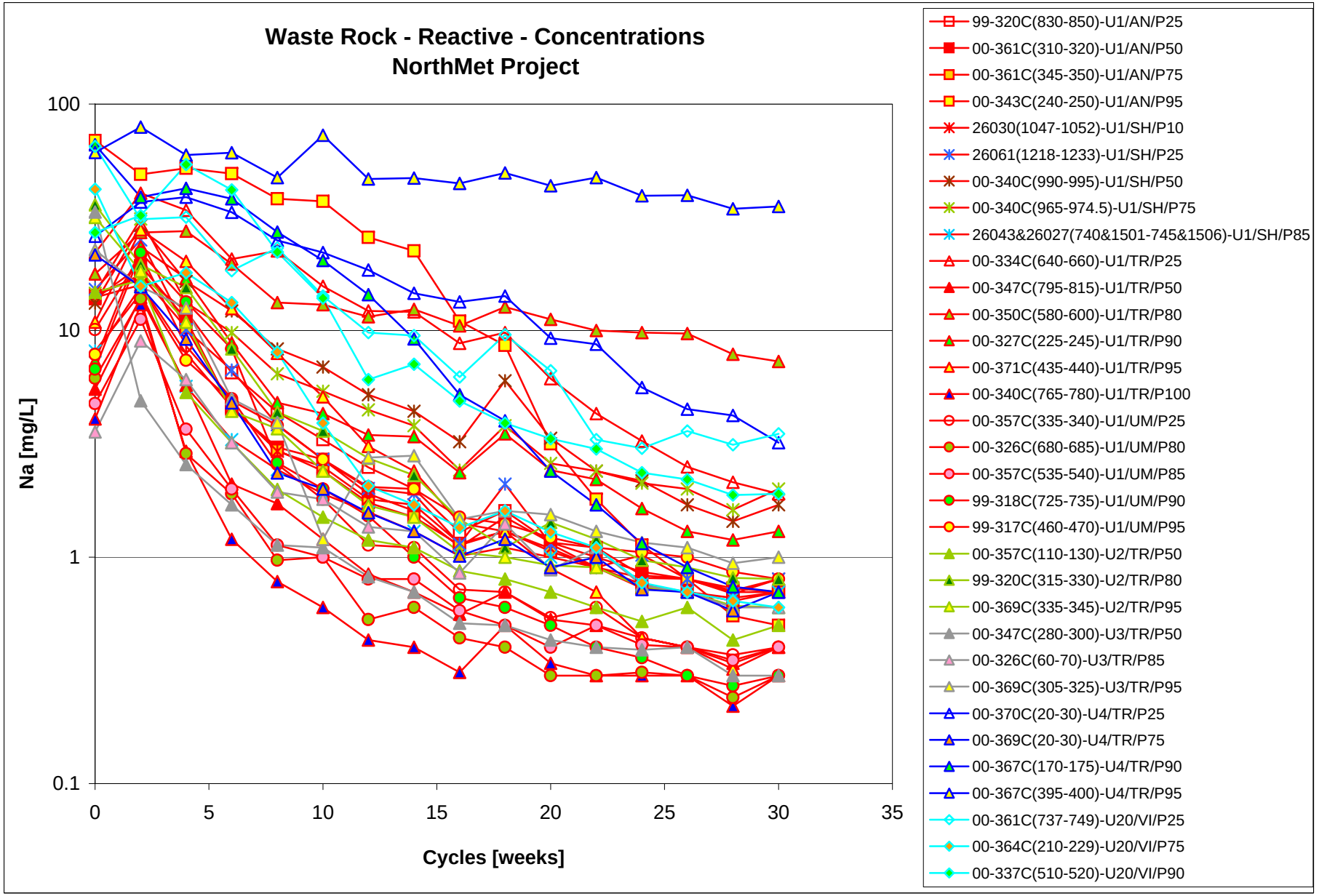

Waste Rock - Reactive - Concentrations
NorthMet Project
K [mg/L]
10
1
0.1
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

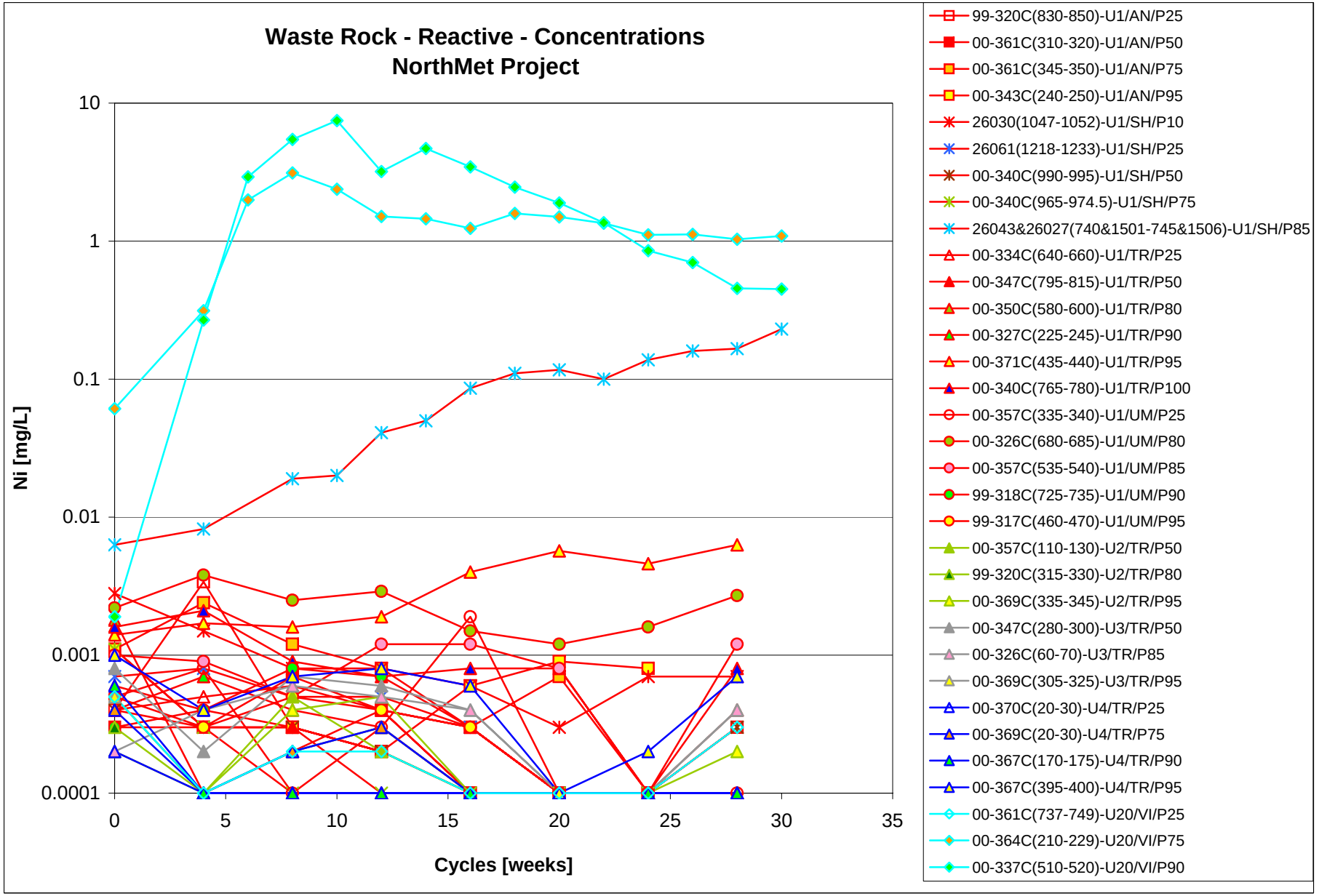

Waste Rock - Reactive - Concentrations
NorthMet Project
Li [mg/L]
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

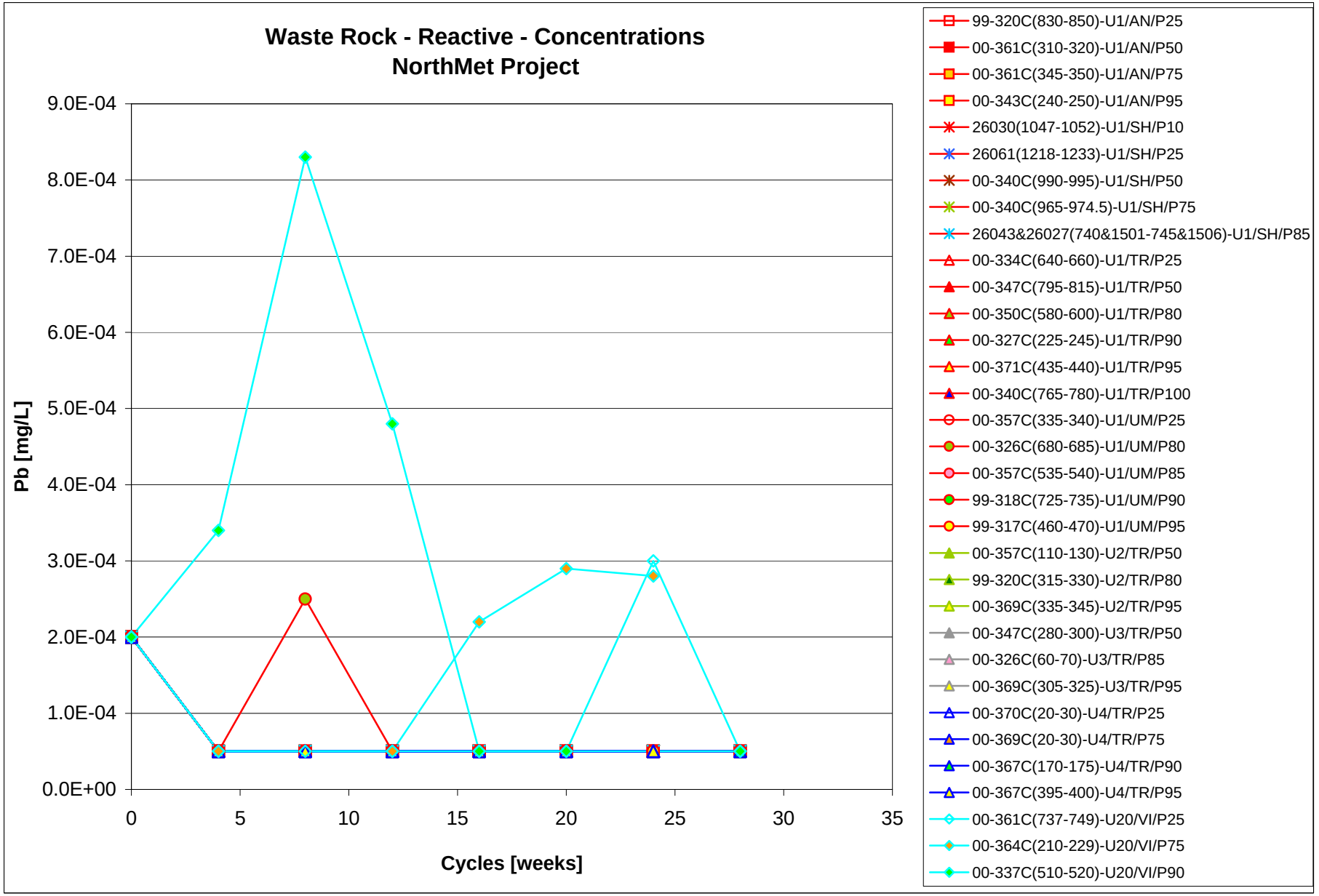

Waste Rock - Reactive - Concentrations
NorthMet Project
Mg [mg/L]
100
10
1
0.1
0.01
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

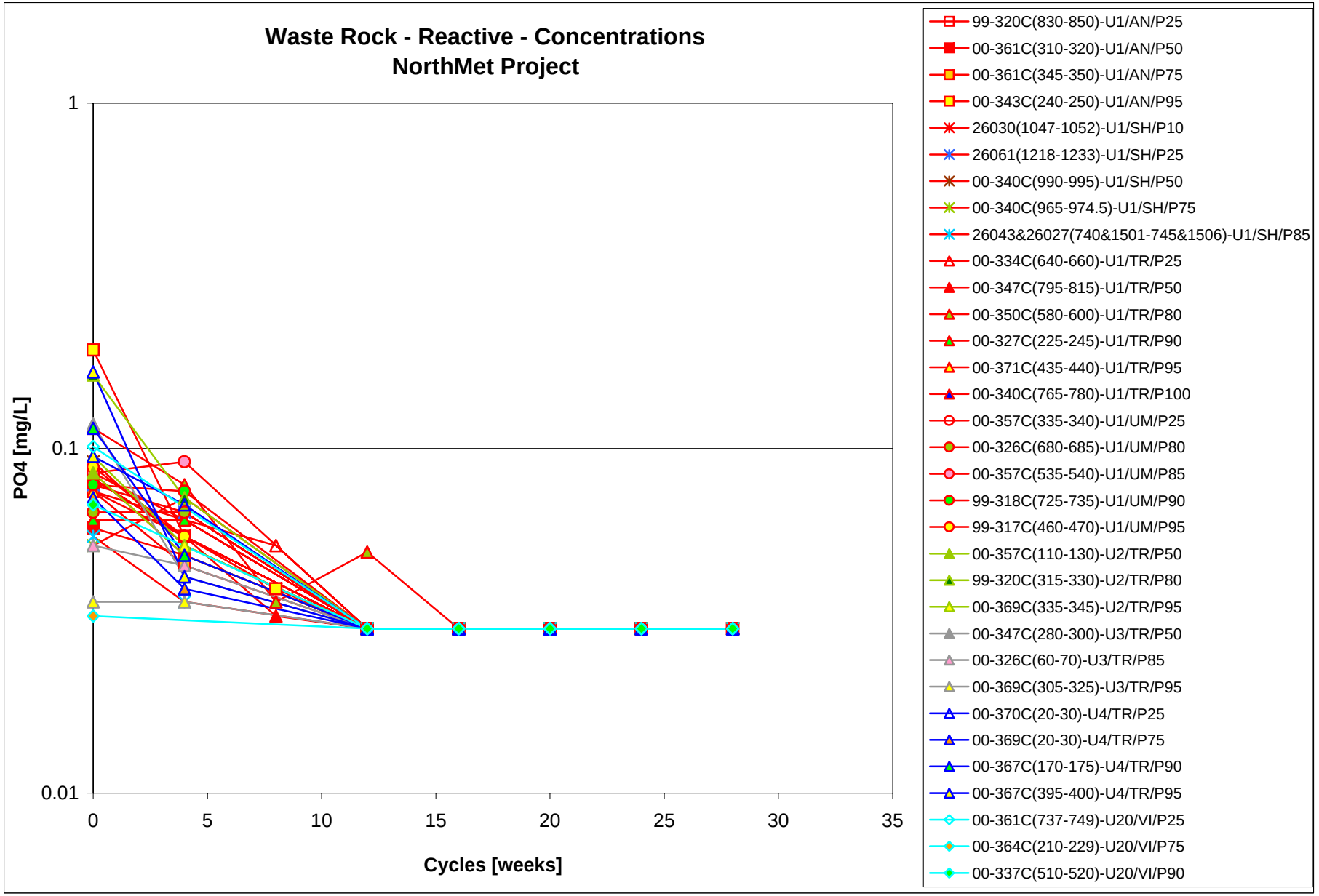

Waste Rock - Reactive - Concentrations
NorthMet Project
Mn [mg/L]
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

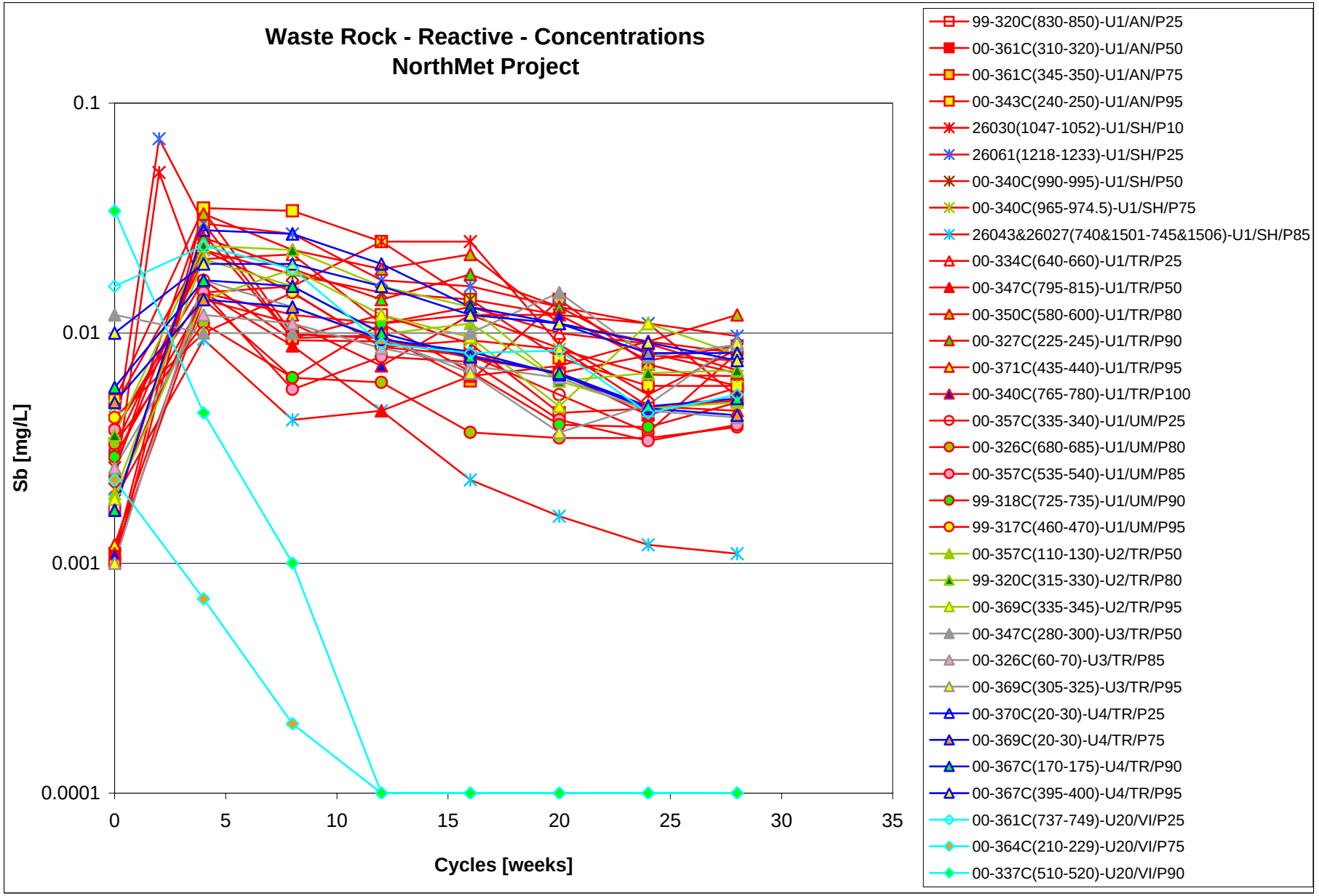

Waste Rock - Reactive - Concentrations
NorthMet Project
Mo [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

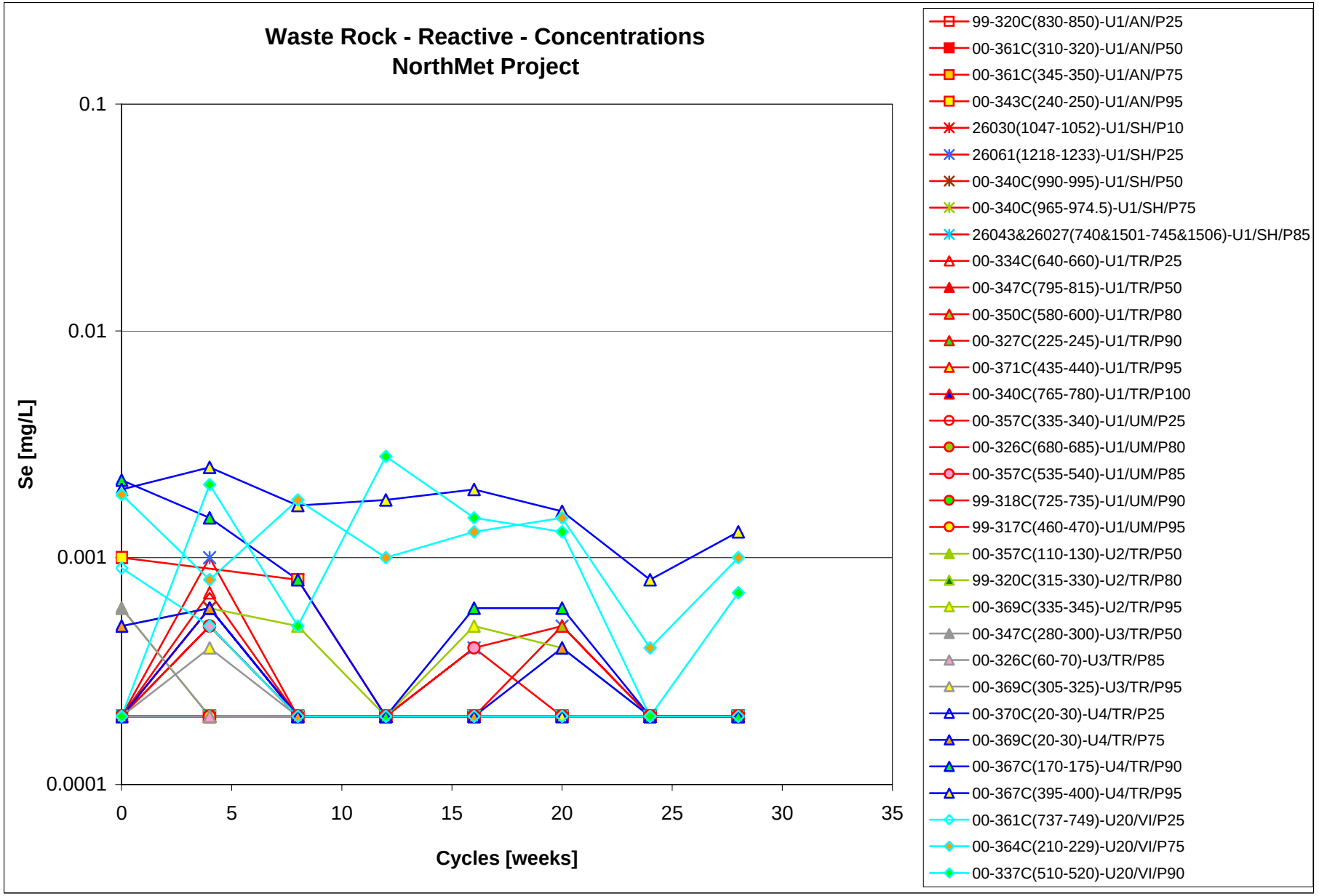

Waste Rock - Reactive - Concentrations
NorthMet Project
Na [mg/L]
100
10
1
0.1
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

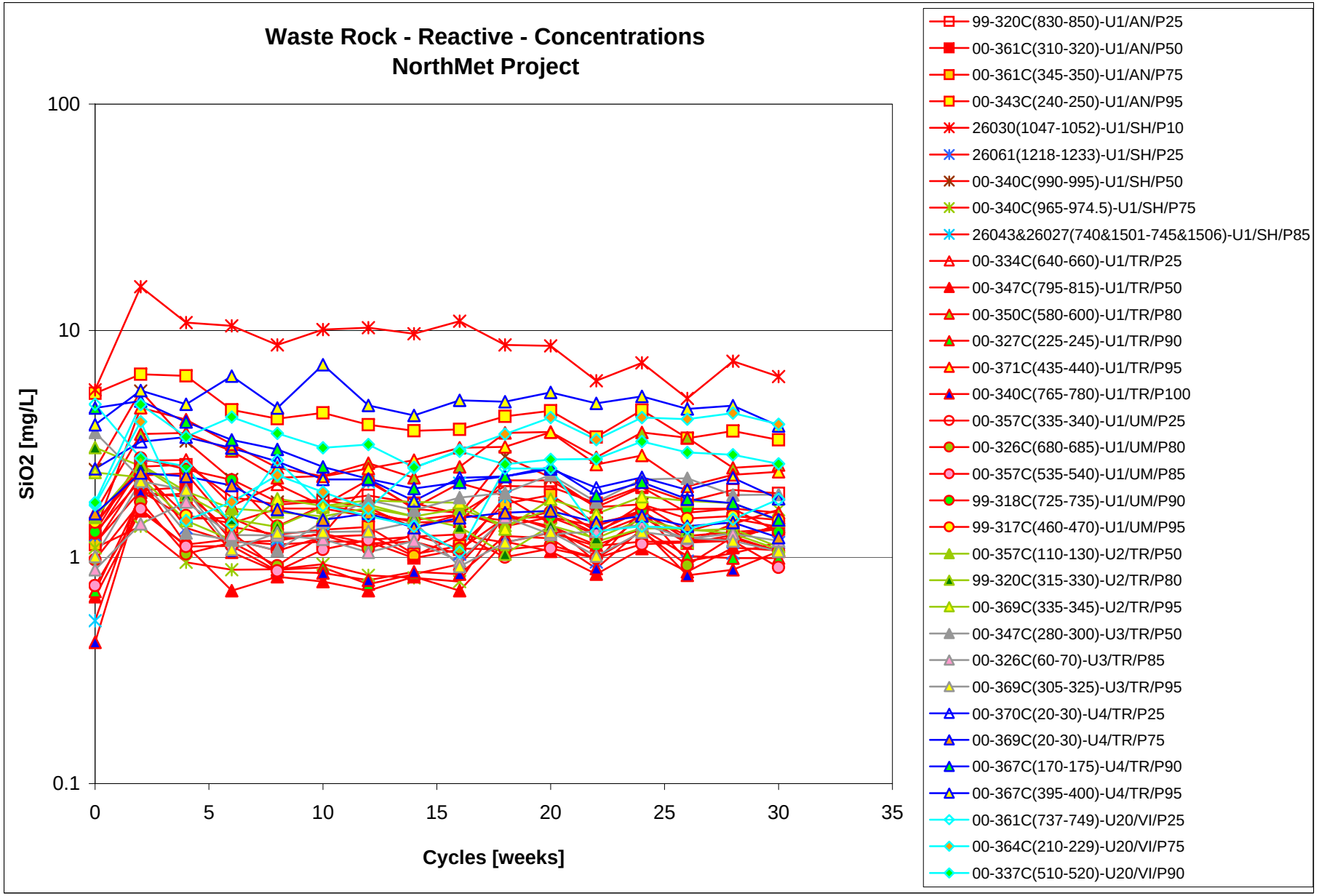

Waste Rock - Reactive - Concentrations
NorthMet Project
Ni [mg/L]
10
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

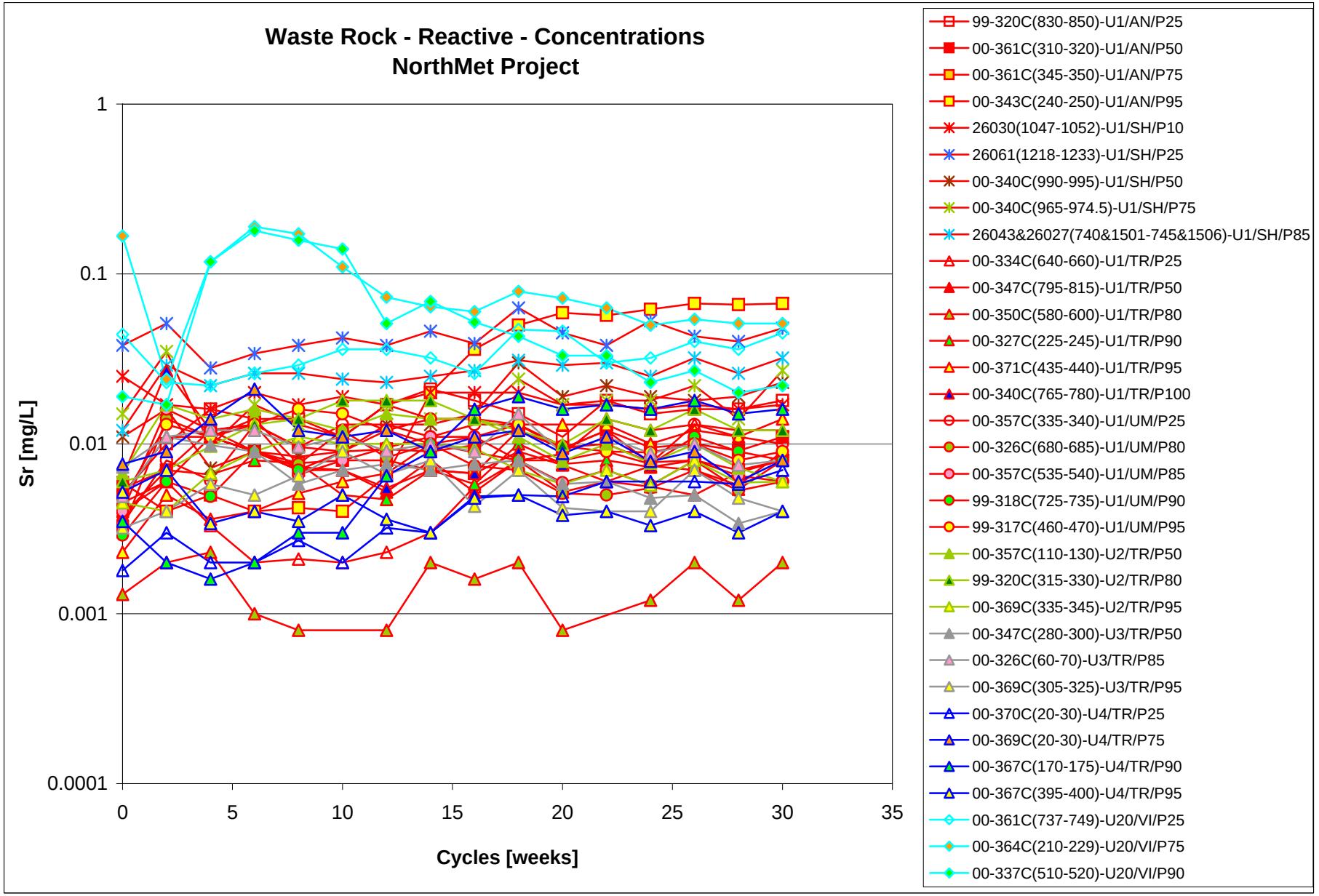

Waste Rock - Reactive - Concentrations
NorthMet Project
Pb [mg/L]
9.0E-04
8.0E-04
7.0E-04
6.0E-04
5.0E-04
4.0E-04
3.0E-04
2.0E-04
1.0E-04
0.0E+00
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

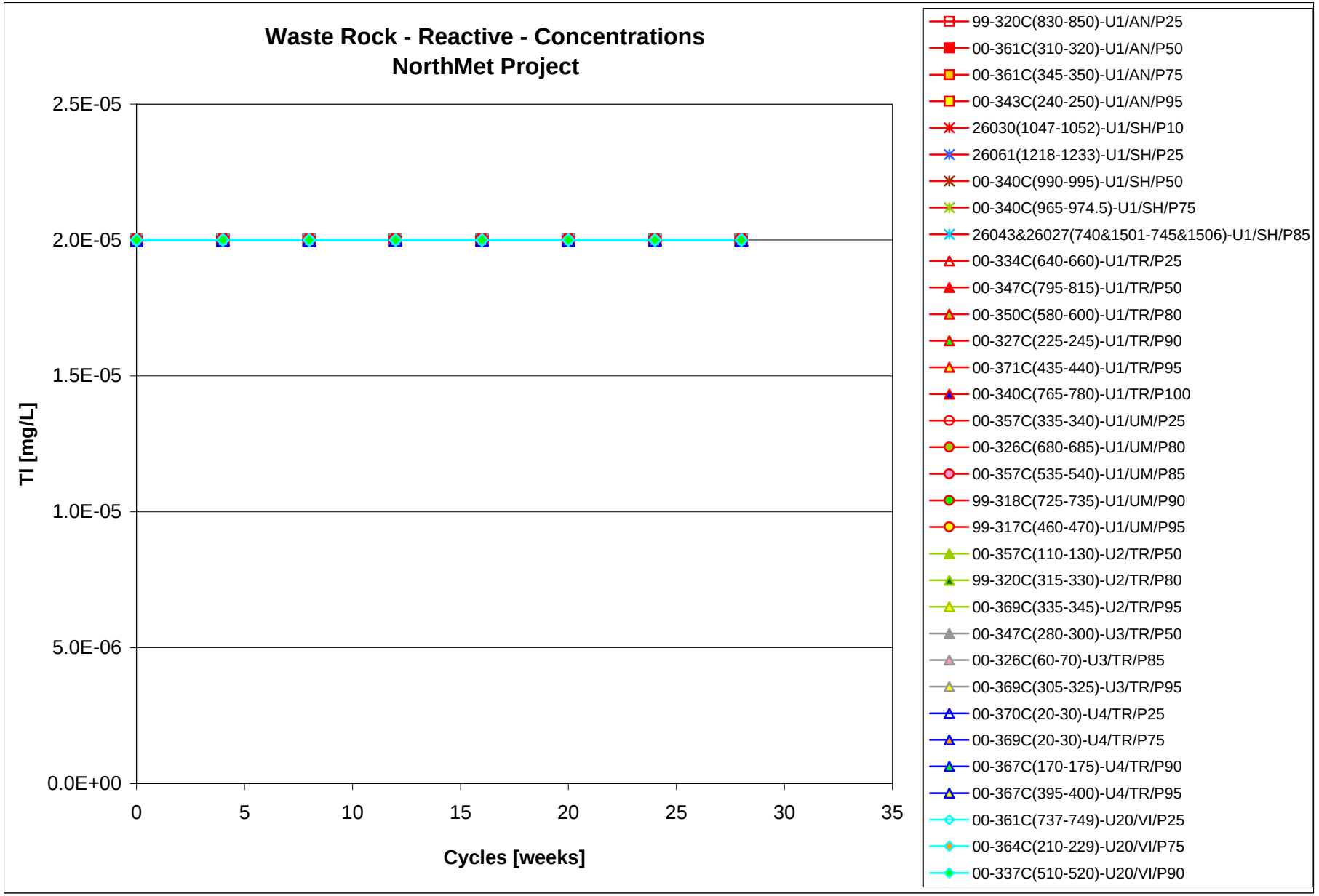

Waste Rock - Reactive - Concentrations
NorthMet Project
PO4 [mg/L]
1
0.1
0.01
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

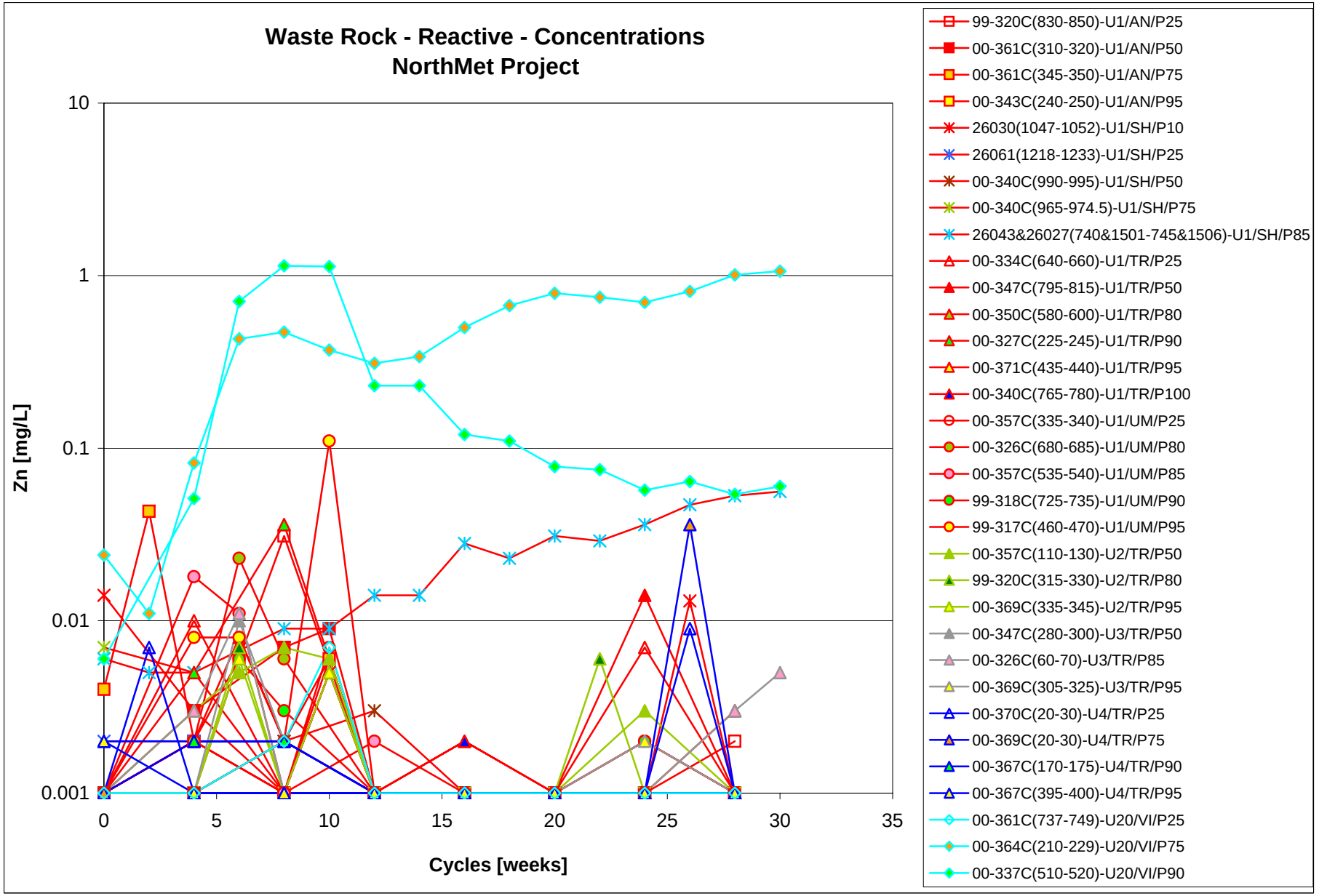

Waste Rock - Reactive - Concentrations
NorthMet Project
Sb [mg/L]
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90


Waste Rock - Reactive - Concentrations
NorthMet Project
Se [mg/L]
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

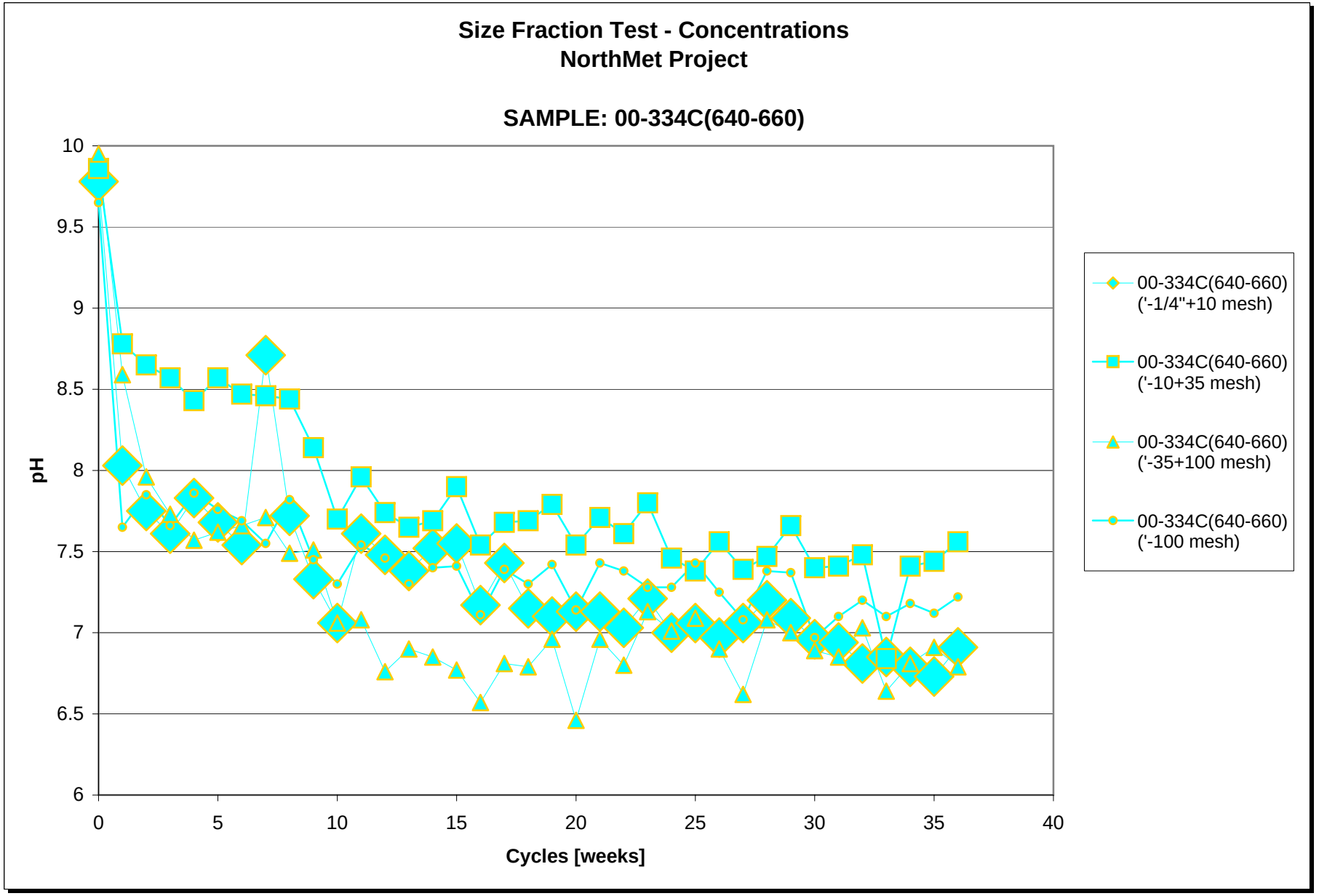

Waste Rock - Reactive - Concentrations
NorthMet Project
SiO2 [mg/L]
100
10
1
0.1
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

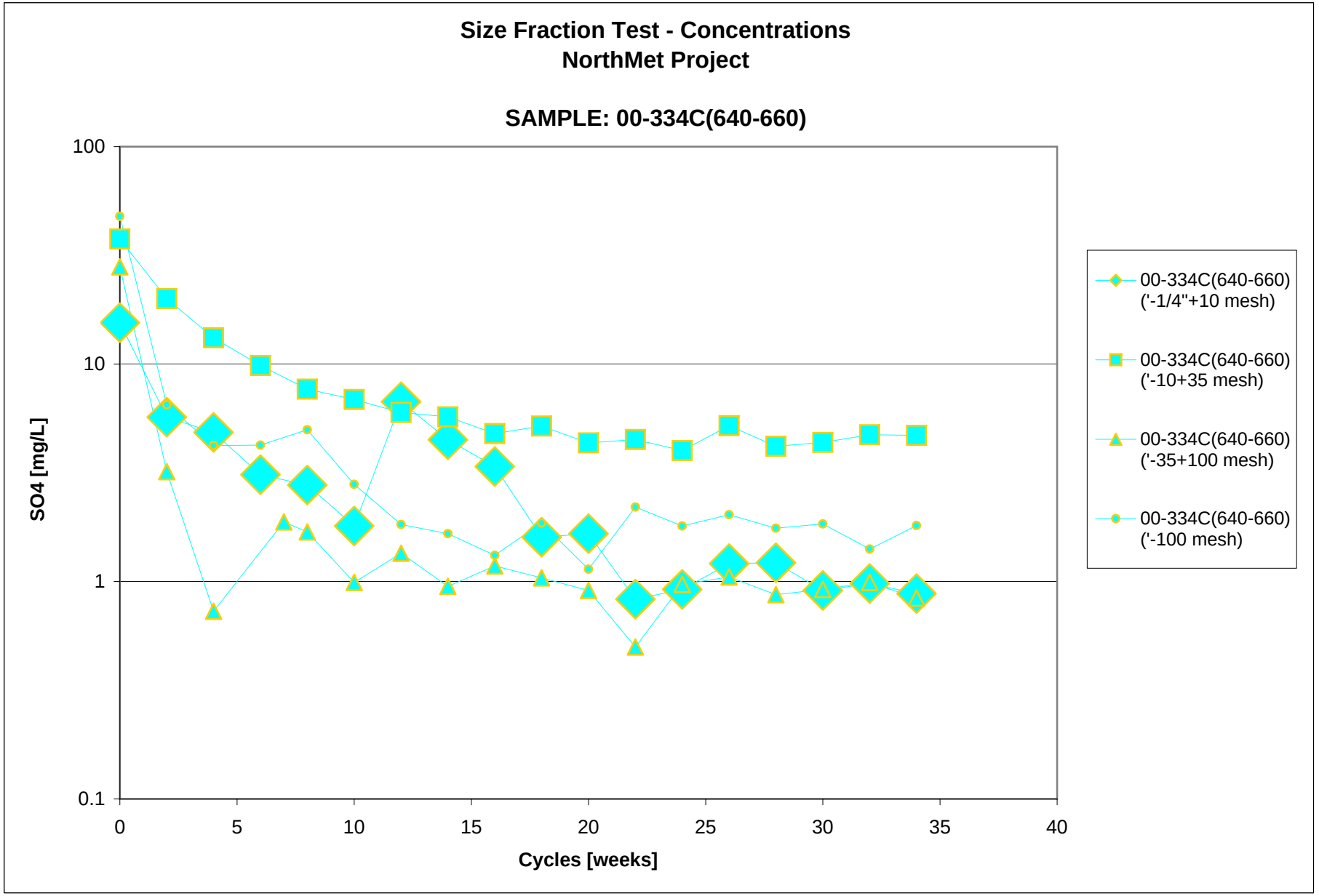

Waste Rock - Reactive - Concentrations
NorthMet Project
Sr [mg/L]
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

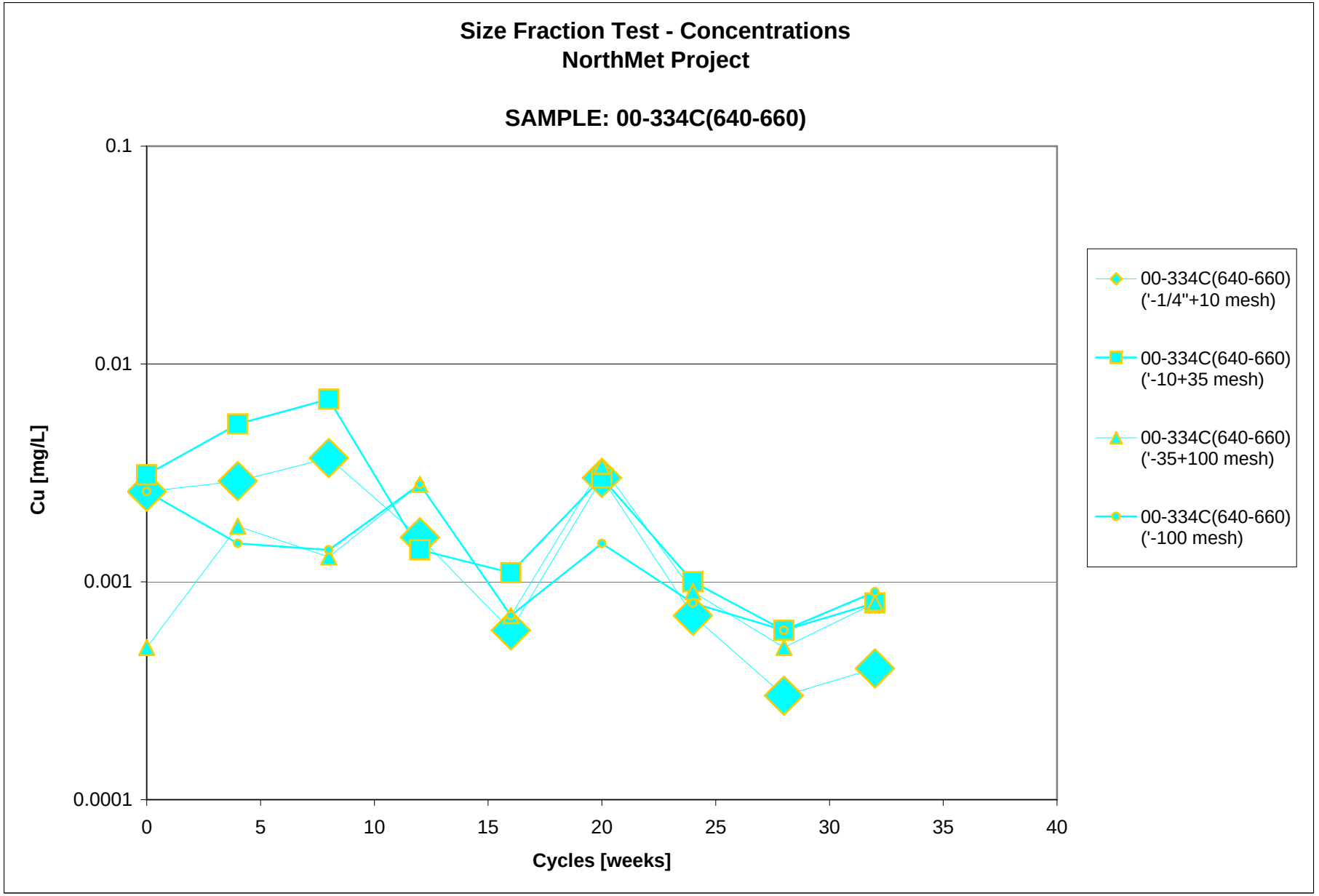

Waste Rock - Reactive - Concentrations
NorthMet Project
Tl [mg/L]
2.5E-05
2.0E-05
1.5E-05
1.0E-05
5.0E-06
0.0E+00
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

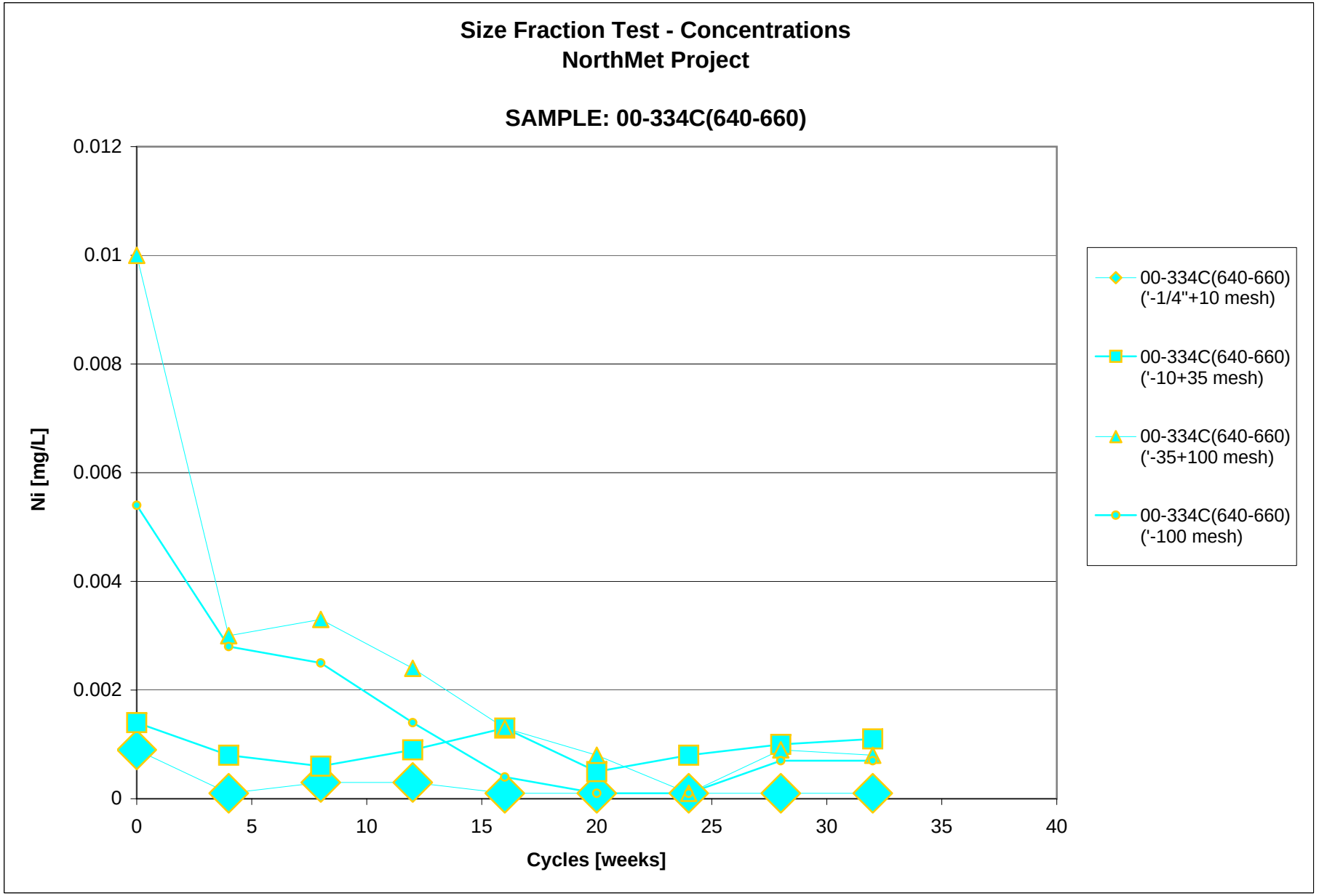

Waste Rock - Reactive - Concentrations
NorthMet Project
Zn [mg/L]
10
1
0.1
0.01
0.001
0
5
10
15
20
25
30
35
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90

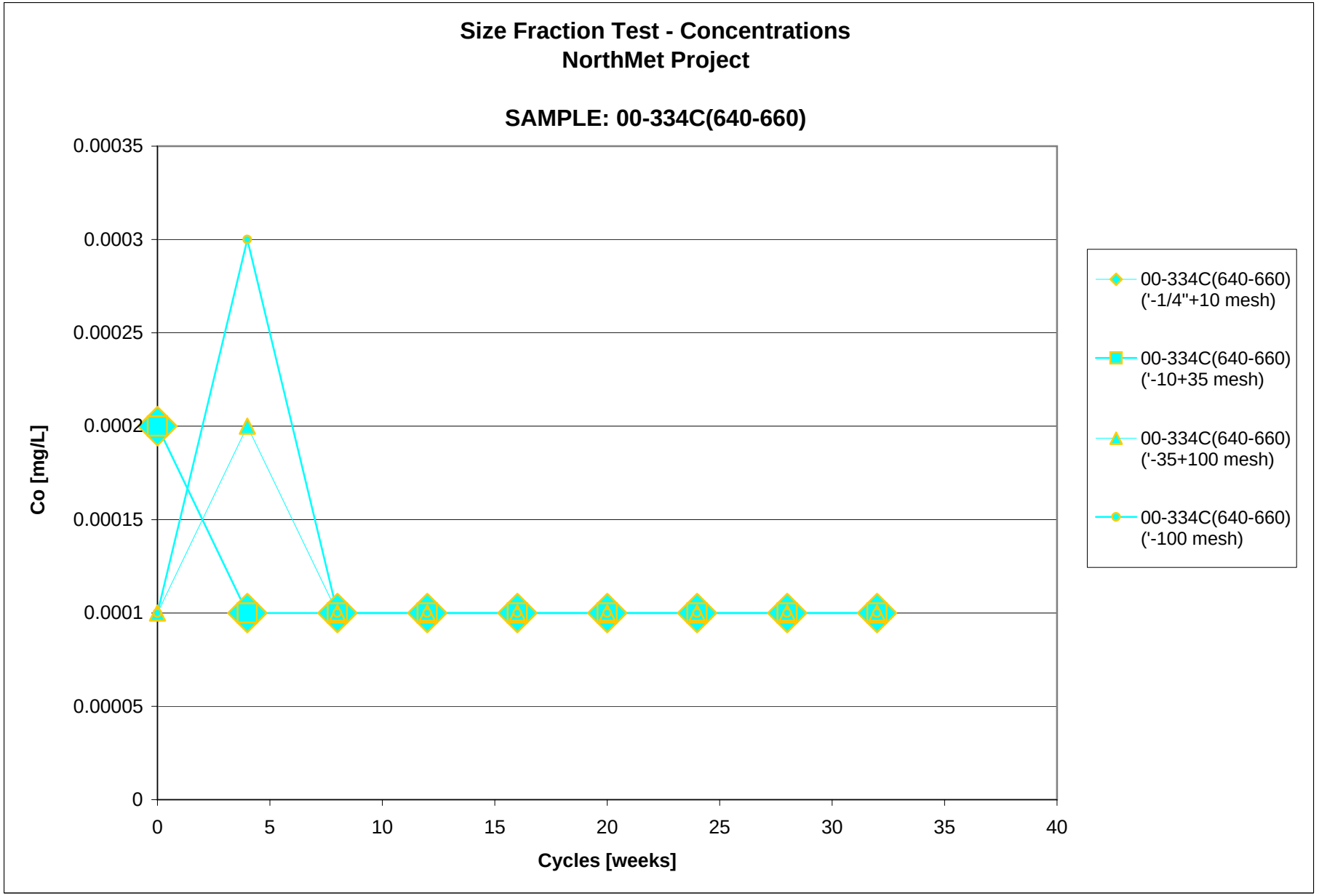
Appendix G
Loading Graphs – Size Fractions

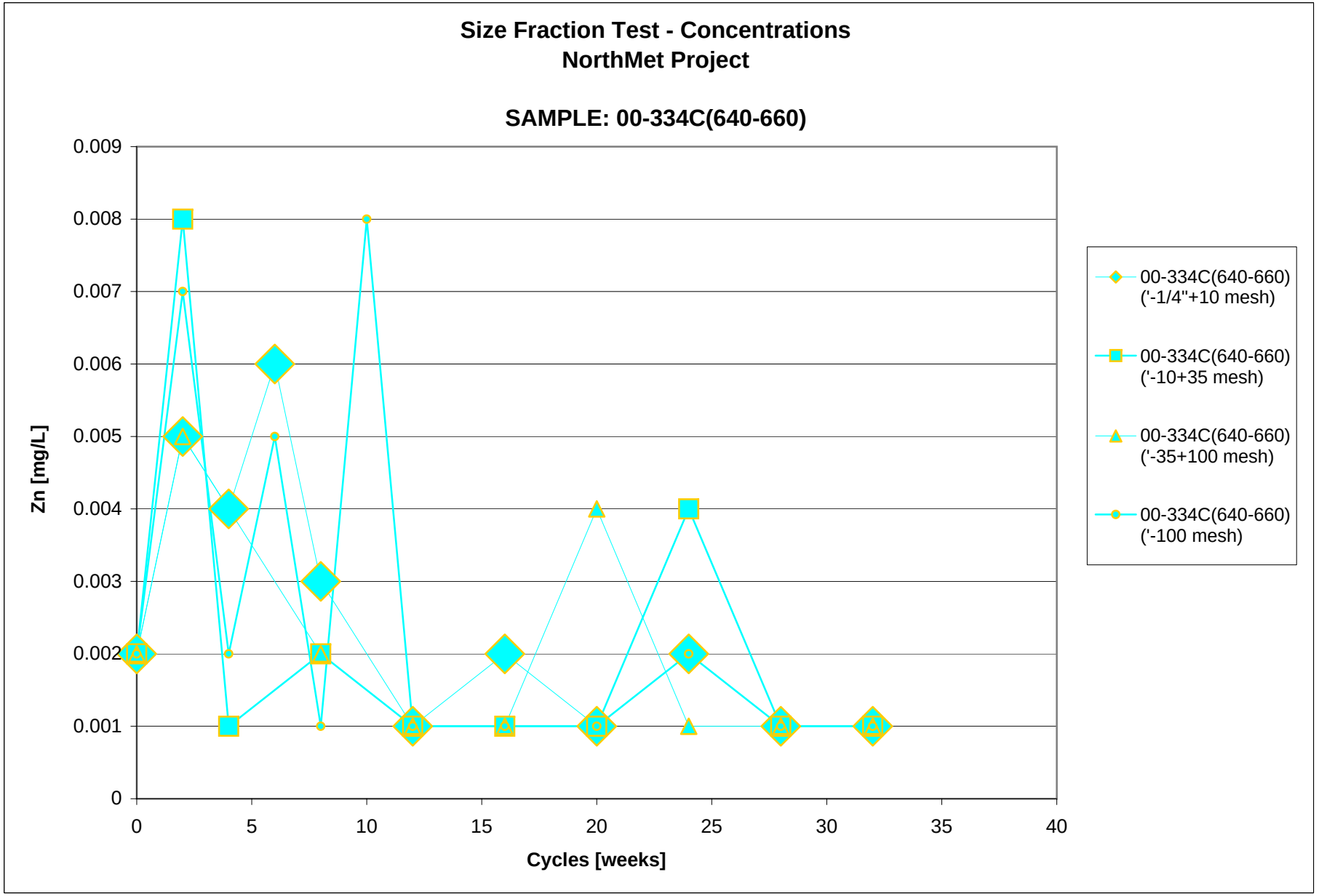

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-334C(640-660)
pH
10
9.5
9
8.5
8
7.5
7
6.5
6
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-334C(640-660) ('-1/4"+10 mesh)
00-334C(640-660) ('-10+35 mesh)
00-334C(640-660) ('-35+100 mesh)
00-334C(640-660) ('-100 mesh)

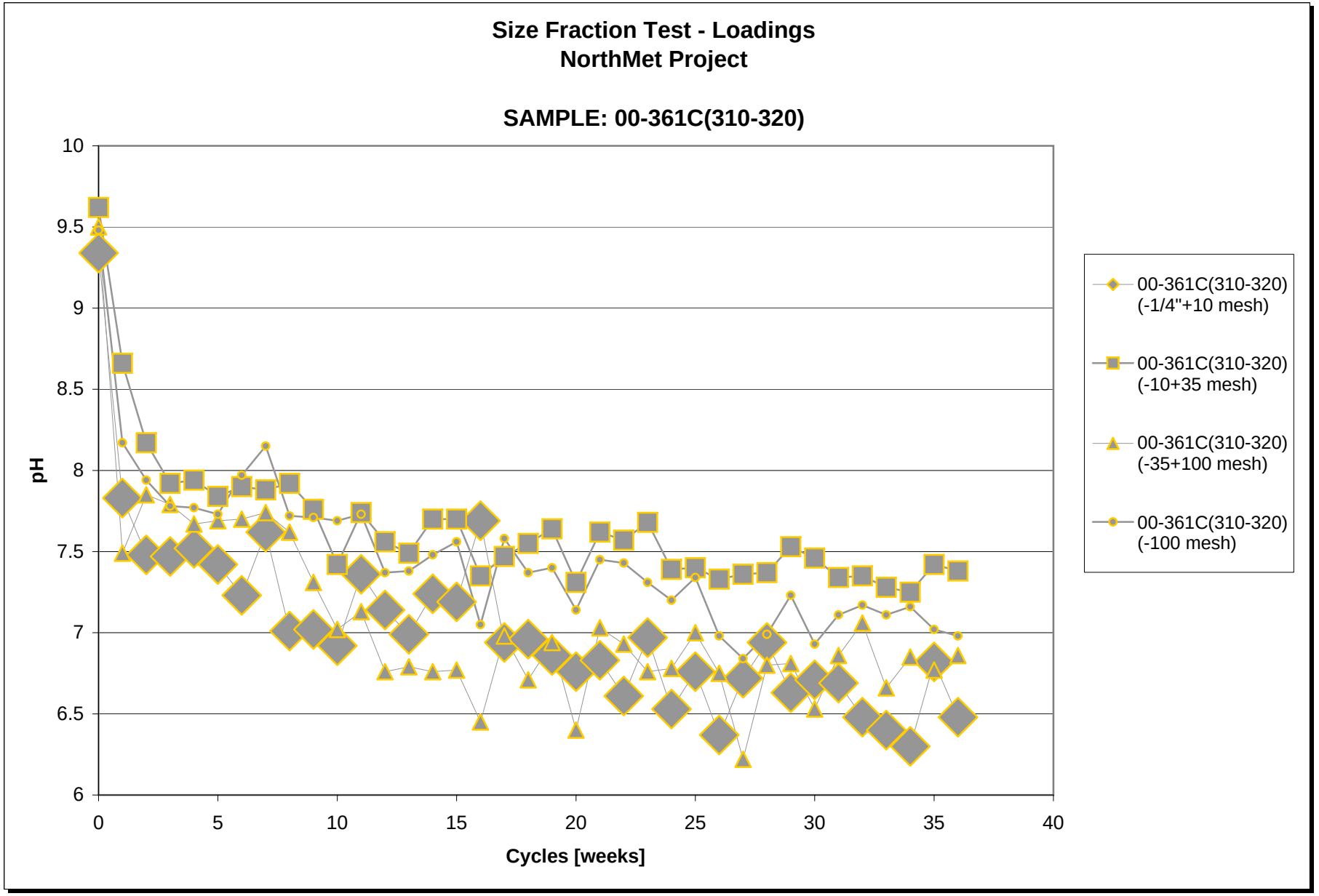

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-334C(640-660)
SO4 [mg/L]
100
10
1
0.1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-334C(640-660) ('-1/4"+10 mesh)
00-334C(640-660) ('-10+35 mesh)
00-334C(640-660) ('-35+100 mesh)
00-334C(640-660) ('-100 mesh)

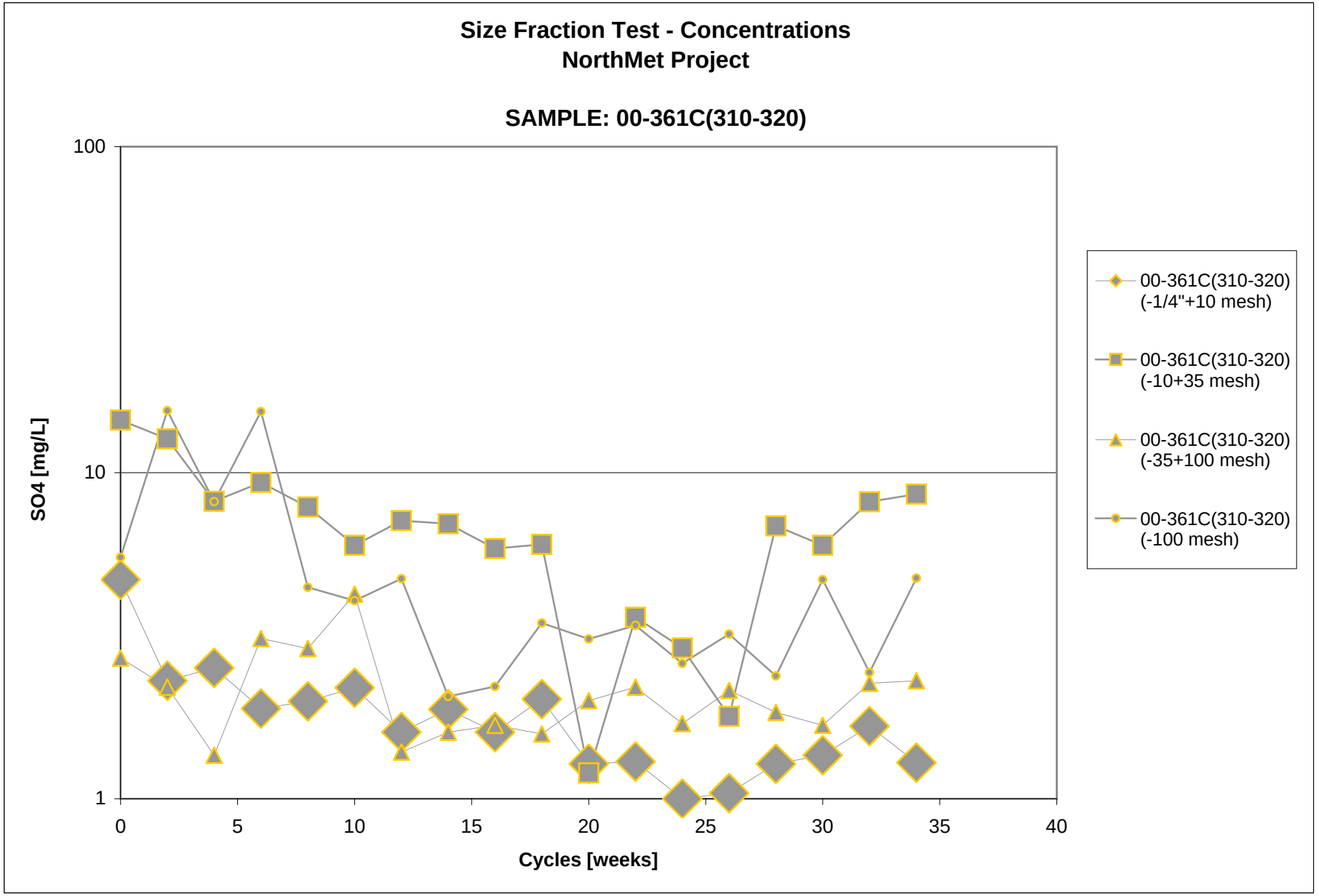

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-334C(640-660)
Cu [mg/L]
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-334C(640-660) ('-1/4"+10 mesh)
00-334C(640-660) ('-10+35 mesh)
00-334C(640-660) ('-35+100 mesh)
00-334C(640-660) ('-100 mesh)

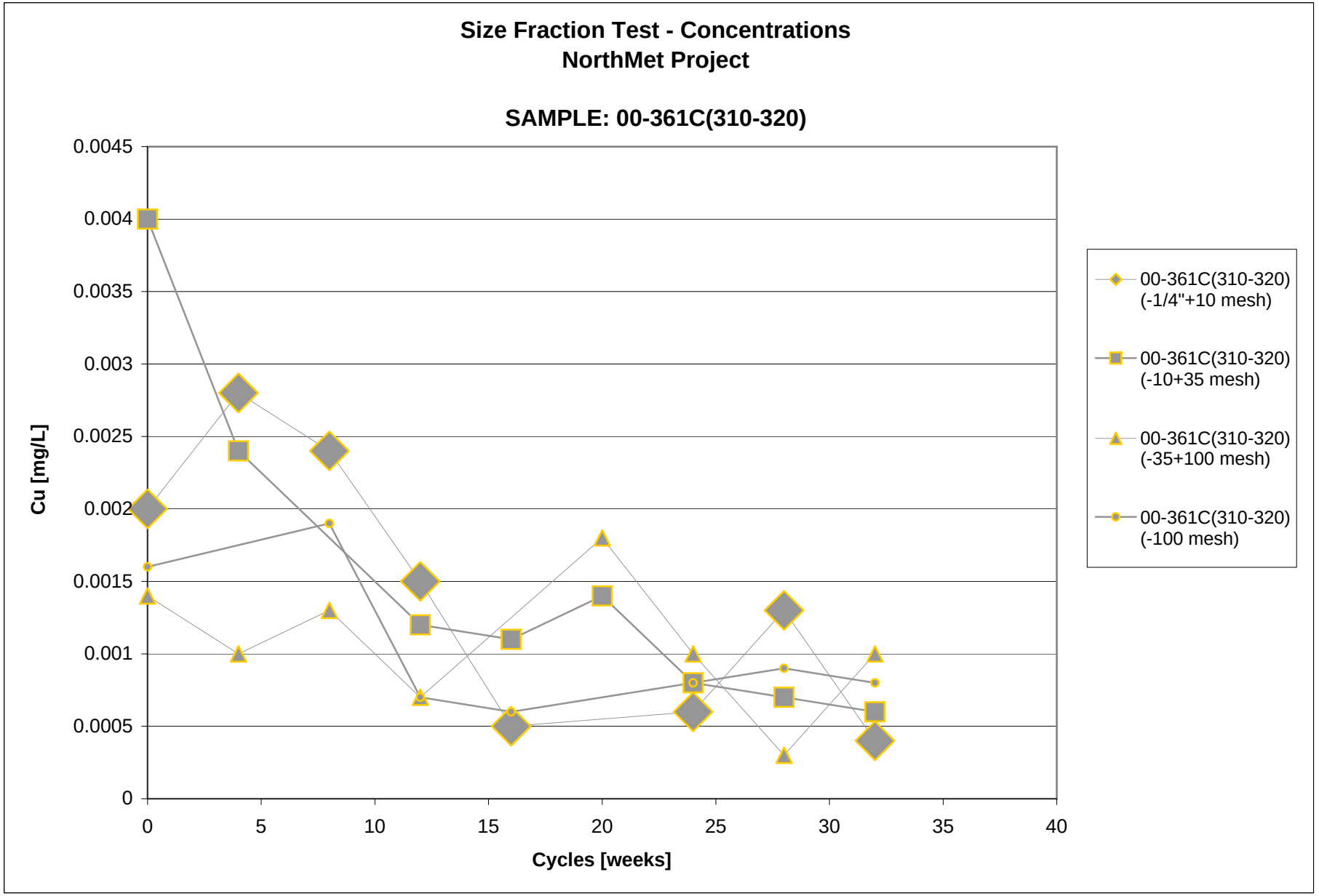

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-334C(640-660)
Ni [mg/L]
0.012
0.01
0.008
0.006
0.004
0.002
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-334C(640-660) ('-1/4"+10 mesh)
00-334C(640-660) ('-10+35 mesh)
00-334C(640-660) ('-35+100 mesh)
00-334C(640-660) ('-100 mesh)

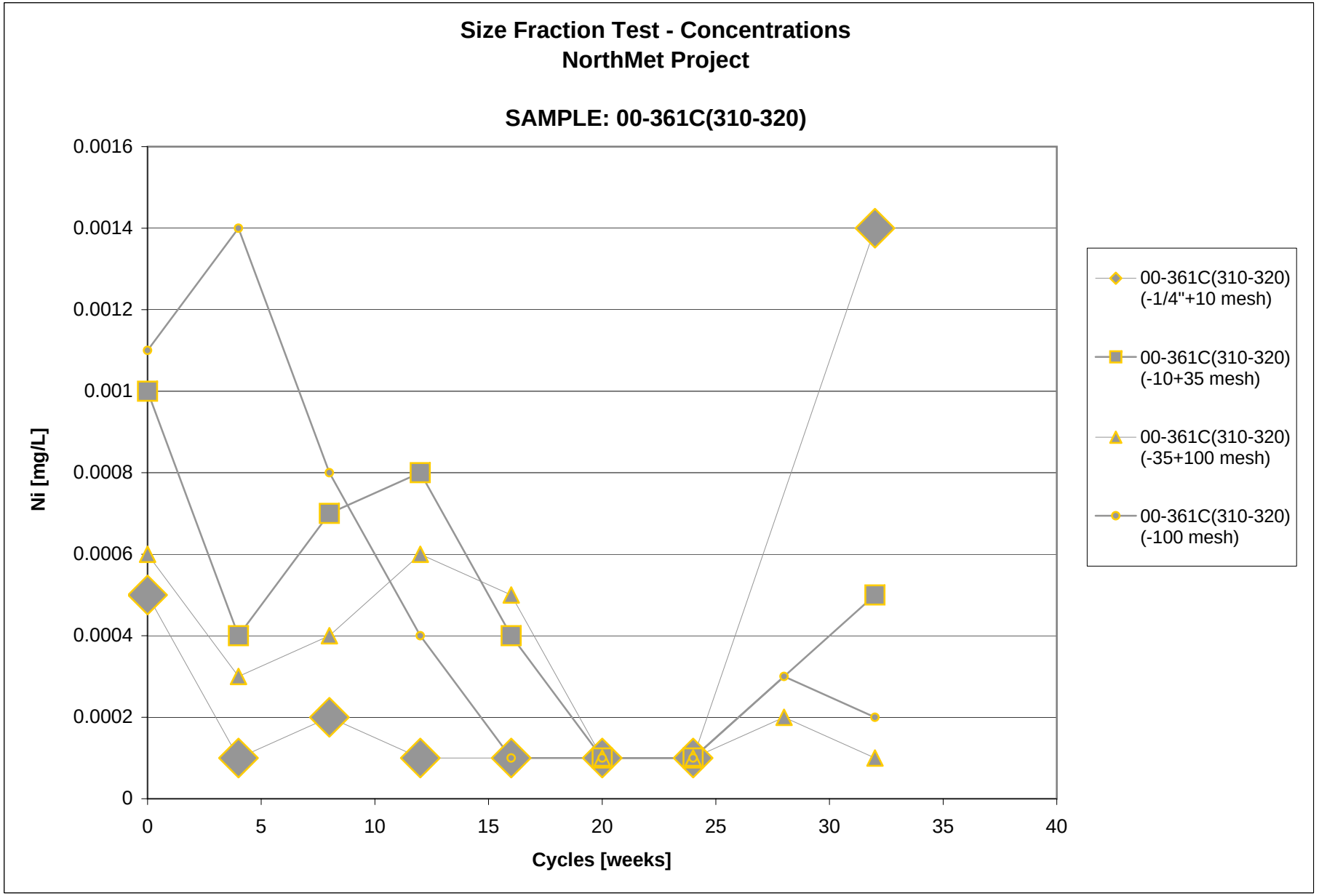

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-334C(640-660)
Co [mg/L]
0.00035
0.0003
0.00025
0.0002
0.00015
0.0001
0.00005
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-334C(640-660) ('-1/4"+10 mesh)
00-334C(640-660) ('-10+35 mesh)
00-334C(640-660) ('-35+100 mesh)
00-334C(640-660) ('-100 mesh)

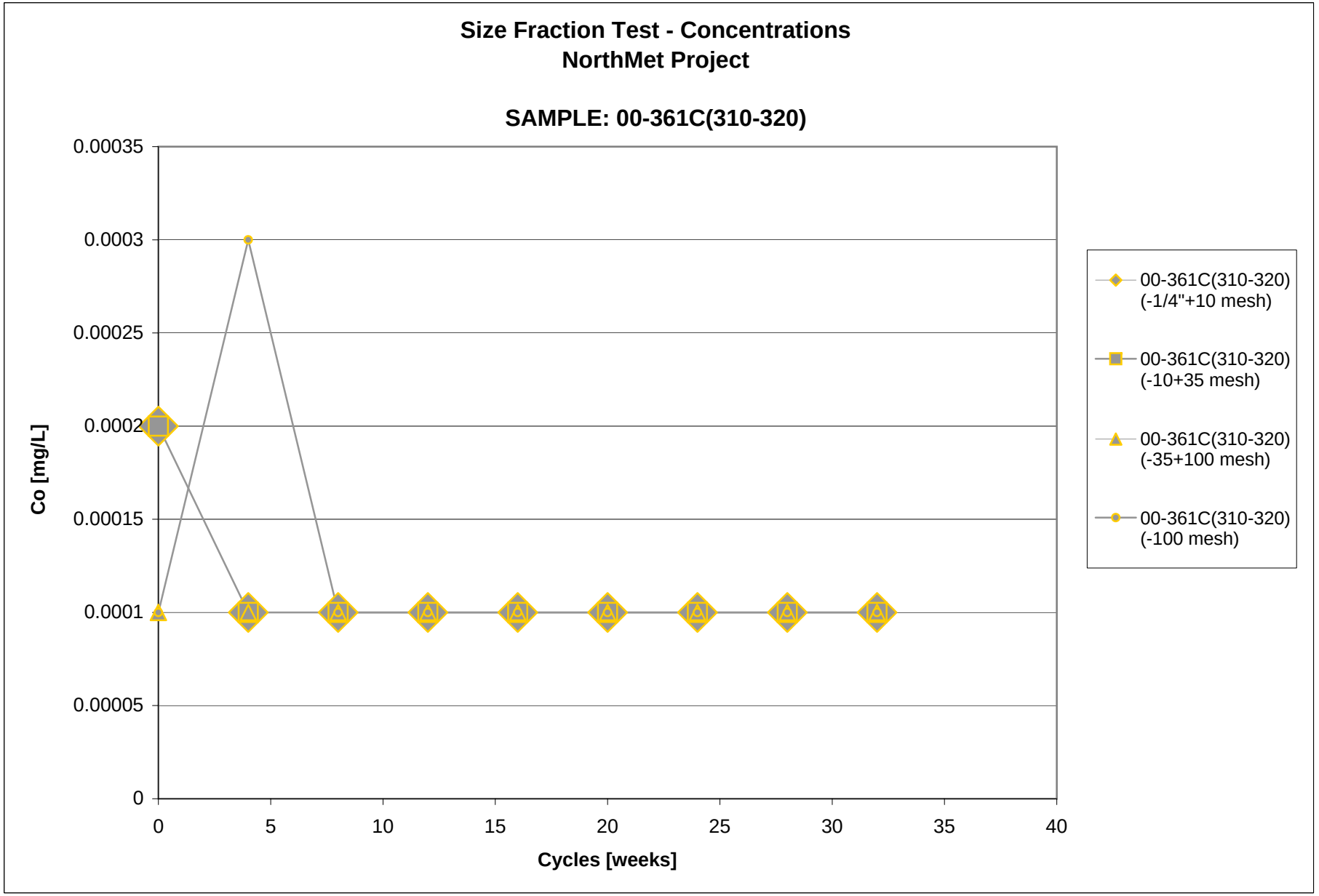

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-334C(640-660)
Zn [mg/L]
0.009
0.008
0.007
0.006
0.005
0.004
0.003
0.002
0.001
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-334C(640-660) ('-1/4"+10 mesh)
00-334C(640-660) ('-10+35 mesh)
00-334C(640-660) ('-35+100 mesh)
00-334C(640-660) ('-100 mesh)

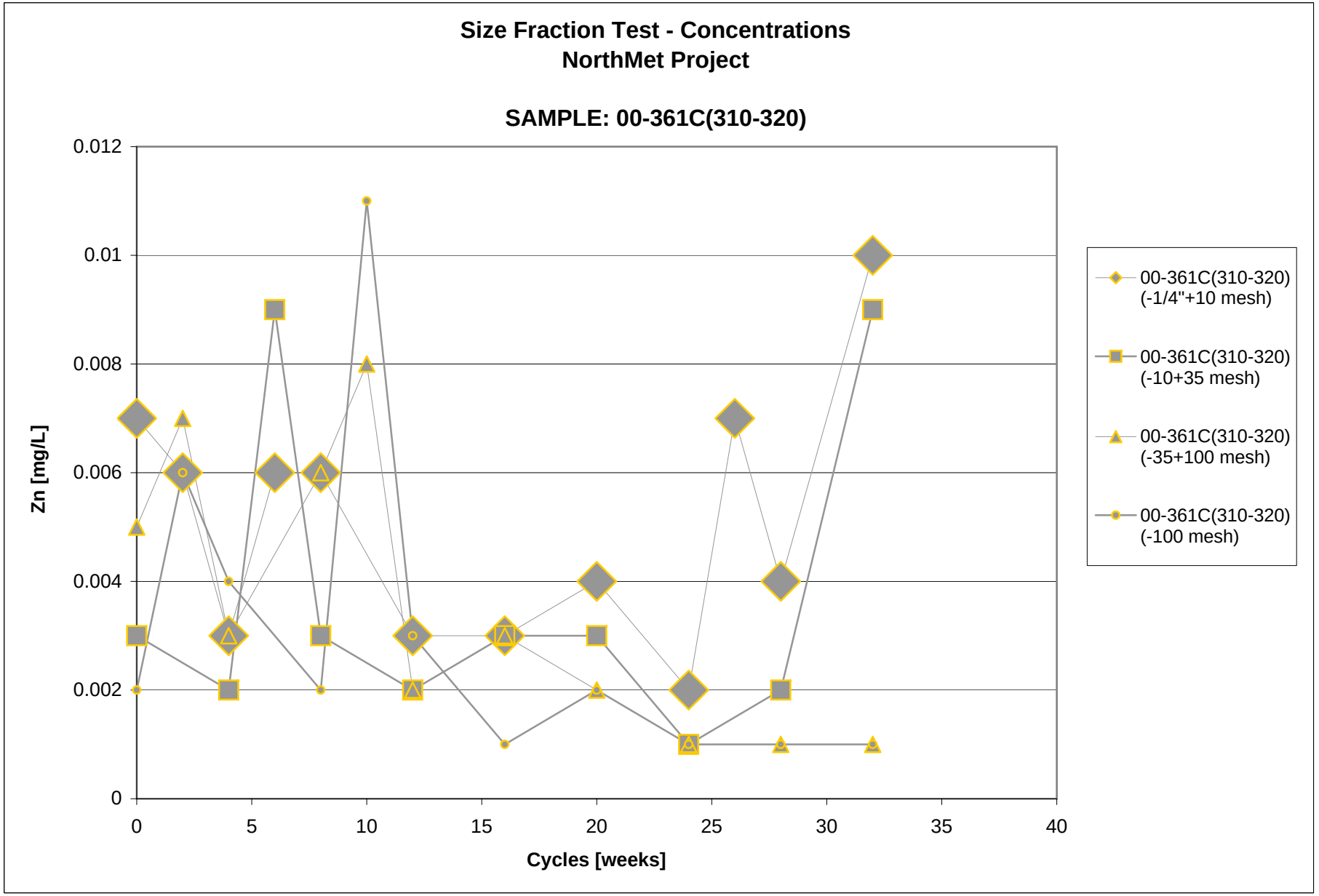

Size Fraction Test - Loadings
NorthMet Project
SAMPLE: 00-361C(310-320)
pH
10
9.5
9
8.5
8
7.5
7
6.5
6
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-361C(310-320) (-1/4"+10 mesh)
00-361C(310-320) (-10+35 mesh)
00-361C(310-320) (-35+100 mesh)
00-361C(310-320) (-100 mesh)

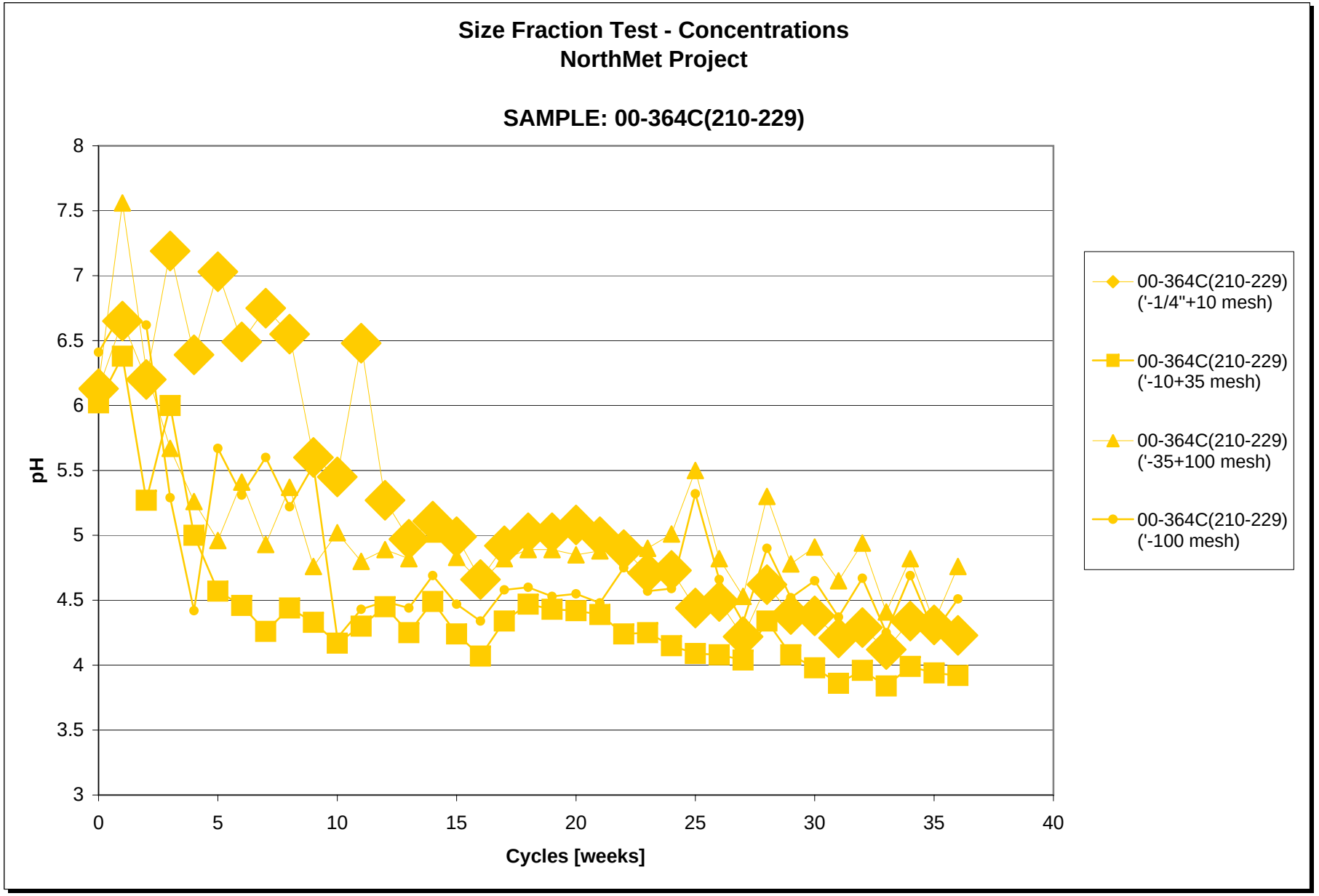

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-361C(310-320)
SO4 [mg/L]
100
10
1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-361C(310-320) (-1/4"+10 mesh)
00-361C(310-320) (-10+35 mesh)
00-361C(310-320) (-35+100 mesh)
00-361C(310-320) (-100 mesh)

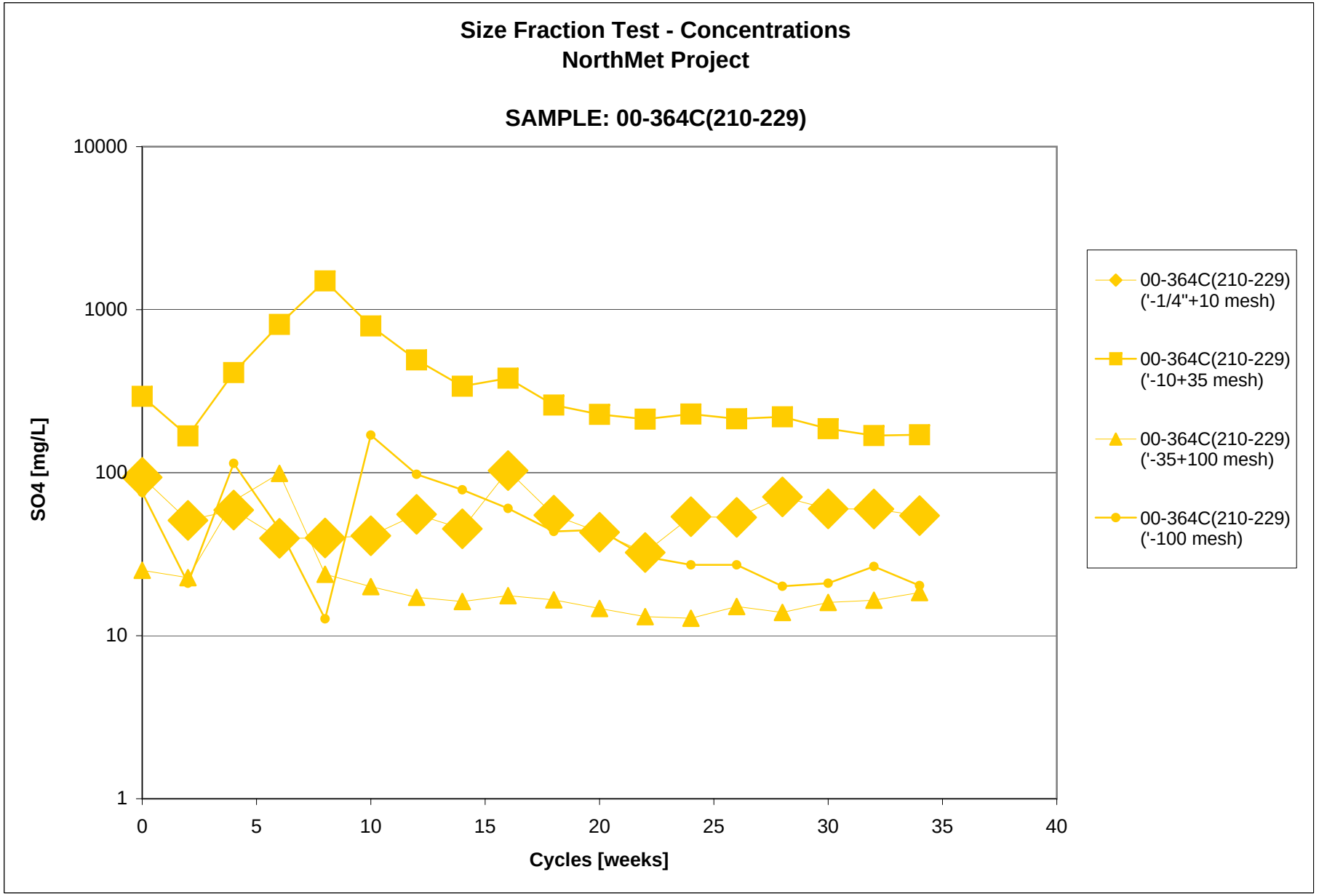

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-361C(310-320)
Cu [mg/L]
0.0045
0.004
0.0035
0.003
0.0025
0.002
0.0015
0.001
0.0005
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-361C(310-320) (-1/4"+10 mesh)
00-361C(310-320) (-10+35 mesh)
00-361C(310-320) (-35+100 mesh)
00-361C(310-320) (-100 mesh)

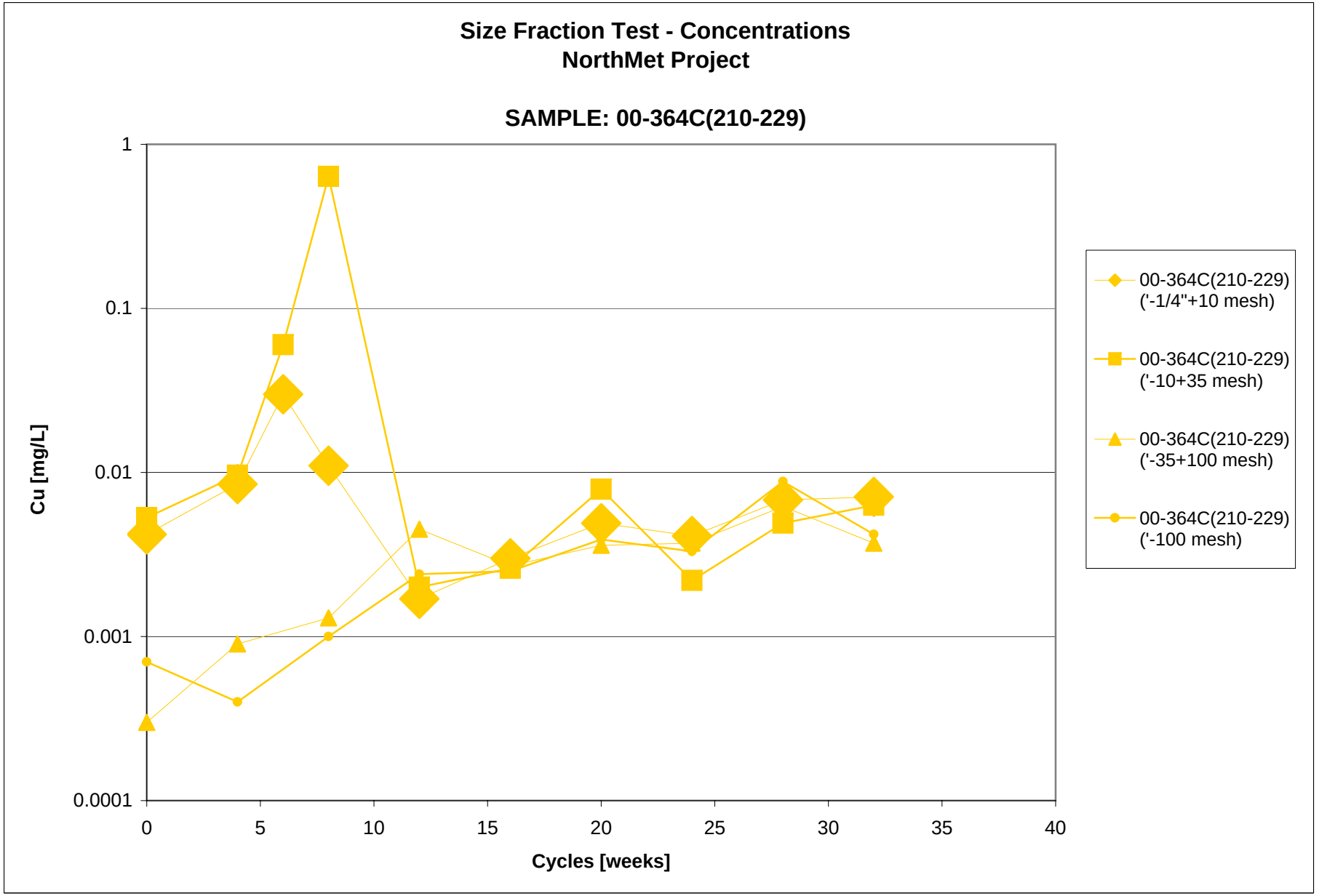

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-361C(310-320)
Ni [mg/L]
0.0016
0.0014
0.0012
0.001
0.0008
0.0006
0.0004
0.0002
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-361C(310-320) (-1/4"+10 mesh)
00-361C(310-320) (-10+35 mesh)
00-361C(310-320) (-35+100 mesh)
00-361C(310-320) (-100 mesh)

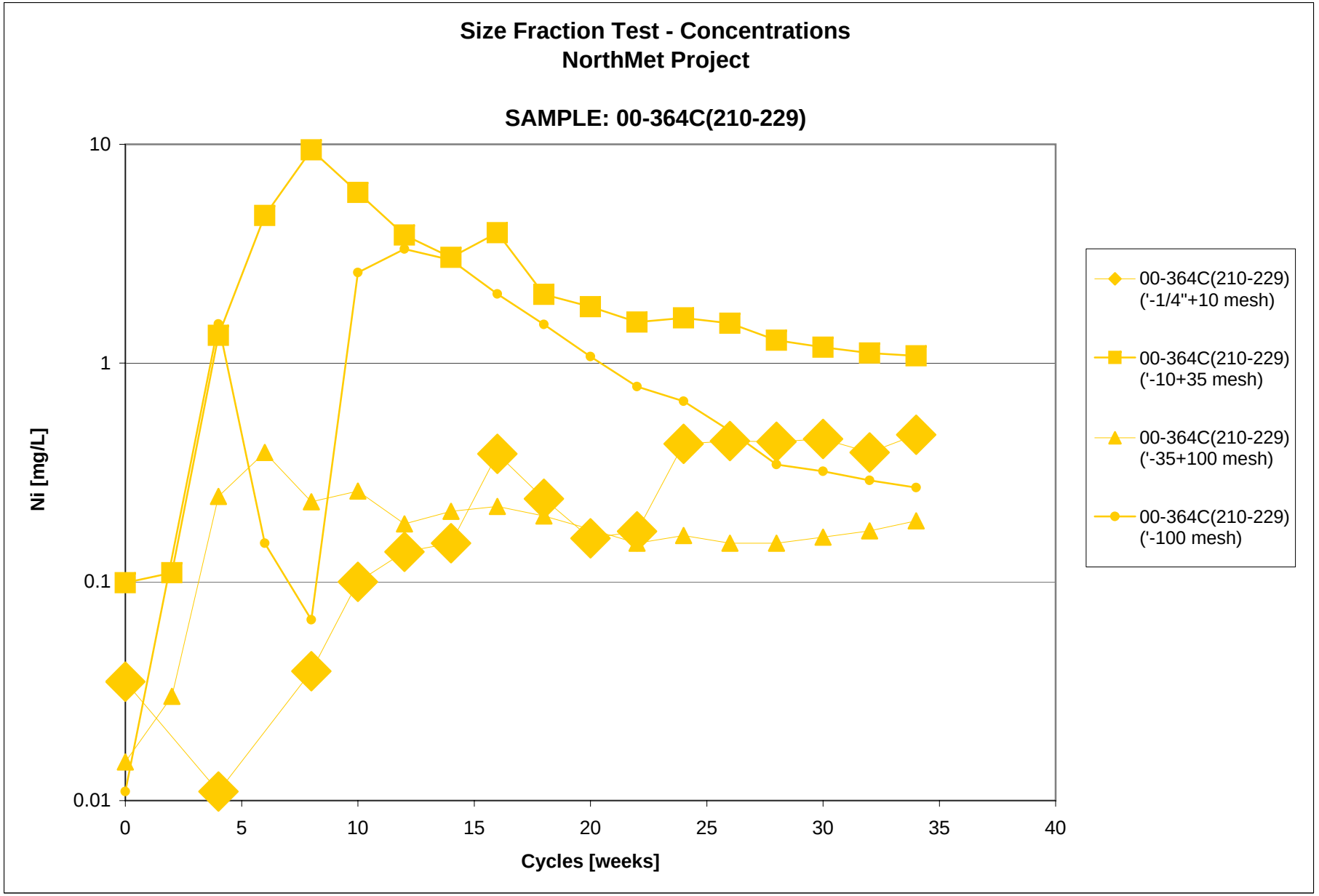

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-361C(310-320)
Co [mg/L]
0.00035
0.0003
0.00025
0.0002
0.00015
0.0001
0.00005
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-361C(310-320) (-1/4"+10 mesh)
00-361C(310-320) (-10+35 mesh)
00-361C(310-320) (-35+100 mesh)
00-361C(310-320) (-100 mesh)

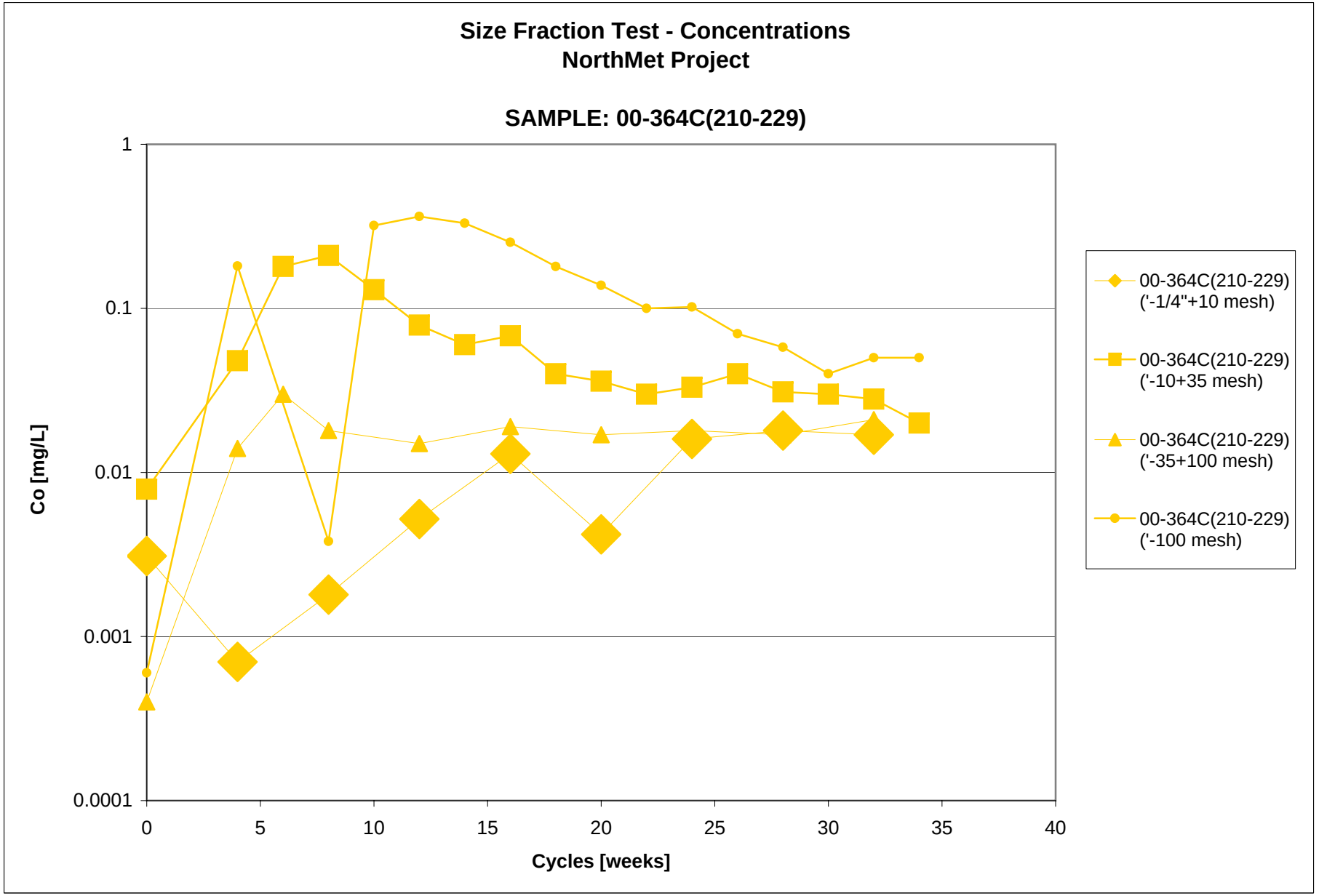

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-361C(310-320)
Zn [mg/L]
0.012
0.01
0.008
0.006
0.004
0.002
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-361C(310-320) (-1/4"+10 mesh)
00-361C(310-320) (-10+35 mesh)
00-361C(310-320) (-35+100 mesh)
00-361C(310-320) (-100 mesh)

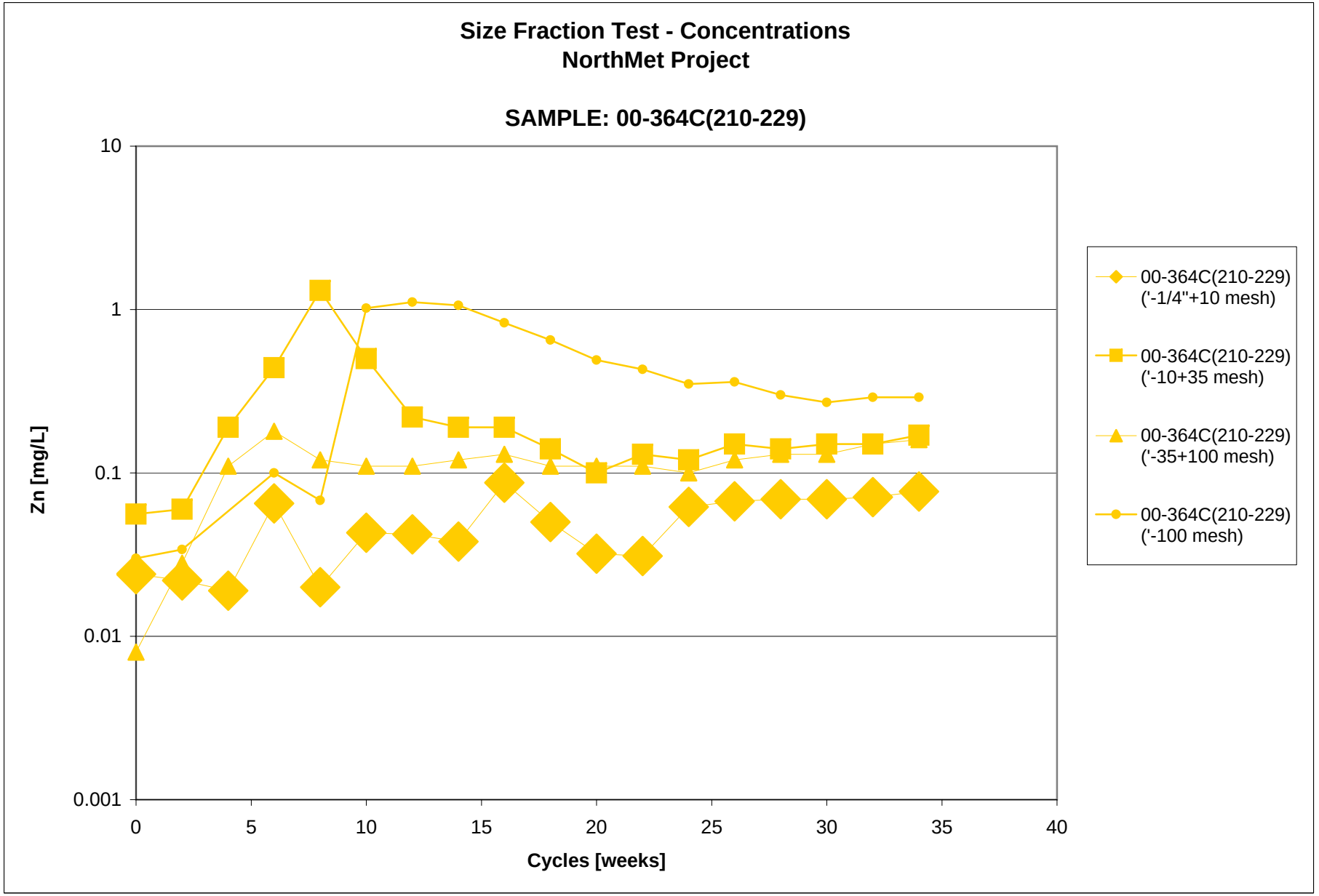

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-364C(210-229)
pH
8
7.5
7
6.5
6
5.5
5
4.5
4
3.5
3
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-364C(210-229) ('-1/4"+10 mesh)
00-364C(210-229) ('-10+35 mesh)
00-364C(210-229) ('-35+100 mesh)
00-364C(210-229) ('-100 mesh)


Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-364C(210-229)
SO4 [mg/L]
10000
1000
100
10
1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-364C(210-229) ('-1/4"+10 mesh)
00-364C(210-229) ('-10+35 mesh)
00-364C(210-229) ('-35+100 mesh)
00-364C(210-229) ('-100 mesh)

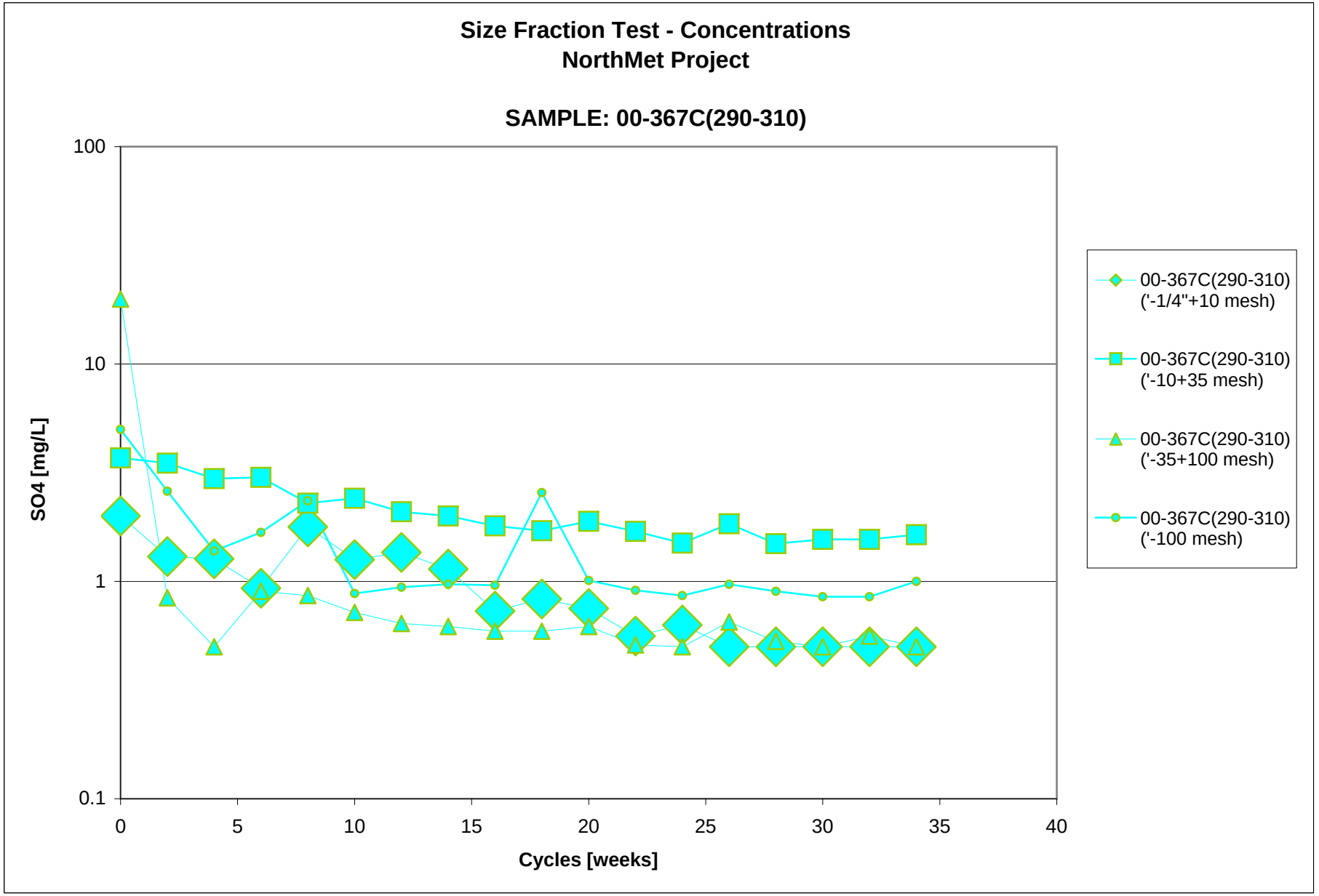

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-364C(210-229)
Cu [mg/L]
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-364C(210-229) ('-1/4"+10 mesh)
00-364C(210-229) ('-10+35 mesh)
00-364C(210-229) ('-35+100 mesh)
00-364C(210-229) ('-100 mesh)

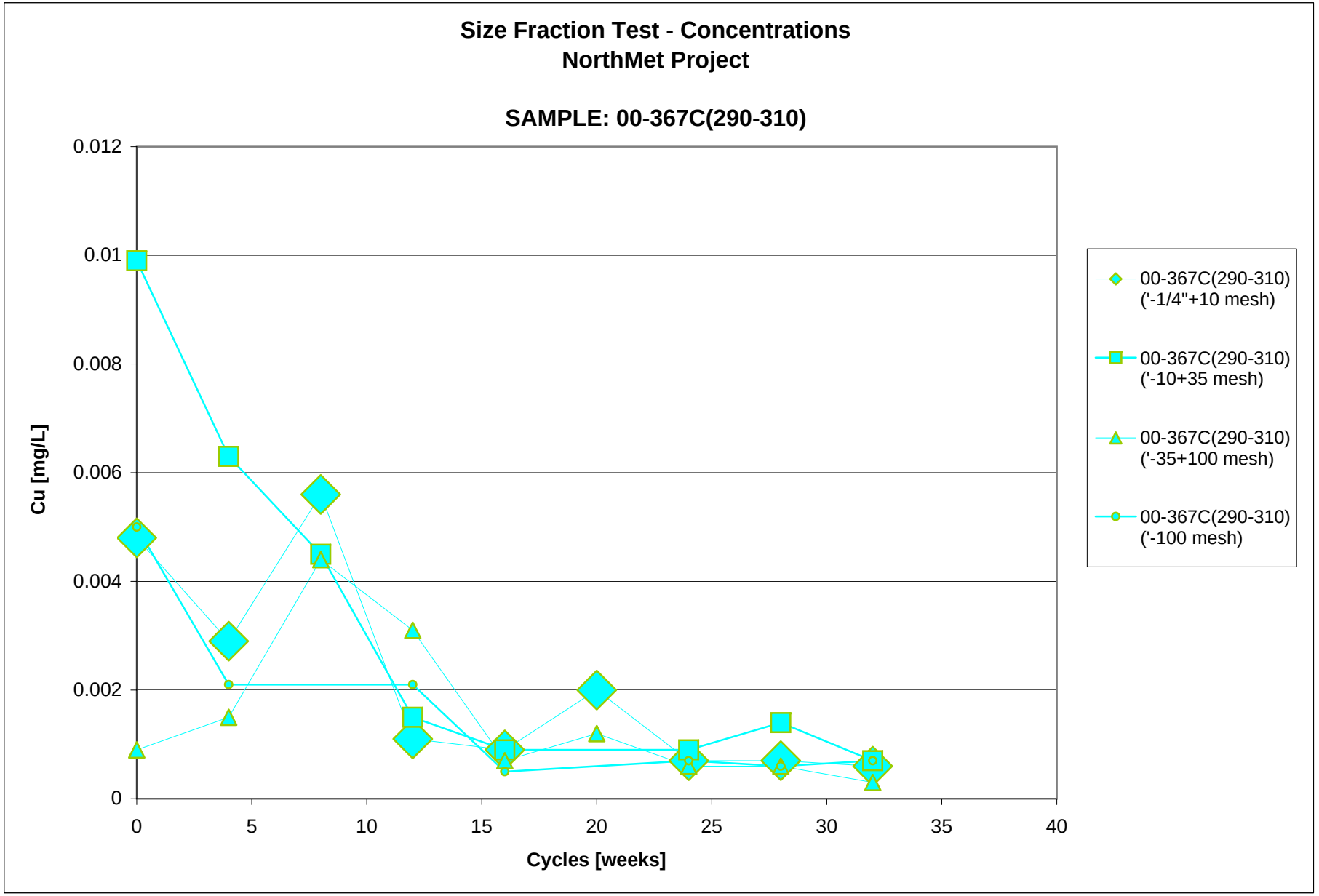

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-364C(210-229)
Ni [mg/L]
10
1
0.1
0.01
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-364C(210-229) ('-1/4"+10 mesh)
00-364C(210-229) ('-10+35 mesh)
00-364C(210-229) ('-35+100 mesh)
00-364C(210-229) ('-100 mesh)

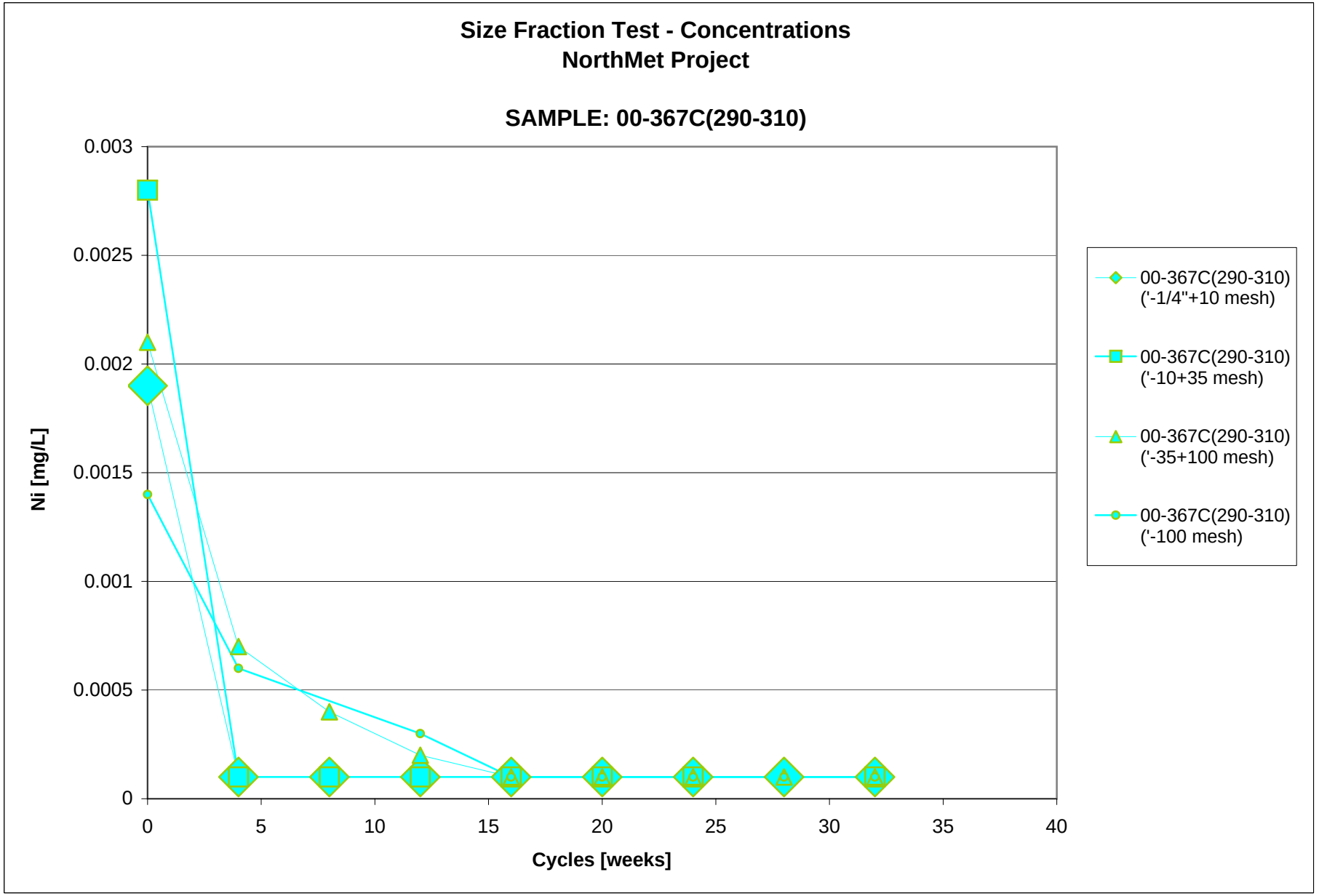

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-364C(210-229)
Co [mg/L]
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-364C(210-229) ('-1/4"+10 mesh)
00-364C(210-229) ('-10+35 mesh)
00-364C(210-229) ('-35+100 mesh)
00-364C(210-229) ('-100 mesh)

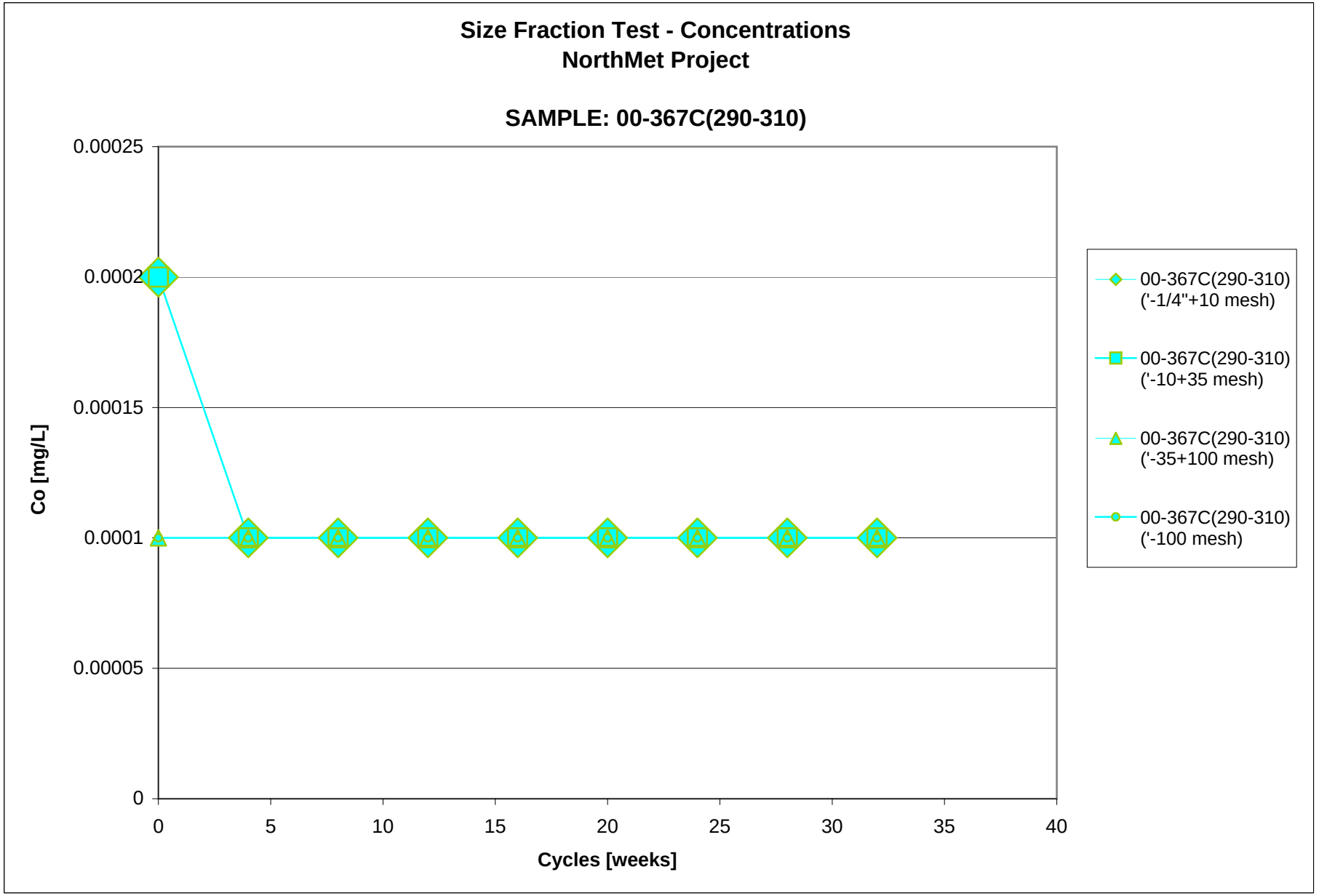

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-364C(210-229)
Zn [mg/L]
10
1
0.1
0.01
0.001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-364C(210-229) ('-1/4"+10 mesh)
00-364C(210-229) ('-10+35 mesh)
00-364C(210-229) ('-35+100 mesh)
00-364C(210-229) ('-100 mesh)

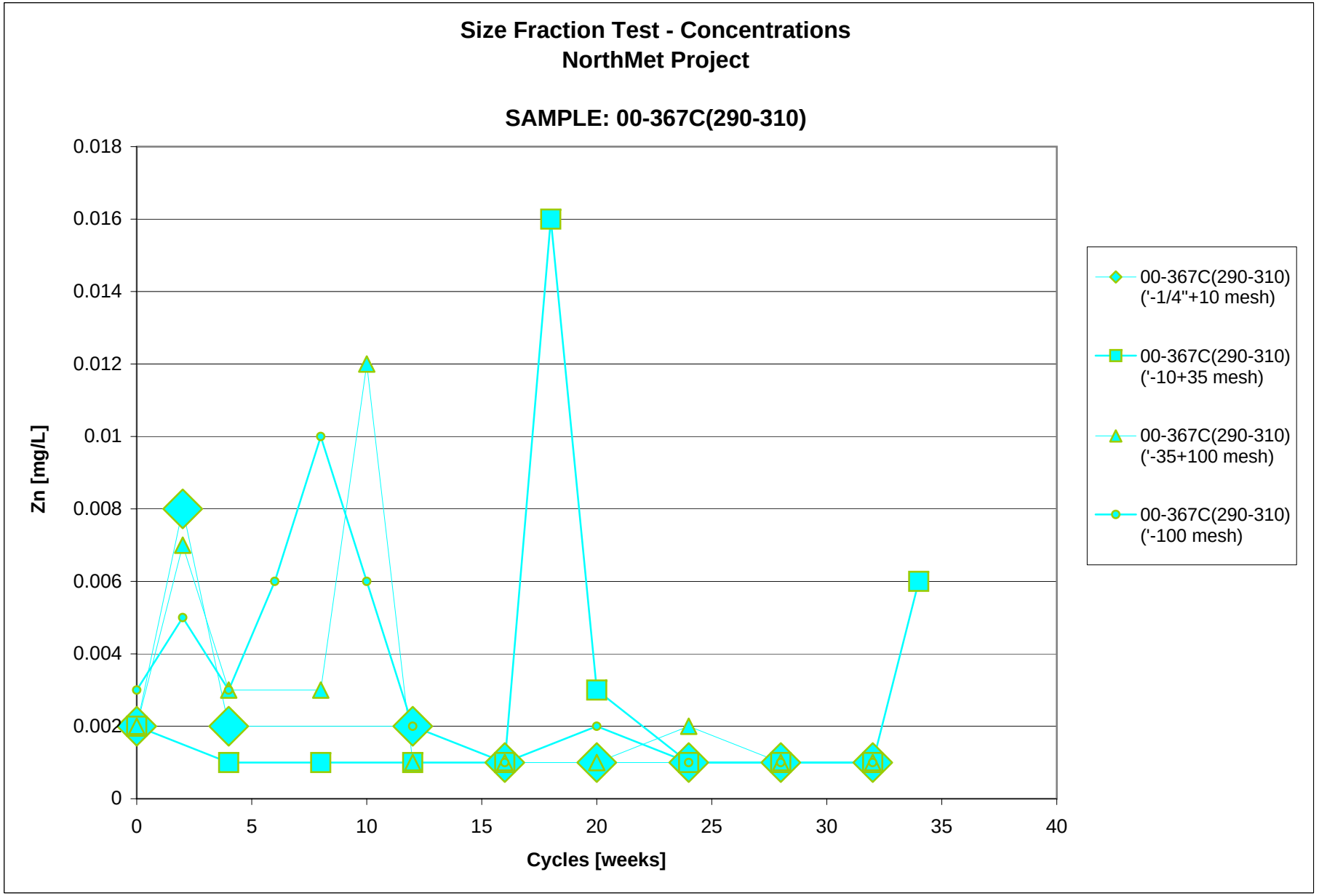
Size Fraction Test - Loadings
NorthMet Project
SAMPLE: 00-367C(290-310)
pH
11
10
9
8
7
6
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-367C(290-310) ('-1/4"+10 mesh)
00-367C(290-310) ('-10+35 mesh)
00-367C(290-310) ('-35+100 mesh)
00-367C(290-310) ('-100 mesh)


Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-367C(290-310)
SO4 [mg/L]
100
10
1
0.1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-367C(290-310) ('-1/4"+10 mesh)
00-367C(290-310) ('-10+35 mesh)
00-367C(290-310) ('-35+100 mesh)
00-367C(290-310) ('-100 mesh)

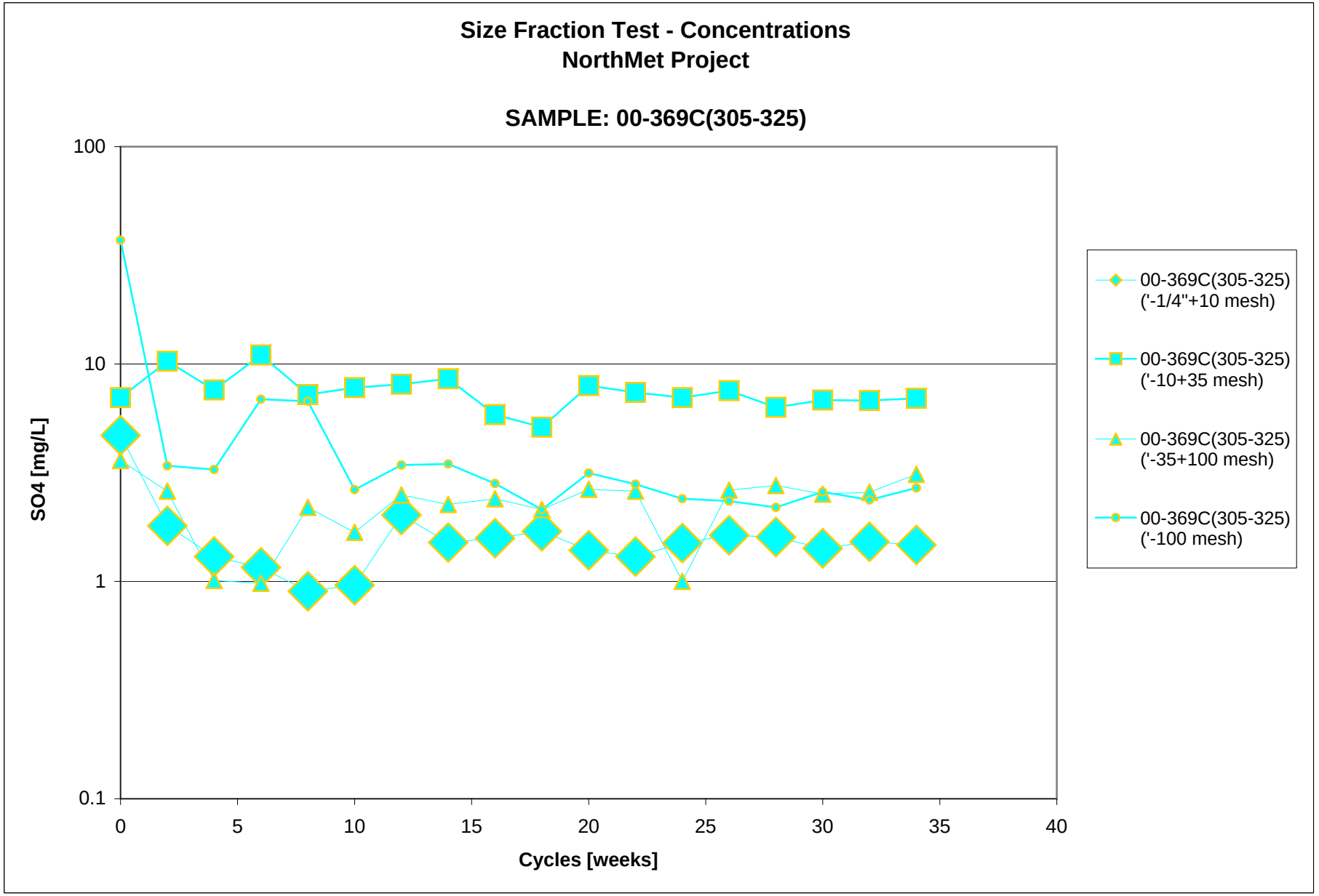

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-367C(290-310)
Cu [mg/L]
0.012
0.01
0.008
0.006
0.004
0.002
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-367C(290-310) ('-1/4"+10 mesh)
00-367C(290-310) ('-10+35 mesh)
00-367C(290-310) ('-35+100 mesh)
00-367C(290-310) ('-100 mesh)

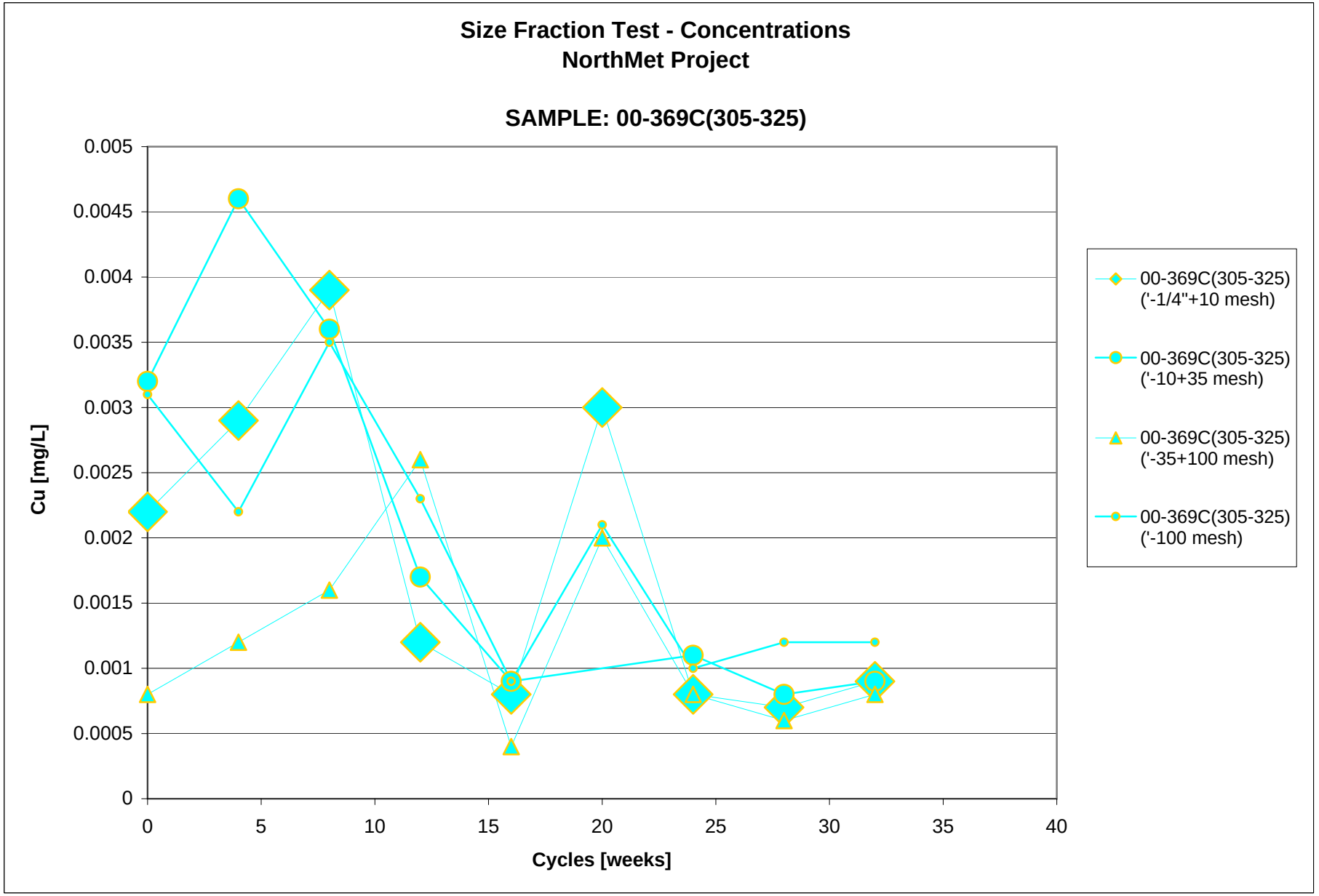

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-367C(290-310)
Ni [mg/L]
0.003
0.0025
0.002
0.0015
0.001
0.0005
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-367C(290-310) ('-1/4"+10 mesh)
00-367C(290-310) ('-10+35 mesh)
00-367C(290-310) ('-35+100 mesh)
00-367C(290-310) ('-100 mesh)

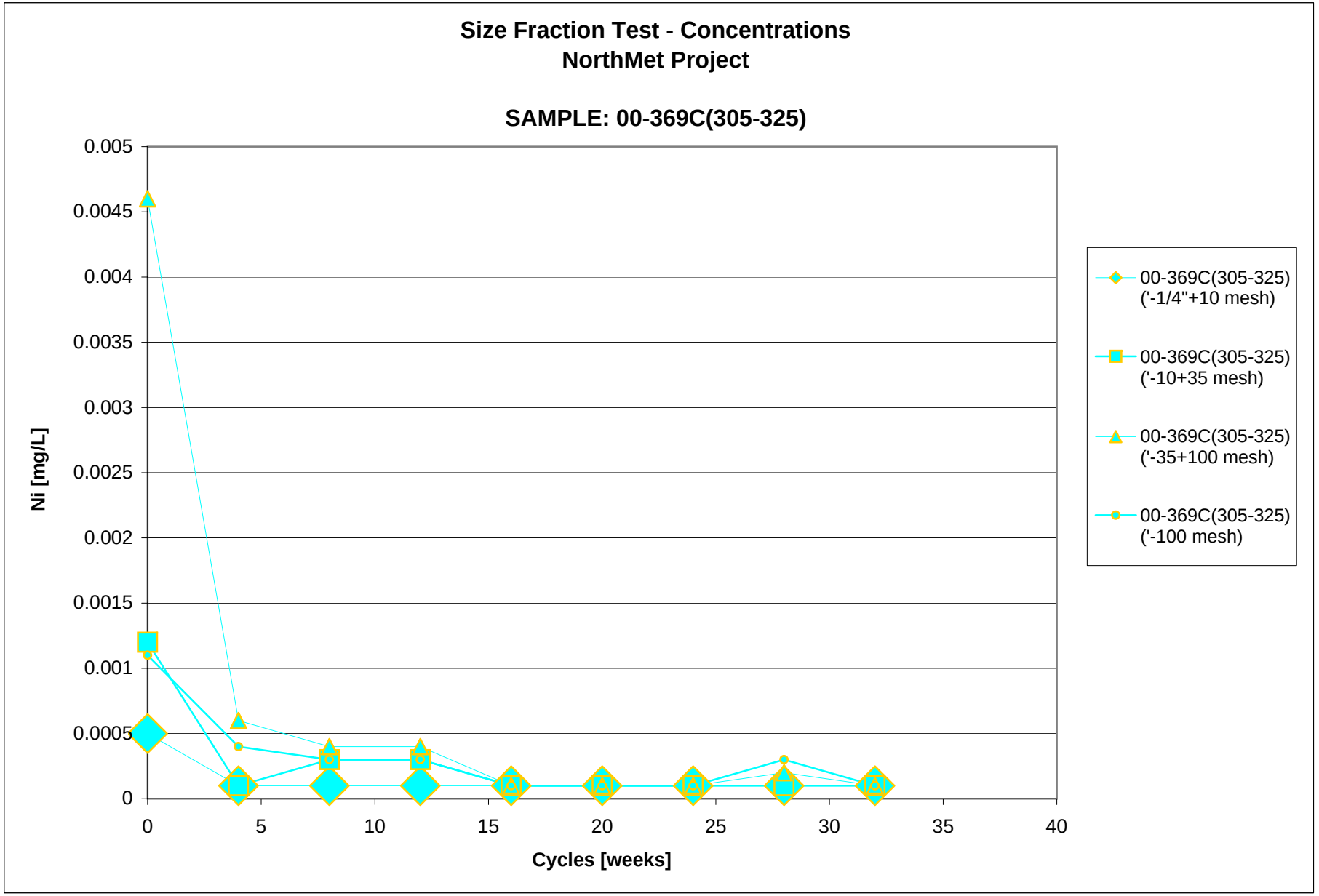

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-367C(290-310)
Co [mg/L]
0.00025
0.0002
0.00015
0.0001
0.00005
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-367C(290-310) ('-1/4"+10 mesh)
00-367C(290-310) ('-10+35 mesh)
00-367C(290-310) ('-35+100 mesh)
00-367C(290-310) ('-100 mesh)

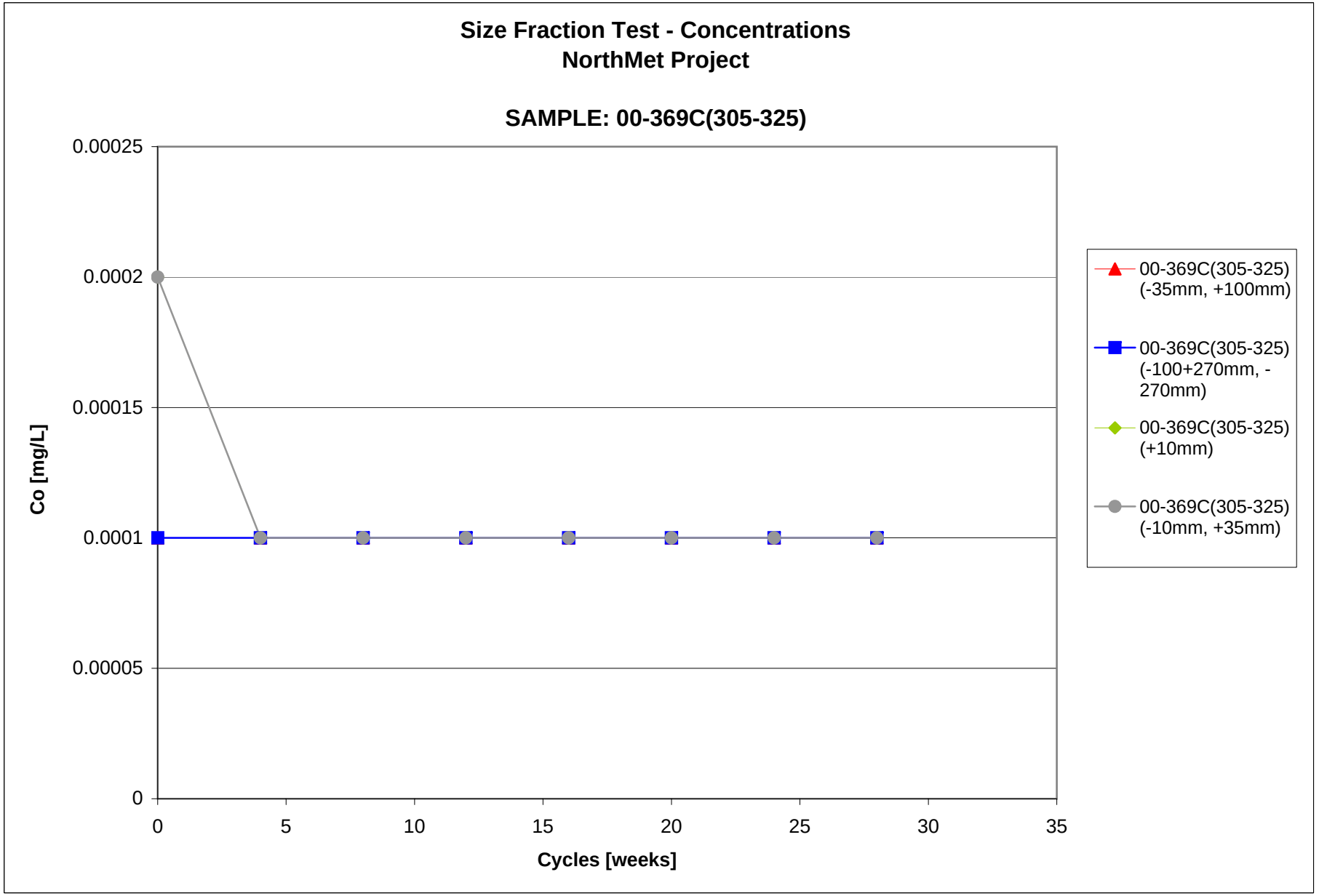

Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-367C(290-310)
Zn [mg/L]
0.018
0.016
0.014
0.012
0.01
0.008
0.006
0.004
0.002
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-367C(290-310) ('-1/4"+10 mesh)
00-367C(290-310) ('-10+35 mesh)
00-367C(290-310) ('-35+100 mesh)
00-367C(290-310) ('-100 mesh)

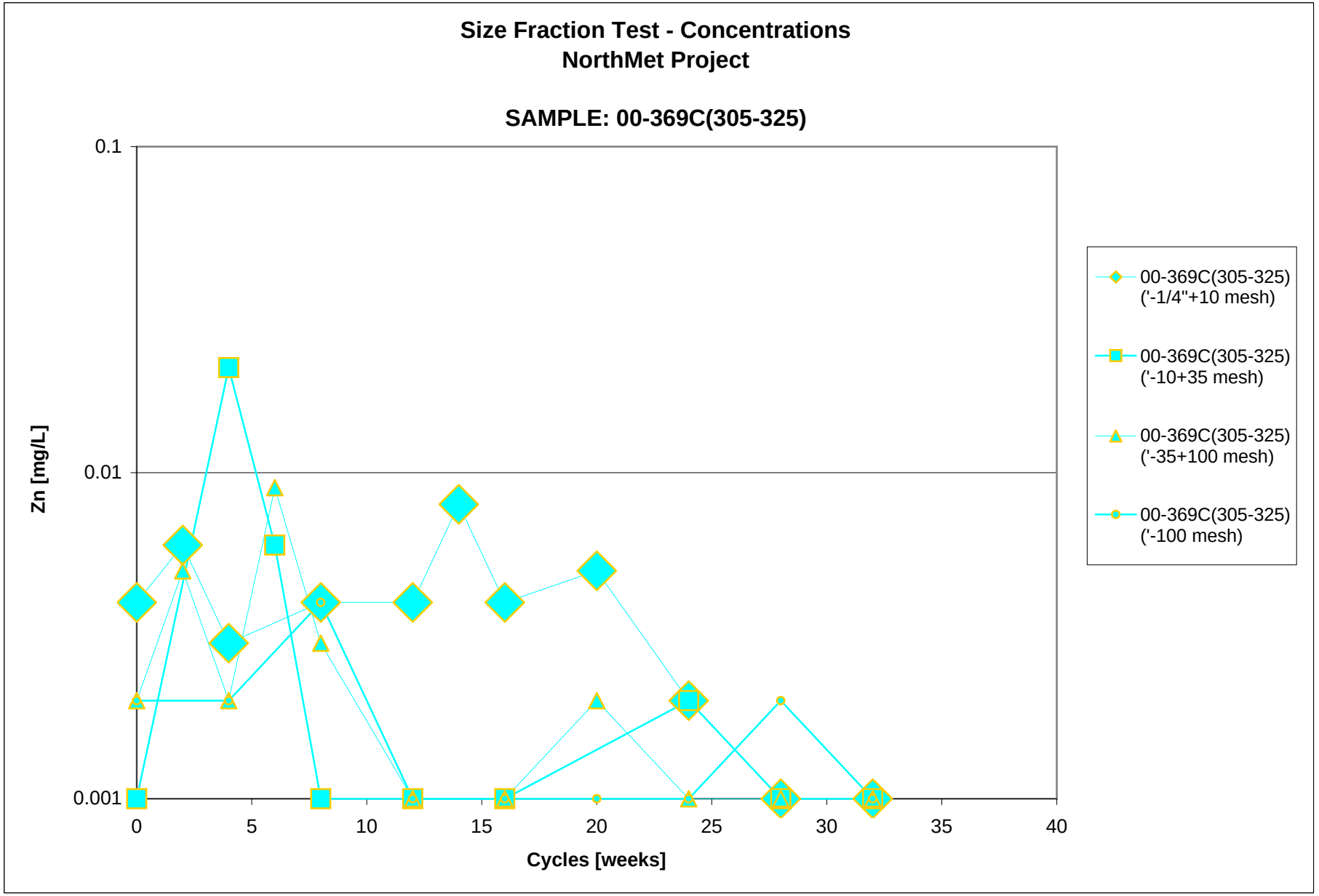
Size Fraction Test - Loadings
NorthMet Project
SAMPLE: 00-369C(305-325)
pH
10
9
8
7
6
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-369C(305-325) ('-1/4"+10 mesh)
00-369C(305-325) ('-10+35 mesh)
00-369C(305-325) ('-35+100 mesh)
00-369C(305-325) ('-100 mesh)


Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-369C(305-325)
SO4 [mg/L]
100
10
1
0.1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-369C(305-325) ('-1/4"+10 mesh)
00-369C(305-325) ('-10+35 mesh)
00-369C(305-325) ('-35+100 mesh)
00-369C(305-325) ('-100 mesh)


Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-369C(305-325)
Cu [mg/L]
0.005
0.0045
0.004
0.0035
0.003
0.0025
0.002
0.0015
0.001
0.0005
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-369C(305-325) ('-1/4"+10 mesh)
00-369C(305-325) ('-10+35 mesh)
00-369C(305-325) ('-35+100 mesh)
00-369C(305-325) ('-100 mesh)


Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-369C(305-325)
Ni [mg/L]
0.005
0.0045
0.004
0.0035
0.003
0.0025
0.002
0.0015
0.001
0.0005
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-369C(305-325) ('-1/4"+10 mesh)
00-369C(305-325) ('-10+35 mesh)
00-369C(305-325) ('-35+100 mesh)
00-369C(305-325) ('-100 mesh)


Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-369C(305-325)
Co [mg/L]
0.00025
0.0002
0.00015
0.0001
0.00005
0
0
5
10
15
20
25
30
35
Cycles [weeks]
00-369C(305-325) (-35mm, +100mm)
00-369C(305-325) (-100+270mm, -270mm)
00-369C(305-325) (+10mm)
00-369C(305-325) (-10mm, +35mm)


Size Fraction Test - Concentrations
NorthMet Project
SAMPLE: 00-369C(305-325)
0.1
0.01
0.001
Zn [mg/L]
0
5
10
15
20
25
30
35
40
Cycles [weeks]
00-369C(305-325) ('-1/4"+10 mesh)
00-369C(305-325) ('-10+35 mesh)
00-369C(305-325) ('-35+100 mesh)
00-369C(305-325) ('-100 mesh)

Attachment 2-11

SRK Consulting. RS33A – Reactive Residues Progress Report – NorthMet Project. May 2006.

Appendix D – Delivery Package 1A-2

RS33A – Reactive Residues Progress Report, NorthMet Project, Minnesota

Prepared for:

Polymet Mining Inc.
#2350 – 1177 West Hastings Street
Vancouver, BC
V6E 2K3

Prepared by:

SRK Consulting
Engineers and Scientists

Project Reference Number
SRK 1UP005.001

May 2006

Appendix D – Delivery Package 1A-2

RS33A – Reactive Residues. Progress Report, NorthMet Project, Minnesota

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SRK Project Number 1UP005.001

May 2006

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Appendix A: Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan
Appendix B: Results of Mineralogy, Bulk Chemical and Leach Test Characterization
Appendix C: Dissolution Tests Charts
C.1 Sequential Shake Flasks
C.2 Humidity Cells
C.3 DNR Reactors

1 Introduction

PolyMet Mining Inc. (PolyMet) is proposing to develop the NorthMet Project (Dunka Road Project of US Steel) near Babbitt, Minnesota. As a part of the Minnesota Department of Natural Resources (MDNR) "Permit to Mine" process a complete "mine waste characterization" will be required (Minnesota Rules Chapter 6132.1000).

The processing and recovery of NorthMet Project ore to recover commodity metals will involve conventional flotation to produce a sulfide concentrate followed by hydrometallurgical treatment of the concentrate.

Processing will result in the generation of the following waste products:

- Flotation tailings (low sulfide).
- Leach residue (silicates).
- Gypsum residue (calcium sulfate).
- Raffinate neutralization residue (calcium sulfate).
- Fe/Al residue (oxide).
- Mg residue (oxide).

The characterization plan for these products was developed in consultation with the Minnesota Department of Natural Resources (SRK 2005) (Appendix A). The plan covers characterization of both tailings from the flotation process and the hydrometallurgical residues from recovery of commodities. Characterization of the hydrometallurgical residues was started in February 2006 following generation of the residues between August 29 and October 11, 2005 and agreement on the characterization methodologies with MDNR. This report is the first of two reports describing characterization results. It covers results of mineralogical characterization, analysis of bulk solids, results of regulatory leach procedures (EPA 1311 and 1312) and initial results from kinetic dissolution tests.

2 Methods

2.1 Background

The sampling and testing plan was developed in consultation with MDNR (Appendix A). The following sections summarize the methods used for characterization of the tailings and hydrometallurgical residues. This report describes the hydrometallurgical methods.

2.2 Sample Collection and Compositing

Sulfide concentrates were prepared by a pilot flotation plant in which the copper sulfate was evaluated as a reagent to improve flotation of sulfide minerals. Two concentrates were prepared with and without the use of copper sulfate, and both residues were leached in the hydrometallurgical process. PolyMet has since decided that copper sulfate will be used. Therefore, it was agreed (in consultation with MDNR, SRK 2006) that characterization testwork would be performed on the residues generated by leaching of the concentrate produced with copper sulfate. The exception is the leach residue for which both residue samples are being tested.

Residues generated by SGS Lakefield were collected under supervision of Barr Engineering and shipped to Canadian Environmental and Metallurgical Inc (CEMI) in Burnaby, British Columbia, Canada for testing. SRK Consulting is responsible for supervision of the testwork described in this report.

Testing is proceeding on individual samples of each type of residue and combined residues. Residue combinations are being tested with and without the gypsum residue to allow for the option of recovery of a separate gypsum product. The proportions by wet weight of each residue in the combined samples are provided in Table 1. The combined residue containing all residues was received directly from the pilot plant (SGS Lakefield). The combined residue without gypsum was prepared from the component residues using the same weight proportions less the gypsum amount.

Table 1: Proportions (by Wet Weight) for Combined Residues

	Actual Pilot Plant Observation		Projected for Commercial Scale
Residue	**All**	**Without Gypsum**	**All**
Leach Residue	27%	45%	41%
Gypsum Residue	40%	0%	27%
Raffinate Residue	18%	31%	13%
Fe/Al Residue	6%	10%	12%
Mg Residue	8%	14%	7%

Subsequent optimization and modeling (MetSim Version S) of the hydrometallurgical process by Bateman indicates that the proportion of the residues may be somewhat different under full scale production (see Table 1). The implications of this difference are inconsequential and are discussed in Section 4.

2.3 Mineralogical Characterization

Mineralogical characterization of the residues included optical mineralogy on the leach residue and Quantitative X-Ray Diffraction (XRD) on all residues. Optical mineralogy is in progress and results should be available in mid-June. The XRD method detects crystalline phases and calculates the percentage of these phases assuming that the sample is entirely crystalline. Amorphous phases are neither detected nor considered in the proportions of phases. Consideration of the bulk element content provides indicators about the presence and composition of the amorphous components.

Results of the mineralogical characterization are provided in Appendix A.

2.4 Chemical Characterization

2.4.1 Bulk Chemistry

Bulk chemical analysis included acid-base accounting parameters (total and sulfate sulfur, neutralization potential, total inorganic carbon and pH) and metals. In samples containing sulfide minerals, the difference between sulfate sulfur and total sulfur is normally attributed to sulfur occurring as sulfide minerals. Since the concentrates are completely oxidized in the hydrometallurgical process and the resulting residues produced by precipitation processes (i.e. all residues except the leach residue) will not contain any sulfide minerals, the difference between total and sulfate sulfur is due to analytical uncertainties rather than sulfide minerals.

The feasibility of determining the chemical content of individual mineral grains will be evaluated when grain mounts are available in June.

Results of the bulk chemical characterization are provided in Appendix B.

2.4.2 Leach Procedures

Two regulatory leach procedures were used:

- **EPA 1311 or Toxicity Characteristic Leaching Procedure (TCLP)** uses an acetic acid leachate with a pH of about 5. The solid to leachate ratio is 1:20. The procedure evaluates the leachability of wastes in contact with an organic acid. The procedure provides the basis for regulatory classification of the waste residues.

- **EPA 1312 or Synthetic Precipitation Leach Procedure (SPLP)** uses a weak sulfuric and nitric acid mixture buffered at about pH 4.2. The solid to leachate ratio is 1:20. This procedure is intended to simulate the effect of acidic rainfall produced by industrial pollution. SPLP is more appropriate than TCLP for evaluation of leaching due to contact with meteoric waters because it uses a mineral acid rather than an inorganic acid.

A third leach procedure was used as the first step of the sequential shake flask leach procedure described in Section 2.3.3. This procedure, developed by Price (1997) for the British Columbia

Ministry of Energy Mines and Petroleum Resources, uses a lower leach ratio (1:3) to improve detection of low levels of metals. The lixiviant is deionized water which typically has a pH between 5 and 6. The leachate pH is not fixed as in the TCLP and SPLP methods.

It was originally planned to also use the Meteoric Water Mobility Procedure (NDEP 1996); however due to sample limitation, the test was not performed. The solid to leachate ratio for this test is 1:1 which bracketed by the Price (1997) method (1:3) and the ASTM humidity cell (1:0.5).

Results of the leachate tests are provided in Appendix A.

2.4.3 Dissolution Tests

Sequential Shake Flask

All residues are being tested using a sequential leach procedure consisting of weekly repetition of the leach procedure developed by Price (1997) in which the solid to liquid ratio is 1:3. The procedure involves weekly leaching of roughly 300 g of solids in a plastic bottle. The leaching step consists of addition of deionized water and agitation for 24 hours. The leachate is then decanted for analysis.

This procedure was requested by the DNR to provide a more aggressive evaluation of residue dissolution than occurs in humidity cells.

Leachates are analyzed using the following schedule:

- Weekly – pH, oxidation-reduction potential, conductivity.
- Bi-weekly – Acidity, alkalinity, inorganic C, hardness, anions (F, Cl, SO_4).
- Four weekly (weeks 0, 4, 8, 12, etc) – Low level element scan using ICP-MS.
- Four weekly (weeks 2, 6, 10, 14, etc) – General element scan using ICP-OES.

Humidity Cell

Residues are being tested in ASTM-style humidity cells in the tailings configuration. Details of the procedure are provided in Appendix A. The solid to liquid ratio is 1:0.5 (1 kg of solids leached with 500 mL of deionized water). Leachates are being analysed using the same schedule as the sequential shake flasks.

DNR Reactor

Residues are also being tested in small (75 g) DNR Reactors for which the solid to liquid ratio is 1:2.7 (75 g of solids leached with 200 mL of deionized water). Details are provided in Appendix A. Leachates are being analysed using the same schedule as the sequential shake flasks.

Testwork Progress

The dissolution tests were started in early February 2006. Up to 11 weeks of data have been provided by the laboratory depending on the parameter and the tests are ongoing.

Graphs illustrating concentrations obtained to date from dissolution tests are provided in Appendix C. Data listings and supporting quality assurance data can be provided if requested.

Since different reporting limits are used for different cycles and different parameters, the following rules were used to plot data and allow trends to be apparent:

- If the parameter was not determined, the result is not plotted. This is the case for alkalinity and acidity which are only determined if the pH is above or below (respectively) specified values.
- If the result is undergoing quality control re-check, the result is not plotted (e.g. sulfate for leach residues and gypsum residues for shake flask tests).
- If the parameter is determined by the same method for each analysis, value below the reporting limit values are plotted as 50% of the reporting limit.
- Since the reporting limits for ICP-MS are below the reporting limit for ICP-ES:
    - If the result was determined by ICP-MS and was below the reporting limit, the value on the graph is 50% of the reporting limit. If the value is at or above the reporting limit, the value is plotted.
    - If the result was determined by ICP-ES and was determined to be below the reporting limit, no value is plotted.
    - If the result was determined by ICP-ES and was determined to be above the reporting limit, the value is plotted.
    - These rules can result in four cycles between plotted results if the parameter is not detected by ICP-ES (e.g. molybdenum in shake flask leachates).

Occasionally, “sawtooth” trends are apparent in which values alternate between high and low for the ICP-ES and ICP-MS analyses. This results from analytical “noise” around the ICP-ES reporting limit when reported values are slightly above the reporting limit. Aluminum is a particular example that commonly shows reported values above the ICP-ES reporting limit of 0.05 mg/L.

Many graphs are plotted on logarithmic axes to allow data spanning a wide range of concentrations to be compared.

More results are available for immediately determined parameters (for example, pH, conductivity) than for metals.

3 Results and Discussion

3.1 Introduction

Results are described below for each residue in the sequence they are produced in the hydrometallurgical process.

3.2 Leach Residues

The leach residues remain from the concentrate following pressure leaching. It is expected that this product will consist of both secondary minerals and resistant primary minerals (silicates).

Leach residues from processing of concentrates produced by sulfide flotation with and without copper sulfate are being tested. The XRD results implied differences in the residual silicate component of the residues. The concentrate produced without copper sulfate had 26% residual plagioclase compared to 6% for the concentrate produced with copper sulfate. The oxidation products produced by the process are shown as mainly natrojarosite, which was confirmed by the sodium content of the samples.

The two leach residue produced from concentrate without use of copper sulfate showed lower pH than the concentrate produced using copper sulfate and the various leach procedures consistently showed pHs below 3.4. The pH decreased as the water to solid ratio in the tests decreased. Lowest pHs were apparent for the ASTM humidity cell (1.9 and 2.3, respectively). The TCLP extraction showed the pH of the lixiviant as required by the procedure.

The low pHs resulted in high metal solubility. The highest metal concentrations were shown for the humidity cells.

Lime demand to neutralize excess acidity was low as shown by neutralization potentials of -17 and -4 kg $CaCO_3$/t. Acidity leached during the first step of the dissolution procedure was comparable (6 kg $CaCO_3$/t) and subsequent analyses of leachates indicated that acidity is highly soluble and decreases very rapidly along with overall ion content of the solutions (conductivity). Leachate pH also increased. The acidity is probably from rinsing of residual process solutions rather than dissolution of secondary minerals though dissolution of these can be expected to result in long term depression of pH.

3.3 Gypsum Residue

Gypsum residue is the first hydromet precipitation product. It is produced by limestone addition to the leach solution following recovery of platinum group metals and prior to copper recovery.

The elevated calcium and sulfate content of the gypsum residue sample confirmed that it was dominantly hydrated calcium sulfate. XRD showed that it was 99.8% gypsum.

This residue was acidic (pH 2.7 in the humidity cell) and leached metals at comparable levels to the leach residue produced from concentrate generated with copper sulfate. Leachate chemistry was consistent with the dissolution of gypsum. Leachate pH increased then stabilized in dissolution tests, and electrical conductivity remained stable and relatively low due to the solubility limitation of gypsum.

3.4 Raffinate Neutralization Residue

This residue is formed by an intermediate neutralization step between copper removal and precipitation of iron and aluminum.

Raffinate neutralization residue was identified as nearly entirely calcium sulfate (gypsum with minor bassanite) by XRD which was confirmed by the dominance of calcium and sulfate in the sample. This residue had low pH (3.6 in humidity cell), though leachate chemistry showed relatively low soluble metal concentrations compared to other residues.

General leachate chemistry in all three types of dissolution tests stabilized within a few weeks.

3.5 Iron and Aluminum Residue

The iron and aluminum residue is formed by two limestone addition steps prior to cobalt, nickel and zinc hydroxide recovery.

Like the raffinate neutralization residue, the Fe/Al residue was mostly gypsum with some iron and aluminum. Goethite was detected by XRD. This residue was also acidic but the pH was higher than the previous residues (for example, 4.8 in humidity cell leachate) and showed a chemical composition consistent with control by aluminum minerals (e.g. alunite). Concentration of cobalt, nickel and zinc in the process solutions at this stage was apparent by the increased concentrations of these metals in the leachates. The chemistry of the leachates appeared to have stabilized.

Lime demand to neutralize excess acidity was low as shown by neutralization potential of -10 kg $CaCO_3$/t.

3.6 Magnesium Residue

The magnesium residue is formed following recovery of the mixed hydroxide product and is the result of final addition of lime to the process solutions. The magnesium originates from the original concentrate leaching step (dissolution of olivine), and addition of magnesium oxide to precipitate the mixed hydroxide product.

The crystalline component of this residue was dominated by gypsum as shown by the elevated sulfate and calcium. The XRD scan showed that brucite (magnesium hydroxide) was present and this was supported by the magnesium content of the sample. The sample showed a strong fizz reaction to hydrochloric acid and carbonate content equivalent to 4% calcium carbonate (40 kg $CaCO_3$/t). This suggests that the hydroxide was partially converted to carbonate by atmospheric reaction.

Neutralization potential was high at 371 kg $CaCO_3$/t probably reflecting the presence of amorphous magnesium and possibly calcium hydroxide. This residue was alkaline (pH 9.6) and consistent with the hydroxide content. As a result of the high pH, concentrations of leachable metals were very low.

The leaching performance of this residue was variable depending on the test type and different from the other residues. Leachate pH was consistently elevated reflecting the dissolution of magnesium hydroxide, but conductivity was elevated compared to the other residues and trended upwards in the DNR reactor, downward in the shake flask and was stable in the humidity cell. The difference in trends appeared to be related to the sulfate concentration and equilibration with respect to gypsum in the presence of magnesium salt at different solid to liquid ratios. The humidity cell showed the highest sulfate and conductivity, whereas the DNR reactor leachate was equilibrating with respect to gypsum, and therefore sulfate was increasing. The shake flask leachate represented the most aggressive condition for dissolution of gypsum and was showing a decreasing trend in sulfate release.

3.7 Combined Residues

3.7.1 All Residues

The largest component of the combined residue was the gypsum residue, followed by leach residue, raffinate residue, Mg residue and Fe/Al residue. As a result the material was dominantly calcium sulfate from the gypsum residue. The XRD result showed the presence of calcite and siderite.

This residue produced alkaline leachate from SPLP (pH 9.2), 1:3 water leach (pH 8.7) and humidity cell (8.5) and the neutralization potential was 10 kg $CaCO_3$/t indicating excess alkalinity. This value is lower than neutralization potential of 28 kg $CaCO_3$/t obtained by arithmetically combining the residues in the proportion indicated in Table 1. The analytical value is being checked because the analytical and arithmetic values for total sulfur, sulfate sulfur and carbonate values are consistent (16 and 16%S, 15.9 and 15.5%S, and 0.10 and 0.15% CO_2, respectively).

The dissolution tests have confirmed that leachate pHs fall initially but remain pH neutral to slightly alkaline. The combined residues also show that the leachable metal concentrations are as low as for the Mg residue.

3.7.2 Combined without Gypsum Residue

Although this combination did not include the gypsum residue, gypsum remained a significant component from the raffinate neutralization, Fe/Al residue and Mg residues.

The leaching characteristics of this material were very similar to the combination of all residues though the lack of the acidic gypsum residue resulted in higher pH and neutralization potential (51 kg $CaCO_3$/t). This value compares well to 47 kg $CaCO_3$/t by arithmetically combining the residues in the proportions indicated in Table 1.

4 Implications to Management and Disposal

The testwork has shown that four of the five residues will be acidic. However, the actual residual leachable acidity appears to be low (amounting to a maximum of 17 kg $CaCO_3$/kg for the leach residue) and leachable metals are most likely contained in trapped process solutions rather than the solids. It can be expected that during operation of the disposal cells, pore water chemistry in the residues will at first be dominated by acidic process solutions. If during full-scale operation the residues are processed to extract process water, then re-pulped for pumping to the disposal cells, the chemistry of this water would reflect interaction of the water used for re-pulping with the residues, rather than the process water.

As the cells are decommissioned, long term leachate would remain acidic due to the lack of acid buffering minerals but leachable metals would decrease and eventually stabilize at the level indicated by the stable pH of the residues. The dominance of gypsum in the residues indicates that major ions in pore water will be calcium and sulfate from dissolution of gypsum.

The effect of combining the residues effectively neutralizes the stored process acidity because the Mg residue has a high neutralization potential (370 kg $CaCO_3$/t) present mainly as magnesium hydroxide. The calculated NP of the combined residues is 28 kg $CaCO_3$/t for the sample received from the pilot plant, and 23 kg $CaCO_3$/t for the expected proportions under commercial operations (Table 1), both of which indicate an excess of neutralization potential for the combined residues. The proportion of Mg residue can fluctuate significantly without affecting the overall pH of the mixed residue. Based on the data currently available, an Mg residue content as low as 0.5% (Pilot Plant proportions) or 0.8% (commercial-scale proportions) would be sufficient to offset the process water acidity and maintain the combined residues in a non-acidic state. This calculation also shows that the variation in the relative proportions of the major acidic residues (Leach and Gypsum) is not an important factor.

These conclusions are subject to confirmation under operational conditions but they indicate the 7 to 8% assumed for the mixed residue is probably much higher than needed to offset the acidity of the acidic residues.

5 Conclusions and Recommendations

Data obtained to date for the reactive residues indicate that:

- All residues except the magnesium residue and the combined residues are expected to have low pH.
- Titratable acidity in the residues is low and probably due to trapped process water rather than leachable acid salts.
- All dissolution tests on acid residues showed an initial rapid release of acidic water with high concentrations of metals, however, within a few leach cycles, acidity and metal concentrations decrease by orders-of-magnitude.
- The Mg residue had high neutralization potential and very low leachable metals due to its high pH.
- The Mg residue was an effective neutralizing agent for the acidic residues. A few percent Mg residue mixed with the acidic residues is sufficient to neutralize the acidity.

The test program currently has a high level of redundancy. The three types of dissolution test are showing very similar results (decreasing concentrations and overall ionic strength), and the concentrations observed are consistent with the differences in solid to liquid ratio. It is also apparent that leachate chemistry is controlled initially by flushing of acidic process water followed by slow leaching of metals at the expected long term moderately acidic pH in the residues. The development of stable conditions has occurred rapidly, and there is no requirement for long term leach testing. The humidity cells provide the best configuration to evaluate longer term leaching conditions for the residues because the liquid to solid ratio is the lowest of the three test types.

The following modifications to the test program are recommended:

- Immediately terminate the sequential shake flask and DNR reactor tests.
- Continue the humidity cell-style tests until stable conditions are demonstrated.
- Once stable conditions are apparent, terminate all individual residue tests but continue the combined residue tests.

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Appendix A
Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan

Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan NorthMet Project, Minnesota

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May 2006

Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan NorthMet Project, Minnesota

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1 Introduction

1.1 Background

PolyMet Mining Inc (PolyMet) is proposing to develop the NorthMet Project (Dunka Road Project of US Steel) near Babbitt, Minnesota. As a part of the Minnesota Department of Natural Resources (MDNR) "Permit to Mine" process a complete "mine waste characterization" will be required (Minnesota Rules Chapter 6132.1000). This document describes the plan developed for testing of flotation tailings and hydrometallurgical residue samples for the NorthMet Project.

The issues associated with tailings and residues at the NorthMet are expected to include acid rock drainage (ARD) and leaching of some heavy metals. The latter in particular are expected to include nickel and cobalt both of which do not require acidic conditions to be mobilized at elevated concentrations.

The objective of this program is to predict the reactivity of tailings and residues in their respective disposal areas for input into waste and water management planning, and environmental impact assessment. A separate Environmental Sampling and Analysis Plan describes characterization methods for effluents and emissions as metallurgical testing proceeds.

1.2 Geological Setting

The NorthMet Deposit is located in the intrusive mafic Duluth Complex of northern Minnesota. Disseminated copper-nickel-iron sulfides (chalcopyrite, cubanite, pentlandite and pyrrhotite) with associated platinum group element (PGE) mineralization will be extracted from several igneous stratigraphic horizons.

1.3 Agency Consultation and Design Process

This document was developed in consultation with staff from the Minnesota Department of Natural Resources (MDNR). The consultation included the following steps:

- June 22, 2005. A draft of the plan was prepared for MDNR Review.
- July 21, 2005. MDNR provided initial comments focussed primarily on flotation tailings.
- August 16, 2005. MDNR provided further comments on the characterization of hydrometallurgical wastes.
- September 9, 2005. SRK responded to the July 21, 2005 letter.
- September 14, 2005. A conference call was held to discuss the July 21, August 16 and September 9 letters.

This document includes responses to comments provided by MDNR, and has been prepared to conclude the design process and seek MDNR approval of PolyMet's plans to respond to the tailings/residue characterization component of requirements under Minnesota Rules 6132.1000.

1.4 Organization of This Document

This document describes:

- Section 2. Design basis for the program.
- Section 3. Analytical methods. This section describes methods used to analysis solids and leachates.
- Section 4. Use of the results in the context of water chemistry predictions.

1.5 Acknowledgements

The following individuals cooperated in the preparation of this plan:

- John Borovsky, Barr Engineering Company;
- Stephen Day, SRK Consulting;
- Paul Eger, MDNR;
- Jennifer Engstrom, MDNR;
- Don Hunter, PolyMet;
- Kim Lapakko, MDNR;
- Richard Patelke, PolyMet; and
- Jim Scott, PolyMet.

1.6 Laboratory Selection

The following laboratories will perform the procedures described in this plan (contact names for each laboratory are shown):

- ALS Chemex, North Vancouver, British Columbia – solids analysis listed in Section 4.1.1 (Bill Anslow);
- Optical – PolyMet or a Contractor (Richard Patelke);
- Sub-Optical Lab – McSwiggen and Associates (Peter McSwiggen);
- Canadian Environmental and Metallurgical Inc, North Vancouver, British Columbia – kinetic testing (Rik Vos); and
- Cantest Inc.. Vancouver, British Columbia - Kinetic test leachate analysis (Richard Jornitz).

2 Characterization Design

2.1 Objective of the Program

The overall objective of the program is to provide geochemical characterization information that can be used as inputs to design of management plans for the tailings and process residues and inputs into the environmental impact study (EIS) for the project.

2.2 Metallurgical Process Background

The processing and recovery of NorthMet Project ore to recover commodity metals will involve conventional flotation to produce a sulfide concentrate followed by hydrometallurgical treatment of the concentrate. The process flow sheets are provided in Appendix A.

Processing will result in the generation of the following waste products:

- Flotation tailings (low sulfide);
- Leach residue (silicates);
- Gypsum residue (calcium sulfate);
- Raffinate neutralization residue (calcium sulfate);
- Fe/Al residue (oxide); and
- Mg residue (oxide).

2.3 Tailings and Metallurgical Residue Disposal

Tailings and metallurgical residues are proposed for disposal in the existing impoundments at the former LTV Steel Mining Company (LTVSMC) operation (Figure 1). Review of the history of deposition at the LTVSMC tailings area indicates that the tailings are a result of processing ore feed from several iron ore pits and working faces in those pits. The ore was blended by truck delivery to loading pockets and then train delivery to crusher. Once in the plant ore was further blended in the coarse ore bins by a coarse ore tripper which continuously spread coarse ore across seven fine crushing lines and in the fine ore bins by a fine ore tripper which continuously spread fine ore across 34 mill lines. The fine ore was then processed through 34 mill lines in parallel. The tailings are a recombination from the 34 mill lines pumped to the basin and discharged at many spigots at the periphery of the basin. The tailings were deposited over many years as many layers in the basin.

Flotation tailings produced by conventional extraction of commodity-bearing sulfide minerals will be disposed in the existing Cells 1E, 2E and 2W. For the first five years of operation, cell 2W will be lined. Discharge methodology has not been determined but will most likely involve conventional discharge from one or more spigots. Hydrometallurgical residues produced by leaching of the sulfide concentrate will be disposed in lined basins within Cell 2W. The method used to transport the

residues to the cells has not been determined. The residues may be combined for disposal, or disposed separately depending on factors such as the possibility of selling some by-products.

The possibility of constructing tailings dams using cycloned tailings is being considered and has been incorporated in this test program.

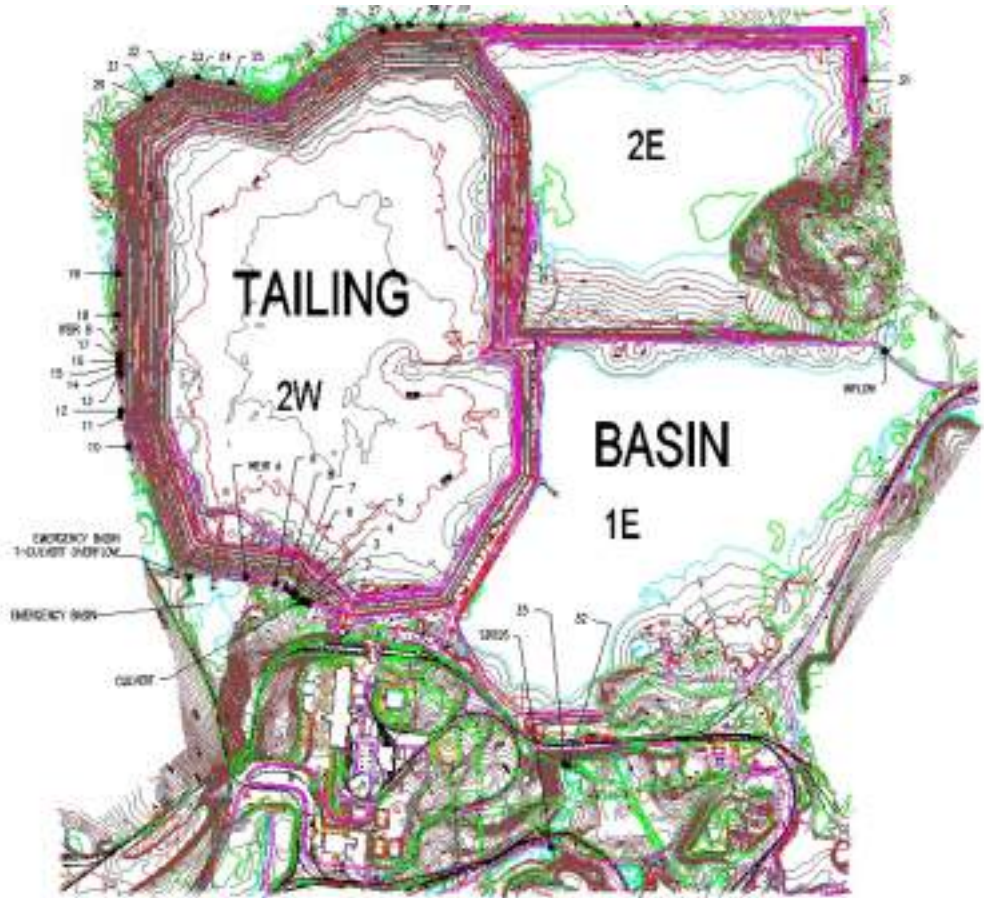

Figure 1: LTV Steel Mining Company Tailings Basins

2.4 Metallurgical Testing

Three ore composites were prepared under PolyMet's direction from diamond drill hole core bracketing head grades expected during mining. Pilot-scale metallurgical testwork was begun in July 2005 and continued into September. Flotation tailings testing was completed in August and included assessment of process alternatives. Addition of copper sulfate to improve sulfide concentrate recovery was evaluated for two ore composites.

A separate Environmental Sampling and Analysis Plan has been prepared to describe monitoring of air, water and solid emissions as the testwork proceeds. That document explains the rationale for preparation of the three ore composites representing copper grades of 0.3%, 0.35% and 0.4%. It also contains details of pilot plant monitoring designed to evaluate variations in tailings geochemical characteristics potentially produced by variations in ore characteristics and process performance. For example, the pilot testing program included frequent (every two hours) monitoring of tailings characteristic.

2.5 Design Basis

2.5.1 Flotation Tailings

Flotation tailings will consist primarily of silicate minerals with small amounts of residual sulfide minerals not recovered by flotation.

The number of variables expected to control reactivity is relatively small. Ore processing naturally results in a well-blended tailings product because the process requires a uniform feed to maximize recovery of commodities. Therefore, variables such as rock type, sulfide mineral type, silicate mineral type and source of ore within the layers of the Duluth Complex will not be significant. Preparation of the ore composite, which has been documented by PolyMet mimics mining at several faces and will result in composites each containing similar distributions of the main silicate and sulfide minerals. Variations will occur reflecting the distribution of commodity-containing minerals. Further, the requirement for grinding to optimize beneficiation of the commodity minerals limits the importance of variables such as mineral particle size and degree of liberation. The remaining variables are therefore expected to be:

- Sulfur content;
- Metal content; and
- Particle size (where separations occur at the deposition site as a result of hydraulic factors) resulting in differences in chemical and mineralogical composition.

The testing of tailings from the three ore composites will allow these variables to be evaluated. It is expected that recovery of sulfide minerals from ore will vary during testwork resulting in tailings containing variable concentrations of sulfur and metals. Results of two-hourly testing were provided in a memorandum to DNR dated January 6, 2006 (Appendix B). The composition of the four tailings samples initiated concurrently with preparation of this plan are shown in Table 1 along with the range of sulfur concentrations indicated by the two-hourly testing.

Table 1: Composition of Four Tailings Samples

Ore Composite	Total Sulphur Content of Ore	Copper Sulfate Used in Flotation	Range of Total Sulphur Concentrations in Tailings	Total Sulphur Content of Tailings Under Test
1	0.86	NO	0.19% to 0.28%	0.23%
		YES	0.09% to 0.13%	0.10%
2	0.90	NO	0.05% to 0.25%	0.20%
3	0.86	YES	0.09% to 0.25%	0.15%

2.5.2 Hydrometallurgical Residues

The residues are mainly chemical products in which the original concentrate components are oxidized to sulfates and hydroxides, and dissolved.

- Sulfur from the sulfide minerals is oxidized to sulfate and precipitated as calcium sulfate through the neutralization process;
- Iron released from sulfide and silicate minerals is precipitated as hydroxide; and
- Magnesium and aluminum released from silicate minerals are precipitated as hydroxides.

Copper is recovered in the process by electro-winning to produce copper anodes. Nickel, cobalt and zinc are recovered as a mixed hydroxide product.

Incomplete dissolution of the silicate and sulfide minerals results in generation of a fourth mineralogical leach residue.

Since these products represent near-complete dissolution, oxidation and neutralization of the concentrate, the chemistry of contact waters is expected to be controlled mainly by the simple dissolution of the compounds. Oxidation of residual sulfide minerals may occur from the leach residue but this effect is expected to minor.

Overall, the process of dissolution of neutralization products is unrelated to the composition of the ore and is not expected to vary in the long term except by depletion.

Pilot scale hydrometallurgical testing will be completed on two composite concentrate samples produced from processing of the three ore samples prepared with and without the use of copper sulfate in the flotation process.

3 Sample Handling and Analysis

3.1 Sample Shipping and Storage

SGS/Lakefield in Lakefield, Ontario, Canada is performing the metallurgical testing. Products will be shipped to Canadian Environmental and Metallurgical Inc. (Vancouver, British Columbia, Canada). Samples will be shipped and stored prior to testing as follows:

- Ore Feed Samples – Refrigerated;
- Sulfide Concentrate – Refrigerated;
- Flotation Tailings – Slurry in sealed pails with sufficient (6 cm) supernatant to ensure the samples are covered by water;
- Leach Residue – Cake, refrigerated; and
- Hydrometallurgical Residues – Cake, refrigerated.

Residual materials remaining after testing will be stored in the same condition as shipped.

3.2 Solids Characterization

3.2.1 Introduction

Table 1 summarizes the types of materials generated by testwork and the chemical testing procedures for each one. Physical testing of these products (including particle size determinations) is described in Environmental Sampling and Analysis Plan for the pilot plant test program. The DNR made several requests for multiple tests on some material types. In reality, the quantity of materials generated by metallurgical testing was limited and restricted the number of tests that could be performed. The 2-hourly testing showed that the sulfur content of the tailings did not vary widely and that the bulk samples under test will characterize the range of sulfur content of tailings.

Details of the test procedures are provided in the following sections.

Table 2: Procedures and Numbers of Samples for Testing

Material	Sulfur Forms	Neutralization Potential	Carbonate	Metals	TCLP	SPLP	Shake Flask	Optical Mineralogy	XRD	Sub-Optical	HCT	Sequential Shake Flask	Column	DNR Reactor Test	Layered Column
Ore feed	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore				
Sulfide Concentrate	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore						
Bulk Flotation Tailings – Without $CuSO_4$[1]	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore			1/Ore	
Bulk Flotation Tailings – With $CuSO_4$[1]	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore			1/ore	Multiple
Cyclone Sands	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore			1/ore	
Tailings Slimes	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		2	1/ore	
Tailings Beaches	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		2	1/ore	
Leach Residue	1	1	1	1	1	1	1	1	1	1	1	1		1	
Gypsum Residue	1	1	1	1	1	1	1		1	1	1	1		1	
Raffinate Neutralization Residue	1	1	1	1	1	1	1		1	1	1	1		1	
Fe/Al Residue	1	1	1	1	1	1	1		1	1	1	1		1	
Mg Residue	1	1	1	1	1	1	1		1	1	1	1		1	
Reactive Residues[2]	1	1	1	1	1	1	1		1	1	1	1		1	
Reactive Residue without Gypsum[3]	1	1	1	1	1	1	1				1	1		1	

Notes:

1. Flotation tailings samples were produced with and without the use of copper sulfate in the process to enhance recovery of sulfide minerals to the sulfide concentrate.

2. Combination of leach, gypsum, raffinate, Fe/Al and Mg residues.

3. Combination of leach, raffinate, Fe/Al and Mg residues.

3.2.2 Bulk Chemical Characterization

A split of each sample will be submitted for an extensive suite of analyses, as follows:

- Sulfur forms (total S, S as sulfate).
- Paste pH.
- Neutralization potential and carbonate.
- 50 elements (mostly metals by ICP scan following aqua regia (nitric and hydrochloric acids) digestion.
- Whole rock oxides. This is provides total concentrations of major elements.

Method detection limits are provided in Appendix C.

3.2.3 Regulatory Leachate Characterization

All samples will be submitted for regulatory leach tests (EPA 1311; EPA 1312) to provide data for waste classification purposes should this be needed. Testing of the ore samples will provide a baseline for comparison to effects from processing.

3.2.4 Shake Flask

A third leach leachate procedure was used as the first step of the sequential shake flask leach procedure described in Section 3.3.3 for hydrometallurgical residues. This procedure, developed by Price (1997) for the British Columbia Ministry of Energy Mines and Petroleum Resources, uses a lower leach ratio (1:3) to improve detection of low levels of metals. The lixiviant is deionized water which typically has a pH between 5 and 6. The leachate is not fixed as in the TCLP and SPLP methods. The extraction is performed by shaking in a glass or plastic container for 24 hours, after which the leachate is extracted and analyzed.

3.2.5 Mineralogical Characterization

Mineralogical characterization will include:

- Optical Analysis on feed, product and mineral wastes (Flotation Tailings, Leach Residue). Other residues will not be examined optically since they are precipitates;
- X-Ray Diffraction on all samples to determine crystalline compounds; and
- Sub-Optical Analysis on ore feed, mineral wastes, and if practical residues to determine the distribution of trace elements in individual minerals prior to and following processing.

3.3 Kinetic Test Methods

3.3.1 Humidity Cell

Humidity cell testing will be performed on ore feed (to characterize three types of ore stockpiles), flotation tailings and leach residues using ASTM Procedure D 5744 – 96 (Reapproved 2001). This procedure was selected for the following reasons:

- Similar procedures have been in use under different names since the late 1980s (e.g. MEND 1991). The results can therefore be evaluated in the context of more than a decade of experience using the procedure.
- It is a standard procedure approved by the ASTM and is therefore defensible as a method.

The ASTM procedure provides some options for varying the test procedure. Appendix D provides a detailed listing of the requirements of the ASTM procedure, options chosen and any variances from the ASTM procedure.

3.3.2 MDNR Reactor

To allow comparison with previous MDNR studies, bulk tailings samples will be tested using a procedure referred to as the "MDNR Reactor" experiment. An apparatus specifically designed by MDNR (Appendix E) contains 75 g of solids.

3.3.3 Sequential Shake Flask Test

All residues are being tested using a sequential leach procedure consisting of weekly repetition of the leach procedure developed by Price (1997) in which the solid to liquid ratio is 1:3 (Section 3.2.4). The procedure involves weekly leaching of roughly 300 g of solids in a plastic bottle. The leaching step consists of addition of deionized water and agitation for 24 hours. The leachate is then decanted for analysis. Between leach steps, the bottle remains open to the atmosphere.

This procedure was requested by the DNR to provide a more aggressive evaluation of hydrometallurgical residue dissolution than occurs in humidity cells.

3.3.4 Leach Columns

A procedure to evaluate the interaction between leachate from NorthMet tailings and LTV tailings was designed and presented to the DNR (Appendix F). The procedure provides for two subaerial columns to generate leachate from the NorthMet tailings.

3.3.5 Leachate Analysis

Leachates from kinetic tests will be analyzed for the parameters indicated in Table 2, which also shows reporting limits. These limits are higher than the detection limits for the analytical instruments. Reporting limits represent the level at which the analytical laboratory (CANTEST) is

confident that the concentrations are quantifiable to an acceptable level. The instrument is able to detect much lower levels but these concentrations carry a very high degree of uncertainty which includes “undetectable”.

Low level leachate analyses for dissolved elements as shown in Table 2 are performed every four weeks (weeks 0, 4, 8 etc). On the intervening even numbered weeks (2, 6, 10 etc.), an ICP scan is performed using a higher detection limit primarily to determine the trend in major ions. pH and conductivity are determined every week. Acidity, alkalinity, inorganic carbon, sulphate, fluoride and chloride are determined every other week.

It is expected that testing of flotation tailings will result in very dilute leachates containing low concentrations of the metals of interest. Back-calculation of metal concentrations from other testwork performed by DNR indicates that cobalt and nickel concentrations could be in the tens of nanograms per litre (ng/L) for nickel and near nanograms per litre for cobalt. Quantification of these low metal concentrations is needed to provide reasonably constrained estimates of metals concentrations in the tailings storage facility.

A number of different approaches are available to quantify low levels of nickel and cobalt:

- The routine leachate analysis will achieve a reporting level of 0.0001 mg/L (100 ng/L). Should concentrations be undetected, detection limits of 50 ng/L can be obtained with additional processing effort using the same routine method.
- Specialist methods can achieve lower detection limits. These are non-routine (for example, evaporation to increase concentrations) and will need to be developed as the need arises.
- Existing testwork demonstrates that good correlations exist between cobalt and nickel concentrations in leachates. Detectable nickel concentrations can be used to estimate cobalt concentrations if this relationship can be demonstrated.
- In the event of undetectable low levels, a scale-up methodology will be agreed upon with MDNR to translate non-detectable concentrations to tailings concentrations. Detection limit values will be used in modeling calculations.

Table 3: List of Parameters for Low Level Analysis of Humidity Cell Leachates

Parameter	Reporting Limit	Parameter	Reporting Limit
pH (standard units)	-	Acidity	1
Conductivity (µS/cm)	1	Alkalinity	1
Chloride	0.2	Sulfate	0.5
Fluoride	0.05	Total Inorganic Carbon	1
ORP (mV)	-		
Dissolved Elements (mg/L)			
Aluminum	0.001	Mercury	0.00005[2]
Antimony	0.0001	Molybdenum	0.00005
Arsenic	0.0001	Nickel	0.0001 (0.00005)[1]
Barium	0.0001	Potassium	0.02
Beryllium	0.0002	Selenium	0.0002
Bismuth	0.0002	Silicon	0.05
Boron	0.005	Silver	0.00005
Cadmium	0.00004	Sodium	0.01
Calcium	0.01	Strontium	0.0001
Chromium	0.0002	Tellurium	0.0002
Cobalt	0.0001 (0.00005)[1]	Thallium	0.00002
Copper	0.0001	Thorium	0.0001
Iron	0.01	Tin	0.0001
Lead	0.00005	Titanium	0.0002
Lithium	0.0002	Uranium	0.00005
Magnesium	0.005	Vanadium	0.0002
Manganese	0.00005	Zinc	0.001

Notes: 1. Low detection limits are available for cobalt and nickel as shown.
2. Lower level mercury analyses will be performed on selected samples.

3.4 Analysis of Remaining Sample Following Dissolution Tests

Analyses of the remaining sample following dissolution tests will be considered depending on the results obtained from the tests. Generally, these analyses can be of value if the test has undergone a major chemical change during the procedure (e.g. change from alkaline to acidic leachate) or if calculations indicate that a large quantity of one or more minerals or elements has been depleted. These is little value in post-test analysis if the depletion quantity is less than the uncertainty that can be expected from sampling of the test residue analysis. Experience also indicates that mineralogical analyses are also of little value unless weathering processes have had a detectable effect on the sample.

Therefore, residue analyses will be performed if:

- A large drop in pH has occurred (for example, from above 7 to below 5).
- Depletion calculations indicate that more than 10% of an important component was removed during the procedure.

Residue analyses will consist of the same procedures performed prior to the test.

3.5 Quality Assurance/Quality Control

A Quality Assurance Project Plan (QAPP) is being prepared for this project. To summarize, QA/QC includes the following components:

- Roughly 10% of all solids analyses will be performed in duplicate as sample availability permits.
- Roughly 10% of all cell and reactor tests will be run as duplicates if sufficient test material is available.
- A blank cell and reactor containing no sample will be operated to check for contamination of leachates by construction materials.
- Individual leachate results will be reviewed.
- Ion balances on leachate results will be reviewed. In general, imbalances of ±10% are considered acceptable. Re-analysis if requested depending on the nature of the imbalance.
- Data trends in kinetic test leachates will be analysed to check for anomalies.

4 Use of Data for Water Quality Predictions

4.1 Introduction

The data obtained from these programs will be used to estimate water quality during operation and closure. The following sections describe the application of the data to making water chemistry predictions.

4.2 Operational Water Quality

4.2.1 Flotation Tailings

Operational water quality in flotation tailings impoundments tends to be dominated by process water since this is the largest volume of water moving into and out of the impoundment by discharge and reclaim. Processes resulting from oxidation are not usually significant because continual placement of fresh tailings covers up older tailings before extensive weathering is initiated. The tailings pond water and trapped pore waters therefore reflect re-circulating process water. Seepage from the impoundments typically has a process water signature modified by anoxic conditions in the saturated tailings and interaction with LTVSMC taconite tailings.

Seepage chemistry will also be influenced by dam construction, particularly whether drains will be needed for stability. If drains are needed, water will be drained horizontally away from the dams, rather than the normal vertical seepage thru the coarse fraction of tailings near the dam.

The method used to estimate tailings pond water during operations is typically a coupled water and load balance that evaluates the effect of build-up of solutes in the tailings pond water due to interactions (e.g. mineral dissolution), reagent addition in the process, unintended additions in the process (e.g. Mo from lubricants), dilution due to rain and snowfall, dilution by run-in, and solute load loss due to encapsulation. If kinetic testing shows a short term leaching effect from beaches and dam faces, these loads are included.

The effect of interaction of saturated tailings with LTVSMC tailings will be evaluated directly by column tests (Appendix F). These tests will indicate whether any significant losses or additions occur as process water moves through the taconite tailings.

4.2.2 Leach Residues

Rinsed leach residues are expected to be relatively soluble and water chemistry associated with the residues will be dominated by equilibration of rinse. Water quality may be affected by the method of disposal (pumped slurry vs truck hauled solids). Since the residues will be continually accumulated, the operational water chemistry will be a result of mixing of rinsate water with precipitation and run-in. The chemistry of contact water during operation will be estimated directly from dissolution test results. Pore water chemistry indicated by testwork will be evaluated using MINTEQA2 or similar thermodynamic equilibrium models.

The effect of small amounts of sulfide minerals in the leach residues will be evaluated.

4.3 Water Quality at Closure and Post Closure

4.3.1 Flotation Tailings

At closure, the main effect is removal of inflows of process water and on-set of oxidation of tailings resulting in metal and possibly acidity loadings. The water and load balance developed for operational conditions is typically modified to evaluate these effects.

The long term closure modeling will need to consider evolution of the tailings profile in response to oxidation. This type of modelling uses humidity cell weathering rates, tailings physical characteristics, and moisture profiles predicted by modelling (HELP and HYDRUS-2D) as inputs. The propagation of the oxidation front and acidity front through the tailings is predicted and used to model the movement of solutes. The migration of the fronts can be used to estimate changes in solute loading in the future due to arrival of chemical fronts at the base of tailings. This type of modeling can be coupled with groundwater models to predict the chemistry of groundwater leaving the site.

4.3.2 Leach Residues

Water associated with the leach residues is expected to evolve as rinsate is displaced by dilute water from rainfall and snowmelt. Humidity cells will show whether this results in changing water quality. Concentrations indicated from humidity cells may be suitable for direct prediction of water chemistry or may be adjusted using MINTEQA2 or similar thermodynamic equilibrium models.

5 References

MEND, 1991. Acid Rock Drainage Prediction Manual. Mine Environment Neutral Drainage Program. Report 1.16.1b.

Nevada Division of Environmental Protection, 1996. Meteoric Water Mobility Procedure (MWMP) Standardized Column Test Procedure. NDEP publication MWMP.ltr. May 3 1996. 6p.

Price, W. 1997. Draft Guidelines and Recommended Methods for the Prediction of Metal Leaching and Acid Rock Drainage at Minesites in British Columbia. Reclamation Section, British Columbia Ministry of Energy and Mines. April 1997.

Sobek A A, Schuller W A, Freeman J R, and Smith R M., 1978. Field and Laboratory Methods Applicable to Overburden and Minesoils. USEPA Report No. 600/2-78-054, 203 pp.

Appendix A

Process Flow Sheets

BATEMAN METALS

ORE CAR TIPPER

COARSE CRUSHER DUMP POCKET

PRIMARY CRUSHER (1)

PRIMARY CRUSHER PRODUCT POCKET

SECONDARY CRUSHER (4)

APRON FEEDER (4)

CONVEYOR

CONVEYOR

COARSE ORE BIN

FEED CHUTE (18)

BELT FEEDER (6)

RETRACTABLE CONVEYOR (3)

TERTIARY CRUSHER (3)

VIBRATING SCREEN FEEDER (6)

VIBRATING SCREEN (6)

QUARTERNARY CRUSHER (6)

CONVEYOR

CONVEYOR

FINE ORE BIN

FEEDER (132)

RODS

BALLS

ROD MILL (12)

PRIMARY CYCLONE (36)

BALL MILL (12)

PAX

AIR

MIBC

ROUGHER FLOTATION (4)

ROUGHER CONDITIONING TANK (2)

AIR

MIBC

PAX

SCAVENGER FLOTATION (12)

REGRIND CYCLONE (9)

FLOCCULANT

FLOTATION TAILS THICKENER (2)

WW1572

PAX

CLEANER 1 CONDITIONING TANK (1)

MIBC

AIR

CLEANER 1 FLOTATION (8)

PAX

REGRIND HOPPER (2)

REGRIND MILL (3)

FLOCCULANT

CONCENTRATE THICKENER (1)

PROCESS WATER TANK (1)

FLOAT TAILINGS

AIR

MIBC

WW1572

CLEANER 2 FLOTATION (4)

PAX

CLEANER 3 CONDITIONING TANK (1)

CLEANER 3 FLOTATION (4)

(2)

(2)

CONCENTRATE GRINDING

FINAL CONCENTRATE SURGE TANK (2)

TO AUTOCLAVE FEED TANKS

BAB403-P-DX-003

FIGURE 1
POLYMET FEASIBILITY STUDY
COMMINUTION AND FLOTATION
SCHEMATIC FLOW DIAGRAM

BATEMAN METALS

1ST STAGE HYDROXIDE PRECIPITATION TANK (3)
AIR
MgO
1ST STAGE HYDROXIDE THICKENER (1)
FLOCCULANT
SERVICE WATER
HYDROXIDE FILTER (1)
MIXED HYDROXIDE PRODUCT
2ND STAGE HYDROXIDE PRECIPITATION TANK (3)
LIME
FLOCCULANT
2ND STAGE HYDROXIDE THICKENER (1)
TO AUTOCLAVE LET DOWN TANK
Mg REMOVAL TANK (4)
LIME
Mg REMOVAL THICKENER (1)
TO HM PROCESS WATER TANK
TO REACTIVE RESIDUES
STEAM
N_2
NaHS
Cu PRECIPITATION TANK (3)
CuS THICKENER (1)
FROM FINAL CONCENTRATE SURGE TANKS
HCl
AUTOCLAVE FEED TANK (3)
OXYGEN
AUTOCLAVE (3)
BLOWDOWN TO FLOTATION TAILINGS
SERVICE WATER
COOLING TOWER
SERVICE WATER
AUTOCLAVE GAS SCRUBBER (3)
HEAT EXCHANGER
TO ATMOSPHERE
SERVICE WATER
VENTS FROM ALL TANKS
FINAL GAS SCRUBBER
AUTOCLAVE FLASH VESSEL (3)
FLOCCULANT
LEACH RESIDUE THICKENER (1)
LIMESTONE
AIR
STEAM
BLEED
Fe REMOVAL TANK (4)
FLOCCULANT
Fe REMOVAL THICKENER (1)
HM PROCESS WATER
LIMESTONE
Fe/Al REMOVAL THICKENER FILTER
Fe/Al RESIDUE TO REACTIVE RESIDUES
Al REMOVAL TANK (2)
FLOCCULANT
Al REMOVAL THICKENER (1)
PRE NEUTRALISATION THICKENER (1)
FLOCCULANT
GYPSUM FILTER (1)
HM PROCESS WATER
GYPSUM RESIDUE TO REACTIVE RESIDUES
LIMESTONE
PRE NEUTRALISATION TANK (4)
FLOCCULANT
THICKENER (1)
SERVICE WATER
PGM FILTER (1)
NaHS
PGM PRECIPITATION (3)
HM PROCESS WATER
LEACH RESIDUE FILTER (3)
SERVICE WATER
HM PROCESS WATER
HM PROCESS WATER TANK
REACTIVE RESIDUES
ORGANIC MAKE-UP
LOADED ORGANIC TANK
COPPER EXTRACTION (3)
COPPER STRIPPING (2)
WASH (1)
SERVICE WATER
GUAR GUM
$CoSO_4$
H_2SO_4
COPPER ELECTROWINNING (160)
PRODUCT COPPER CATHODES
H_2SO_4
OXYGEN
RELEACH VESSEL (2)
TO ATMOSPHERE
SERVICE WATER
EW MIST SCRUBBER
DISCHARGE TO SOLVENT EXTRACTION
SERVICE WATER
PGM CONCENTRATE FILTER (1)
PRODUCT PGM CONCENTRATE
LIMESTONE
FLOCCULANT
RAFFINATE NEUTRALISATION THICKENER (1)
HM PROCESS WATER
RAFFINATE NEUTRALISATION (3)
RAFFINATE NEUTRALISATION FILTER (1)
GYPSUM RESIDUE
CRUD REMOVAL
SOLIDS TO REACTIVE RESIDUES
LIQUID TO LOADED ORGANIC TANK
BAB403-P-DX-004

FIGURE 2
POLYMET FEASIBILITY STUDY
HYDROMET PROCESS PLANT
OPTION 1 - MIXED Ni/Co
HYDROXIDE PRECIPITATION
SCHEMATIC FLOW DIAGRAM

Appendix B

Update on Sample Selection from 24 Hour Testwork

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Memo

To:	Jennifer Engstrom, MDNR	**Date:**	January 6, 2006
cc:	John Borovsky, Barr	**From:**	Stephen Day
	Jim Scott, PolyMet		
	Don Hunter, PolyMet		
Subject:	NorthMet Project Tailings and Hydromet Residue Testwork – Update on Sample Selection from 24 Hour Testwork	**Project #:**	1UP005.001

Jennifer

We have now received the 2-hourly tailings total sulfur analyses from the pilot plant testwork. These results allow final recommendations to be made for the selection of samples for tailings and hydrometallurgical testwork.

1 Results and Implications of 2-Hourly Sulfur Analyses

Results of the 2-hourly sulfur analyses are shown by the coloured solid lines in Figure 1. The broken horizontal lines are the concentration of sulfur in the composite tailings samples currently being tested in humidity cells. The sulfur content of the ore composites was very uniform (Parcel 1, 2 and 3, 0.86%, 0.9% and 0.86%, respectively) as shown by the solid black lines in Figure 1.

The trend in sulfur results in tailings is explained by the chronology of the testwork and evaluation of addition of copper sulfate as a reagent:

- Flotation testwork began on July 17 with Parcel 2 without the use of copper sulfate. Parcel 2 was processed entirely without using copper sulfate. As shown, sulfur concentrations varied from 0.05% to 0.25% reflecting adjustment of the process conditions early in the testwork. The average was 0.19%. The composite tailings sample has a sulfur content of 0.2% closely representing the average.

- Testwork continued with Parcel 1 without using copper sulphate. Processing was continuous so one point is shared between Parcel 2 and Parcel 1. The range of sulfur concentrations was 0.19% to 0.28% with an average of 0.24%. The composite sample was 0.23% and is close to the average.

- Pilot plant testwork was suspended on July 19 to allow for further bench scale testing on recovery of metals.

- The pilot plant resumed on August 8 using Parcel 1. Addition of copper sulfate was evaluated. This reagent causes activation of the sulfide mineral surfaces and improves bulk sulfide flotation. The effect of copper sulfate on tailings characteristics was immediately apparent for Parcel 1.

Total sulfur concentrations decreased to a range of 0.09% to 0.13% (average 0.1%) and the resulting tailings composite was 0.1%.

- Processing continued with Parcel 3 using the copper sulfate additive. Sulfur content of the tailings varied over a wider range (0.09% to 0.25%, average 0.18%) though the range was comparable to the total range indicated by processing of other ore packages. The resulting composite had a total sulfur content of 0.15%.

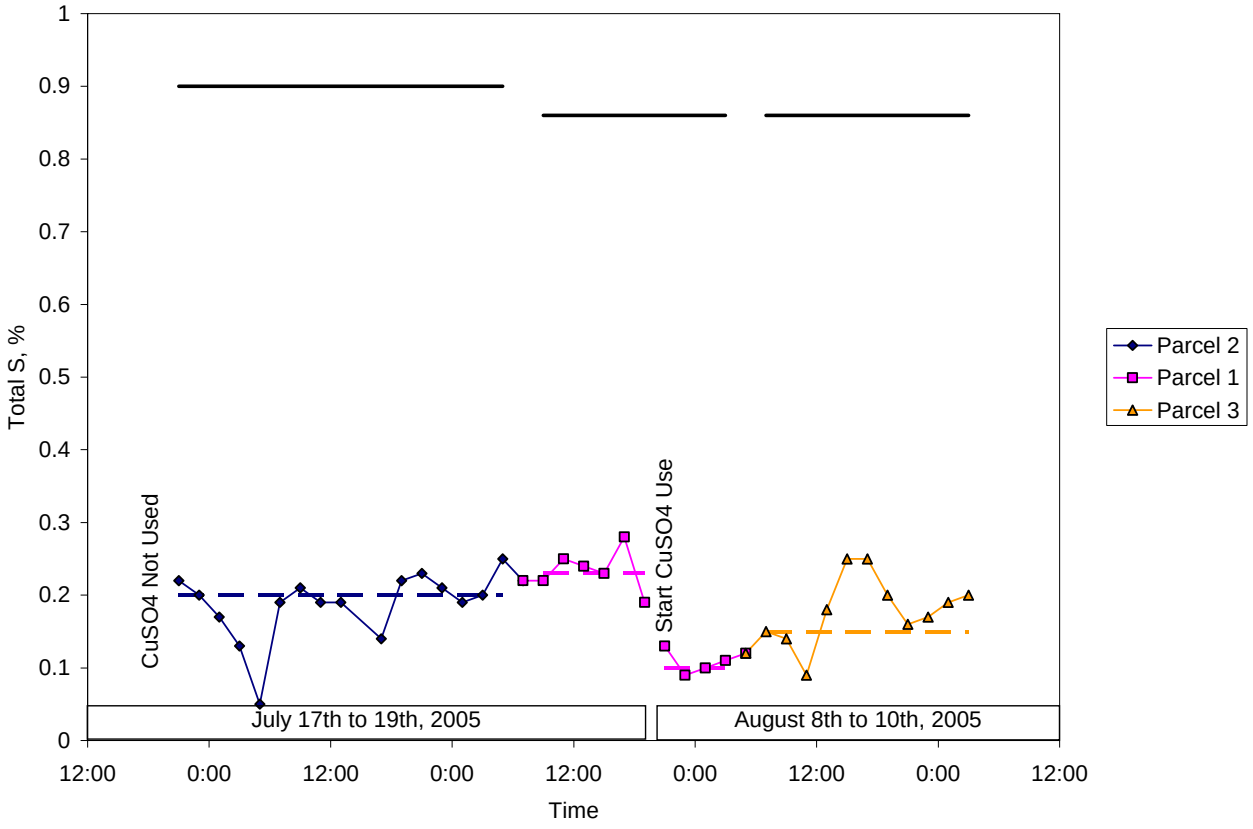


Figure 1. Results of 2-Hourly Total Sulfur Analyses. Solid lines and points connect 2-hourly results. Broken solid lines are sulfur concentrations in composite tailings samples representing each stage of testwork. Solid horizontal lines are the respective ore composite sulfur contents.

Based on the process testwork, Polymet has the made decision to advance the project with the use of copper sulfate to optimize overall sulfide mineral flotation. This decision is beneficial for the tailings since it is expected to lower the overall sulfide content.

The process testwork showed that sulfur concentrations in the tailings can be expected to vary in response to changes in process conditions including the use of copper sulfate. Parcel 3 showed that the use of copper sulfate may not always result in low sulfur content in tailings, and therefore there is need to capture sulfur concentrations approaching 0.25% in the kinetic testwork. The samples generated without copper sulfate provide the required range and can be tested to represent the potential for higher sulfur concentrations in the tailings. The lack of copper sulfate for the Parcel 2 and 1 samples is not expected to have significantly affected the reactivity of the residual sulfide minerals in the tailings:

It is therefore concluded that:

- Kinetic testing of all four tailings samples should be continued.
- No additional samples are needed to represent the range of sulfur content expected in tailings.

2 Testing of HydroMet Residues

2.1 Source of Sulfide Concentrate for Hydromet Process Evaluation

Evaluation of the hydromet process was performed using two bulk sulfide concentrates produced by processing of ore parcels 2 and 1 (without copper sulfate) and ore parcels 1 and 3 (with copper sulfate). The sulfide concentrates contained the following total sulfur concentrations:

- No Copper Sulfate
  - Parcel 2 – 23.6%
  - Parcel 1 – 21.3%
- With Copper Sulfate
  - Parcel 1 – 22.1%
  - Parcel 3 – 21.6%

It is apparent that the sulfur content of the concentrates does not vary significantly though the effect of copper sulfate on concentrate sulfur content for Parcel 1 is apparent and corresponds with the matching decrease in sulfur content of the tailings. Since the decision has been made to proceed with the use of copper sulfate, only the residues produced from sulfide concentrate generated using copper sulfide should be tested.

2.2 HydroMet Residues

All the expected HydroMet Residues were produced by processing of the sulfide concentrate generated using copper sulfate.

A difference exists between the way that the residues were recovered in the pilot test compared to actual operating conditions.

To summarize, the first step in the process is the leaching of the sulfide concentrate to produce a low pH pregnant solution containing all the commodity metals. Subsequent recovery of the metals involves a series of pH adjustments to the leach solution that results in precipitation of products and residues. The products are then refined to recover the contained metals (copper, nickel, cobalt, PGM, zinc). The residues contain entrained leach solutions that have to be recovered to optimize recovery of commodity metals. Under full-scale operating conditions, recovery of the leach solutions from the residues will occur by rinsing the residue cakes with pH-adjusted re-cycled final process water to displace the leach solutions. The pH adjustment is required to ensure that metals in the leach solution are not lost to the solids.

However, under pilot plant conditions, the recycled process water was not available because the processing of the leach solutions occurred in a stepwise rather than continuous fashion. Each metal recovery step was performed and completed before proceeding to the next. The final process solution that will be used for rinsing at full-scale was only generated at the end of the pilot plant and was therefore not available for the residue rinsing steps. The difference between full-scale and pilot plant conditions represents a practicality of metallurgical testing in that operation the pilot plant continuously is not an option with the available quantity feed concentrate.

Residues generated by the pilot plant were rinsed with locally obtained river water. No additional rinsing of the residues is proposed for the dissolution testwork. Rinsing with river water was less aggressive in displacing metal-laden leach solutions than can be expected with pH-adjusted process water. The residues can therefore be expected to contain higher metal content than under operating conditions and indicate greater leachable metals in dissolution tests. The testwork will tend to over-estimate rather than under-estimate water quality for water management planning and impact assessment.

3 Conclusions

The following actions are proposed:

- Kinetic testing of all four existing tailings samples in dissolution tests will continue.
- Testing of residues produced by hydromet testing of sulfide concentrate (with copper sulfate) will be started as described in the "Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan"

Appendix C

Parameter Lists and Detection Limits for Analysis of Solids

ALS/CHEMEX METHOD CODE	ITEM	UINTS	CHEMEX DETECTION LIMIT
ME-ICP61 (four acid)	CU%	%	0.001
ME-ICP61 (four acid)	NI%	%	0.001
S-IR08 (LECO SULFUR)	S%TOT	%	0.01
ME-ICP61 (four acid)	S%ICP	%	0.01
PGM-ICP23 (30 GRAM)	PT_PPB	PPB	5
PGM-ICP23 (30 GRAM)	PD_PPB	PPB	1
PGM-ICP23 (30 GRAM)	AU_PPB	PPB	1
ME-ICP61 (four acid)	CO_PPM	PPM	1
ME-ICP61 (four acid)	AG_PPM	PPM	0.5
ME-ICP61 (four acid)	ZN_PPM	PPM	2
ME-ICP61 (four acid)	CD_PPM	PPM	0.5
ME-ICP61 (four acid)	MO_PPM	PPM	1
ME-ICP61 (four acid)	PB_PPM	PPM	2
ME-ICP61 (four acid)	AS_PPM	PPM	5
ME-ICP61 (four acid)	CR_PPM	PPM	1
ME-ICP61 (four acid)	V_PPM	PPM	1
ME-ICP61 (four acid)	TI%	%	0.01
ME-ICP61 (four acid)	AL%	%	0.01
ME-ICP61 (four acid)	CA%	%	0.01
ME-ICP61 (four acid)	FE%	%	0.01
ME-ICP61 (four acid)	K%	%	0.01
ME-ICP61 (four acid)	NA%	%	0.01
ME-ICP61 (four acid)	MG%	%	0.01
ME-ICP61 (four acid)	MN_PPM	PPM	5
ME-ICP61 (four acid)	P_PPM	PPM	10
ME-ICP61 (four acid)	BA_PPM	PPM	10
ME-ICP61 (four acid)	BE_PPM	PPM	0.5
ME-ICP61 (four acid)	BI_PPM	PPM	2
ME-ICP61 (four acid)	SB_PPM	PPM	5
ME-ICP61 (four acid)	SR_PPM	PPM	1
ME-ICP61 (four acid)	W_PPM	PPM	10

ALS/CHEMEX METHOD CODE	ITEM	UINTS	CHEMEX DETECTION LIMIT
ME-ICP41 (Aqua regia digestion)	CU%	%	0.001
ME-ICP41 (Aqua regia digestion)	NI%	%	0.001
ME-ICP41 (Aqua regia digestion)	S%ICP	%	0.01
ME-ICP41 (Aqua regia digestion)	CO_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	AG_PPM	PPM	0.2
ME-ICP41 (Aqua regia digestion)	ZN_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	CD_PPM	PPM	0.5
ME-ICP41 (Aqua regia digestion)	MO_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	PB_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	AS_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	CR_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	V_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	TI%	%	0.01
ME-ICP41 (Aqua regia digestion)	AL%	%	0.01
ME-ICP41 (Aqua regia digestion)	CA%	%	0.01
ME-ICP41 (Aqua regia digestion)	FE%	%	0.01
ME-ICP41 (Aqua regia digestion)	K%	%	0.01
ME-ICP41 (Aqua regia digestion)	NA%	%	0.01
ME-ICP41 (Aqua regia digestion)	MG%	%	0.01
ME-ICP41 (Aqua regia digestion)	MN_PPM	PPM	5
ME-ICP41 (Aqua regia digestion)	P_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	B_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	BA_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	BE_PPM	PPM	0.5
ME-ICP41 (Aqua regia digestion)	BI_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	GA_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	HG_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	LA_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	SB_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	SC_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	SR_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	W_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	TL_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	U_PPM	PPM	10

ALS/CHEMEX METHOD CODE	ITEM	UINTS	CHEMEX DETECTION LIMIT
ME-ICP06--whole rock geochemisrty by ICP-AES	SIO2	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	AL203	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	TIO2	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	FE2O3	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	CAO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	MGO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	MNO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	NA2O	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	K2O	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	P2O5	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	BAO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	SRO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	LOI	%	0.01

Appendix D

Options and Variance in ASTM Humidity Cell Procedure

9. Sample Preparation

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
9.1	Air dry as-received bulk samples of solid material to prevent the additional oxidation of reactive minerals or compounds. If air drying is not practicable, oven dry the solid material at a maximum temperature of 50 ± 2°C for 24 h, or until a constant weight is reached.	Samples were air-dried at room temperature (~ 20 °C).	
9.1.1	If exploration-generated or run-of-mine solid material samples are not readily available, archived dried and crushed samples from geological exploratory or development drilling programs may be used for preliminary evaluations of ore and waste rock from new operations; this is provided that the available solid material samples are not significantly finer than 95 % passing a No. 12 (1.7-mm) sieve. Document the sample drying and preparation procedures used during the drill sampling program in order to interpret the results properly. Evaluate the effects of drying temperature on metals volatilization (for example, mercury in cinnabar vaporizes at temperatures exceeding 80 to 90°C) and mineral morphology and chemistry modifications (for example, on heating at temperatures exceeding 100°C, chalcocite changes crystal form and is oxidized subsequently from Cu_2S to CuO, $CuSO_4$, and SO_2). Especially ensure that the effects of particle size distribution changes resulting from the more finely crushed sample are considered in the interpretation (this is, the potential for increased liberation of acid-producing and acid-consuming minerals with an attendant increase in mineral surface area).	NA	
9.1.2	In mining waste evaluations, the particle size for mill tailings will be significantly finer (commonly less than 150 µm/100 mesh) than the particle size distributions from ore and waste rock. Pilot plant tailings should be used if mill tailings are not available.	NA	
9.2	Screen the air-dried bulk samples through a 6.3-mm (¼-in.) screen in accordance with Test Method E 276. Crush any oversize material so that 100 % passes the screen.	NA	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
Note 7	Caution: Recent accelerated weathering studies of run-of mine waste rock from metal mines demonstrate that crushing a bulk sample so it passes a 6.3-mm (¼-in.) screen may change the character of the sample by artificially increasing liberation and consequent surface areas of acid-producing and acid-consuming minerals contained in the + 6.3-mm (¼-in.) material. A suggestion for avoiding this problem is to segregate the - 6.3-mm (¼-in.) fraction by screening rather than crushing, and to test that fraction according to the protocol and equipment described in this text. The + 6.3-mm (¼-in.) material can be tested separately (for example, Brodie, et al (10) describe a large-scale humidity cell test that would accommodate – 75-mm material). Samples from the drill core and cuttings also present material sizing problems, which must be considered when interpreting drill core and cuttings accelerated data. The drill core must be crushed to -6.3-mm (¼-in.) to fit the cell described in this test method. The resulting size distribution from crushing will differ from that of run-of-mine due to differences in fracture patterns inherent to blasting practices that produce run-of-mine material. By contrast, drill cuttings size fractions are commonly less than 6.3-mm (¼-in.) due to the rotary-percussive nature of obtaining the sample.	NA	
9.3	Mix and divide the bulk sample to obtain a representative test unit with a weight in the range of 8 to 10 kg, using a riffle splitter with 1-in. (2.54-cm) chutes. Divide the test unit into eight nominal 1-kg specimens. Seal each test specimen in a moisture-barrier bag.	NA	
Note 8	The dried sample should be mixed through the riffle splitter at least once before making any splits; recombine the splits resulting from the sample mixing exercise by pouring individual splits either over each other or through the splitter again. Once the actual split is made, it is wise to re-mix it (according to the above procedure) prior to making the next split.		Samples were mixed through the riffle splitter once.
9.4	Select one test specimen at random, and determine the moisture content by weighing and drying to constant weight at 80 ±5°C.		Determined at 20 °C
9.4.1	Crush the dried test specimen so that at least 95 % passes a 1.7-mm (10-mesh) screen, in accordance with Test Method E 276.	NA	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
9.4.2	Divide the crushed test specimen in half twice, using a riffle splitter with 6.35-mm (¼-in.) chutes, and select a ¼ subsample at random.	NA	
9.4.3	Transfer the selected subsample to a ring and puck grinding mill and grind to a nominal of 95 % passing a 150-µm (100-mesh) screen, in accordance with Test Method E 276. Use the subsample for chemical and mineralogical characterization of the test unit.	NA	
9.5	Select one test specimen at random, and determine the particle size distribution in accordance with Test Method E 276.	ASTM	
9.6	Select one test specimen at random for use in the accelerated test method. Divide the test specimen into four nominal 250-g subsamples using the riffle splitter with 25.4-mm (1-in.) chutes, and label and store in vapor-barrier bags until it is time to load the humidity cells.	NA	
9.7	Reserve the remaining test specimens for replicated testing or to resolve disputed results.	NA	

10. Apparatus Assembly

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
10.1	The humidity cells are table-mounted at a height sufficient to accommodate the placement of both the humidifier and one Erlenmeyer flask for effluent collection from the bottom of each cell. During the water-saturated and dry-air portions of each weekly cycle, feed air is metered to the bottom of each cell at the selected rate (1 to 10L/min). Feed air for the three-day dry–air portion is routed first through a desiccant column and then to each of the cells through a dry-air manifold. Feed air for the water-saturated air portion is routed through a water-filled humidifier by means of aeration stones or gas dispersion fritted cylinders/disks, and then to each humidity cell lid air exit port to prevent the short circuiting of air through cells containing more permeable solid material samples. A separatory funnel rack is mounted on the table that holds the cells if the weekly water leach is applied dropwise (drip trickle). Multiple separatory funnels (one for each cell) are held in the rack during the drip trickle leach that is performed on the seventh day of each weekly cycle. The separatory funnel can be used to meter the required water volume slowly down the sides of the cell wall until the sample is flooded if the weekly leach is to be a flooded leach.	Humidity cells are constructed of acrylic tubing with an inside diameter of four inches and an overall height of twelve inches, with an acrylic base plate. The base plate is glued to the tube and threaded with a nylon hose adapter to which a length of tubing is attached to allow for leachate drainage into a collection container. A perforated PVC support plate is positioned inside the cell, one inch above the base plate and covered with six layers of nylon mesh. A nylon adapter is threaded into the side of the cell between the support plate and the base plate and a length of tubing was connected from the side adapter to the humidifier to facilitate the inflow of humid air to the cell. A dry air line is also connected to each cell. Each cell is covered with a removable acrylic lid.	Approximately 16 cells per humidifier Flood leaching: peristaltic pump using a peristaltic pump Temperature: 20 ± 2°C. Feed air rate to be determined.

11. Procedure

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.1	Cell Loading:		
11.1.1	If more than one humidity cell is used at one time, label each with a sequential number, and use the same number for the matching collection vessel (Erlenmeyer flask).	ASTM	
11.1.2	Weigh each humidity cell (without its lid) and each collection vessel; record the tare weights of each to the nearest 0.1 g.	ASTM	
11.1.3	Cut the filter media (such as 12-oz/yd^2 polypropylene described in 6.11) to the humidity cell's inside diameter dimensions so that it fits snugly yet lies flat on the perforated support.		Shark Skin filter paper (320mm)
11.1.4	Re-weigh the humidity cell, and record the resulting tare to the nearest 0.1 g; the original cell tare (11.1.2) minus the new cell tare is the weight of the filter media.	ASTM	
11.1.5	Transfer the contents from each of the four bags containing the 250-g samples (9.6) into the humidity cell. Prior to the transfer, mix the contents of each bag by gentle rolling to eliminate possible stratification that may have occurred during sample storage.	ASTM	
11.1.6	Re-weigh the loaded cell, and record the weight to the nearest 0.1 g; the loaded cell weight minus the combined cell and filter-media tare weight is the weight of the sample charge.	ASTM	
11.2	First Leach:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.2.1	The first leach (whether drip trickle or flooded), designated as the Week 0 leach, initiates the 20-week long humidity cell test and establishes the starting or initial characteristics of the leachate. Either a 500-mL or 1-L volume of water may be used for the weekly leachates, depending on the weekly pore volume desired or the quantity of solution required for analytical purposes; however, once a weekly volume has been selected, that weekly volume must remain constant throughout the 20-week testing period. A centrifuged cell culture of *Thiobacillus ferrooxidans* may be used in the first leach in order to ensure that optimum conditions for accelerates weathering are present at the beginning of the test.	500 mL Flood Leach	
Note 9	In the testing of mining wastes, cation (including metals and trace metals) and anion loadings are commonly high in the Week 0 leachate due to the dissolution of pre-existing soluble oxidation salts present in the sample prior to sample collection. The average number of weekly accelerated weathering cycles required to flush these pre-existing salts ranges from 3 to 5 weeks. Oxidation products observed during these 3 to 5 weeks are principally from the pre-existing salts, while those products observed after this period are considered to be solely a function of the accelerated weathering procedure. A method for estimating the amount of pre-existing oxidation salts present in a solid material sample is described by Sobek, et al (6). A comparison of estimated salt storage data obtained using this method with the first thee weeks of humidity cell effluent loadings from three different samples is describes by White and Jeffers (7).	NA	
11.2.2	Fill a separatory funnel with for each cell with de-ionized water using a volumetric flask. If the leach is to be performed using the drip trickle method, set each separatory funnel above its corresponding cell, and adjust the drip rate (approximately 3 to 4 L/min) so that the solid material sample is wetted thoroughly but not flooded.	NA	
11.2.3	A minimum of 2 to 3 h is commonly required to complete the drip trickle leach.	NA	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.2.4	If the leach is to be performed by flooding, the separatory funnel can be used to meter the selected water volume slowly down the sides of the cell wall until the sample is flooded. This application method reduces hydraulic agitation of the sample surface commonly caused by pouring liquid from an open-mouthed vessel. Alternatively, flooding may be accomplished by any application apparatus (for example, a peristaltic pump) that supplies the selected volume of leachant at a reasonable rate without causing agitation and suspension of the finer fractions contained in the sample charge.	ASTM	
11.2.4.1	Allow the flooded cell to sit for a period of 1 h before draining the leachate into the Erlenmeyer collection flask. The 1-h leach time commences after all of the leachant has been placed in the cell. The solid material sample should be saturated and covered with leachant to a depth sufficient to maintain sample saturation. In testing mining wastes, the observed depth of leachant cover from a 500-mL flooded leach performed in 10.2-cm (4.0-in.) ID cells is approximately 2.5 cm (1.0 in.).	ASTM	
11.2.5	The following is performed once the leaching process has been completed: to reduce the effects of evaporation, and to prevent the contamination of each cell by airborne contaminants, place the lids on their corresponding cells and let the cells complete the leachate draining process for the remainder of the leaching day and overnight.	ASTM	
11.2.6	Disconnect the cells on the day following the leach, and weigh and record the weight of each cell and Erlenmeyer collection flask. Set each filled collection flask aside for leachate analyses. (Measurements of pH and Eh and sample preservation procedures must be performed as soon as possible after leachate collection.) Return each cell, replace the filled collection flasks with clean, tared Erlenmeyer flasks, hook up all connections, and begin the dry-air cycle.	ASTM	
11.3	Dry-Air Cycle:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.3.1	The commencement of the three-day dry-air period marks the beginning of each new weekly cycle of the accelerated weathering humidity cell test; the first full-week cycle after the first leaching is designated Week 1; subsequent weeks (commencing with the second dry-air period) are designated as Week 2, Week 3 … . Week n, etc.	ASTM	
11.3.2	To perform the dry-air cycle, feed air is metered to the humidity cell array with a flowmeter (see 6.3) set at a target rate in the range of 1 to 10 L/min per cell, depending on the objectives of the testing. The air flow rate must be checked daily and adjusted to the target value ± 0.5 L/min.	ASTM	
11.3.3	Feed air from the flowmeter is routed first through a desiccant column and then to each of the sells through a dry-air manifold. Air exiting the desiccant column should have a relative humidity of less than 10 % as measured with a hygrometer (see 6.23).	ASTM	
11.3.4	To maintain similar positive air pressure through the cells, attach a water-bubbling vessel to each humidity cell air exit port coming out of the humidity cell lid; a 50-mL Erlenmeyer flask with a rubber stopper containing a vent and air inlet tube serves as a simple and efficient bubbler.	ASTM	
11.3.5	The dry air is passed through each humidity cell for three days. Air flow rates from each of the cells should be checked each day, recorded, and adjusted, if necessary. See also Note 10.	ASTM	
11.4	Wet-Air Cycle:		
11.4.1	The three-day wet-air period commences on the fourth day of each weekly cycle.	ASTM	
11.4.2	To perform the wet-air cycle of the method, feed air is routed through a water-filled humidifier via aeration stones or gas dispersion fritted cylinders/disks and then to each humidity cell.	ASTM	
11.4.3	The water temperature in the humidifier is maintained at 30 ± 2°C to ensure that the sparged air maintains a relative humidity of approximately 95 % as measured with a hygrometer (see 6.23) from one of the humidifier exit lines. Air flow rates to each of the cells should be checked each day, recorded, and adjusted, if necessary.	ASTM	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
Note 10	It is good practice to measure the air flow rates and relative humidity of the air exiting each humidity cell during each day of the three-day dry- and wet-air periods; the measurements should be taken at the same time each day from the humidity cell air exit port; these measurements can be accomplished by installing a quick-disconnect fitting in the tubing that connects the air exit port to the bubbler.	NA	
Note 11	Coals spoils in eastern states are commonly saturated; Caruccio (10) has suggested the following geographic control alternative to the dry-air versus saturated-air scheduling: (1) Eastern States Samples – Six days of saturated air (versus three days dry/three days wet); and (2) Western States Samples – Three days dry/three days wet.	NA	
11.5	Subsequent Weekly Leaches:		
11.5.1	A second leach with water is performed on the day following the end of the three-day wet-air period (that is, day seven of the first weekly cycle). This leach marks the end of the first weekly cycle and is designated as the Week 1 leach.	ASTM	
11.5.2	Subsequent leaches are designates as Week2, Week 3 … Week n, and they mark the end of the weekly cycle for that numbered week. Perform each weekly leach as described in 11.2.2 – 11.2.5. Weekly weighing of the test cells is optional.	ASTM	No weekly weighing of the cells.
11.6	It is recommended that the weekly accelerated weathering cycles described in 11.2, 11.3, 11.4 and 11.5 be performed for a minimum of 20 weeks.	ASTM	
Note 12	Additional weeks of accelerated weathering may be required to demonstrate the nature of the material, depending on the chemical composition of the solid material. For some metal mining wastes, researchers have shown that as much as 60 to 120 weeks of accelerated weathering data may be required to demonstrate the complete weathering characteristics of a particular sample (7, 12). The criteria for ending the testing may be site specific and should be agreed before initiating the testing.	ASTM	
11.7	Leachate Analyses:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.7.1	Analyze the leachates for specific constituents or properties, or use them for biological testing procedures as desired, using (1) appropriate ASTM test methods or (2) methods accepted for the site where disposal will occur. Where no appropriate ASTM test method exists, other test methods may be used and recorded in the report, provided that they are sufficiently sensitive to assess potential water quality impacts at the proposed disposal site. Suggested minimum weekly analyses should include pH, Eh, conductivity, and selected metals could be analyzed less frequently (for example, at Weeks 0, 1, 2, 4, 8, 12, 16, and 20), especially if changes in leachate chemistry are slow. Whether visible phase separation during storage of the leachates occurs or not, appropriate mixing should be used to ensure the homogeneity of the leachates prior to their use in such analyses.	At the end of weekly cycle the volume of leachate collected is recorded. The leachate is filtered through a Gelman magnetic filter funnel fitted with a membrane filter with pore size of 0.45 microns and analyzed for the parameters listed in Table 2 of the RFP. Filtered leachate samples will be submitted to ALS Environmental/Cantest Ltd. for dissolved metals analysis as requested in Table 4 of the Waste Rock and Lean Ore Geochemical Characterization Plan. Conductivity, Eh, and pH are measured in the CEMI laboratory using standard procedures. An aliquot of filtered leachate is titrated with standardized sulphuric acid to pH 4.5 to calculate total alkalinity. Standardized sodium hydroxide is used to titrate an aliquot of leachate to pH 4.5 and to pH 8.3 to calculate total acidity. Analysis frequency: pH, cond, Eh every cycle; SO4, Cl, F, alkalinity, TIC, acidity cycle 0, 2, 4, 6 etc.; ICP-MS including Hg and Si cycle 0, 4, 8, 12, etc., ICP-ES including Si cycle 2, 6, 10, 14, etc.	
11.7.2	Table 1 is an example of a spreadsheet format used for recording 20 weeks of leachate analytical data.	ASTM	
11.7.3	Fig. 5 is an example of a method used to plot the temporal variation (by week) of leachate pH, sulfate load, and cumulative sulfate load from 21 weeks of accelerated load and release rates).	ASTM	
11.8	Weathered Solid Material Analyses:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.8.1	Weigh the humidity cell after collection of the final effluent and completion of a three-day dry-air period.	ASTM	
11.8.2	Transfer the weathered residue and filter media to a clean drying pan, and dry to constant weight at 50 ± 5°C. Record the final weight.	ASTM	
Note 13	Perform any gross sample examination (for example, sample texture and weathering product mineralogic characterization) desired for the weathered residues prior to pulverization. To facilitate such an examination, empty the humidity cell contains into a clean drying pan carefully by pushing gently on the bottom of the perforated plate with a wooden dowel until the sample exits the cell mouth. The perforate plate is accessed through the humidity cell drain port.	NA	
11.8.3	Identify and mark the top versus bottom portions of the sample for gross sampling purposes. Formations of cemented lumps of sample termed “ferricrete” that result from the accelerated weathering process arte common in iron-sulfide-mineral rich samples. Depending on the sample mineralogy, the degree of “ferricrete” cementation may vary vertically within the sample, and the investigator may wish to segregate the sample into upper, middle, and lower thirds to document and characterize such changes.	Procedure to be determined	
11.8.4	After drying to constant weight and prior to splitting, use an instrument such as a rolling pin to break up cemented lumps in the sample (if the cemented lumps cannot be sufficiently reduced to pass through the chutes of a riffle splitter, remove, record, and weigh separately):	ASTM	
11.8.4.1	Split the sample into halves using a riffle splitter with 2.54-cm (1-in.) chutes, and reserve one half to determine the particle size distribution in accordance with Test Method E 276.		Repeat same screen assay method as for pre-test characterization (s.9.5)
11.8.4.2	Split the remaining half sample into two quarters using a riffle splitter with 2.54-cm (1-in.) chutes, and submit one quarter for mineralogical characterization; pulverize the other quarter in either a ring-and-puck or disk-pulverizing machine to 95 % passing a 150-µm (100-mesh) screen in accordance with Test Method E 276.	Procedure to be determined	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.8.5	Mix the pulverized residue in a blender or on a rolling cloth. Use the prepared residue for chemical characterization and for comparison with the pre-weathered solid material sample.	Procedure to be determined	

Appendix E

Design of MDNR Reactor

Day, Stephen

From: Kim Lapakko [kim.lapakko@dnr.state.mn.us]
Sent: Tuesday, May 17, 2005 9:55 AM
To: Stephen Day
Cc: Dave Antonson; Jennifer Engstrom; Paul Eger
Subject: RE: Small reactor

Attachments: MN DNR psize methods 050517.doc

MN DNR psize methods 050517.do...

Steve,

Attached is a description of the reactors, masses, and rinse volumes used for various size fractions of Duluth Complex rock in our particle size experiment. As indicated in the attachment, I won't have access to the trace metal data from that experiment until tomorrow. I will need to examine this to help evaluate the expected metal concentrations in drainage relative to detection limits. I'm not sure it will give us as much as hoped because the sulfur contents of the samples typically were on the order of 0.9% to 1.3%. This may make extrapolation by more than an order of magnitude tenuous. It will be another pertinent piece of information.

Kim

>>> "Stephen Day" <sday@srk.com> 5/17/2005 11:18:50 AM >>>
Dave

A design drawing should be fine along with description of the procedure.

The main question is what do you do to scale-up the sample mass as the particle size increases? I want to copy your procedure exactly.

Thanks
Steve.

-----Original Message-----
From: Kim Lapakko [mailto:kim.lapakko@dnr.state.mn.us]
Sent: Tuesday, May 17, 2005 8:38 AM
To: Stephen Day
Cc: Dave Antonson
Subject: Small reactor

Steve,

Dave Antonson will email a figure depicting our small reactor, along with some design details (perforated plate, adehesive, filter). He could also send a reactor. Please contact him directly, with an address to send it, if you think that would be helpful.

Kim

17 May 2005

Steve,

In our particle size tests we used a small reactor and 75-g mass for particle sizes of –270, +270/-100, and +100/-35 mesh. We used the ASTM cell and 1000-g mass for +35/-10, +10/-0.25 inch, and +0.25/-0.75 inch particle sizes. For rinse volumes, we used 200 mL for the 75-g samples and 300 mL for the 1000-g samples. The 300-mL rinse volume was determined as the quantity of water, rounded up to the nearest 100 mL, required to submerge the solids.

I won't have access to the metal release data for the particle size experiment until tomorrow. As mentioned on the phone, sulfate release rates appear to vary linearly with surface area. It seems likely that nickel release rates will vary similarly, and I'll look into this further tomorrow. Hopefully this information will shed some light on the maximum particle size question.

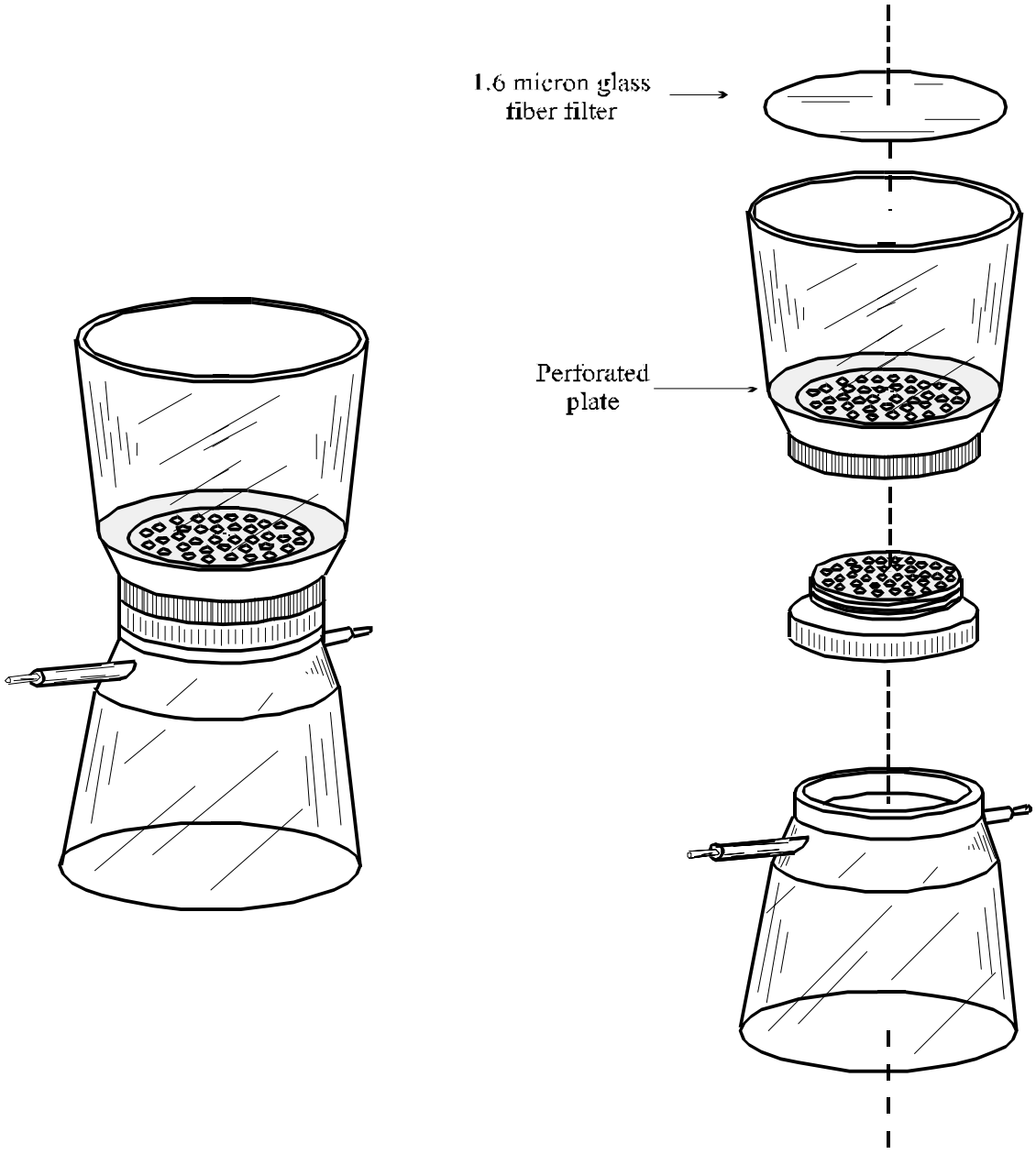
Day, Stephen

From: Dave Antonson [dave.antonson@dnr.state.mn.us]
Sent: Tuesday, May 17, 2005 11:53 AM
To: Kim Lapakko
Subject: reactor

Attachments: small reactor.doc

small reactor.doc
(271 KB)

see if this makes any sense. you can edit it if you want. if it seems adequate you can forward it to steve. maybe he doesn't need a sample of the base.


1.6 micron glass
fiber filter
Perforated
plate

The reactors were purchased from Millipore Corporation (1-800-645-5476). They are 47 mm Sterifil aseptic systems. You will need the 250 ml receiver flask, 250 ml funnel (top), silicone o-rings, and the filter holder base and support screen.

The perforated acrylic plastic base was purchased as flat stock and fabricated to fit the top funnel. The plates are 1/8" thick, 2 1/4" in diameter and tapered to fit into the reactor top. Approximately sixteen 1/16" holes were drilled in the plate. The plate was glued into the reactor using acrylic solvent cement purchased from United States Plastics (1-800-537-9724). Catalog # 44629 for 5 oz. tube. The acrylic flat stock was also purchased from United States Plastics.

After the plate is glued into the top of the reactor there should be approximatly a 3/8" gap between the bottom of the perforated plate and the top of the support screen of the filter unit.

The filter that rests on the perforated plate is a 55 mm Whatman GF/A glass microfibre filter (catalog # 1820 055).

Note: Before adding the solids to the filter you should wet the filter slightly with distilled water so no solids escape around the filter.

Appendix F

Design of Column Testing on Interaction between NorthMet and LTV Tailings

SRK Consulting (Canada) Inc.
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Memo

To:	Jennifer Engstrom, MDNR	**Date:**	December 8, 2005
cc:	Kim Lapakko, MDNR Paul Eger, MDNR	**From:**	Stephen Day
Subject:	Design of Column Testing on Interaction Between NorthMet and LTV Tailings	**Project #:**	1UP005.001

Jennifer

During our conference call on Oct 25, 2005, I agreed to provide design details for column testing to evaluate the effect of contact of leach waters from the NorthMet Tailings with LTV tailings when we received characterization data for the samples obtained from the LTV impoundment in September. Complete results have now been received.

This memorandum describes:

- Characteristics of LTV tailings.
- Characteristics of available tailings sample material.
- Proposed testwork.

The original proposed program was described in a memorandum dated September 23, 2005. SRK is seeking comments from MDNR on this proposal. It is acknowledged, as we discussed during conference call that this column testwork may represent a preliminary assessment. Additional testing may be required as the design for the tailings basin progresses.

1 Characteristics of LTV Tailings

1.1 Sample Analysis

Seven holes were drilled in the tailings to a depth exceeding 60 feet using a geoprobe. Samples have been analyzed from five holes with tailings having the following textural characteristics:

- GP-1 – Mainly coarse sand.
- GP-2 – Interlayered fine sand and slimes.
- GP-3 – Coarse sand grading into fine sand and slimes.
- GP-4 – Interlayered coarse and fine sands.
- GP-5 – Interlayered fine sand and slimes.

Samples were obtained as core and shipped whole to Canadian Environmental and Metallurgical Inc. At the laboratory, all discrete textural layers were tested for rate of HCl reaction (ie a “fizz” test) and qualitative magnetism as an indicator of magnetite content. Samples were selected from each hole to to represent the surface material (ie potentially weathered) and two samples of each textural type from each hole. These samples were submitted for relative density, moisture content and particle size determinations, quantitative mineralogy by x-ray diffraction and chemical analysis.

1.2 Results

Table 1 shows selected data sorted by the main textural groups (coarse sand, fine sand and slimes).

The dominant mineral in all samples was quartz which varied from 58 to 79% (by weight) but was not different in the three textural groups. Hematite and magnetite were present as expected. Magnetite was lower in the slimes samples likely resulting from density segregation as the tailings were deposited. Carbonates were a significant mineralogical component varying from 5 to 14%. Total carbonate content was greater in the slimes fraction compared to the coarse sands. Ankerite and siderite dominated and occurred in about equal amounts. The calcite content was lower than either ankerite or siderite.

Pyrite was detected in most samples but at very low levels. The sulphur content of the samples varied from 0.02 to 0.04% equivalent to pyrite content of 0.04 to 0.08%.

Silicates occurring in all samples were hydrobioitite, kaolinite, amphibole (cummingtonite ± grunerite), diopside, ferripyrophyllite (possibly minnesotaite) and albite. Other minerals occurring in a few samples were pyrophyllite, muscovite and hydroxylapatite. There was no evidence that the mineral distribution was related to particle size.

Distribution of metals was also unrelated to particle size with the possible exception of manganese which appeared to be elevated in the slimes. This is consistent with the higher carbonate content and indicates that manganese is associated with the carbonates.

2 Characteristics of PolyMet Tailings

Four tailings samples have been tested. These samples originated from processing of three ore composites nominally containing 0.35%, 0.4% and 0.45% copper (Parcels 1, 2 and 3, respectively). Flotation testing also considered addition of copper sulphate as an activator of pyrite. For Parcel 1 and 3, copper sulphate was evaluated. Also for Parcel 1, and Parcel 2 processing without the addition of copper sulphate was evaluated. The addition of copper sulphate does not materially affect the copper and sulphur content of the tailings directly but changes the mineralogical composition due to the selective recovery of pyrite to the bulk sulfide concentrate.

The characteristics of the four samples are shown in Table 2. Very little difference between the tailings was apparent. As expected, addition of copper sulphate in the process resulted in lower sulphur content in the tailings (0.1% and 0.15%, Parcels 1 and 3, respectively) compared to 0.23% and 0.2% (Parcels 1 and 2, respectively) without copper sulphate. Copper showed the widest variation in concentration (223 to 527 mg/kg). Variations of cobalt, nickel and zinc were small.

Table 1. Characteristics of LTV Tailings Samples

Core	Sampled		Material	Q	Py	Cal	Ank	Sid	Hem	Mag	Bio	Kao	Fe-Pyr	Albite low	As	Cd	Co	Cr	Cu	Fe	Mn	Ni	P	Pb	S	Zn
	Start ft	Finish ft		%	%	%	%	%	%	%	%	%	%	%	%	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	ppm
Coarse Sand																										
GP-1	8	12	Coarse Sand	72	0.2	0.3	3	3	3	4	1	1	2	2	31	-0.5	12	100	14	15.85	5970	5	240	7	0.02	14
GP-1	20	40	Coarse Sand	71	0.2	0.2	5	8	2	2	2	1	2	2	36	-0.5	14	90	25	15.5	7110	8	250	5	0.04	13
GP-3	8	12	Coarse Sand	58	0	0.4	6	6	2	3	5	1	6	4	16	-0.5	7	42	7	13.7	3420	3	250	4	0.02	9
GP-4	4	16	Coarse Sand	59	0.1	0.7	2	4	2	3	10	2	2	3	19	-0.5	7	57	9	15.45	3890	1	240	7	0.02	9
GP-4	20	24	Coarse Sand	79	0	0.1	3	2	2	3	3	1	2	1	21	-0.5	10	45	8	12.85	4010	3	250	5	0.02	34
n				5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Min				58	0	0.1	2	2	2	2	1	1	2	1	16	-0.5	7	42	7	12.85	3420	1	240	4	0.02	9
Median				71	0.1	0.3	3	4	2	3	3	1	2	2	21	-0.5	10	57	9	15.45	4010	3	250	5	0.02	13
Max				79	0.2	0.7	6	8	3	4	10	2	6	4	36	-0.5	14	100	25	15.85	7110	8	250	7	0.04	34
Fine Sand																										
GP-1	60	72	Fine Sand	72	0.1	0.7	3	2	1	1	2	2	2	4	22	-0.5	11	77	20	12.05	7010	3	330	3	0.04	14
GP-2	0	1	Fine Sand	60	0	0.2	6	7	2	2	11	4	3	0	15	-0.5	9	27	7	14.4	4270	-1	490	-2	0.02	10
GP-2	24	28	Fine Sand	62	0.4	1	6	3	1	2	5	2	3	3	22	-0.5	11	61	13	14.55	5340	3	550	7	0.02	14
GP-3	44	60	Fine Sand	68	0.2	0.5	4	3	3	2	4	1	3	1	15	-0.5	12	45	17	13.4	8510	4	400	7	0.03	12
GP-4	18	20	Fine Sand	73	0	0.2	7	3	2	2	4	2	2	1	43	-0.5	10	41	12	13.3	4020	2	290	8	0.02	13
GP-5	8	20	Fine Sand	78	0.1	0.1	4	2	3	3	3	1	2	1	14	-0.5	7	42	7	13.45	3630	2	270	2	0.02	7
GP-5	36	48	Fine Sand	73	0.4	0.4	4	2	2	3	2	1	3	2	14	-0.5	9	53	14	13.55	5820	2	290	4	0.02	8
n				7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7
Min				60	0	0.1	3	2	1	1	2	1	2	0	14	-0.5	7	27	7	12.05	3630	-1	270	-2	0.02	7
Median				72	0.1	0.4	4	3	2	2	4	2	3	1	15	-0.5	10	45	13	13.45	5340	2	330	4	0.02	12
Max				78	0.4	1	7	7	3	3	11	4	3	4	43	-0.5	12	77	20	14.55	8510	4	550	8	0.04	14
Slimes																										
GP-2	28	32	Slimes	62	0	1	4	4	2	1	5	2	3	5	18	-0.5	14	31	10	14.2	7390	3	530	4	0.02	13
GP-3	60	72	Slimes	62	0	0.7	8	5	3	1	7	2	2	1	15	-0.5	14	33	20	12	10050	4	590	2	0.04	14
GP-5	20	24	Slimes	70	0.1	0.3	6	4	3	2	6	2	3	1	25	-0.5	9	38	7	13.75	4830	1	460	4	0.02	8
GP-5	48	52	Slimes	72	0	0.6	4	6	2	1	4	1	2	1	16	-0.5	16	40	19	13.1	12400	3	550	4	0.03	11
n				4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Min				62	0	0.3	4	4	2	1	4	1	2	1	15	-0.5	9	31	7	12	4830	1	460	2	0.02	8
Median				66	0	0.65	5	4.5	2.5	1	5.5	2	2.5	1	17	-0.5	14	35.5	14.5	13.425	8720	3	540	4	0.025	12
Max				72	0.1	1	8	6	3	2	7	2	3	5	25	-0.5	16	40	20	14.2	12400	4	590	4	0.04	14

Notes
Minerals Q = Quartz, Ank = Ankerite, Hem = Hematite, Mag = Magnetite, Sid = siderite, Bio = hydrobiotite, kao = kaolinite, Fe-Pyr = Ferriprophyllite, Cal = Calcite, py =pyrite

Table 2. Characteristics of NorthMet Tailings Samples

Sample Number	Source and Process	Total S %	As ppm	Cd ppm	Co ppm	Cr ppm	Cu ppm	Mn ppm	Ni ppm	P ppm	Zn ppm
PIS-ACS TO DSC COMP	Parcel 1, $CuSO_4$ Added	0.10	<5	<0.5	59	186	223	1130	319	790	95
PISA TO D COMP	Parcel 1, $CuSO_4$ Not Added	0.23	<5	<0.5	62	188	248	1140	329	800	97
P2SA TO D COMP	Parcel 2, $CuSO_4$ Not Added	0.20	<5	<0.5	61	199	527	1125	385	750	93
P3S-A TO D COMP	Parcel 3, $CuSO_4$ Added	0.15	<5	<0.5	60	175	418	1110	372	770	92

3 Proposed Column Testwork

3.1 Design Basis

Based on the characteristics of the NorthMet and LTV tailings, the following chemical processes can be expected to occur within the layered tailings basins:

- In NorthMet Tailings
  - Near surface oxidation of residual sulfide minerals resulting in release acidity, iron, sulphate and trace elements (copper and nickel).
  - Development and migration of an oxidation front due to consumption of oxygen near the surface.
  - Attenuation of metals as a result of interaction between pore fluids and mineral grains.
- LTV Tailings
  - Enhanced dissolution of ankerite and siderite under saturated conditions resulting in release of calcium, magnesium, ferrous iron, reduced manganese and bicarbonate alkalinity.
  - Localized re-precipitation of ferric hydroxides and manganese oxides due to variations in pH and oxidation-reduction potential.
- Interaction Between NorthMet Tailings Pore Water and LTV Tailings
  - Possible sorption of metals by ferric hydroxides and manganese oxides, particularly in the immediate contact zone where LTV tailing are probably partially oxidized.
  - Precipitation of metal carbonates due to alkaline conditions.

Testing of the LTV tailings indicates little significant variation in mineralogical and chemical content. The primary variable expected to influence the degree to metal attenuation occurs, if at all, is particle size which will control the availability of adsorption sites, oxidation-reduction conditions and contact time.

3.2 Proposed Testwork

The proposed leach column design includes the following main features (Figure 1):

- In series leach columns designed to generate NorthMet tailings pore water as feed into LTV tailings.

- First column containing 10 kg of unsaturated NorthMet Tailings and second receiving column containing 5 kg of LTV tailings. The larger volume of NorthMet tailings is intended to optimize development of near equilibrium pore water chemistry below an oxidation zone in the NorthMet tailings. The smaller volume of LTV tailings may allow breakthrough of NorthMet tailings water chemistry to be observed.
- NorthMet tailings open to atmosphere.
- Connecting pipe between columns and sampling "T" operated to exclude oxygen.
- Side sampling port in LTV tailings column to enable sampling of pore water just below the entry point.
- Sampling of final effluent.
- Application of 2 L of deionized water every week to allow withdrawal of up to 250 mL of water from each of the two intermediate location and 1.5 L of the final effluent. This application rate represents approximately one pore volume every 4 weeks.
- Analysis of intermediate sampling points for pH and Eh every week and composite sample for anions and cations every other week (including sulfur).
- Analysis of final effluent for same parameters, sulphate, alkalinity and anion scan.

Approximately 11 kg of each of the tailings samples shown in Table 2 are available for additional testing. The following matrix summarises six proposed tests (Table 3). It is preferred to test slimes and coarse sands as two extreme characteristics of LTV tailings. From experience, it is unlikely that the slimes will transmit sufficient water for the experiment. Therefore, a fine sand composite would be used instead.

Each "X" in Table 3 represents an in-series column pair. The two control experiments will operate without NortMet tailings in the first column to evaluate leachate chemistry from LTV tailings. This will allow comparison with seepage chemistry in the existing basin.

Table 3. Matrix of Proposed Tests

		NorthMet Tailings Samples		
		Control No NorthMet Tailings	**Lower S Composite (Parcels 1 and 3, $CuSO_4$ Added)**	**Higher S Composite (Parcels 1 and 2, $CuSO_4$ added)**
LTV Samples	Fine sand or Slimes Composite	X	X	X
	Sand Composite	X	X	X

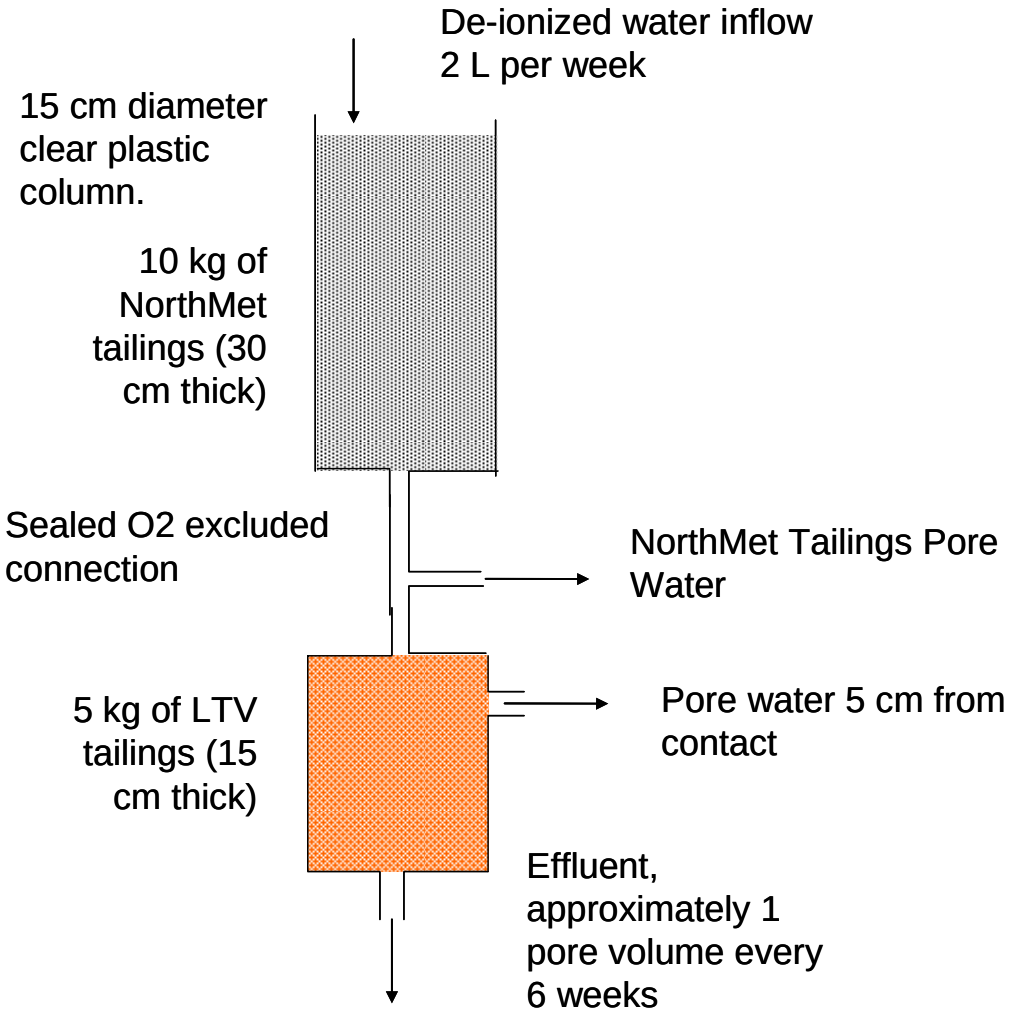


Figure 1. Schematic of In-Series Column Design

Appendix B
Results of Mineralogy, Bulk Chemical and Leach Test Characterization

			Residues							
Procedure	Parameter	Unit	Leach, no CuSO4	Leach, with CuSO4	Gypsum	Raffinate Neutralization	Fe/Al	Mg	Combined	Combined no Gypsum
XRD Mineralogy Results	Quartz	%	2.1	3	0.2	0.3	0.3	0.2	2.2	3.4
	Plagioclase	%	25.6	5.8						2.2
	Gypsum	%	8.1	6.9	99.8	96.3	98.9	76.8	73.4	53.3
	Calcite	%							2.9	
	Bassanite	%				3.4				
	Siderite	%							1.2	
	Actinolite	%	1.3							
	Talc	%	4.3	3.9					3.2	3.2
	Natrojarosite	%	34	63					9.6	29.3
	Hematite	%	24.6	17.4					6	8.6
	Goethite	%					0.8			
	Butlerite	%							1.5	
	Brucite	%						22.2		
	Halite	%						0.8		
Acid-Base Accounting	Fizz Rating	-	1	1	1	1	1	3	1	2
	Paste pH	-	2.6	3.3	3.8	5.4	4.6	9.6	-	9.4
	Total S	%	6.24	9.8	19.55	18.95	17.9	14.4	16.5	13.65
	S as SO4	%	6.1	9.3	18.35	18.55	17.35	13.65	15.9	13.5
	S as Non-SO4	%	0.18	0.17	0.59	0.55	0.57	0.29	0.43	0.18
	C	%	<0.05	<0.05	<0.05	<0.05	<0.05	0.48	<0.05	0.08
	CO2	%	<0.2	<0.2	<0.2	<0.2	<0.2	1.8	<0.2	0.3
	AP	kg CaCO3/t	195	306.3	610.9	592.2	559.4	450	515.6	426.6
	NNP	kg CaCO3/t	-212	-310	-611	-592	-569	-79	-506	-376
	NP	kg CaCO3/t	-17	-4	0	0	-10	371	10	51
	NP/AP	-	-0.09	-0.01	0	0	-0.02	0.82	0.02	0.12

Notes
1. #N/A - result checking in progress
2. Negative value indicates result is below reporting limit

			Residues							
Procedure	**Parameter**	**Unit**	**Leach, no CuSO4**	**Leach, with CuSO4**	**Gypsum**	**Raffinate Neutralization**	**Fe/Al**	**Mg**	**Combined**	**Combined no Gypsum**
Metals	**Ag**	**ppm**	11.05	23.8	0.23	0.9	0.33	0.12	4.93	11.05
	Al	**%**	2.82	1.97	0.02	0.02	0.46	0.03	0.54	1.01
	As	**ppm**	35.4	56.3	3	6	15	5	110	34
	B	**ppm**	<10	<10	<10	<10	<10	<10	<10	<10
	Ba	**ppm**	40	30	<10	<10	<10	<10	10	20
	Be	**ppm**	0.05	<0.05	<0.05	<0.05	0.28	<0.05	<0.05	<0.05
	Bi	**ppm**	4.89	6.84	0.5	0.8	0.81	0.04	1.94	3.53
	Ca	**%**	1.88	1.41	14.5	17.1	18.8	15.1	16.35	11.3
	Cd	**ppm**	0.66	0.19	0.22	0.73	0.48	0.44	0.29	0.23
	Ce	**ppm**	8.21	5.45	1.71	0.6	0.85	1.6	2.35	2.93
	Co	**ppm**	133.5	17.9	5.1	9.3	105.5	13.6	6.9	23.6
	Cr	**ppm**	112	154	8	11	457	5	59	116
	Cs	**ppm**	0.1	<0.05	<0.05	<0.05	0.09	<0.05	<0.05	<0.05
	Cu	**ppm**	7380	1280	184.5	22.5	2960	26.9	519	945
	Fe	**%**	17.3	26.9	0.05	0.08	1.88	0.05	5.21	11.75
	Ga	**ppm**	6.47	5.06	2.54	0.07	0.57	0.09	1.22	2.43
	Ge	**ppm**	0.41	0.47	<0.05	<0.05	0.08	<0.05	0.11	0.22
	Hf	**ppm**	0.12	0.06	<0.02	<0.02	0.12	<0.02	0.04	0.04
	Hg	**ppm**	0.04	0.02	0.12	0.35	0.07	0.05	0.11	0.03
	In	**ppm**	0.3	0.355	23.3	0.011	1.01	0.083	0.164	0.28
	K	**%**	0.11	0.1	<0.01	<0.01	0.01	0.01	0.02	0.05
	La	**ppm**	3.6	2.5	0.7	0.5	0.5	2.3	1.1	1.6
	Li	**ppm**	1	0.6	0.1	0.1	0.5	0.5	0.2	0.2
	Mg	**%**	0.28	0.25	0.01	0.01	0.14	9.44	0.39	1.47
	Mn	**ppm**	35	<5	<5	5	23	83	<5	<5
	Mo	**ppm**	21.8	28.4	0.47	0.85	42.3	0.38	9.35	18.9
	Na	**%**	1.28	3	0.02	0.04	0.13	1.04	0.58	1.62
	Nb	**ppm**	0.18	0.21	<0.05	0.06	0.46	0.06	0.11	0.17
	Ni	**ppm**	3270	410	99.9	192.5	2710	1230	260	674
	P	**ppm**	290	110	<10	20	160	40	70	90
	Pb	**ppm**	48.5	56.1	8.3	86.1	20.4	12.5	133.5	32
	Rb	**ppm**	1.7	1	0.2	0.3	0.5	0.3	0.4	0.7
	Re	**ppm**	0.012	0.002	0.015	0.022	0.013	<0.001	0.009	0.009
	S	**%**	5.47	9.22	>10.0	>10.0	>10.0	>10.0	>10.0	>10.0
	Sb	**ppm**	1.22	1.81	0.18	1.25	1.47	0.4	3.99	1.14
	Sc	**ppm**	1.3	0.8	0.4	0.4	4.4	0.5	0.8	1.1
	Se	**ppm**	49.8	73.4	0.3	0.4	0.6	0.5	20.4	39.2
	Sn	**ppm**	4.8	6.1	0.2	0.5	1.6	<0.2	2	3.2
	Sr	**ppm**	73.1	63.9	60.6	106	83.7	86.5	77.3	85.2
	Ta	**ppm**	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
	Te	**ppm**	2.01	2.91	0.01	0.07	0.09	<0.01	0.83	1.48
	Th	**ppm**	1.1	0.9	<0.2	<0.2	0.3	<0.2	0.3	0.4
	Ti	**%**	0.066	0.042	<0.005	<0.005	0.007	<0.005	0.012	0.019
	Tl	**ppm**	0.1	0.34	0.03	0.03	0.24	0.02	0.06	0.19
	U	**ppm**	0.06	<0.05	<0.05	<0.05	0.55	0.1	0.06	0.09
	V	**ppm**	39	41	1	1	16	1	11	21
	W	**ppm**	0.57	0.49	0.09	0.18	1.12	<0.05	0.26	0.45
	Y	**ppm**	3.88	3.29	1.28	1.14	1.06	7.64	2	2.86
	Zn	**ppm**	171	15	30	98	164	23	32	36
	Zr	**ppm**	<0.5	0.7	1.5	4.2	1.2	<0.5	9.8	1.9

Notes
1. #N/A - result checking in progress
2. Negative value indicates result is below reporting limit

			Residues							
Procedure	**Parameter**	**Unit**	**Leach, no CuSO4**	**Leach, with CuSO4**	**Gypsum**	**Raffinate Neutralization**	**Fe/Al**	**Mg**	**Combined**	**Combined no Gypsum**
SPLP Leachate Extraction Results	**pH**	**-**	2.79	3.38	4.12	4.91	5.03	9.72	9.18	9.42
	Hardness	**mgCaCO3/L**	1520	1480	1360	1290	1460	1880	1480	1650
	Al	**mg/L**	9.24	0.27	0.27	0.3	0.21	<0.005	<0.005	<0.005
	Sb	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	As	**mg/L**	<0.001	<0.001	0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Ba	**mg/L**	<0.001	<0.001	<0.001	0.009	<0.001	0.003	<0.001	<0.001
	Be	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Bi	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	B	**mg/L**	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
	Cd	**mg/L**	0.022	0.0058	<0.0002	0.0017	0.012	<0.0002	<0.0002	<0.0002
	Ca	**mg/L**	487	492	541	511	500	476	519	499
	Cr	**mg/L**	0.1	0.006	0.005	0.003	<0.001	<0.001	0.002	<0.001
	Co	**mg/L**	4.19	0.21	0.14	0.33	3.75	<0.001	<0.001	<0.001
	Cu	**mg/L**	241	6.67	5.38	0.5	13.9	0.006	0.009	0.005
	Fe	**mg/L**	59.6	0.26	0.5	0.69	<0.05	<0.05	<0.05	<0.05
	Pb	**mg/L**	0.004	0.002	0.009	0.093	0.001	<0.001	<0.001	<0.001
	Li	**mg/L**	0.023	<0.001	<0.001	<0.001	0.015	0.008	<0.001	<0.001
	Mg	**mg/L**	74.1	61.1	1.8	2.85	50.7	168	44.7	97
	Mn	**mg/L**	1.29	0.055	0.026	0.046	0.82	<0.001	<0.001	<0.001
	Hg	**ug/L**	0.13	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
	Mo	**mg/L**	<0.0005	<0.0005	0.0005	<0.0005	0.0014	<0.0005	0.0074	0.013
	Ni	**mg/L**	103	3.95	3.09	6.84	79	0.004	0.015	0.008
	PO4	**mg P/L**	<0.15	<0.15	<0.15	<0.15	<0.15	<0.15	<0.15	<0.15
	K	**mg/L**	<0.1	1.4	<0.1	<0.1	0.3	2.6	0.4	0.7
	Se	**mg/L**	0.005	0.003	0.002	<0.001	0.002	0.003	0.01	0.024
	Si	**mg SiO2/L**	0.8	0.8	<0.25	0.5	1.4	<0.25	15.8	10.5
	Ag	**mg/L**	0.0064	0.0077	<0.00025	<0.00025	0.0012	<0.00025	<0.00025	<0.00025
	Na	**mg/L**	47.2	147	2.8	4.63	41.9	238	51.4	97.6
	Sr	**mg/L**	0.26	0.32	0.16	0.3	0.19	0.24	0.18	0.26
	Te	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Tl	**mg/L**	0.0006	<0.0001	0.0002	0.0001	0.0012	<0.0001	<0.0001	<0.0001
	Th	**mg/L**	0.033	0.01	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
	Sn	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Ti	**mg/L**	0.044	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	U	**mg/L**	0.0017	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
	V	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Zn	**mg/L**	5.85	0.17	0.095	0.24	4.4	<0.005	<0.005	<0.005
	Zr	**mg/L**	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01

Notes
1. #N/A - result checking in progress
2. Negative value indicates result is below reporting limit

			Residues							
Procedure	**Parameter**	**Unit**	**Leach, no CuSO4**	**Leach, with CuSO4**	**Gypsum**	**Raffinate Neutralization**	**Fe/Al**	**Mg**	**Combined**	**Combined no Gypsum**
TCLP Leachate Extraction Results	**pH**	**-**	4.74	4.94	4.95	4.96	4.98	9.8	5.23	8.98
	Hardness	**mgCaCO3/L**	1500	1260	1490	1900	1610	3270	2400	3710
	Al	**mg/L**	2.6	0.008	0.22	0.28	1.51	<0.005	0.088	<0.005
	Sb	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	As	**mg/L**	<0.001	<0.001	0.001	0.001	<0.001	<0.001	<0.001	<0.001
	Ba	**mg/L**	<0.001	0.001	0.001	0.014	0.001	0.004	0.007	0.003
	Be	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Bi	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	B	**mg/L**	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
	Cd	**mg/L**	0.021	0.0055	<0.0002	0.0018	0.011	<0.0002	0.0025	<0.0002
	Ca	**mg/L**	451	402	594	754	562	586	733	718
	Cr	**mg/L**	0.031	0.002	0.006	0.007	0.13	0.002	0.022	0.004
	Co	**mg/L**	4.48	0.21	0.14	0.32	3.37	0.001	0.096	0.001
	Cu	**mg/L**	244	5.75	5.28	0.46	46.8	0.018	2.41	0.005
	Fe	**mg/L**	3.9	<0.05	0.42	0.55	0.06	<0.05	0.14	<0.05
	Pb	**mg/L**	0.001	<0.001	0.012	0.34	<0.001	<0.001	0.004	<0.001
	Li	**mg/L**	0.025	0.002	0.001	0.002	0.014	0.008	<0.001	0.002
	Mg	**mg/L**	91	62	2	2.74	50	438	137	464
	Mn	**mg/L**	1.39	0.057	0.029	0.046	0.76	<0.001	0.11	<0.001
	Hg	**ug/L**	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
	Mo	**mg/L**	<0.0005	<0.0005	0.0007	0.0006	0.0017	<0.0005	0.0006	0.027
	Ni	**mg/L**	107	4	3.15	6.35	73.4	0.018	3.02	0.093
	P	**mg PO4/L**	<0.15	<0.15	<0.15	1	<0.15	<0.15	<0.15	<0.15
	K	**mg/L**	0.6	2.2	0.5	0.6	0.7	3	1.2	1.7
	Se	**mg/L**	0.004	0.003	0.002	0.002	0.002	0.002	0.002	0.033
	Si	**mg SIO2/L**	3.5	4	1.1	1.5	2.7	0.8	14.9	14.1
	Ag	**mg/L**	0.0074	0.0093	<0.00025	<0.00025	0.0015	<0.00025	<0.00025	<0.00025
	Na	**mg/L**	1730	1680	1560	1600	1540	1710	1590	1710
	Sr	**mg/L**	0.29	0.35	0.22	0.46	0.29	0.41	0.34	0.47
	Te	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Tl	**mg/L**	0.0009	<0.0001	0.0002	0.0003	0.0016	<0.0001	0.0003	0.0001
	Th	**mg/L**	0.002	0.0013	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
	Sn	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	0.001	<0.001
	Ti	**mg/L**	0.009	<0.001	<0.001	<0.001	<0.001	<0.001	0.001	<0.001
	U	**mg/L**	0.0016	<0.0005	<0.0005	<0.0005	0.0069	<0.0005	0.0008	<0.0005
	V	**mg/L**	0.003	0.003	0.003	0.003	0.001	0.003	0.002	0.003
	Zn	**mg/L**	5.49	0.18	0.12	0.25	4.12	<0.005	0.22	<0.005
	Zr	**mg/L**	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01

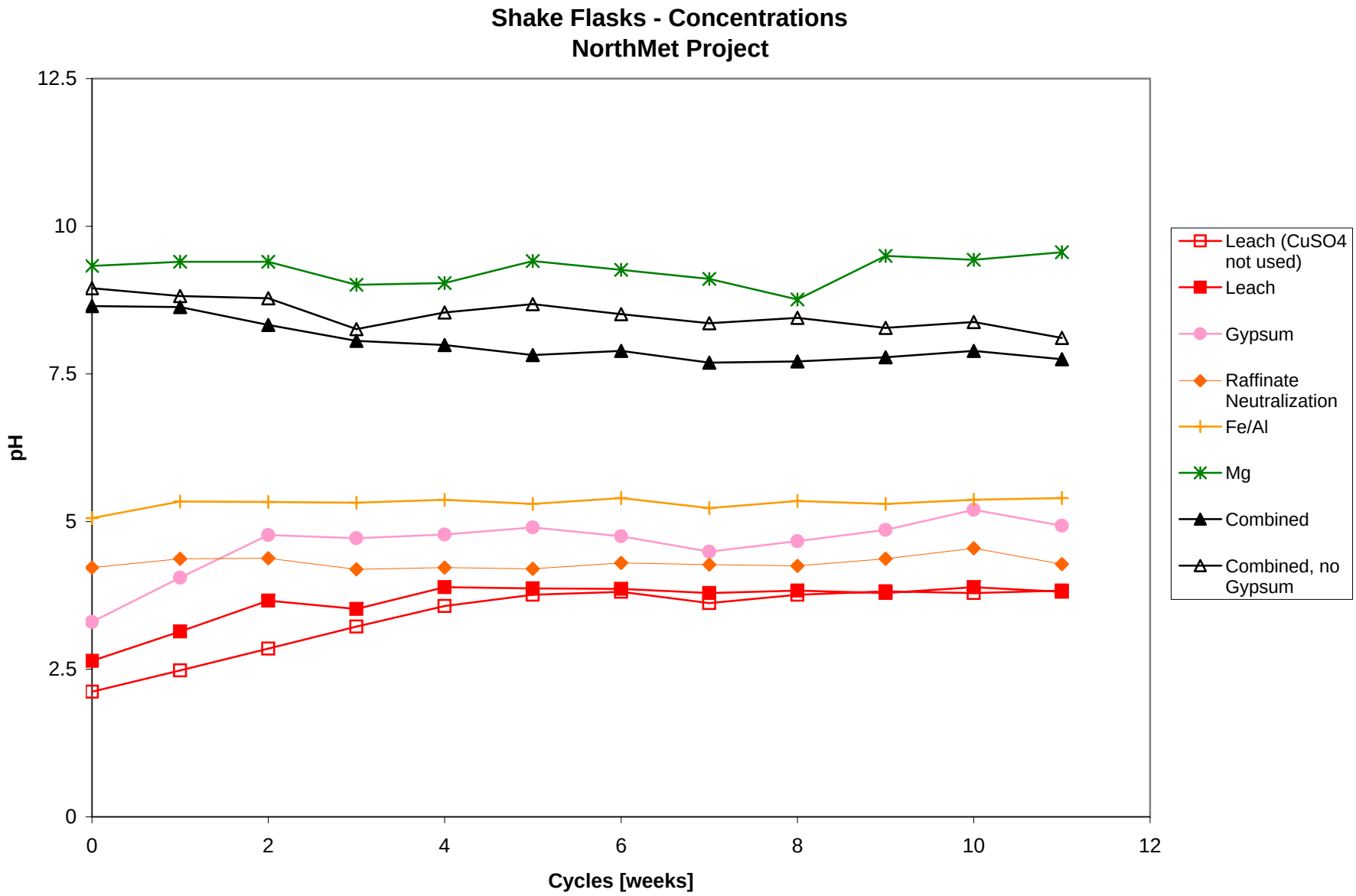
Notes
1. #N/A - result checking in progress
2. Negative value indicates result is below reporting limit

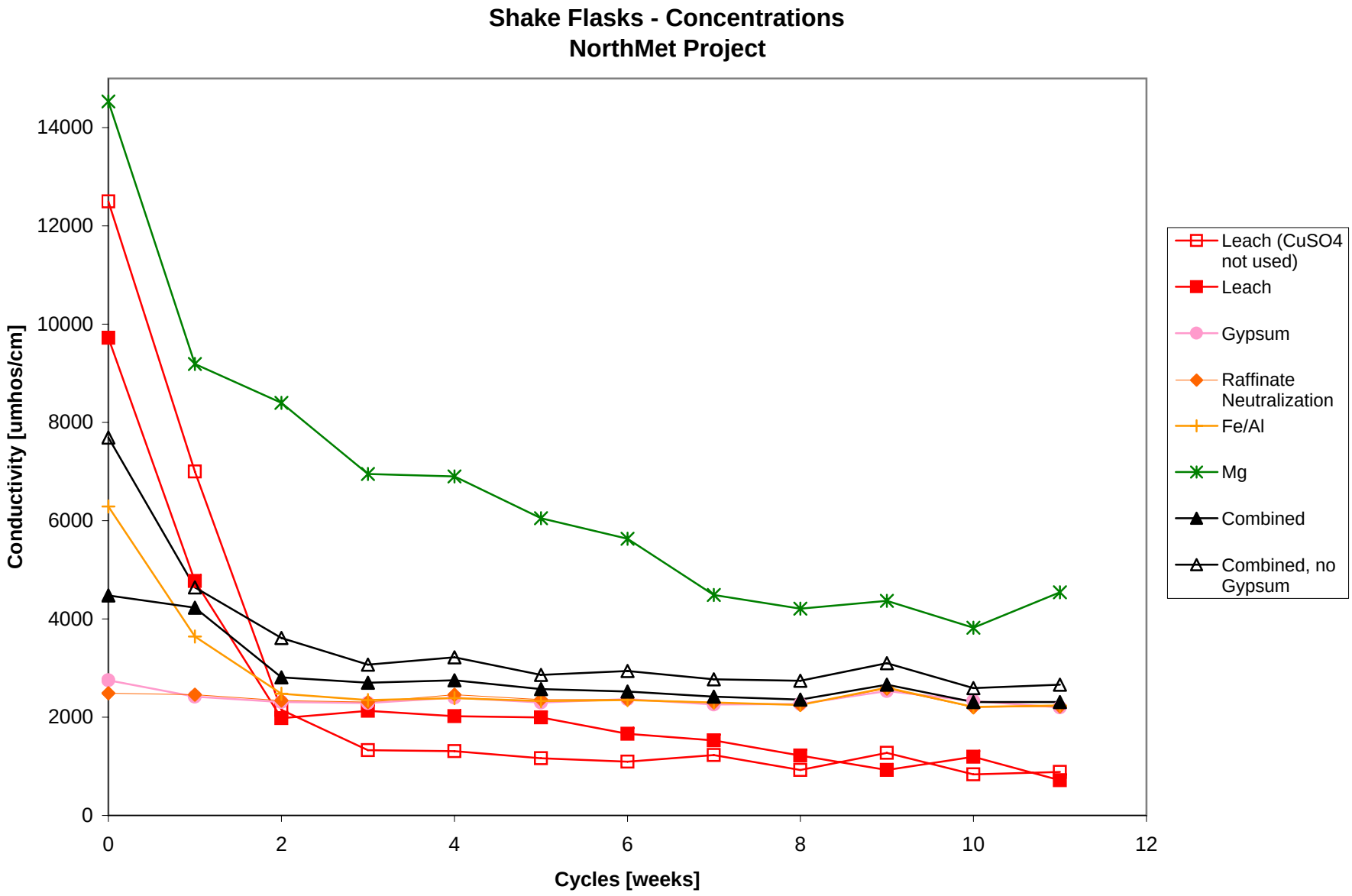
			Residues							
Procedure	**Parameter**	**Unit**	**Leach, no CuSO4**	**Leach, with CuSO4**	**Gypsum**	**Raffinate Neutralization**	**Fe/Al**	**Mg**	**Combined**	**Combined no Gypsum**
3:1 Water Leach (First Week)	**pH**	**-**	2.12	2.64	3.30	4.22	5.06	9.33	8.65	8.95
	ORP	**mV**	625	601	568	530	490	277	390	262
	Cond.	**µS/cm**	12500	9720	13	2490	6290	14530	4480	7690
	Acidity to pH 4.5	**mg CaCO3/L**	1807.5	252.0	38.0	2.25	#N/A	#N/A	#N/A	#N/A
	Total Acidity	**mg CaCO3/L**	4362.5	365.0	119.0	58.5	400.0	#N/A	#N/A	#N/A
	Alkalinity	**mg CaCO3/L**	#N/A	#N/A	#N/A	#N/A	5.0	52.0	39.5	46.0
	Inorganic C	**mg CaCO3/L**	<1	<1	<1	<1	<1	7	9	8
	Hardness	**mg CaCO3/L**	2640	2810	1360	1300	3000	6020	2000	3730
	F	**mg/L**	<1	<0.5	<0.25	3.6	1.6	<1	<5	<0.5
	Cl	**mg/L**	717	717	21.7	23.5	473	1950	392	635
	SO4	**mg/L**	9485	4507	1600	1670	4490	7940	2430	4280
	Al	**mg/L**	47.9	1.34	1.35	1.69	1.06	<0.005	<0.005	<0.005
	Sb	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	As	**mg/L**	0.012	<0.001	0.004	0.003	0.002	<0.001	<0.001	<0.001
	Ba	**mg/L**	<0.001	<0.001	0.006	0.012	0.002	0.005	0.001	0.002
	Be	**mg/L**	0.003	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Bi	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	B	**mg/L**	0.07	<0.05	<0.05	<0.05	0.11	<0.05	<0.05	<0.05
	Cd	**mg/L**	0.102	0.031	0.0035	0.0082	0.064	<0.0002	<0.0002	<0.0002
	Ca	**mg/L**	322	434	522	494	463	458	450	453
	Cr	**mg/L**	0.54	0.03	0.02	0.013	0.001	0.001	0.006	0.002
	Co	**mg/L**	30.2	0.99	0.79	1.8	29.1	0.001	<0.001	<0.001
	Cu	**mg/L**	1300	51.1	31.4	2.81	52	0.025	0.002	0.006
	Fe	**mg/L**	511	12.6	2.72	3.94	0.12	<0.05	<0.05	<0.05
	Pb	**mg/L**	0.009	0.006	0.018	0.3	0.002	<0.001	<0.001	<0.001
	Li	**mg/L**	0.14	0.005	0.003	0.005	0.096	0.048	<0.001	0.001
	Mg	**mg/L**	446	419	10.1	15.9	446	955	213	630
	Mn	**mg/L**	6.69	0.28	0.13	0.28	4.83	<0.001	<0.001	<0.001
	Hg	**mg/L**	1.18	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
	Mo	**mg/L**	<0.0005	<0.0005	0.0013	<0.0005	0.0023	<0.0005	0.027	0.03
	Ni	**mg/L**	616	19	15.7	37.9	616	0.034	0.028	0.054
	P	**mg/L**	0.7	<0.15	<0.15	<0.15	<0.15	<0.15	<0.15	<0.15
	K	**mg/L**	0.4	7.6	0.1	0.5	2.2	20.3	1.8	3.9
	Se	**mg/L**	0.01	0.003	<0.001	0.001	0.003	<0.001	0.011	0.021
	Si	**mg/L**	3.3	3.2	1.2	2.7	4.5	<0.25	14.8	5.7
	Ag	**mg/L**	0.022	0.025	0.0003	<0.00025	0.013	<0.00025	<0.00025	<0.00025
	Na	**mg/L**	290	616	16	20.9	284	2020	255	717
	Sr	**mg/L**	0.17	0.38	0.2	0.46	0.22	0.3	0.2	0.27
	Te	**mg/L**	<0.001	<0.001	0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Tl	**mg/L**	0.0029	0.0005	0.0006	0.0005	0.0049	0.0004	<0.0001	0.0001
	Th	**mg/L**	0.145	0.109	0.0007	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
	Sn	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Ti	**mg/L**	0.42	0.005	0.002	0.001	0.001	<0.001	<0.001	<0.001
	U	**mg/L**	0.0081	<0.0005	<0.0005	0.0009	0.001	<0.0005	<0.0005	<0.0005
	V	**mg/L**	0.12	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Zn	**mg/L**	48.6	0.92	0.15	1.34	34.7	<0.005	<0.005	<0.005
	Zr	**mg/L**	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01

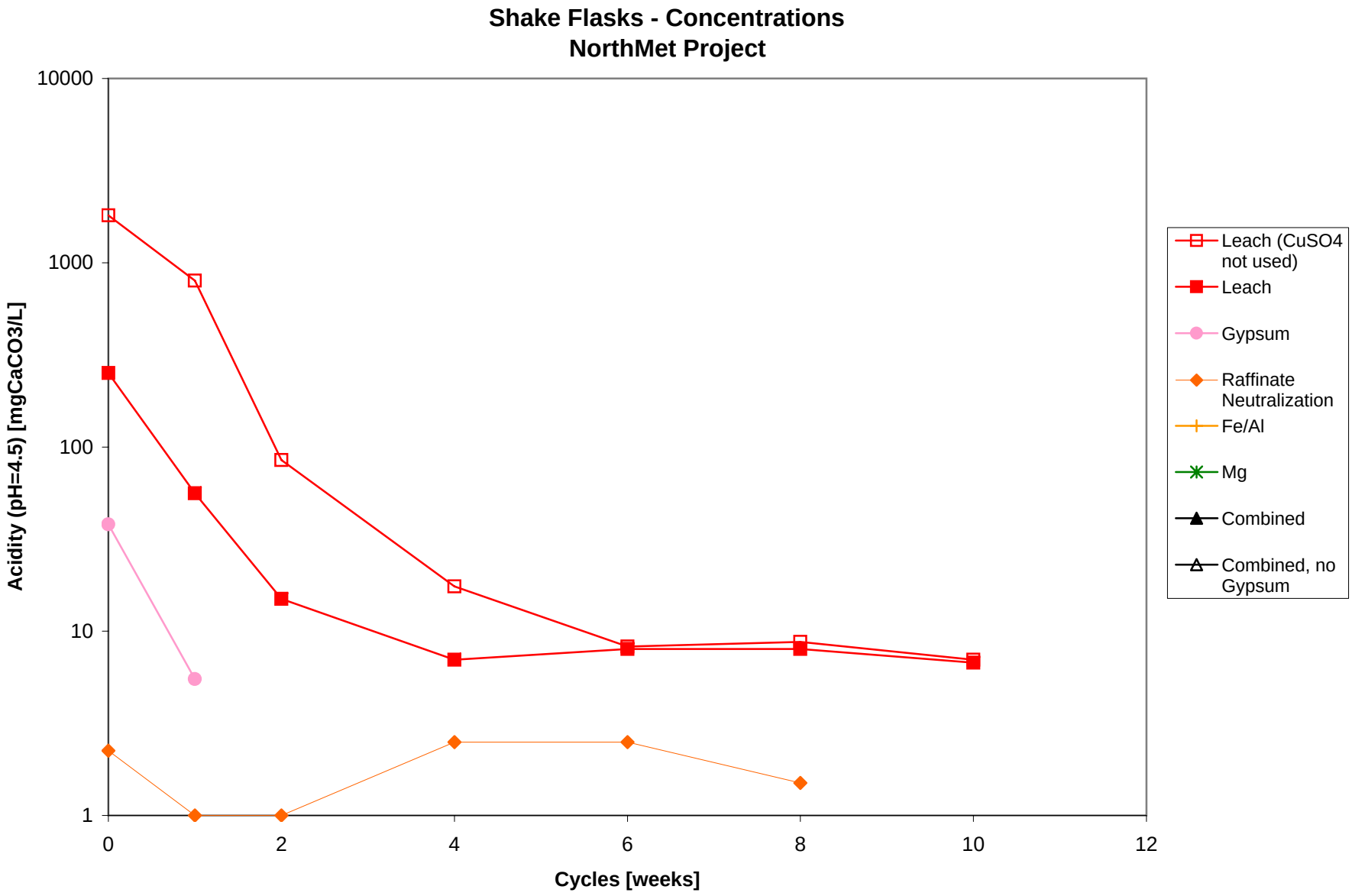
Notes
1. #N/A - result checking in progress
2. Negative value indicates result is below reporting limit

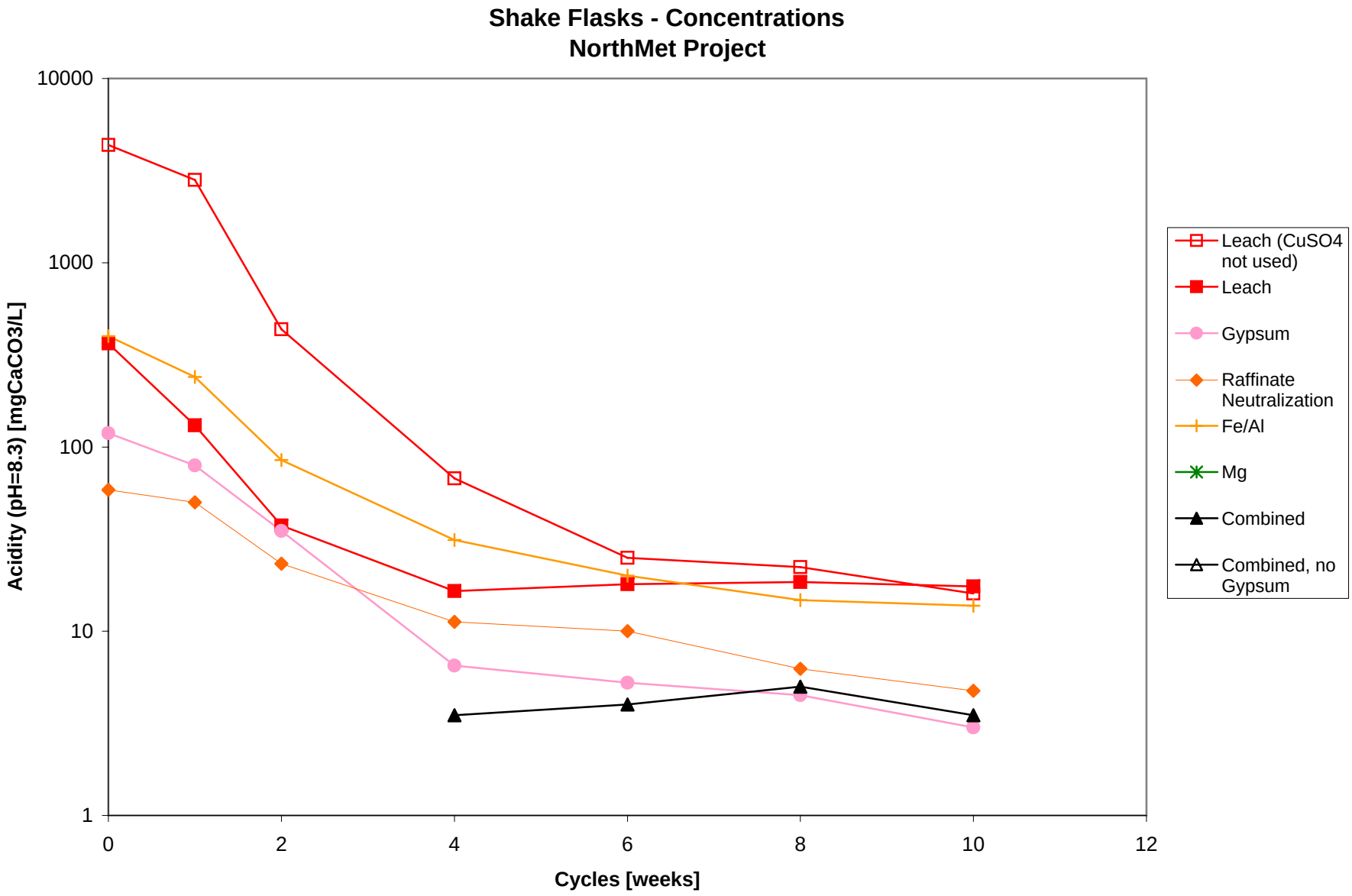
Appendix C
Dissolution Tests Charts

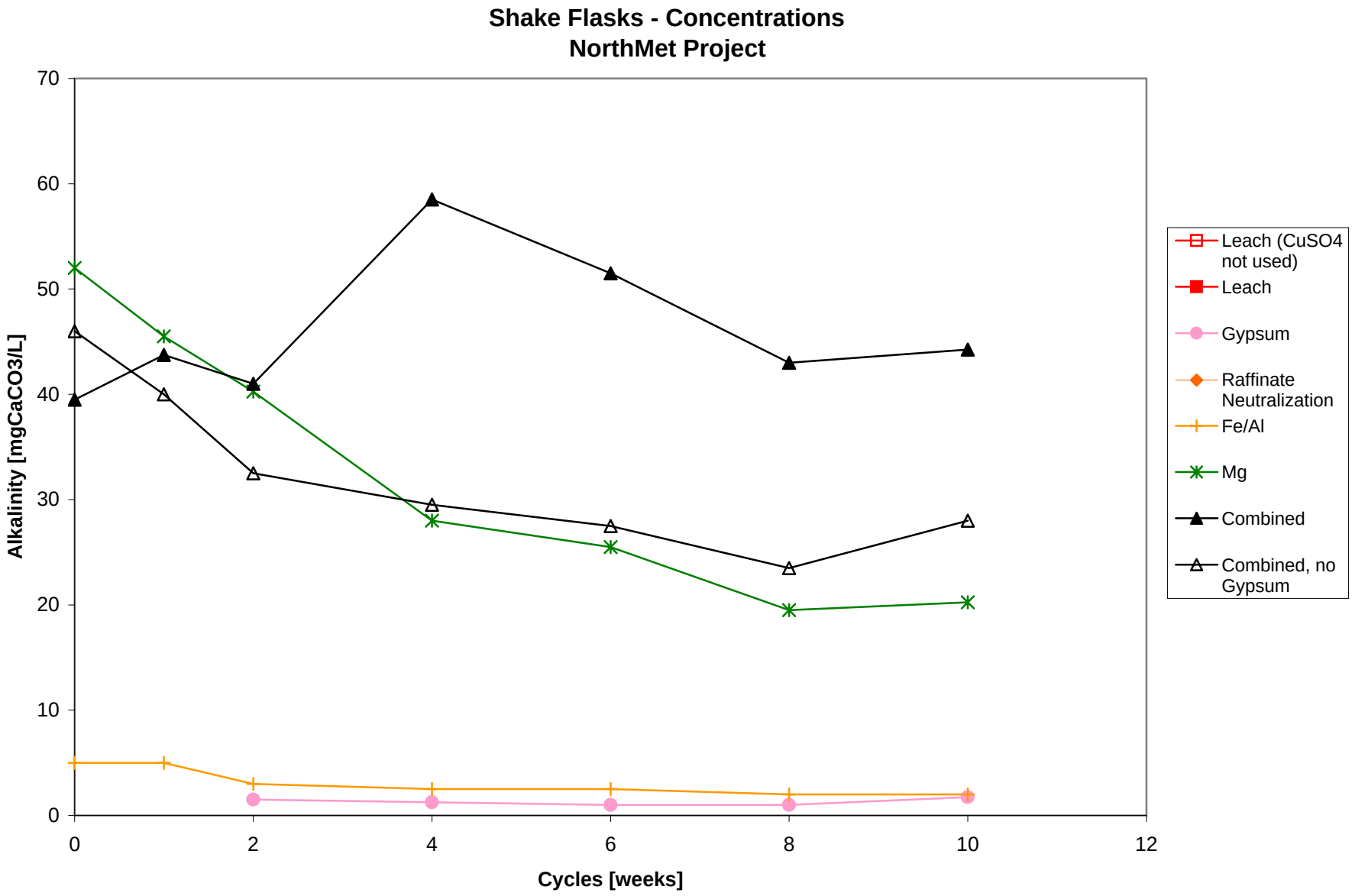
Appendix C.1
Sequential Shake Flasks

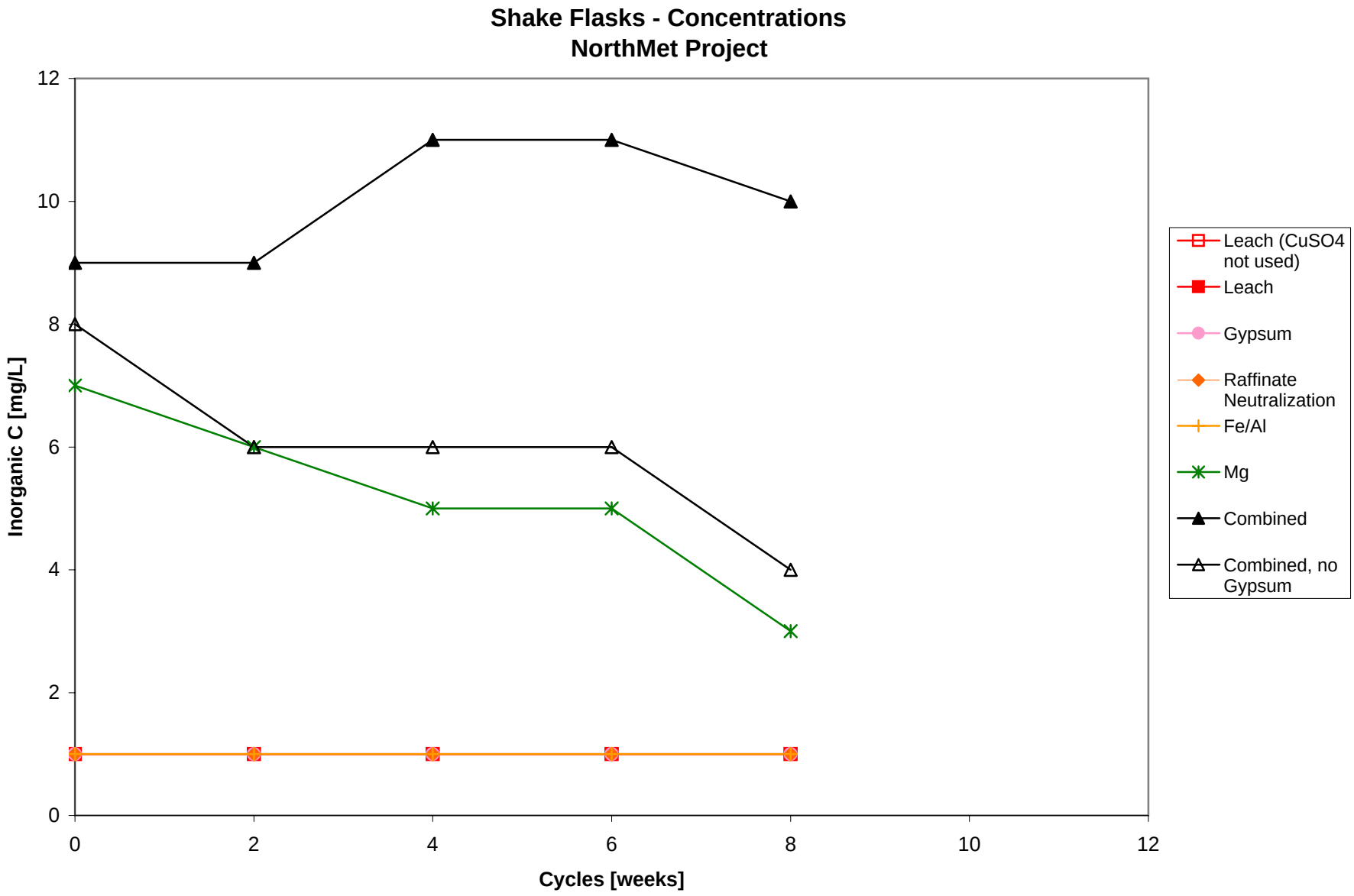

Shake Flasks - Concentrations
NorthMet Project
pH
12.5
10
7.5
5
2.5
0
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

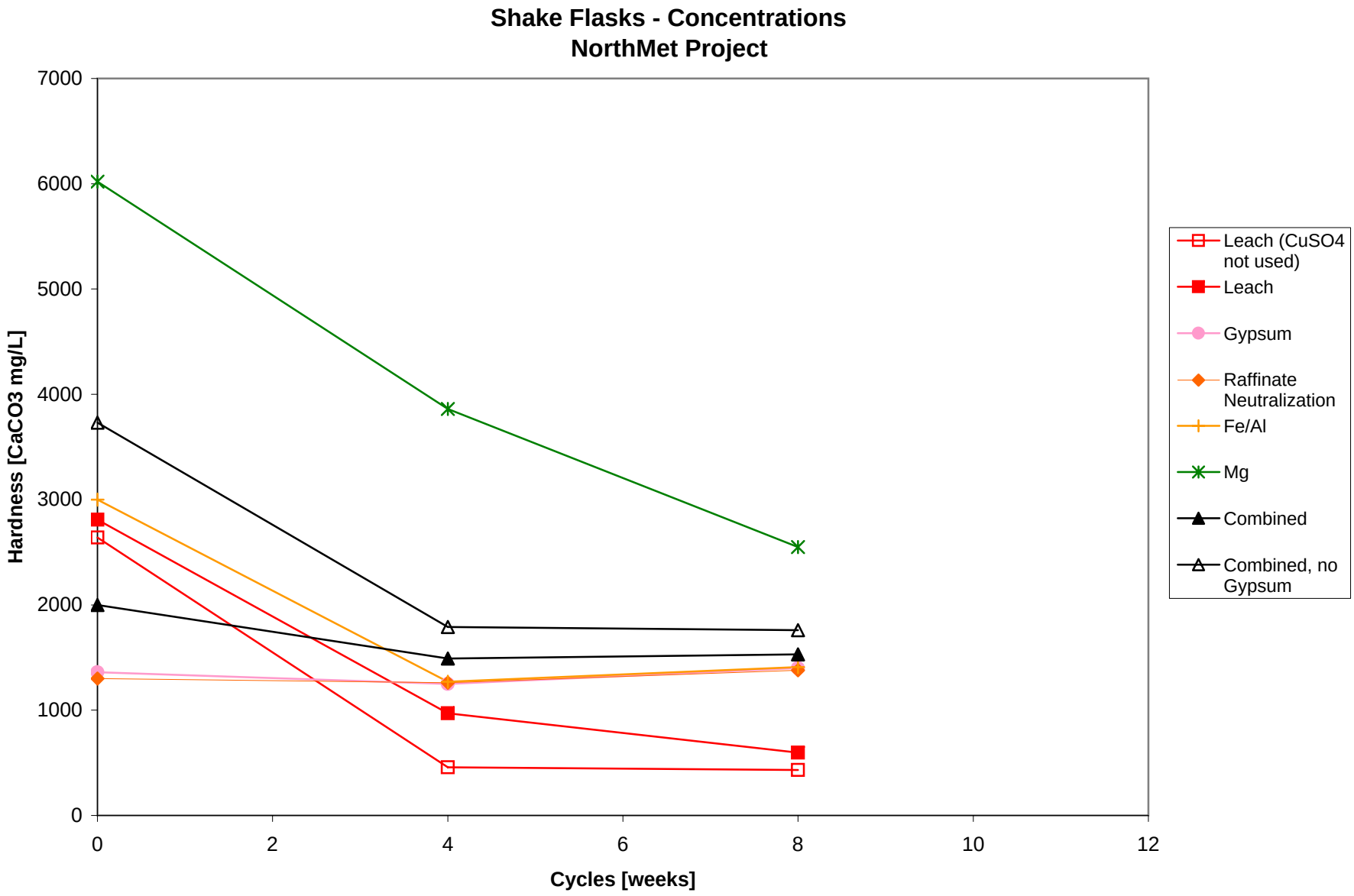

Shake Flasks - Concentrations
NorthMet Project
Conductivity [umhos/cm]
0
2000
4000
6000
8000
10000
12000
14000
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

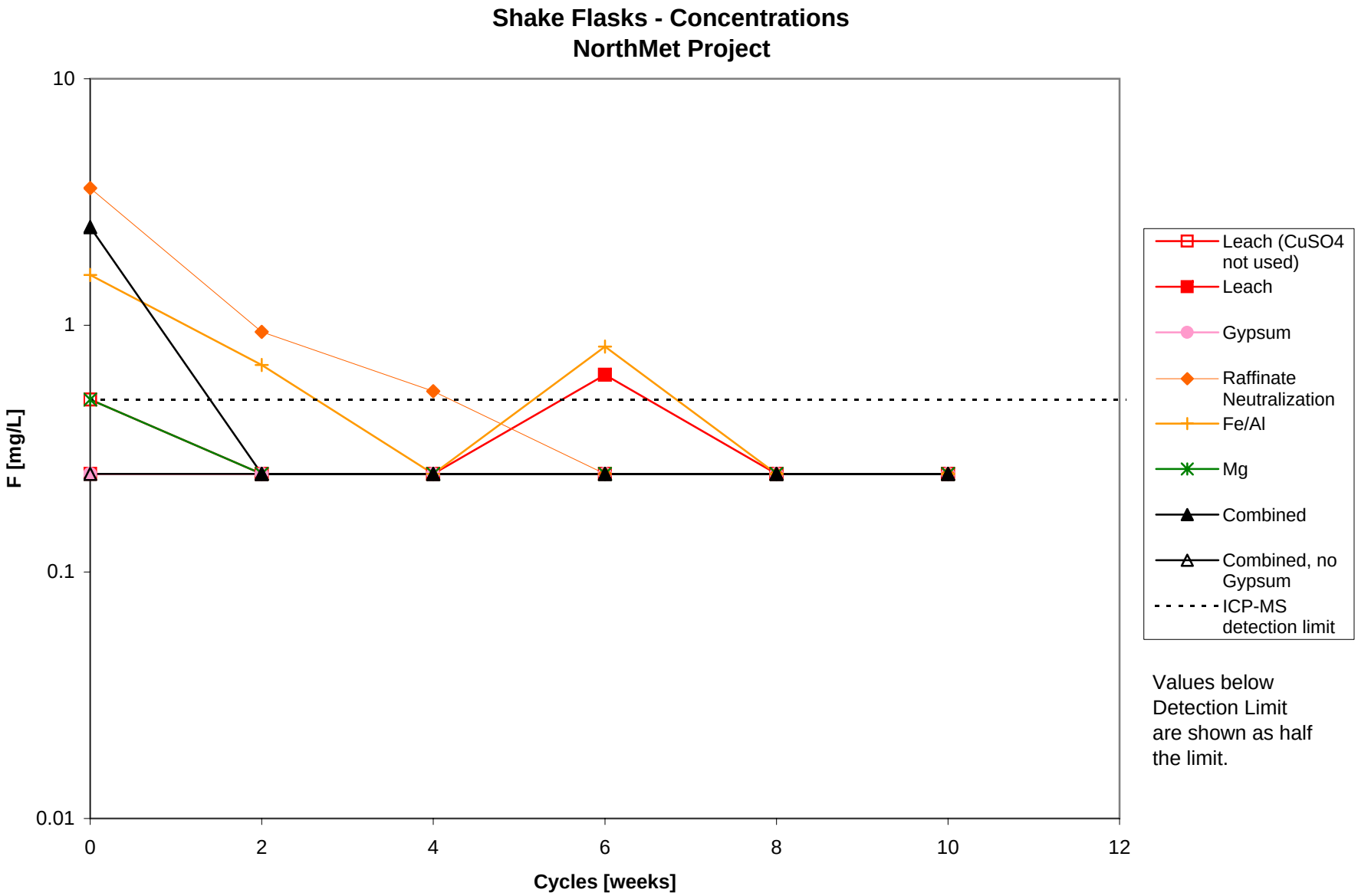

Shake Flasks - Concentrations
NorthMet Project
Acidity (pH=4.5) [mgCaCO3/L]
10000
1000
100
10
1
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

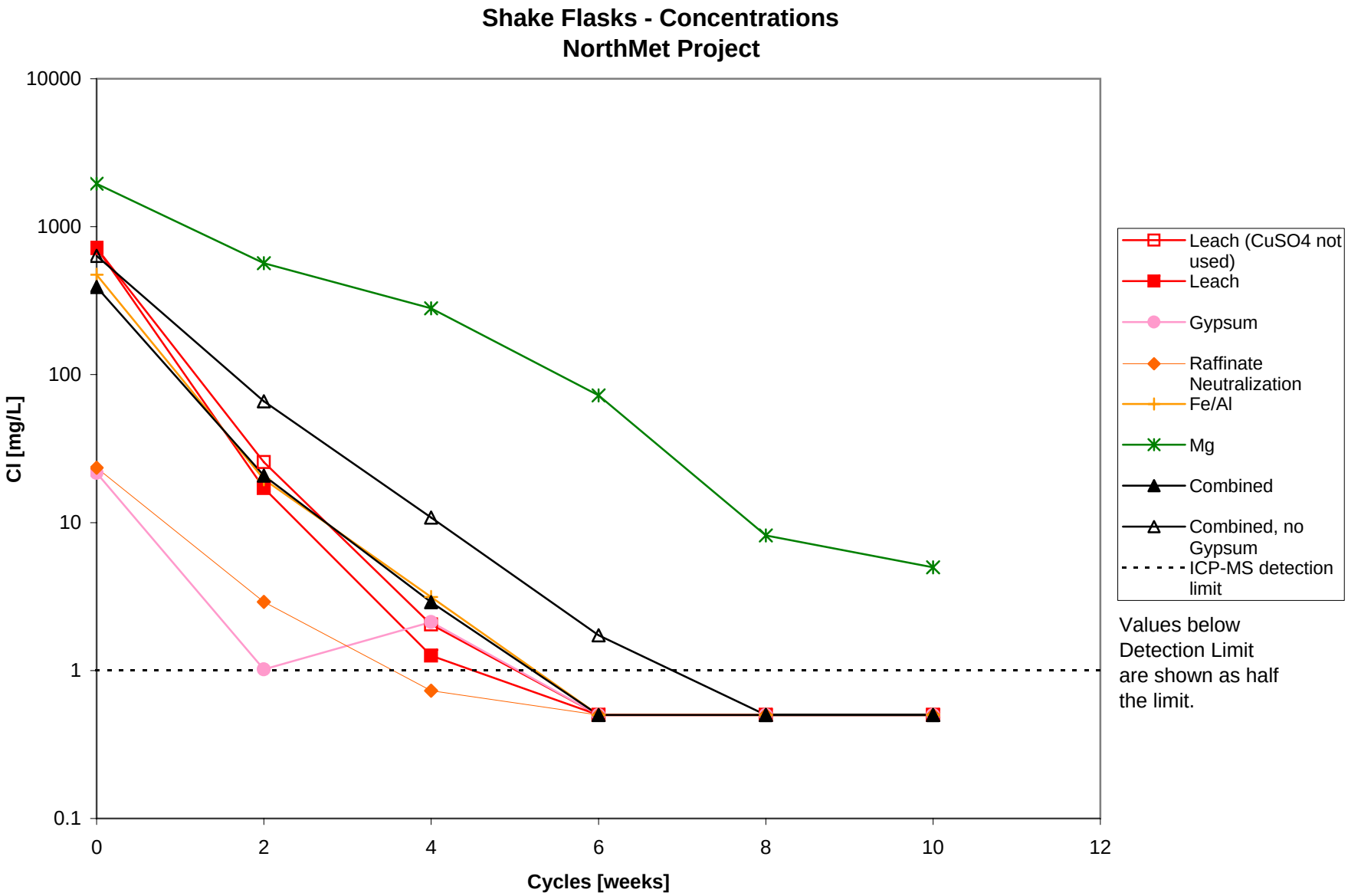

Shake Flasks - Concentrations
NorthMet Project
Acidity (pH=8.3) [mgCaCO3/L]
10000
1000
100
10
1
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

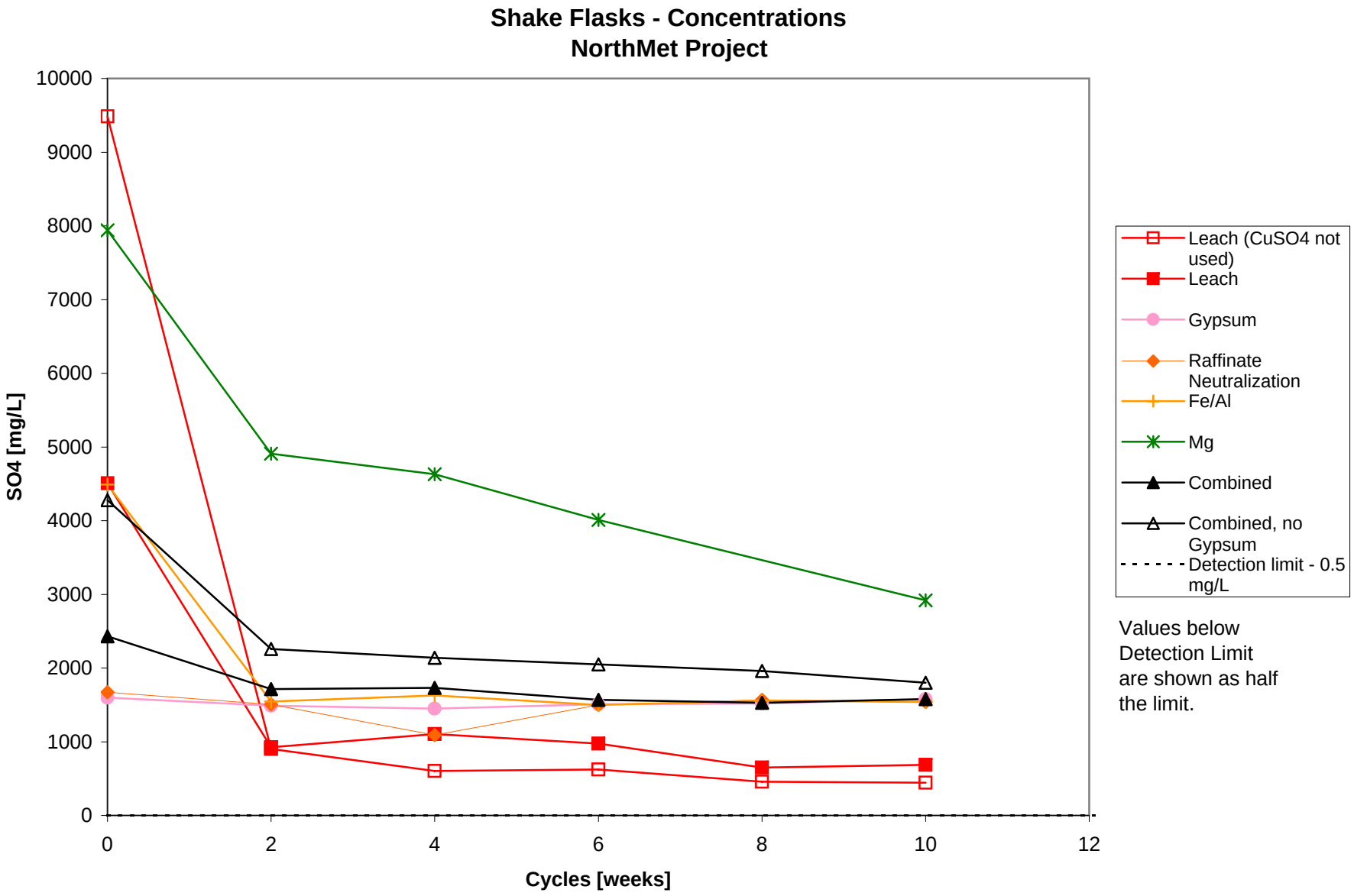

Shake Flasks - Concentrations
NorthMet Project
Alkalinity [mgCaCO3/L]
Cycles [weeks]
0
10
20
30
40
50
60
70
0
2
4
6
8
10
12
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

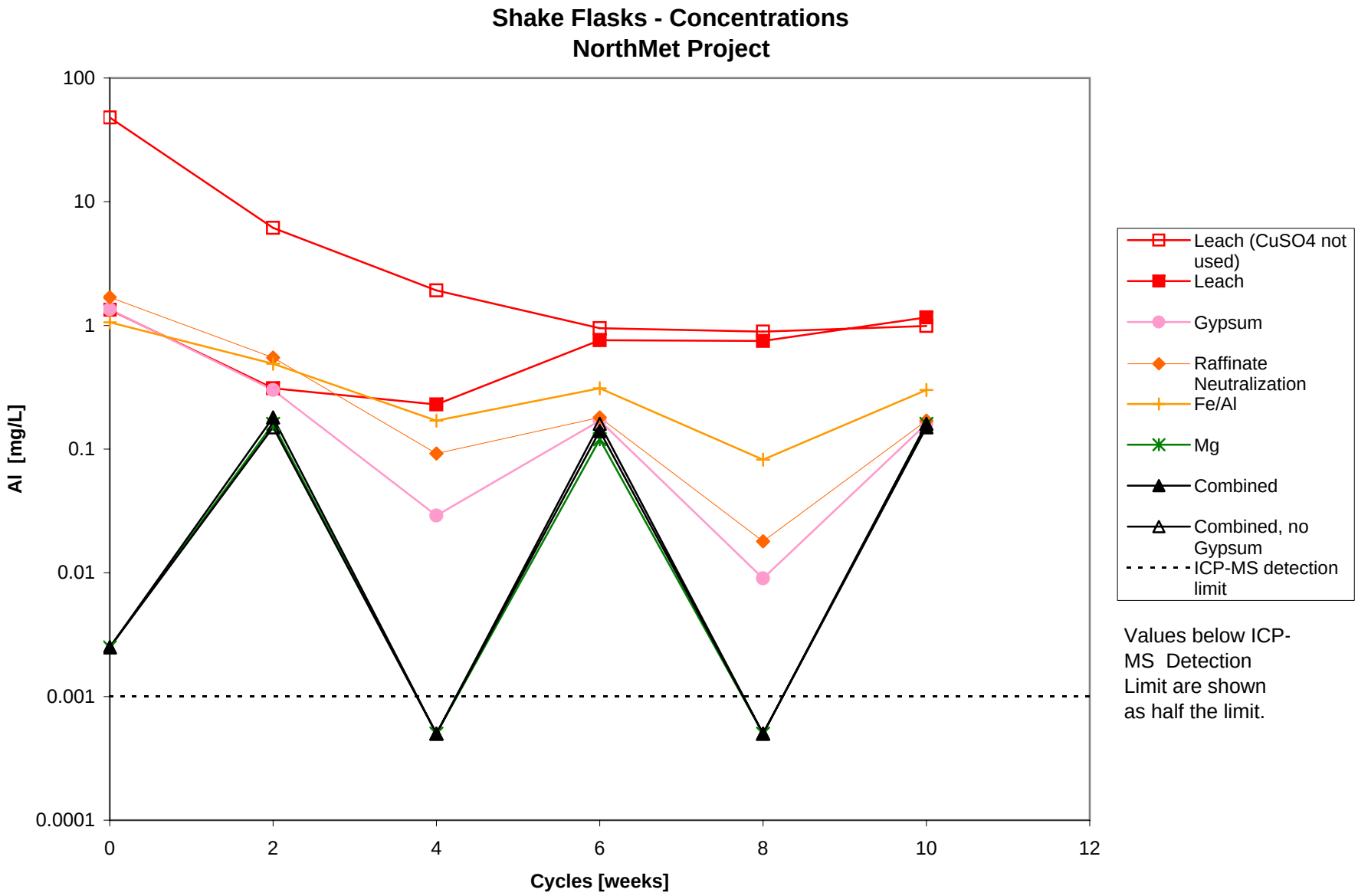

Shake Flasks - Concentrations
NorthMet Project
Inorganic C [mg/L]
Cycles [weeks]
0
2
4
6
8
10
12
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum


Shake Flasks - Concentrations
NorthMet Project
Hardness [CaCO3 mg/L]
7000
6000
5000
4000
3000
2000
1000
0
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

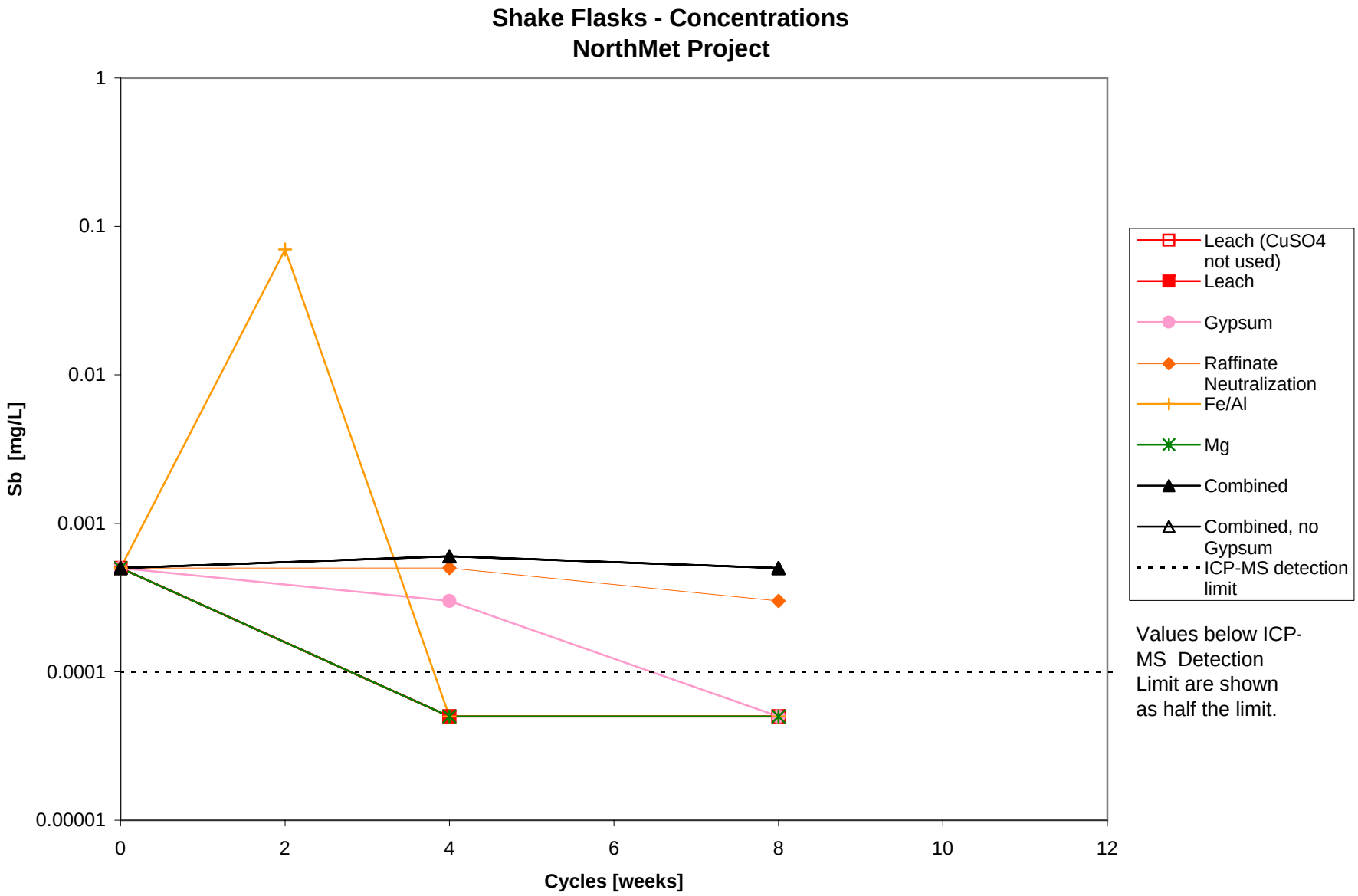

Shake Flasks - Concentrations
NorthMet Project
10
1
0.1
0.01
F [mg/L]
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below Detection Limit are shown as half the limit.

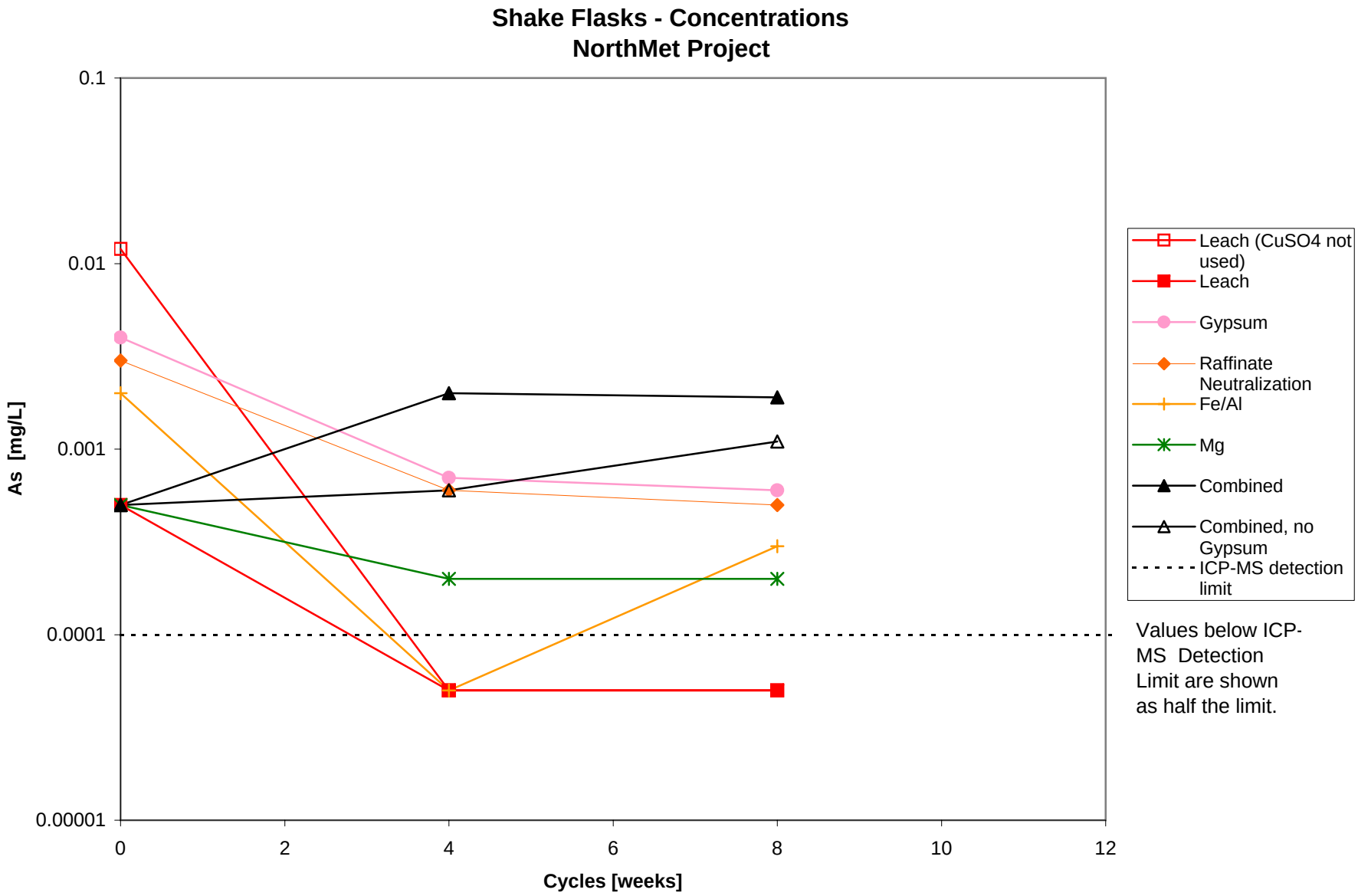

Shake Flasks - Concentrations
NorthMet Project
Cl [mg/L]
10000
1000
100
10
1
0.1
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below Detection Limit are shown as half the limit.

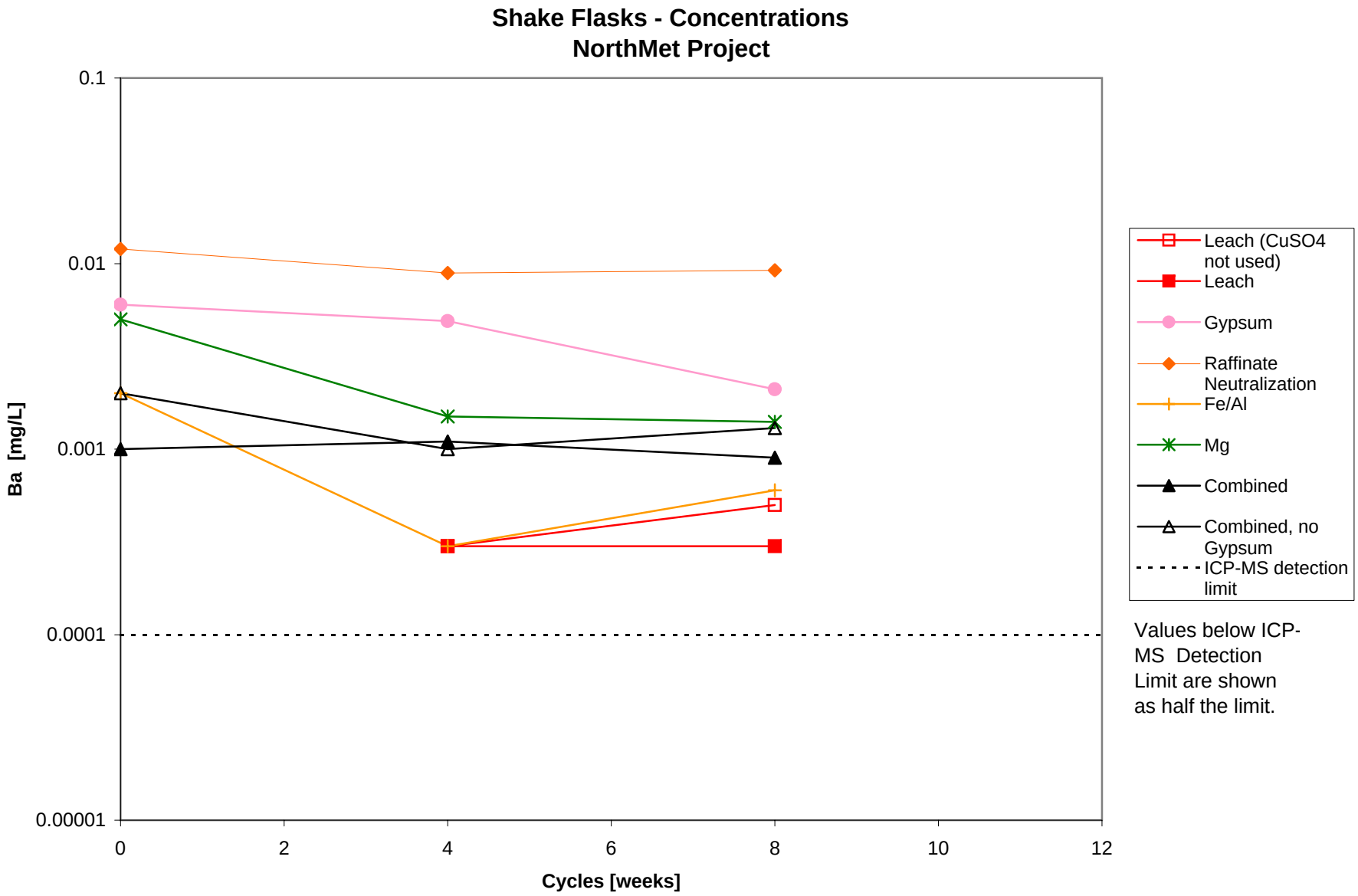

Shake Flasks - Concentrations
NorthMet Project
SO4 [mg/L]
Cycles [weeks]
0
1000
2000
3000
4000
5000
6000
7000
8000
9000
10000
0
2
4
6
8
10
12
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
Detection limit - 0.5 mg/L
Values below Detection Limit are shown as half the limit.

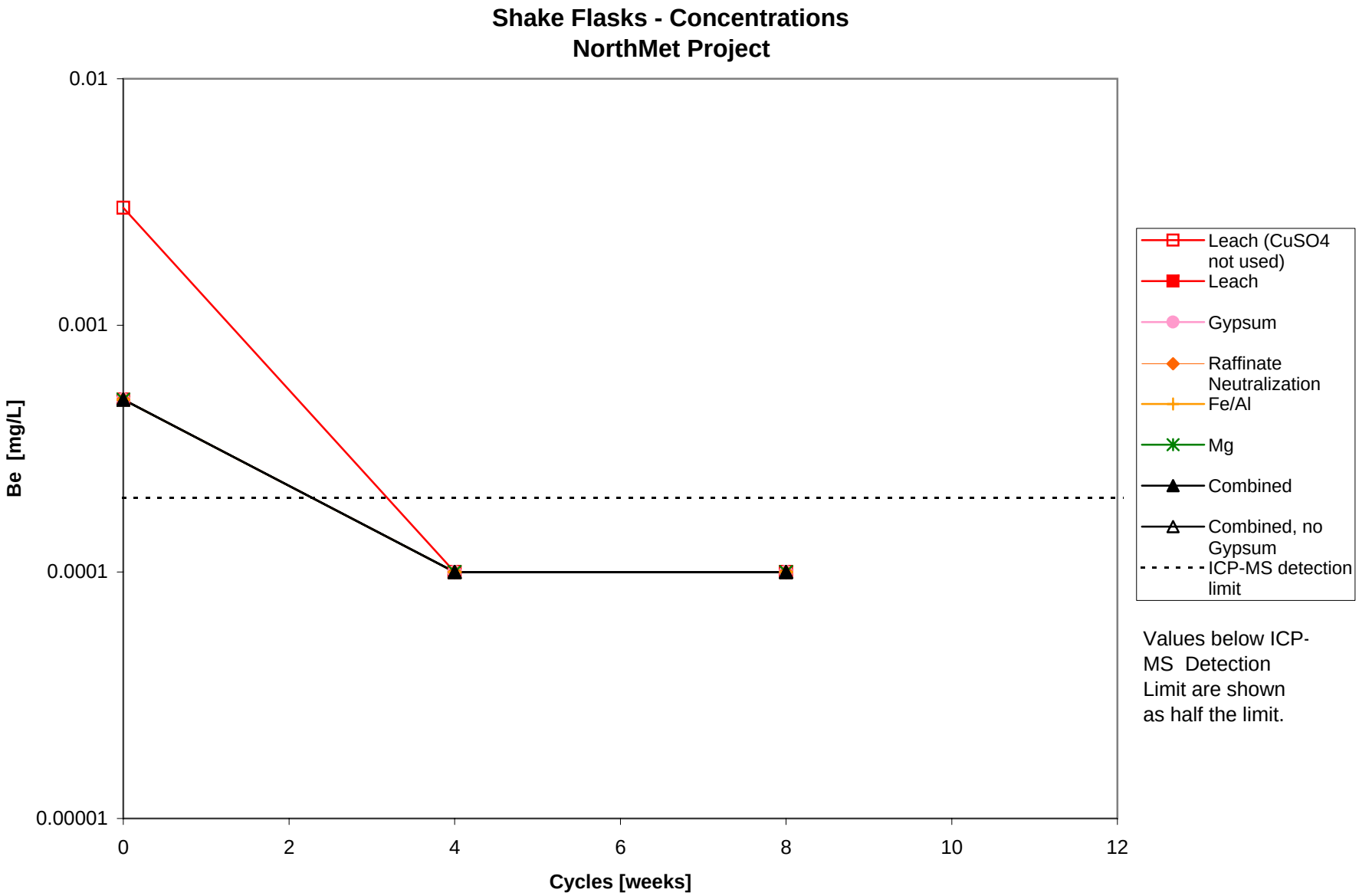


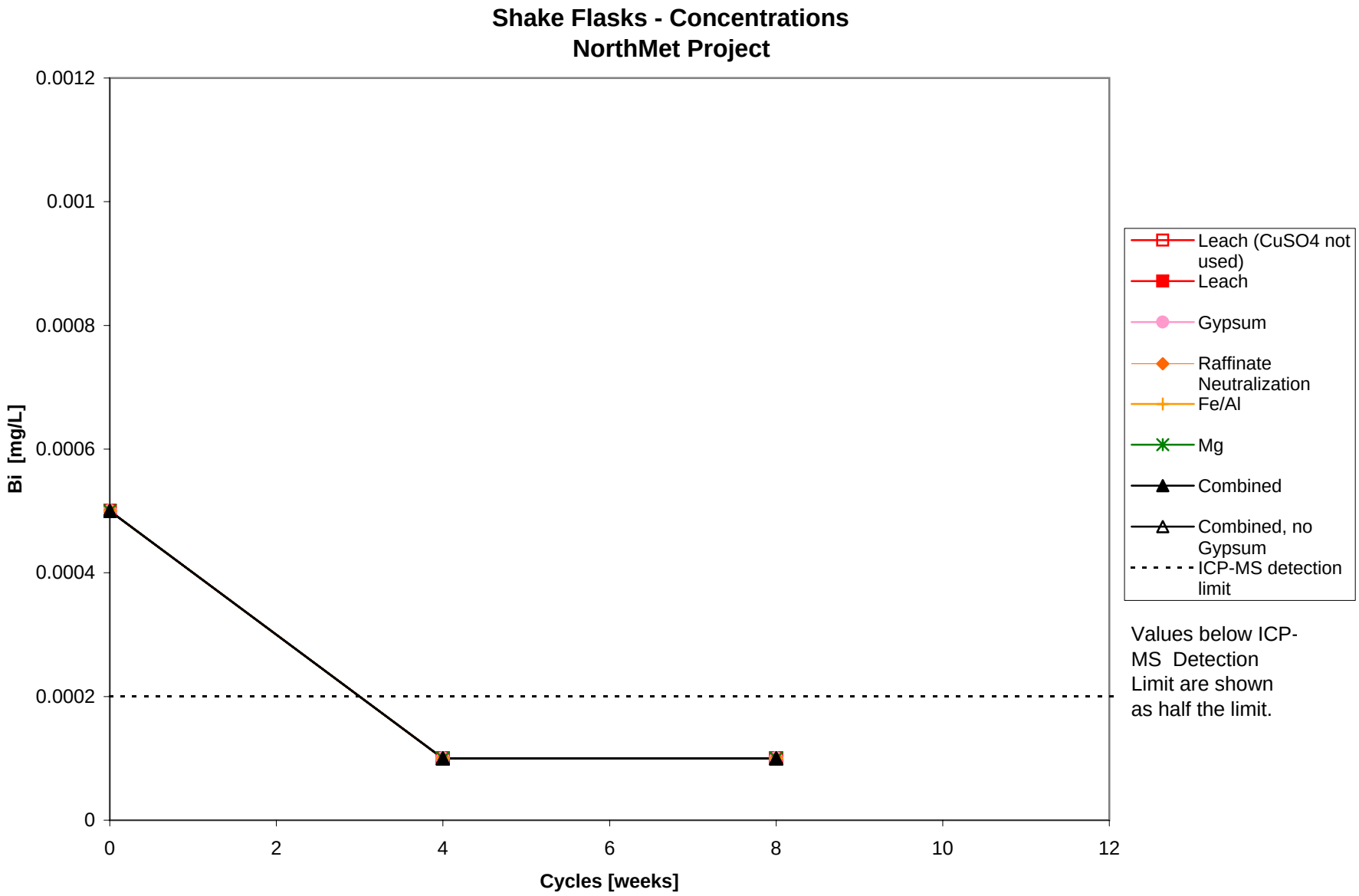
Values below ICP-MS Detection Limit are shown as half the limit.

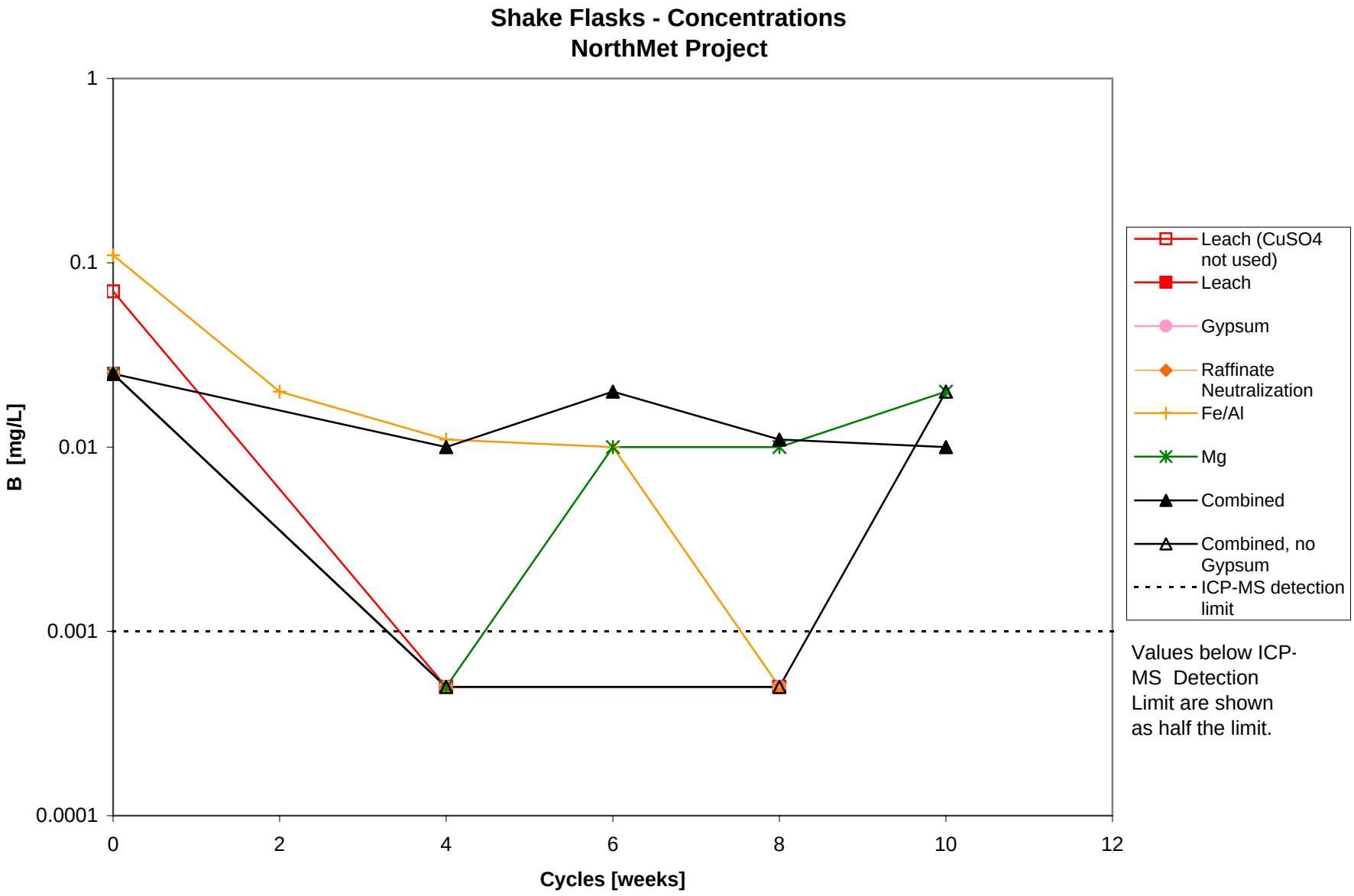

Shake Flasks - Concentrations
NorthMet Project
Sb [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

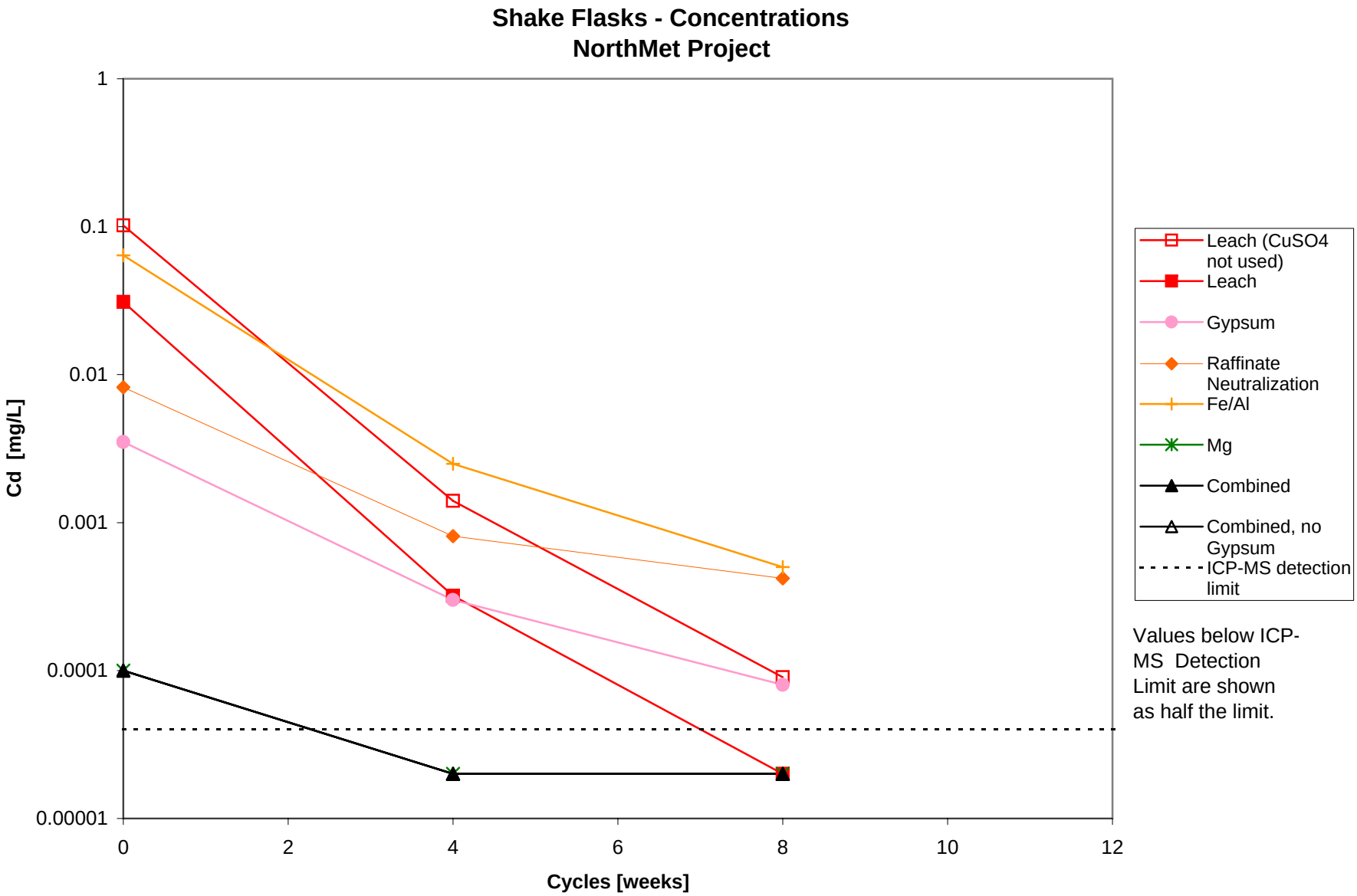

Shake Flasks - Concentrations
NorthMet Project
As [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

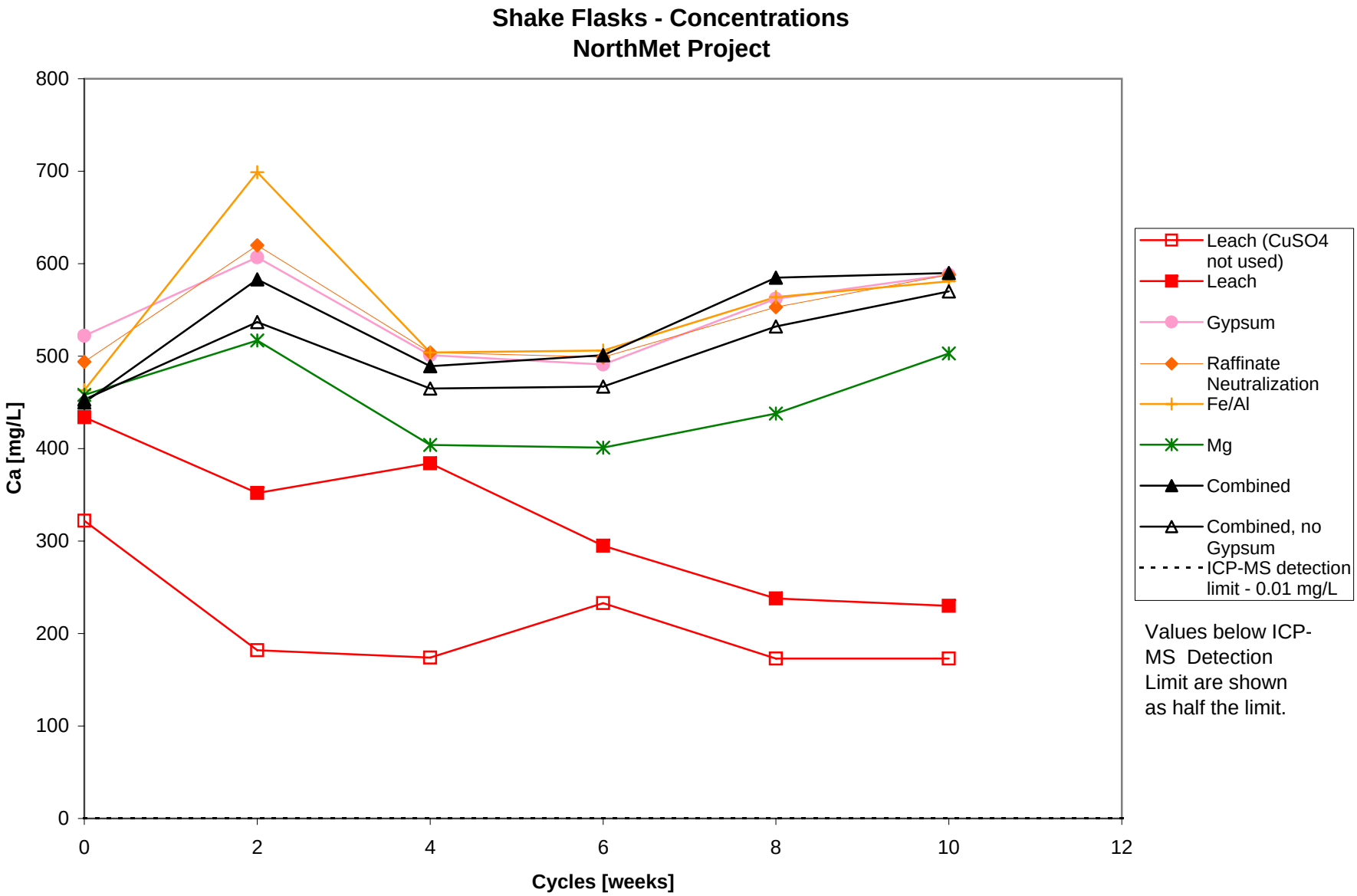

Shake Flasks - Concentrations
NorthMet Project
Ba [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

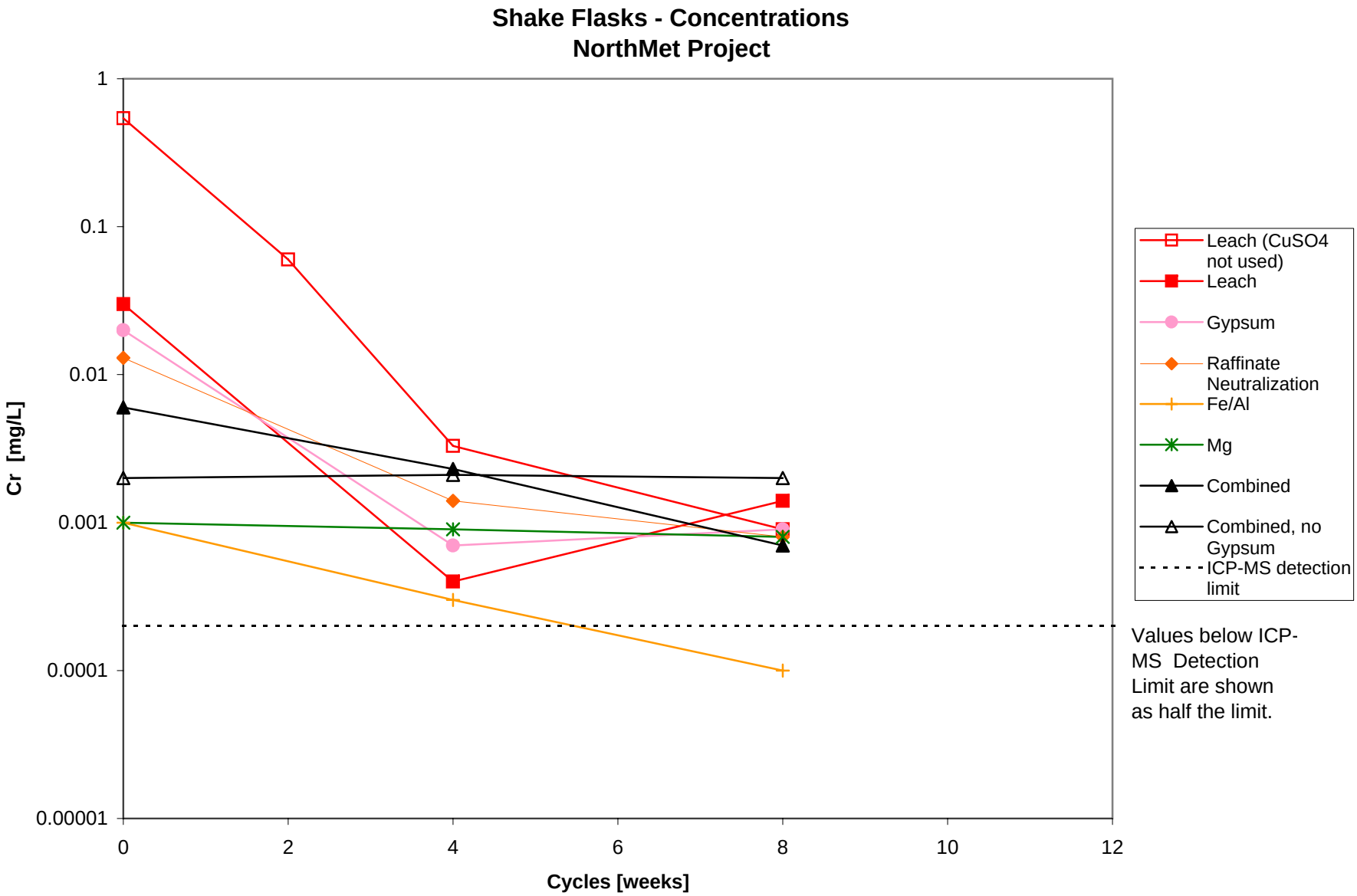

Shake Flasks - Concentrations
NorthMet Project
0.01
0.001
0.0001
0.00001
Be [mg/L]
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

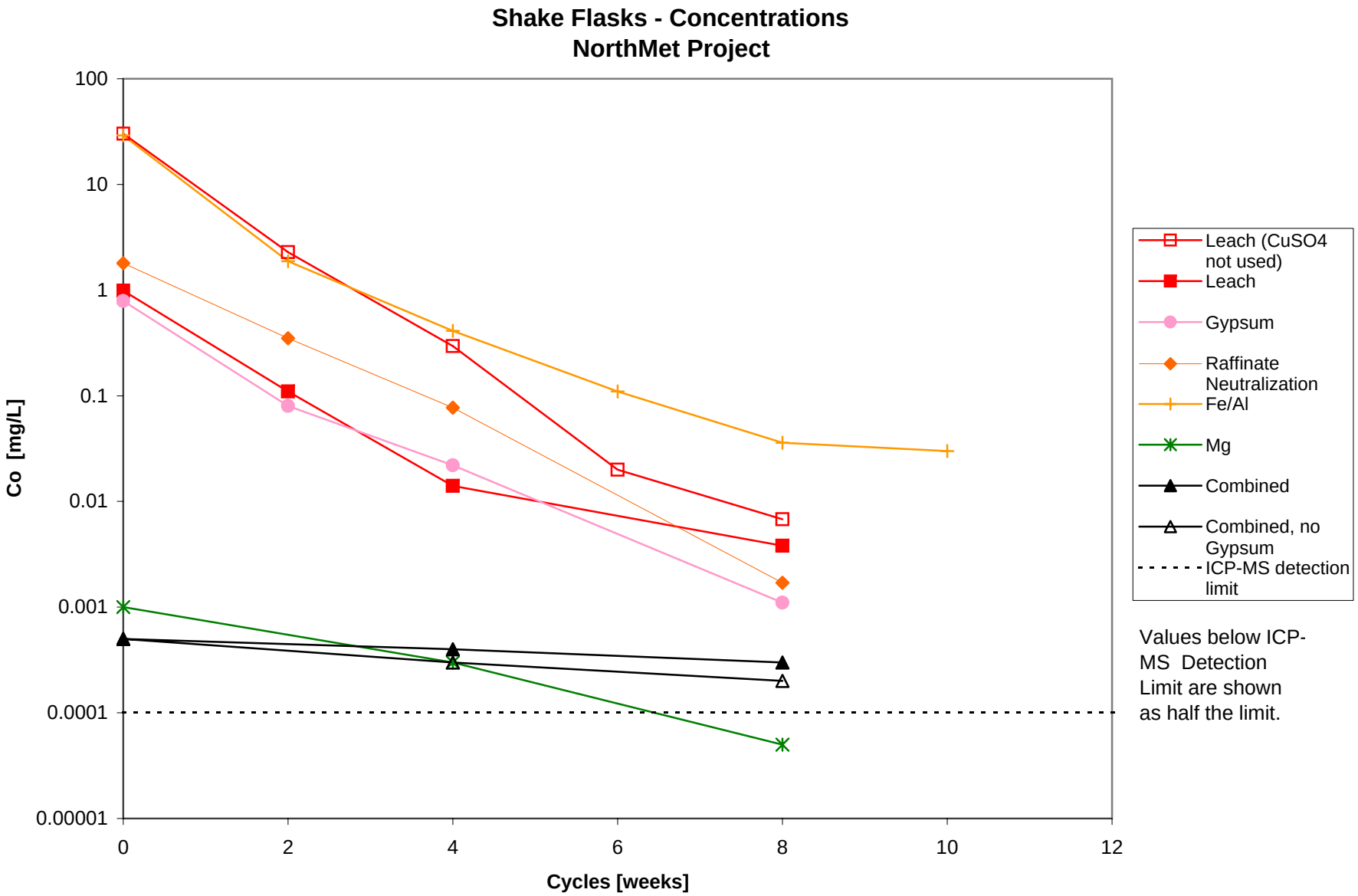

Shake Flasks - Concentrations
NorthMet Project
Bi [mg/L]
0.0012
0.001
0.0008
0.0006
0.0004
0.0002
0
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

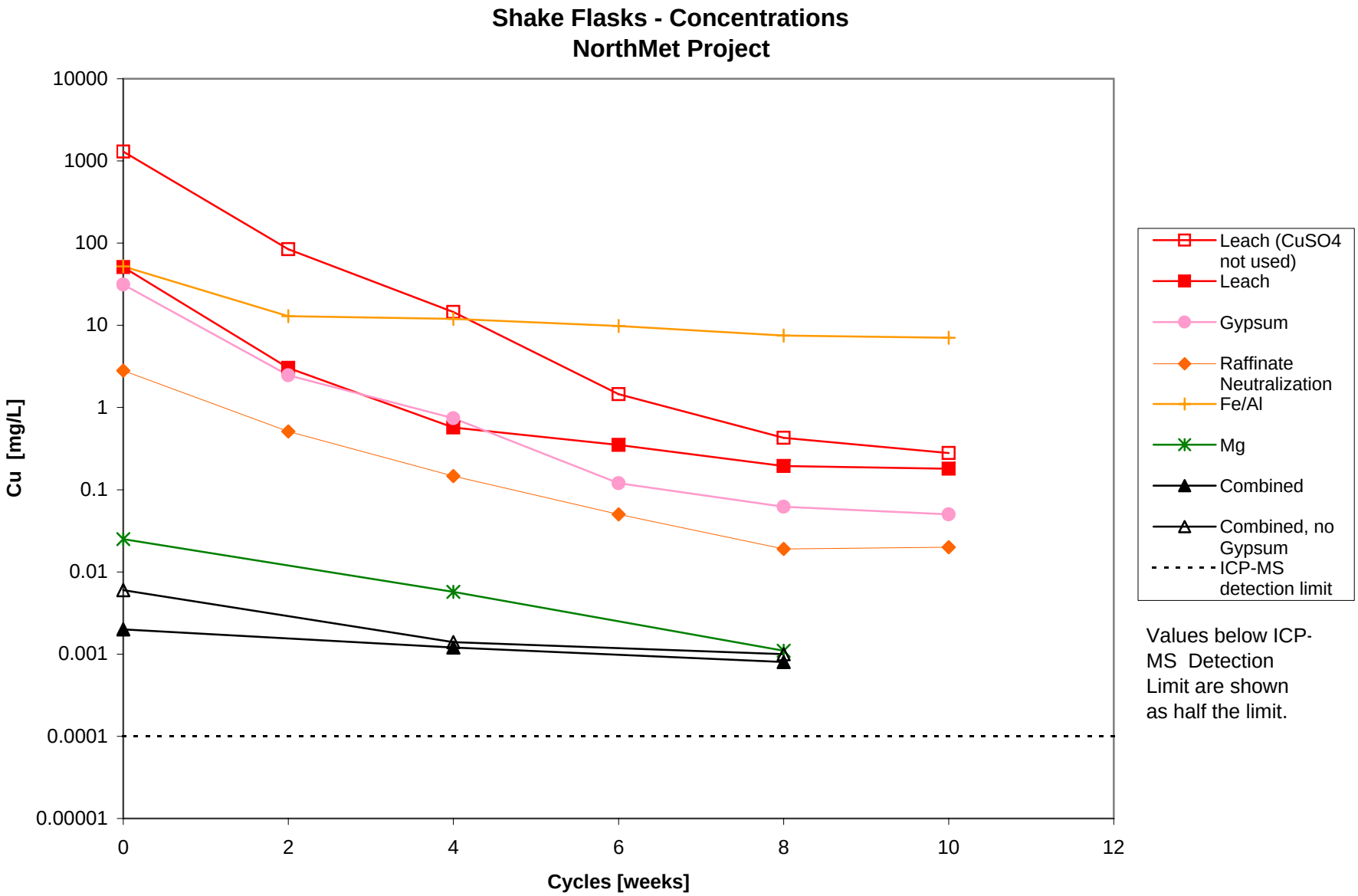

Shake Flasks - Concentrations
NorthMet Project
B [mg/L]
1
0.1
0.01
0.001
0.0001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.


Shake Flasks - Concentrations
NorthMet Project
Cd [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

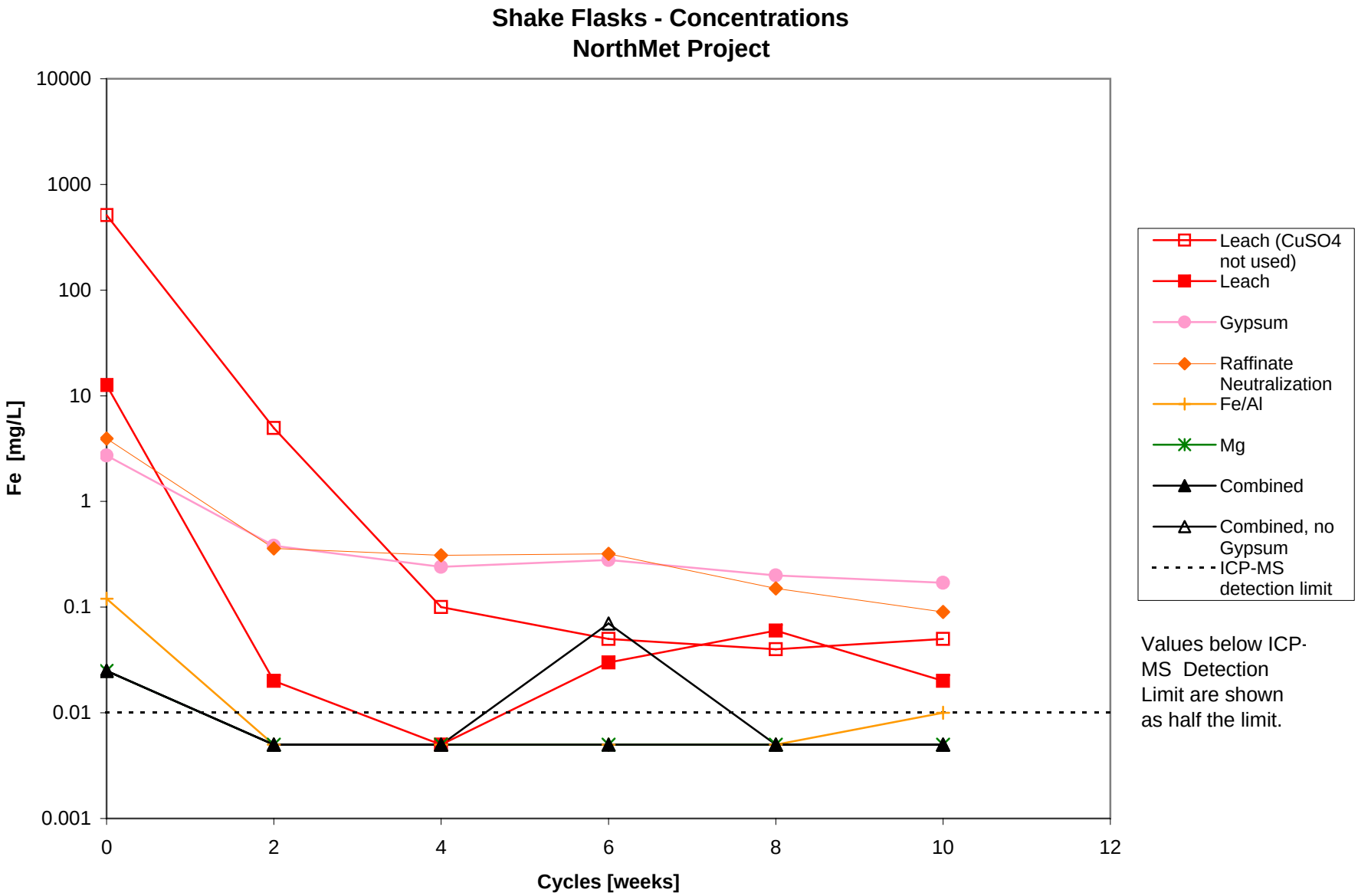

Shake Flasks - Concentrations
NorthMet Project
Ca [mg/L]
800
700
600
500
400
300
200
100
0
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit - 0.01 mg/L
Values below ICP-MS Detection Limit are shown as half the limit.

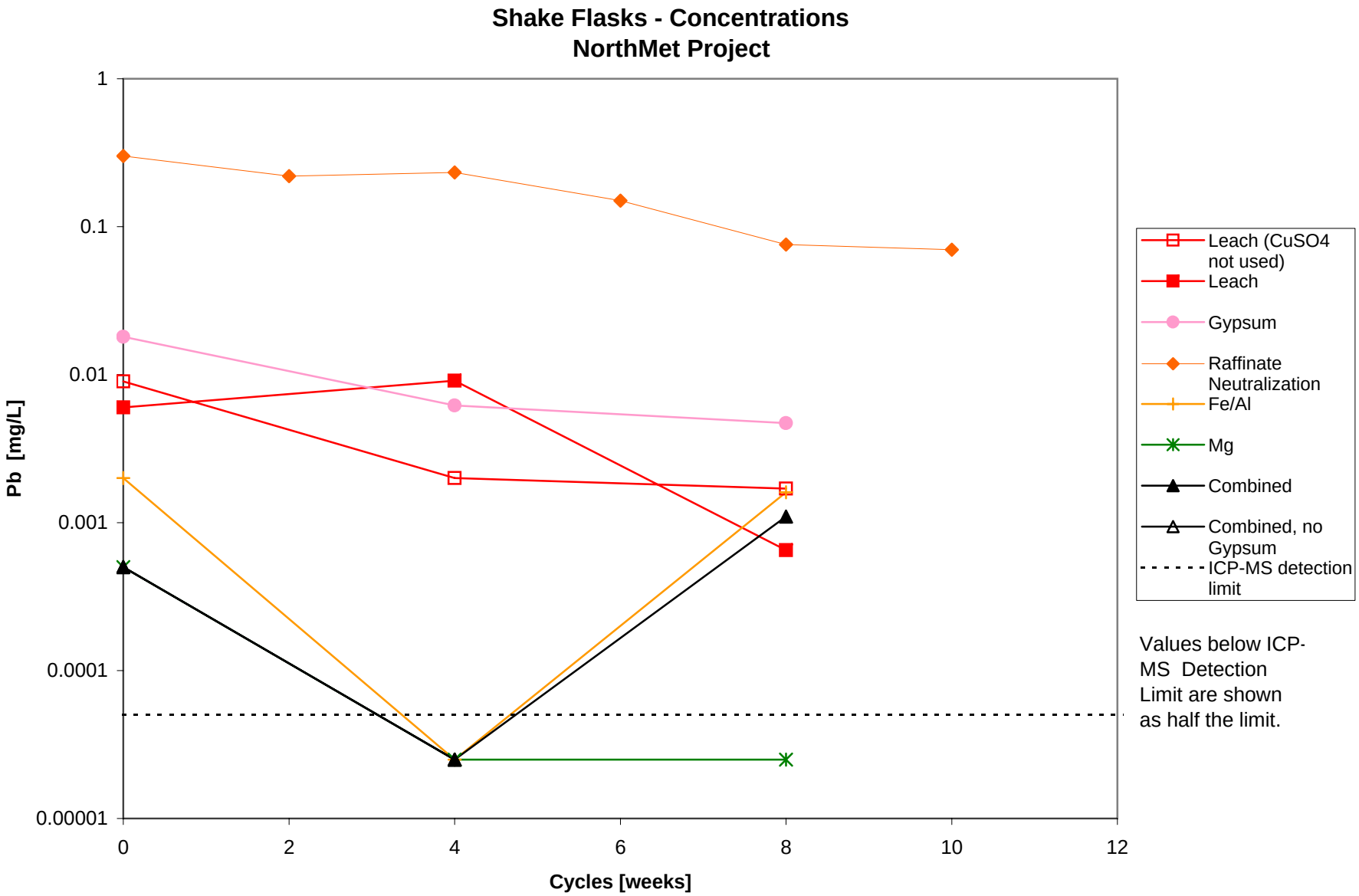

Shake Flasks - Concentrations
NorthMet Project
Cr [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

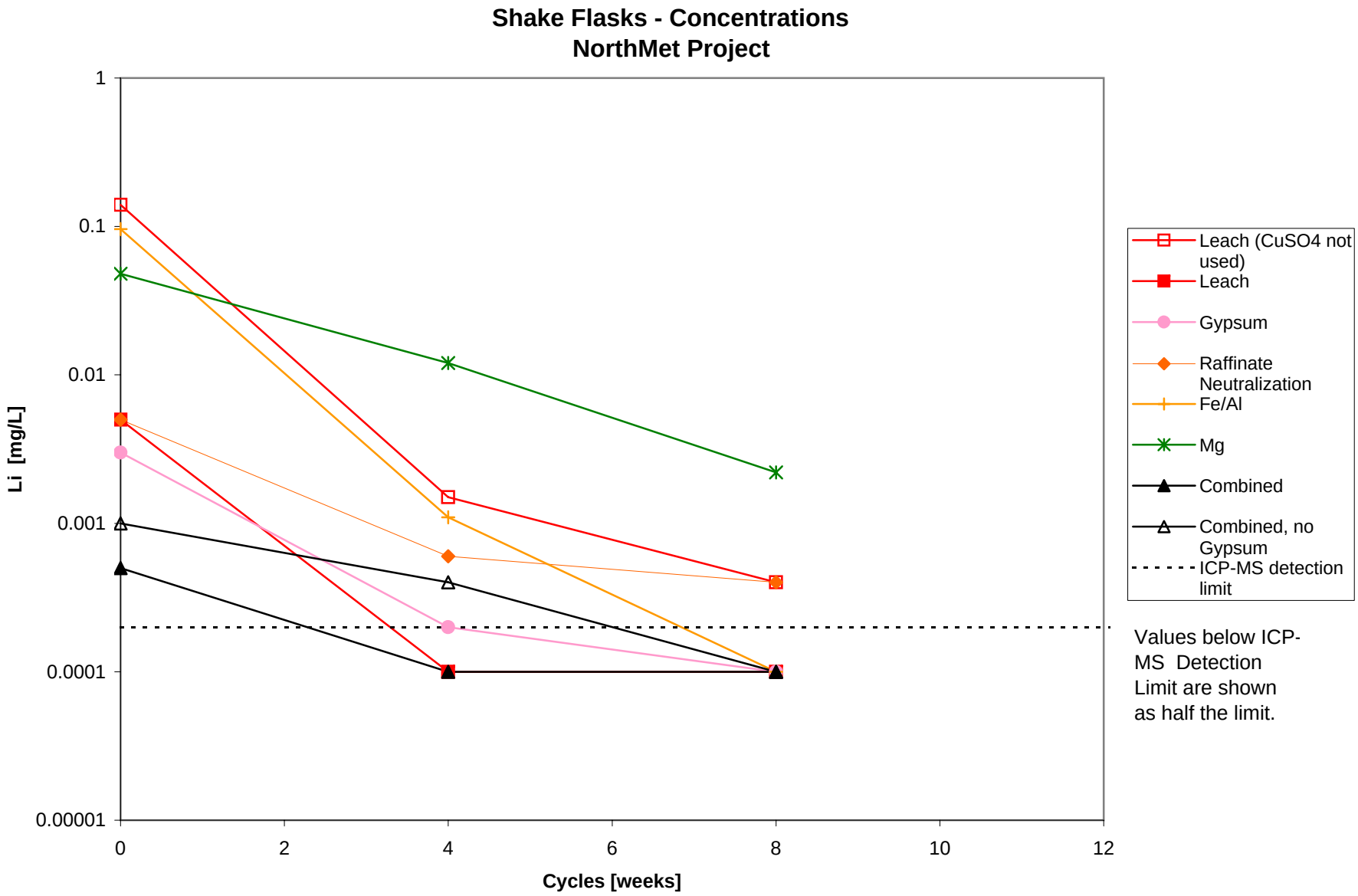

Shake Flasks - Concentrations
NorthMet Project
Co [mg/L]
100
10
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

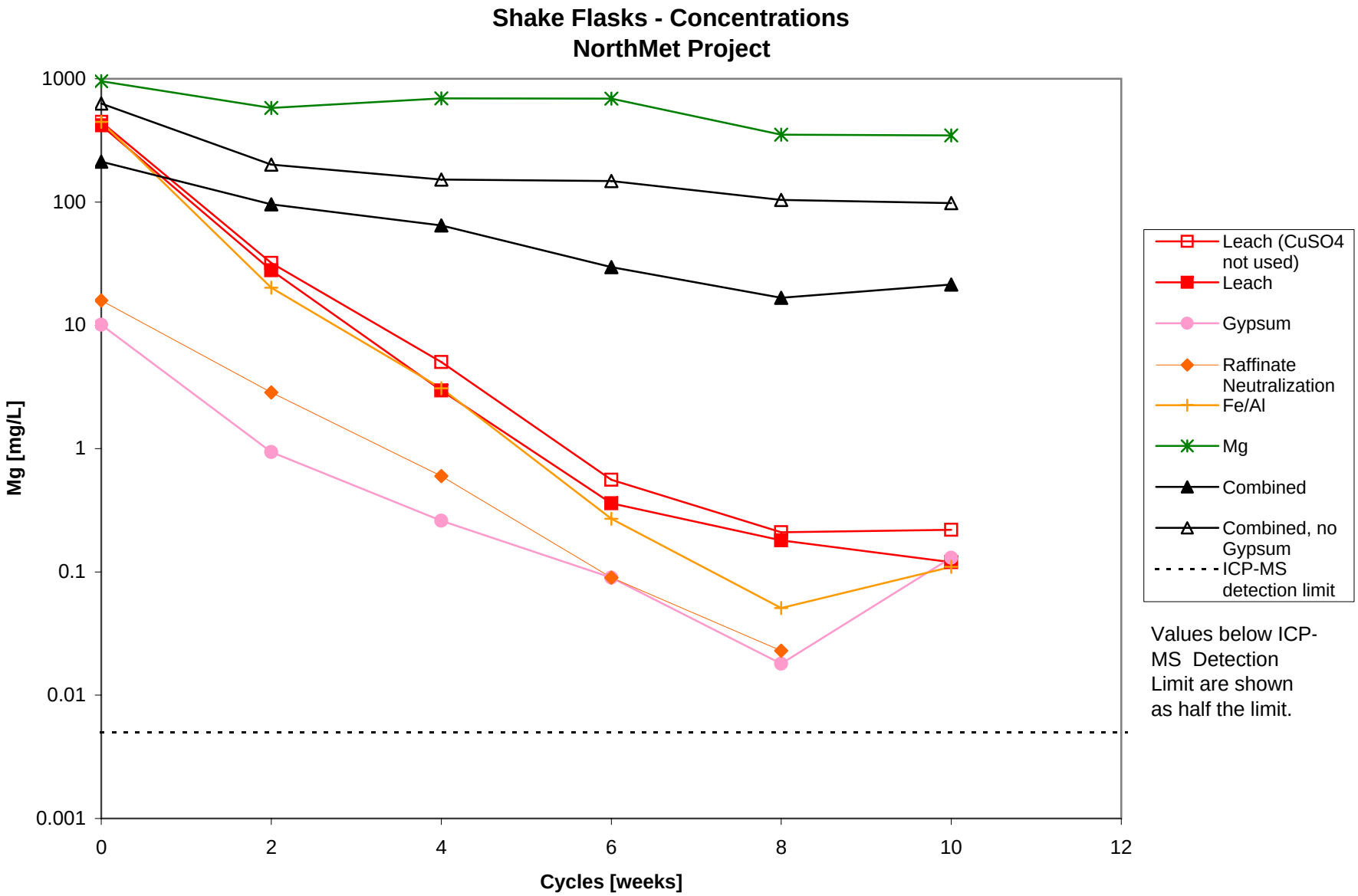


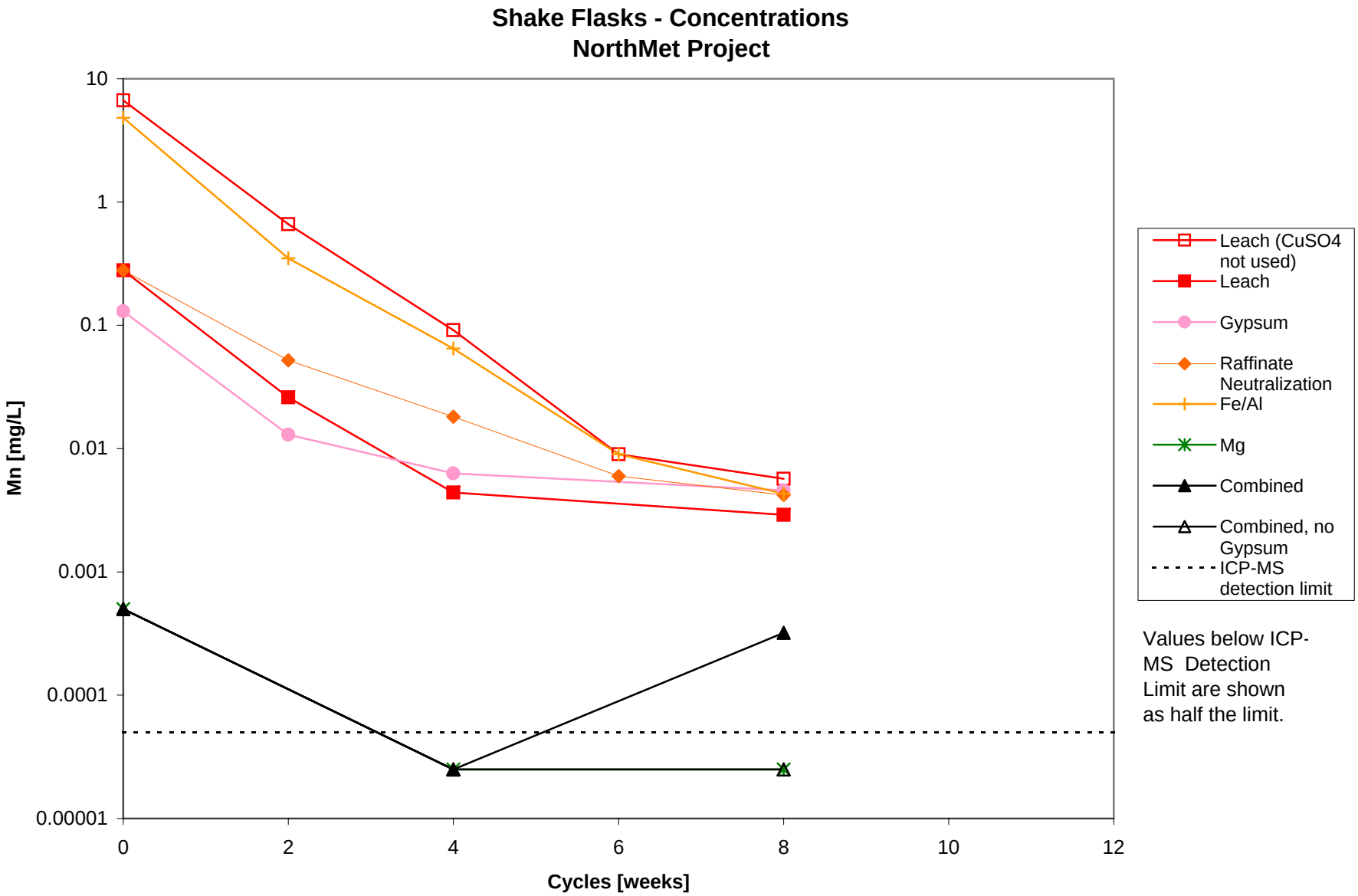
Values below ICP-MS Detection Limit are shown as half the limit.

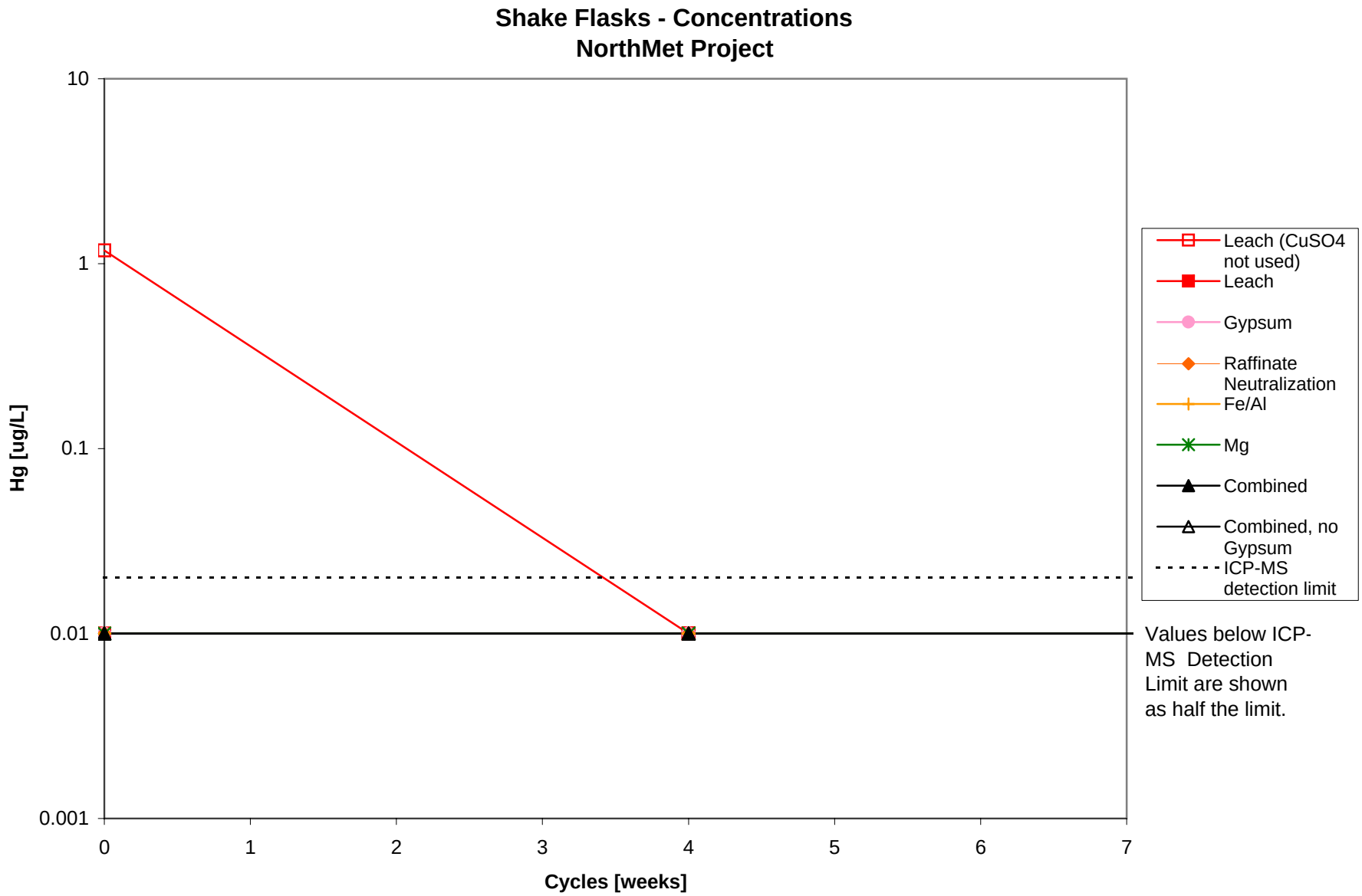

Shake Flasks - Concentrations
NorthMet Project
Fe [mg/L]
10000
1000
100
10
1
0.1
0.01
0.001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

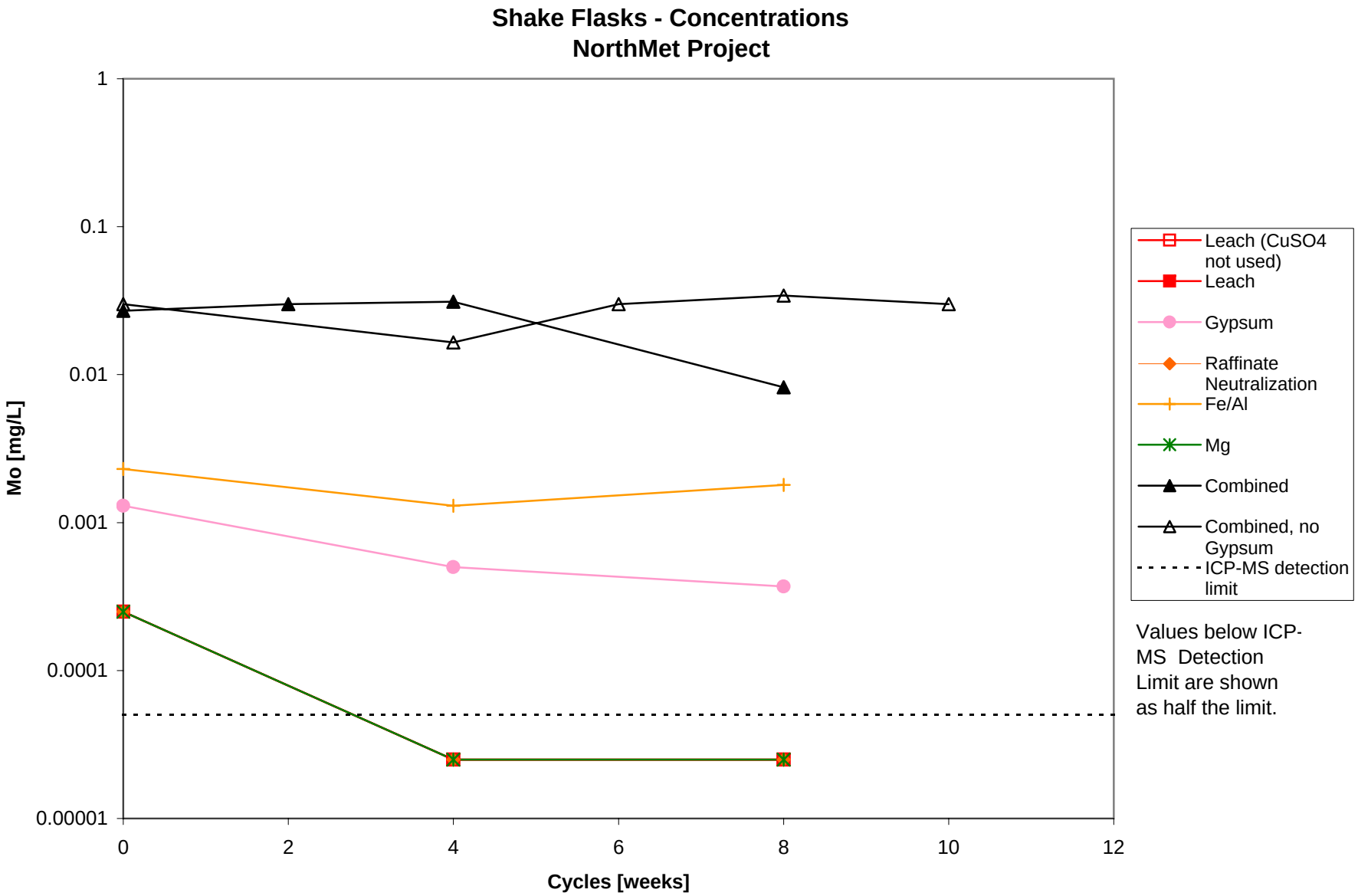

Shake Flasks - Concentrations
NorthMet Project
Pb [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

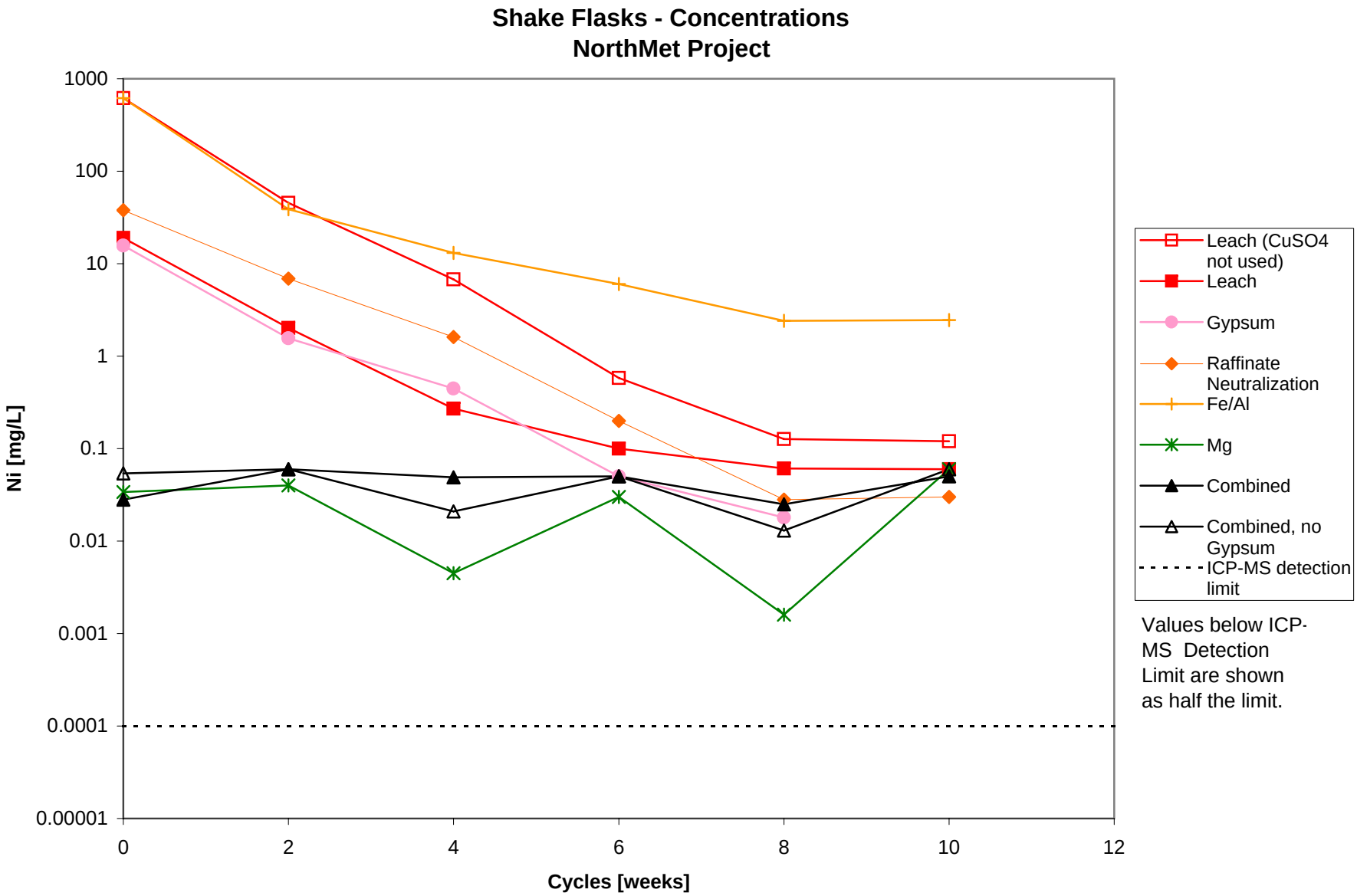

Shake Flasks - Concentrations
NorthMet Project
Li [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

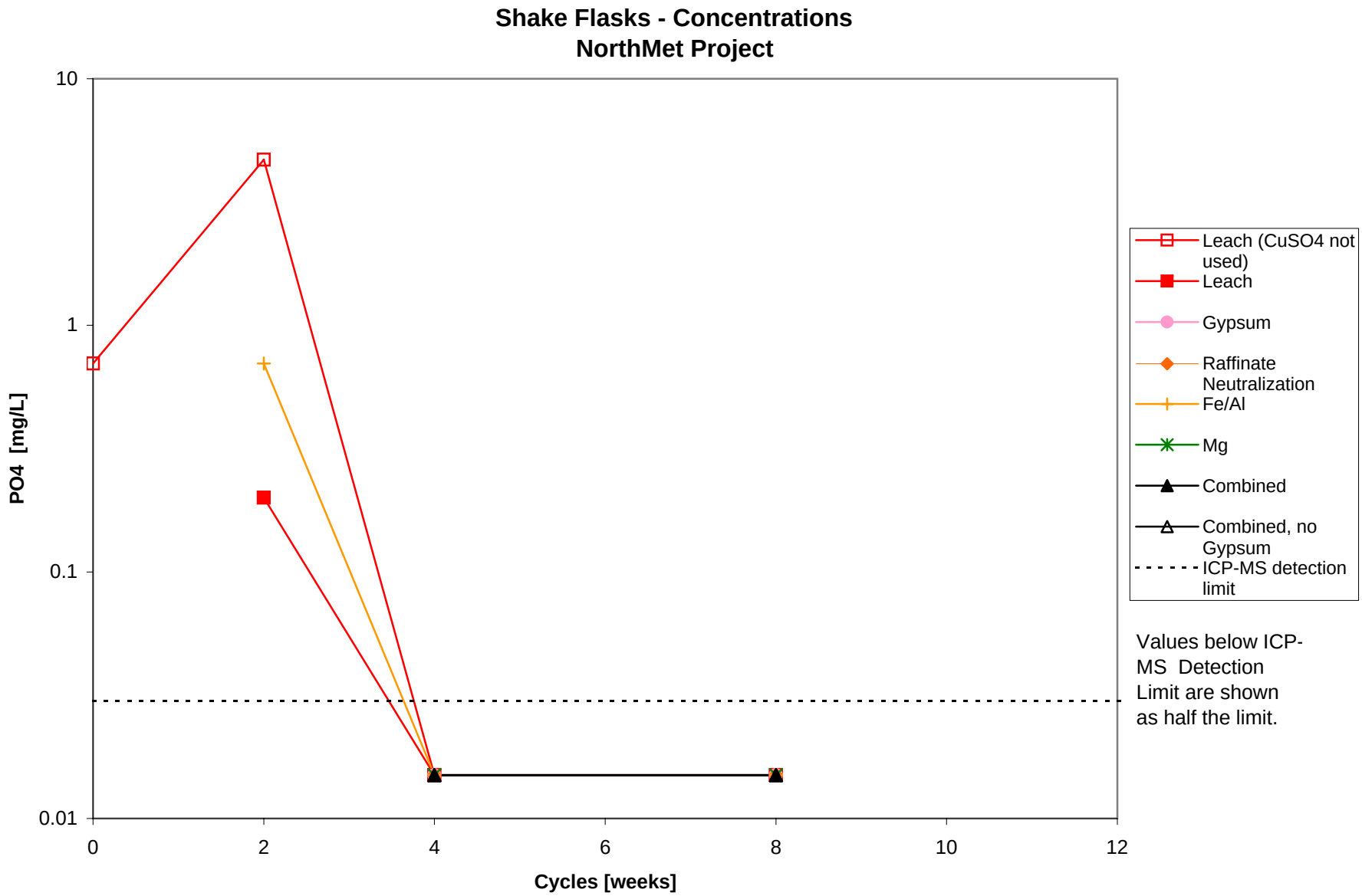

Shake Flasks - Concentrations
NorthMet Project
Mg [mg/L]
1000
100
10
1
0.1
0.01
0.001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

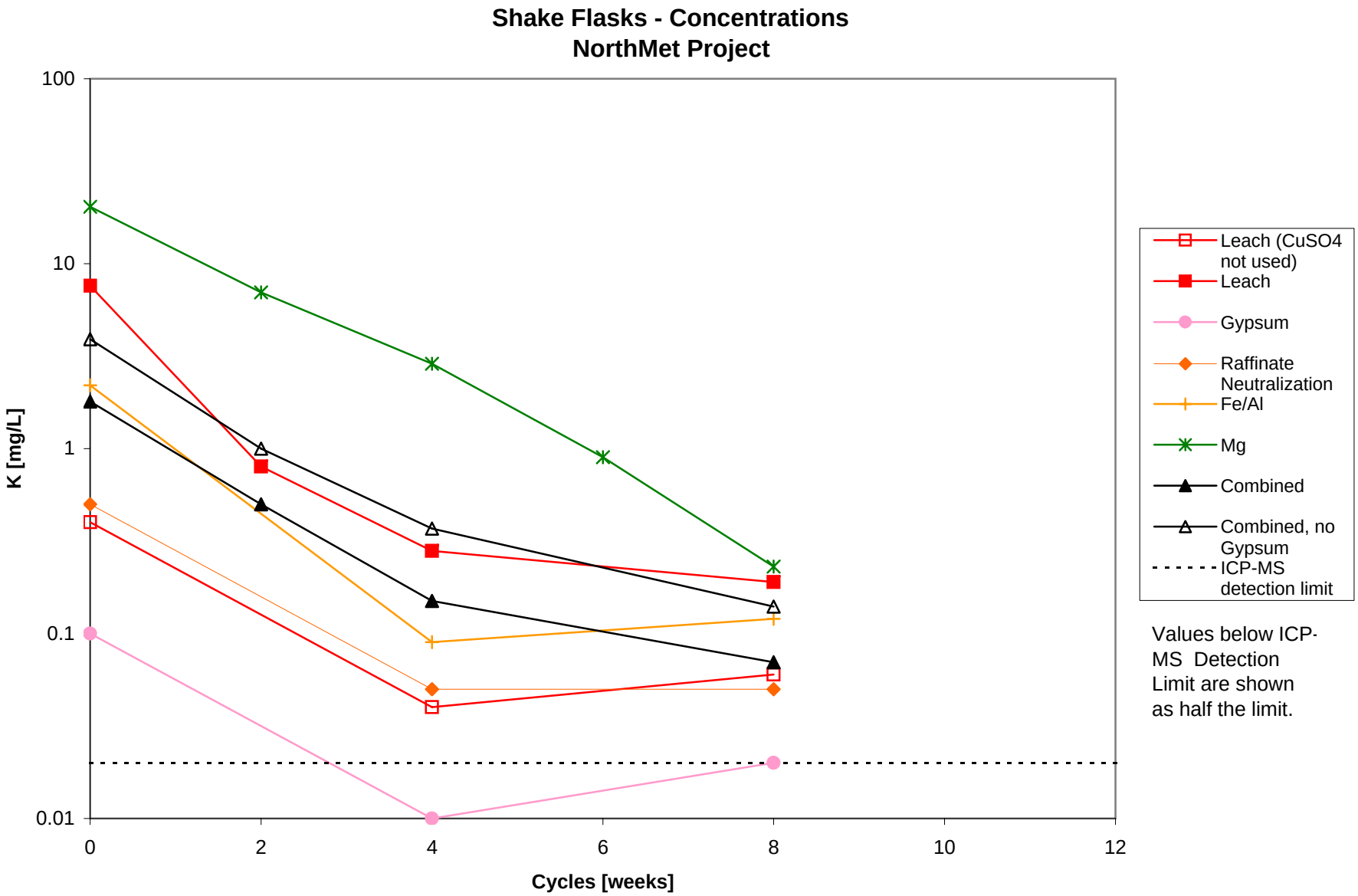

Shake Flasks - Concentrations
NorthMet Project
Mn [mg/L]
10
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

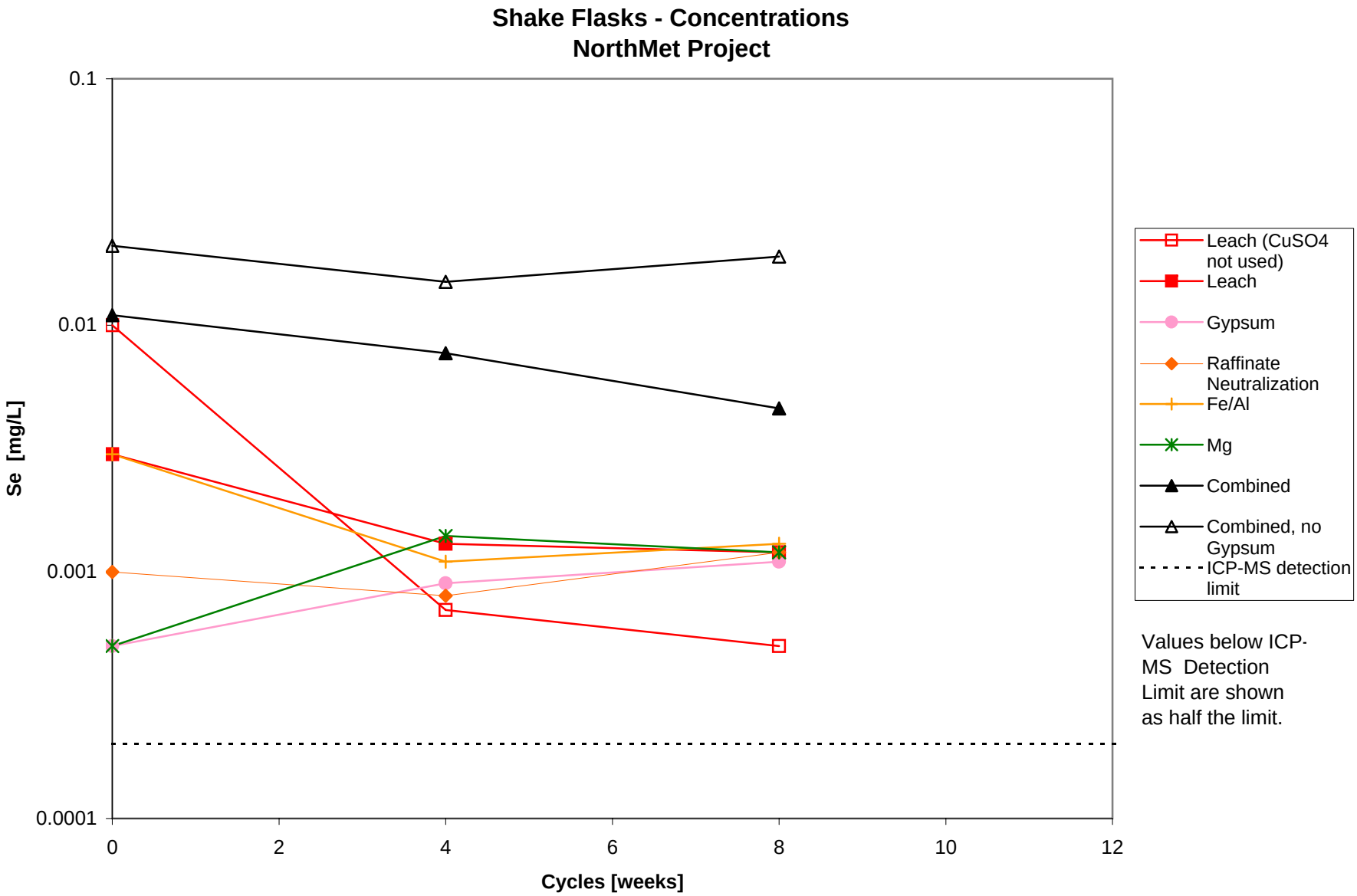

Shake Flasks - Concentrations
NorthMet Project
Hg [ug/L]
10
1
0.1
0.01
0.001
0
1
2
3
4
5
6
7
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

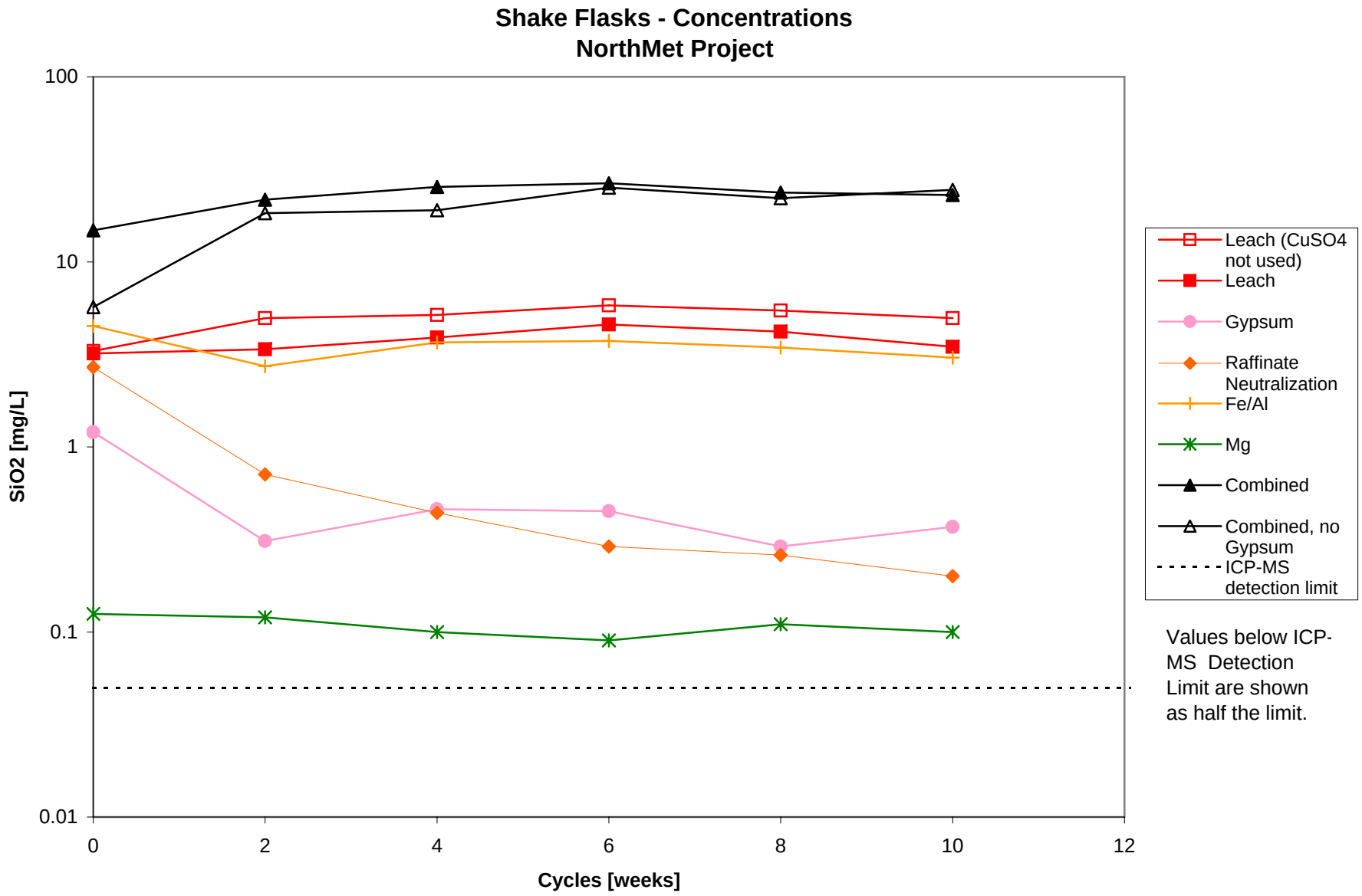

Shake Flasks - Concentrations
NorthMet Project
Mo [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

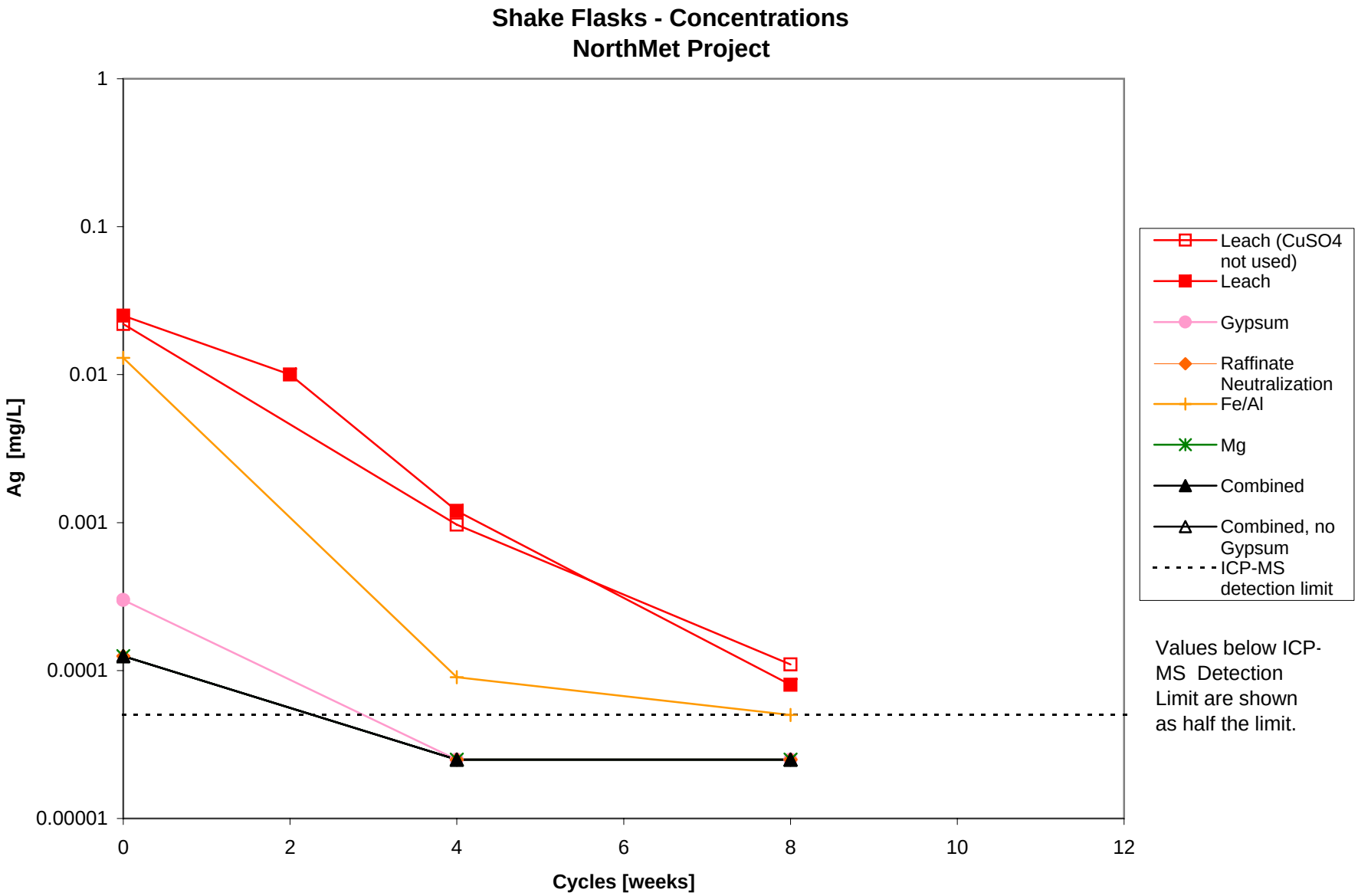

Shake Flasks - Concentrations
NorthMet Project
Ni [mg/L]
1000
100
10
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

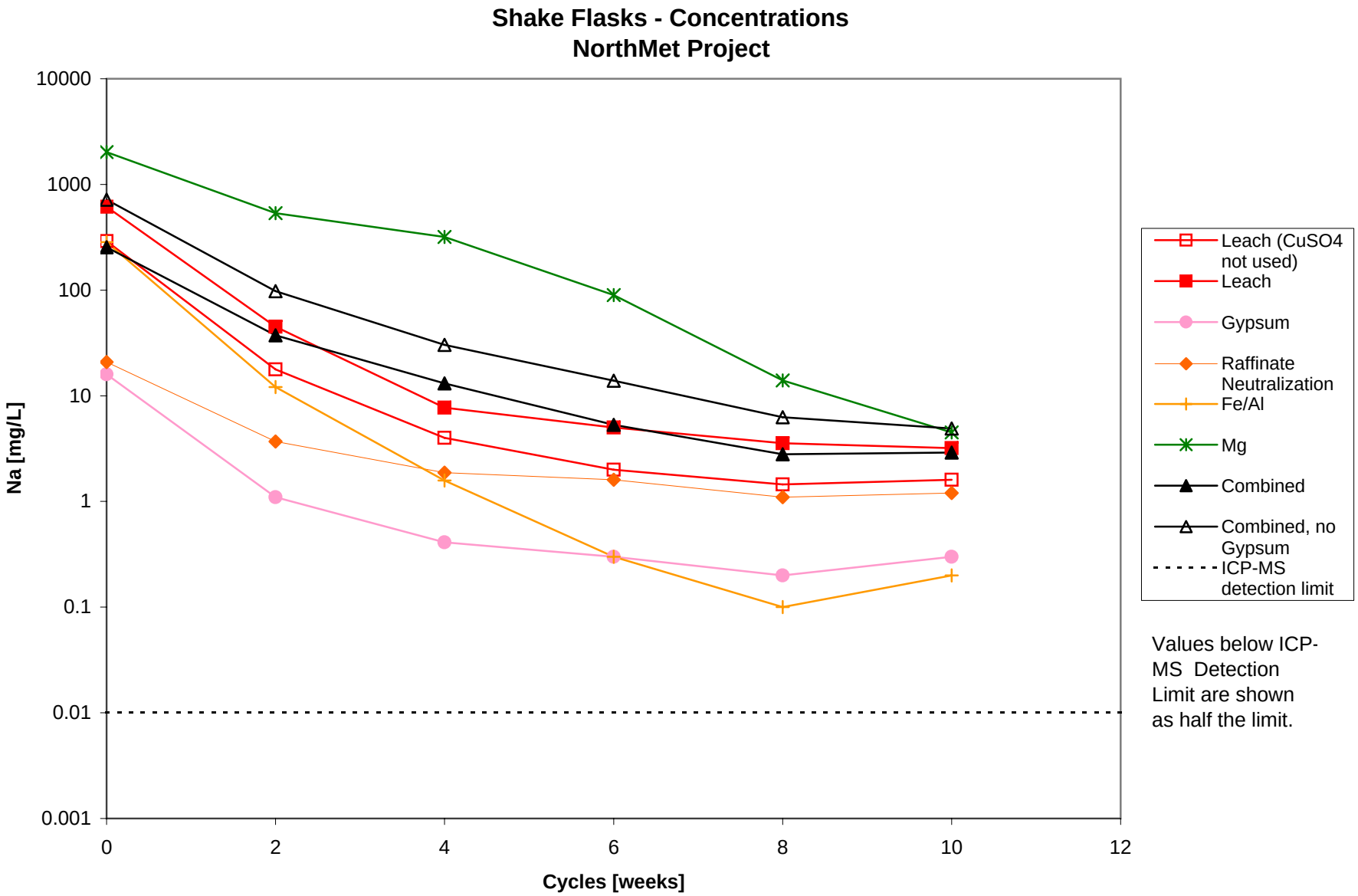

Shake Flasks - Concentrations
NorthMet Project
PO4 [mg/L]
10
1
0.1
0.01
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.


Shake Flasks - Concentrations
NorthMet Project
K [mg/L]
100
10
1
0.1
0.01
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

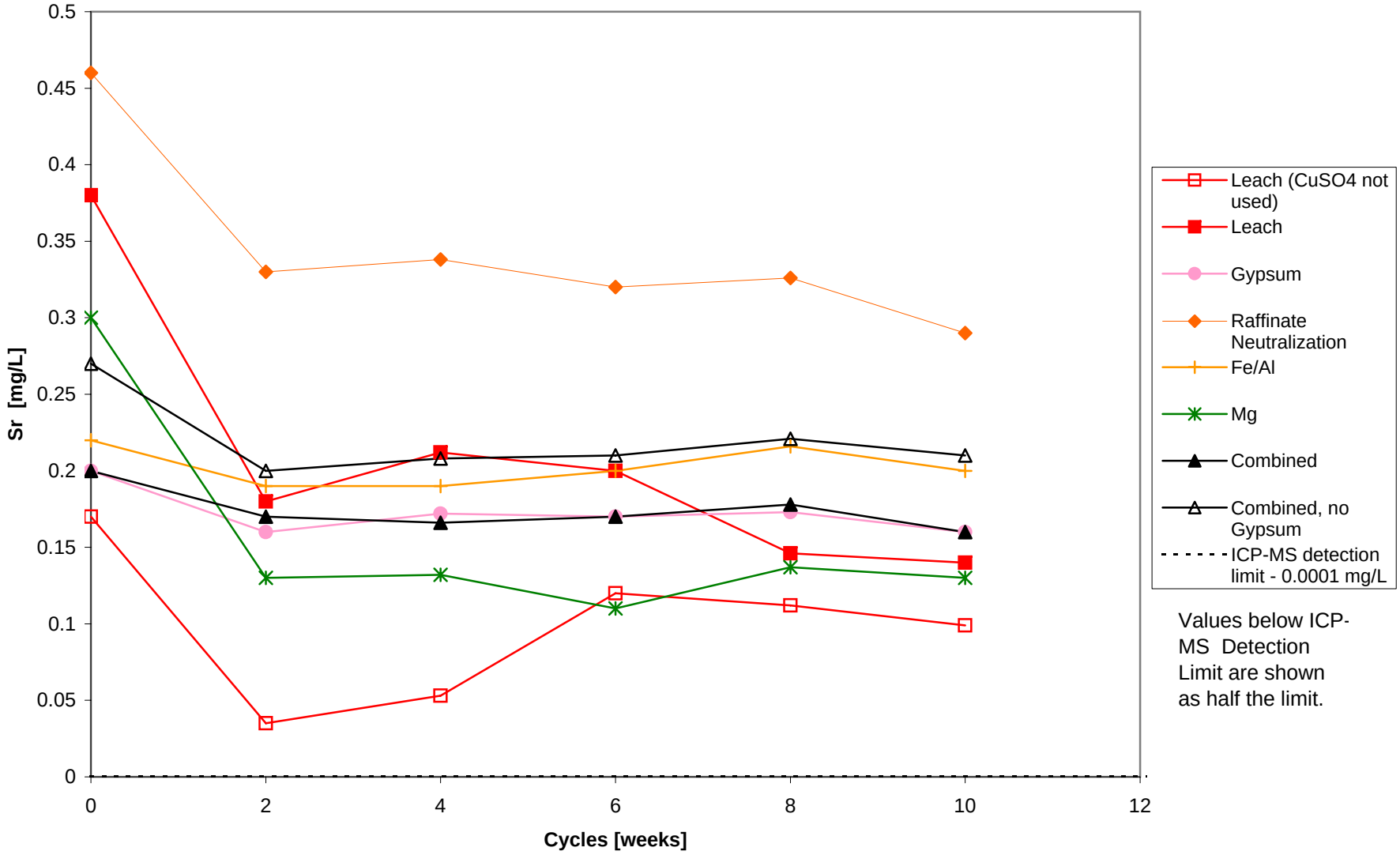

Shake Flasks - Concentrations
NorthMet Project
0.1
0.01
0.001
0.0001
Se [mg/L]
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.


Shake Flasks - Concentrations
NorthMet Project
SiO2 [mg/L]
100
10
1
0.1
0.01
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.


Shake Flasks - Concentrations
NorthMet Project
Ag [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

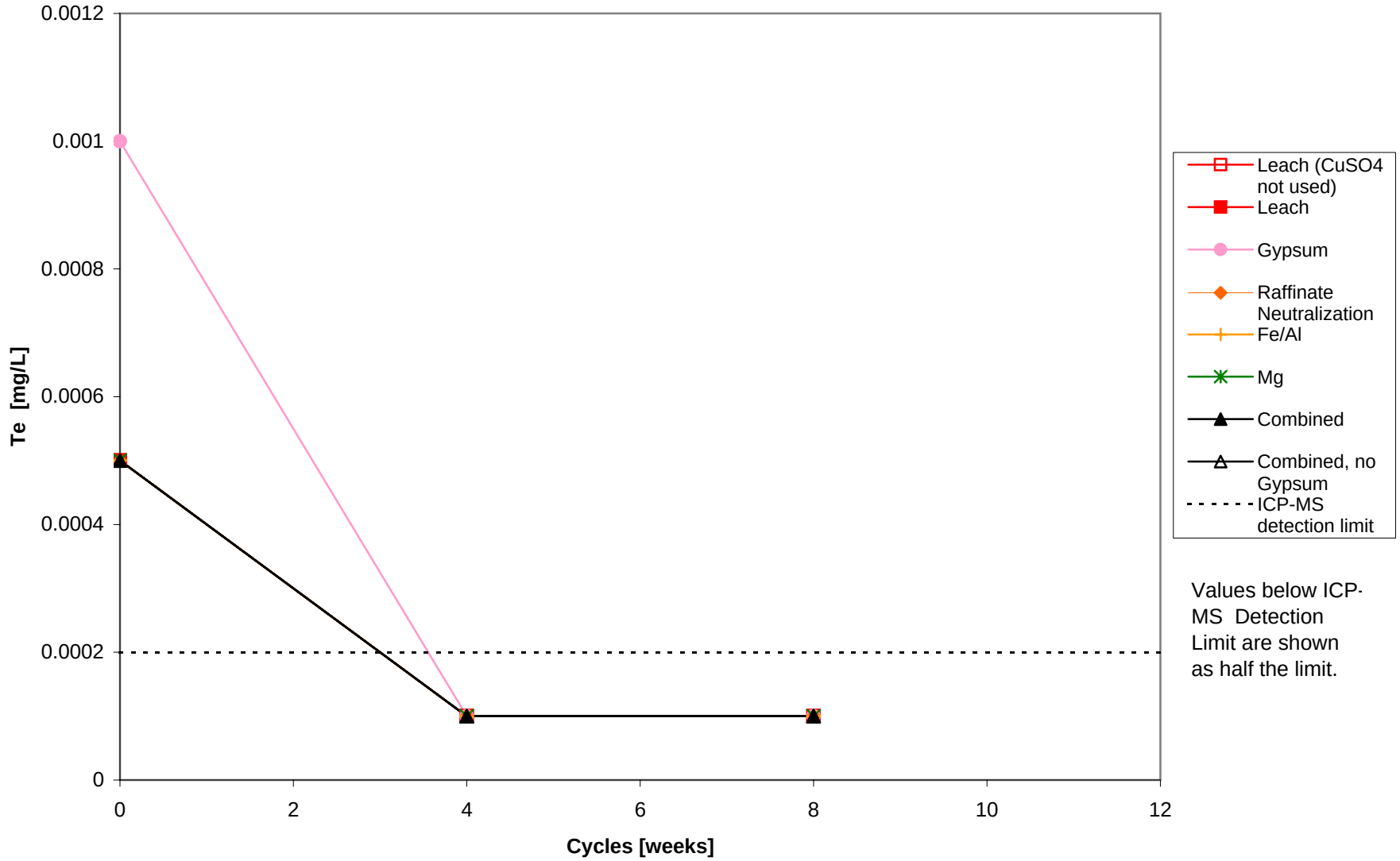

Shake Flasks - Concentrations
NorthMet Project
Na [mg/L]
10000
1000
100
10
1
0.1
0.01
0.001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

Shake Flasks - Concentrations
NorthMet Project

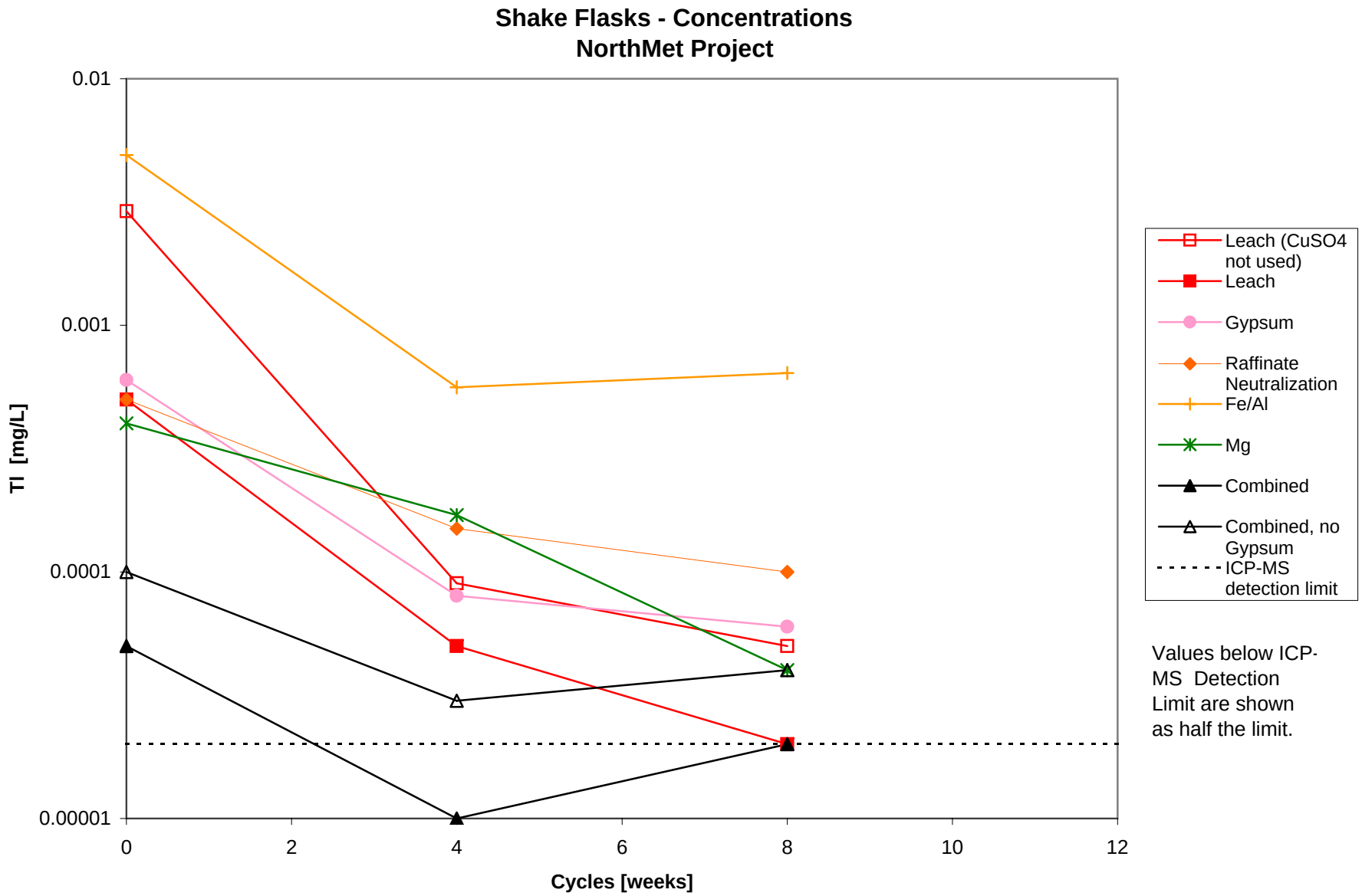


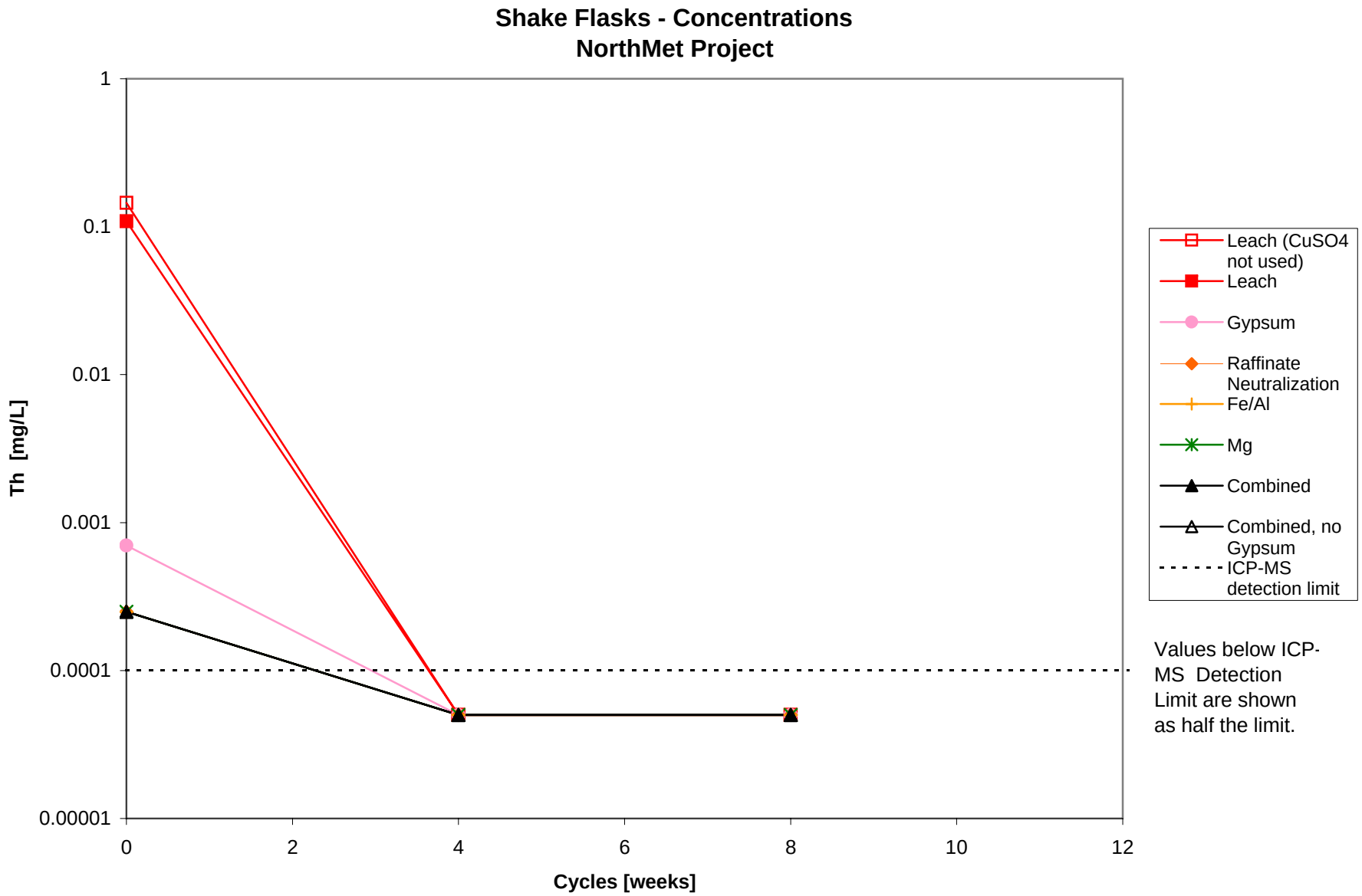
Values below ICP-MS Detection Limit are shown as half the limit.

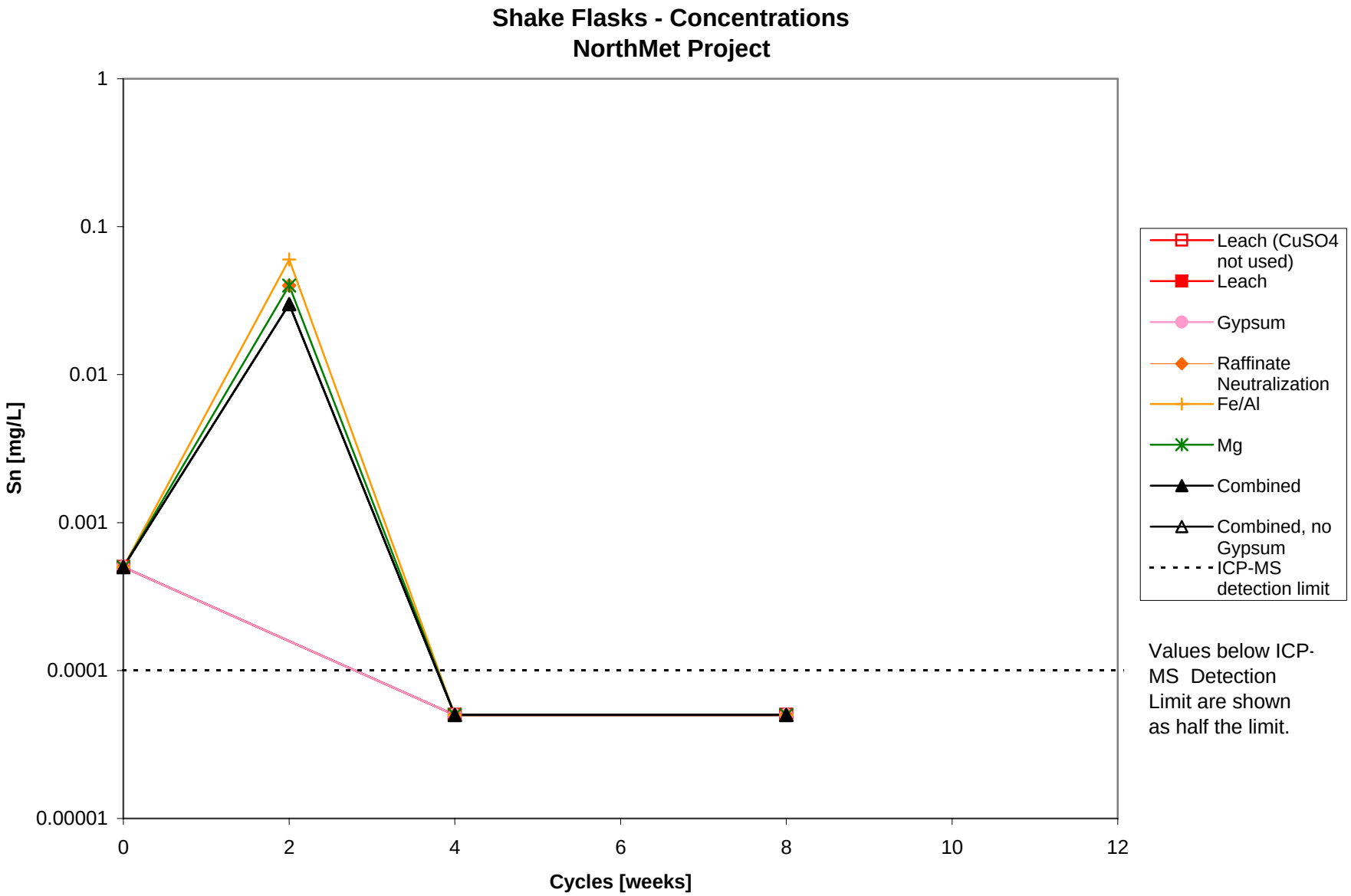
Shake Flasks - Concentrations
NorthMet Project

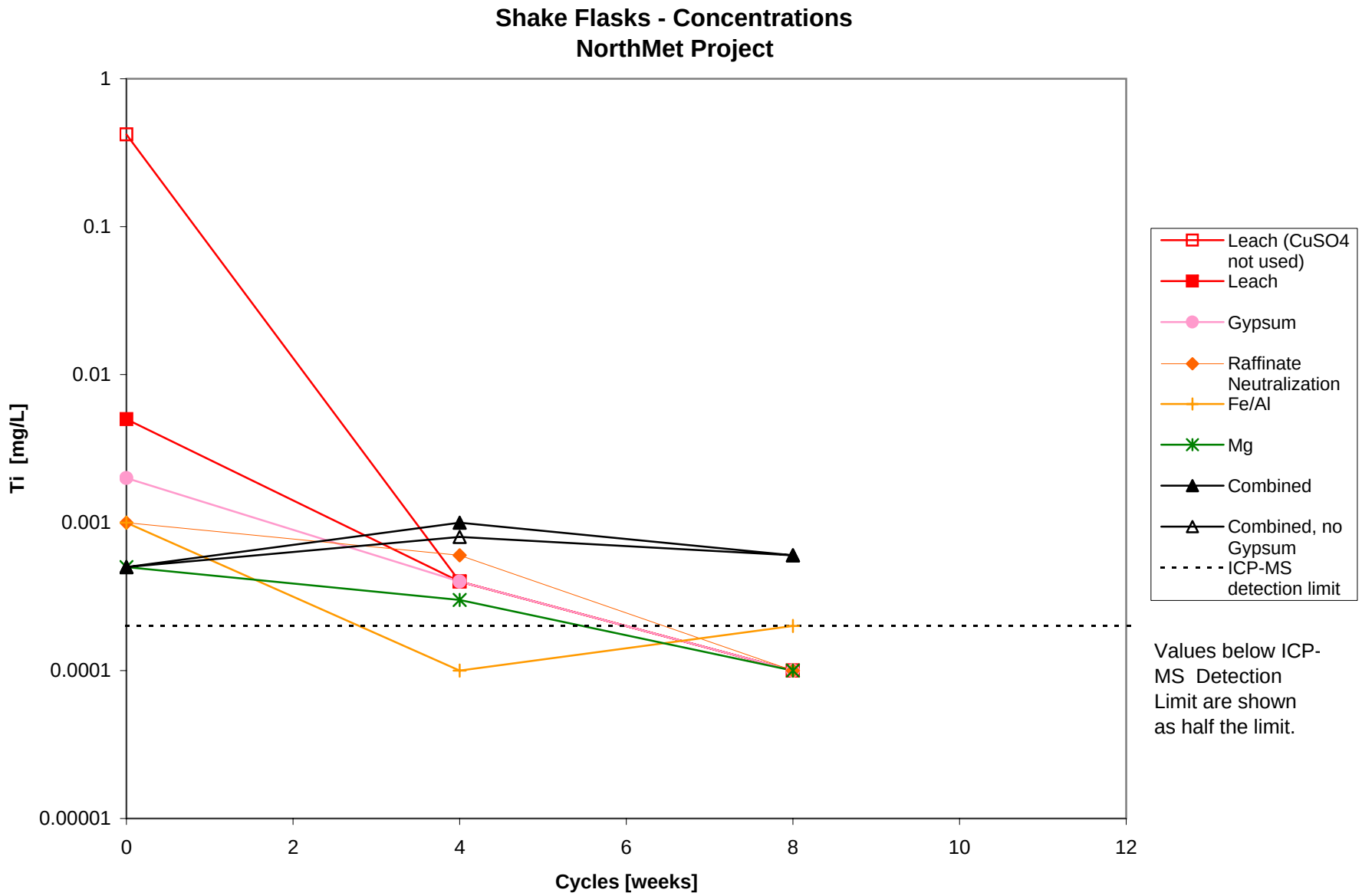


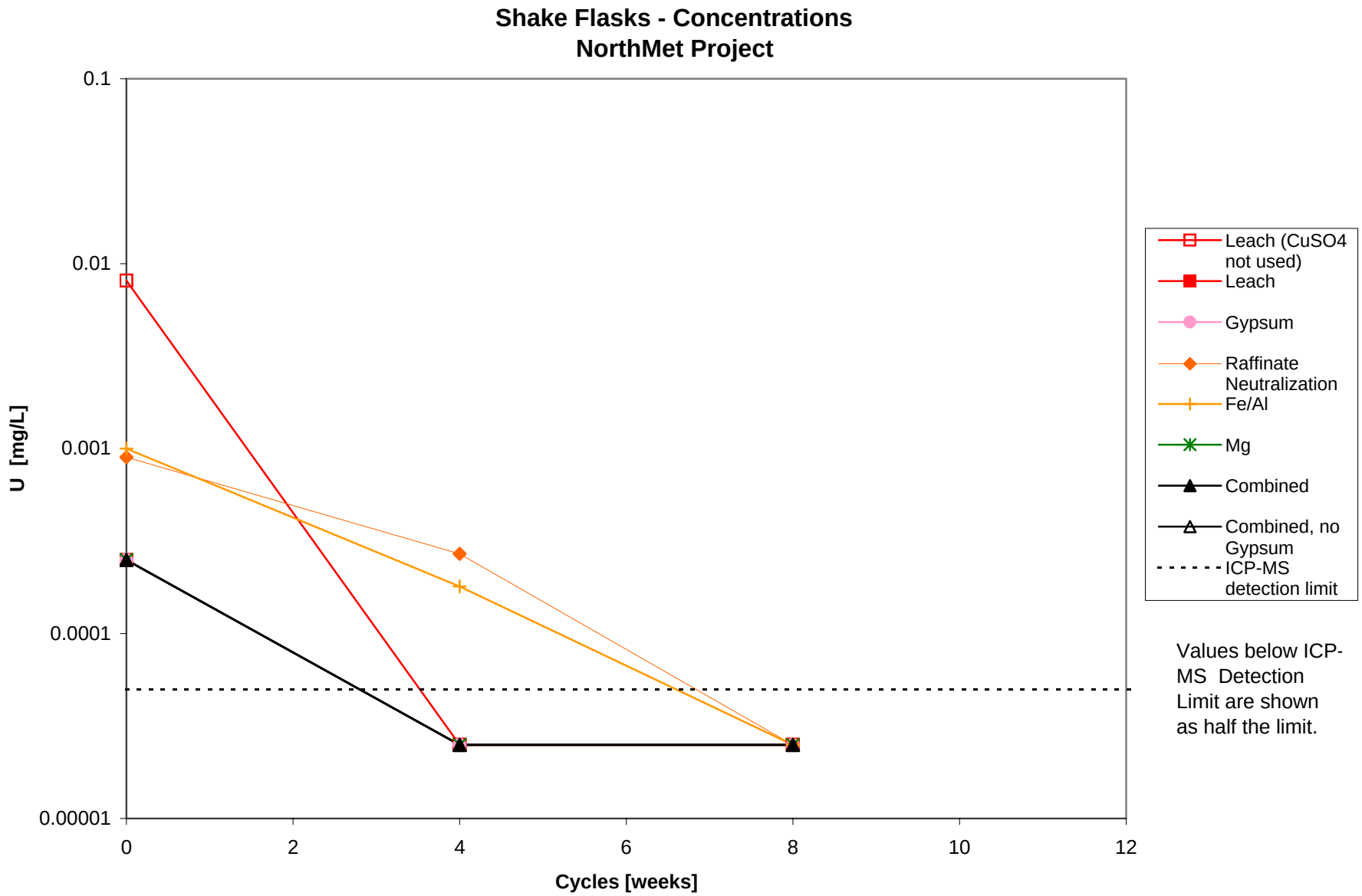
Values below ICP-MS Detection Limit are shown as half the limit.

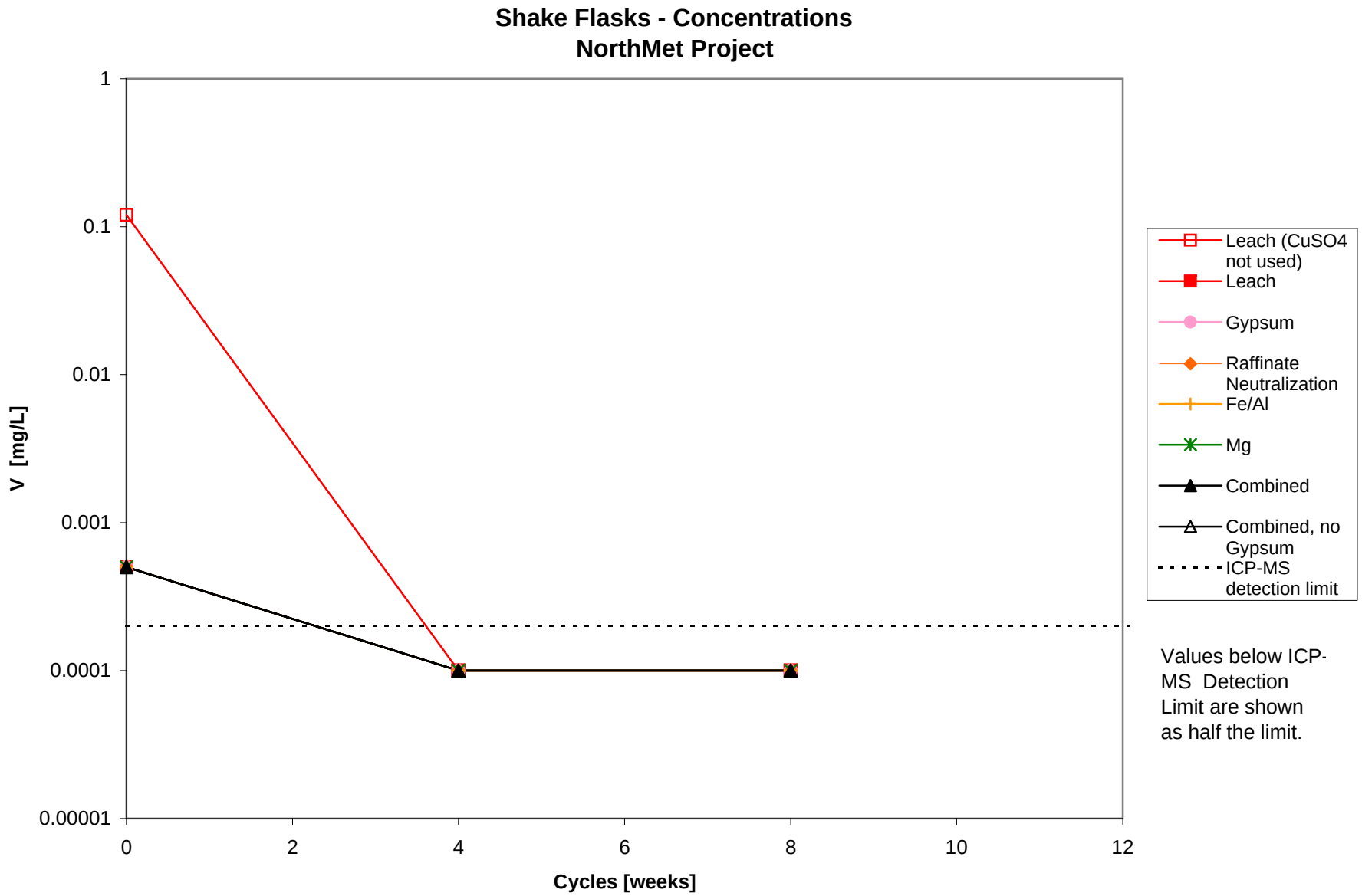

Shake Flasks - Concentrations
NorthMet Project
0.01
0.001
0.0001
0.00001
Tl [mg/L]
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

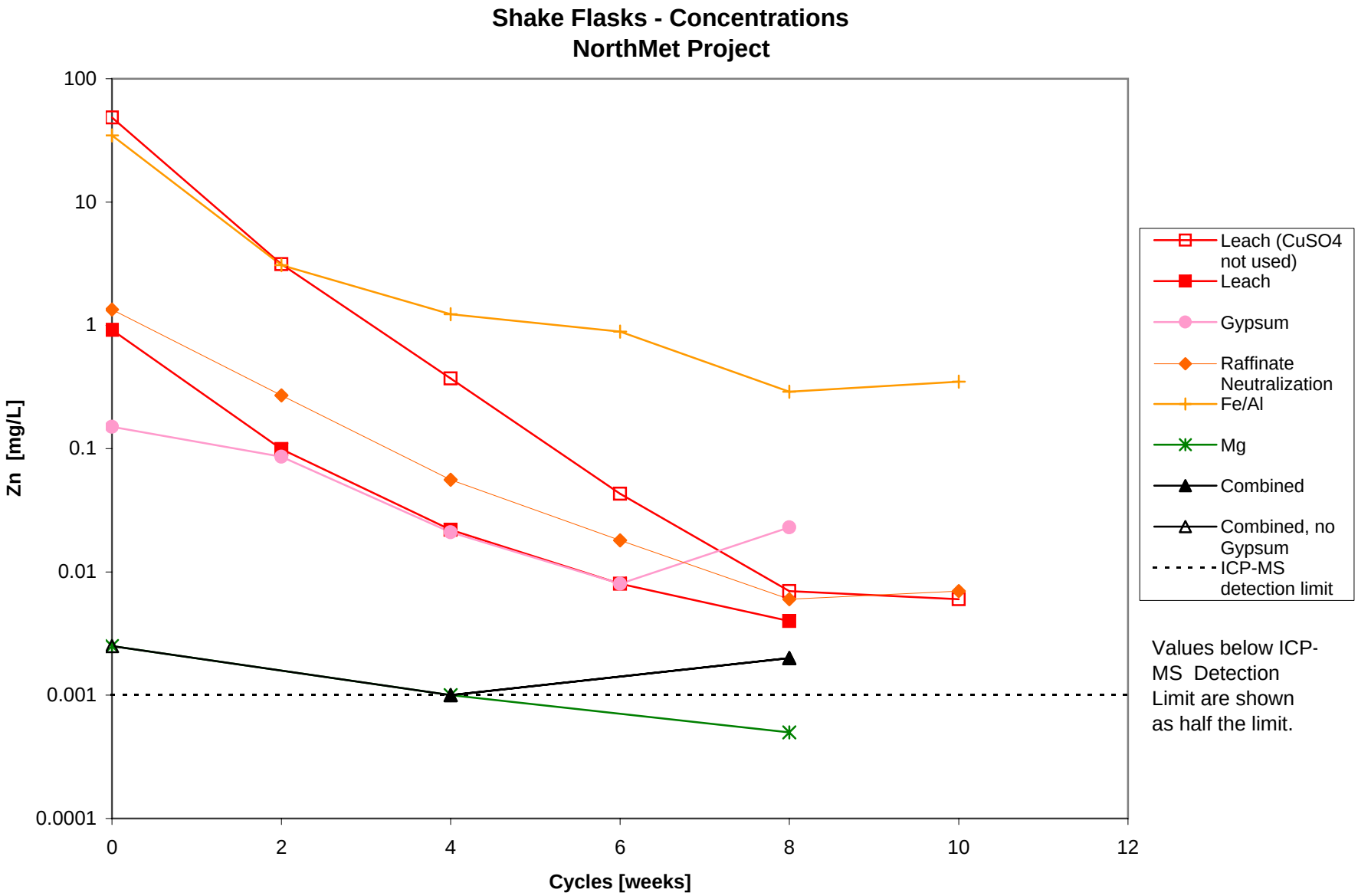

Shake Flasks - Concentrations
NorthMet Project
Th [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

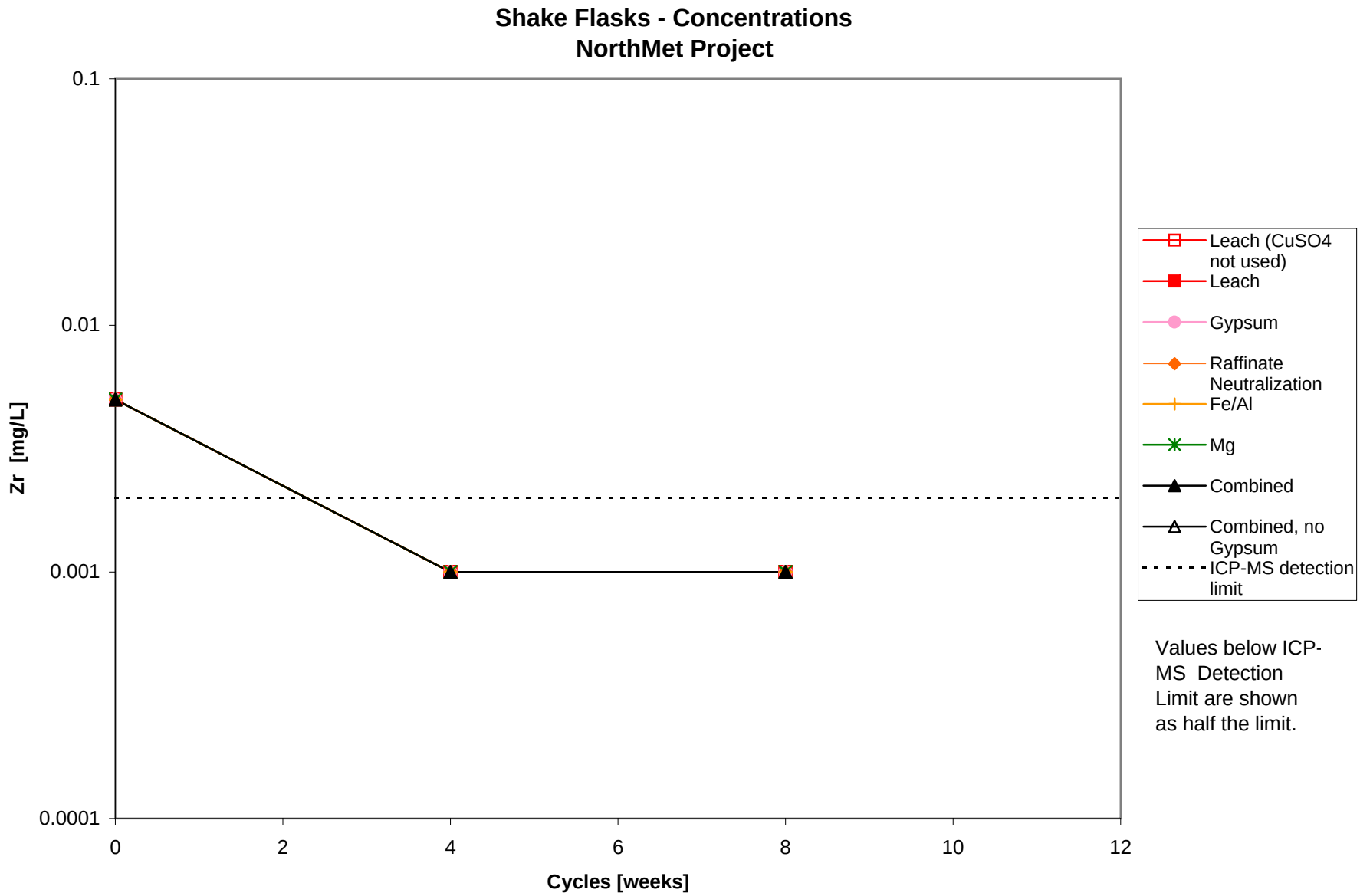

Shake Flasks - Concentrations
NorthMet Project
Sn [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.


Shake Flasks - Concentrations
NorthMet Project
Ti [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

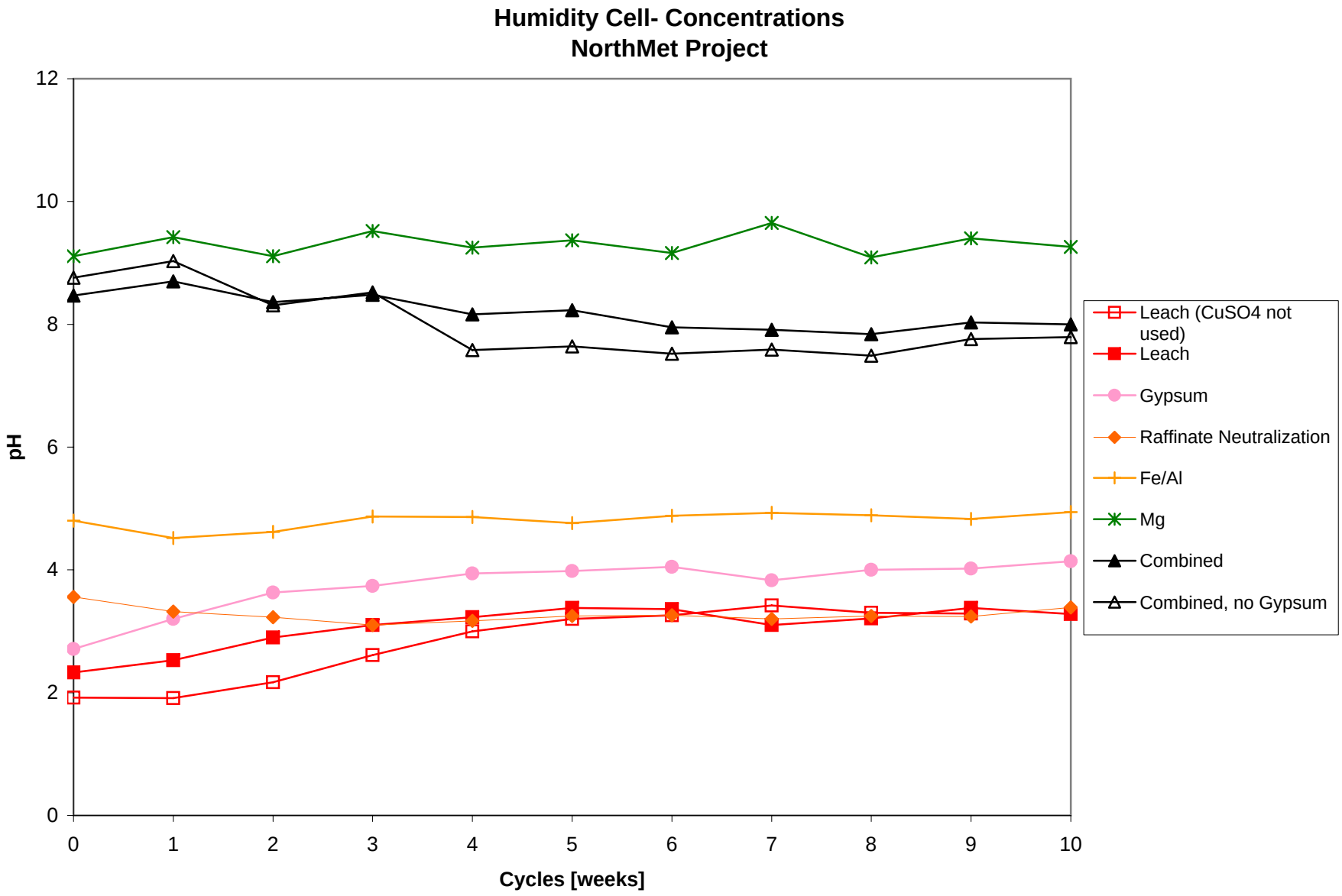

Shake Flasks - Concentrations
NorthMet Project
U [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

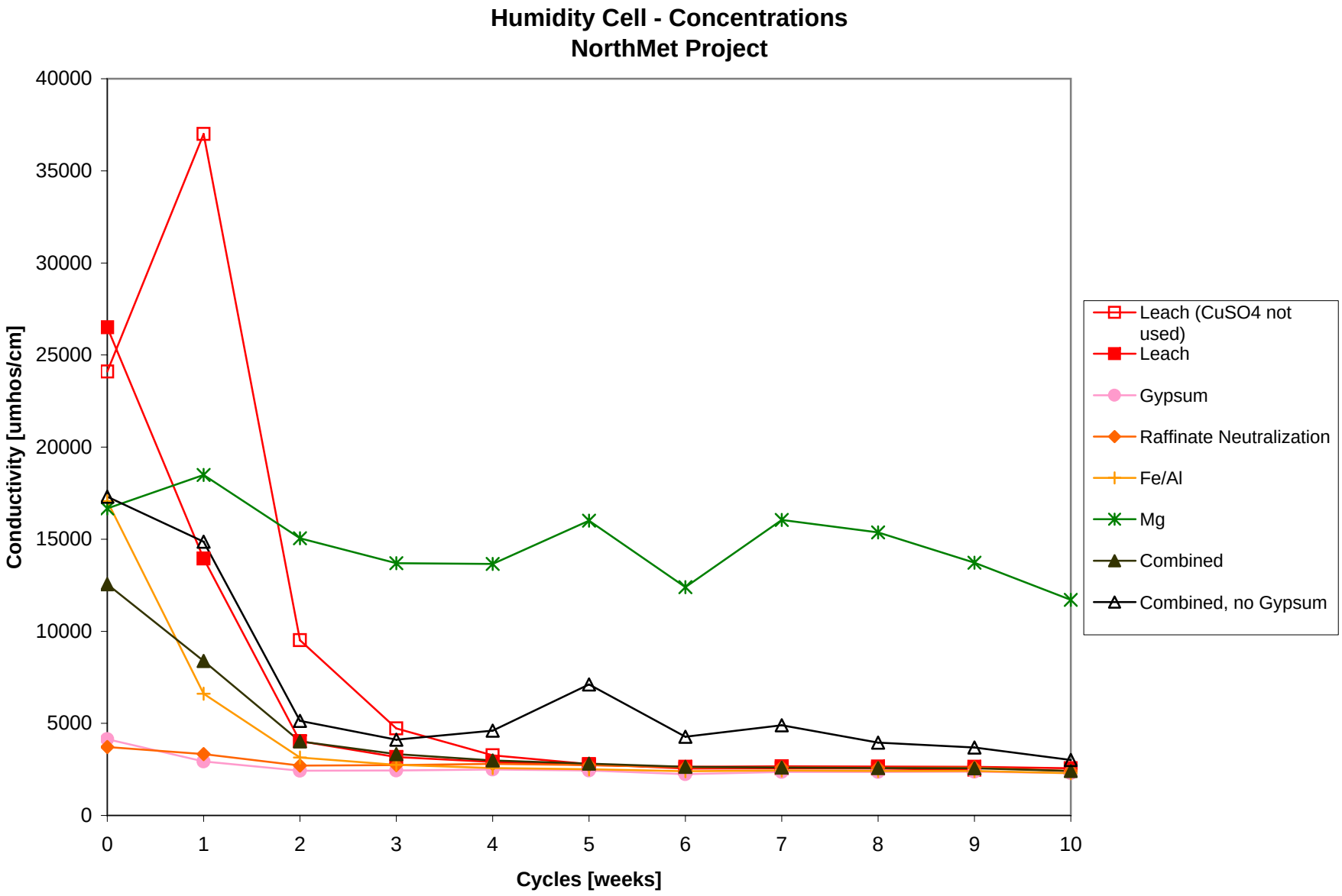

Shake Flasks - Concentrations
NorthMet Project
V [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

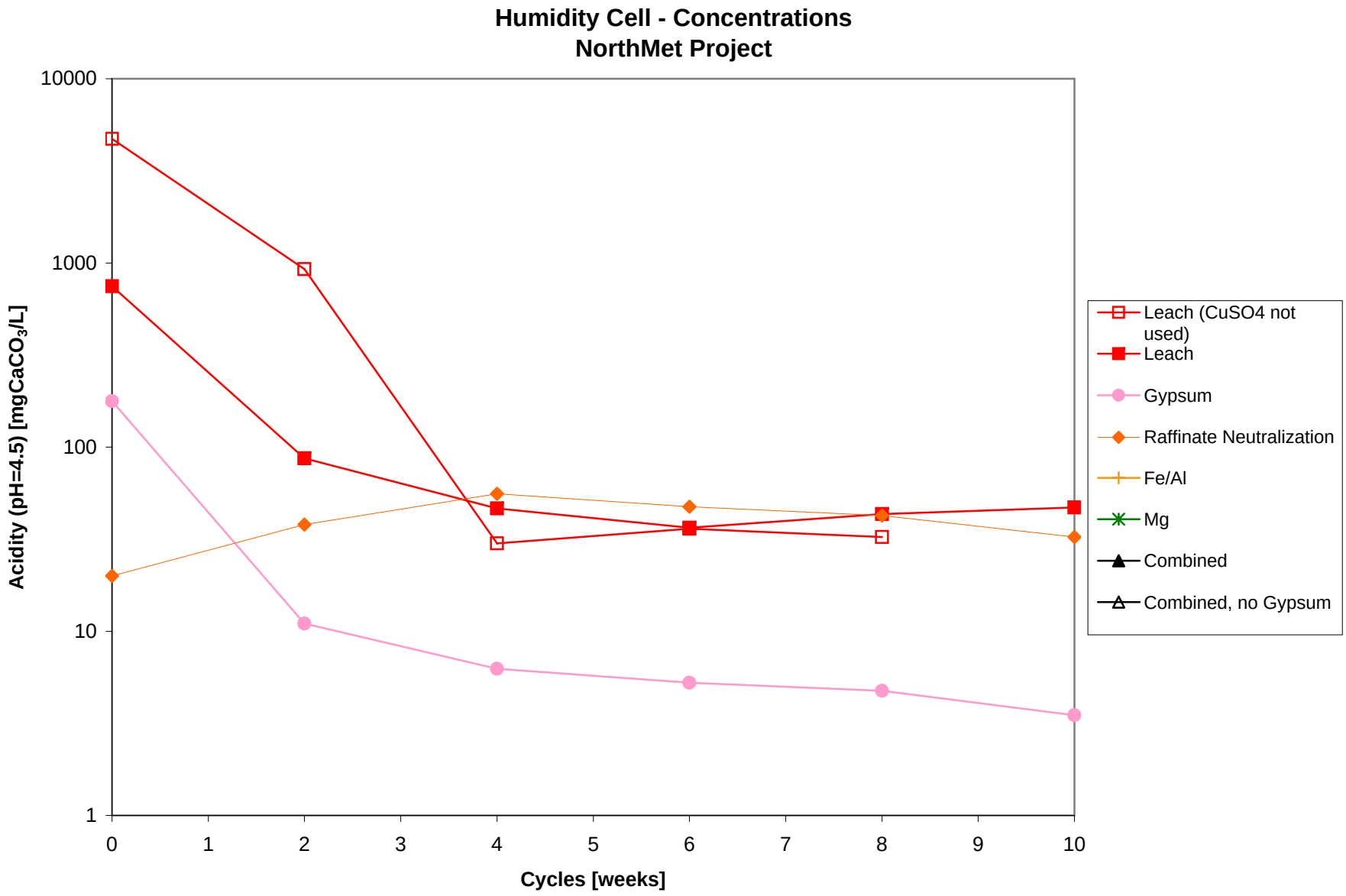

Shake Flasks - Concentrations
NorthMet Project
Zn [mg/L]
100
10
1
0.1
0.01
0.001
0.0001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

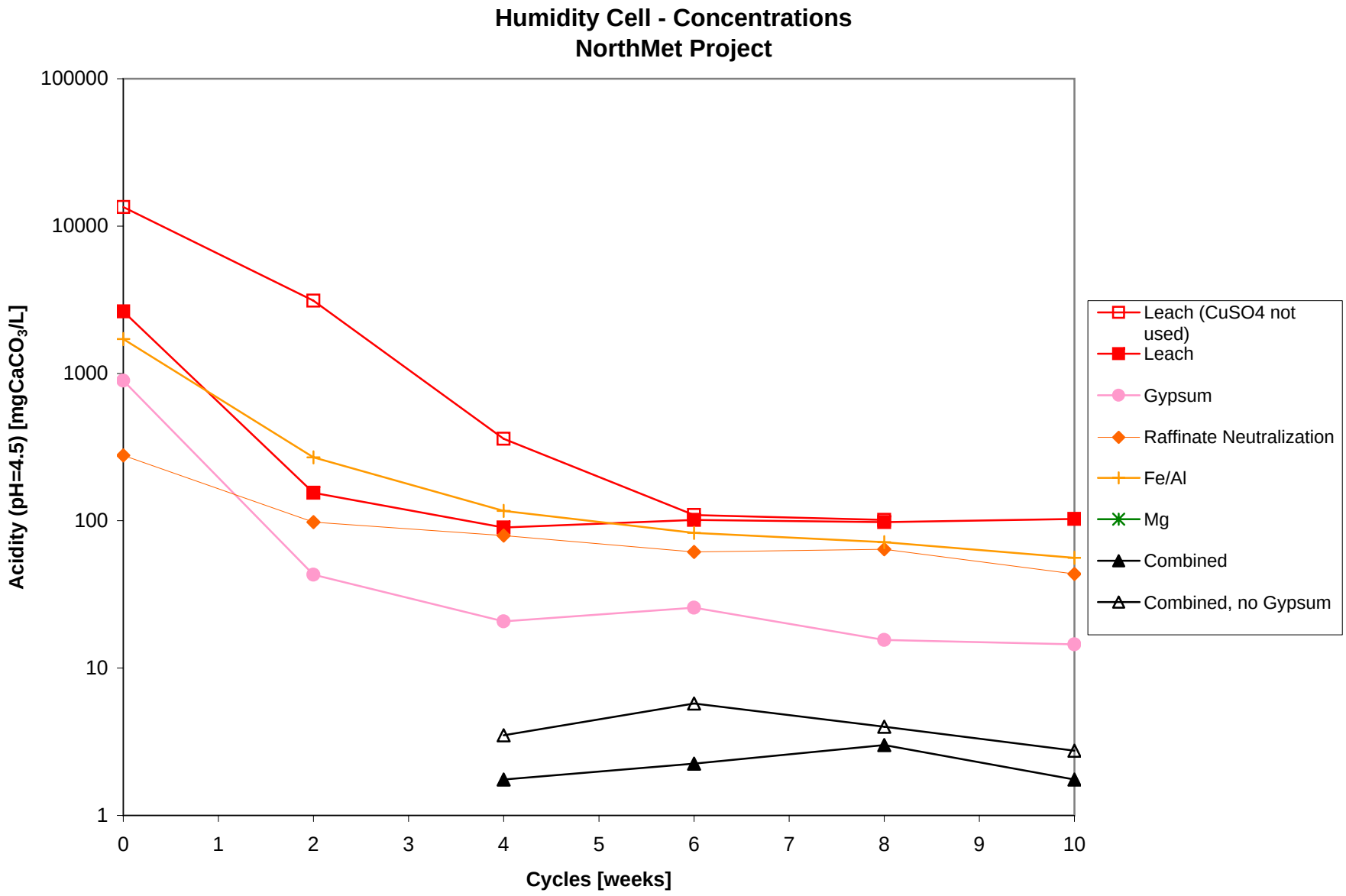

Shake Flasks - Concentrations
NorthMet Project
0.1
0.01
0.001
0.0001
Zr [mg/L]
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

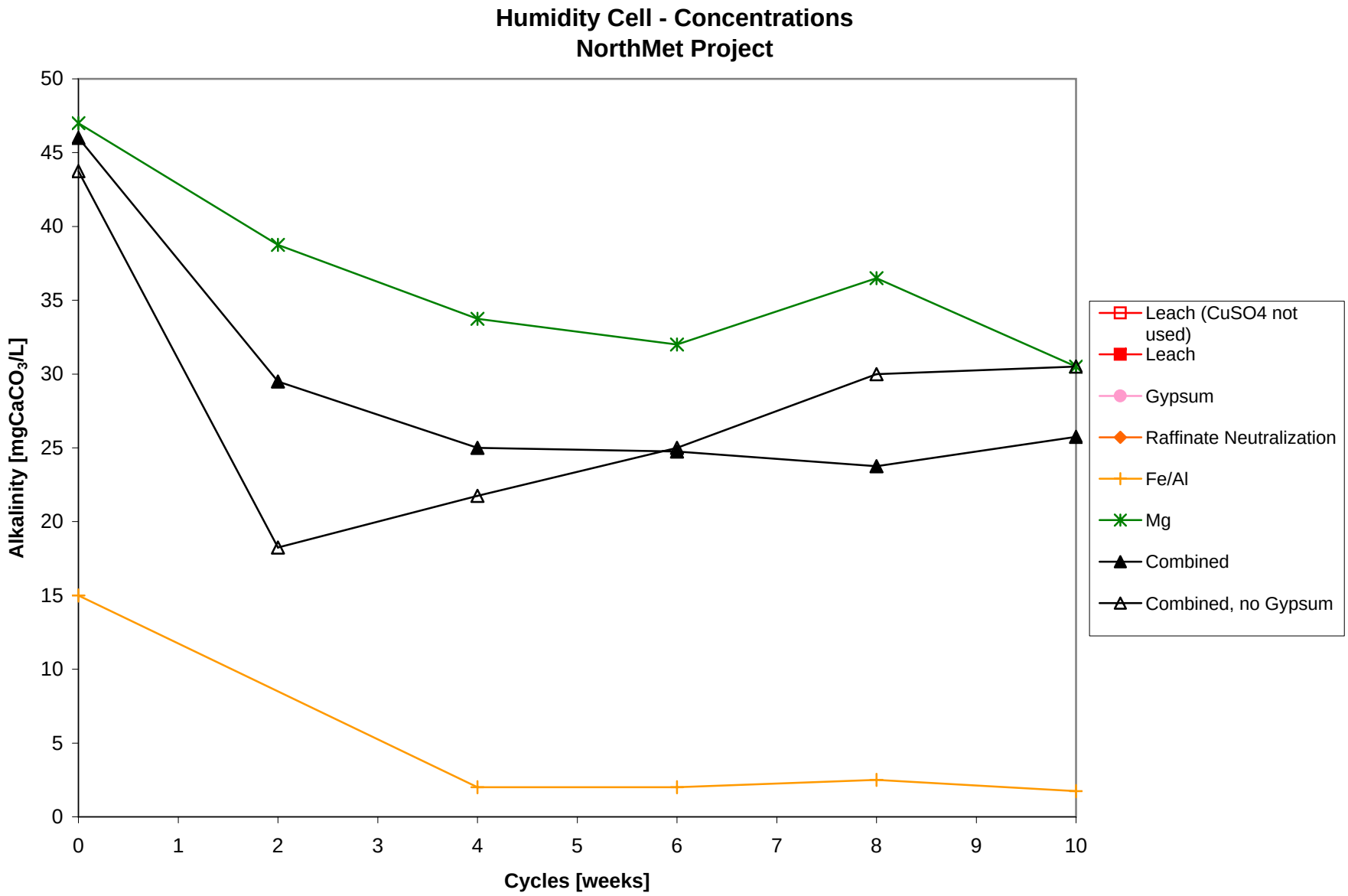
Appendix C.2
Humidity Cells

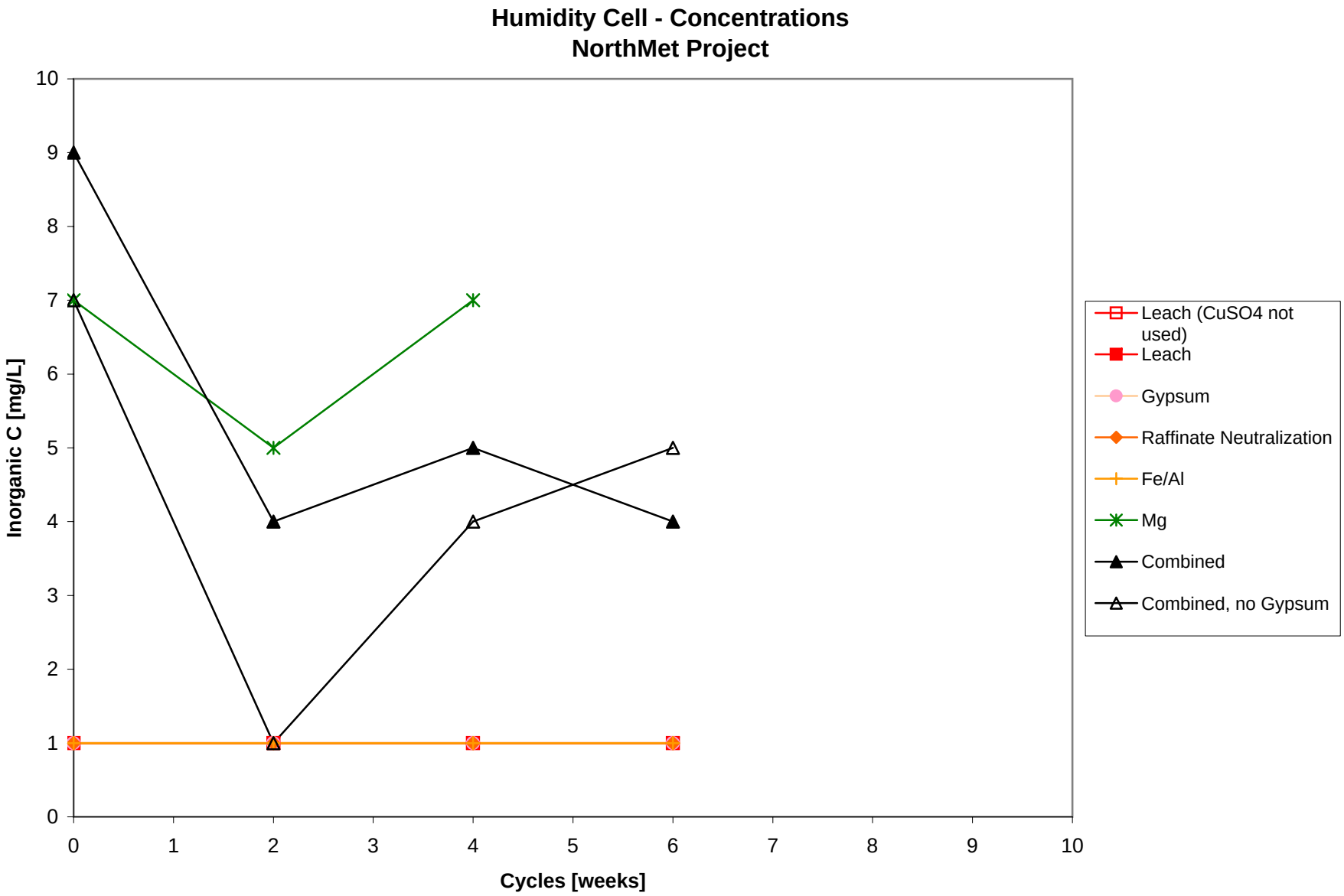

Humidity Cell- Concentrations
NorthMet Project
pH
Cycles [weeks]
0
2
4
6
8
10
12
0
1
2
3
4
5
6
7
8
9
10
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

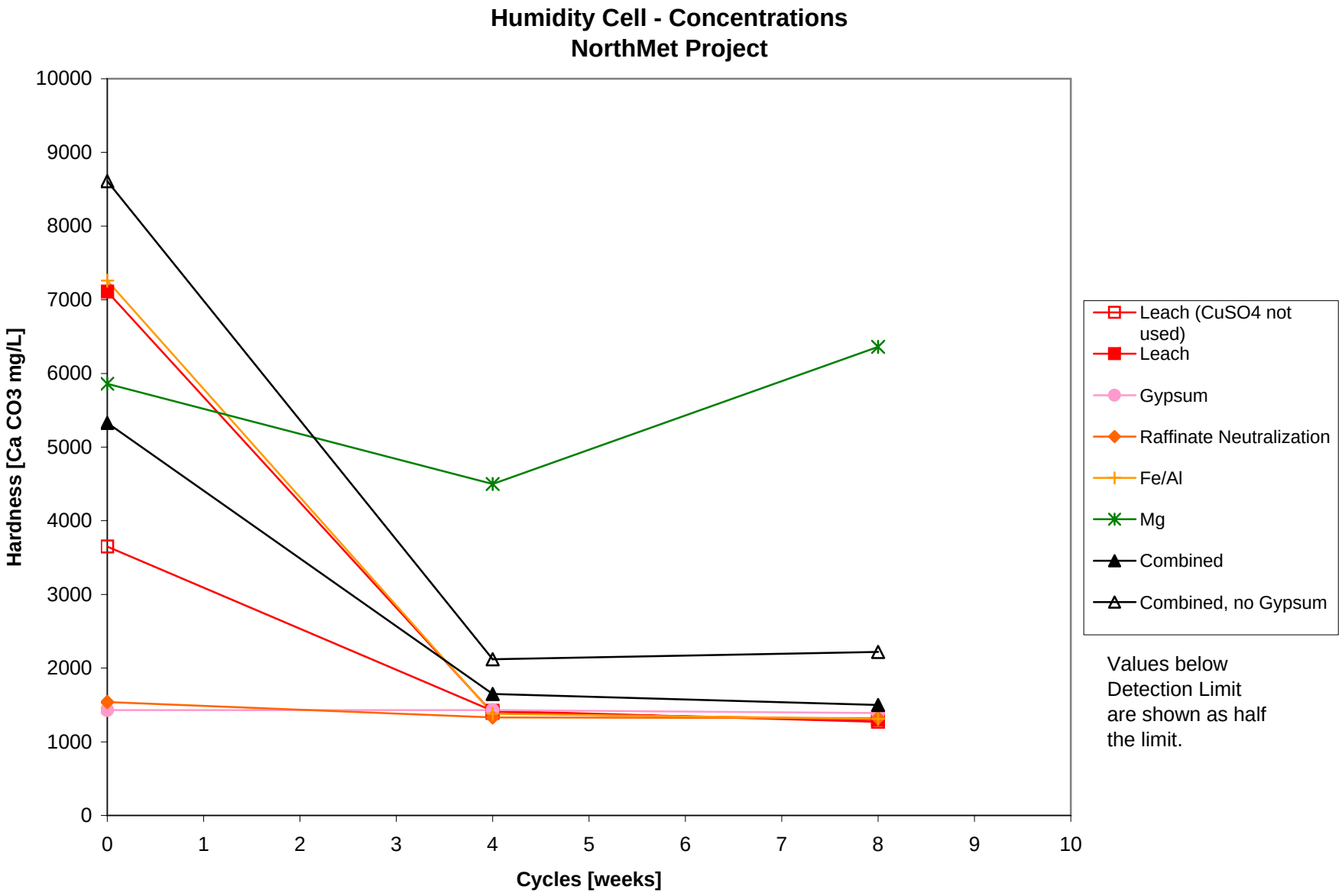

Humidity Cell - Concentrations
NorthMet Project
Conductivity [umhos/cm]
40000
35000
30000
25000
20000
15000
10000
5000
0
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

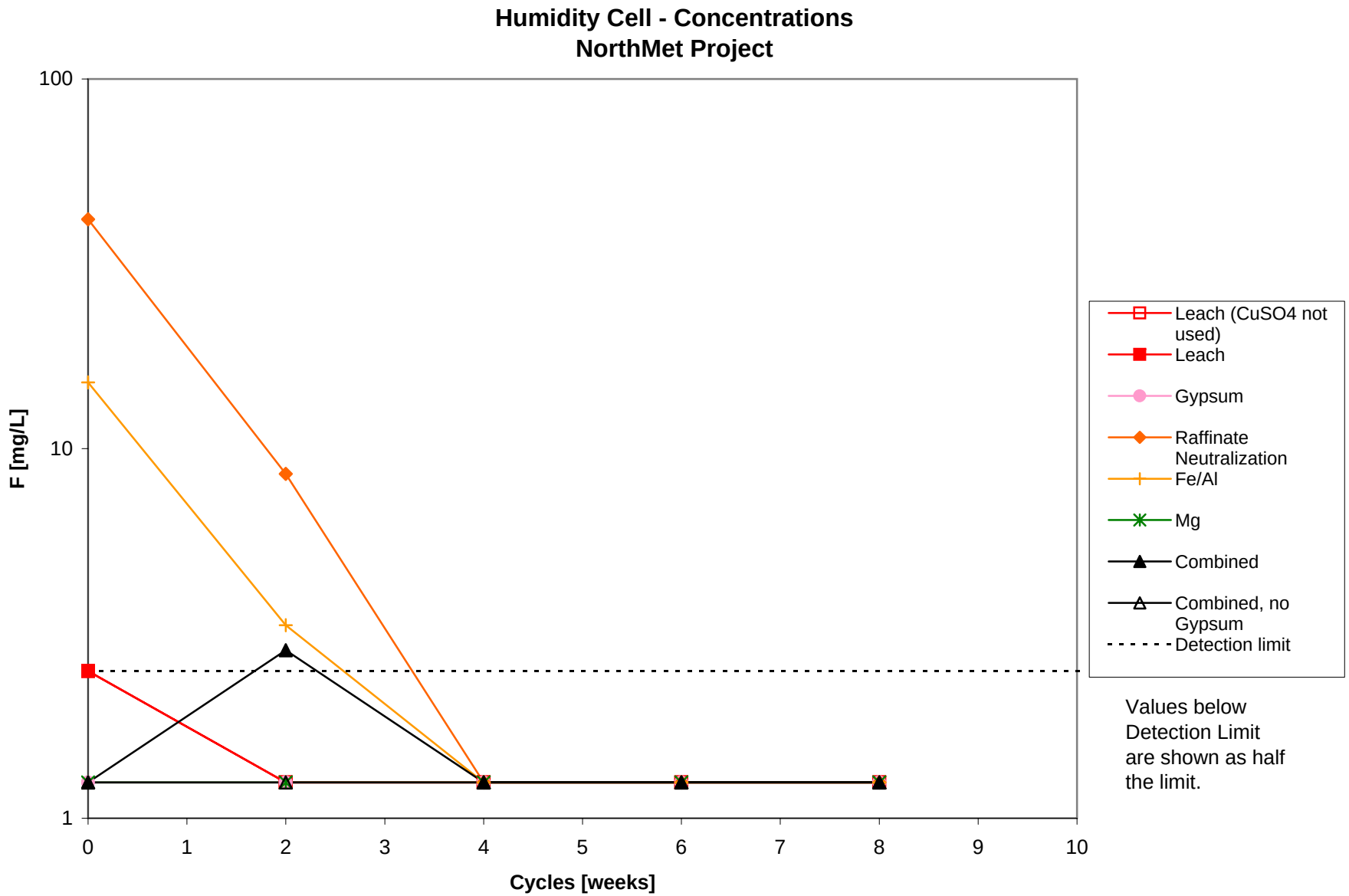

Humidity Cell - Concentrations
NorthMet Project
Acidity (pH=4.5) [mgCaCO3/L]
10000
1000
100
10
1
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

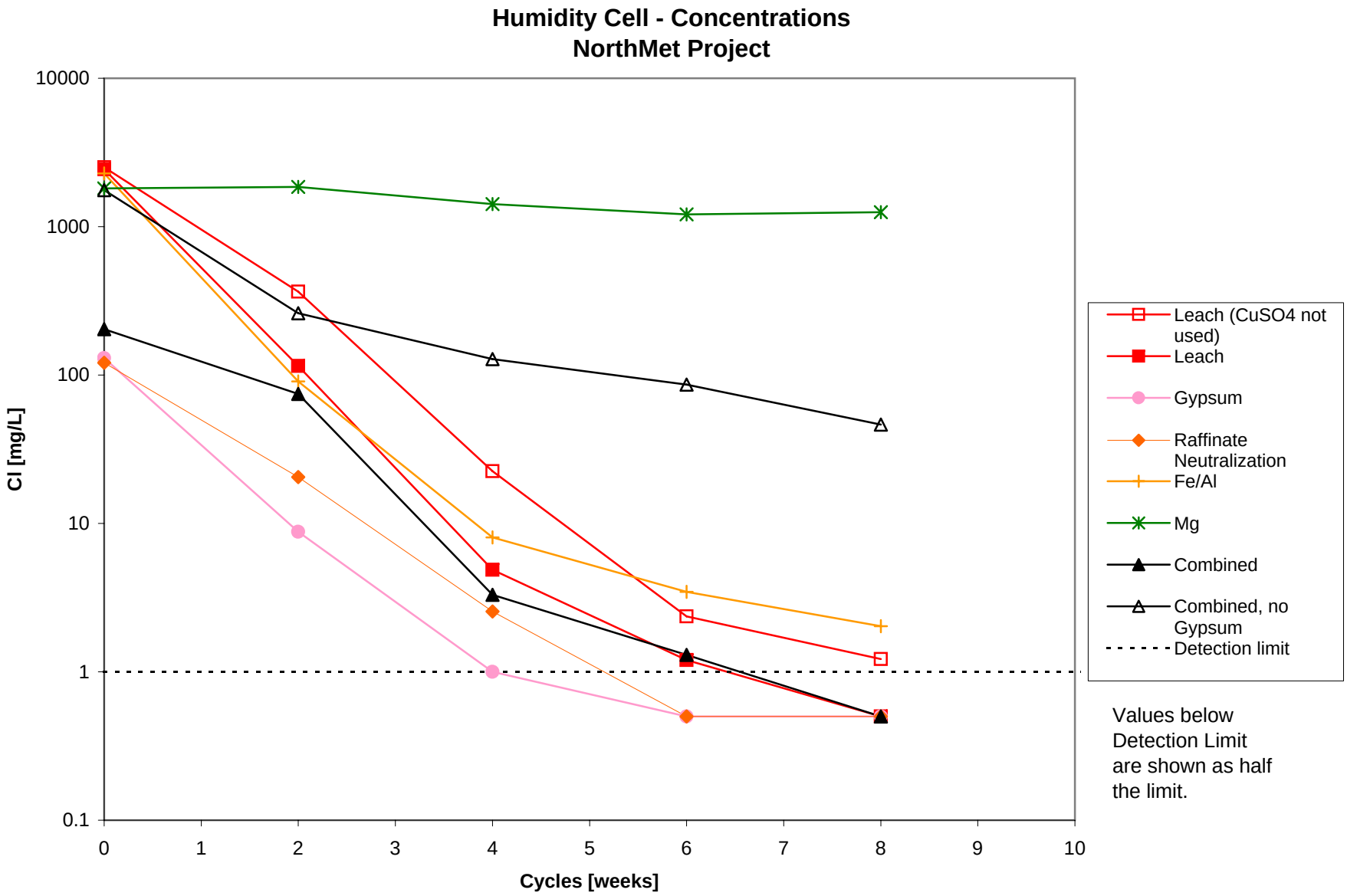

Humidity Cell - Concentrations
NorthMet Project
Acidity (pH=4.5) [mgCaCO3/L]
100000
10000
1000
100
10
1
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

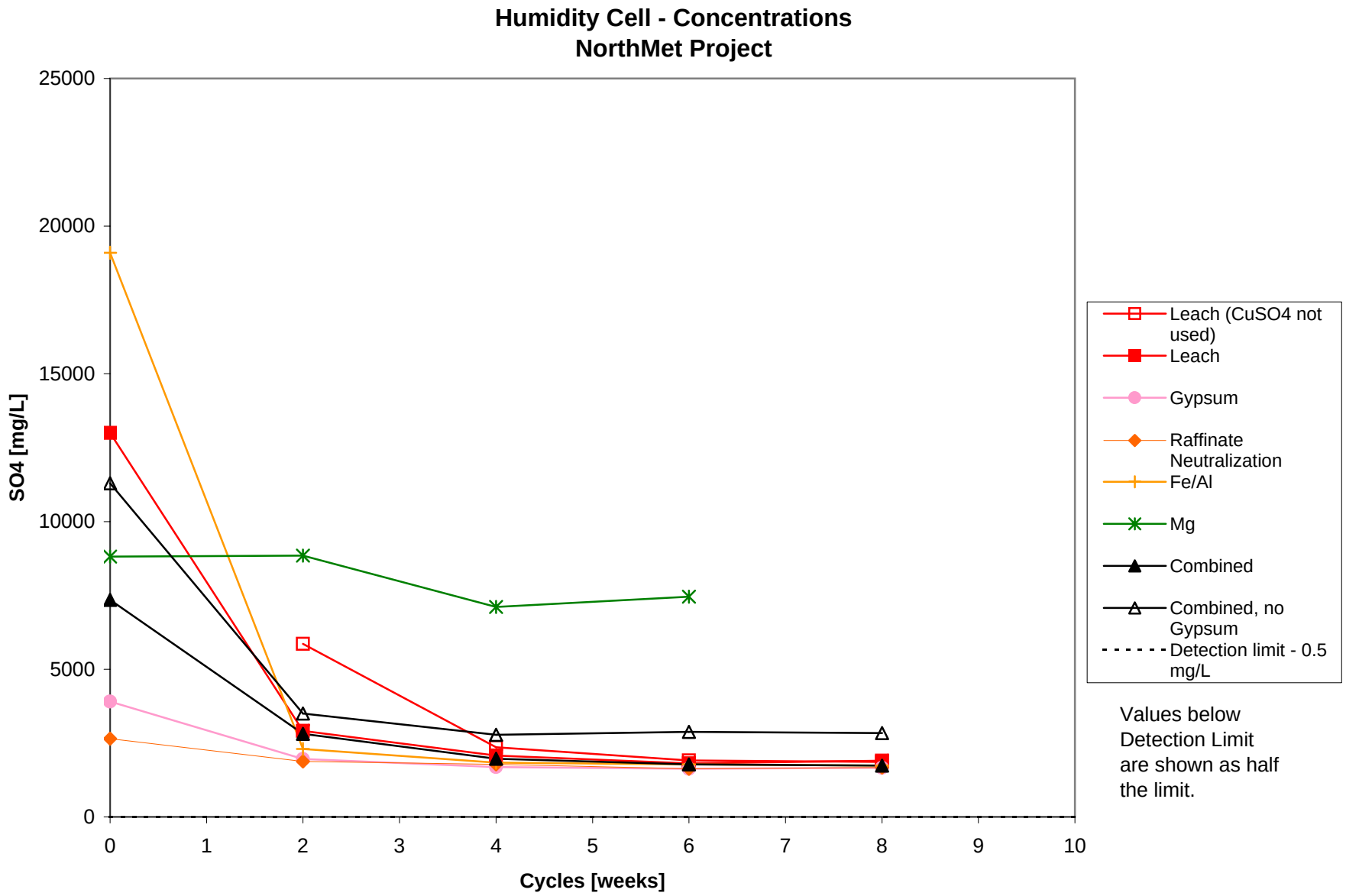

Humidity Cell - Concentrations
NorthMet Project
Alkalinity [mgCaCO3/L]
Cycles [weeks]
0
5
10
15
20
25
30
35
40
45
50
0
1
2
3
4
5
6
7
8
9
10
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

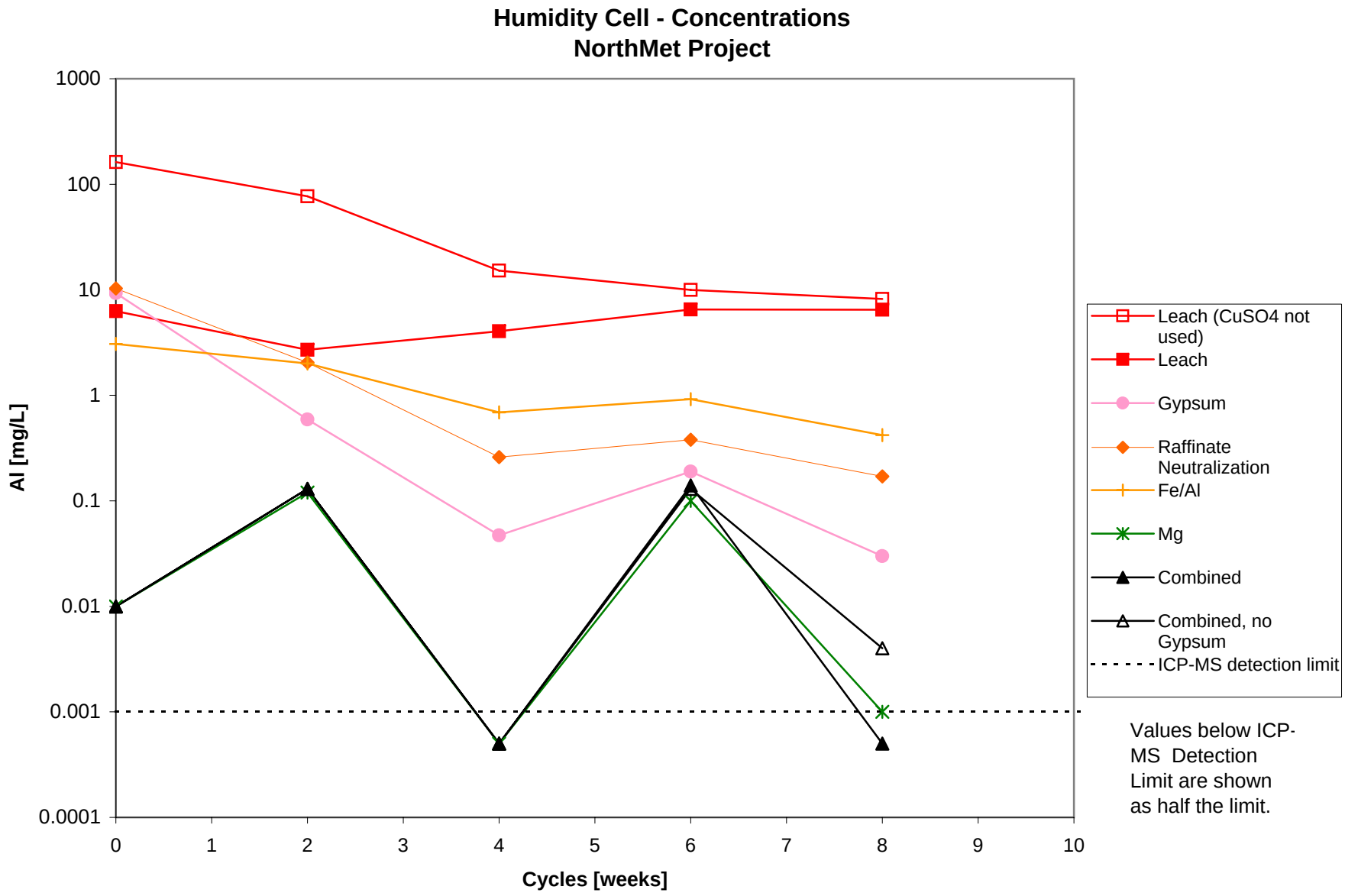

Humidity Cell - Concentrations
NorthMet Project
Inorganic C [mg/L]
Cycles [weeks]
0
1
2
3
4
5
6
7
8
9
10
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

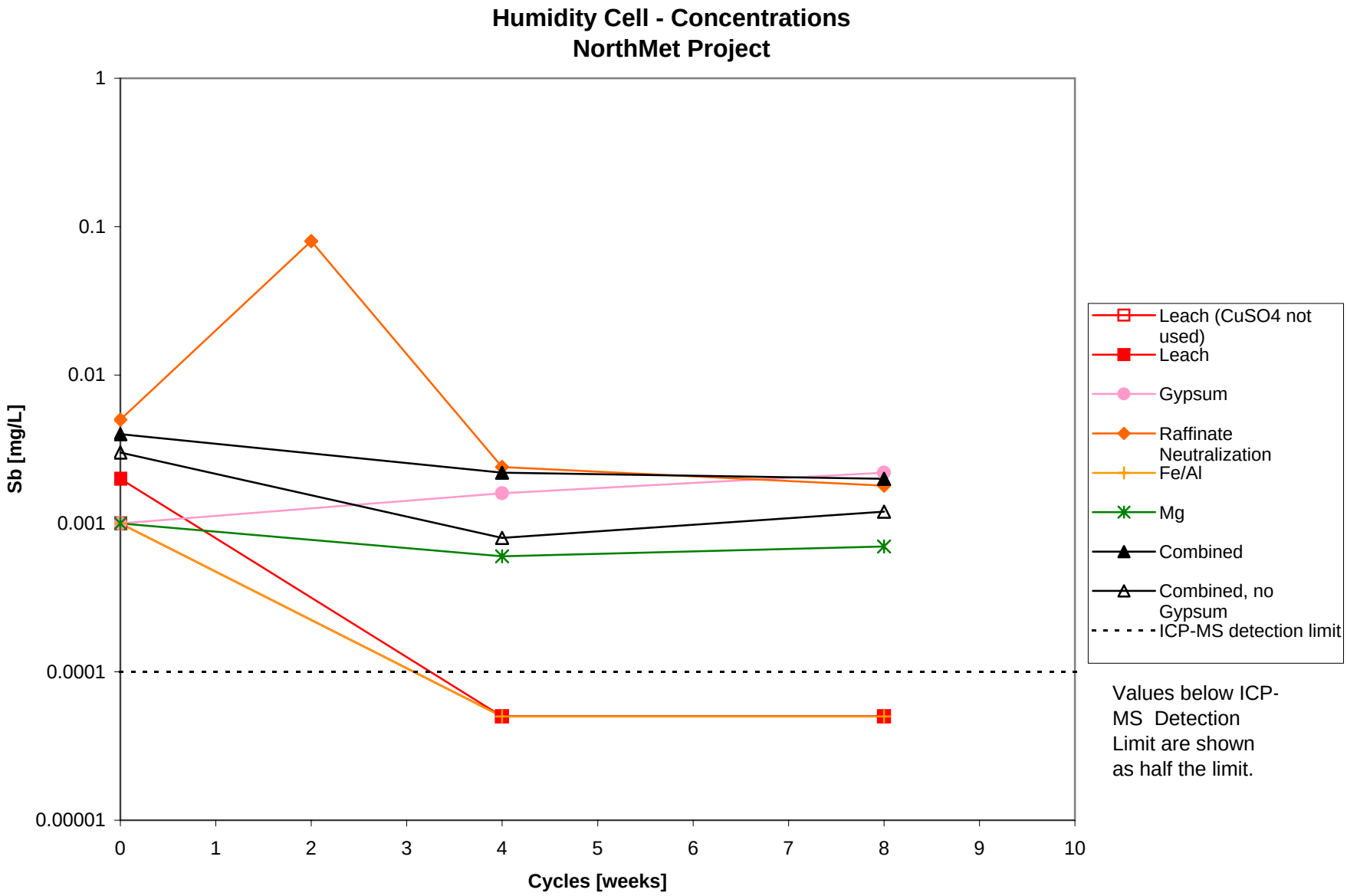

Humidity Cell - Concentrations
NorthMet Project
Hardness [Ca CO3 mg/L]
10000
9000
8000
7000
6000
5000
4000
3000
2000
1000
0
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
Values below Detection Limit are shown as half the limit.

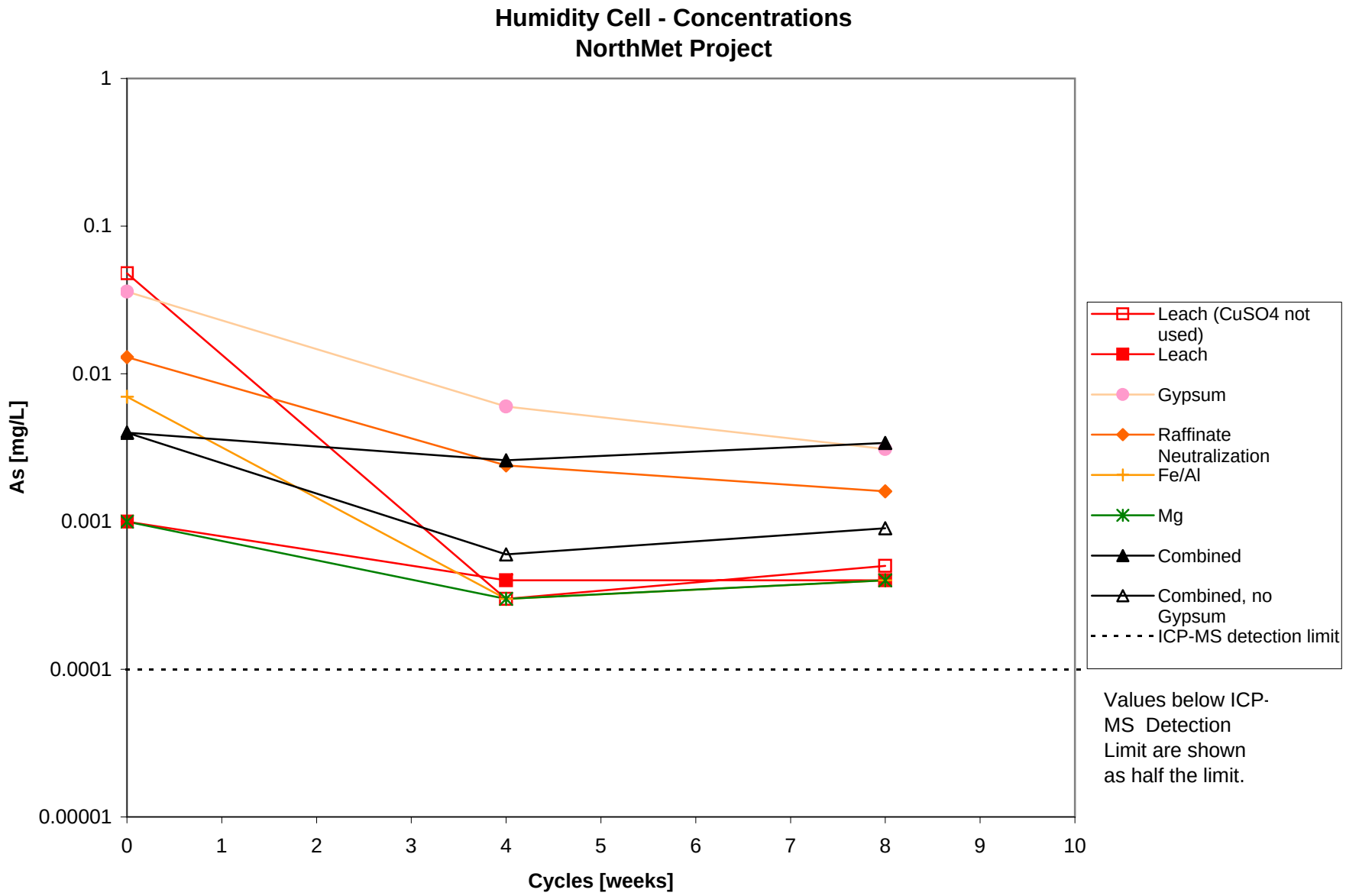

Humidity Cell - Concentrations
NorthMet Project
F [mg/L]
100
10
1
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
Detection limit
Values below Detection Limit are shown as half the limit.

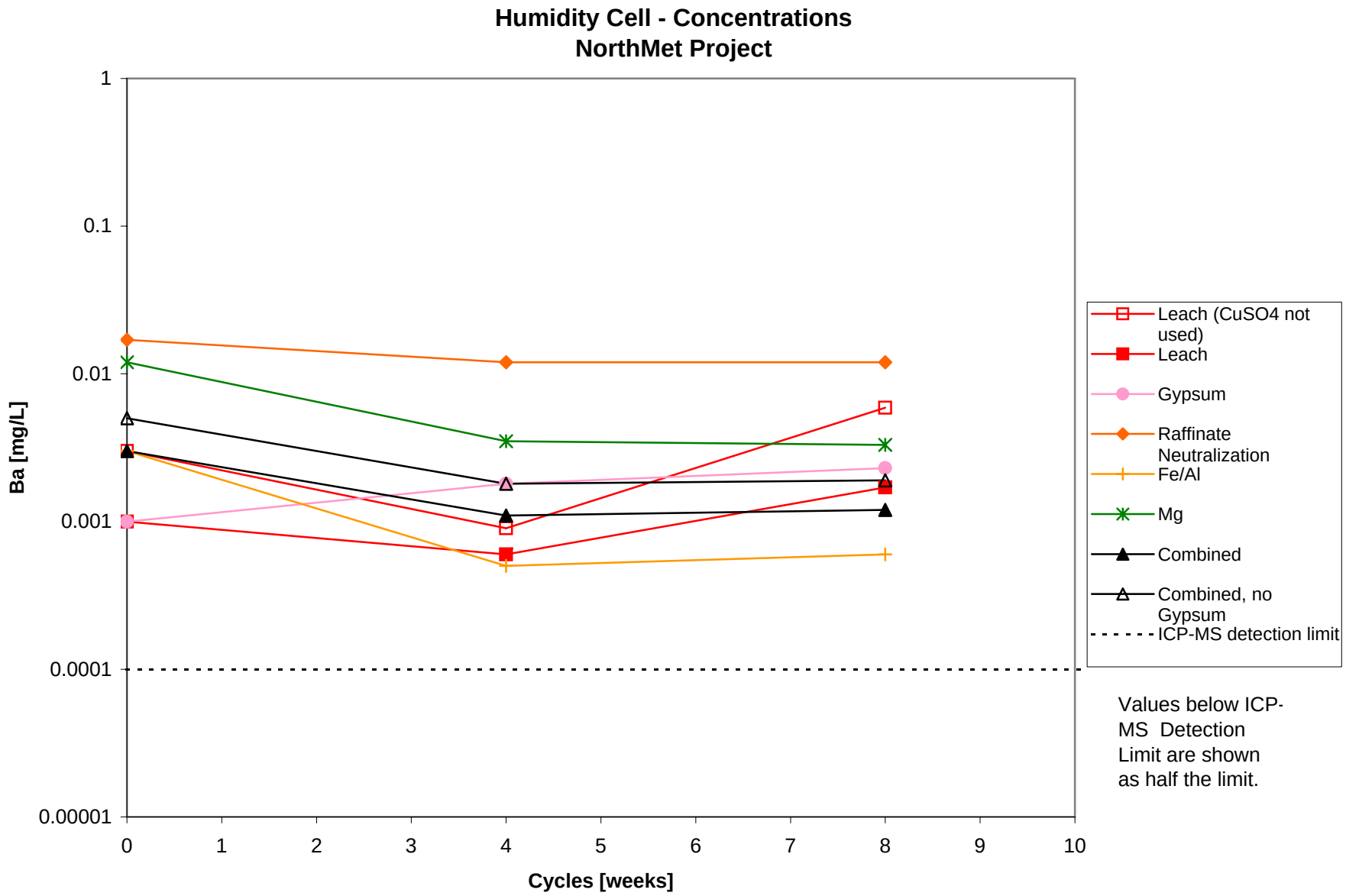

Humidity Cell - Concentrations
NorthMet Project
Cl [mg/L]
10000
1000
100
10
1
0.1
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
Detection limit
Values below Detection Limit are shown as half the limit.

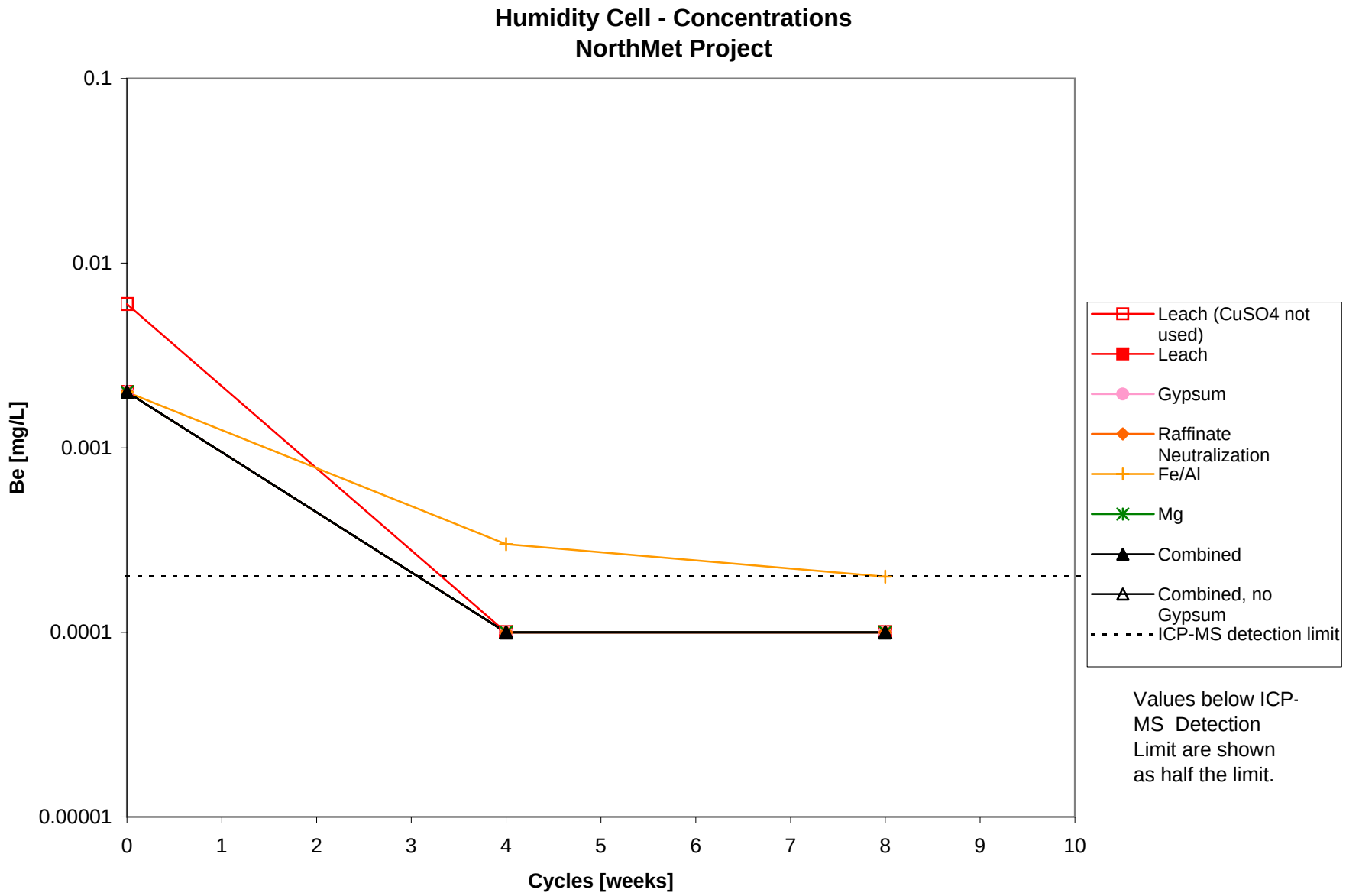

Humidity Cell - Concentrations
NorthMet Project
SO4 [mg/L]
25000
20000
15000
10000
5000
0
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
Detection limit - 0.5 mg/L
Values below Detection Limit are shown as half the limit.

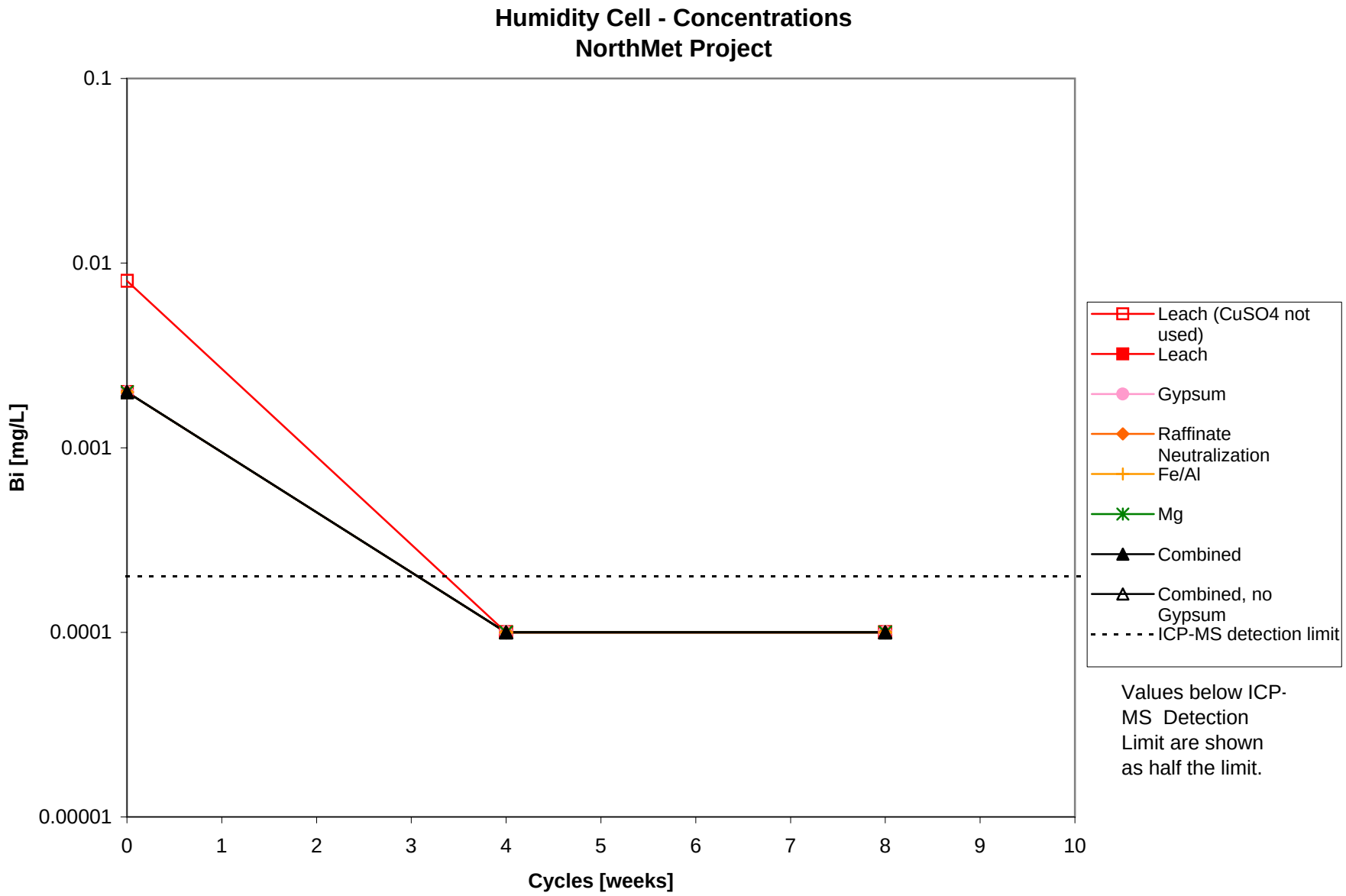

Humidity Cell - Concentrations
NorthMet Project
Al [mg/L]
1000
100
10
1
0.1
0.01
0.001
0.0001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

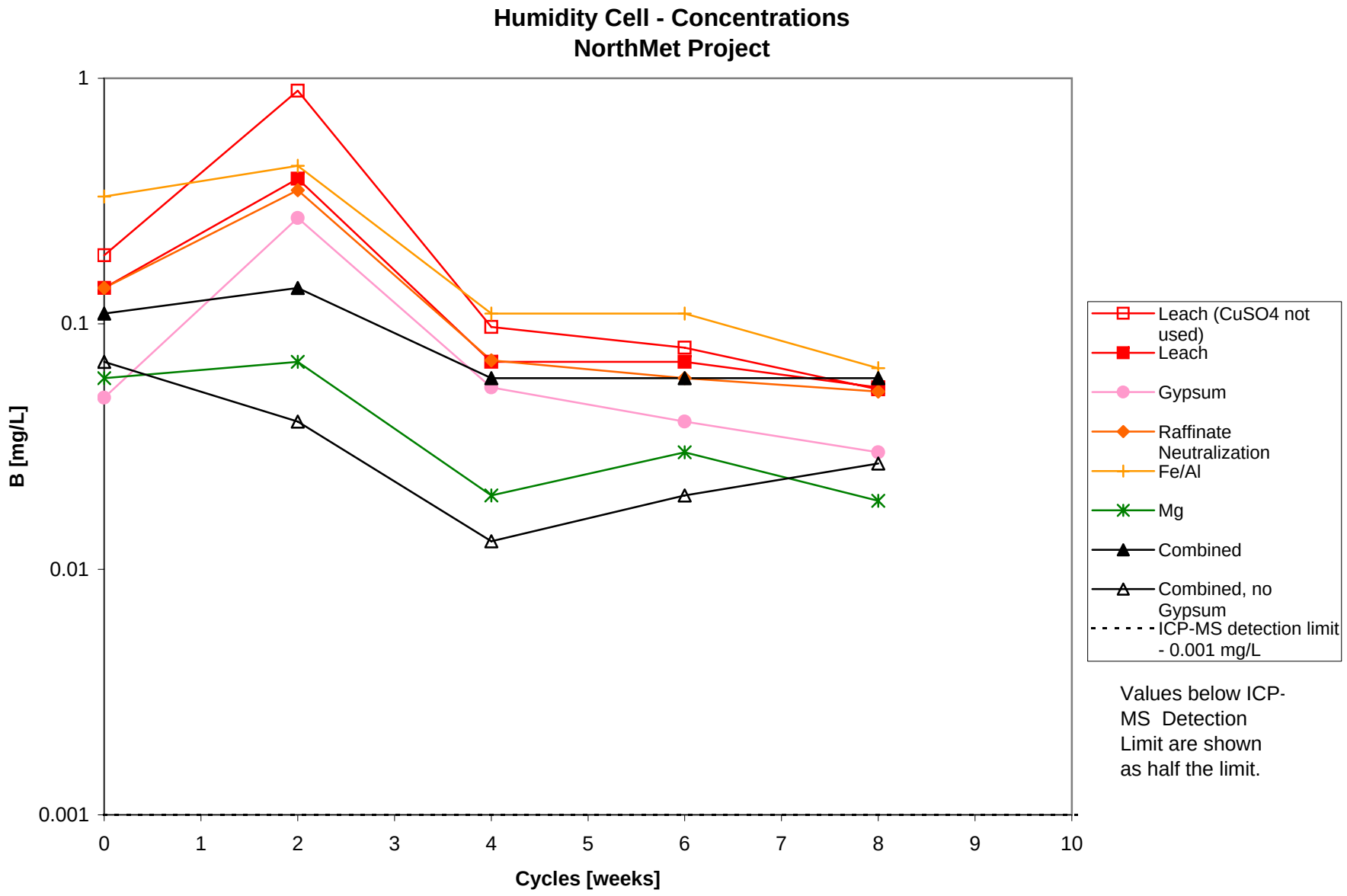

Humidity Cell - Concentrations
NorthMet Project
Sb [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

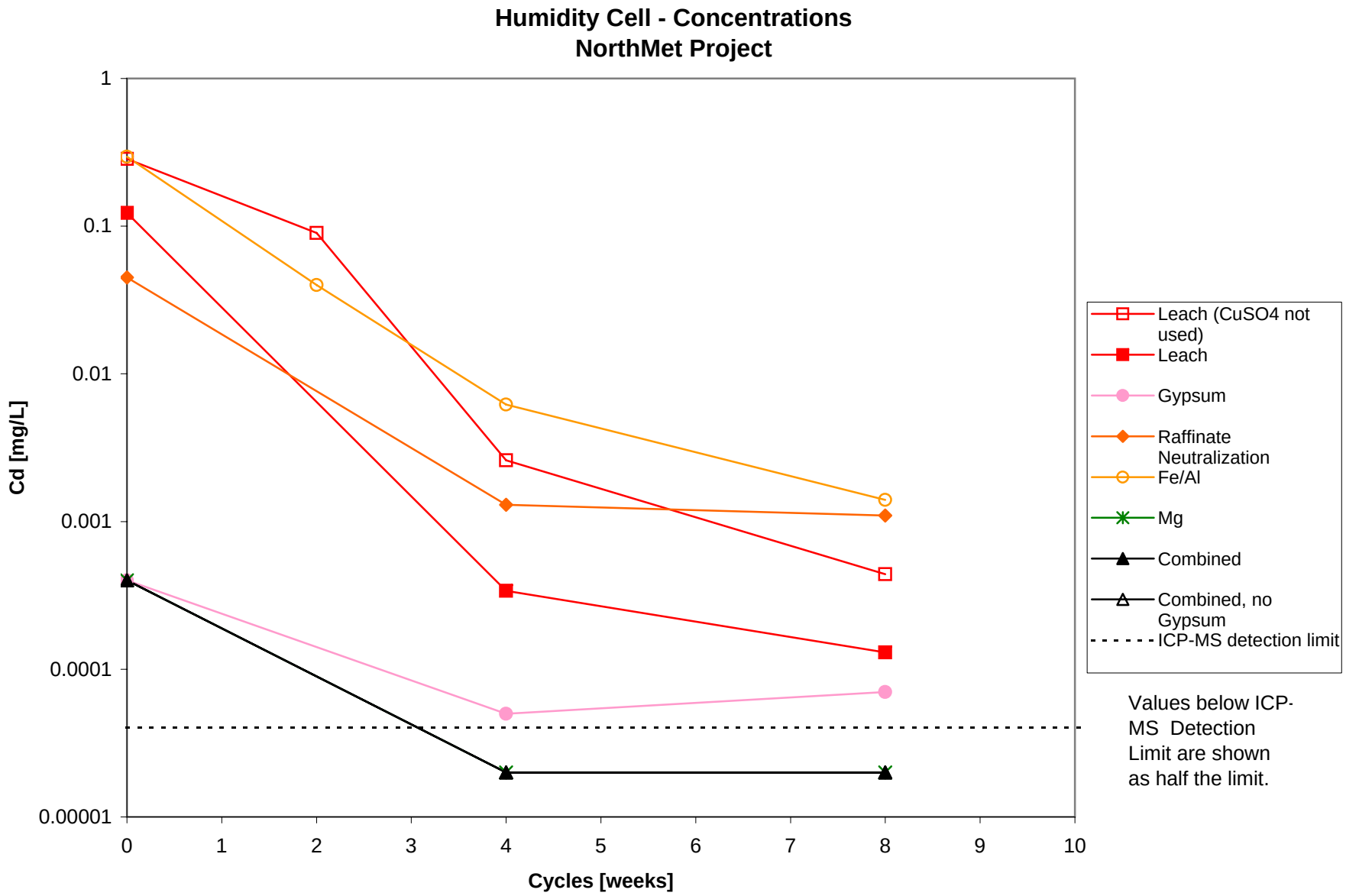

Humidity Cell - Concentrations
NorthMet Project
As [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

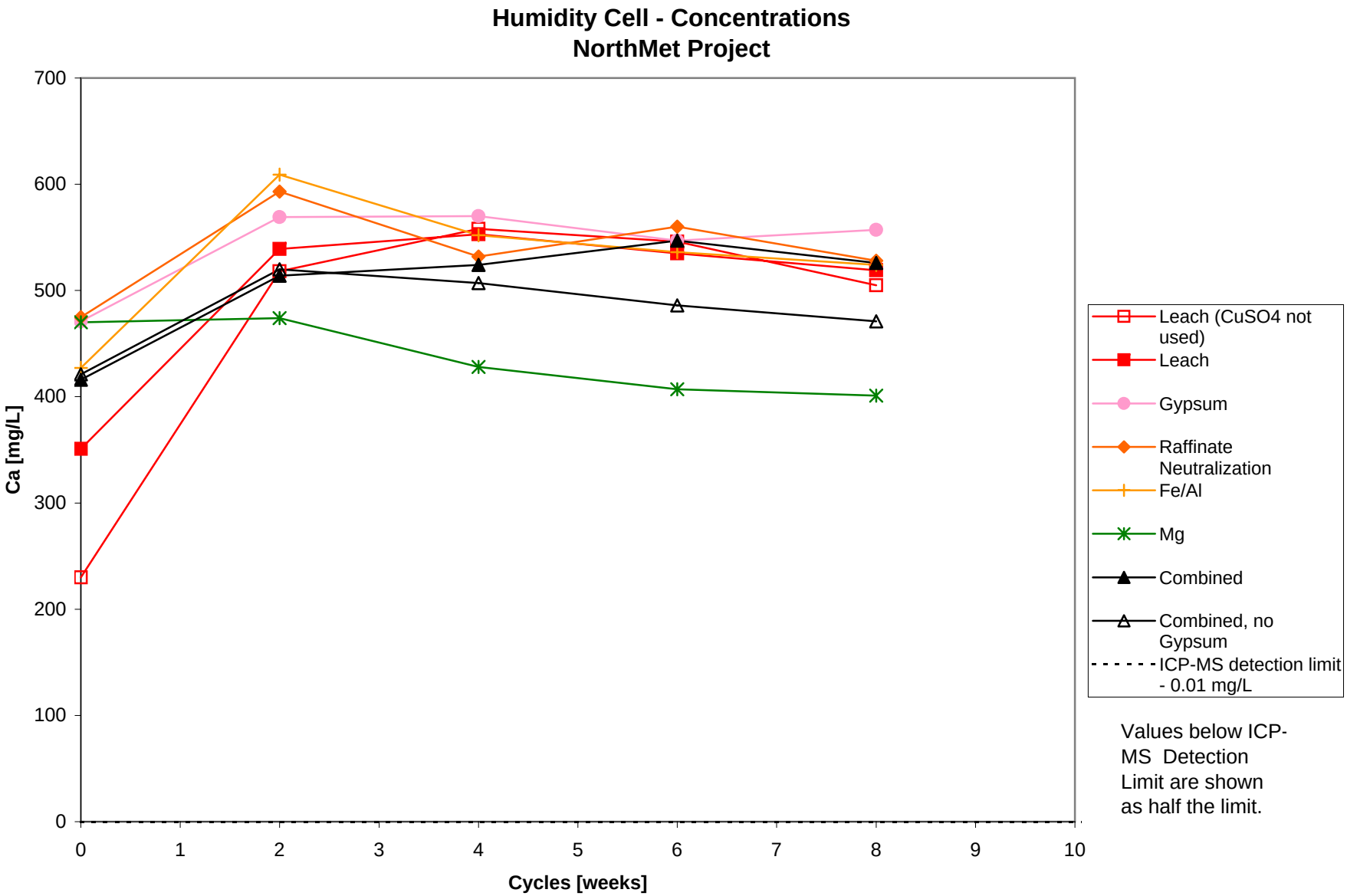

Humidity Cell - Concentrations
NorthMet Project
Ba [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

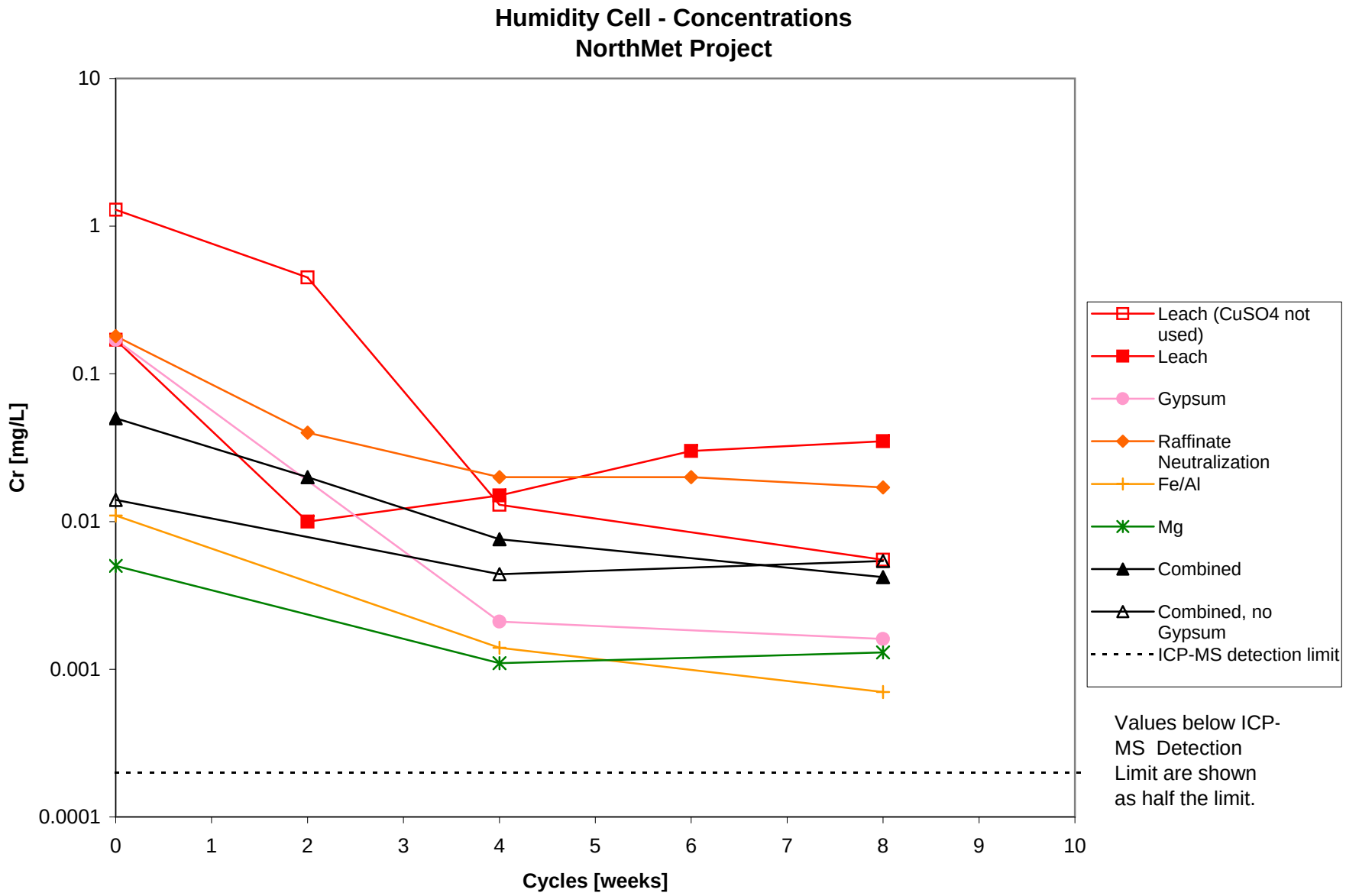

Humidity Cell - Concentrations
NorthMet Project
Be [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

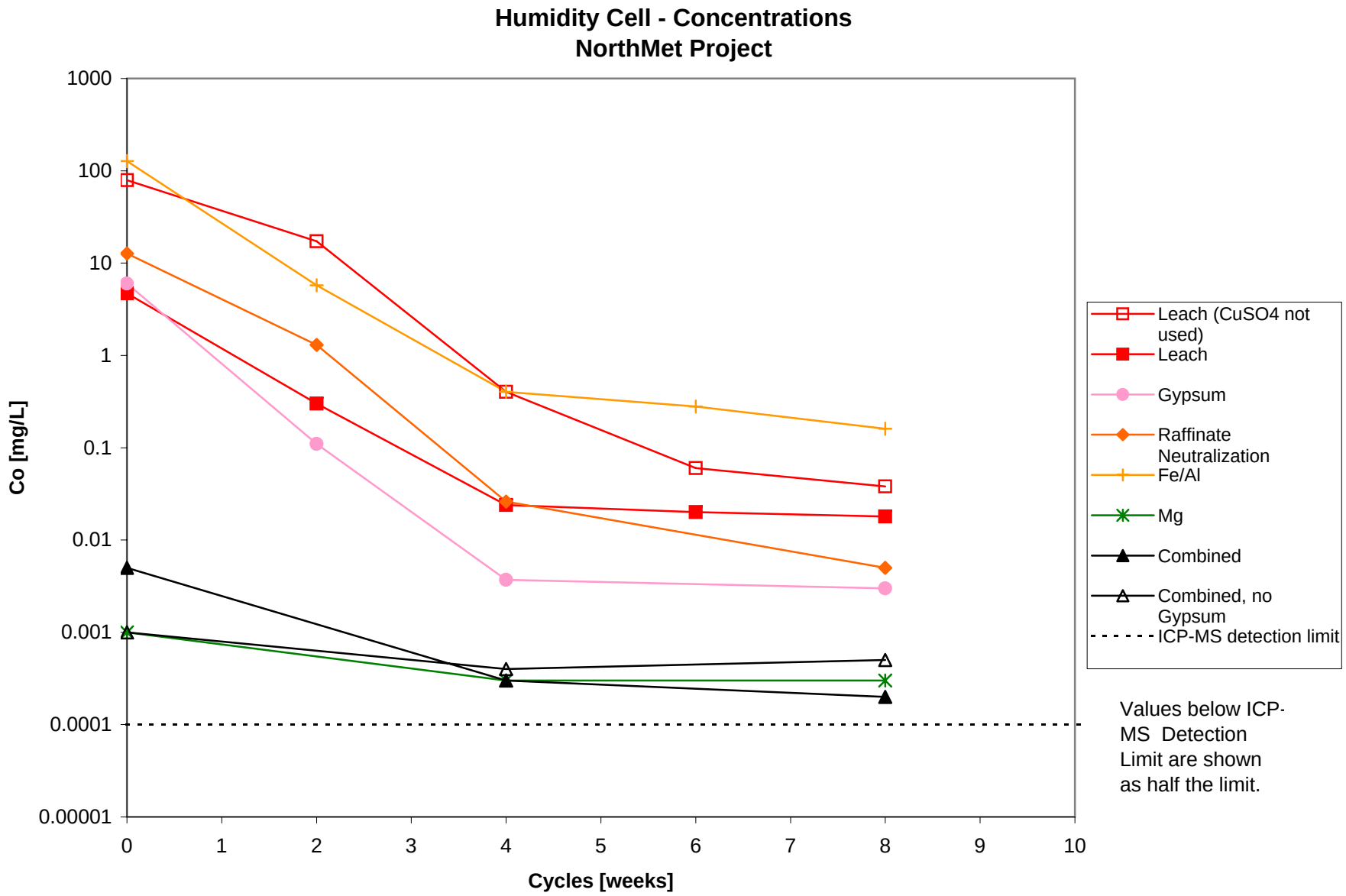

Humidity Cell - Concentrations
NorthMet Project
Bi [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

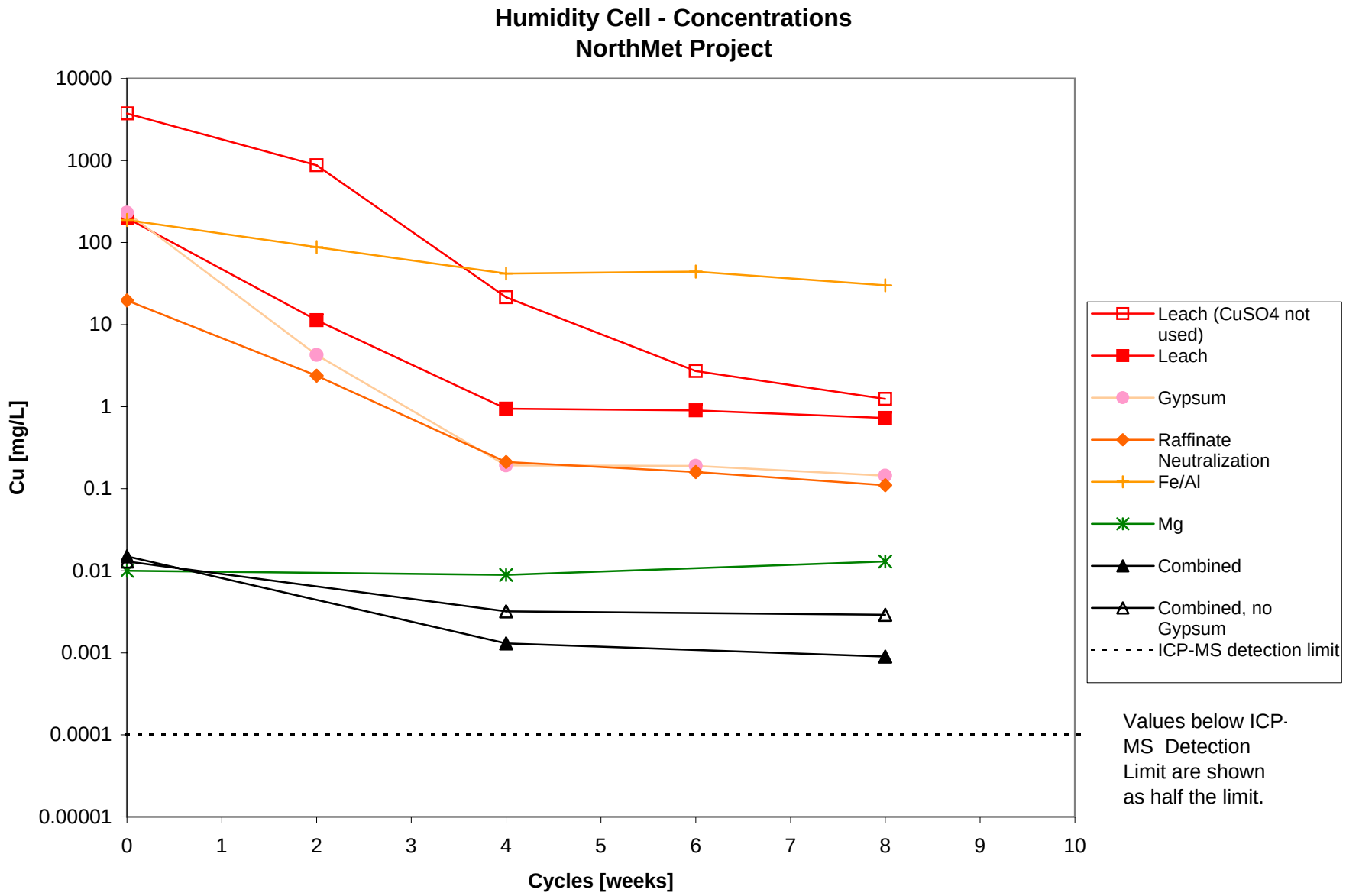

Humidity Cell - Concentrations
NorthMet Project
B [mg/L]
1
0.1
0.01
0.001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit - 0.001 mg/L
Values below ICP-MS Detection Limit are shown as half the limit.

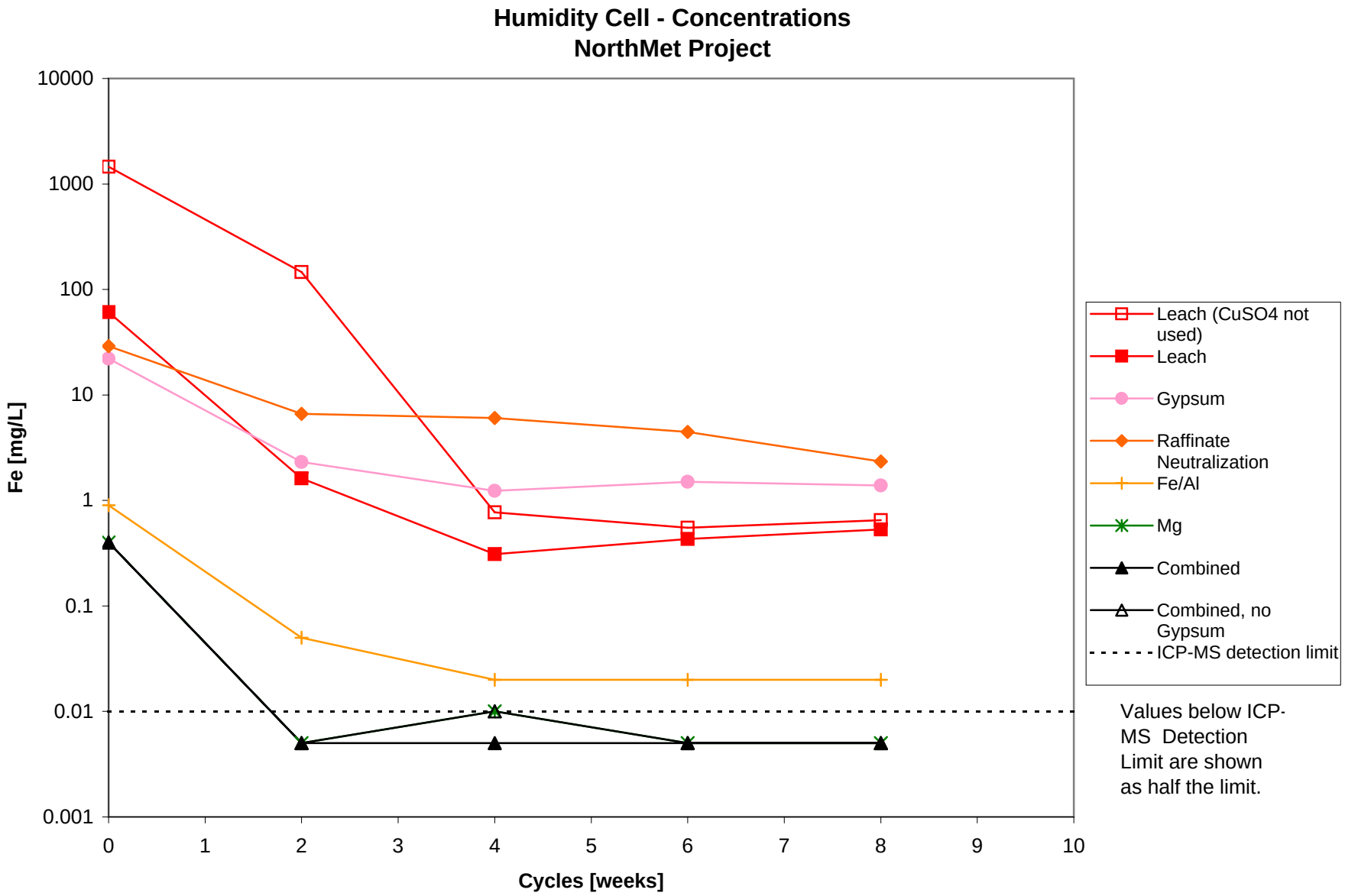

Humidity Cell - Concentrations
NorthMet Project
Cd [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

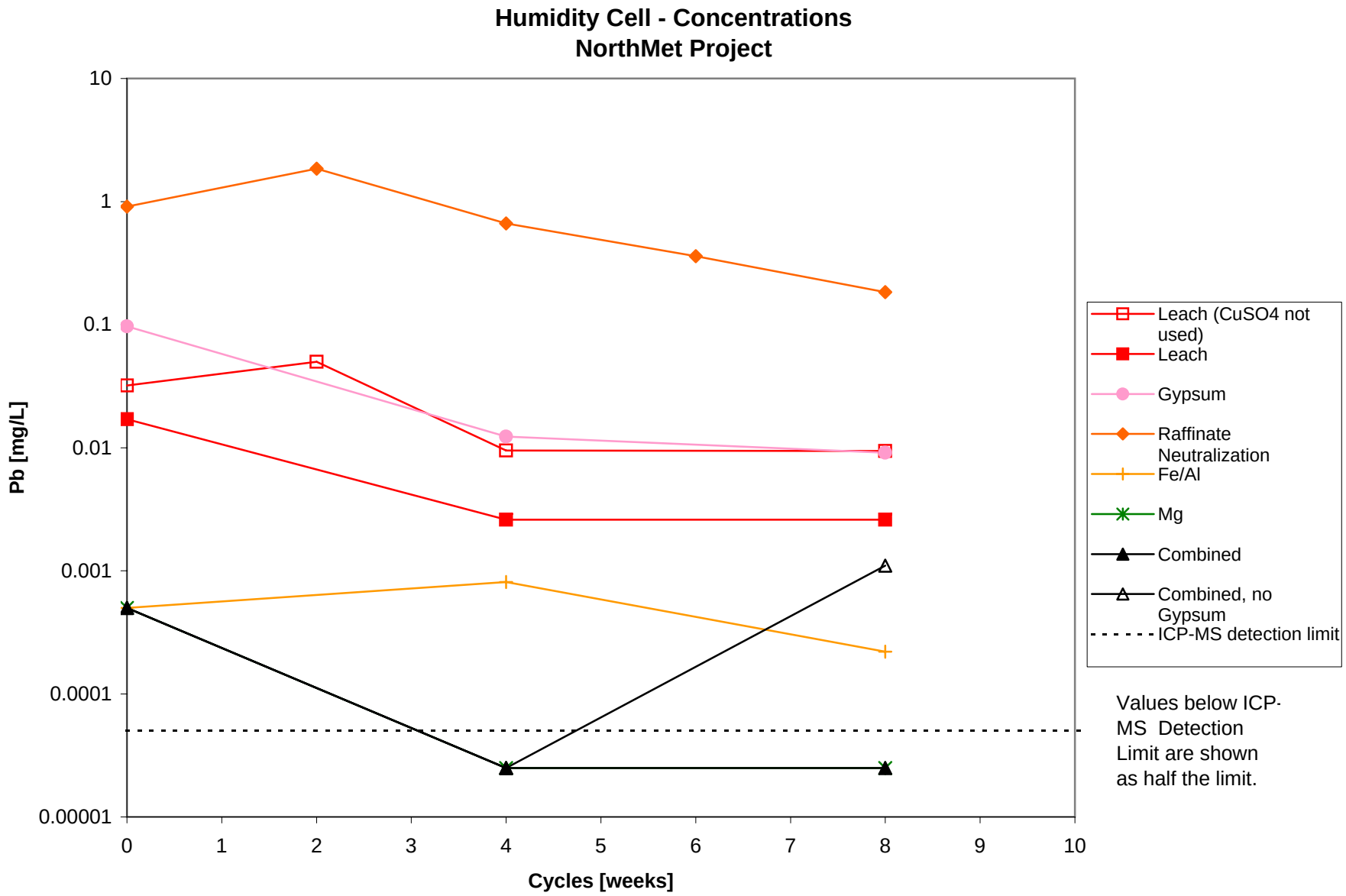

Humidity Cell - Concentrations
NorthMet Project
Ca [mg/L]
Cycles [weeks]
0
100
200
300
400
500
600
700
0
1
2
3
4
5
6
7
8
9
10
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit - 0.01 mg/L
Values below ICP-MS Detection Limit are shown as half the limit.

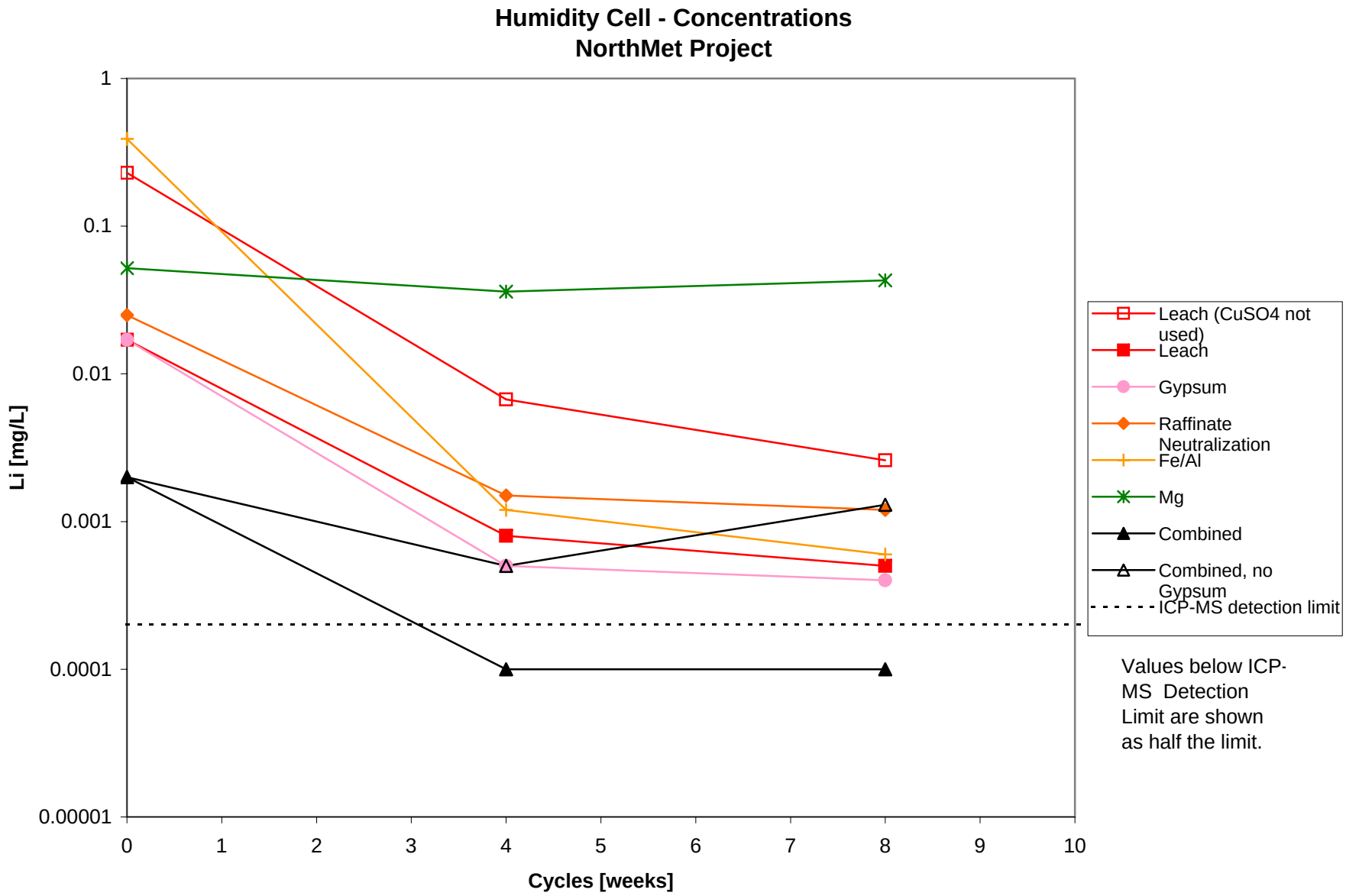

Humidity Cell - Concentrations
NorthMet Project
Cr [mg/L]
10
1
0.1
0.01
0.001
0.0001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

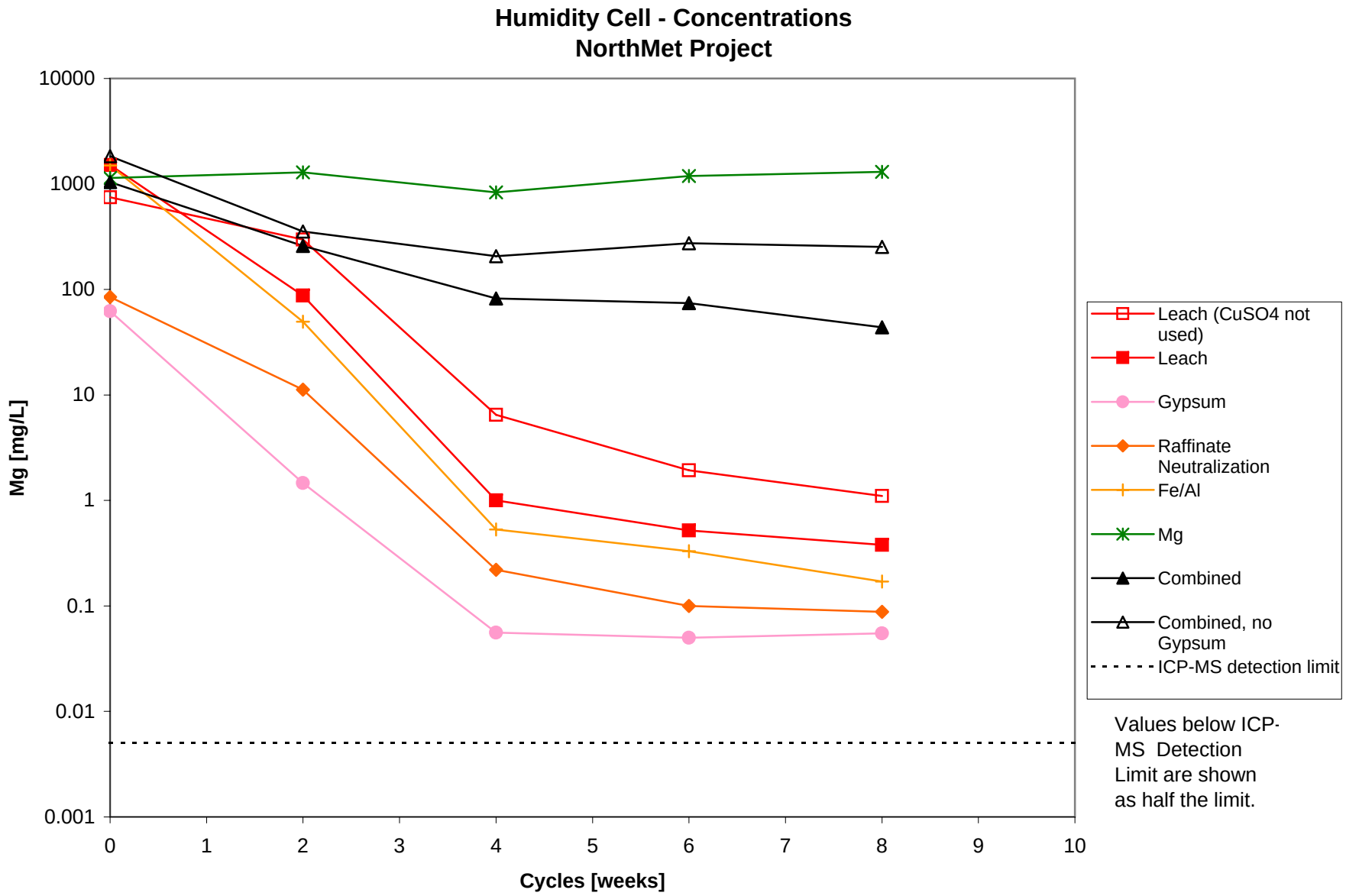

Humidity Cell - Concentrations
NorthMet Project
Co [mg/L]
1000
100
10
1
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

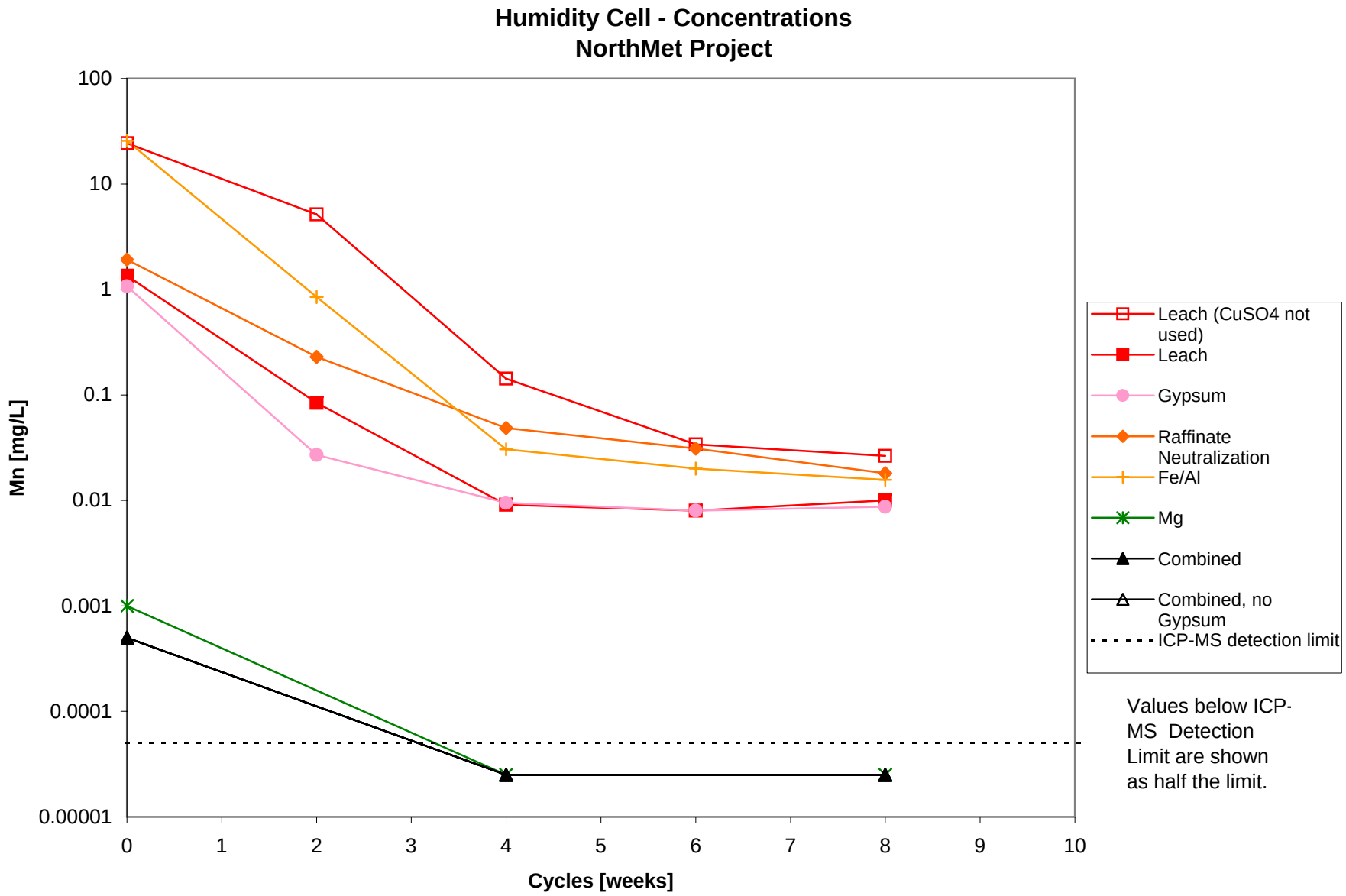

Humidity Cell - Concentrations
NorthMet Project
Cu [mg/L]
10000
1000
100
10
1
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

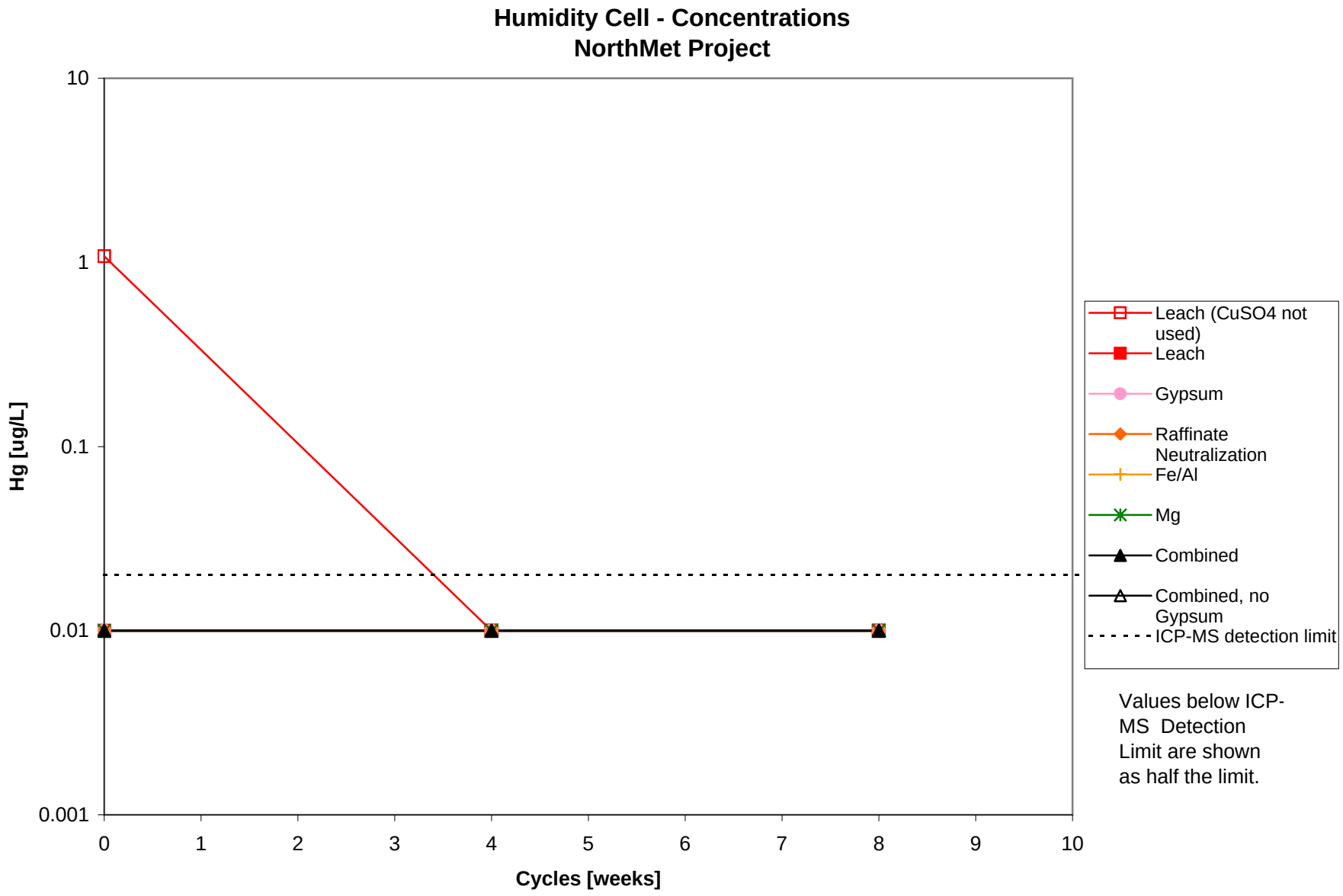

Humidity Cell - Concentrations
NorthMet Project
Fe [mg/L]
10000
1000
100
10
1
0.1
0.01
0.001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

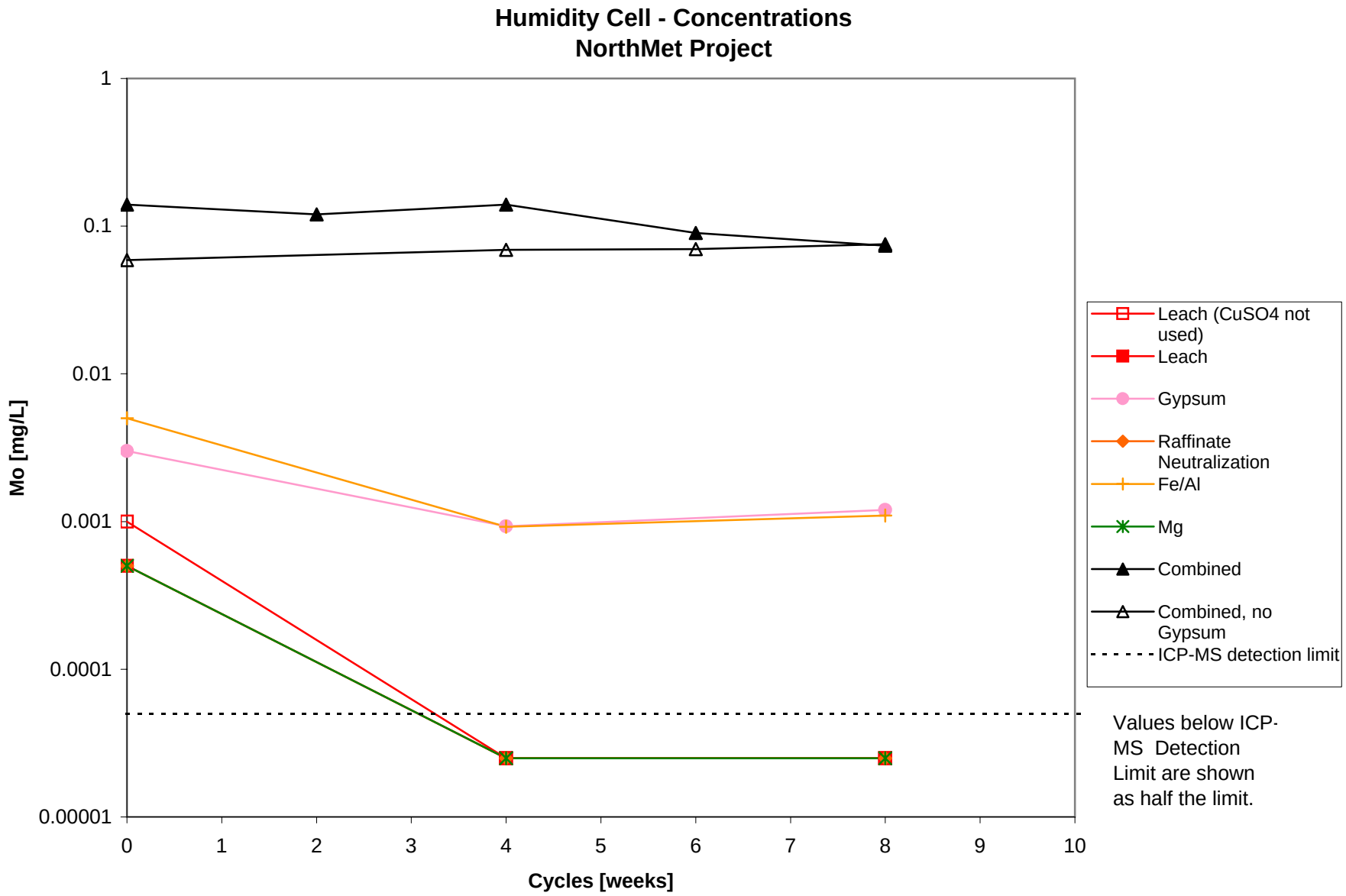

Humidity Cell - Concentrations
NorthMet Project
Pb [mg/L]
10
1
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

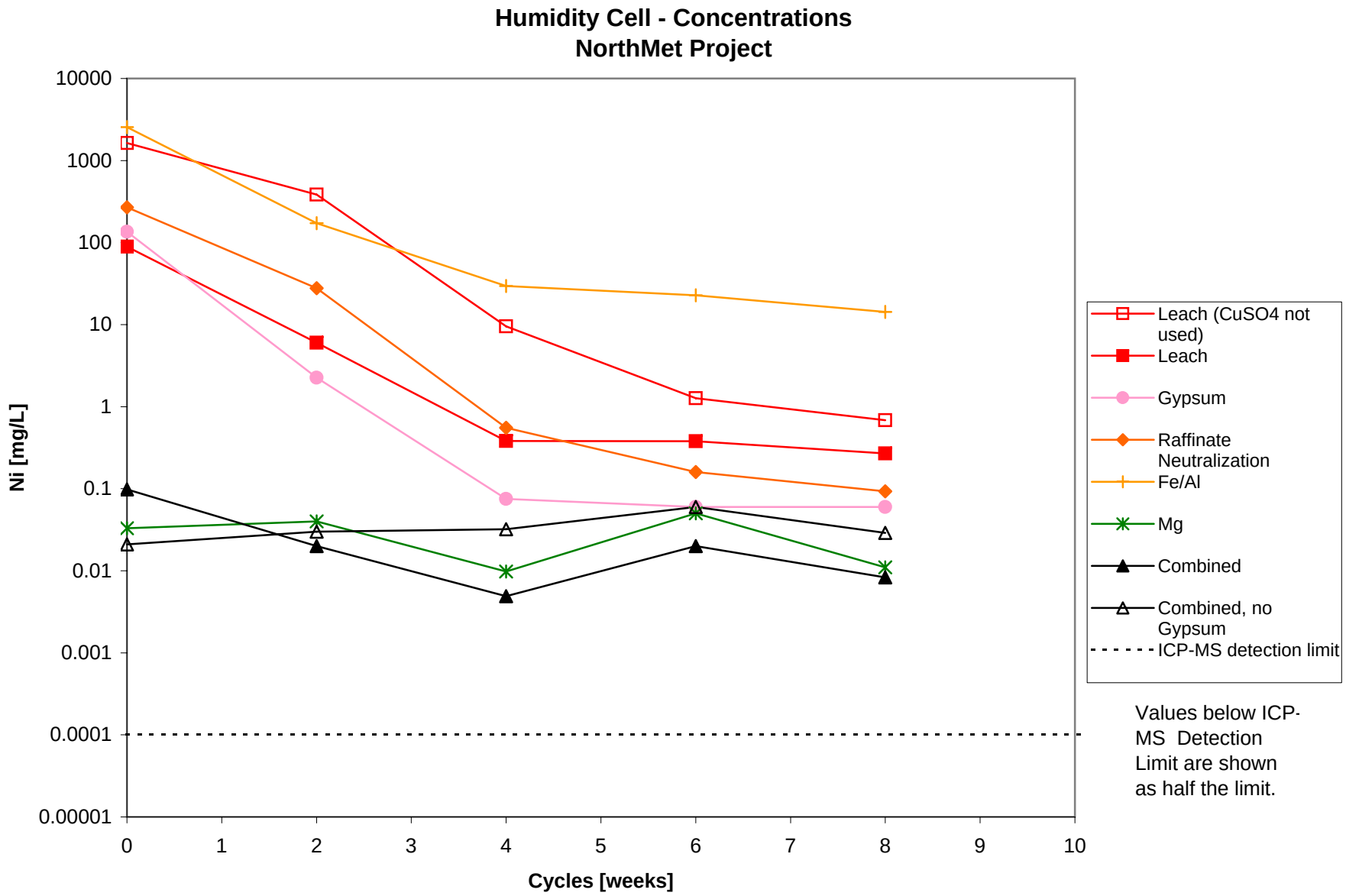

Humidity Cell - Concentrations
NorthMet Project
Li [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

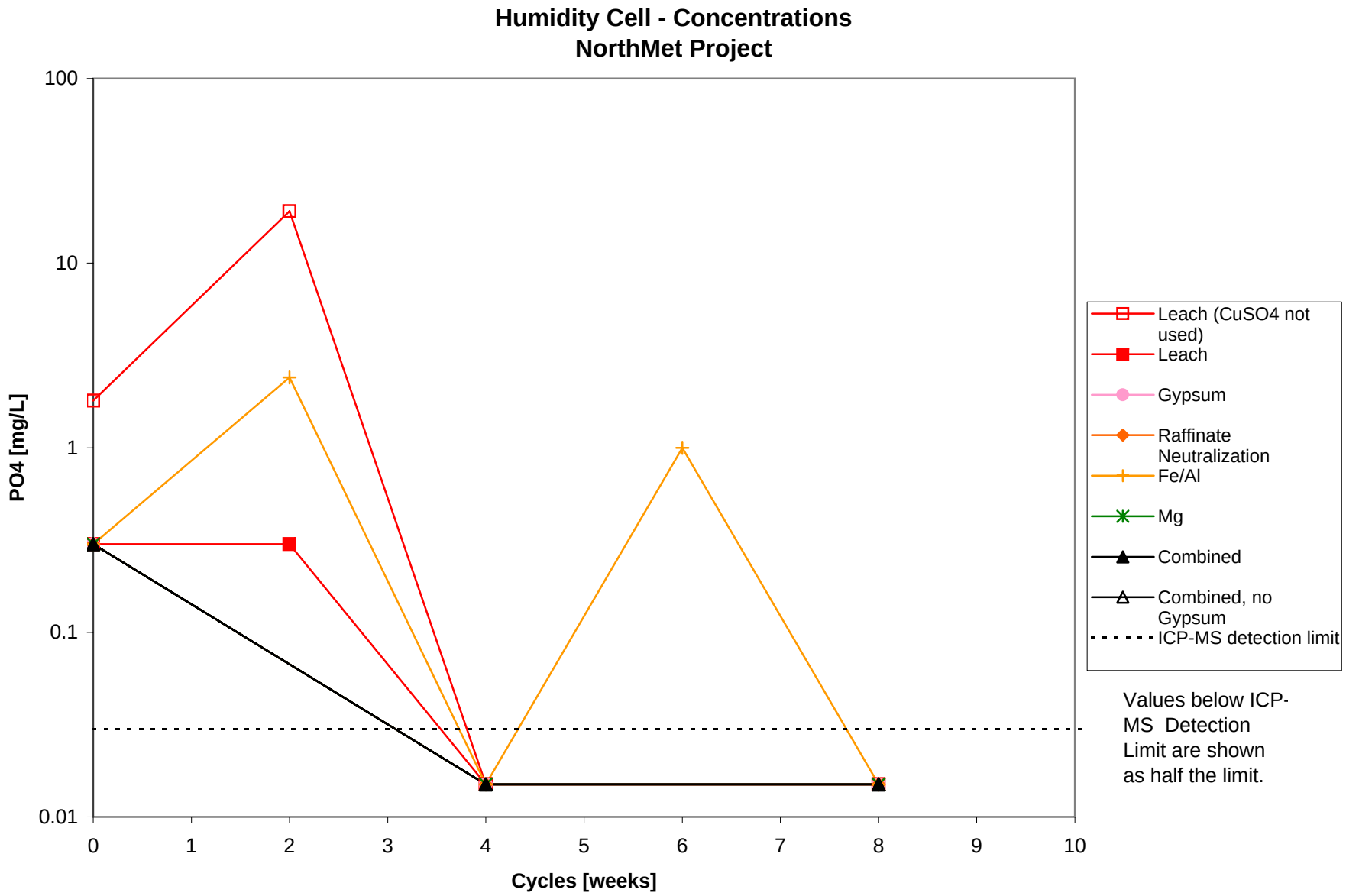

Humidity Cell - Concentrations
NorthMet Project
Mg [mg/L]
10000
1000
100
10
1
0.1
0.01
0.001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

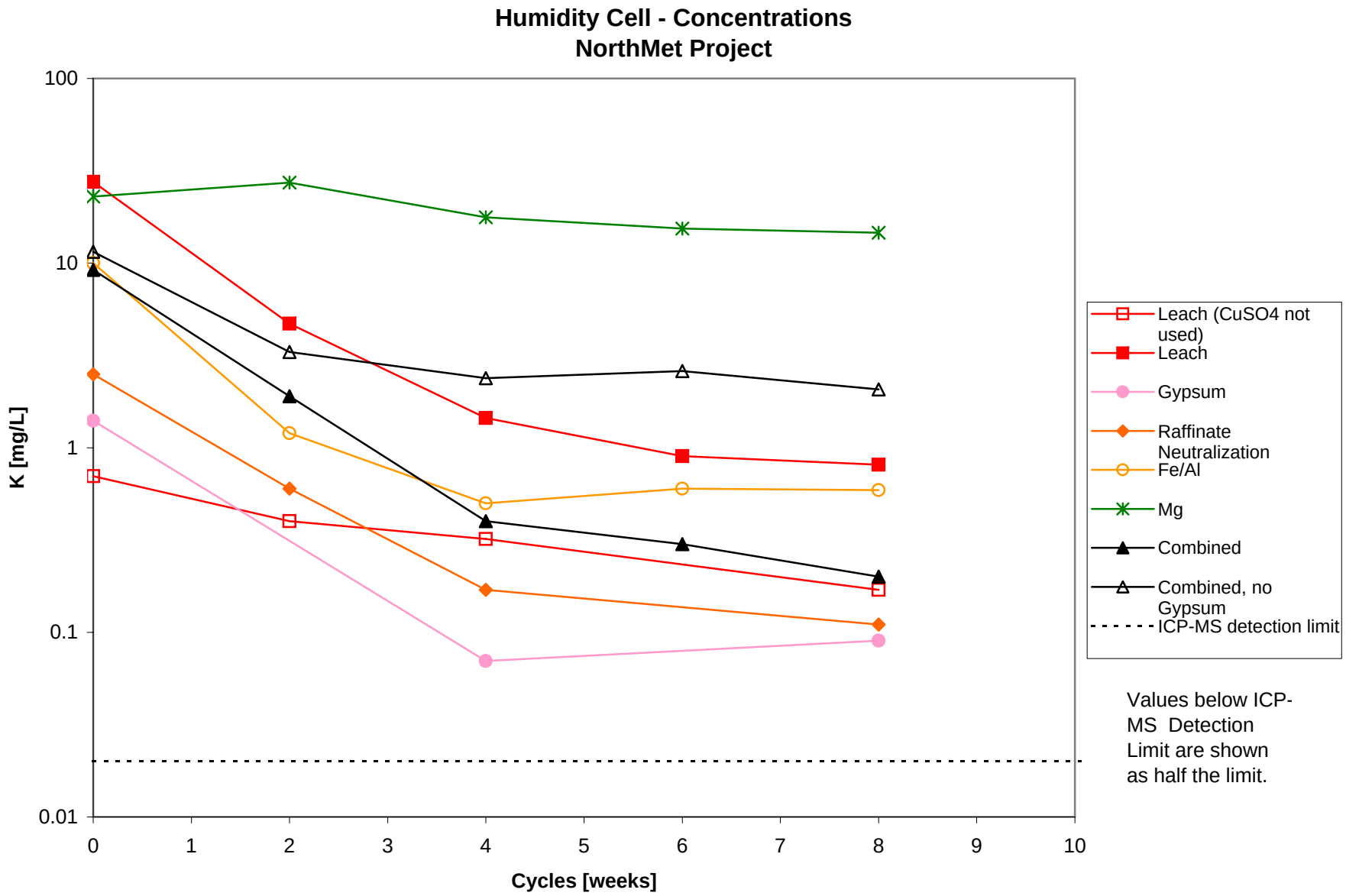

Humidity Cell - Concentrations
NorthMet Project
Mn [mg/L]
100
10
1
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

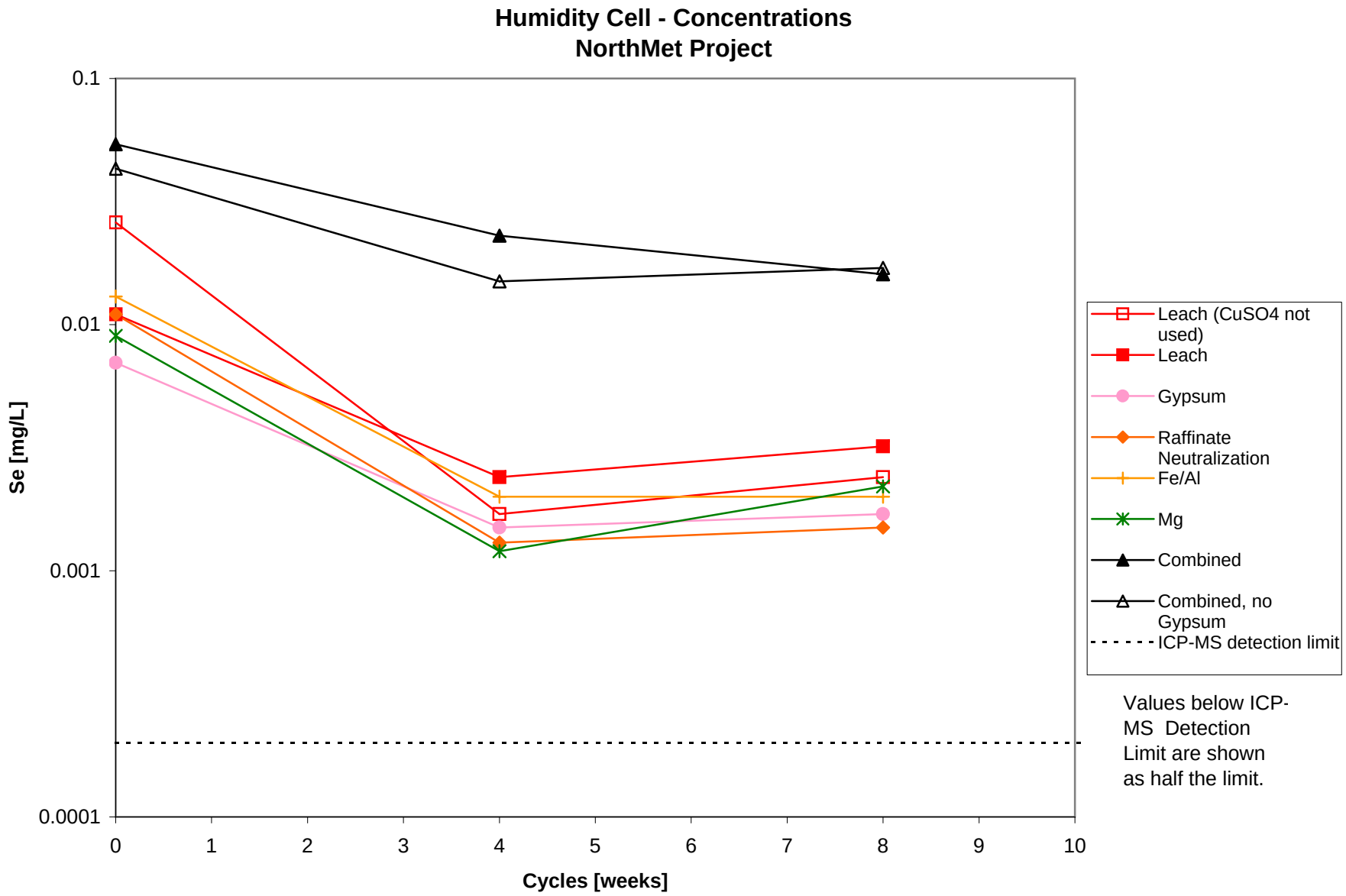

Humidity Cell - Concentrations
NorthMet Project
Hg [ug/L]
10
1
0.1
0.01
0.001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

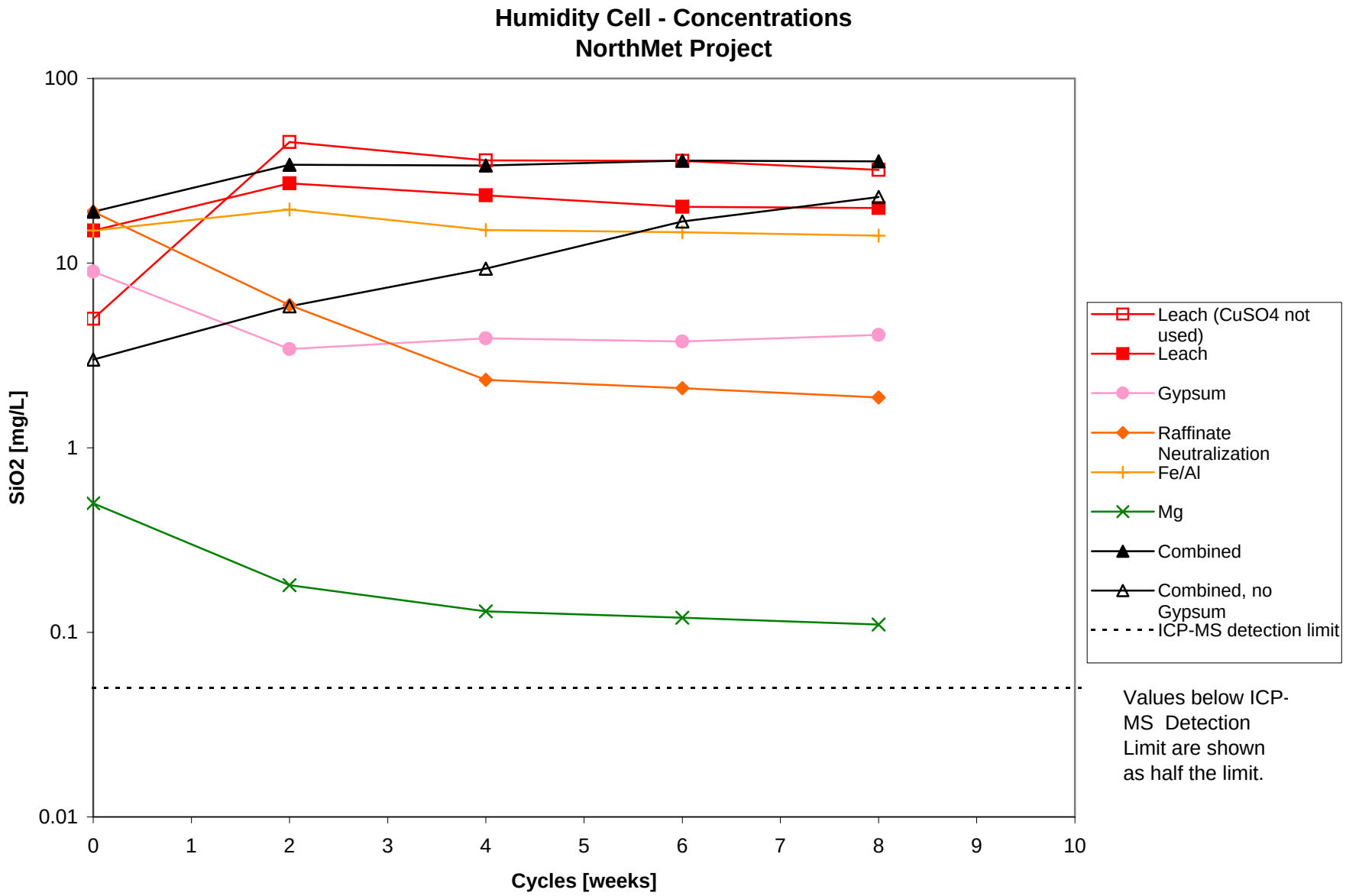

Humidity Cell - Concentrations
NorthMet Project
Mo [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

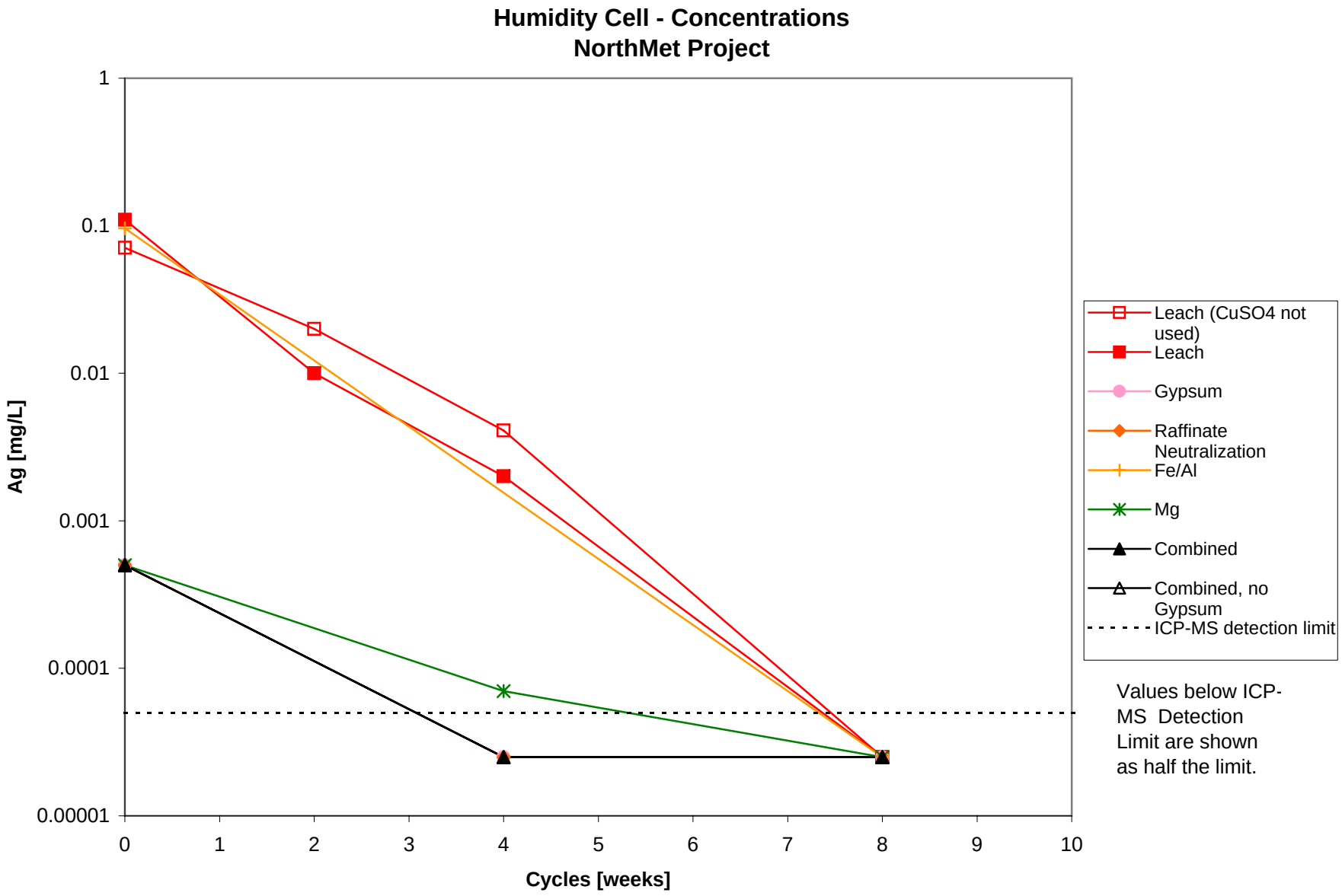

Humidity Cell - Concentrations
NorthMet Project
Ni [mg/L]
10000
1000
100
10
1
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

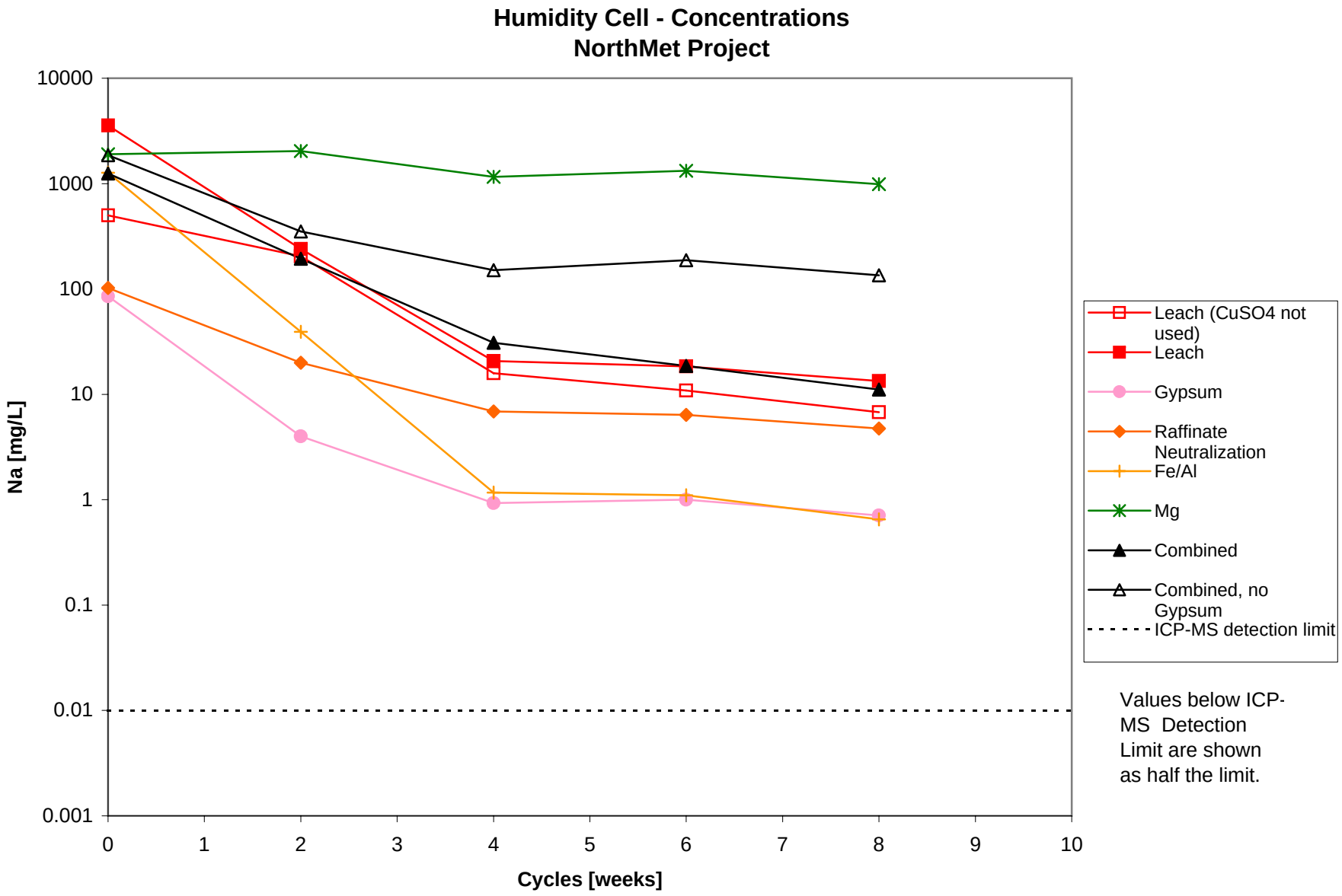

Humidity Cell - Concentrations
NorthMet Project
PO4 [mg/L]
100
10
1
0.1
0.01
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

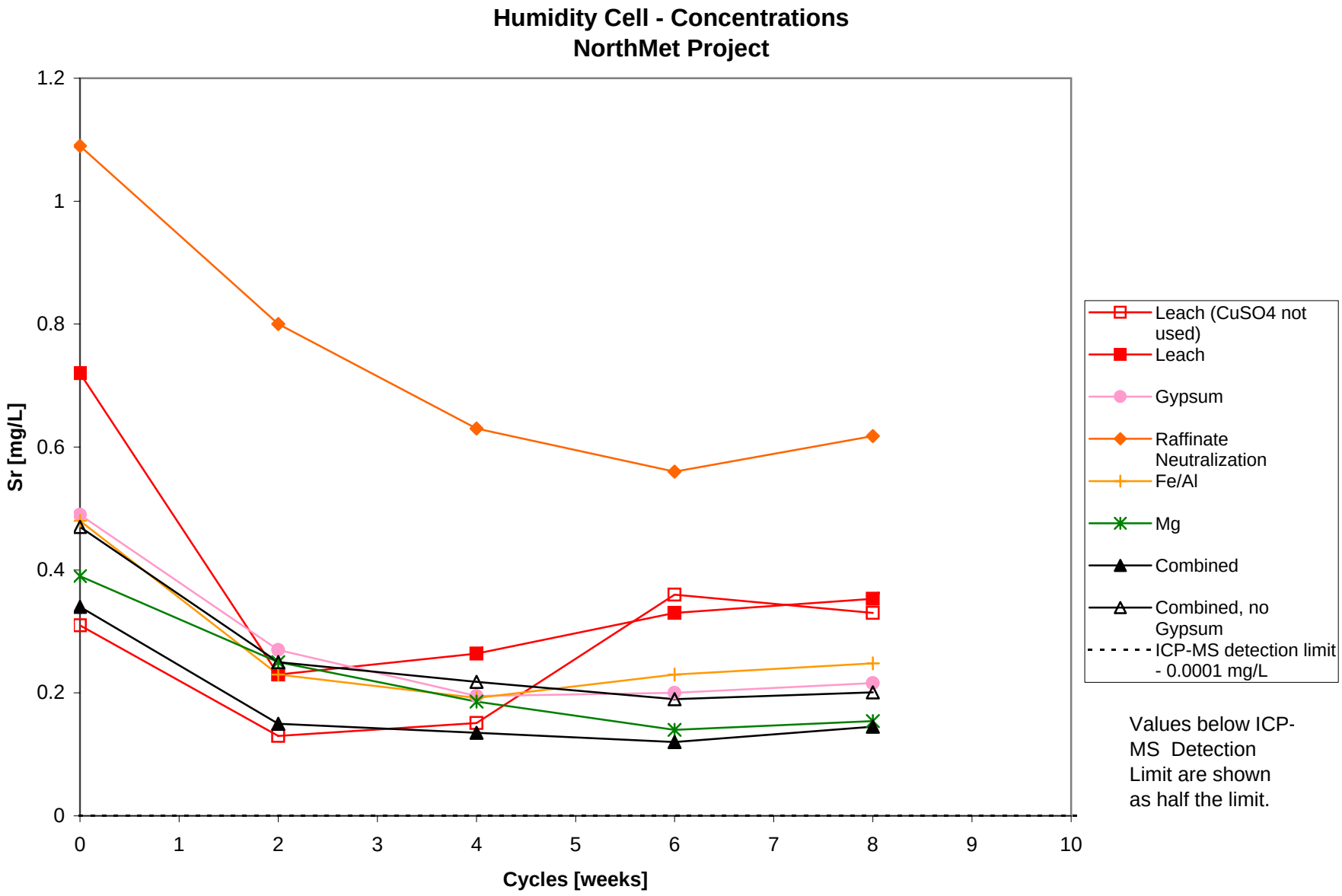

Humidity Cell - Concentrations
NorthMet Project
K [mg/L]
100
10
1
0.1
0.01
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.


Humidity Cell - Concentrations
NorthMet Project
Se [mg/L]
0.1
0.01
0.001
0.0001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

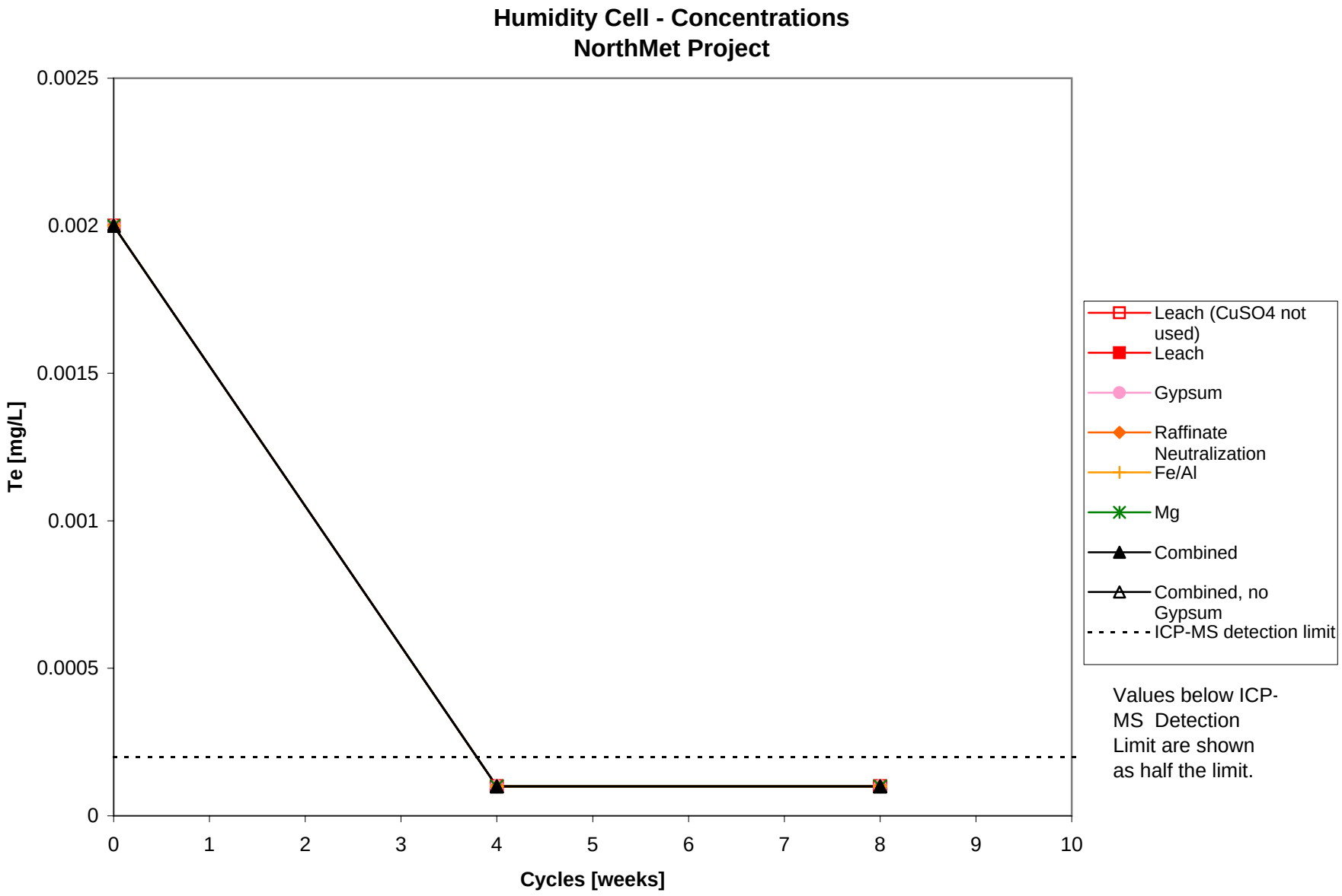

Humidity Cell - Concentrations
NorthMet Project
SiO2 [mg/L]
100
10
1
0.1
0.01
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

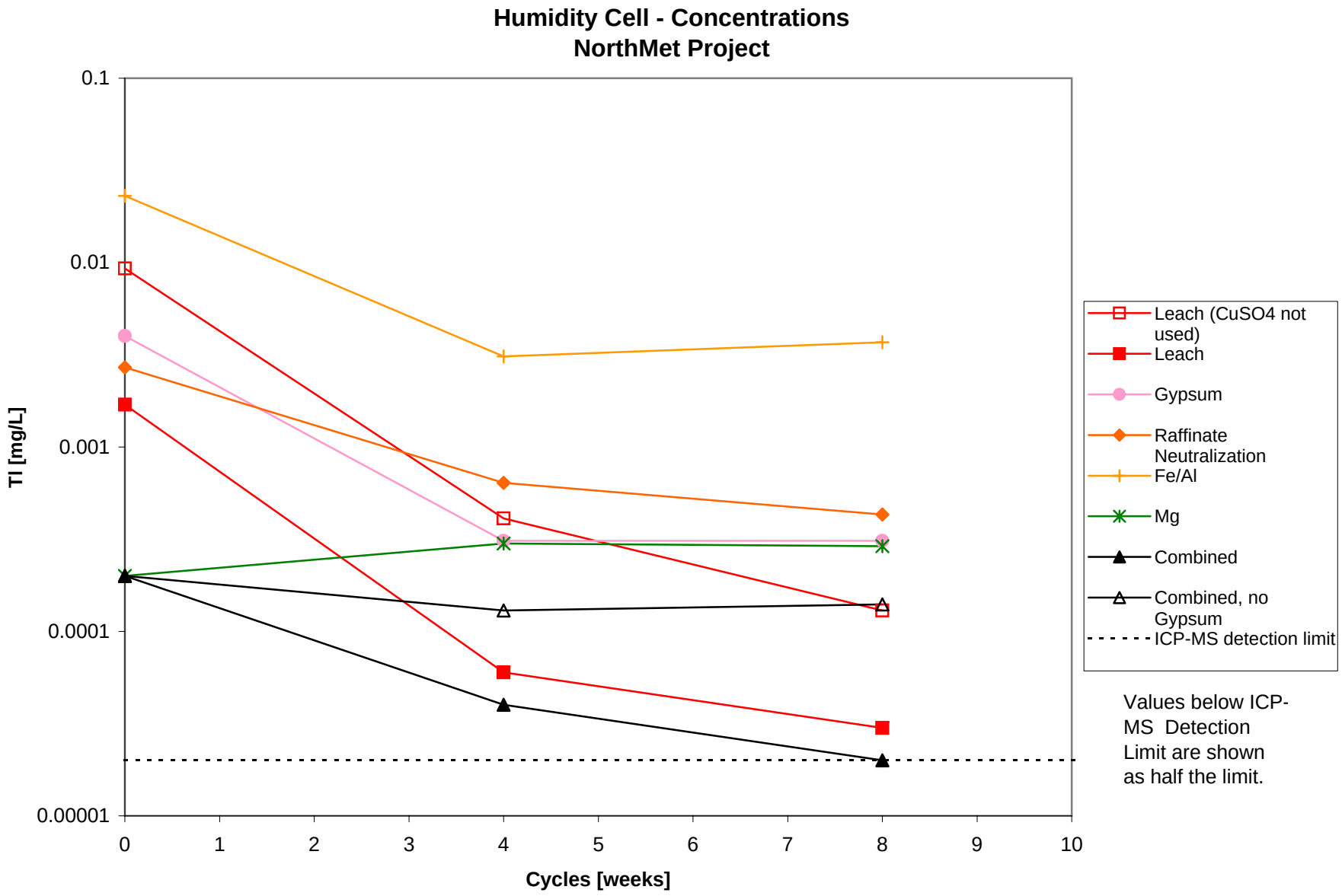

Humidity Cell - Concentrations
NorthMet Project
Ag [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

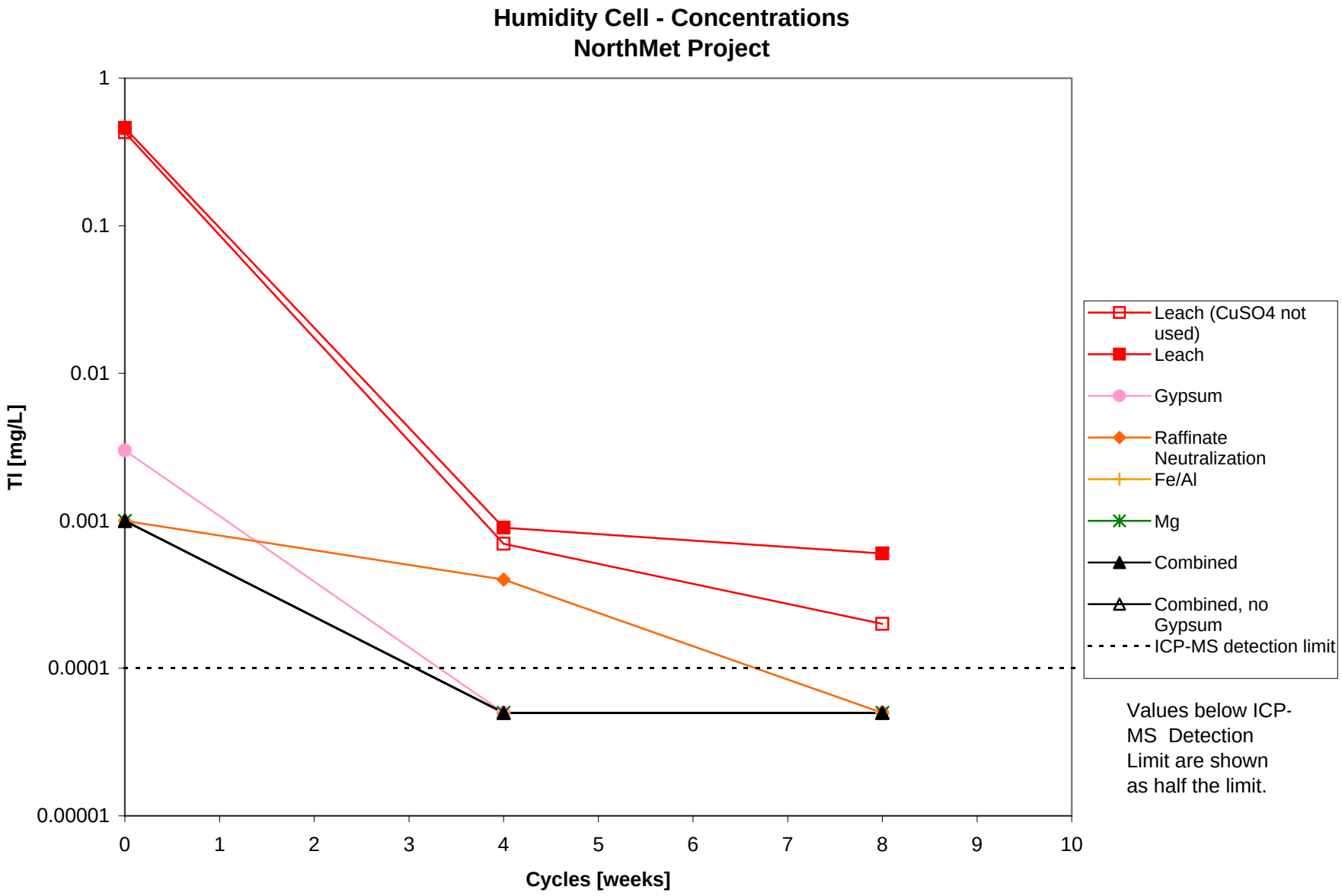

Humidity Cell - Concentrations
NorthMet Project
Na [mg/L]
10000
1000
100
10
1
0.1
0.01
0.001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

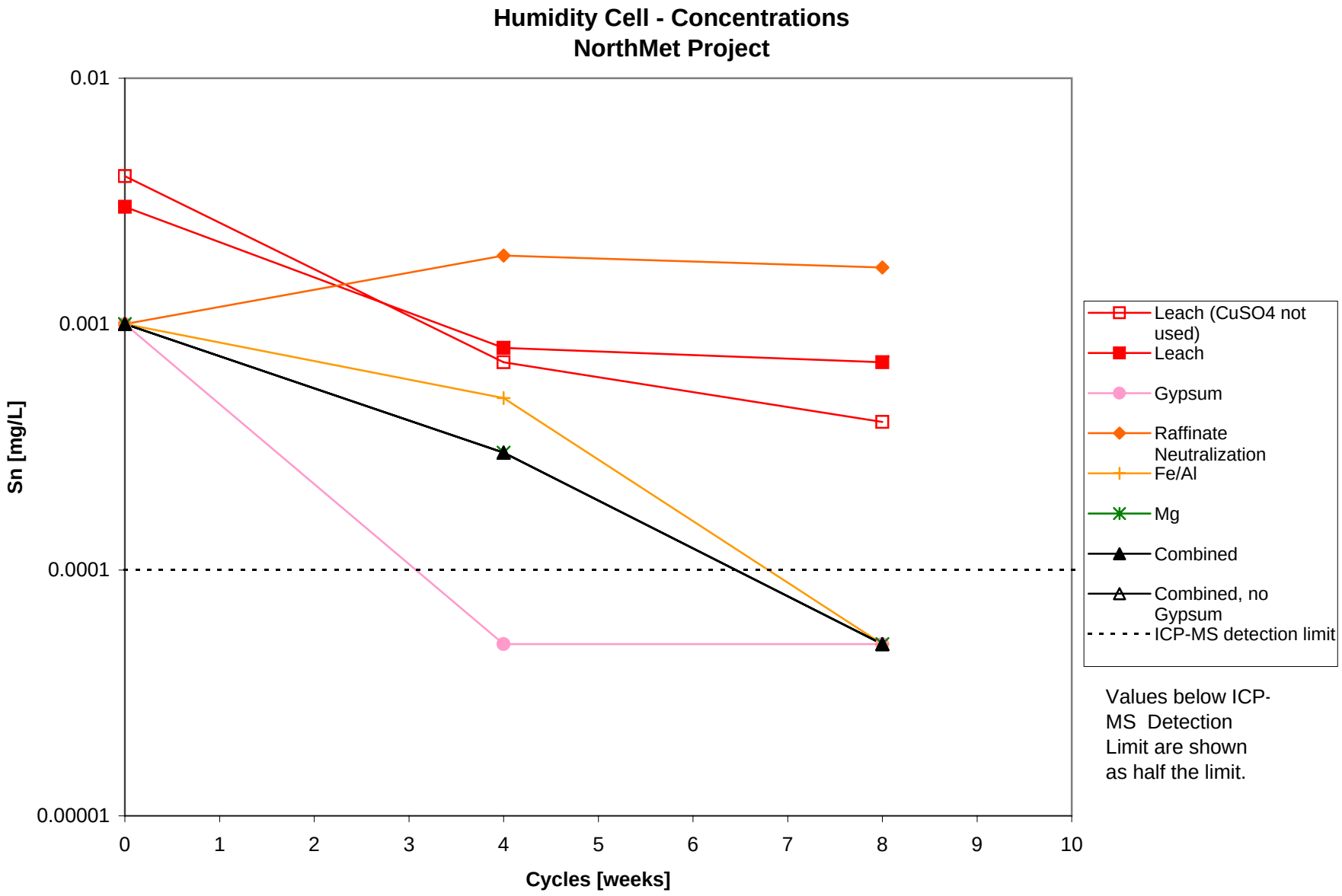


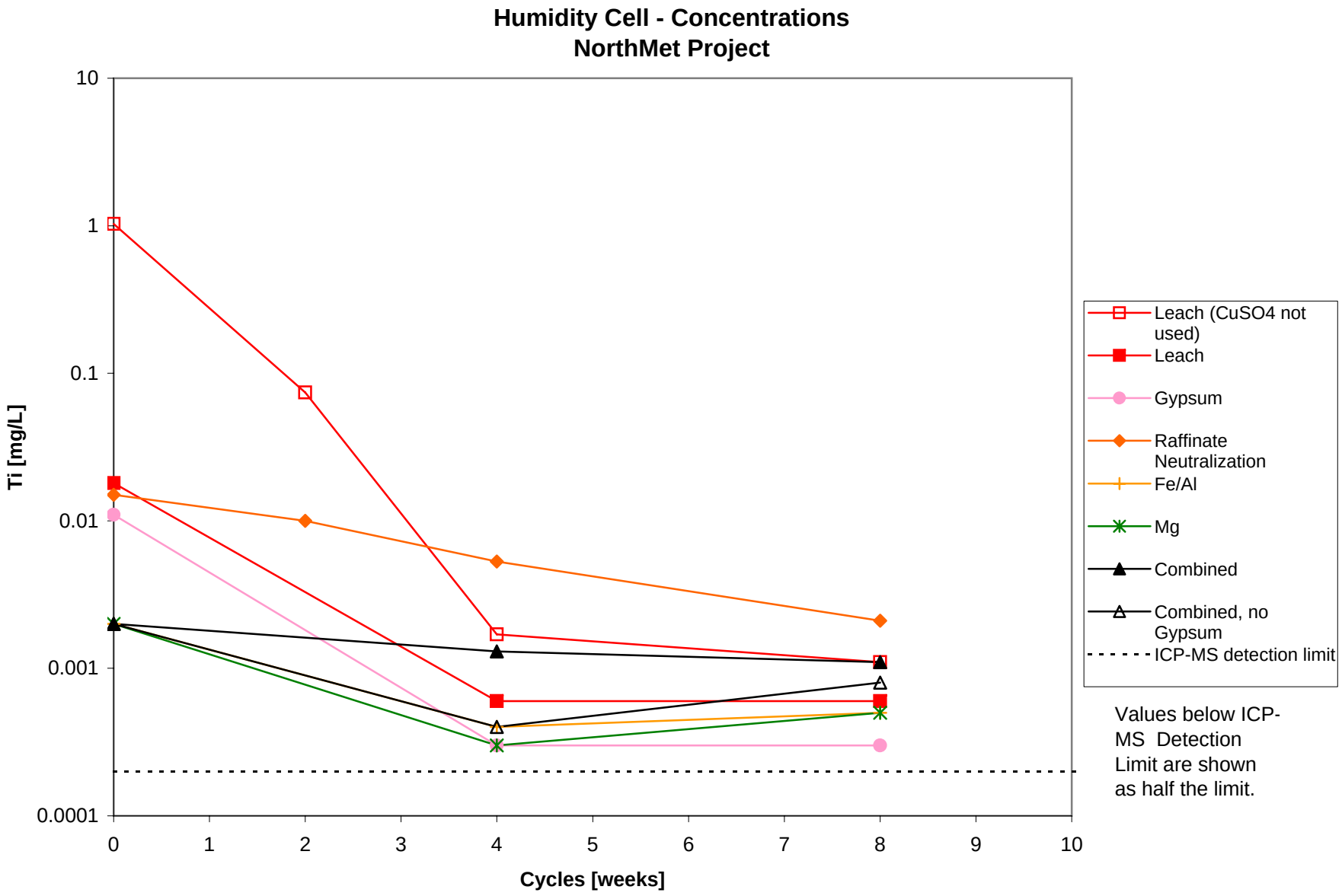
Values below ICP-MS Detection Limit are shown as half the limit.

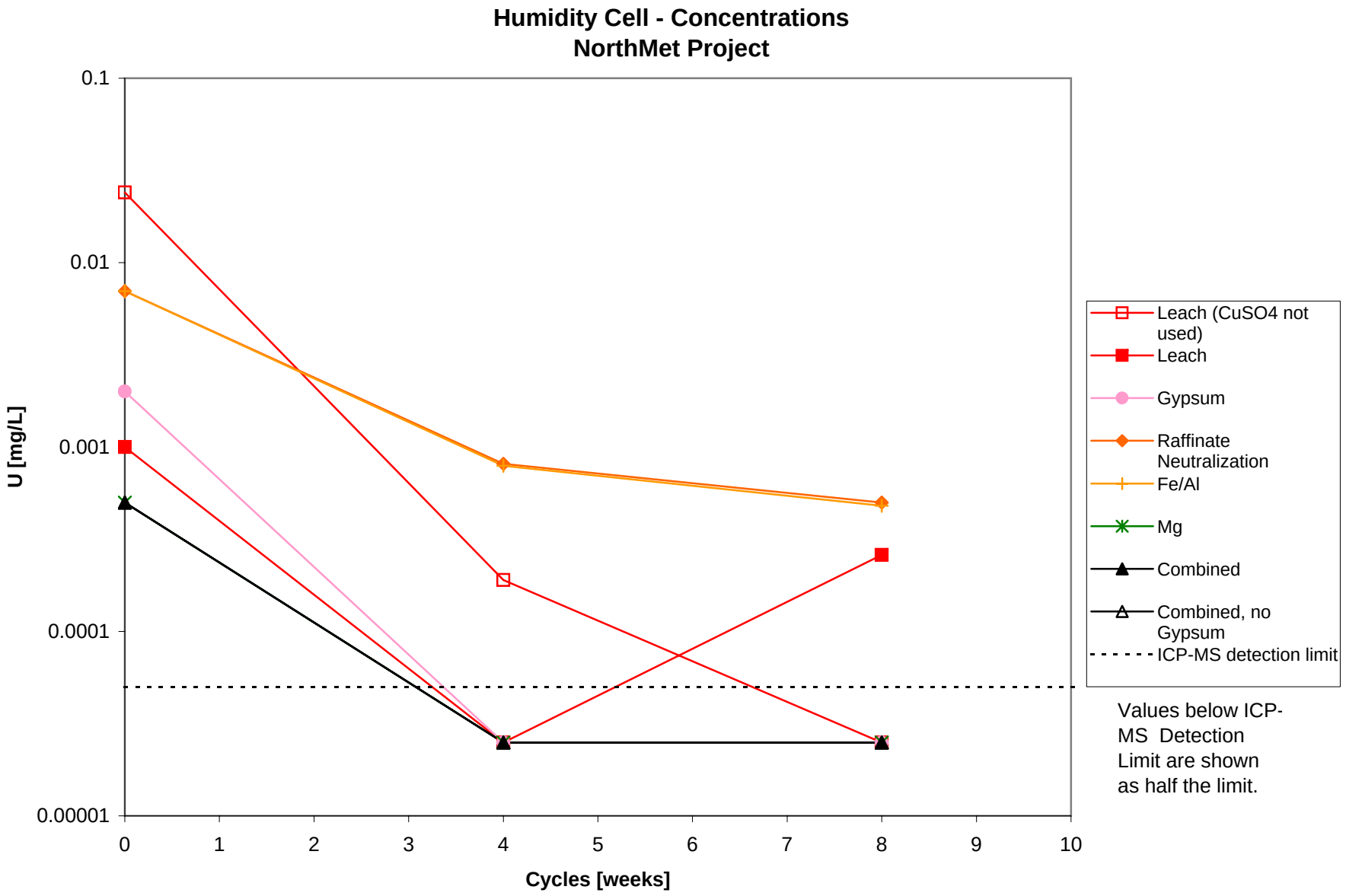

Humidity Cell - Concentrations
NorthMet Project
Te [mg/L]
0.0025
0.002
0.0015
0.001
0.0005
0
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

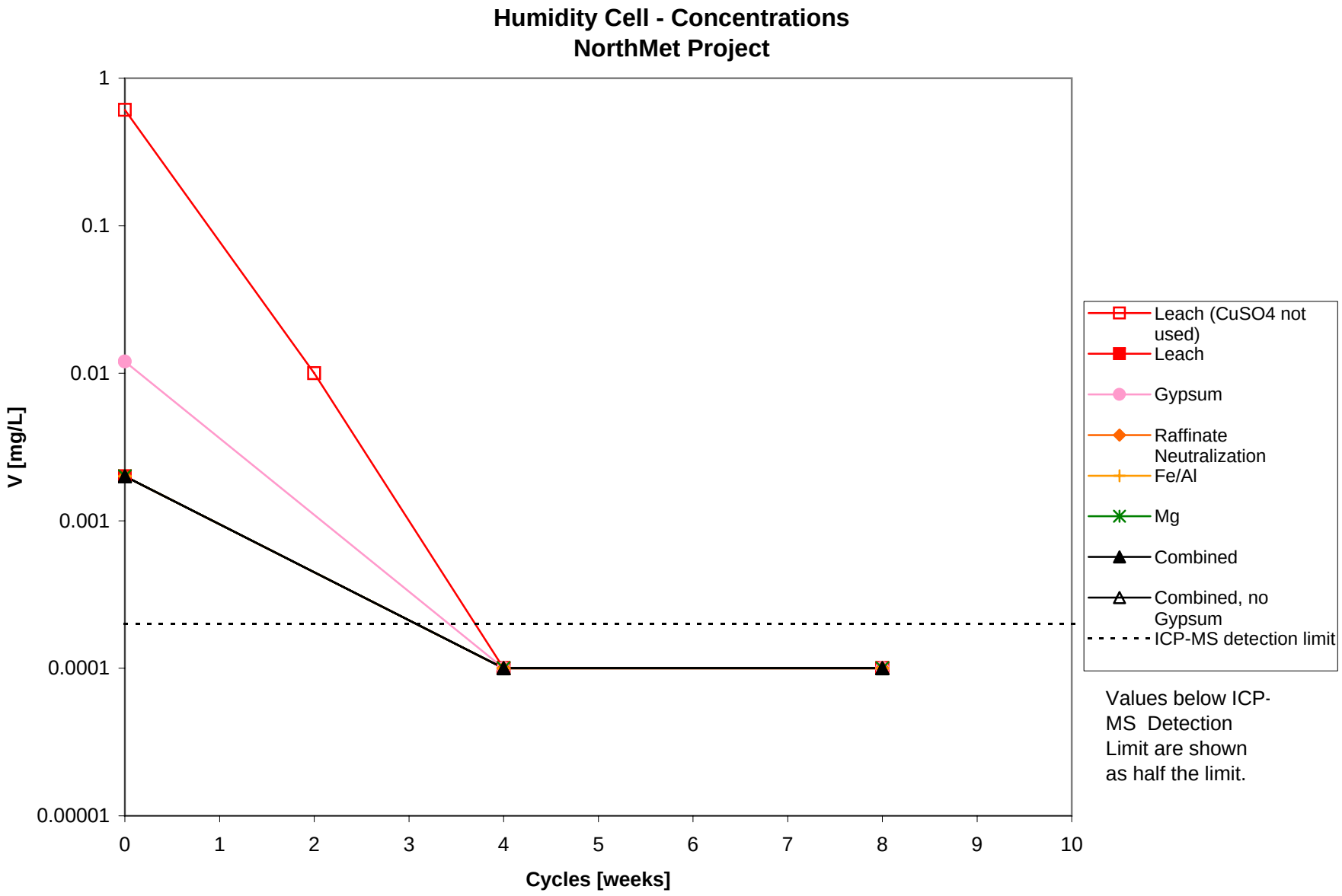

Humidity Cell - Concentrations
NorthMet Project
Tl [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

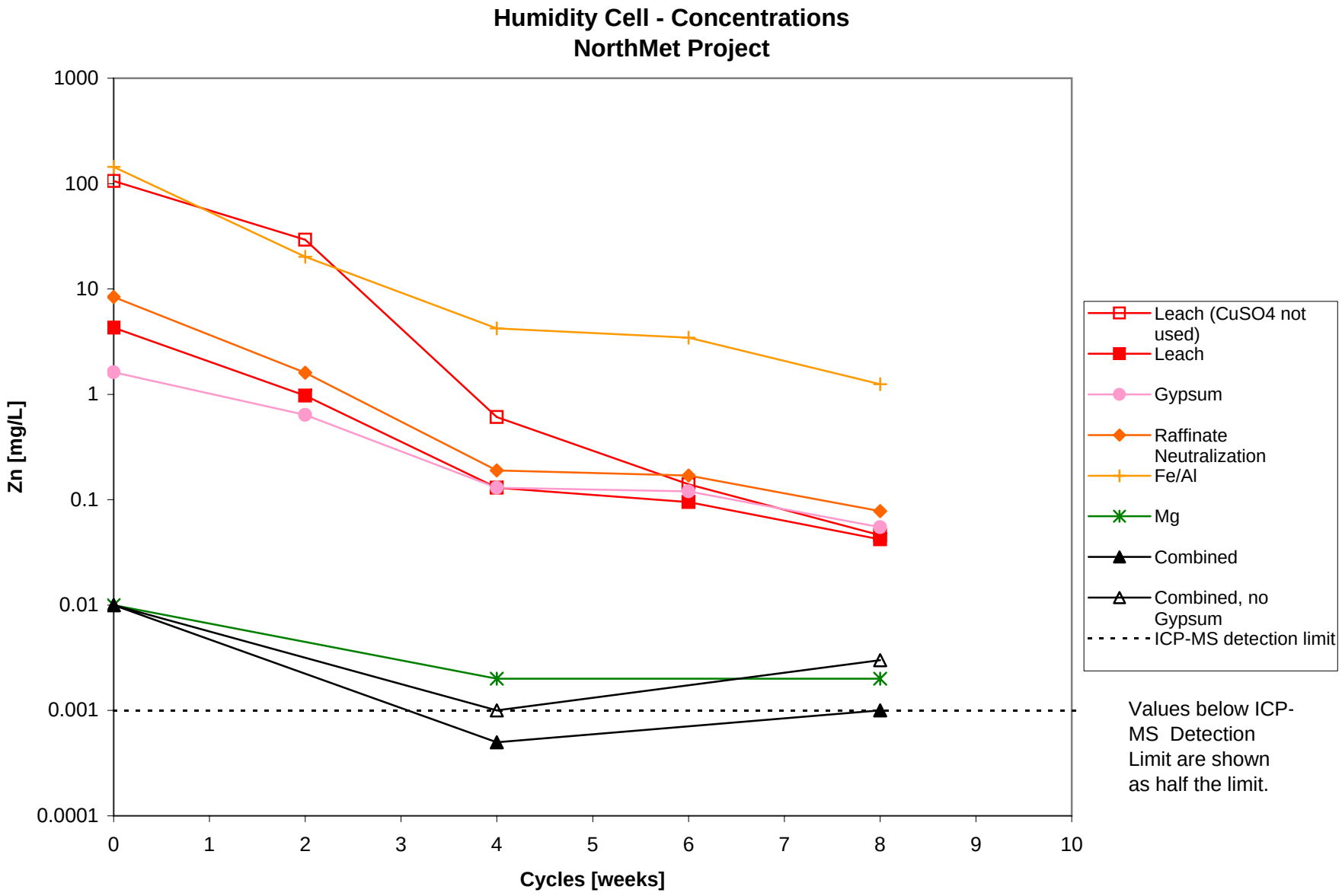

Humidity Cell - Concentrations
NorthMet Project
Tl [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

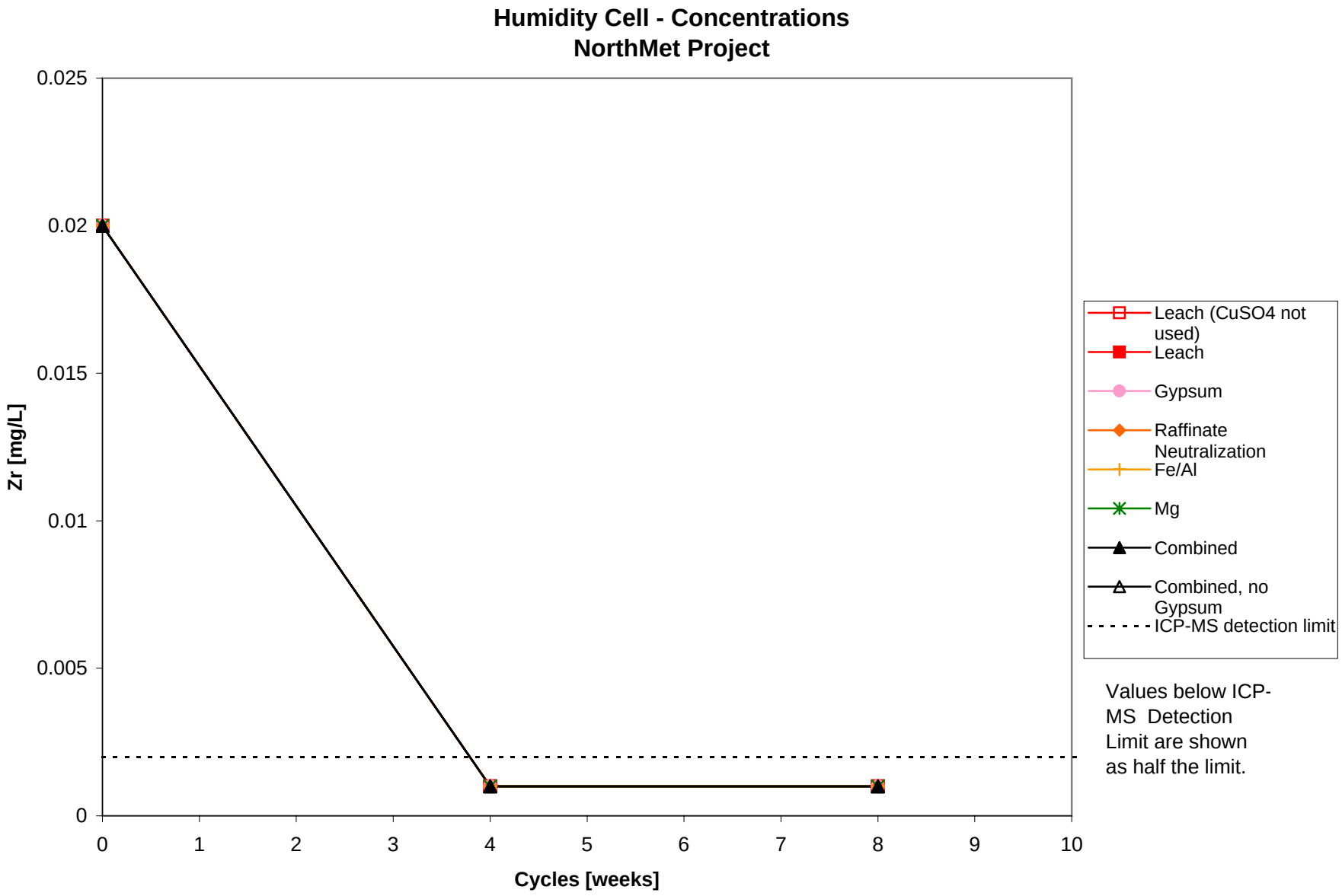

Humidity Cell - Concentrations
NorthMet Project
0.01
0.001
0.0001
0.00001
Sn [mg/L]
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.


Humidity Cell - Concentrations
NorthMet Project
Ti [mg/L]
10
1
0.1
0.01
0.001
0.0001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

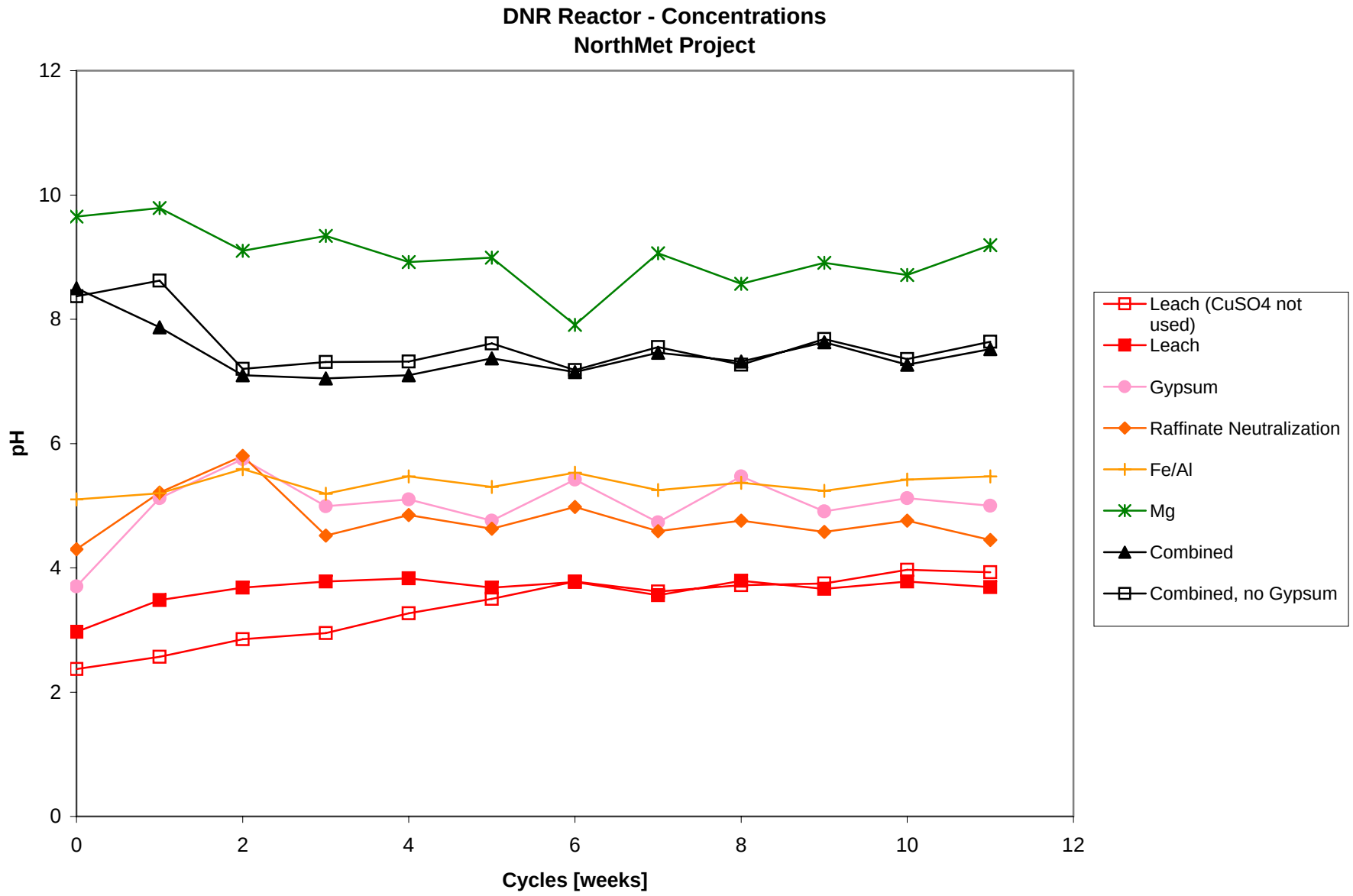

Humidity Cell - Concentrations
NorthMet Project
U [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

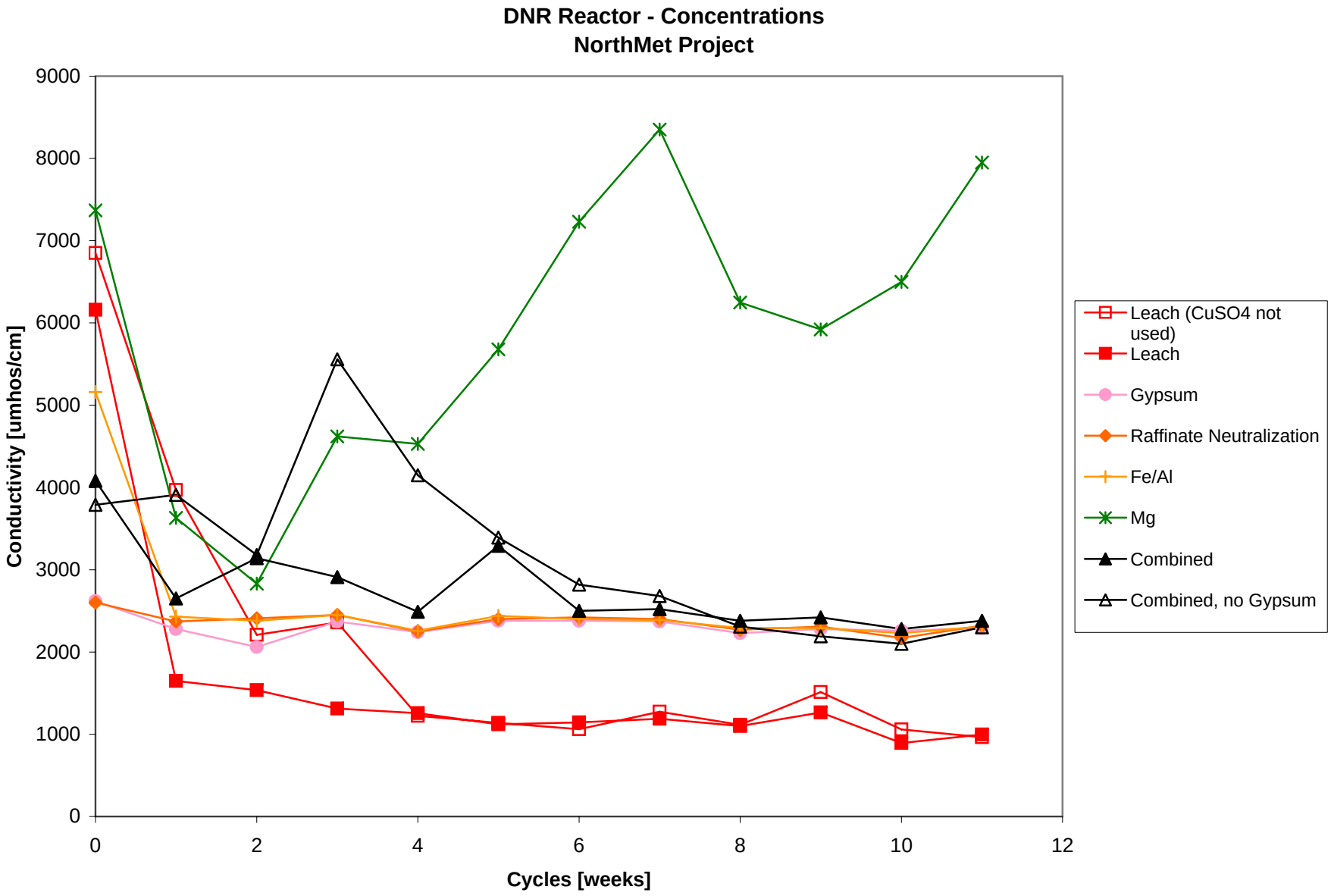

Humidity Cell - Concentrations
NorthMet Project
V [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

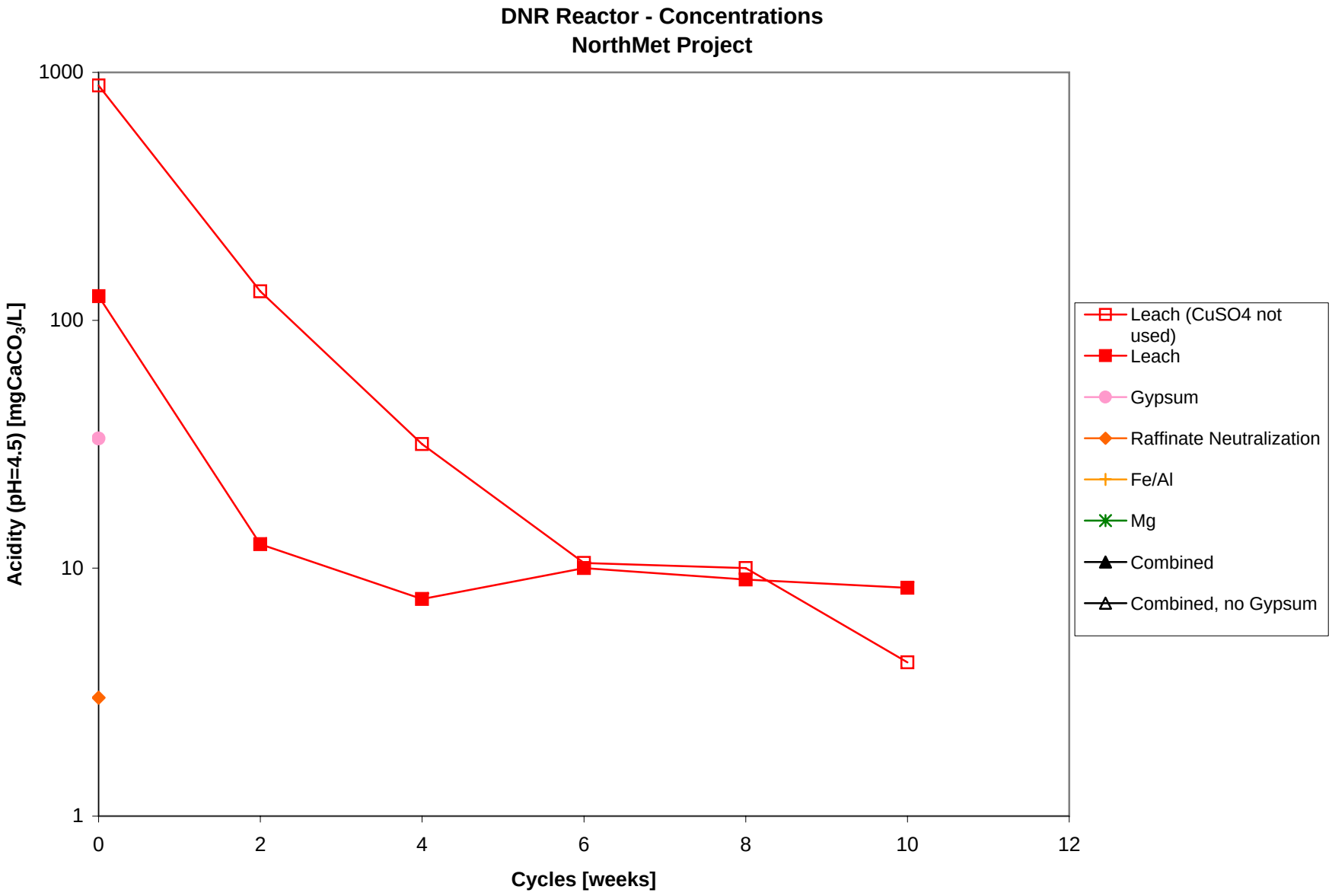

Humidity Cell - Concentrations
NorthMet Project
Zn [mg/L]
1000
100
10
1
0.1
0.01
0.001
0.0001
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

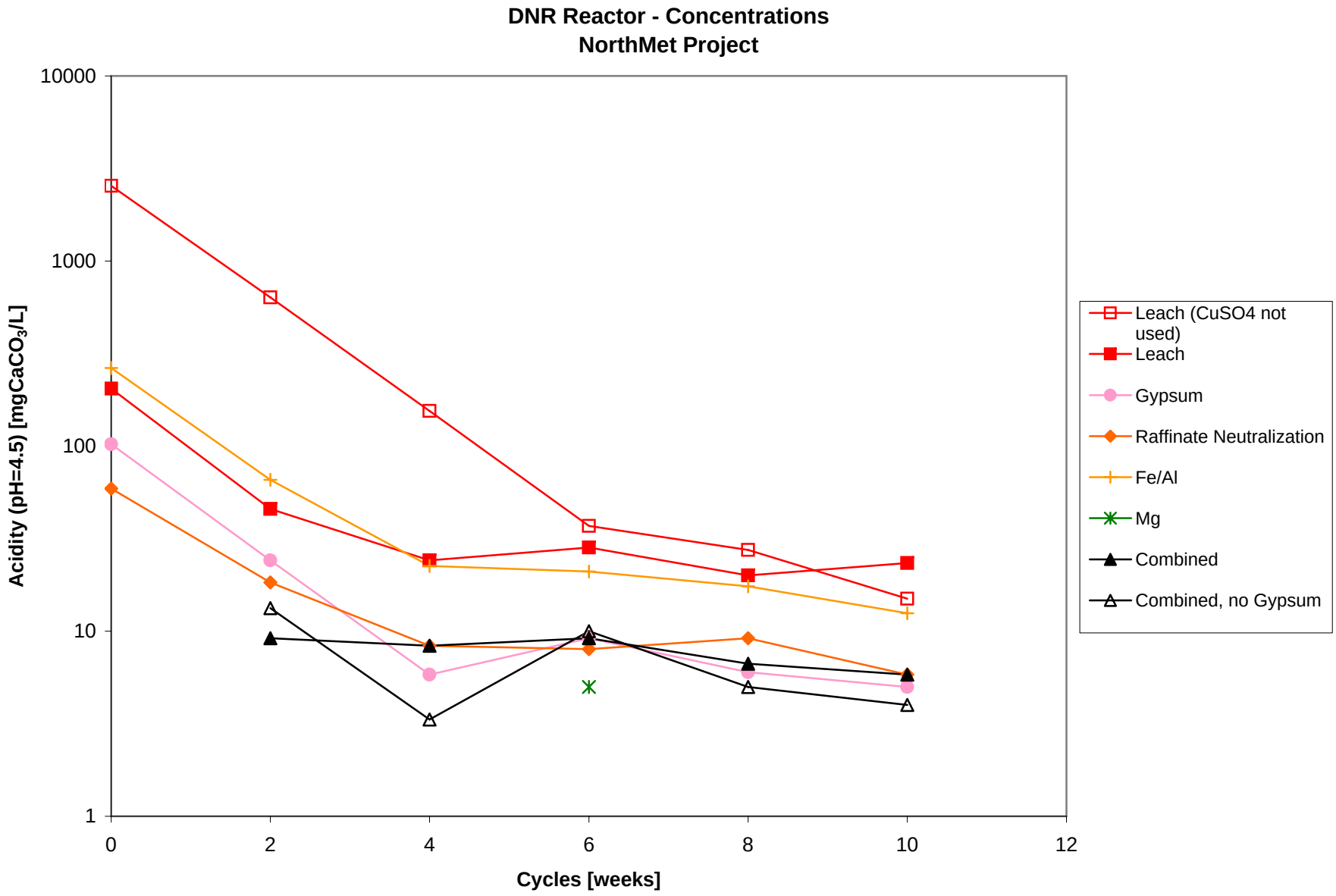

Humidity Cell - Concentrations
NorthMet Project
Zr [mg/L]
0.025
0.02
0.015
0.01
0.005
0
0
1
2
3
4
5
6
7
8
9
10
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

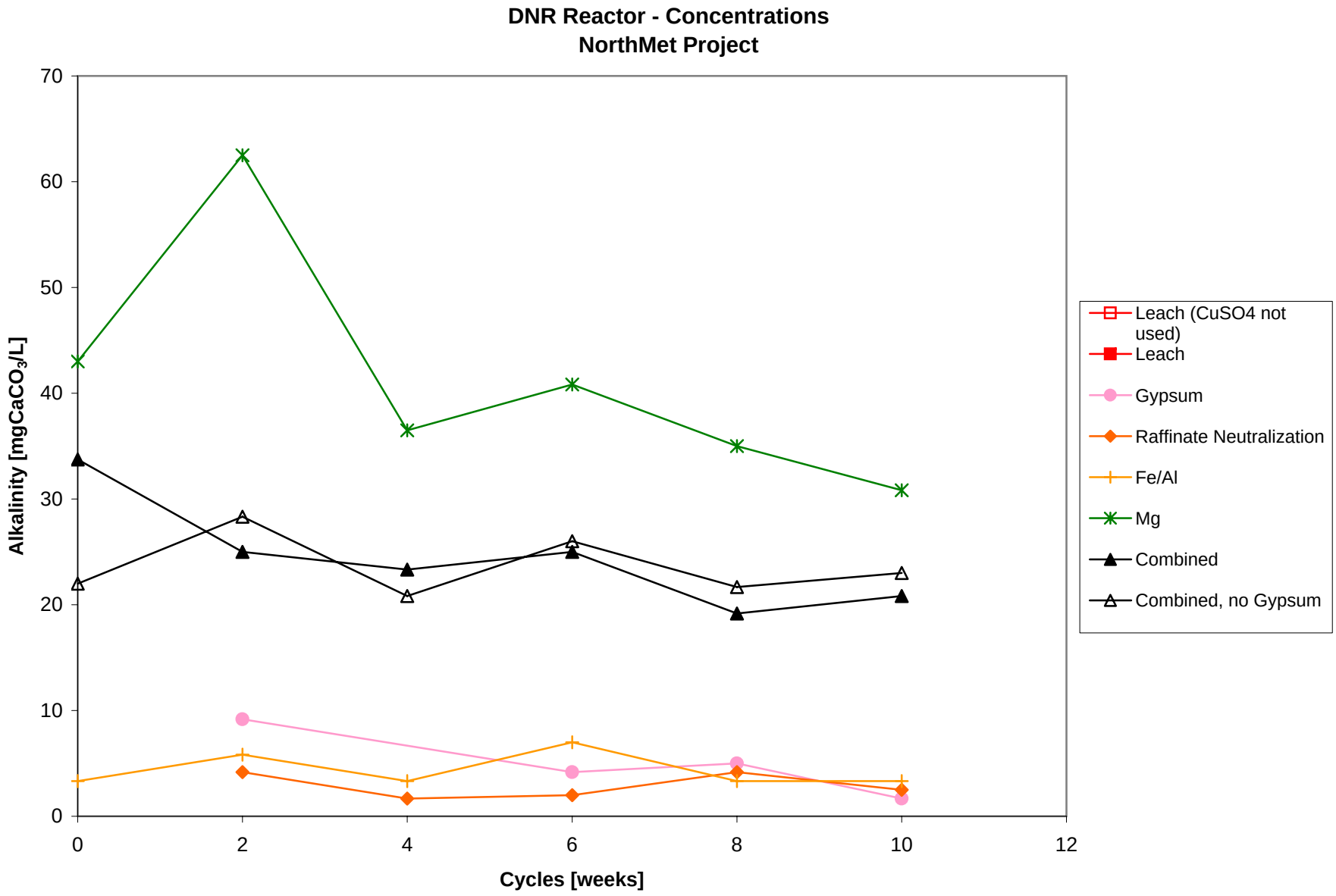
Appendix C.3
DNR Reactors

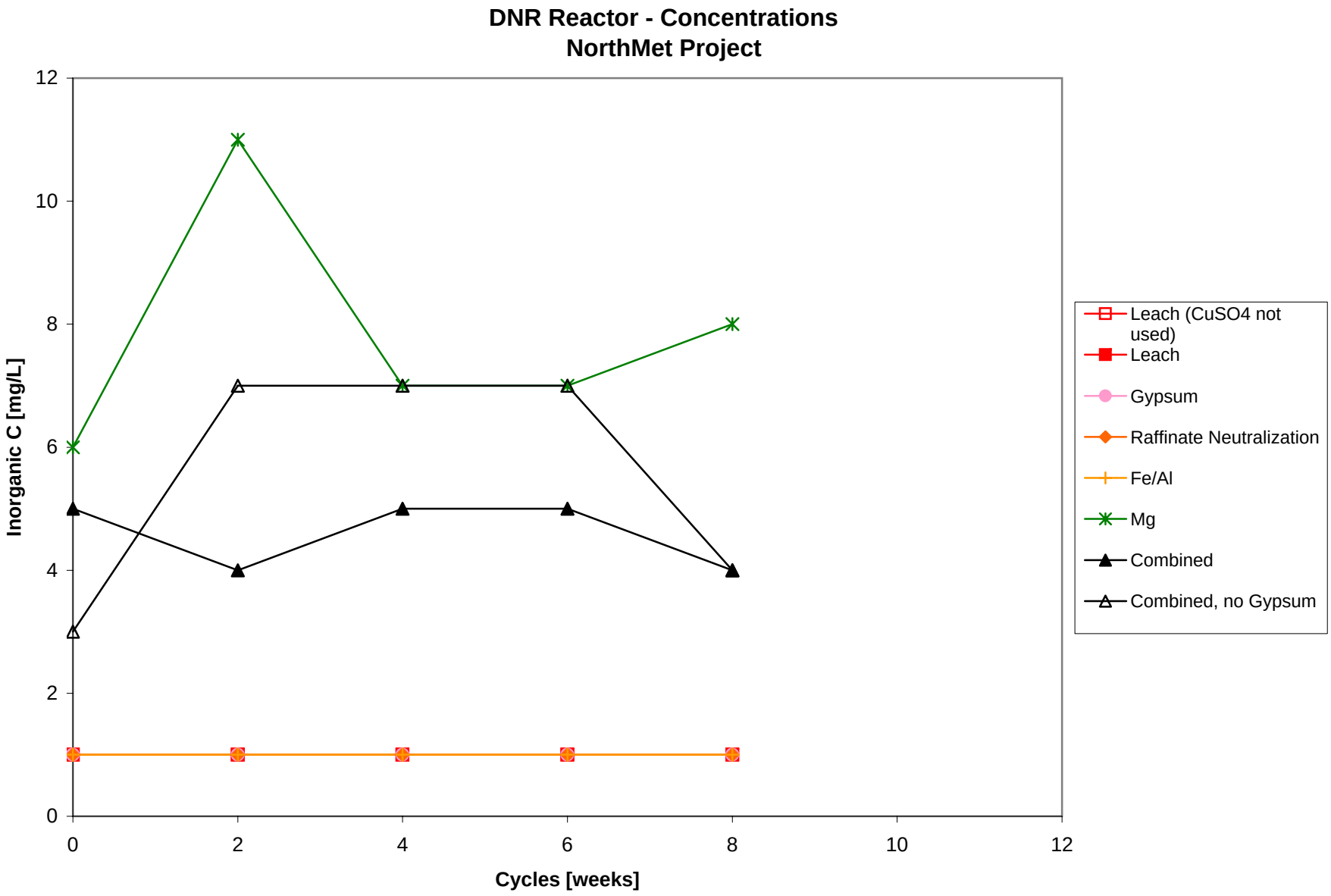

DNR Reactor - Concentrations
NorthMet Project
pH
Cycles [weeks]
0
2
4
6
8
10
12
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

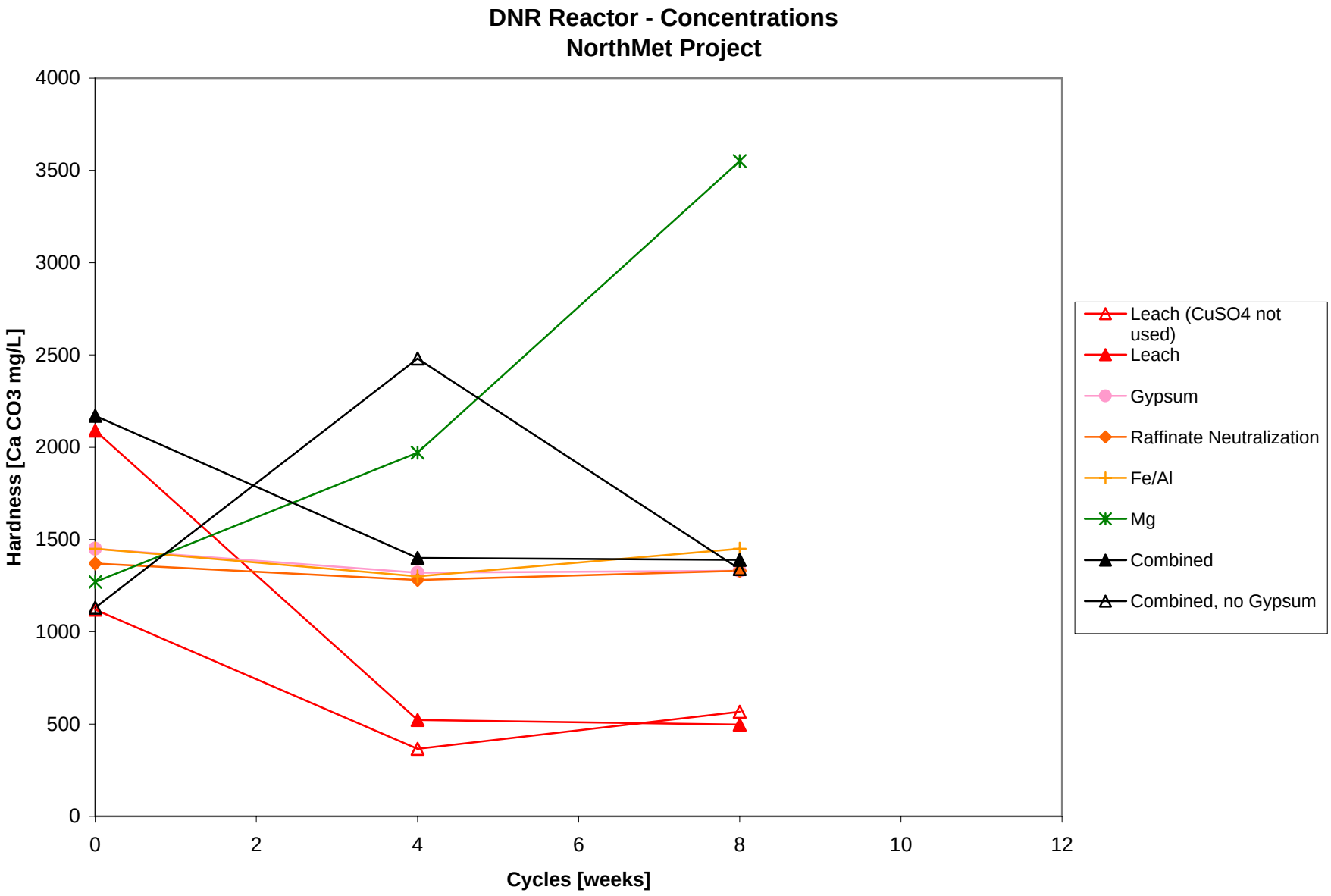

DNR Reactor - Concentrations
NorthMet Project
Conductivity [umhos/cm]
Cycles [weeks]
0
1000
2000
3000
4000
5000
6000
7000
8000
9000
0
2
4
6
8
10
12
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

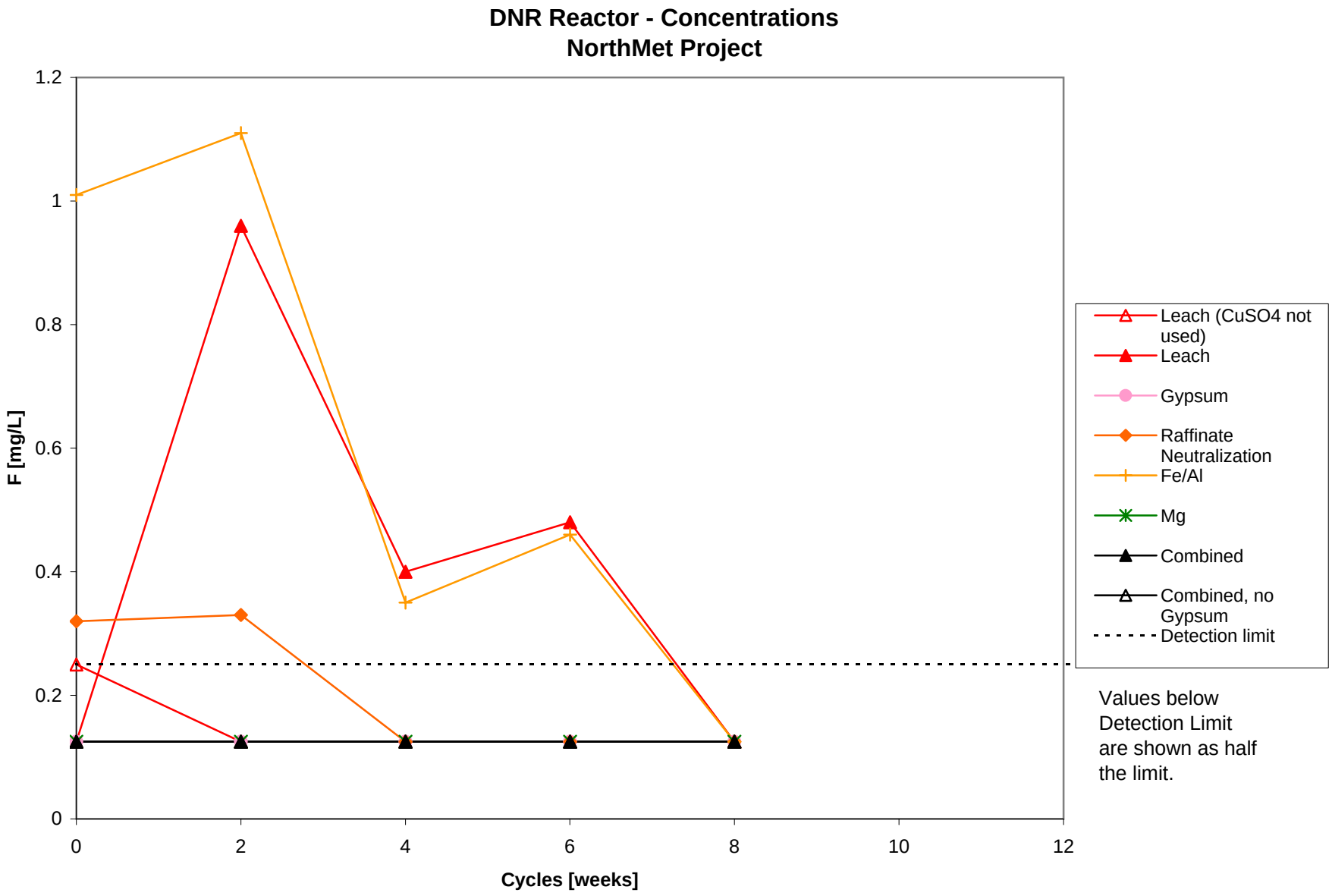

DNR Reactor - Concentrations
NorthMet Project
Acidity (pH=4.5) [mgCaCO3/L]
1000
100
10
1
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

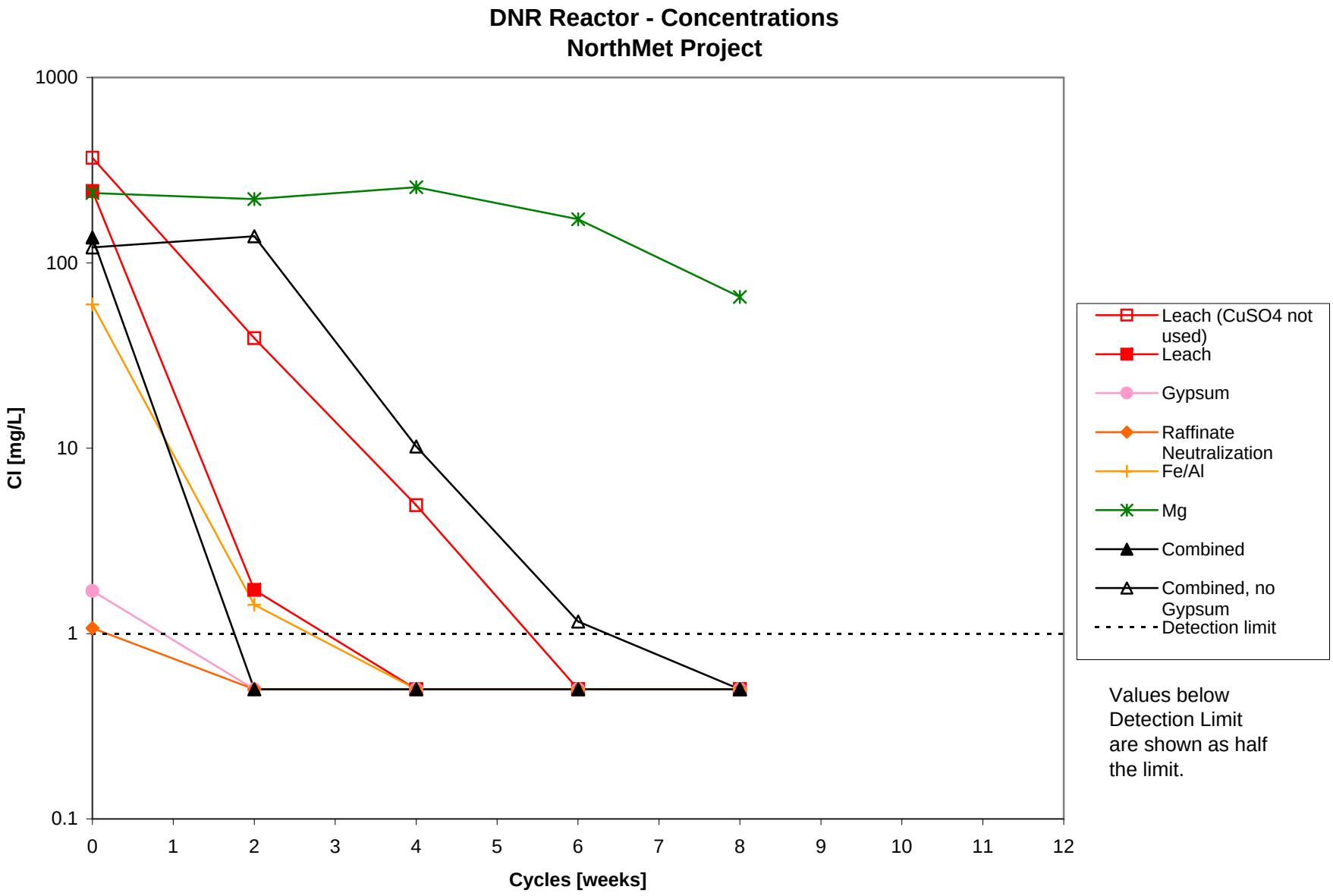

DNR Reactor - Concentrations
NorthMet Project
Acidity (pH=4.5) [mgCaCO3/L]
10000
1000
100
10
1
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

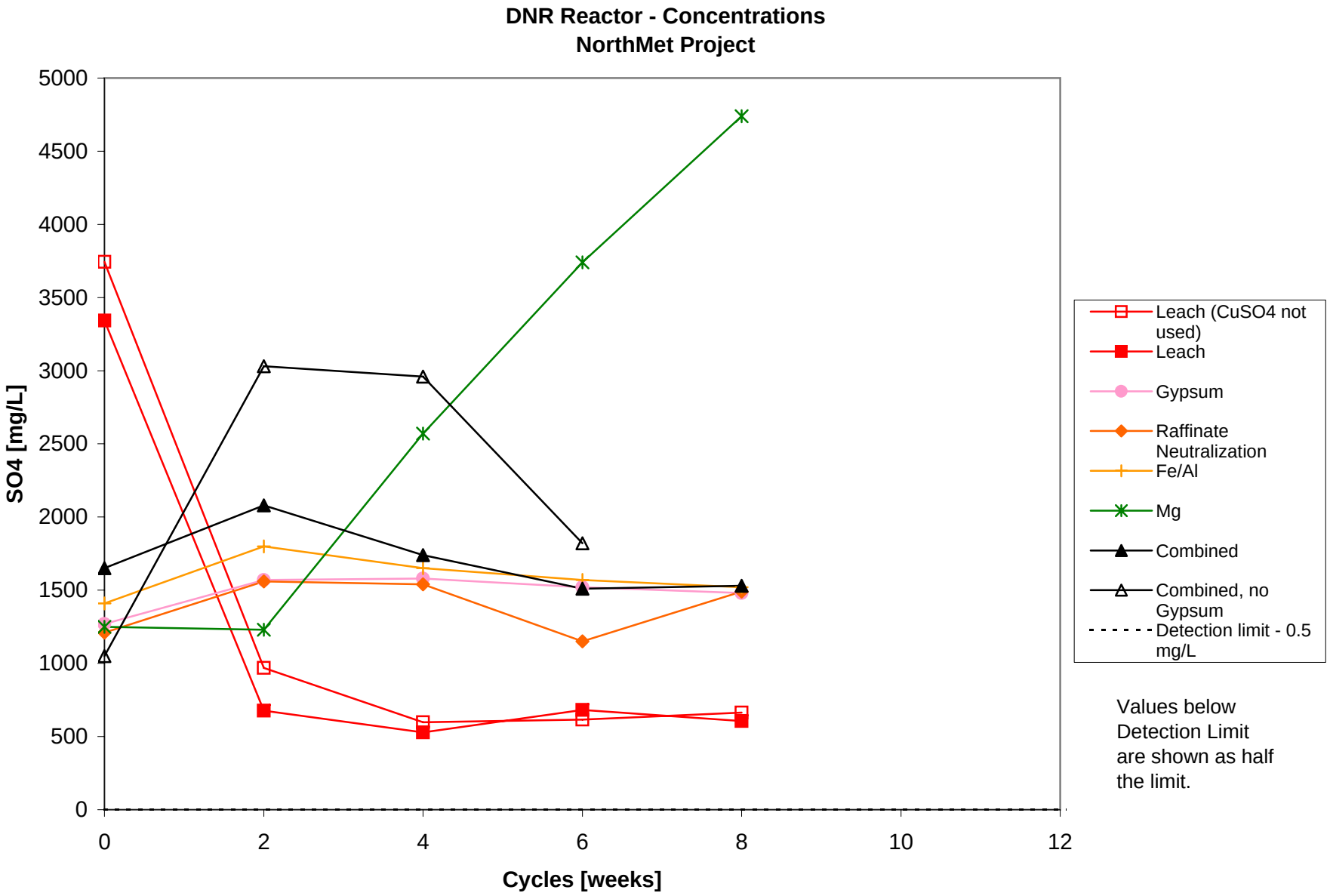

DNR Reactor - Concentrations
NorthMet Project
Alkalinity [mgCaCO3/L]
Cycles [weeks]
0
10
20
30
40
50
60
70
0
2
4
6
8
10
12
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum


DNR Reactor - Concentrations
NorthMet Project
Inorganic C [mg/L]
Cycles [weeks]
0
2
4
6
8
10
12
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

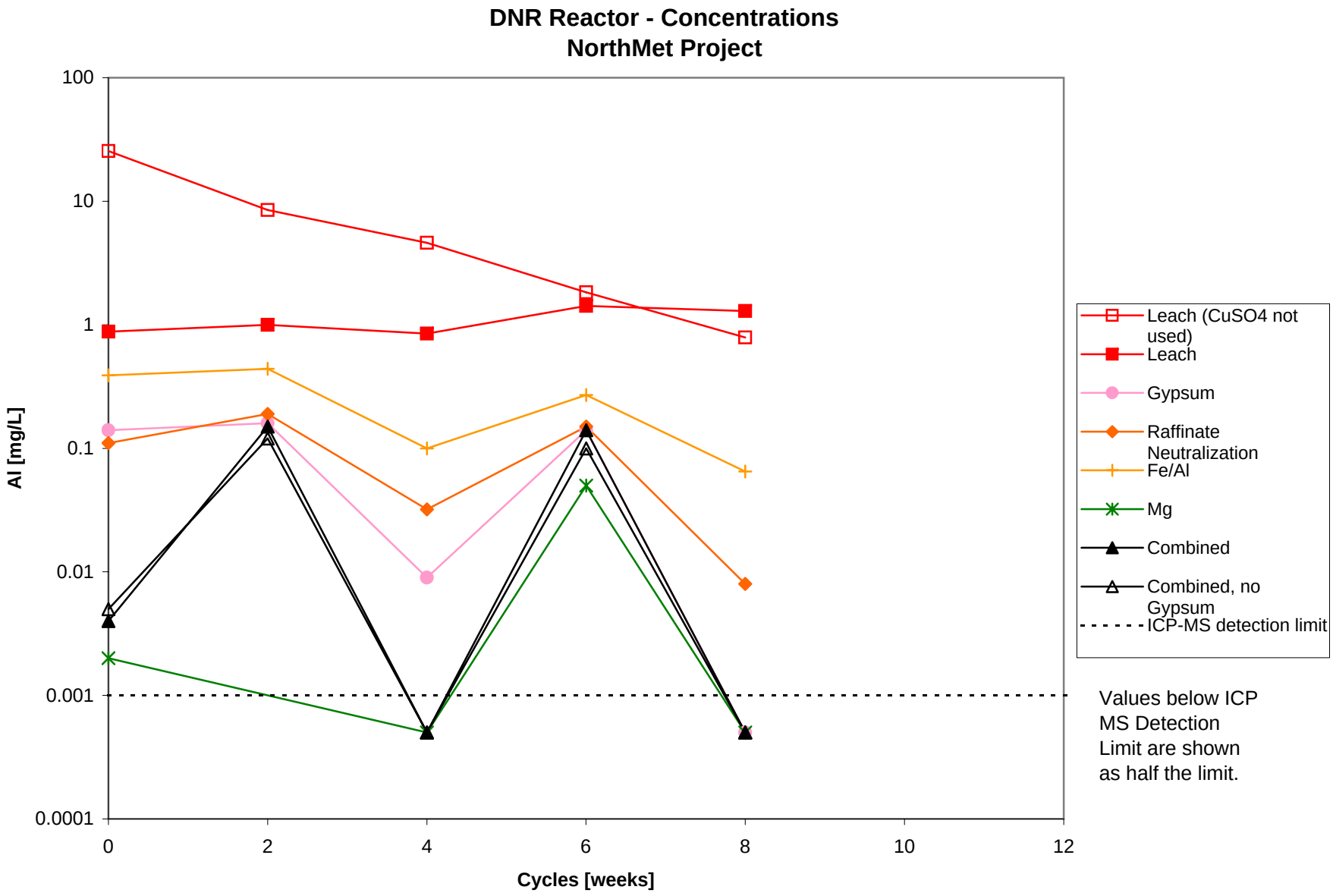

DNR Reactor - Concentrations
NorthMet Project
Hardness [Ca CO3 mg/L]
4000
3500
3000
2500
2000
1500
1000
500
0
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

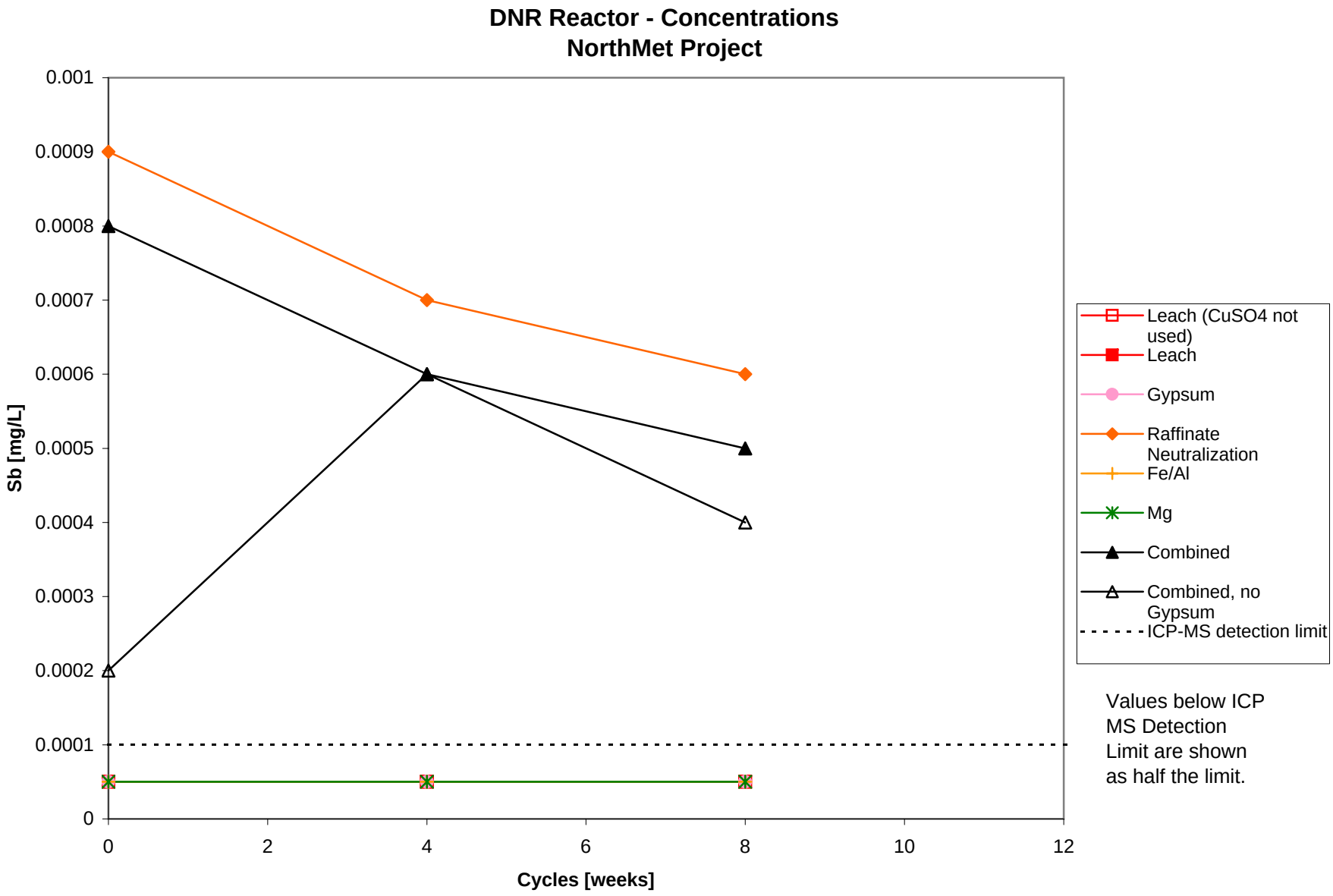

DNR Reactor - Concentrations
NorthMet Project
F [mg/L]
1.2
1
0.8
0.6
0.4
0.2
0
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
Detection limit
Values below Detection Limit are shown as half the limit.

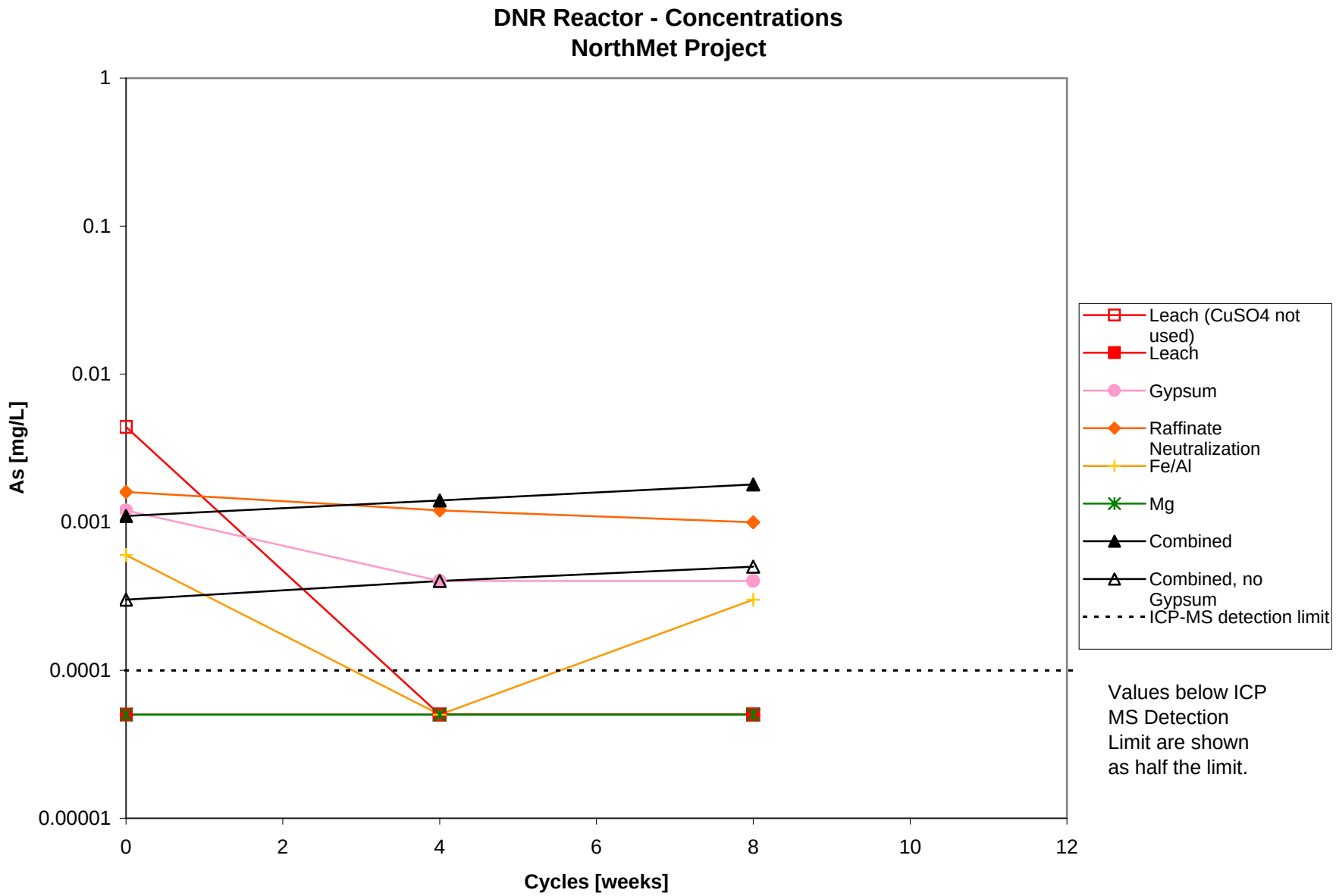

DNR Reactor - Concentrations
NorthMet Project
Cl [mg/L]
1000
100
10
1
0.1
0
1
2
3
4
5
6
7
8
9
10
11
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
Detection limit
Values below Detection Limit are shown as half the limit.

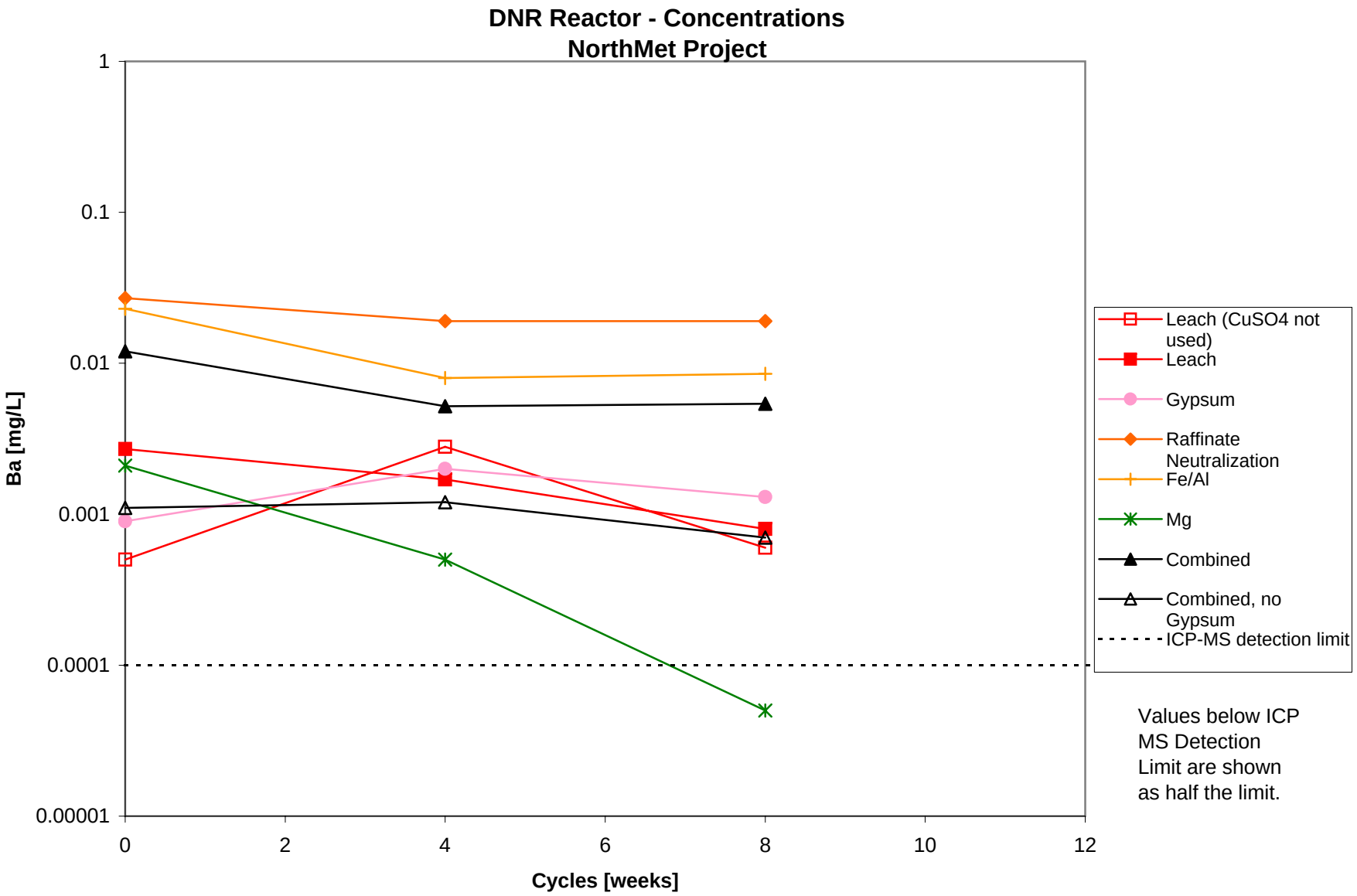


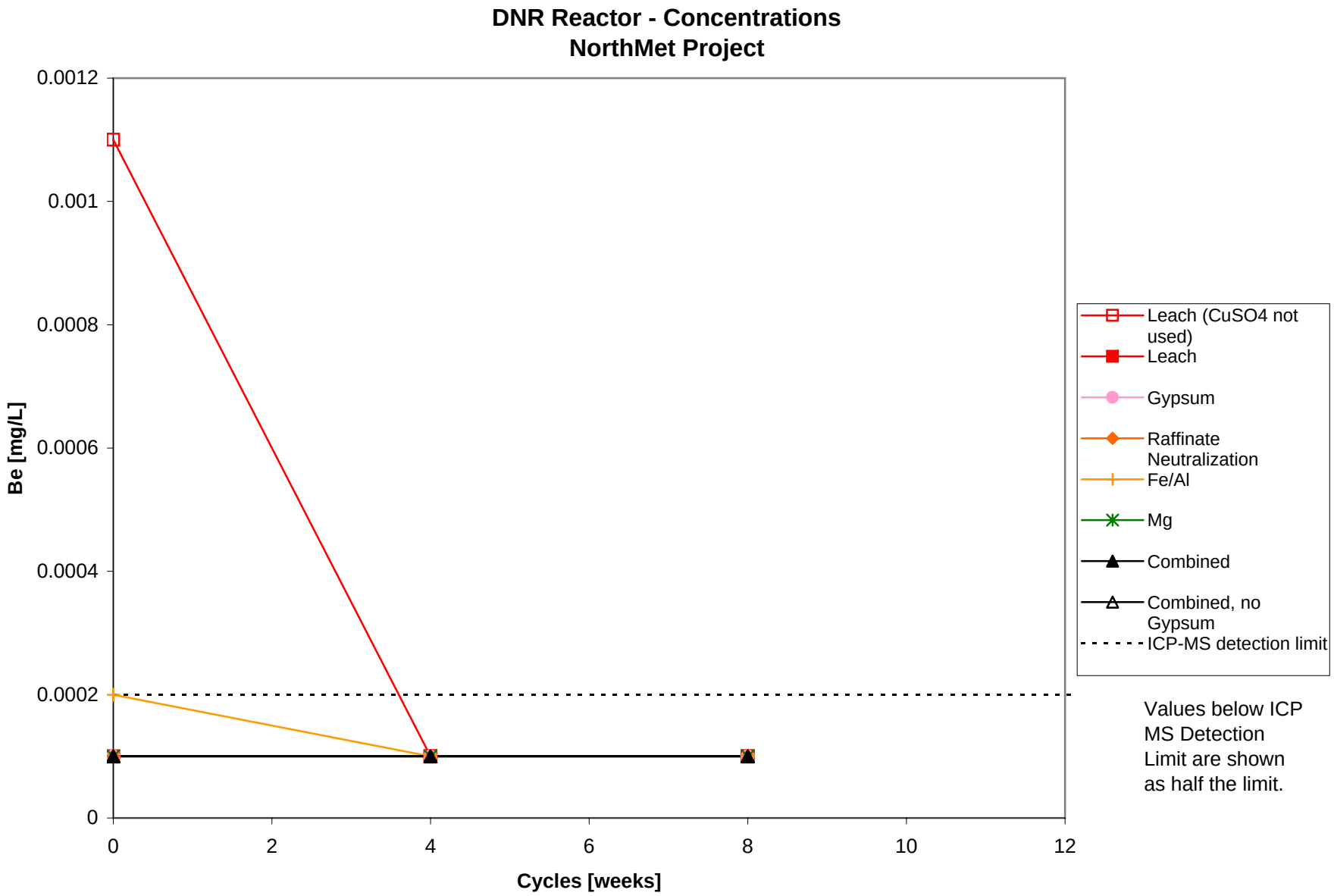
Values below Detection Limit are shown as half the limit.

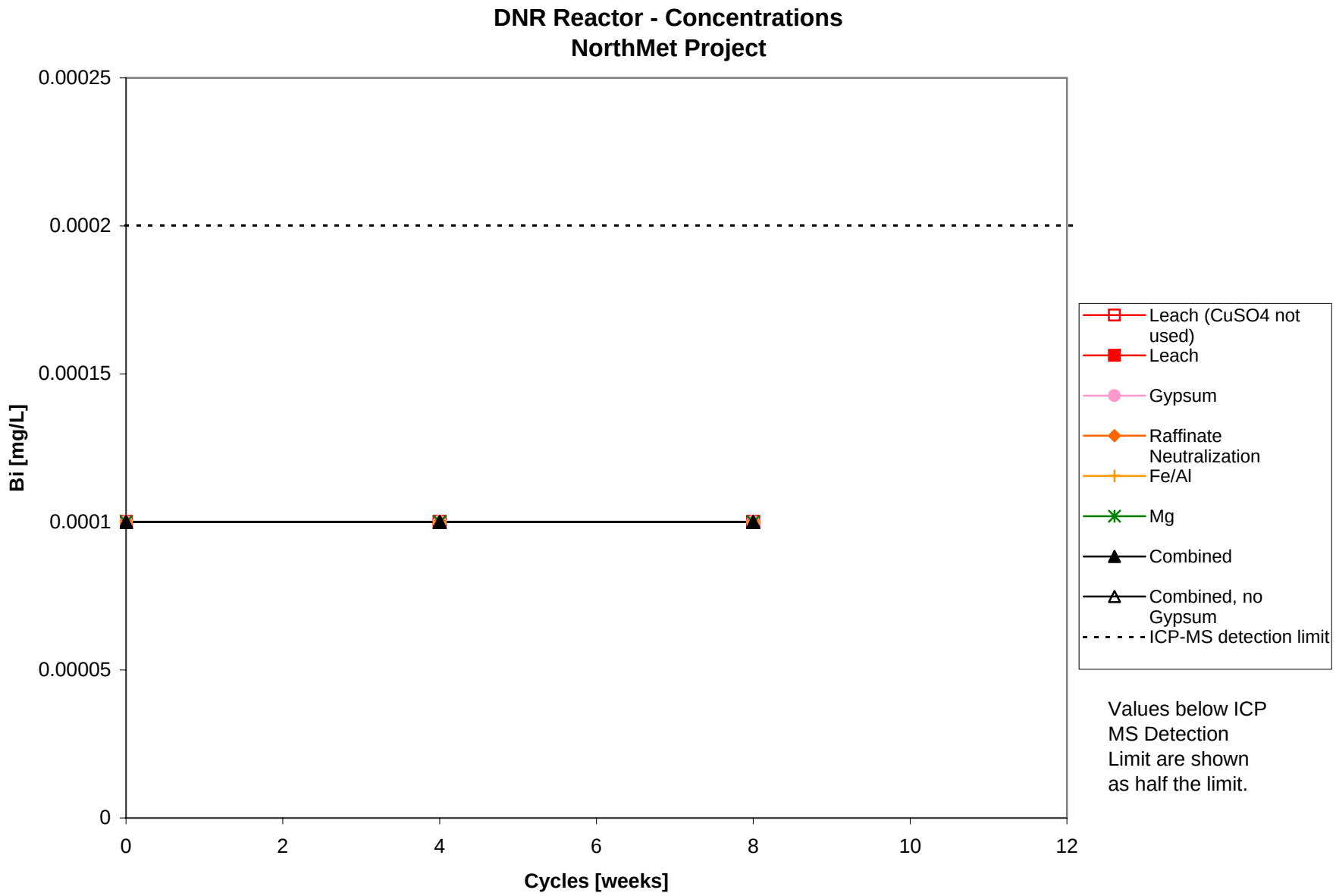

DNR Reactor - Concentrations
NorthMet Project
Al [mg/L]
100
10
1
0.1
0.01
0.001
0.0001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

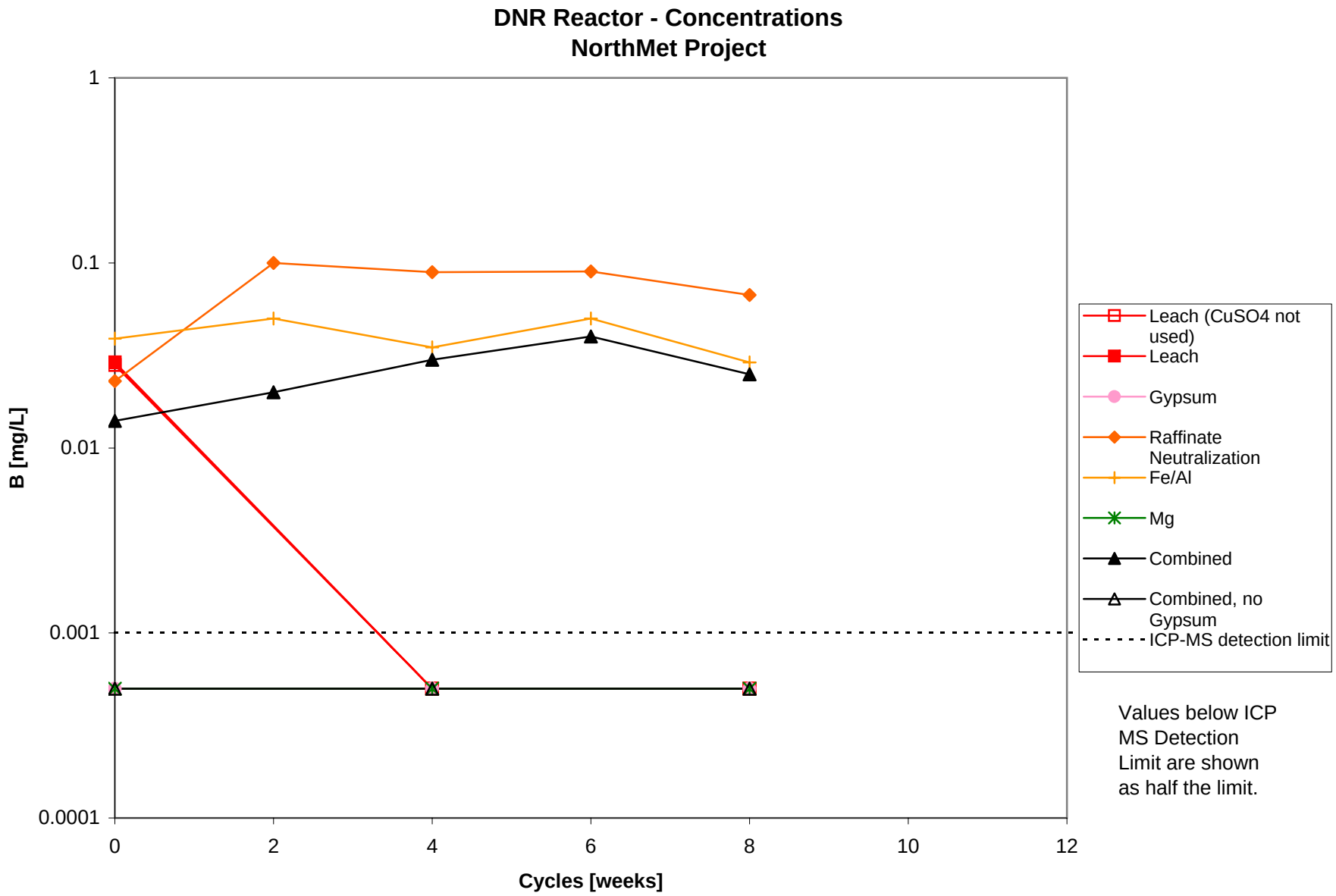

DNR Reactor - Concentrations
NorthMet Project
Sb [mg/L]
0.001
0.0009
0.0008
0.0007
0.0006
0.0005
0.0004
0.0003
0.0002
0.0001
0
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

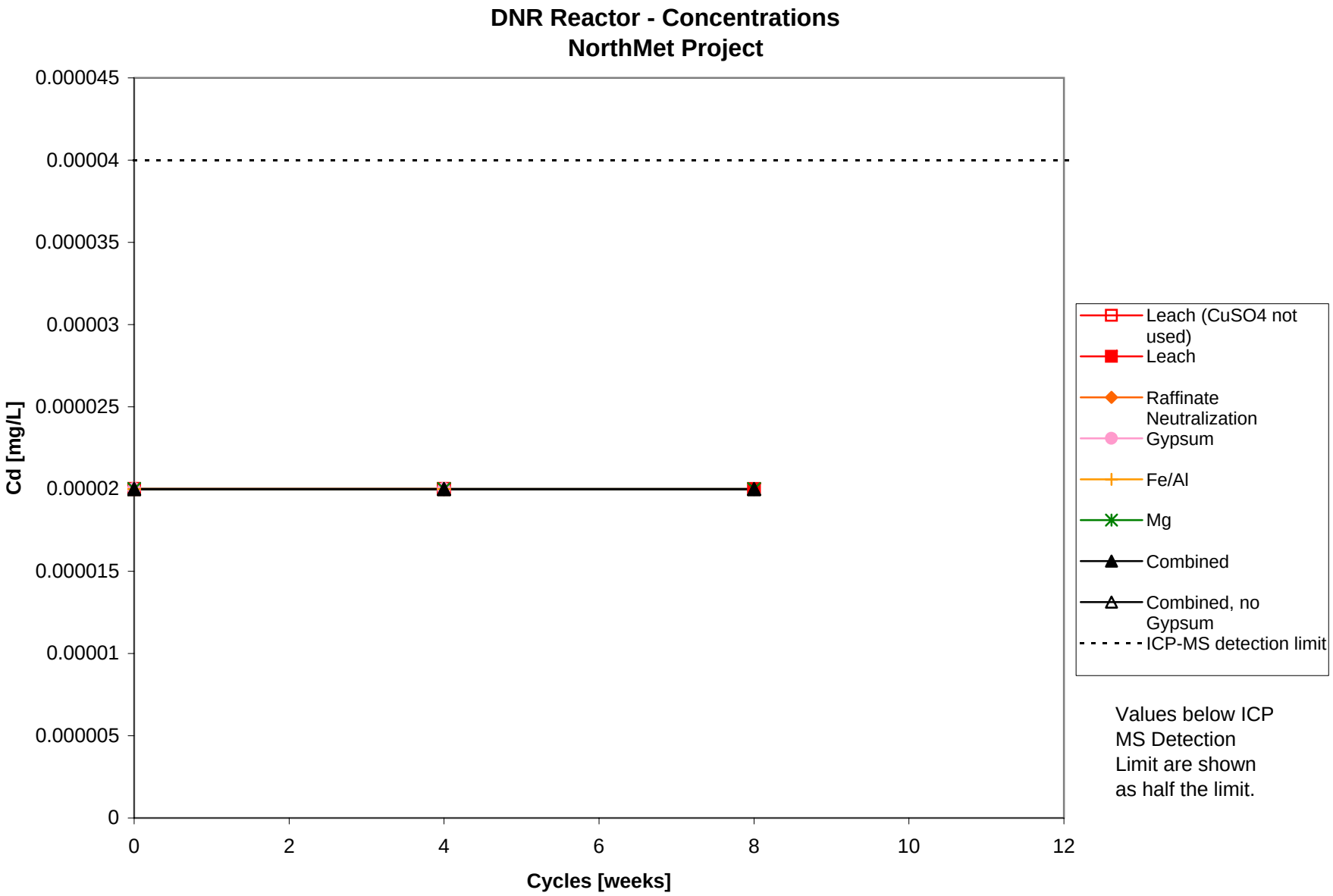

DNR Reactor - Concentrations
NorthMet Project
As [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

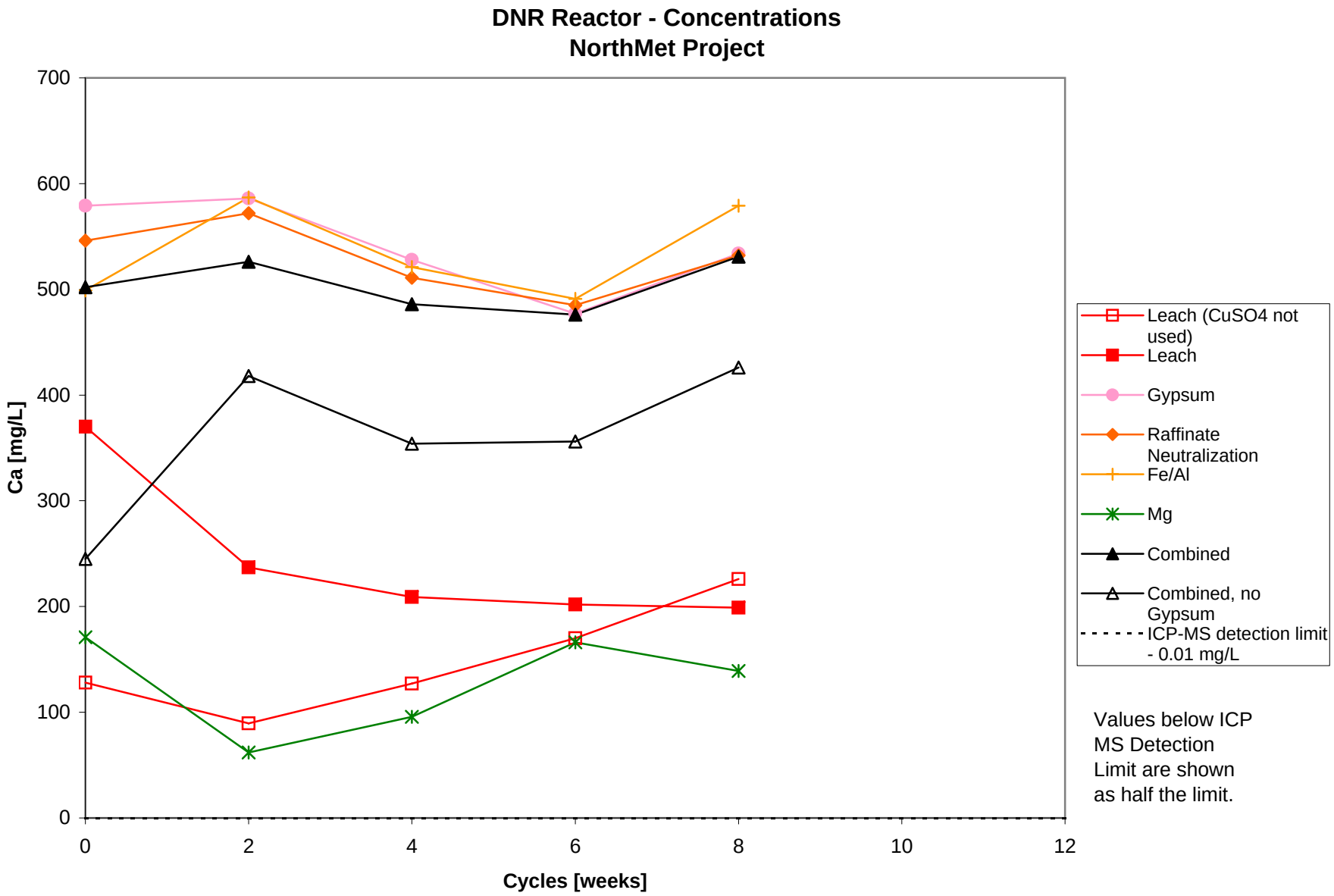

DNR Reactor - Concentrations
NorthMet Project
Ba [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

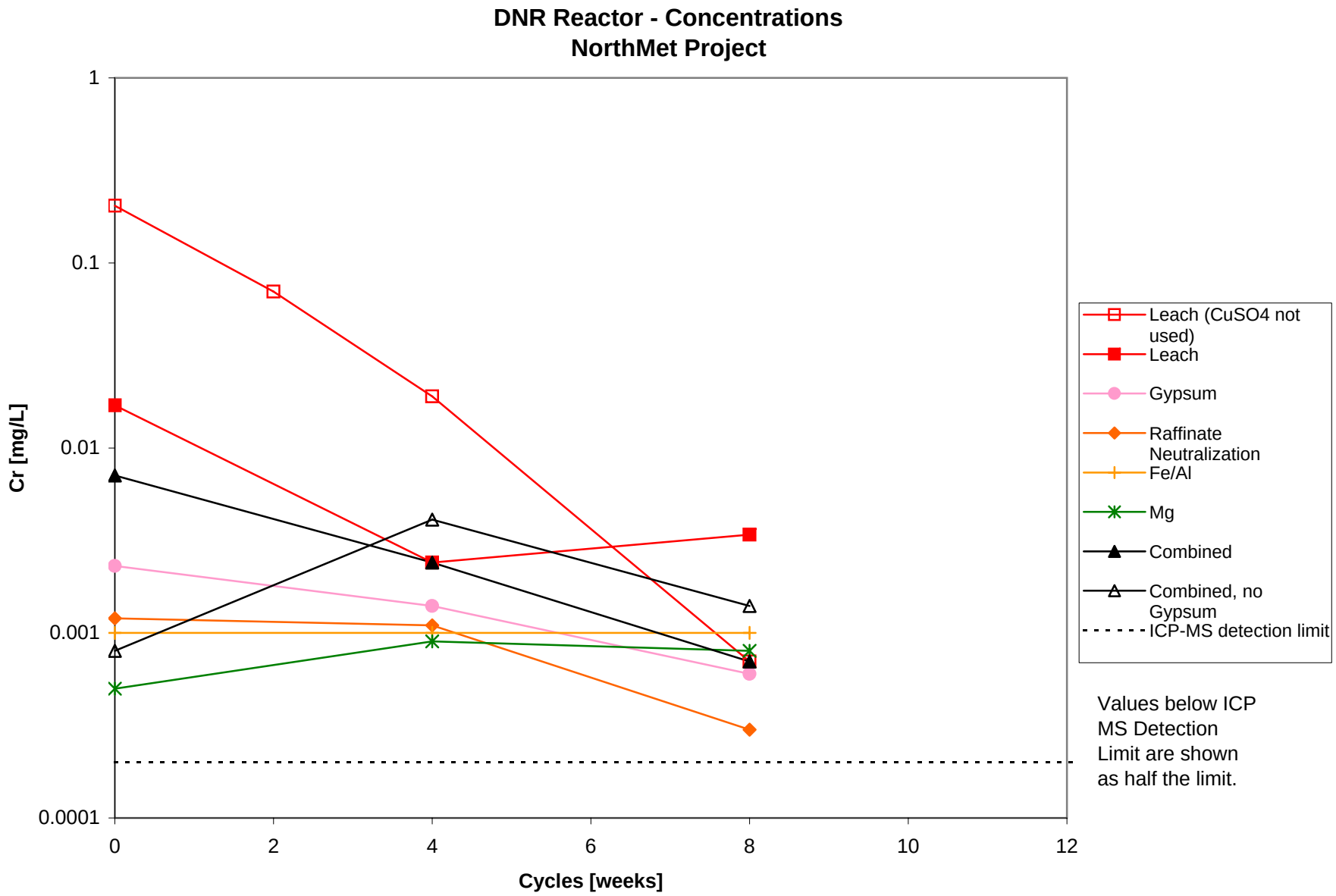

DNR Reactor - Concentrations
NorthMet Project
Be [mg/L]
0.0012
0.001
0.0008
0.0006
0.0004
0.0002
0
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

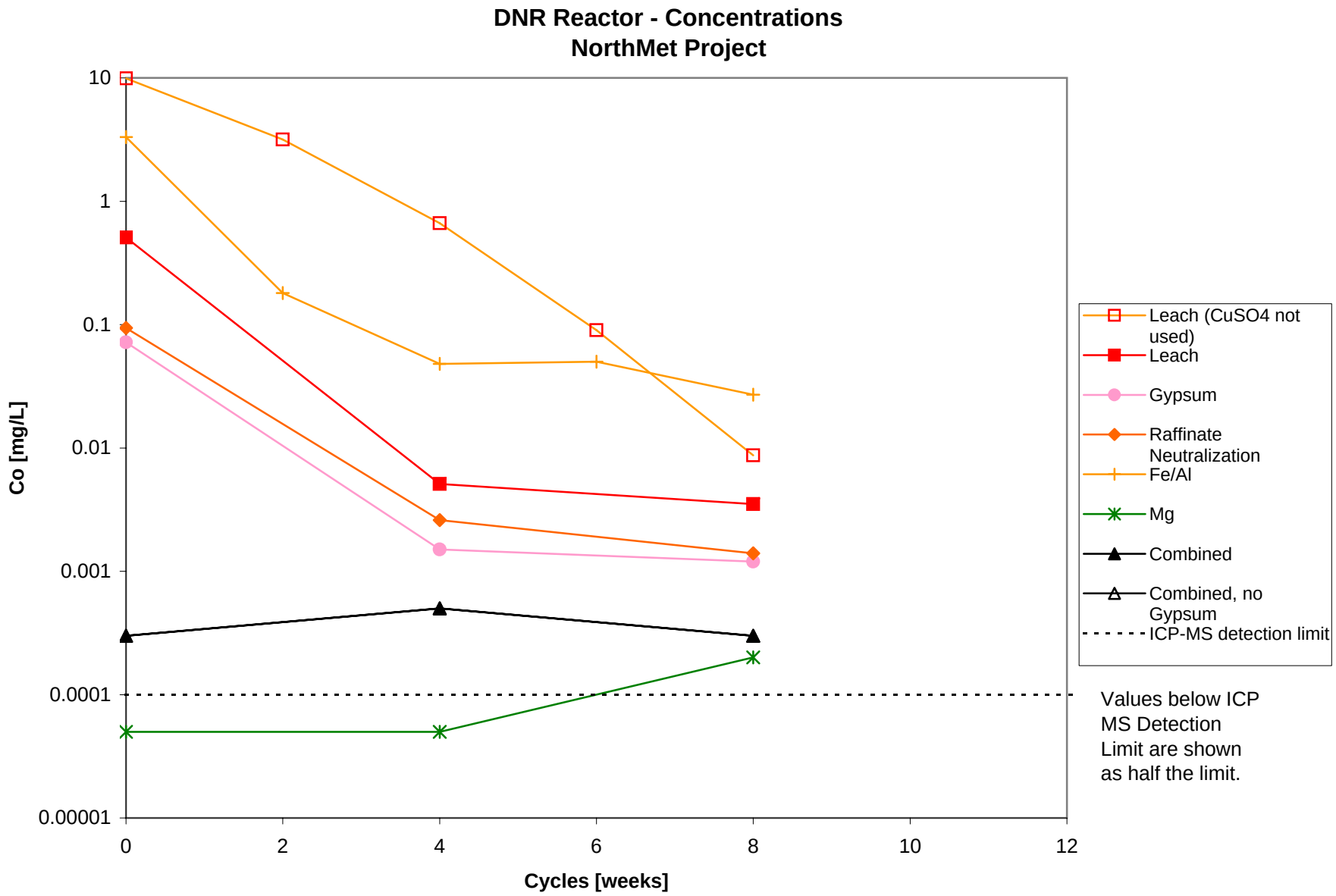

DNR Reactor - Concentrations
NorthMet Project
0.00025
0.0002
0.00015
0.0001
0.00005
0
Bi [mg/L]
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

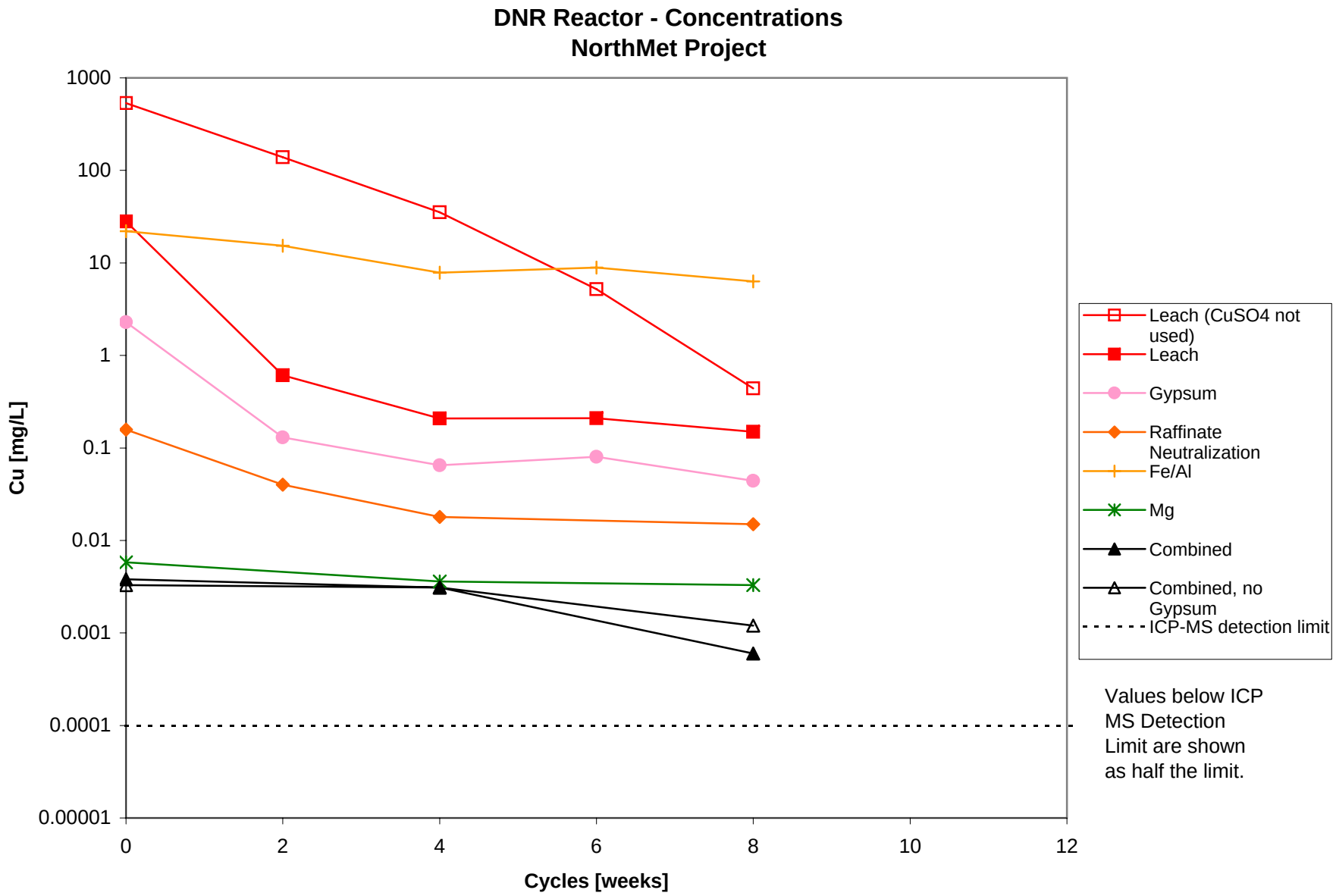

DNR Reactor - Concentrations
NorthMet Project
B [mg/L]
1
0.1
0.01
0.001
0.0001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

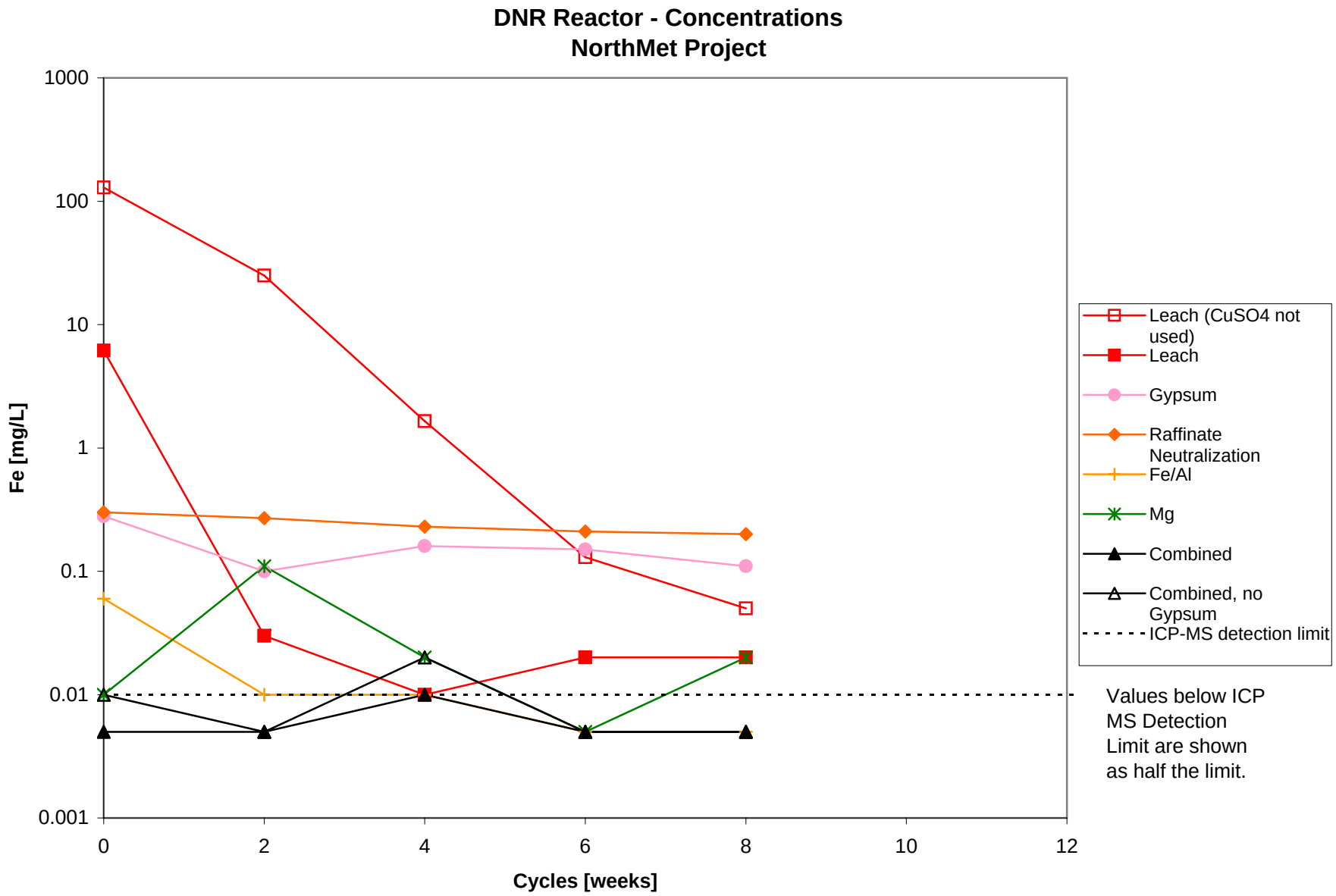

DNR Reactor - Concentrations
NorthMet Project
Cd [mg/L]
0.000045
0.00004
0.000035
0.00003
0.000025
0.00002
0.000015
0.00001
0.000005
0
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Raffinate Neutralization
Gypsum
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

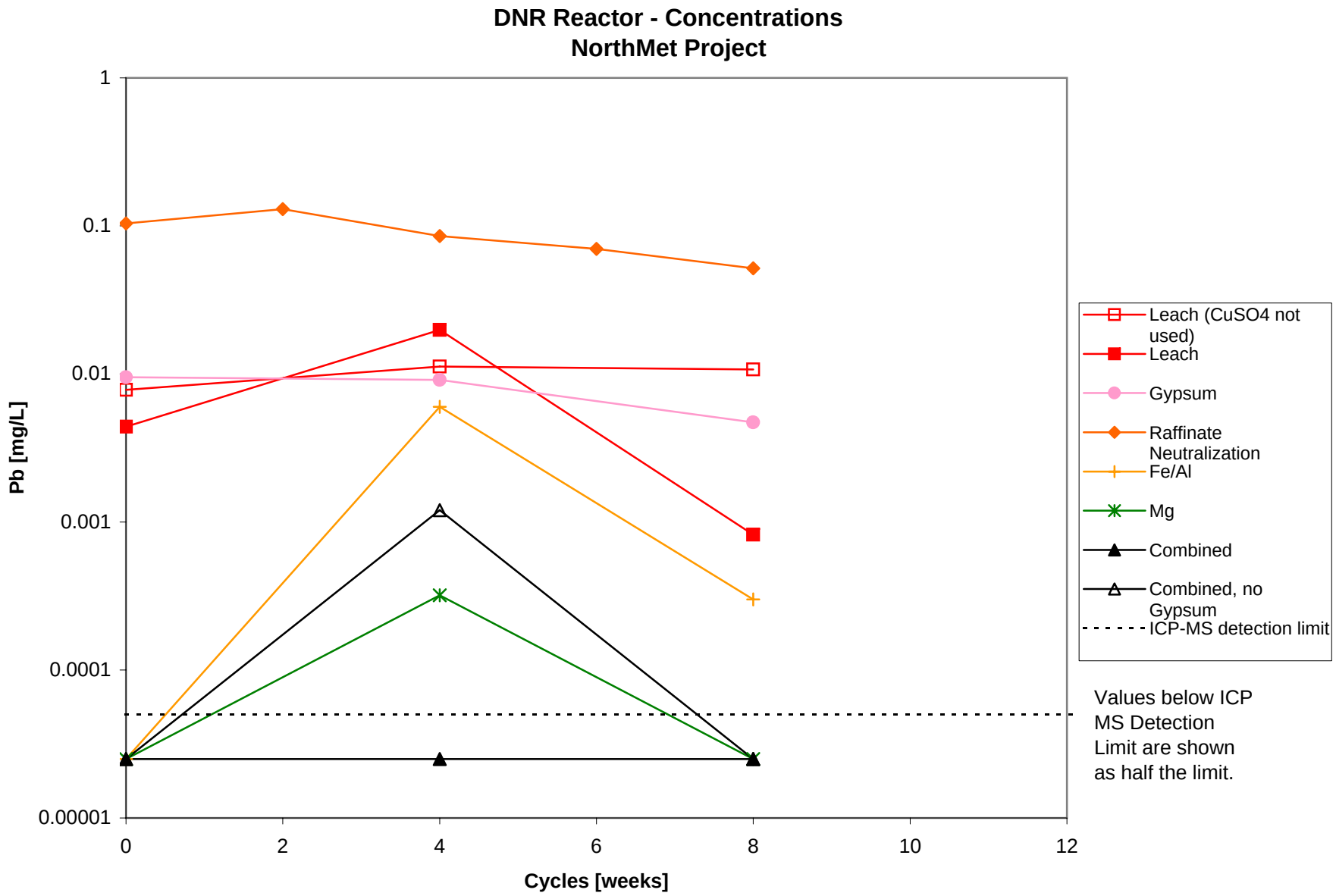

DNR Reactor - Concentrations
NorthMet Project
Ca [mg/L]
Cycles [weeks]
0
100
200
300
400
500
600
700
0
2
4
6
8
10
12
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit - 0.01 mg/L
Values below ICP MS Detection Limit are shown as half the limit.

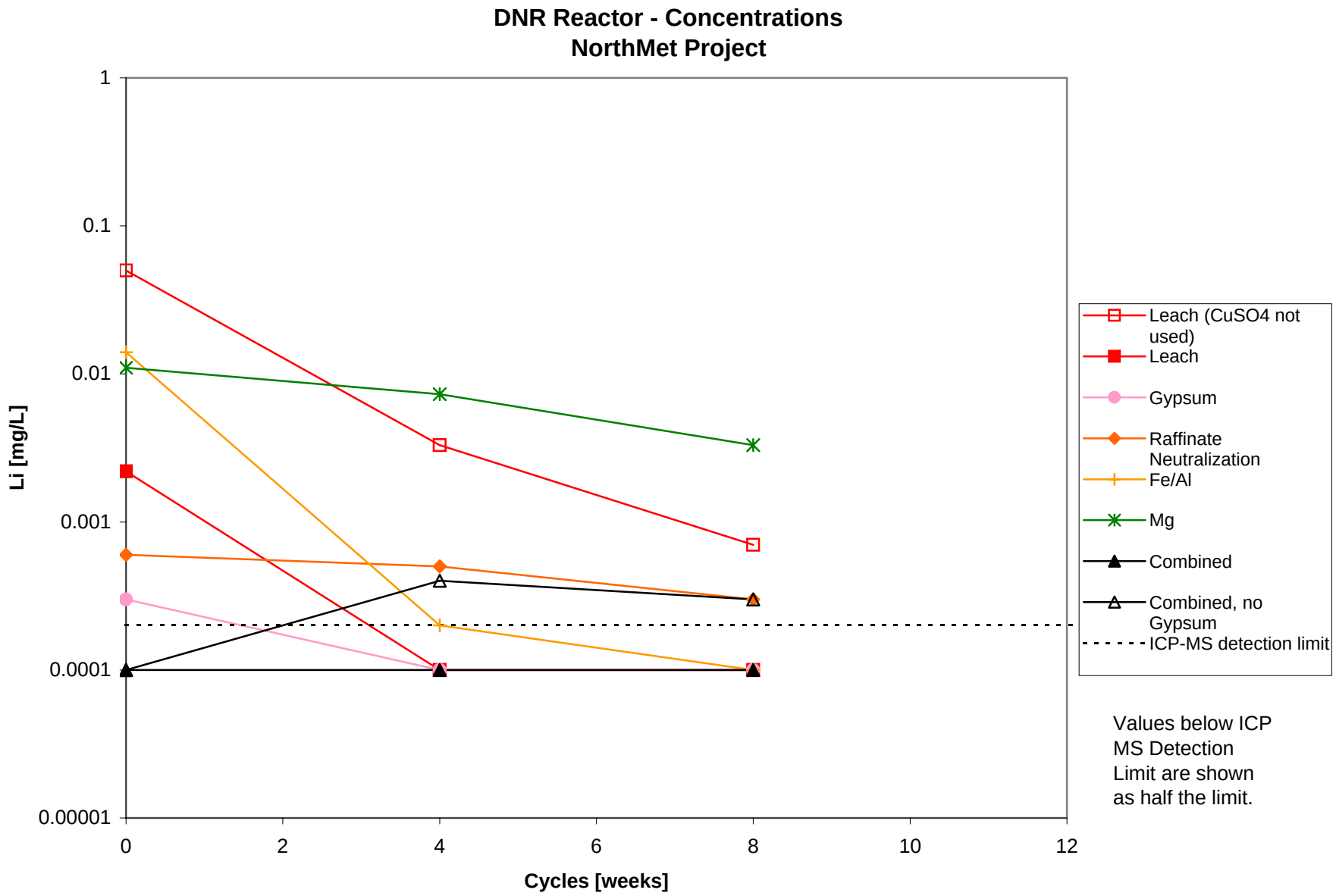

DNR Reactor - Concentrations
NorthMet Project
Cr [mg/L]
1
0.1
0.01
0.001
0.0001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

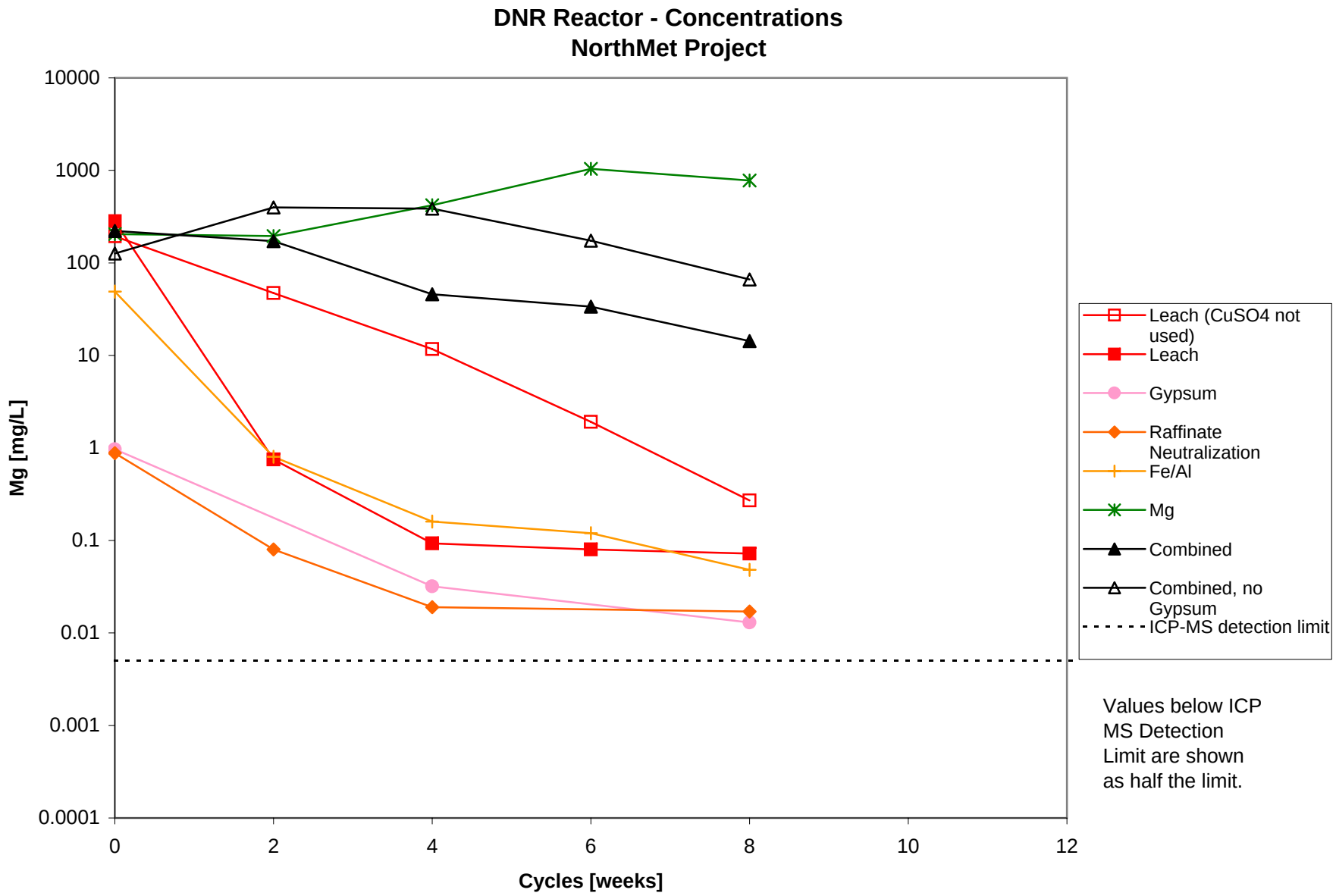

DNR Reactor - Concentrations
NorthMet Project
Co [mg/L]
10
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

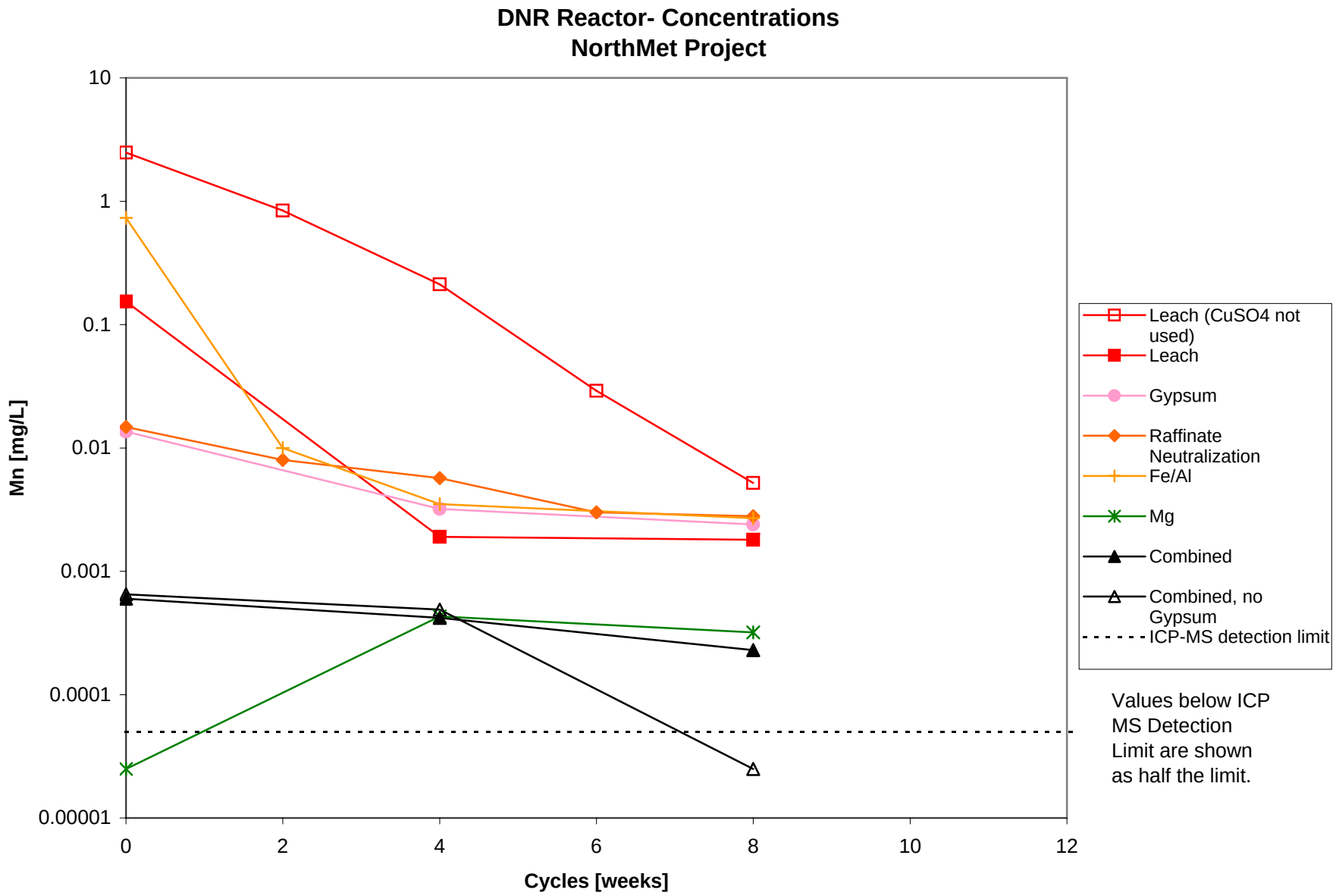

DNR Reactor - Concentrations
NorthMet Project
Cu [mg/L]
1000
100
10
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

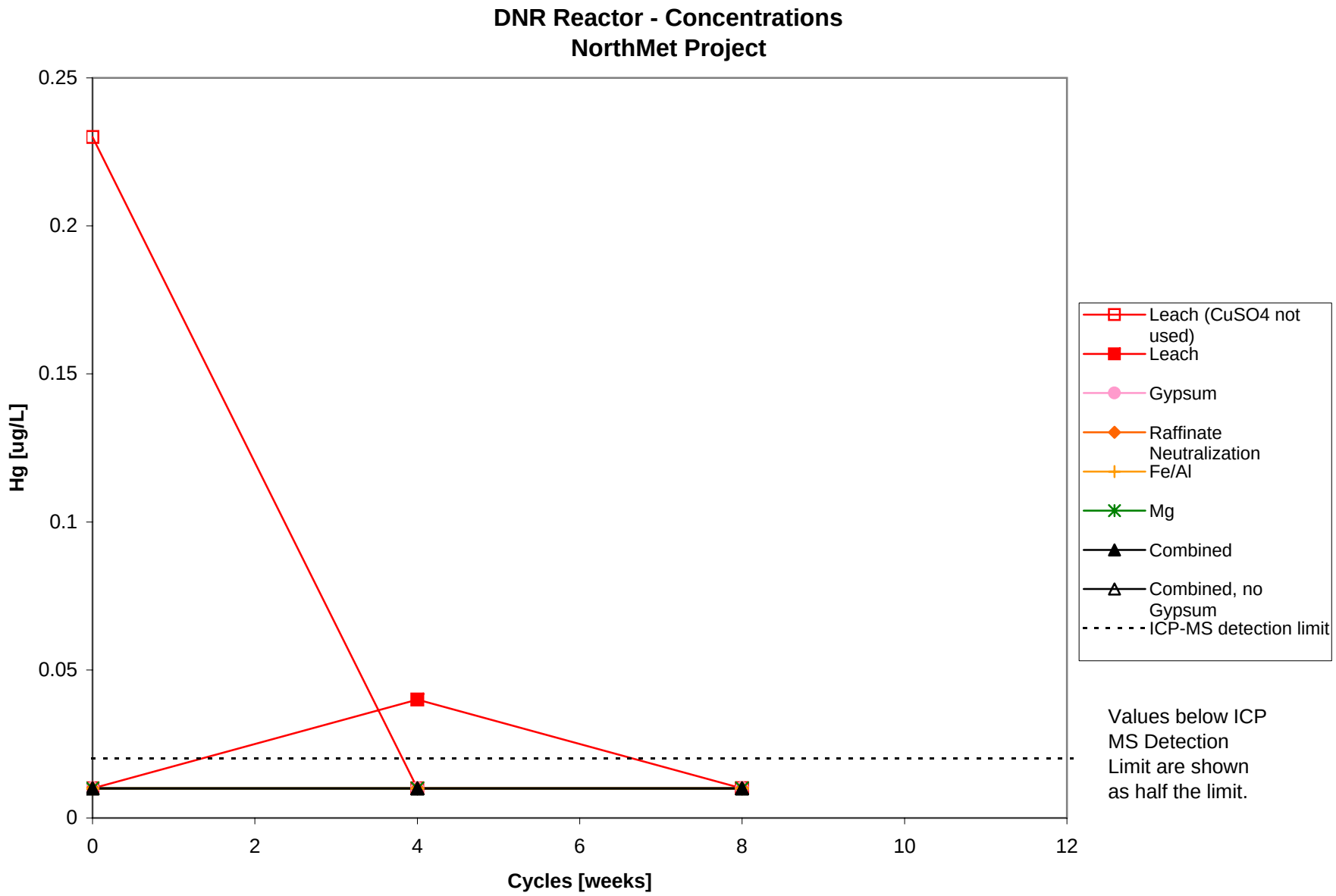

DNR Reactor - Concentrations
NorthMet Project
Fe [mg/L]
1000
100
10
1
0.1
0.01
0.001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

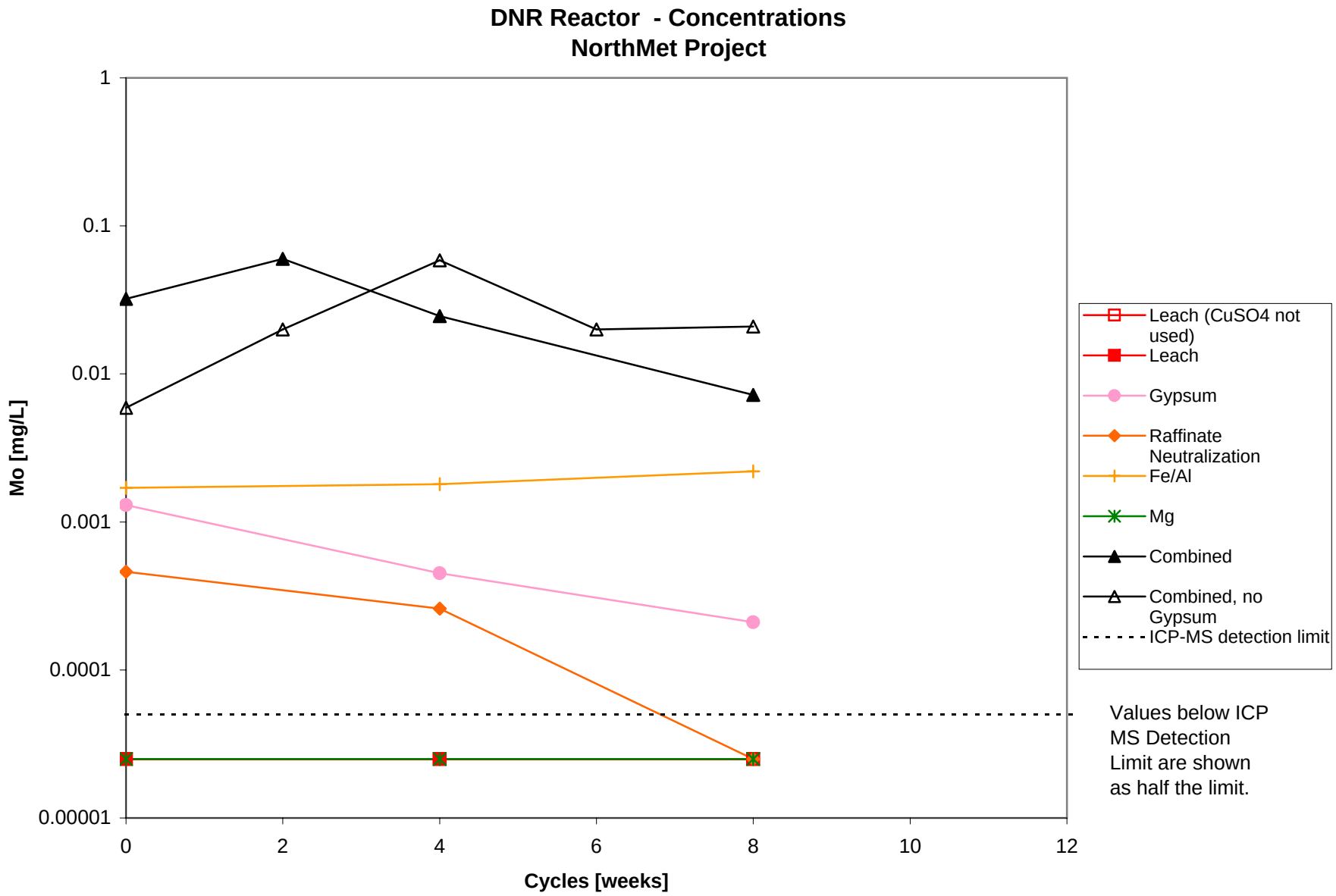

DNR Reactor - Concentrations
NorthMet Project
Pb [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

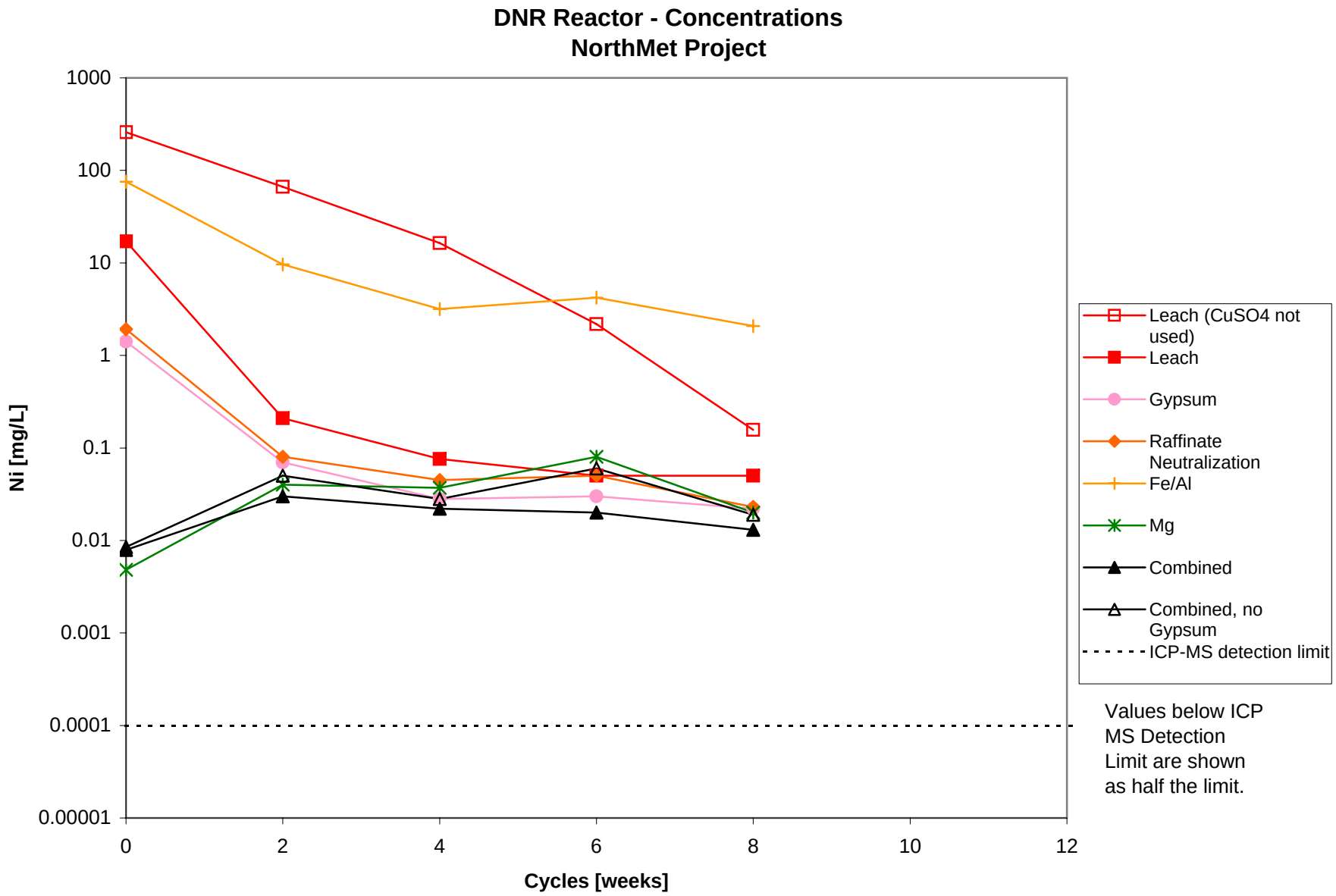

DNR Reactor - Concentrations
NorthMet Project
Li [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

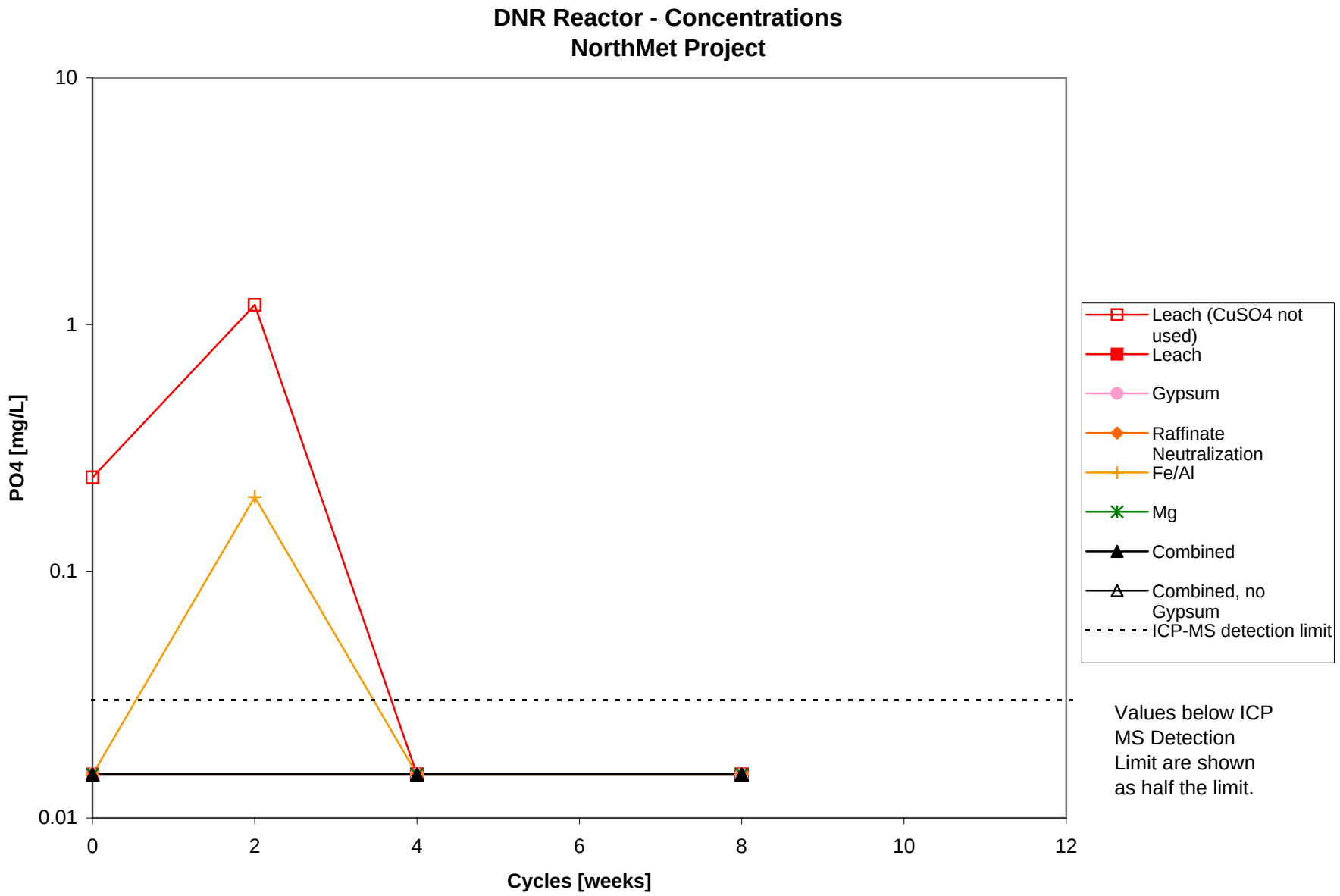

DNR Reactor - Concentrations
NorthMet Project
Mg [mg/L]
10000
1000
100
10
1
0.1
0.01
0.001
0.0001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

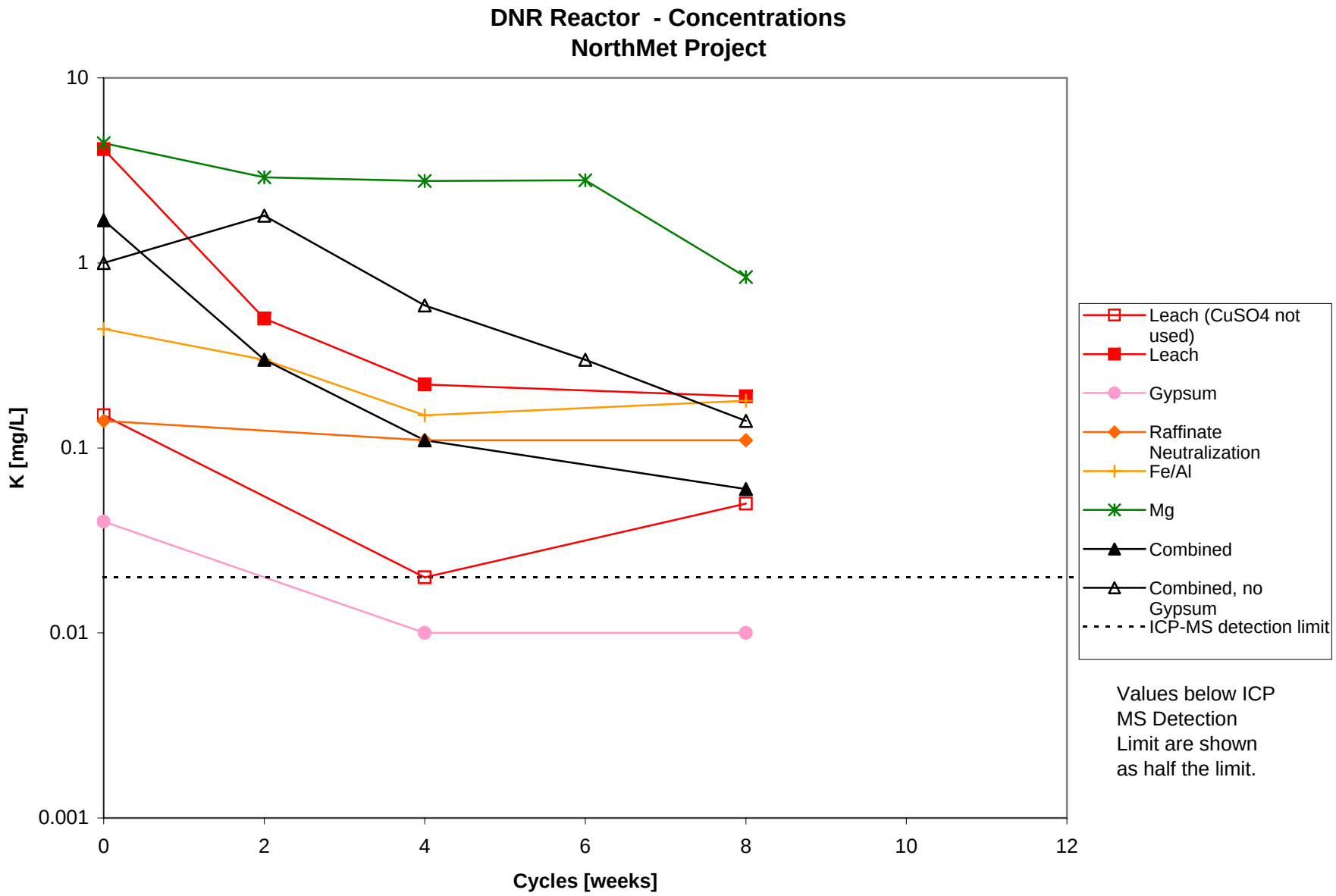

DNR Reactor- Concentrations
NorthMet Project
Mn [mg/L]
10
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

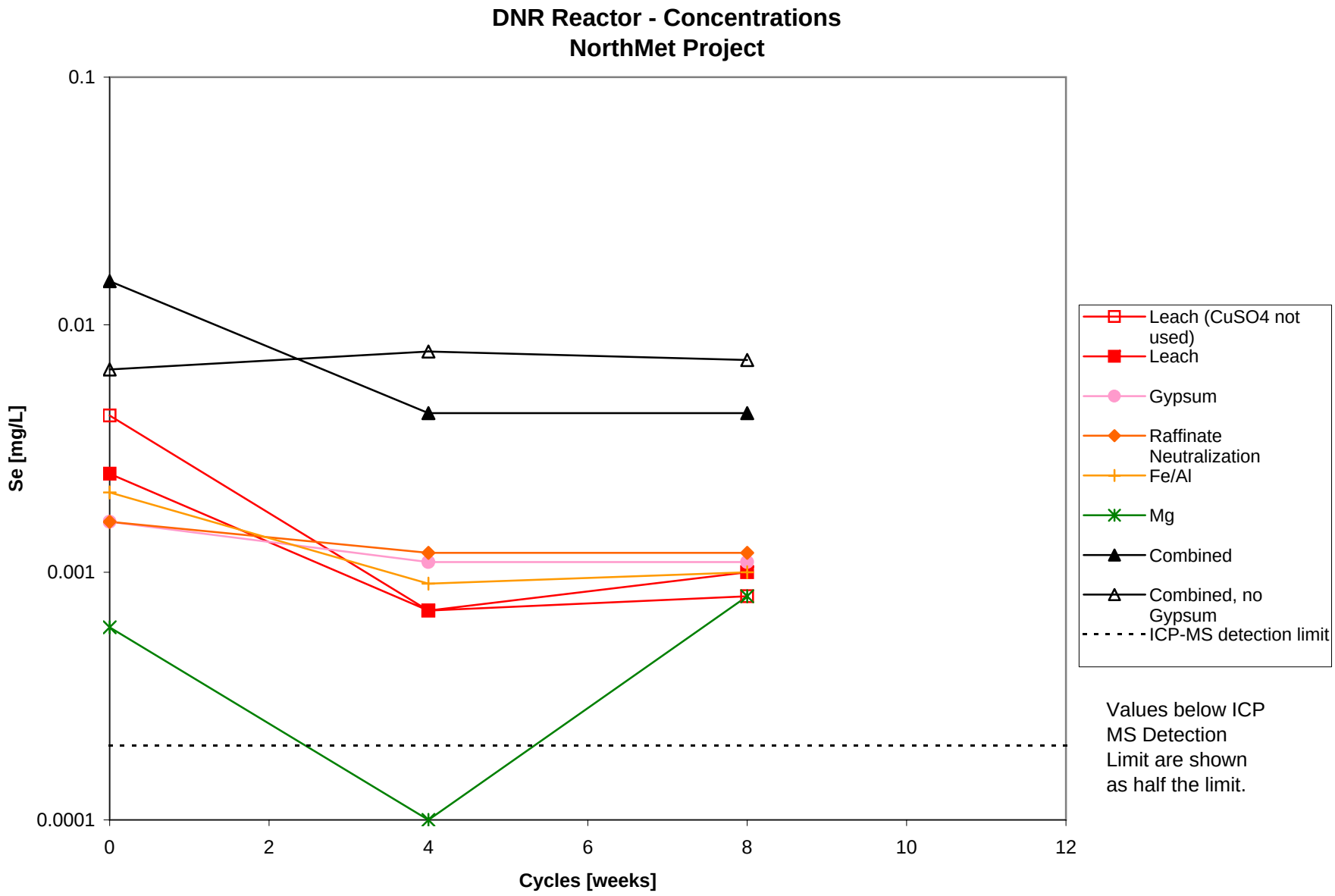

DNR Reactor - Concentrations
NorthMet Project
Hg [ug/L]
0.25
0.2
0.15
0.1
0.05
0
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

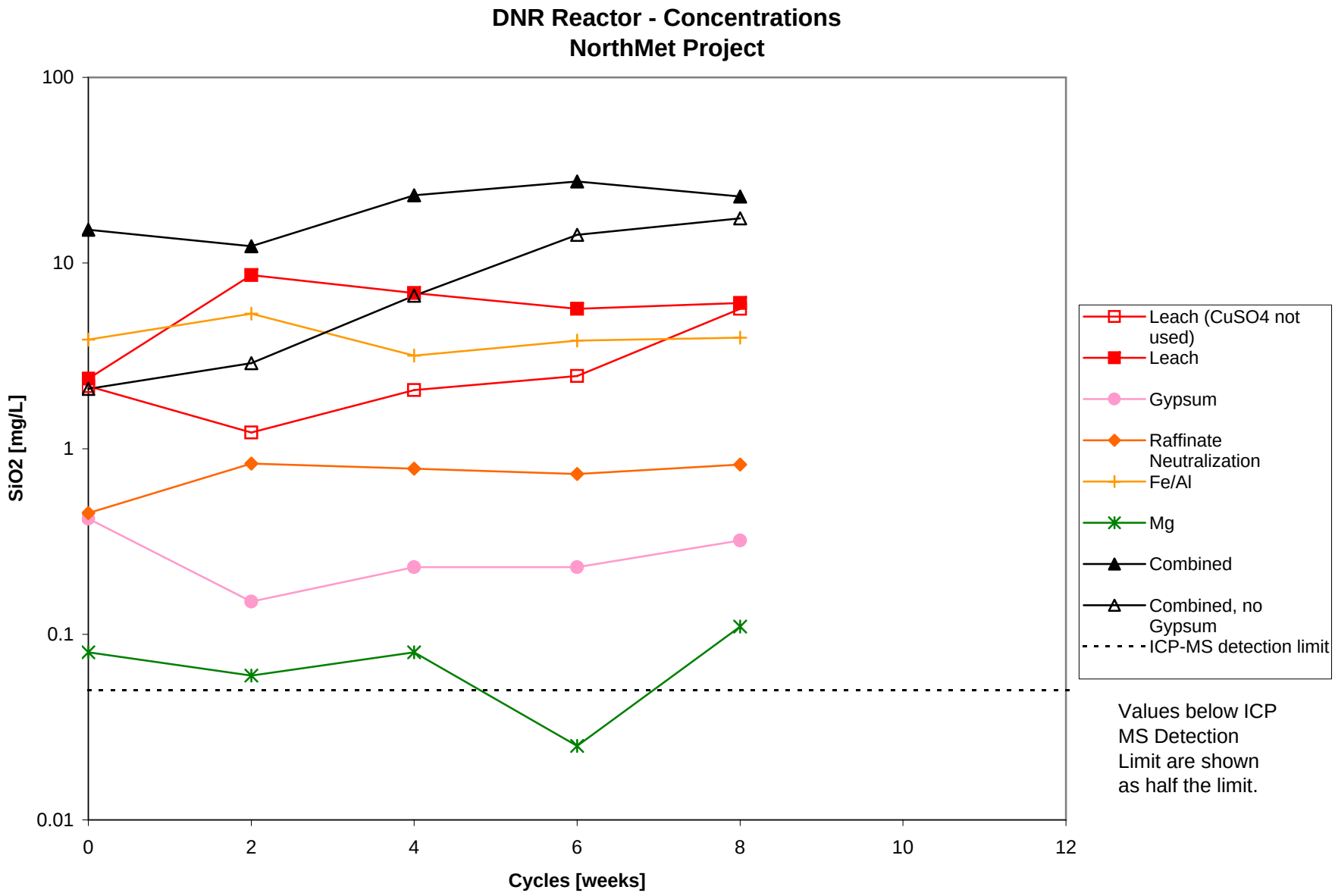

DNR Reactor - Concentrations
NorthMet Project
Mo [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

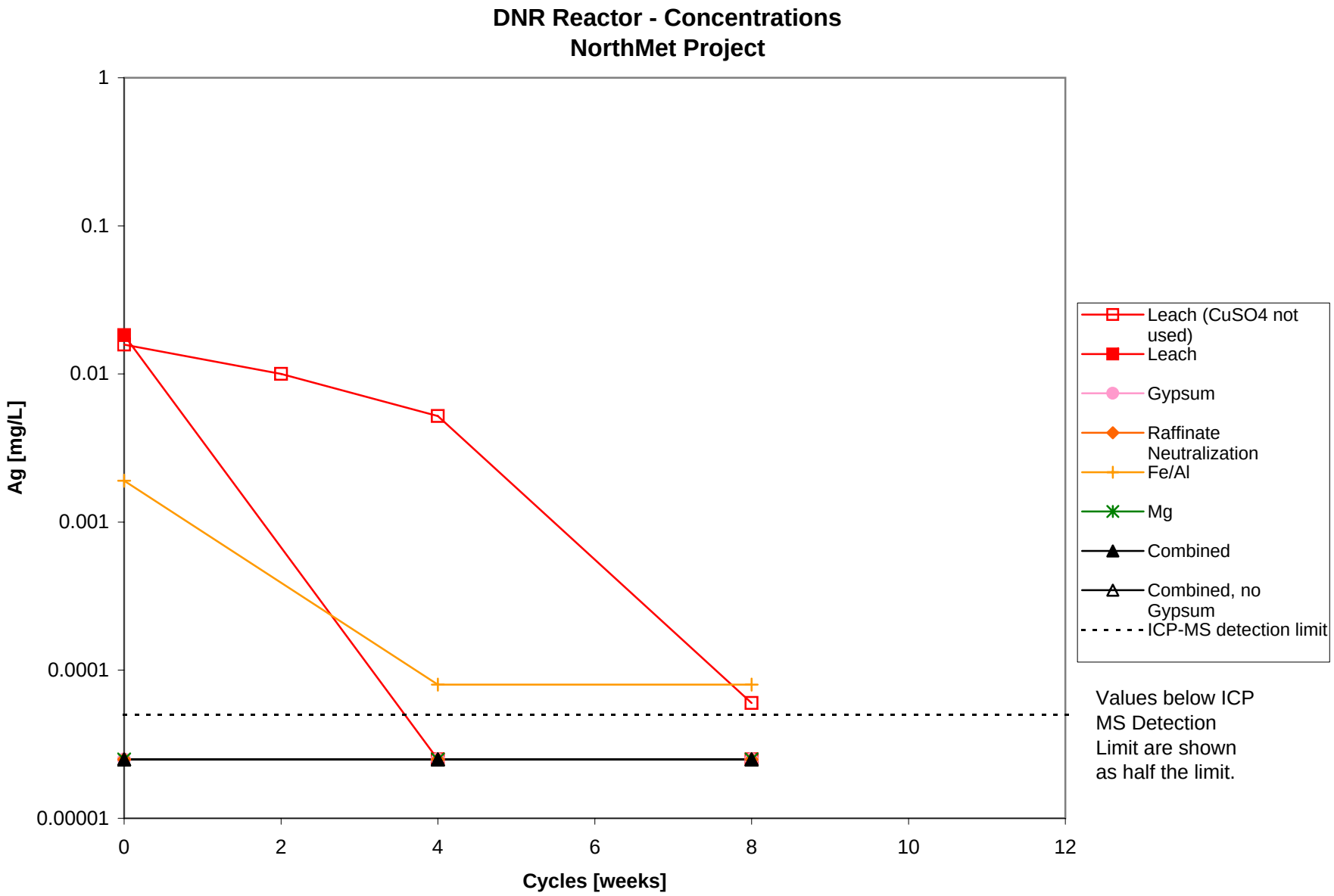

DNR Reactor - Concentrations
NorthMet Project
Ni [mg/L]
1000
100
10
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

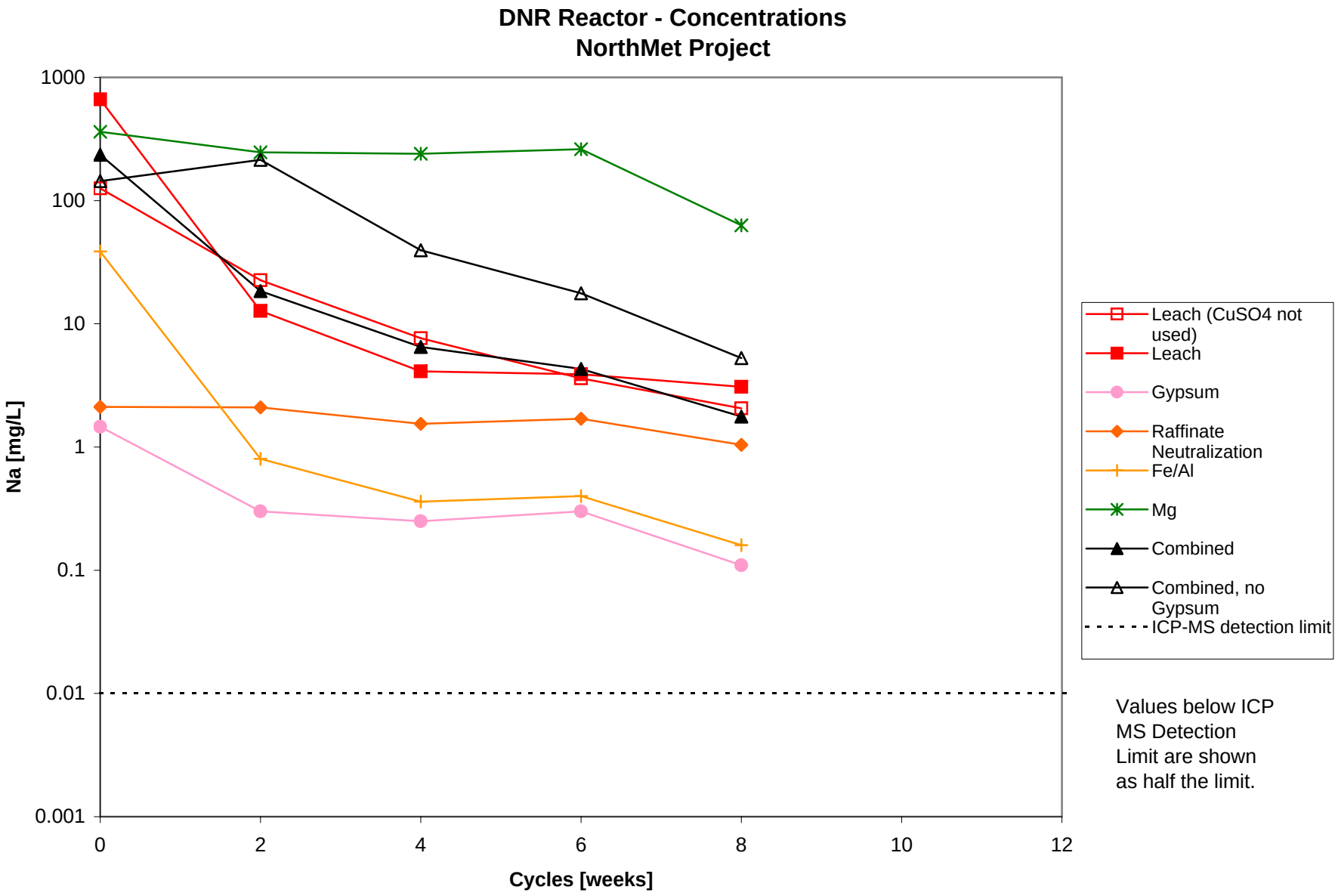

DNR Reactor - Concentrations
NorthMet Project
PO4 [mg/L]
10
1
0.1
0.01
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

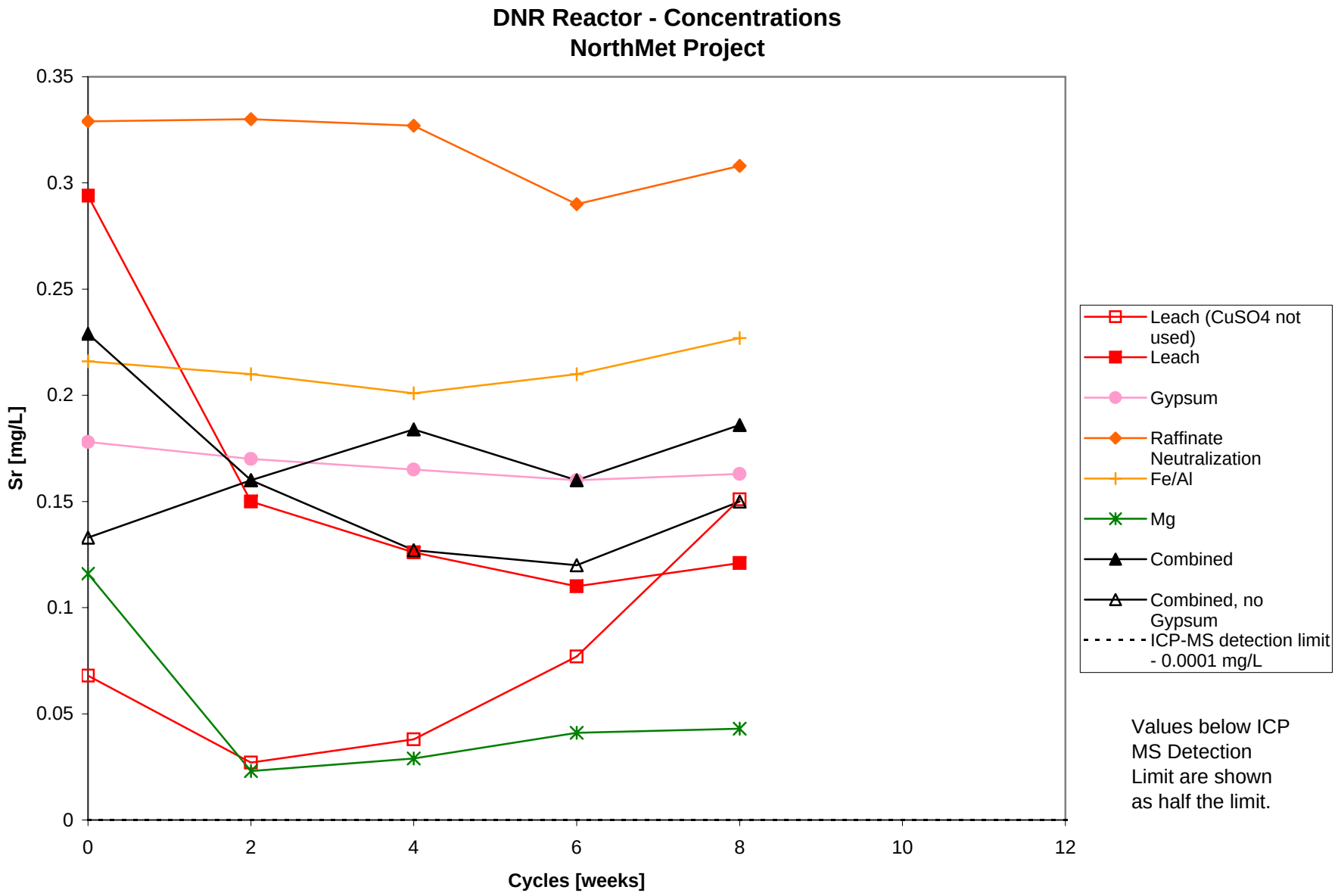

DNR Reactor - Concentrations
NorthMet Project
K [mg/L]
10
1
0.1
0.01
0.001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

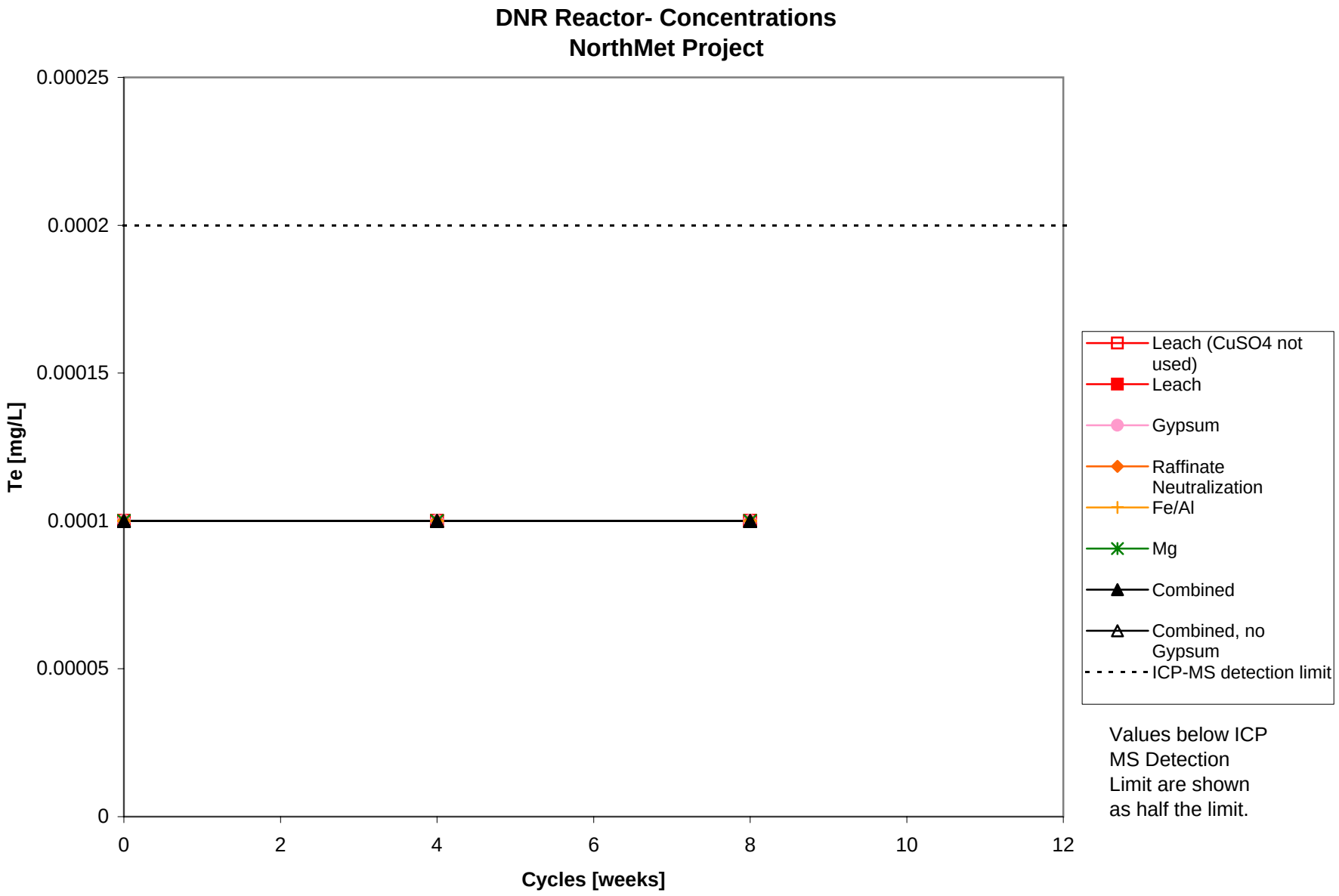

DNR Reactor - Concentrations
NorthMet Project
Se [mg/L]
0.1
0.01
0.001
0.0001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

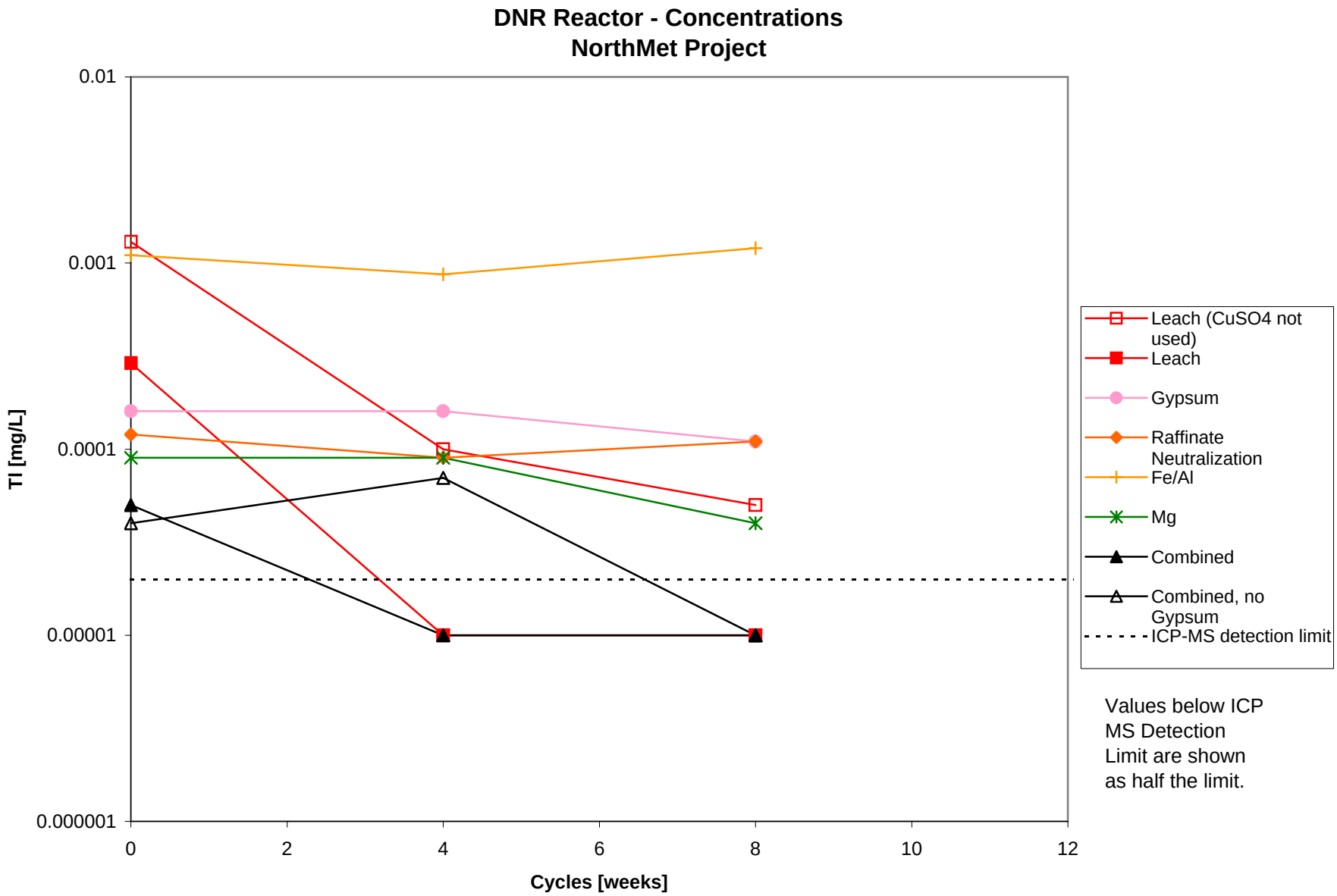

DNR Reactor - Concentrations
NorthMet Project
SiO2 [mg/L]
100
10
1
0.1
0.01
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.


DNR Reactor - Concentrations
NorthMet Project
Ag [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

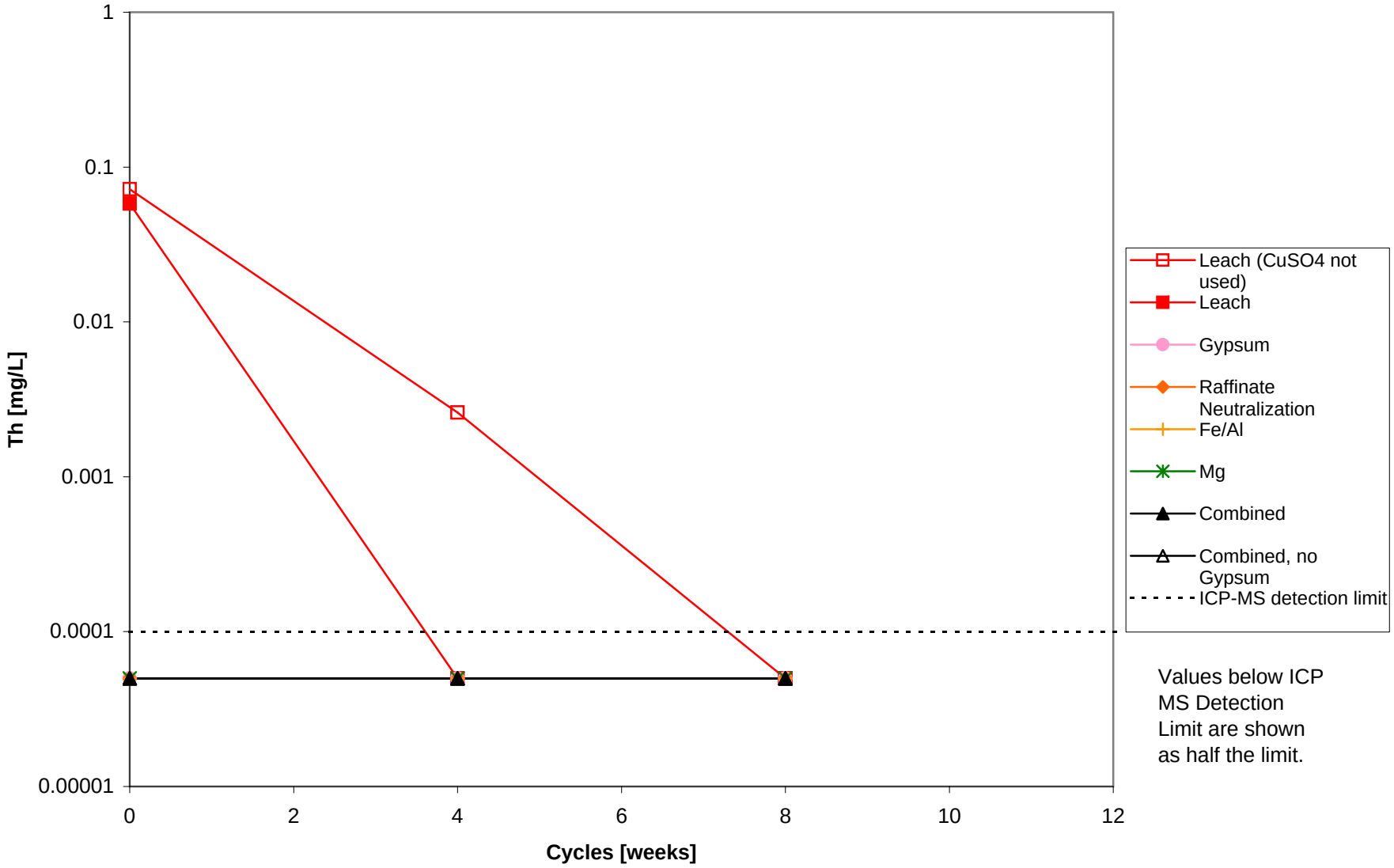

DNR Reactor - Concentrations
NorthMet Project
Na [mg/L]
1000
100
10
1
0.1
0.01
0.001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

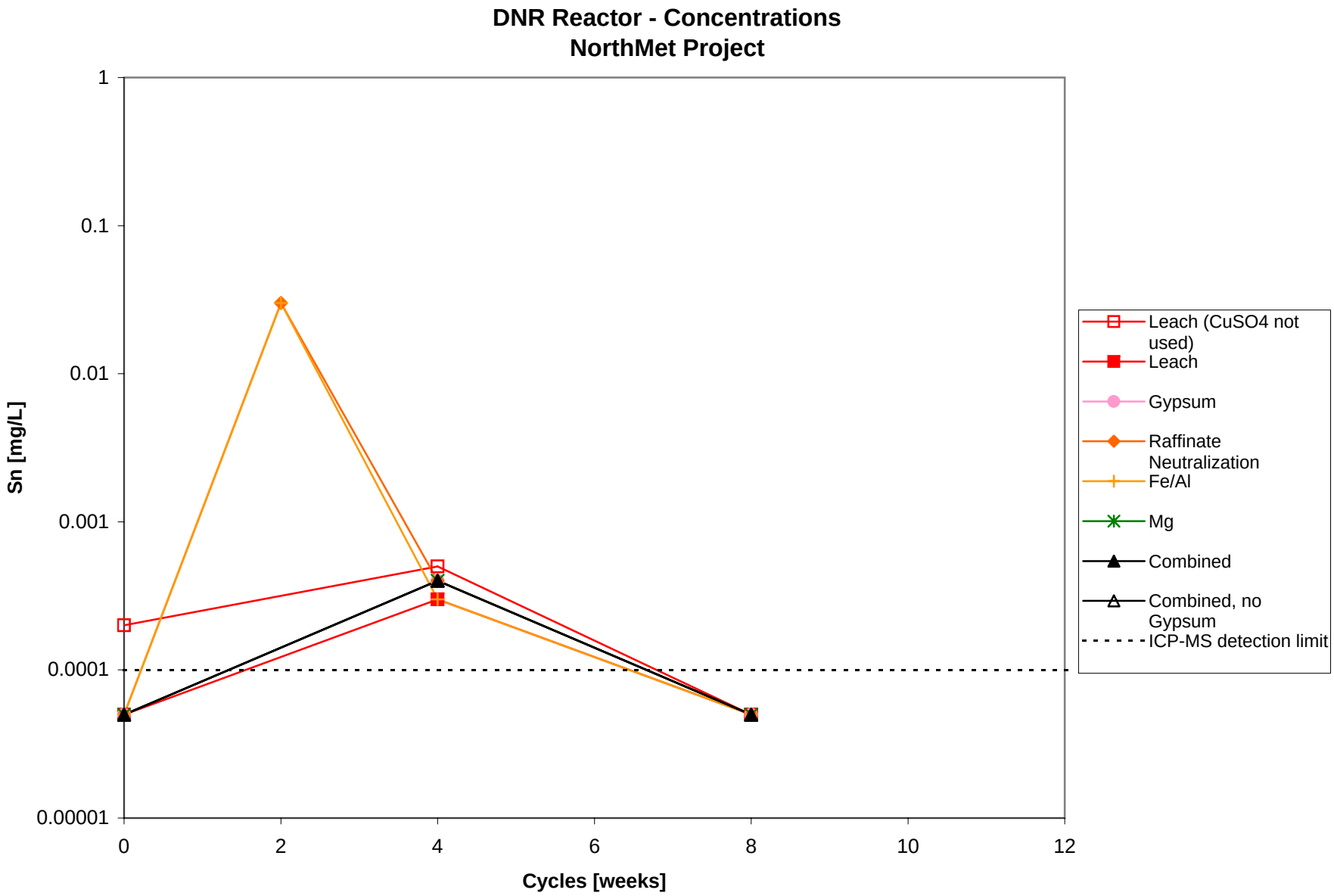

DNR Reactor - Concentrations
NorthMet Project
Sr [mg/L]
0.35
0.3
0.25
0.2
0.15
0.1
0.05
0
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit - 0.0001 mg/L
Values below ICP MS Detection Limit are shown as half the limit.

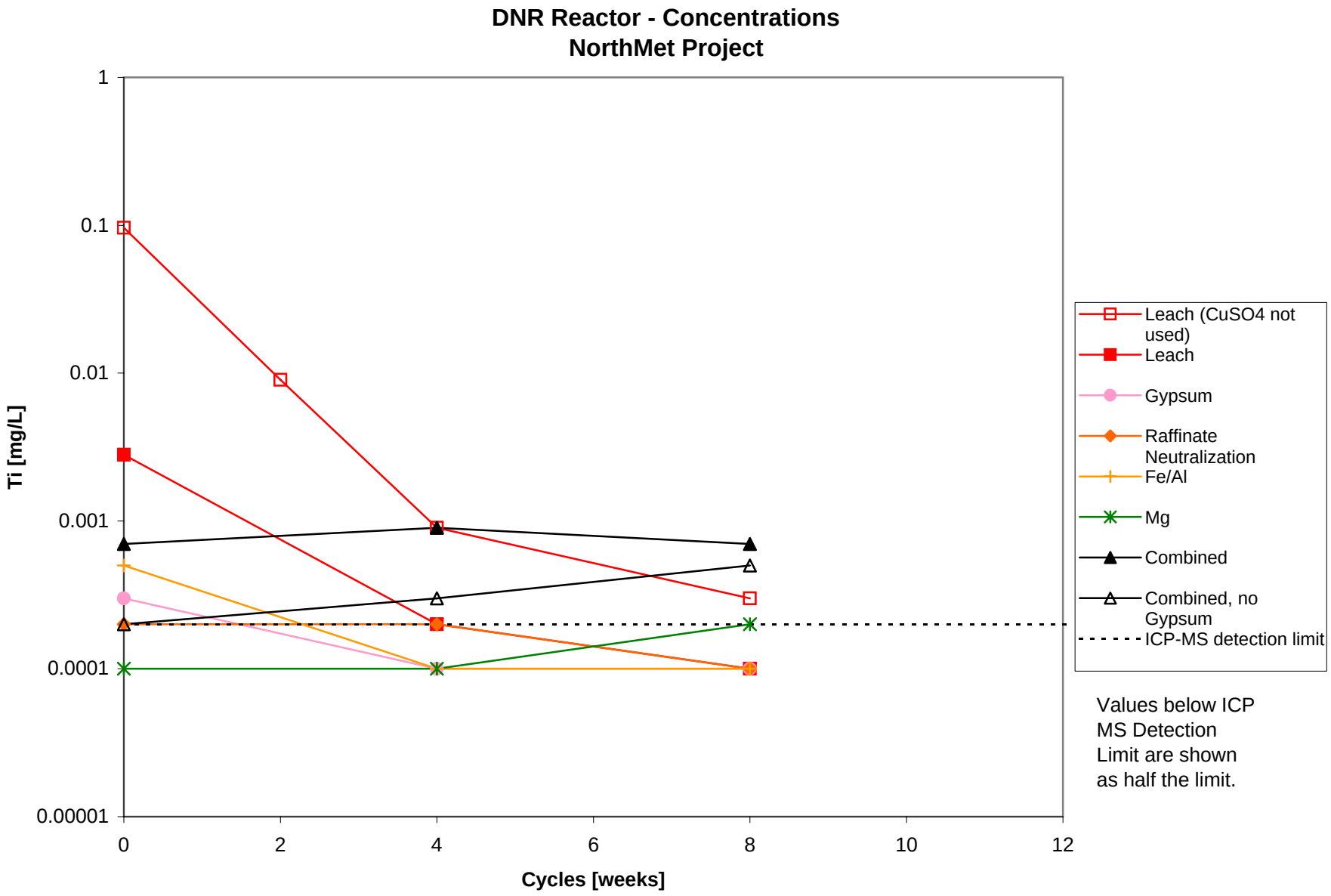

DNR Reactor- Concentrations
NorthMet Project
0.00025
0.0002
0.00015
0.0001
0.00005
0
Te [mg/L]
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

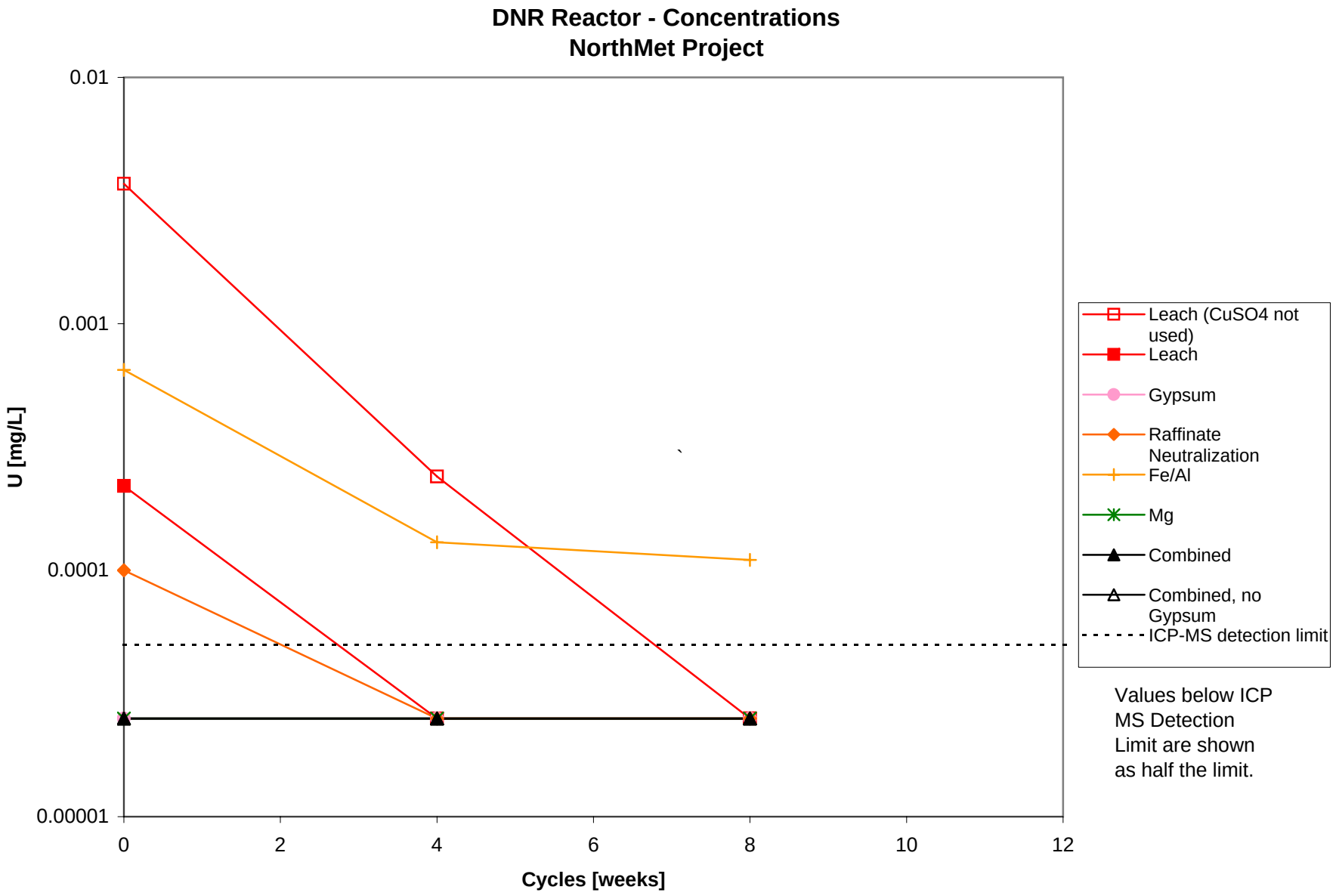

DNR Reactor - Concentrations
NorthMet Project
Tl [mg/L]
0.01
0.001
0.0001
0.00001
0.000001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

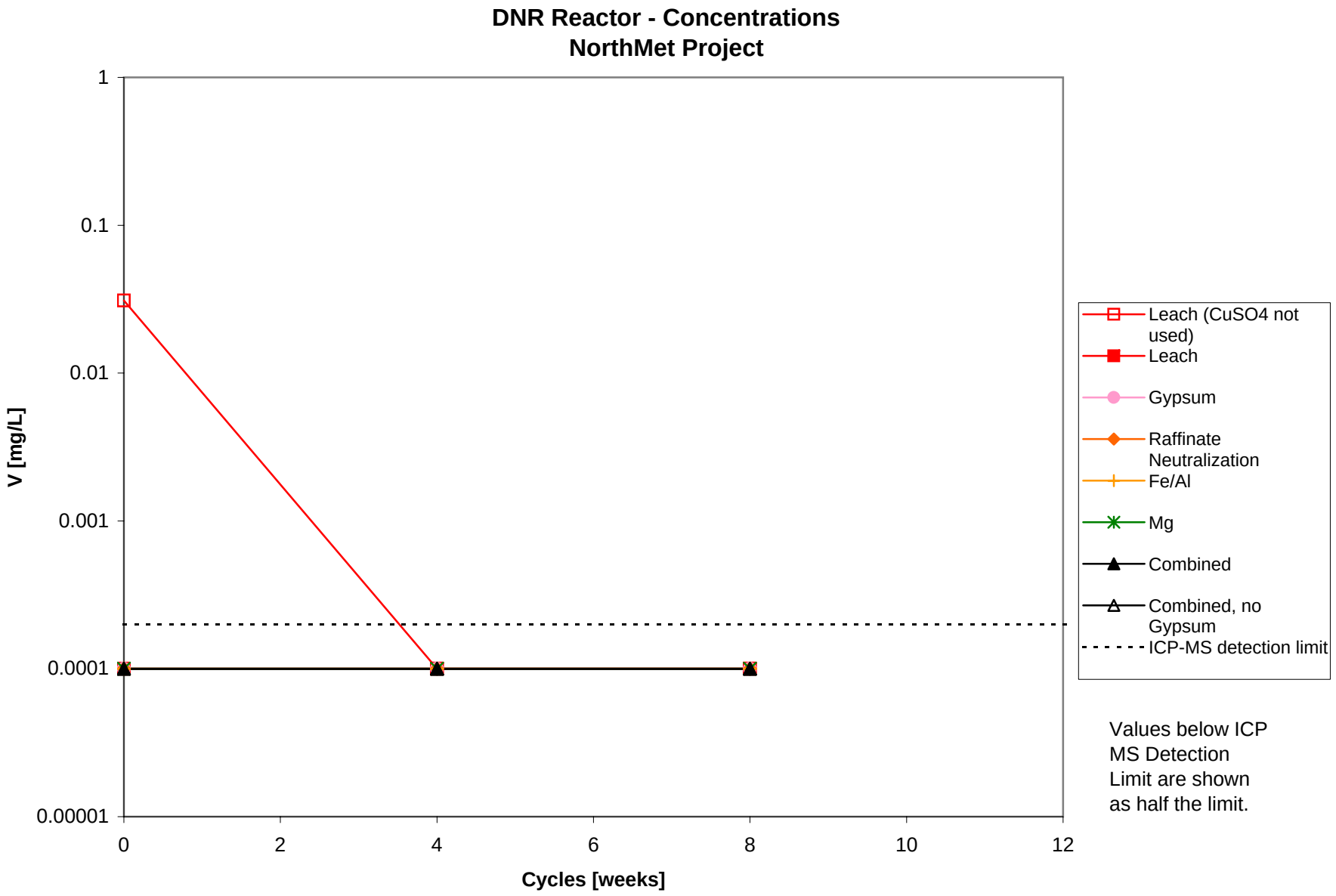
DNR Reactor - Concentrations
NorthMet Project

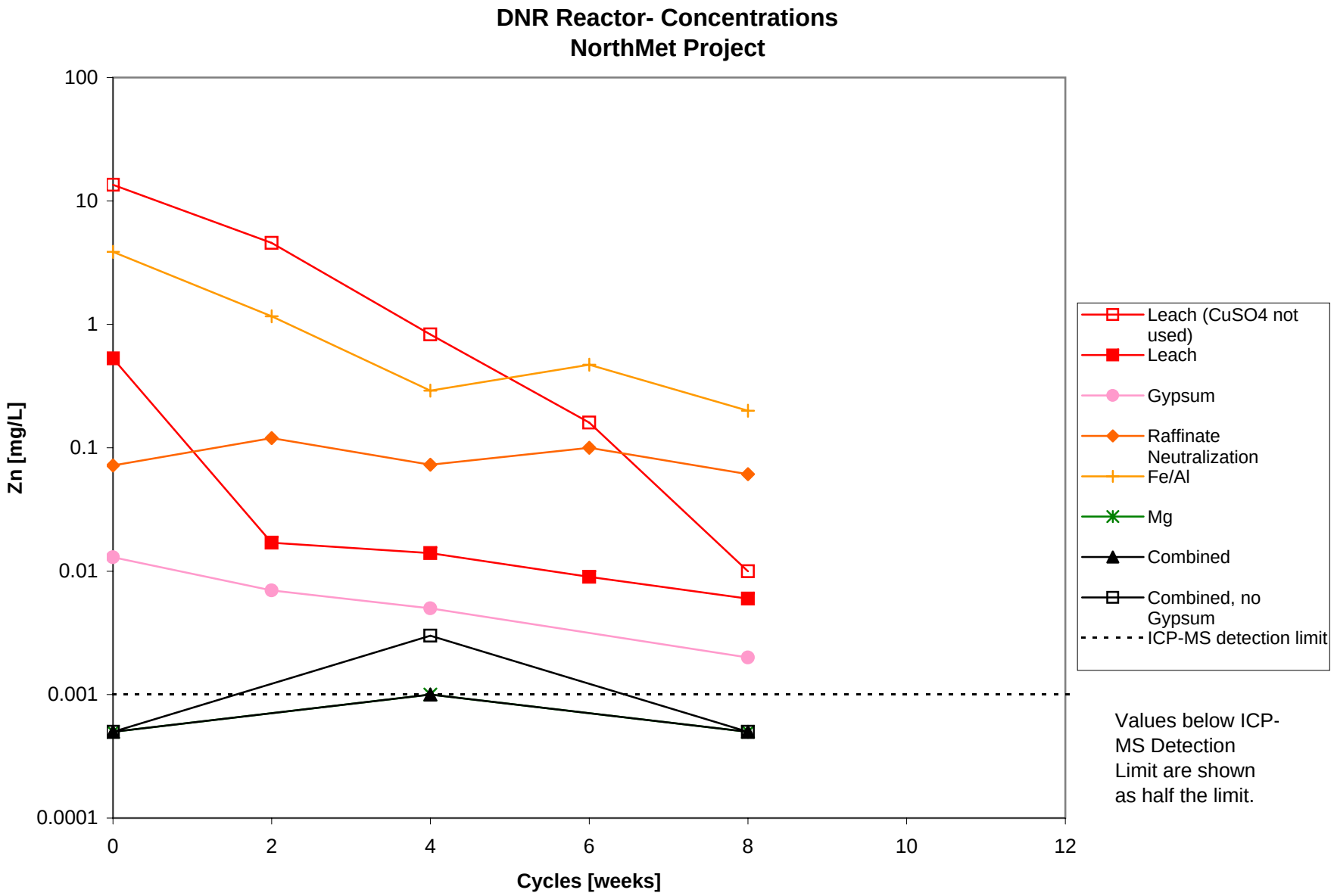



DNR Reactor - Concentrations
NorthMet Project
Sn [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit

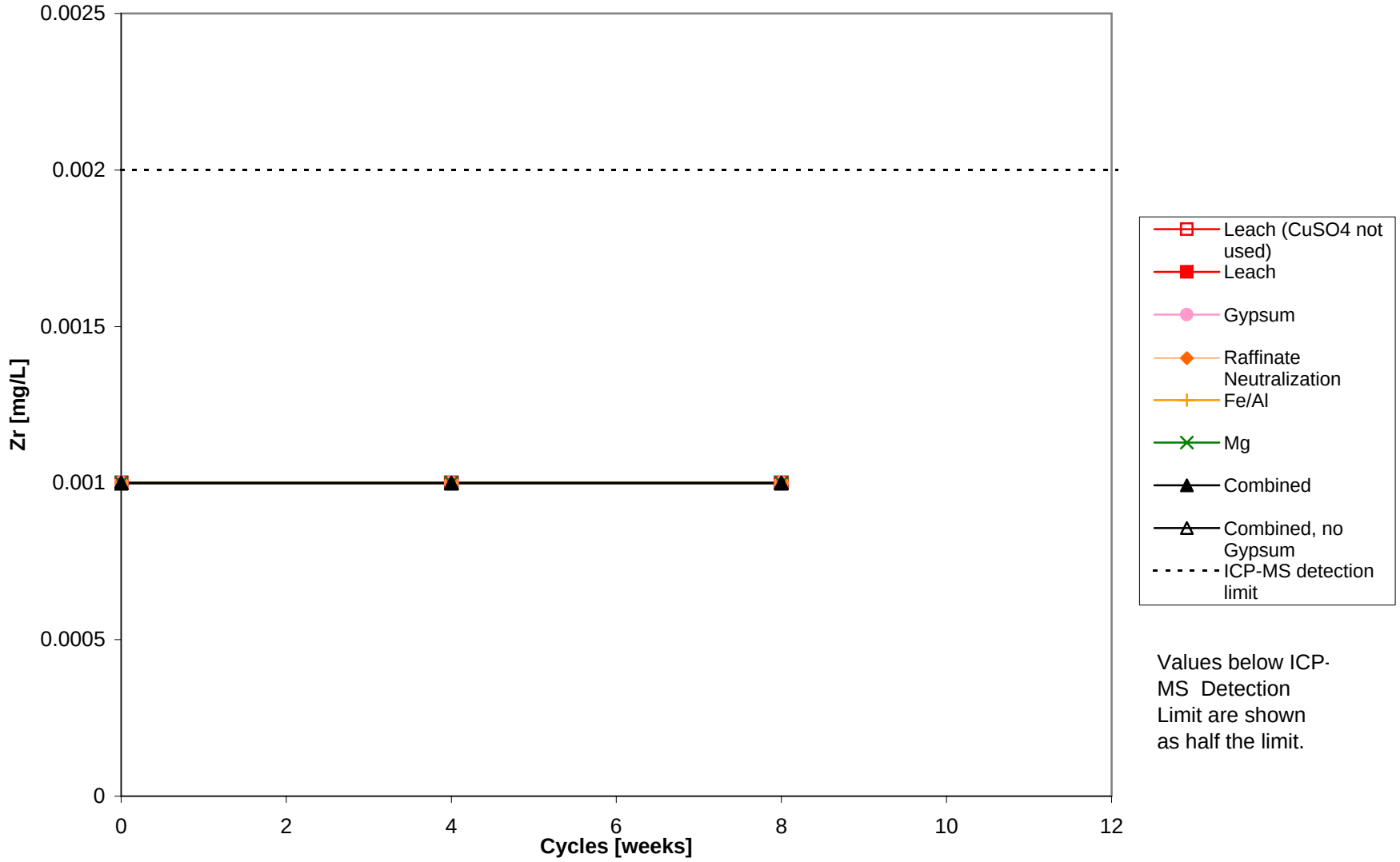

DNR Reactor - Concentrations
NorthMet Project
Ti [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.


DNR Reactor - Concentrations
NorthMet Project
U [mg/L]
0.01
0.001
0.0001
0.00001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.


DNR Reactor - Concentrations
NorthMet Project
1
0.1
0.01
0.001
0.0001
0.00001
V [mg/L]
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.


DNR Reactor- Concentrations
NorthMet Project
Zn [mg/L]
100
10
1
0.1
0.01
0.001
0.0001
0
2
4
6
8
10
12
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

DNR Reactor - Concentrations
NorthMet Project



Values below ICP-MS Detection Limit are shown as half the limit.

Attachment 2-12

SRK Consulting. RS33/RS65 - Hydrometallurgical Residue Characterization and Water Quality Model – DRAFT, NorthMet Project. February 21, 2007.

RS33/RS65

Draft-01

February 21, 2007

RS33/RS65 – Hydrometallurgical Residue Characterization and Water Quality Model – NorthMet Project - DRAFT

Report Prepared for

PolyMet Mining Corp.

Report Prepared by

SRK Consulting
Engineers and Scientists

February 2007

RS33/RS65

Draft-01

February 21, 2007

RS33/RS65 – Hydrometallurgical Residue Characterization and Water Quality Model – NorthMet Project - DRAFT

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SRK Project Number 1UP005.001

February 2007

Executive Summary

The processing and recovery of NorthMet Project ore to recover commodity metals will involve conventional flotation to produce a sulfide concentrate followed by hydrometallurgical treatment of the concentrate to recovery commodity metals. The hydrometallurgical process will produce five residues:

- Leach residue (silicates);
- Gypsum residue (calcium sulfate);
- Raffinate neutralization residue (calcium sulfate);
- Fe/Al residue (oxide); and
- Mg residue (oxide).

For final disposal, all residues will be mixed to create a single "combined residue" product which will be placed in lined disposal cells.

Samples of the residues were obtained by pilot testing and subjected to mineralogical and chemical tests to understand the composition and leaching characteristics of the residues. Except for the leach residue which is material remaining after initial leaching of the mineral concentrate, the residues are composed dominantly of calcium sulfate (mainly gypsum). The leach residue contains natrojarosite. The combined residue is also composed mainly of gypsum but natrojarosite will also be a component of this residue.

Except for the Mg residue, the individual residues were acidic to varying degrees. The combined residues were non-acidic in the time frame of the tests due to buffering minerals in the Mg residue but the presence of natrojarosite indicated that the combined residues could be acidic at some time in the future. To ensure that the combined residues do not become acidic resulting in accelerated leaching of metals held in the oxide components, additional base material (e.g. lime or limestone) will be added to offset the acid potential of the natrojarosite.

Results of the EPA1311 test shows that the residues are not classified as hazardous.

A variety of leaching tests have shown that leachate chemistry reached chemical equilibrium with the solids, and that the initial leach produced the highest concentrations of most parameters. The initial contact solution chemistry from the combined residues was used to indicate the chemistry of pore water in the hydrometallurgical residue disposal cells both during operation and at closure.

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Report CD: Leachate Chemistry Results

1 Introduction

1.1 Background

PolyMet Mining Inc. (PolyMet) is proposing to develop the NorthMet Project (Dunka Road Project of US Steel) near Babbitt, Minnesota. As a part of the Minnesota Department of Natural Resources (MDNR) "Permit to Mine" process a complete "mine waste characterization" is required (Minnesota Rules Chapter 6132.1000).

The processing and recovery of NorthMet Project ore to recover commodity metals will involve conventional flotation to produce a sulfide concentrate followed by hydrometallurgical treatment of the concentrate to recovery commodity metals.

The residues from hydrometallurgical treatment will be disposed as a single combined residue. As there are indications (but no certainty) that a market can be found for the gypsum residue, this report focuses on the disposal of combination of all residues with gypsum residue as described in the Detailed Project Description (PolyMet 2007) but with reference to testwork results for combined residue without gypsum.

All residues will be disposed in lined cells. Design concepts for the cells are provided in RS28T (Barr 2007).

1.2 Objective

The objectives of the characterization studies were to evaluate the mineralogical and chemical properties of the residues and predict the chemistry of pore waters in the residues. This information is an input into the prediction of overall water quality at the hydrometallurgical residue disposal cells.

1.3 Design and Consultation Process

The characterization plan for the residues was developed in consultation with the MDNR (SRK 2005) (Appendix A). The plan covers characterization of both tailings from the flotation process and the hydrometallurgical residues from recovery of commodity metals. Characterization of the hydrometallurgical residues was started in February 2006 following generation of the residues from pilot plant testing between August 29 and October 11, 2005 and agreement on the characterization methodologies with MDNR.

1.4 Structure of Report

This report combines results of two studies. RS33 provides results of characterization of the hydrometallurgical residues whereas RS65 is the prediction of pore water chemistry.

The structure of the RS65 report, which was a combination of RS33 and RS65 was agreed with the MDNR. The final version of the report outline was transmitted to the MDNR on April 26, 2006. The agreed outline has been followed. If any sections are redundant, the section heading is shown with a brief note to explain why the section is no longer relevant.

2 Water Chemistry Prediction Methods

2.1 Theoretical Method

The theoretical method considers the rate at which components of the residues might dissolve and their final solubility. This approach is potentially applicable to the hydrometallurgical residues because they are composed primarily of discrete fully oxidized mineral phases. Solution chemistry in the pores is expected to be controlled mainly by dissolution of these phases possibly with formation of new minerals. Oxidation reactions are not expected to be significant because the residues are the product of a strongly oxidizing process.

Numerous computer modeling programs are available to predict solubility of minerals including MINTEQ (e.g. Allison et al 1991) and PHREEQC (Parkhurst and Appelo 1999). Other programs couple solubility, reaction paths and reaction rates (e.g. React Bethke 2005). The main limitations of these programs are thermodynamic databases and the rate expressions for mineral reactivity. For common minerals, solubility constants are well-established but for trace elements, the solubility of co-precipitated phases are site-specific and not provided in the database, which limits the value of the theoretical method as a standalone approach. However, it can be combined with the empirical approach, as described below.

2.2 Analog and Empirical Methods

2.2.1 Analog

The analog method involves direct prediction of the dissolution of residues by comparison with similar hydrometallurgical processes involving pressure oxidation of copper sulfide concentrates. Similar processes include the CESL Copper Process (TeckCominco, Undated) which involves moderate pressure oxidation of copper concentrates using chloride as a catalyst and production of residues containing leach residue, iron oxide, elemental sulfur and gypsum, and Outokumpu's HydroCopper^{TM} process (Ootokumpu 2006) which also produces iron oxide and elemental sulfur but no gypsum. These are comparable to the NorthMet process, which will result in production of jarosite as a product of sulfide oxidation and gypsum due to acid neutralization. The differences in the processes obviously limit the direct comparison of the performance of residues.

2.2.2 Empirical

The empirical method involves the use of testwork to simulate the leaching behavior of residues. The limitation of this approach is usually that the ratio of leachate to solid in testwork is often much higher than under field conditions due to the practical need to produce enough water for analysis. The results can be scaled up to field conditions by assuming that the concentration will increase in proportion to the decrease in liquid to solid ratio. Such scale-up calculations can produce very high concentrations that are not realistic.

2.3 Method Selected

The selected method is primarily empirical (i.e. based on testwork) but with evaluation of data using thermodynamic considerations, if practical, to place constraints on concentrations indicated by scale-up of laboratory results. The steps used to generate water quality predictions are therefore:

- Evaluation of overall leachate chemistry using the SpecE8 module of the Geochemist's Workbench (Bethke 2005) to determine if major elements are near chemical saturation.
- Comparison of trace metal concentrations with pH to determine if concentrations are constrained by pH.
- Scale-up of metal concentrations to reflect under-saturation indicated by SpecE8 and the pH relationships.
- Development of final predictions for pore water chemistry.

3 Program Design

3.1 Process Background

Hydrometallurgical processing will result in production of the following residues composed dominantly of the indicated minerals or mineral groups:

- Leach residue (silicates);
- Gypsum residue (calcium sulfate);
- Raffinate neutralization residue (calcium sulfate);
- Fe/Al (Iron/Aluminum) residue (oxide); and
- Mg residue (oxide).

The process flow sheets are provided in Appendix A. The pilot plant run that generated the residues for testing had a separate Fe/Al Removal Stage. Subsequent refinement of the flowsheet has combined this stage with Raffinate Neutralization which means that the raffinate neutralization residue and the Fe/Al residue will be combined in the full scale plant and are generally referred to singularly as raffinate neutralization residue.

3.2 Geochemical Background

The residues are mainly chemical products in which the original concentrate components are oxidized to sulfates and hydroxides, and dissolved.

- Sulfur from the sulfide minerals is oxidized to jarosite and soluble sulfate. The latter is precipitated as calcium sulfate through the neutralization process;
- Iron released from sulfide and silicate minerals is precipitated as hydroxide; and
- Magnesium and aluminum released from silicate minerals are precipitated as hydroxides.

Copper is recovered in the process by electro-winning to produce copper anodes. Nickel, cobalt and zinc are recovered as a mixed hydroxide product. Platinum group metals and gold are recovered as concentrate product.

Incomplete dissolution of the silicate and sulfide minerals results in the generation of a fourth residue; a mineralogical leach residue.

Because these products represent near-complete dissolution, oxidation and neutralization of the concentrate, the chemistry of contact waters is expected to be controlled mainly by the simple

dissolution of the compounds. Oxidation of residual sulfide minerals may occur from the leach residue but this effect is expected to be minor.

Overall, the process of dissolution of neutralization products is unrelated to the composition of the ore and is not expected to vary in the long term except by depletion.

3.3 Data Requirements

The primary requirements for the test program were therefore:

- Mineralogical characterization of residues to provide a basis for understanding the dissolution of the residues; and
- Dissolution tests to investigate the solubility of the minerals.

3.4 Overall Program Design

The overall program design included the following components:

- Production of individual and combined residues using a pilot plant (see RS32 Part III, Barr 2006).
- Mineralogical characterization of residues.
- Determination of solids elemental characteristics.
- Performance of various types of leaching experiments at different solid to liquid ratios to determine the dissolution behavior of the residues.

4 Sampling and Analytical Methods

4.1 Metallurgical Program

4.1.1 Ore Composite Preparation

Preparation of the ore composite is described in ER03 (PolyMet, in preparation).

4.1.2 Generation of Residue Samples

Sulfide concentrates were prepared by a pilot flotation plant in which the copper sulfate was evaluated as a reagent to improve flotation of sulfide minerals. Two concentrates were prepared in the pilot plant. One concentrate was prepared with the use of copper sulfate, and a second was prepared without the use of copper sulfate. Both residues were subsequently leached in the hydrometallurgical process. PolyMet has since decided that copper sulfate will be used. Therefore, it was agreed (in consultation with MDNR, SRK 2005) that characterization testwork would be performed on the residues generated by leaching of the concentrate that was produced with copper sulfate. The exception is the leach residue for which both residue samples are being tested.

Residues generated during the pilot plant run at SGS Lakefield were collected under supervision of Barr Engineering and shipped to Canadian Environmental and Metallurgical Inc (CEMI) in Burnaby, British Columbia, Canada for testing. SRK Consulting is responsible for supervision of the testwork described in this report.

Testing is proceeding on individual samples of each type of residue and combined residues. Residue combinations are being tested with and without the gypsum residue to allow for the option of recovery of a separate gypsum product. The proportions by wet weight of each residue in the combined samples are provided in Table 4-1. The combined residue containing all residues was received directly from the pilot plant (SGS Lakefield). The combined residue without gypsum was prepared from the component residues using the same weight proportions less the gypsum amount.

Table 4-1: Proportions (by Wet Weight) for Combined Residues

	Actual Pilot Plant Observation		Projected for Commercial Scale
Residue	**All**	**Without Gypsum**	**All**
Leach Residue	27%	45%	41%
Gypsum Residue	40%	0%	33%
Raffinate Residue	18%	31%	20%
Fe/Al Residue	6%	10%	NA, combined w/Raffinate
Mg Residue	8%	14%	6%

Subsequent optimization and modeling (MetSim Version U3) of the hydrometallurgical process by Bateman indicated that the proportion of the residues will be somewhat different under full scale production (see Table 4-1). Specifically, the pilot plant run that generated the residues for testing had a separate Fe/Al Removal Stage. The commercial scale flowsheet has combined this stage with Raffinate Neutralization which means that the raffinate neutralization residue and the Fe/Al residue will be combined in the full scale plant. This difference will affect the overall mineralogical make-up of the residues but will not affect overall leaching performance. Further discussion is provided in Section 6.4.

4.2 Dissolution Testwork

4.2.1 Mineralogy

Mineralogical characterization included:

- Optical Analysis on feed, product and mineral wastes (Leach Residue). Other residues were not examined optically because they are precipitates; and
- X-Ray Diffraction on all samples to determine crystalline compounds.

Residues were not examined sub-optically due to the nature of the materials.

4.2.2 Analytical Methods

Solids Characterization

A split of each sample was submitted for:

- Sulfur forms (total S, S as sulfate).
- Paste pH.
- Neutralization potential and carbonate.
- 50 elements (mostly metals by ICP scan following aqua regia (nitric and hydrochloric acids) digestion).
- Whole rock oxides. This provides total concentrations of major elements.

Leachate Extraction Tests

All samples were submitted for regulatory leach tests (EPA 1311; EPA 1312) to provide data for waste classification purposes.

A third leach procedure was used as the first step of the sequential shake flask leach procedure described below. This procedure, developed by Price (1997) for the British Columbia (Canada) Ministry of Energy Mines and Petroleum Resources, uses a lower leach ratio (1:3) to improve

detection of low levels of metals. The lixiviant was deionized water, which typically has a pH between 5 and 6. The leachate pH was not fixed as in the TCLP and SPLP methods. The extraction was performed by shaking in a glass or plastic container for 24 hours, after which the leachate was extracted and analyzed.

Extraction test results are provided in Appendix B.

Sequential Shake Flask

All samples were tested using a sequential leach procedure consisting of weekly repetition of the leach procedure developed by Price (1997) in which the solid to liquid ratio is 1:3. The procedure involved weekly leaching of roughly 300 g of solids in a plastic bottle. The leaching step consisted of addition of deionized water and agitation for 24 hours. The leachate was then decanted for analysis.

This procedure was requested by the MDNR to provide a more aggressive evaluation of residue dissolution than was perceived to occur in humidity cells.

Leachates were analyzed using the following schedule:

- Weekly – pH, oxidation-reduction potential, conductivity.
- Bi-weekly – Acidity, alkalinity, inorganic C, hardness, anions (F, Cl, SO4).
- Four weekly (weeks 0, 4, 8, 12, etc) – Low level element scan using ICP-MS.
- Four weekly (weeks 2, 6, 10, 14, etc) – General element scan using ICP-OES.

Humidity Cell

Samples were tested in ASTM-style humidity cells in the tailings configuration. Details of the procedure are provided in Appendix A. The solid to liquid ratio is 1:0.5 (1 kg of solids leached with 500 mL of deionized water). Leachates were analyzed using the same schedule as the sequential shake flasks.

MDNR Reactor

Samples were also tested in small (75 g) MDNR Reactors for which the solid to liquid ratio is 1:2.7 (75 g of solids leached with 200 mL of deionized water). Details are provided in Appendix A. Leachates were analyzed using the same schedule as the sequential shake flasks.

Kinetic Testwork Duration and Data Management

The dissolution tests were started in early February 2006. Up to 38 weeks of data are available depending on the parameter. Tests are ongoing. Graphs illustrating concentrations obtained are

provided in Appendix C. This report is based on data collected and checked for quality to the end of October 2006.

Because different reporting limits were used for different cycles and different parameters, the following rules were used to plot data and allow trends to be apparent:

- If the parameter was not determined, the result is not plotted. This is the case for alkalinity and acidity which are only determined if the pH is above or below (respectively) specified values.
- If the result is undergoing quality control re-check, the result is not plotted.
- If the parameter is determined by the same method for each analysis, values below the reporting limit values are plotted as 50% of the reporting limit.
- Because the reporting limits for ICP-MS are below the reporting limit for ICP-ES:
  - If the result was determined by ICP-MS and was below the reporting limit, the value on the graph is 50% of the reporting limit. If the value is at or above the reporting limit, the value is plotted.
  - If the result was determined by ICP-ES and was determined to be below the reporting limit, no value is plotted.
  - If the result was determined by ICP-ES and was determined to be above the reporting limit, the value is plotted.
  - These rules can result in four cycles between plotted results if the parameter is not detected by ICP-ES (e.g. molybdenum in shake flask leachates).

Occasionally, "sawtooth" trends are apparent in which values alternate between high and low for the ICP-ES and ICP-MS analyses. This results from analytical "noise" around the ICP-ES reporting limit when reported values are slightly above the reporting limit. Aluminum is a particular example that commonly shows reported values above the ICP-ES reporting limit of 0.05 mg/L.

Many graphs are plotted on logarithmic axes to allow data spanning a wide range of concentrations to be compared.

4.2.3 Quality Assurance/Quality Control

In addition to Quality Assurance/Quality Control (QA/QC) of leachate chemistry performed by the analytical laboratory, QA/QC on the dissolution test procedure and overall leachate chemistry included blanks, duplicates, leachate ion balances and visual trend analysis to identify severe outliers. These measures were designed mainly in conjunction with the waste rock characterization plan, the results of which are described in RS42.

Quality assurance review for the waste rock and tailings programs identified two specific issues with respect to pH measurements and antimony leaching that affected the subsequent interpretation of the results. These issues also applied to the residue program and are discussed in the following paragraphs.

Trend analysis of pH measurements indicated a "sawtooth" trend in which values alternated between higher and lower values every other week. The reason for the pattern was that pH measurements were performed on filtered and unfiltered leachates on alternate weeks depending on whether samples were being collected for metals analysis. This was consistent with the analytical method. Because vacuum filtration potentially causes weakly buffered leachates to respond to changes in pressure by taking up or releasing carbon dioxide, determination of pH of filtered leachates was discontinued when the concern was identified. For results prior to this point in the test program, pH measurements on filtered leachates were discarded. Where pH results were needed for interpretation of other chemical parameters, the two nearby results were averaged (i.e. assuming the results could be interpolated linearly).

Review of antimony data indicated that test apparatus components of humidity cell tests constructed from polyvinyl chloride (PVC) were leaching antimony due to the use of antimony oxide in manufacturing. Antimony results from humidity cell results were therefore discarded.

4.2.4 Interpretation Methods

Results were interpreted using three methods:

- Leachate chemistry trends were examined as a function of time.
- All leachate results were combined and evaluated with respect to pH as the primary control on concentrations.
- Selected leachate results were input into a thermodynamic chemical equilibrium model to evaluate chemical saturation.

5 Results

5.1 Solids Characteristics

Table 5-1 provides quantitative mineralogy determined by Rietveld x-ray diffraction. Results are expressed as weight percentage of crystalline phases. If amorphous phases are present, the quantities of the indicated crystalline phases will be lower than indicated.

Table 5-2 provides elemental composition of the residues.

Table 5-3 provides sulfur forms from analysis and mineralogy, neutralization potential, carbonate analyses and acid-base accounts. The distribution of sulfate indicated by mineralogy was calculated as follows:

- The proportions indicated by mineralogy for natrojarosite, gypsum and bassanite were used to calculate the quantity of sulfate indicated by mineralogy using the formula weights of these minerals.
- The total sulfate indicated by mineralogy was calculated and compared to the analyzed sulfate. As shown in Table 5-3, there is a strong correspondence between the analytical and mineralogical sulfate amounts.
- The proportion of the analytical sulfate as gypsum and bassanite was calculated based on the proportion of these minerals indicated by mineralogy.

Acid potential (AP) in Table 5-3 was calculated based on sulfur not occurring as calcium sulfate based on the assumption that sulfur as natrojarosite is the source of acid. The factor for converting sulfur as natrojarosite to AP is 23.44 compared to 31.25 for sulfur as iron sulfide.

Table 5-1: Mineralogy of Residues

Mineral	Unit	Leach, no $CuSO_4$	Leach, with $CuSO_4$	Gypsum	Raffinate Neutralization	Fe/Al	Mg	Combined	Combined no Gypsum
Quartz	%	2.1	3	0.2	0.3	0.3	0.2	2.2	3.4
Plagioclase	%	25.6	5.8						2.2
Gypsum	%	8.1	6.9	99.8	96.3	98.9	76.8	73.4	53.3
Calcite	%							2.9	
Bassanite	%				3.4				
Siderite	%							1.2	
Actinolite	%	1.3							
Talc	%	4.3	3.9					3.2	3.2
Natrojarosite	%	34	63					9.6	29.3
Hematite	%	24.6	17.4					6	8.6
Goethite	%					0.8			
Butlerite	%							1.5	
Brucite	%						22.2		
Halite	%						0.8		

Leach Residues

The leach residues remain from the concentrate following pressure leaching. It was expected that this product would consist of both secondary minerals and resistant primary minerals (silicates).

Leach residues from processing of concentrates produced by sulfide flotation with and without copper sulfate were tested. The XRD results implied differences in the residual silicate component of the residues. The concentrate produced without copper sulfate had 26% residual plagioclase compared to 6% for the concentrate produced with copper sulfate. The oxidation products produced by the process are shown as mainly natrojarosite, which was also confirmed by the sodium content of the samples (Table 5-2). Hematite (iron oxide) and talc (basic magnesium silicate) were also present.

The residual metal content of the leach residues was significantly lower when the feed concentrate was generated using copper sulfate. Nickel, cobalt and zinc were an order-of-magnitude lower.

As expected, the leach residues were acidic and contained residual acidity shown by negative NP. In addition, the presence of natrojarosite resulted in higher acid potential than the other residues (111 and 199 kg $CaCO_3$/t).

Table 5-2: Elemental Composition of Residues

Parameter	Unit	Leach, no $CuSO_4$	Leach, with $CuSO_4$	Gypsum	Raffinate Neutralization	Fe/Al	Mg	Combined	Combined no Gypsum
Ag	ppm	11.05	23.8	0.23	0.9	0.33	0.12	4.93	11.05
Al	%	2.82	1.97	0.02	0.02	0.46	0.03	0.54	1.01
As	ppm	35.4	56.3	3	6	15	5	110	34
B	ppm	<10	<10	<10	<10	<10	<10	<10	<10
Ba	ppm	40	30	<10	<10	<10	<10	10	20
Be	ppm	0.05	<0.05	<0.05	<0.05	0.28	<0.05	<0.05	<0.05
Bi	ppm	4.89	6.84	0.5	0.8	0.81	0.04	1.94	3.53
Ca	%	1.88	1.41	14.5	17.1	18.8	15.1	16.35	11.3
Cd	ppm	0.66	0.19	0.22	0.73	0.48	0.44	0.29	0.23
Ce	ppm	8.21	5.45	1.71	0.6	0.85	1.6	2.35	2.93
Co	ppm	133.5	17.9	5.1	9.3	105.5	13.6	6.9	23.6
Cr	ppm	112	154	8	11	457	5	59	116
Cs	ppm	0.1	<0.05	<0.05	<0.05	0.09	<0.05	<0.05	<0.05
Cu	ppm	7380	1280	184.5	22.5	2960	26.9	519	945
Fe	%	17.3	26.9	0.05	0.08	1.88	0.05	5.21	11.75
Ga	ppm	6.47	5.06	2.54	0.07	0.57	0.09	1.22	2.43
Ge	ppm	0.41	0.47	<0.05	<0.05	0.08	<0.05	0.11	0.22
Hf	ppm	0.12	0.06	<0.02	<0.02	0.12	<0.02	0.04	0.04
Hg	ppm	0.04	0.02	0.12	0.35	0.07	0.05	0.11	0.03
In	ppm	0.3	0.355	23.3	0.011	1.01	0.083	0.164	0.28
K	%	0.11	0.1	<0.01	<0.01	0.01	0.01	0.02	0.05
La	ppm	3.6	2.5	0.7	0.5	0.5	2.3	1.1	1.6
Li	ppm	1	0.6	0.1	0.1	0.5	0.5	0.2	0.2
Mg	%	0.28	0.25	0.01	0.01	0.14	9.44	0.39	1.47
Mn	ppm	35	<5	<5	5	23	83	<5	<5
Mo	ppm	21.8	28.4	0.47	0.85	42.3	0.38	9.35	18.9
Na	%	1.28	3	0.02	0.04	0.13	1.04	0.58	1.62
Nb	ppm	0.18	0.21	<0.05	0.06	0.46	0.06	0.11	0.17
Ni	ppm	3270	410	99.9	192.5	2710	1230	260	674
P	ppm	290	110	<10	20	160	40	70	90
Pb	ppm	48.5	56.1	8.3	86.1	20.4	12.5	133.5	32
Rb	ppm	1.7	1	0.2	0.3	0.5	0.3	0.4	0.7
Re	ppm	0.012	0.002	0.015	0.022	0.013	<0.001	0.009	0.009
S	%	5.47	9.22	>10.0	>10.0	>10.0	>10.0	>10.0	>10.0
Sb	ppm	1.22	1.81	0.18	1.25	1.47	0.4	3.99	1.14
Sc	ppm	1.3	0.8	0.4	0.4	4.4	0.5	0.8	1.1
Se	ppm	49.8	73.4	0.3	0.4	0.6	0.5	20.4	39.2
Sn	ppm	4.8	6.1	0.2	0.5	1.6	<0.2	2	3.2
Sr	ppm	73.1	63.9	60.6	106	83.7	86.5	77.3	85.2
Ta	ppm	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Te	ppm	2.01	2.91	0.01	0.07	0.09	<0.01	0.83	1.48
Th	ppm	1.1	0.9	<0.2	<0.2	0.3	<0.2	0.3	0.4
Ti	%	0.066	0.042	<0.005	<0.005	0.007	<0.005	0.012	0.019
Tl	ppm	0.1	0.34	0.03	0.03	0.24	0.02	0.06	0.19
U	ppm	0.06	<0.05	<0.05	<0.05	0.55	0.1	0.06	0.09
V	ppm	39	41	1	1	16	1	11	21
W	ppm	0.57	0.49	0.09	0.18	1.12	<0.05	0.26	0.45
Y	ppm	3.88	3.29	1.28	1.14	1.06	7.64	2	2.86
Zn	ppm	171	15	30	98	164	23	32	36
Zr	ppm	<0.5	0.7	1.5	4.2	1.2	<0.5	9.8	1.9

Table 5-3: Acid-Base Accounting Results

Parameter	Unit	Leach, no $CuSO_4$	Leach, with $CuSO_4$	Gypsum	Raffinate Neutralization	Fe/Al	Mg	Combined	Combined no Gypsum
Paste pH	-	2.6	3.3	3.8	5.4	4.6	9.6	-	9.4
Sulphur Forms									
Total S	%	6.24	9.8	19.55	18.95	17.9	14.4	16.5	13.65
S as SO_4	%, S	6.1	9.3	18.35	18.55	17.35	13.65	15.9	13.5
Calculated Sulfur Forms From XRD Mineralogy									
S as Gypsum	%, S	1.51	1.28	18.55	17.90	18.38	14.27	13.64	9.90
S as Natrojarsoite	%, S	4.49	8.32	0.00	0.00	0.00	0.00	1.27	3.87
Total SO_4	%, S	5.99	9.61	18.55	17.90	18.38	14.27	14.91	13.78
Calculated Sulfur Not Present as Calcium Sulfate									
Non-Gypsum	%, S	4.73	8.52	1.00	1.05	-0.48	0.13	2.86	3.75
Neutralization Potential									
Fizz Rating	-	1	1	1	1	1	3	1	2
Neutralization Potential (NP)	kg $CaCO_3$/t	-17	-4	0	0	-10	371	10	51
CO_2	%	-0.2	-0.2	-0.2	-0.2	-0.2	1.8	-0.2	0.3
C	kg$CaCO_3$/t	2	2	2	2	2	40	2	7
Acid Potential (AP)	kg $CaCO_3$/t	111	199	23	25	-11	3	67	88
Acid-Base Accounting									
NP-AP	kg $CaCO_3$/t	-164	-271	-38	-13	-27	348	-51	-72
NP/AP	-	-	-	0.0	0.0	-	124	0.1	0.6

Gypsum Residue

Gypsum residue is the first hydrometallurgical precipitation product. It is produced by limestone addition to the leach solution following recovery of platinum group metals and prior to copper recovery.

The elevated calcium and sulfate content of the gypsum residue sample confirmed that it was dominantly hydrated calcium sulfate. XRD showed that it was 99.8% gypsum.

The metal content of this residue was very low. Acid-base accounting indicated that the dominant sulfur form was sulfate. About 1.2% of the sulfur was not accounted for by sulfate analysis, but XRD failed to recognize any other sulfur minerals. It is likely therefore that the difference reflects analytical uncertainties rather than unknown mineral content.

Raffinate Neutralization Residue

This residue is formed by an intermediate neutralization step between copper removal and precipitation of iron and aluminum.

Raffinate neutralization residue was identified as nearly entirely calcium sulfate (gypsum with minor bassanite) by XRD, which was confirmed by the dominance of calcium and sulfate in the sample. Like the gypsum residue, the metal content of this residue was very low and the sulfur forms analysis was consistent with the dominance of sulfate.

Fe/Al Residue

The Fe/Al residue is formed by two limestone addition steps prior to cobalt, nickel and zinc hydroxide recovery. Note that the full scale plant design combines Fe/Al removal with raffinate neutralization which means that the Fe/Al residue will be combined with the raffinate neutralization residue.

Like the raffinate neutralization residue, the Fe/Al residue was mostly gypsum with some iron and aluminum. Goethite was detected by XRD and iron and aluminum were both an order of magnitude higher than the other two dominantly calcium sulfate residues. The Fe/Al residue contained higher concentrations of copper and nickel compared to the gypsum and raffinate neutralization residues.

Magnesium Residue

The magnesium residue is formed following recovery of the mixed hydroxide product and is the result of final addition of lime to the process solutions. The magnesium originates from the original concentrate leaching step (dissolution of olivine), and addition of magnesium oxide to precipitate the mixed hydroxide product.

The crystalline component of this residue was dominated by gypsum as shown by the elevated sulfate and calcium. The XRD scan showed that brucite (magnesium hydroxide) was present and this was supported by the magnesium content of the sample. The sample showed a strong fizz reaction to hydrochloric acid and carbonate content was equivalent to 4% calcium carbonate (40 kg $CaCO_3$/t). This suggests that the hydroxide was partially converted to carbonate by atmospheric reaction.

Neutralization potential was high at 371 kg $CaCO_3$/t compared to the other residues; probably reflecting the presence of amorphous magnesium and possibly calcium hydroxide. This residue was alkaline (pH 9.6) and consistent with the hydroxide content.

The dominant trace metal in this residue was nickel (1230 mg/kg).

Combined Residues including Gypsum Residue

The largest component of the combined residue was the gypsum residue, followed by leach residue, raffinate residue, Mg residue and Fe/Al residue. As a result the material was dominantly calcium sulfate from the gypsum residue. The XRD result also showed the presence of calcite and siderite, presumably due to the magnesium residue.

The overall acid-base account for this sample indicated AP of 67 kg $CaCO_3$/t and NP of 10 kg $CaCO_3$/t. Because carbonate content was relatively low compared to NP, most NP was present as hydroxide from the magnesium residue. As a result, the residue was predicted to be potentially acid generating (NP/AP of 0.1). The acid generation potential is produced by natrojarosite which yields acidic leachate when dissolved. This process does not require an oxidant.

Combined Residues without Gypsum Residue

Although this combination did not include the gypsum residue, gypsum remained a significant component from the raffinate neutralization, Fe/Al residue and Mg residues. The overall characteristics of this material are similar to the combination of all residues though both AP and NP were higher in this residue due to the higher proportions of leach and magnesium residues respectively. The NP/AP of the residue was calculated to be 0.6.

5.2 Description of Leachate Chemistry

5.2.1 EPA 1311

Results for EPA 1311 (TCLP) are provided in Appendix B. Concentrations for all parameters were below regulated limits. None of the residues (individually or combined or combined without the gypsum residue) are classified as hazardous wastes.

5.2.2 EPA 1312

Results for EPA 1312 (SPLP) are provided in Appendix B. The use of weakly buffered acidic solution resulted in a range of final leachate pHs that reflect the stage of neutralization in the process. The leach residues had the lowest pH (2.79 and 3.38), followed by gypsum (4.1), raffinate neutralization (4.9), Fe/Al (5.0) and magnesium (9.7). The combined residues had pHs of 9.2 (all residues) and 9.4 (no gypsum residue). The variation in pH also resulted in different metal concentrations in solution. The leach residues had the highest metal concentrations. The two gypsum dominated residues had much lower metal concentrations (despite the lower pH). The higher metal content of the Fe/Al residue resulted in higher leachate metal concentrations. The three non-acidic leachates had low metal concentrations.

5.2.3 Price (1997) Method

Results for Price (1997) method are provided in Appendix B. This method uses deionized water as its extractant and a low liquid to solid ratio (3:1) compared to SPLP and TCLP which both use 20:1. The Price (1997) method is closely related to the SPLP method which uses deionized water weakly buffered by sulfuric and nitric acids. Both of these methods are different from the TCLP method which uses acetic acid to buffer pH.

As a result of the low liquid to solid ratio in the Price (1997) method, leachate pHs were lower than for the SPLP for the more acidic residues. The two leach residues had pHs of 2.1 and 2.6, followed by gypsum (3.3) and raffinate neutralization (4.2). For the three non-acidic residues, the pH was slightly lower with the Price (1997) method.

Differences in metal concentrations between the SPLP and Price (1997) methods appear to reflect the different liquid to solid ratios. Both copper and nickel concentrations were 4 to 7 times higher in the Price (1997) method leachates for the leach, gypsum, raffinate neutralization and Fe/Al residues. Concentrations of these elements in leachates from the three non-acidic residues were relatively low which obscured differences between the leachates.

5.3 Description of Kinetic Test Leachate Chemistry

5.3.1 Sequential Shake Flasks

Sequential shake flask charts are provided in Appendix C.1. Data are provided on a CD included in the report pocket.

Samples of leach residues, gypsum residue, raffinate neutralization residue and Fe/Al residue consistently produced acidic leachate but pH steadily increased for all residues except for the leach residue from processing of concentrate produced using copper sulfate. The raffinate neutralization residue showed erratic increase in pH reaching high values above 7. Leachates were dominated by calcium and sulfate. Sulfate concentrations were lowest for the leach residues (60 and 200 mg/L in most recent samples) but stable at about 1600 mg/L for the other acidic residues. Trends for other parameters included:

- Aluminum concentrations were greatest for the leach residues and slowly increased up to 3.4 mg/L for the residue produced from concentrate using copper sulfate. Other tests showed declining and low concentrations.
- Cobalt, copper, nickel and zinc concentrations were greatest for the Fe/Al residue as the test proceeded (0.003 mg Co/L, 1.4 mg Cu/L, 0.2 mg Ni/L and 0.02 mg Zn/L in most recent leachate) and steadily declined in all cases.
- Iron concentrations were greatest but declining for the gypsum residue (1.7 mg/L in a recent leachate).
- Lead concentrations were greatest and declining for the raffinate neutralization residue (0.01 mg/L in latest sample).
- Sodium concentrations were greatest for the leach residues and slightly increasing (7 mg/L in recent leachate).

Magnesium residues showed declining pH from 9.6 initially to 6.4 in recent leachates. Leachates were dominated by sulfate concentrations at higher levels than other residues (2,500 to 7,900 mg/L)

and magnesium rather than calcium as the dominant cation. Sodium and chloride concentrations were initially elevated (both above 1000 mg/L) then near to 1 mg/L. Concentrations of metal ions were very low and stable or declining. The exception was boron, which showed increasing concentrations reaching 0.16 mg/L in recent leachates.

Combined residues showed stable or very slightly declining pH reaching about 7. Leachate chemistry was dominated by calcium and sulfate at narrowly constrained stable concentrations. Alkalinity leaching declined but appeared to stabilize at about 30 mg $CaCO_3$/L. Major element chemistry reflected mixing of waters from the individual residues. Metal concentrations were generally low. The following exceptions were apparent:

- Arsenic leaching from both residues was greater than other residues and appeared to be stable. Greatest concentrations were 0.003 mg/L.
- Molybdenum leaching in these residues was greater than other residues. Maximum concentrations were 0.03 mg/L in both combined residues, but these concentrations declined. The combined residue without gypsum showed higher concentrations than the combined residue.
- Selenium concentrations were also greatest for these residues and showed a stable trend following decreases. The combined residue without gypsum showed higher concentrations (maximum of 0.019 mg/L). Concentrations in most recent samples were 0.007 mg/L.

5.3.2 Humidity Cells

Humidity cell results charts are provided in Appendix C.2.

Samples of leach residues, gypsum residue, raffinate neutralization residue and Fe/Al residue consistently produced acidic leachate but pH steadily increased for all residues, except for the leach residue from processing of concentrate produced using copper sulfate. The raffinate neutralization residue showed the greatest increase in pH (from 3.3 to 4.6, except for one point at 6.0). Leachates were dominated by calcium and sulfate. Sulfate concentrations rapidly stabilized between 1600 and 1900 mg/L, becoming 1600 mg/L as the test proceeded. Concentrations of other parameters either remained stable or decreased, for example:

- Aluminum concentrations were greatest for the most acidic leach residues (up to 14 mg/L following an initial flush) but remained relatively stable.
- Cobalt, copper, nickel and zinc concentrations were greatest for the Fe/Al residue as the test proceeded (0.03 mg Co/L, 10 mg Cu/L, 2.8 mg Ni/L and 0.25 mg Zn/L in most recent leachate) and steadily declined in all cases.
- Iron concentrations were greatest and relatively stable for the gypsum residue (1.7 mg/L in a recent leachate).
- Lead concentrations were greatest and declining for the raffinate neutralization residue (0.05 mg/L in latest sample).

- Sodium concentrations were greatest for the leach residues and stable or slightly decreasing.

Acidity of these leachates was stable or slowly declining for the leach residues at near 100 mg $CaCO_3$/L, and declining at lower levels for the gypsum dominated residues.

Magnesium residues showed declining pH from 9.4 initially to 7.4. Leachates were dominated by sulfate concentrations at higher levels than other residues (5,000 to 10,000 mg/L) and magnesium rather than calcium as the dominant cation. Sodium and chloride concentrations were initially elevated (both above 1000 mg/L) then declined to less than 10 mg/L. Concentrations of metal ions were very low and stable or declining.

Combined residues showed stable or very slightly declining pH near 7 after about 20 weeks. Leachate chemistry was dominated by calcium and sulfate. Alkalinity leached at low levels (about 23 mg $CaCO_3$/L in recent leachates). Major element chemistry reflected mixing of waters from the individual residues. For example, chloride leaching initially followed the same trend as the magnesium residue for the combined residue without gypsum. Metal concentrations were generally low. The following exceptions were apparent:

- Arsenic leaching from both residues showed an increasing trend though at low levels. The combined residue showed the greatest concentration in any residue leachate (0.007 mg/L).
- Molybdenum leaching in these residues was greater than any other residue. Maximum concentrations were 0.14 mg/L in the combined residue, but these concentrations declined. The combined residue without gypsum showed stable concentrations between 0.03 and 0.09 mg/L.
- Selenium concentrations were also greatest for these residues and showed a slowly increasing trend for the combined residue without gypsum. Concentrations in most recent samples were 0.03 mg/L.

5.3.3 MDNR Reactors

MDNR reactor results charts are provided in Appendix C.3.

Samples of leach residues, gypsum residue, raffinate neutralization residue and Fe/Al residue consistently produced acidic leachate but pH steadily increased for all residues except for the leach residue from processing of concentrate produced using copper sulfate. Like other tests, the raffinate neutralization residue showed the greatest increase in pH (from 3.3 to 4.6). Leachates were dominated by calcium and sulfate. Like the sequential shake flask results, sulfate concentrations were lowest for the leach residues (90 and 200 mg/L in most recent samples) but stable between 1400 and 1600 mg/L for the other acidic residues. Concentrations of other parameters either remained stable or decreased, for example:

- Aluminum concentrations were greatest for the most acidic leach residues (up to 2.8 mg/L following an initial flush) but remained relatively stable.

- Cobalt, copper and nickel concentrations were greatest for the Fe/Al residue as the test proceeded (0.006 mg Co/L, 3.1 mg Cu/L, and 0.5 mg Ni/L in most recent leachate) and steadily declined in all cases.

- Zinc concentrations were comparable in Fe/Al residue and raffinate neutralization residue as the test proceeded. Zinc concentrations in most recent leachates were about 0.04 mg/L.

- Iron concentrations were similar and relatively stable after about 20 weeks for the leach residue (copper sulfate used for concentrate), gypsum residue and raffinate neutralization residues (highest concentrations of about 0.2 mg/L).

- Lead concentrations were greatest and declining for the raffinate neutralization residue (0.02 mg/L in latest sample).

- Sodium concentrations were greatest for the leach residues and stable or slightly decreasing.

Magnesium residues showed declining pH from 9.8 initially to 7.4. Leachates were dominated by sulfate concentrations generally at higher levels than other residues (up to 5,000 mg/L) and magnesium rather than calcium as the dominant cation. Sodium and chloride concentrations were initially elevated (both above 200 mg/L) then declined to near or less than 1 mg/L. Concentrations of metal ions were very low and stable or declining.

Combined residues showed stable or very slightly declining pH between 7 and 7.4 after about 9 weeks. Leachate chemistry was dominated by calcium and sulfate. Alkalinity leached at low levels (about 23 mg $CaCO_3$/L in recent leachates). Major element chemistry reflected mixing of waters from the individual residues. For example, chloride leaching initially followed the same trend as the magnesium residue for the combined residue without gypsum. Metal concentrations were generally low. The following exceptions were apparent:

- Arsenic leaching from both residues was greater than other residues and appeared to be stable. Greatest concentrations were 0.002 mg/L.

- Boron leaching from the combined residue showed an increasing trend reaching a maximum concentration of 0.27 mg/L.

- Molybdenum leaching in these residues was greater than any other residue at first but then decreased and showed similar concentrations to the Fe/Al Residue. Maximum concentrations were 0.06 mg/L in the residue without gypsum, but these concentrations declined.

- Selenium concentrations were also greatest for these residues and showed a stable trend with maximum concentrations of 0.008 mg/L.

6 Interpretation of Dissolution Testwork

6.1 General Interpretation of Leachate Chemistry

6.1.1 Saturation Indices

In order to interpret the testwork chemistry, selected leachate chemistry data from all tests were input into Geochemists' Workbench (Bethke 2005) to evaluate whether the leachates were in equilibrium with any of the known mineral components of the residues. As shown in the foregoing descriptions, the humidity cells yielded the highest concentrations of metals. This is consistent with the low applied liquid to solid ratio in these tests (0.5 mL/g) compared to the sequential shake flasks (3 mL/g) and MDNR Reactors (2.7 mL/g). The humidity cells are therefore most likely to show chemical saturation. Table 6-1 shows saturation indices for minerals identified by XRD and other minerals that may control solubility. Color coding shows leachates that are close to chemical saturation (green shading for $-0.5<SI<0.5$) and well over-saturated (amber for $SI>0.5$). Saturation indices were calculated for the initial release and chemistry after 28 weeks following the decrease that typically occurred in the early weeks.

Leachate from leach residues appeared to be constrained by dissolution of silica and gypsum both initially and as the test proceeded. While it is likely that natrojarosite is dissolving releasing ferric iron, the low pH of the leachates means that the solubility of natrojarosite is not limited and ferric hydroxide does not precipitate. The gypsum and raffinate neutralization residues show much the same result though it appears that under initial conditions fluorite was also dissolving or forming in the raffinate neutralization residue as shown by SI above 1. Fluoride concentrations were 42 mg/L initially. In the higher pH Fe/Al residues, gypsum continued to be an important overall chemistry control, but ferric hydroxide was indicated as at saturation meaning that iron oxides were probably controlling solution chemistry. Also, the SI for tenorite was -1.2 compared to lower than -4.4 for the more acidic residues. For the magnesium residue, brucite and carbonates were near or well saturated consistent with the mineralogy of the samples. This effect was most apparent in the early weeks but diminished in week 28.

For the combined residues, the effect of gypsum dissolution was again apparent for the entire testwork period. In the first week, dissolution of natrojarosite to form ferric hydroxide appeared to be occurring as shown by the SI values. Week 28 showed the same effect but in reality iron was not detected so the SI's are maximum values and do not confirm that ferric hydroxide formed. Tenorite was over-saturated in the first week but not subsequently. Dissolution of carbonates also appeared to be an important effect.

In summary, the evaluation of saturation indices indicates that leachates were consistently in equilibrium with silica and gypsum and in the early stages of testing were probably also in equilibrium with iron and copper oxides. The interpretation did not provide any indication of possible constraints on the solubility of other potential contaminants such as cobalt, nickel and zinc. As a result, metal concentrations were compared to pH for all tests to evaluate pH control on metal concentrations.

Table 6-1: Saturation Indices for Humidity Cell Leachates

	Initial Release (week 1)								Release at Week 28							
Mineral	**Leach Residue, no $CuSO_4$**	**Leach Residue**	**Gypsum**	**Raffinate Neutralization Residue**	**Fe /Al**	**Mg Residue**	**Combined Residue**	**Comb. Residue, no Gypsum**	**Leach Residue, no $CuSO_4$**	**Leach Residue**	**Gypsum**	**Raffinate Neutralization Residue**	**Fe /Al**	**Mg Residue**	**Combined Residue**	**Comb. Residue, no Gypsum**
Hematite	2.7	2.7	5.2	6.9	10.0	13.4	13.6	13.5	3.3	2.4	5.9	7.2	8.2	10.4	10.4	10.4
Quartz	0.3	0.8	0.5	0.8	0.8	-0.7	0.8	-0.1	1.0	0.8	-0.3	-0.1	0.8	-1.5	1.2	1.1
Fluorite	-6.3	-3.2	-4.4	0.6	0.6	-0.2	-0.2	-0.5	-6.3	-6.6	-1.6	-1.5	-1.7	-0.9	-1.4	-1.4
Chalcedony	0.0	0.5	0.2	0.6	0.5	-0.9	0.5	-0.3	0.8	0.5	-0.5	-0.4	0.5	-1.8	0.9	0.9
Barite	-0.5	-0.7	-0.8	0.2	-0.5	0.0	-0.6	-0.3	-1.0	-1.2	-0.3	-0.1	-0.9	-0.8	-0.9	-1.1
Jarosite-K	-1.0	-0.9	-0.4	0.1	2.5	-5.4	-3.6	-4.3	-7.4	-6.6	-6.1	-4.7	-4.4	-7.9	-7.7	-7.6
Gypsum	-0.4	-0.3	-0.2	-0.2	-0.1	-0.1	-0.2	-0.2	-0.2	-0.2	-0.1	-0.2	-0.1	-0.1	-0.2	-0.1
Anhydrite	-0.6	-0.4	-0.4	-0.4	-0.3	-0.3	-0.3	-0.3	-0.3	-0.4	-0.3	-0.3	-0.3	-0.3	-0.3	-0.3
Amorphous Silica	-1.0	-0.5	-0.8	-0.4	-0.5	-1.9	-0.5	-1.3	-0.2	-0.5	-1.5	-1.4	-0.5	-2.8	-0.1	-0.1
Bassanite	-1.2	-1.1	-1.0	-1.0	-0.9	-0.9	-1.0	-1.0	-1.0	-1.0	-0.9	-1.0	-1.0	-0.9	-1.0	-0.9
Fe(OH)3(ppd)	-3.5	-3.5	-2.3	-1.4	0.1	1.8	1.9	1.9	-3.2	-3.7	-1.9	-1.3	-0.8	0.3	0.3	0.3
Jarosite-Na	-1.9	-2.5	-2.4	-2.0	0.8	-7.2	-5.2	-5.9	-9.4	-8.9	-8.8	-7.1	-8.3	-10.2	-9.8	-9.8
Epsomite	-2.2	-1.9	-3.5	-3.3	-1.9	-2.1	-2.1	-1.9	-5.4	-5.4	-6.6	-6.6	-6.4	-2.1	-4.1	-3.7
Tenorite	-5.7	-6.1	-5.0	-4.4	-1.2	1.1	0.7	0.9	-5.6	-6.7	-5.0	-5.4	-1.0	-1.3	-1.4	-2.0
Gibbsite	-6.4	-6.9	-4.7	-6.0	-3.3	-1.2	-0.5	-0.8	-1.5	-3.0	-3.2	-2.8	0.2	-1.0	-0.6	-0.7
Halite	-4.8	-3.9	-6.7	-6.6	-4.5	-4.3	-5.4	-4.4	-9.8	-9.5	-11.0	-10.5	-11.2	-9.7	-10.2	-10.0
Ni(OH)2(s)	-11.3	-11.7	-10.5	-8.5	-5.4	-1.6	-2.3	-2.5	-11.1	-12.3	-10.6	-10.2	-6.8	-5.4	-6.1	-5.7
Kaolinite	-11.8	-11.8	-7.9	-9.7	-4.6	-3.2	1.0	-1.3	-0.5	-3.9	-6.5	-5.3	2.4	-4.5	1.6	1.4
Brucite	-14.9	-13.7	-14.1	-12.3	-8.9	-0.3	-1.6	-0.8	-13.9	-14.9	-13.8	-13.3	-11.8	-3.2	-5.5	-5.0
Alunite	-10.2	-11.4	-8.0	-13.9	-8.2	-14.8	-11.4	-12.9	-2.6	-4.8	-10.3	-9.5	-1.8	-12.2	-10.7	-11.0
Cuprite	-20.9	-21.7	-19.2	-17.6	-12.3	-14.2	-14.9	-14.3	-15.7	-19.0	-12.0	-17.9	-11.1	-9.9	-13.1	-15.6
Talc	-31.9	-26.4	-28.5	-21.8	-11.9	8.3	10.3	9.1	-25.8	-30.0	-30.7	-28.4	-20.6	-4.1	-0.2	1.3
Dolomite[2]	-	-	-	-	-8.8	3.5	2.5	2.9	-	-	-16.3	-	-12.4	-0.2	-1.5	-1.4
Dolomite-ord[2]	-	-	-	-	-8.8	3.5	2.5	2.9	-	-	-16.3	-	-12.4	-0.2	-1.5	-1.4
Dolomite-dis[2]	-	-	-	-	-10.3	1.9	0.9	1.4	-	-	-17.9	-	-14.0	-1.8	-3.1	-2.9
Calcite[2]	-	-	-	-	-5.3	0.9	0.4	0.5	-	-	-6.7	-	-4.8	-0.9	-0.6	-0.7

Notes:

1. Green shading indicates saturation indices between -0.5 and 0.5. Amber shading indicates saturation indices greater than 0.5.
2. Carbonate not reported in leachates at low pH. Saturation indices for carbonate minerals not calculated.

6.1.2 pH Control on Metal Concentrations

The overall assumption in evaluating these plots is that if similar concentrations are indicated by tests operating at different liquid to solid ratios, and there is a relationship with pH that is consistent with first principles, it can be concluded that a solubility control is operating. If tests operating at low liquid to solid ratios yield higher concentrations than tests operating at high ratios, and the difference in concentrations can be accounted for by the ratio, it is more likely that the availability of soluble components is limited. The following bullets indicate observations for individual parameters.

Graphs are provided in Appendix D.

- Aluminum. A strong relationship between pH and Al concentration was defined mainly by humidity cell leachates but also by some MDNR reactor and shake flask leachates for the low pH samples. A strong control at neutral to basic pH was indicated by all test types.

- Arsenic. The leachate data indicated a pH relationship in which there was a pH minimum at about 6. Concentrations were comparable at both higher and lower pHs. At low pH, the relationship was defined by humidity cells. Lower concentrations were indicated for shake flask and MDNR reactor leachates. It is likely that solubility at acidic pHs is greater than indicated by the testwork. At pH greater than 6, arsenic concentrations were positively correlated to pH and the relationship was defined by all test types. The data are a good indication of arsenic constraints under these conditions.

- Cadmium. The results showed that cadmium was much more soluble under acidic conditions than neutral to basic conditions. The Fe/Al residues showed greatest solubility of cadmium at lowest pHs. Cadmium is probably more soluble at acidic pH than indicated by testwork. The presence of non-detectable cadmium at neutral to basic pH indicates that the detection limit is a reasonable indication of cadmium concentrations under these conditions.

- Cobalt. A strong relationship was shown for pH and cobalt concentrations. The bulk of the data describe increasing cobalt concentrations spanning several orders of magnitude as pH decreases from 9 to 3. The Fe/Al residues show a distinctive group at higher concentrations than the other residues. This group was defined by all three types of tests and implies a common solubility control for cobalt which is probably co-precipitated cobalt associated with iron oxides in this residue. Because this residue is also present in the combined residues and concentrations at neutral to basic pH are similar for all three tests, the data are a reliable indication of cobalt concentrations at neutral and moderately acidic pH when the Fe/Al residue is being leached.

- Chromium. Chromium results defined a strong pH relationship with all types of tests. Chromium concentrations showed a minimum at about pH 6 and higher concentrations for lower and higher pHs.

- Copper. The relationship for copper was similar to cobalt. At low pH, the Fe/Al residues showed the greatest concentrations in all test types. At higher pHs, copper concentrations were consistent for different residues. It appears likely that a copper oxide, co-precipitated with the iron oxide solids in the Fe/Al residue will control copper concentrations.

- Iron. Iron concentrations were mostly negatively correlated with pH except at the highest pH for which iron concentrations increased. Iron concentrations are higher than the ideal solubility of ferric hydroxide for pH greater than 4. This implies that ferric hydroxide was forming but that it formed as colloids which passed through the filter.

- Manganese. A strong negative correlation was apparent. Like Co and Cu, the Fe/Al residues showed the highest concentrations at lower pH implying that these residues were a source of co-precipitate. Because the relationship is indicated by all test types, the data appears to be a reliable indictor of manganese concentrations.

- Molybdenum. A positive relationship between molybdenum and pH was indicated for combined residues, Fe/Al residues and gypsum residues. The relationship at higher pHs was indicated by MDNR reactors and humidity cell samples.

- Nickel. The relationship for nickel was very similar to copper. The Fe/Al residues showed higher leachable nickel than other test types at acidic pH indicating that these residues with their elevated nickel concentrations probably contained co-precipitated nickel. Some leachates from the magnesium residues appeared to contain higher nickel concentrations than the combined residues possibly indicating a different solubility control (such as nickel hydroxide). The data for the combined residues which reflects the effect of leaching of Fe/Al residues imply lower nickel leaching from this control.

- Lead. Leaching of lead was apparent under acidic conditions but not neutral to basic conditions. Highest lead concentrations were apparent for the raffinate neutralization residue. The source of lead is unknown though this material contained the second highest lead concentration of any residue. Similar elevated lead concentrations were indicated for shake flasks and MDNR reactors but not for humidity cells.

- Selenium. Data for selenium showed that selenium leaching was greatest at higher pH and that concentrations were correlated with pH. The neutral to basic relationship was defined by all test types whereas under acidic conditions, greatest concentrations were shown by humidity cells implying that selenium was more soluble under these conditions than shown by the testwork.

- Thallium. In general, the relationship resembled nickel. The Fe/Al residue appeared to be the main source of leachable thallium and the relationship between pH and thallium concentration at neutral to basic pH was indicated by humidity cell and MDNR reactor leachates.

- Zinc. Like many other elements, the relationship for zinc appeared to be controlled by leaching of the Fe/Al residue as a source of co-precipitated zinc. Zinc concentrations at neutral to basic pH were consistently indicated by several different types of testwork.

Evaluation of metal concentrations with respect to pH indicates that under pH neutral to basic conditions, the concentrations indicated by humidity cells in combined residues are a reliable indicator of expected near equilibrium leaching. Maximum concentrations indicated in humidity cell leachates on combined residues (with gypsum) at neutral pH are shown in Table 6-2.

Leaching under acidic conditions is well-defined for the Fe/Al residue because solubility appears to be limited by the dissolution of iron oxides containing co-precipitated metal oxides of elements such as cadmium, cobalt, copper, manganese, nickel and zinc. For the acidic leach residue, upper limit solubility was not defined by the testwork; however, separate disposal of this waste is not contemplated.

Table 6-2: Summary of Maximum Concentrations Observed in Humidity Cell Leachates

Parameter	Concentration (mg/L)	Test Type
pH Range	6.6 to 8.5	All
SO_4	7347	Humidity Cell
Al	0.18	Shake flask
As[1]	0.004	Humidity Cell
Cd	0.0004	Humidity Cell
Co	0.005	Humidity Cell
Cr	0.05	Humidity Cell
Cu	0.015	Humidity Cell
Fe	0.4	Humidity Cell
Mn	0.0023	Humidity Cell
Mo	0.14	Humidity Cell
Ni	0.098	Humidity Cell
Pb	0.0005	Humidity Cell
Se	0.054	Humidity Cell
Tl	0.0002	Humidity Cell
Zn	0.01	Humidity Cell

Notes:

1. Arsenic concentrations increased to 0.0075 mg/L as the test proceeded. The arsenic concentrations shown are for initial leachates for which maxima occurred for other parameters.

6.2 Trend Evaluation

6.2.1 Trends in Leachate Chemistry Shown by Testwork

As discussed in Section 5.3, various trends in leach chemistry have been observed. These include:

- pH which has trended steadily upward for most acidic residues and steadily downward for most non-acidic residues.

- Parameters showing general stable trends without apparent increases or decreases, for example sulfate and calcium for most test materials using all three test protocols. Some metals have also shown stable leaching.
- Parameters showing steady downward trends in concentrations, which include most metals.
- Parameters showing steady upward trends in concentrations, for example, arsenic in humidity cell leachates and boron in sequential shake flask and MDNR reactor tests.
- Undetectable trends due to concentrations below detection limits.

The main factors expected to contribute to trends in leachate chemistry are the presence of process solutions that presumably equilibrated with the solids prior to testing, the progressive dissolution of the solids by weekly additions of deionized water and removal of the leachate, and interaction between parameters that affect solubility (mainly pH).

The initial flushing of process waters typically yielded water containing high concentrations of sulphate, chloride, magnesium and sodium. Calcium concentrations were relatively low in the humidity cell leachates during this phase. Subsequent stable concentrations of some parameters can indicate that the solution is in chemical equilibrium with the solid(s) that are the source of the parameter, but also may indicate that the quantity available for dissolution during the leaching cycle is the same each week. The latter is not a chemical equilibrium. Sulfate is controlled by the dissolution of gypsum and SI's indicate that the solutions are in chemical equilibrium with gypsum in most cases. The exception is shown by the MDNR reactors and sequential shake flask tests on leach residues which yielded sulfate concentrations below the level expected for gypsum. This is probably a result of the lower concentrations of gypsum in this material.

General declining trends in metal concentrations may be due to shifts in pH (both downward and upward) and declining availability of readily leachable solids. Dissolution is controlled by available surface area which is expected to decline as the tests proceeded. In addition, as oxides age they become more crystalline trapping the co-precipitated metals and making the metals less leachable.

The upward trend in arsenic concentrations leaching from humidity cells containing combined residues occurs at low concentrations but appears to be related to the presence or absence of gypsum. The trend could be a result of breakdown of calcium arsenate which occurs as it equilibrates with carbon dioxide in the atmosphere:

$$Ca_3(AsO_4)_2 + 3H_2O + 3CO_2(g) \rightarrow 2AsO_4^{3-} + 3CaCO_3 + 6H^+$$

Calcium arsenate is not known to occur in the residues but arsenic concentrations are too low to allow detection of the compound directly.

6.2.2 Trends Beyond Testwork Time Frame

The proposed final disposal method will involve combination of all residues. The following discussion considers the long term pore water chemistry expected for the combined residues.

In the time frame of the tests, the combined residues were non-acidic because buffering capacity from brucite and possibly also calcite were present to offset the acidity produced by dissolution of natrojarosite:

$$NaFe_3(SO_4)_2(OH)_6 + 3H_2O \rightarrow 2SO_4^{2-} + Na^+ + 3H^+ + 3Fe(OH)_3$$

Eventually, it is expected that acid buffering minerals will be exhausted and the residues will become acidic unless additional buffering capacity is added. This conclusion was confirmed by modeling the complete dissolution of all the mineralogical components of the residues in any of the proportions shown in Table 4-1 using React (Bethke 2005). As pH drops, metal mobility can also be expected to increase due to accelerated dissolution of the Fe/Al residue. Because the leach residue is dominantly natrojarosite and it is also the major component of the residue it is expected that acidification will result from dissolution of only a small proportion of the natrojarosite. However, under field conditions, dissolution of natrojarosite will be slow due to the slow movement of water through the residue mass.

PolyMet has proposed to add additional limestone or lime to the combined residues to ensure that they do not become acidic. The theoretical quantities of alkaline material needed can be calculated assuming the complete dissolution of natrojarosite balanced by calcium carbonate or hydroxide:

$$NaFe_3(SO_4)_2(OH)_6 + 3H_2O + 3CaCO_3 \rightarrow 2SO_4^{2-} + Na^+ + 3Fe(OH)_3 + 3Ca^{2+} + 3HCO_3^-$$

$$NaFe_3(SO_4)_2(OH)_6 + 3H_2O + 3/2Ca(OH)_2 \rightarrow 2SO_4^{2-} + Na^+ + 3Fe(OH)_3 + 3/2Ca^{2+}$$

Using these reactions, the quantities of limestone or lime required are 6.2 and 2.3 g/kg per percent natrojarosite, respectively. This does not consider the effect of buffering by brucite from the Mg residue, which would reduce the requirement. These calculations assume the complete dissolution of natrojarosite. In reality, the process will not proceed to this endpoint because the conversion to ferric hydroxide will result in a decrease in volume and formation of iron hardpan. The latter will act as a barrier to water movement and dissolution of the residues.

6.3 Comparison of Results with Other Testwork Programs

As indicated in Section 2.2.1, other hydrometallurgical processes exist for recovery of metals from copper sulfide concentrates but none are directly comparable to the process proposed for the NorthMet Project. No comparisons with other testwork programs can be made.

6.4 Effect of Process Variations

The main process variation expected to occur is in the proportion of the various residues and the mineralogical composition of the residues. Changes in the proportion of leach and magnesium residues will result in a need to adjust the amount of basic material (i.e. limestone or lime) to be added. The magnesium residue would need to constitute about 50% of the combined residue to eliminate the need for additional basic material.

6.5 Conclusions

Characterization of hydrometallurgical residues has shown:

- Four of the residues (leach, gypsum, raffinate neutralization and Fe/Al) are expected to be acidic. The magnesium residue will be basic.
- The dominant mineral in the leach residue is jarosite, which generates acidic water when dissolved. Leach residue is the dominant component of the combined residues.
- The other residues (including magnesium) are mainly gypsum. The gypsum and raffinate neutralization residues are nearly entirely calcium sulfate. The Fe/Al residue also contains goethite, and the magnesium residue contains brucite.
- The Fe/Al residue also contains iron probably in amorphous form that contains co-precipitated metals.
- None of the residues were classified as hazardous wastes using the EPA 1311 protocol.
- Kinetic leach tests using three different protocols showed very similar results. Dissolution effects generally decreased with time though dissolution of gypsum exerted a very strong effect.
- Neutral to basic leachates produced by combined residues are believed to represent chemical saturation conditions suitable for prediction of pore water chemistry.
- The combined residue produced non-acidic leachate during the test but is expected to become acidic in the future unless additional basic material is added. PolyMet has proposed to add additional basic material as required to offset the acid potential of natrojarosite.

7 Pore Water Chemistry Prediction

7.1 Operational Model

7.1.1 Explanation of Modeling Approach

Because humidity cell test leachates had equilibrated with the residues, the testwork leachates are assumed to be an analogue for the residue pore water chemistry. No adjustment to the water chemistry is assumed to scale-up from test to site conditions.

7.1.2 Inputs to Water Quality Model

The main variable that could cause variations in water chemistry is inflow rate. However, because the concentrations in testwork leachates were found to be in equilibrium with the solids, concentrations will be independent of flow rate.

7.1.3 Results

Table 6-2 indicates predicted maximum expected concentrations in pore waters provided that pore waters remain non-acidic due to the addition of additional basic material to offset acidity produced by long term dissolution of natrojarosite. It is proposed that these are maximum values because they were produced by initial contact of leachates with the residues.

Leachate chemistry is expected to be dominated by the dissolution of gypsum, which will result in elevated sulfate concentrations. The presence of magnesium due to the dissolution of brucite ($Mg(OH)_2$) will support higher sulfate concentrations than occurs for dissolution of pure gypsum.

The performance of combine residue without the gypsum residue is expected to be similar to combined residue with gypsum.

7.1.4 Conclusions

The chemistry of leachates in contact with combined residues in testwork has been estimated using results from leaching experiments.

7.2 Closure and Post-Closure Model

A separate prediction was not performed for closure conditions. Leachate chemistry as shown in Table 6-2 is expected to persist to closure. Slow movement of water through the residues is not expected to result in significant long term depletion of residues.

8 Conclusions

This report provides predicted pore water chemistry for the proposed disposal of combined residues in the hydrometallurgical residue cells. The results will be used to assess the requirement for treatment of residue leachates drained from cells at cell closure and the potential water quality impacts of leakage from the cells.

This report **"1UP005.01 – RS33/RS65 – Hydrometallurgical Residue Characterization and Water Quality Model – NorthMet Project - DRAFT"**, has been prepared by SRK Consulting (Canada) Inc.

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Appendices

Appendix A

Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan

Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan NorthMet Project, Minnesota

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Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan NorthMet Project, Minnesota

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1 Introduction

1.1 Background

PolyMet Mining Inc (PolyMet) is proposing to develop the NorthMet Project (Dunka Road Project of US Steel) near Babbitt, Minnesota. As a part of the Minnesota Department of Natural Resources (MDNR) "Permit to Mine" process a complete "mine waste characterization" will be required (Minnesota Rules Chapter 6132.1000). This document describes the plan developed for testing of flotation tailings and hydrometallurgical residue samples for the NorthMet Project.

The issues associated with tailings and residues at the NorthMet are expected to include acid rock drainage (ARD) and leaching of some heavy metals. The latter in particular are expected to include nickel and cobalt both of which do not require acidic conditions to be mobilized at elevated concentrations.

The objective of this program is to predict the reactivity of tailings and residues in their respective disposal areas for input into waste and water management planning, and environmental impact assessment. A separate Environmental Sampling and Analysis Plan describes characterization methods for effluents and emissions as metallurgical testing proceeds.

1.2 Geological Setting

The NorthMet Deposit is located in the intrusive mafic Duluth Complex of northern Minnesota. Disseminated copper-nickel-iron sulfides (chalcopyrite, cubanite, pentlandite and pyrrhotite) with associated platinum group element (PGE) mineralization will be extracted from several igneous stratigraphic horizons.

1.3 Agency Consultation and Design Process

This document was developed in consultation with staff from the Minnesota Department of Natural Resources (MDNR). The consultation included the following steps:

- June 22, 2005. A draft of the plan was prepared for MDNR Review.
- July 21, 2005. MDNR provided initial comments focussed primarily on flotation tailings.
- August 16, 2005. MDNR provided further comments on the characterization of hydrometallurgical wastes.
- September 9, 2005. SRK responded to the July 21, 2005 letter.
- September 14, 2005. A conference call was held to discuss the July 21, August 16 and September 9 letters.

This document includes responses to comments provided by MDNR, and has been prepared to conclude the design process and seek MDNR approval of PolyMet's plans to respond to the tailings/residue characterization component of requirements under Minnesota Rules 6132.1000.

1.4 Organization of This Document

This document describes:

- Section 2. Design basis for the program.
- Section 3. Analytical methods. This section describes methods used to analysis solids and leachates.
- Section 4. Use of the results in the context of water chemistry predictions.

1.5 Acknowledgements

The following individuals cooperated in the preparation of this plan:

- John Borovsky, Barr Engineering Company;
- Stephen Day, SRK Consulting;
- Paul Eger, MDNR;
- Jennifer Engstrom, MDNR;
- Don Hunter, PolyMet;
- Kim Lapakko, MDNR;
- Richard Patelke, PolyMet; and
- Jim Scott, PolyMet.

1.6 Laboratory Selection

The following laboratories will perform the procedures described in this plan (contact names for each laboratory are shown):

- ALS Chemex, North Vancouver, British Columbia – solids analysis listed in Section 4.1.1 (Bill Anslow);
- Optical – PolyMet or a Contractor (Richard Patelke);
- Sub-Optical Lab – McSwiggen and Associates (Peter McSwiggen);
- Canadian Environmental and Metallurgical Inc, North Vancouver, British Columbia – kinetic testing (Rik Vos); and
- Cantest Inc.. Vancouver, British Columbia - Kinetic test leachate analysis (Richard Jornitz).

2 Characterization Design

2.1 Objective of the Program

The overall objective of the program is to provide geochemical characterization information that can be used as inputs to design of management plans for the tailings and process residues and inputs into the environmental impact study (EIS) for the project.

2.2 Metallurgical Process Background

The processing and recovery of NorthMet Project ore to recover commodity metals will involve conventional flotation to produce a sulfide concentrate followed by hydrometallurgical treatment of the concentrate. The process flow sheets are provided in Appendix A.

Processing will result in the generation of the following waste products:

- Flotation tailings (low sulfide);
- Leach residue (silicates);
- Gypsum residue (calcium sulfate);
- Raffinate neutralization residue (calcium sulfate);
- Fe/Al residue (oxide); and
- Mg residue (oxide).

2.3 Tailings and Metallurgical Residue Disposal

Tailings and metallurgical residues are proposed for disposal in the existing impoundments at the former LTV Steel Mining Company (LTVSMC) operation (Figure 1). Review of the history of deposition at the LTVSMC tailings area indicates that the tailings are a result of processing ore feed from several iron ore pits and working faces in those pits. The ore was blended by truck delivery to loading pockets and then train delivery to crusher. Once in the plant ore was further blended in the coarse ore bins by a coarse ore tripper which continuously spread coarse ore across seven fine crushing lines and in the fine ore bins by a fine ore tripper which continuously spread fine ore across 34 mill lines. The fine ore was then processed through 34 mill lines in parallel. The tailings are a recombination from the 34 mill lines pumped to the basin and discharged at many spigots at the periphery of the basin. The tailings were deposited over many years as many layers in the basin.

Flotation tailings produced by conventional extraction of commodity-bearing sulfide minerals will be disposed in the existing Cells 1E, 2E and 2W. For the first five years of operation, cell 2W will be lined. Discharge methodology has not been determined but will most likely involve conventional discharge from one or more spigots. Hydrometallurgical residues produced by leaching of the sulfide concentrate will be disposed in lined basins within Cell 2W. The method used to transport the

residues to the cells has not been determined. The residues may be combined for disposal, or disposed separately depending on factors such as the possibility of selling some by-products.

The possibility of constructing tailings dams using cycloned tailings is being considered and has been incorporated in this test program.

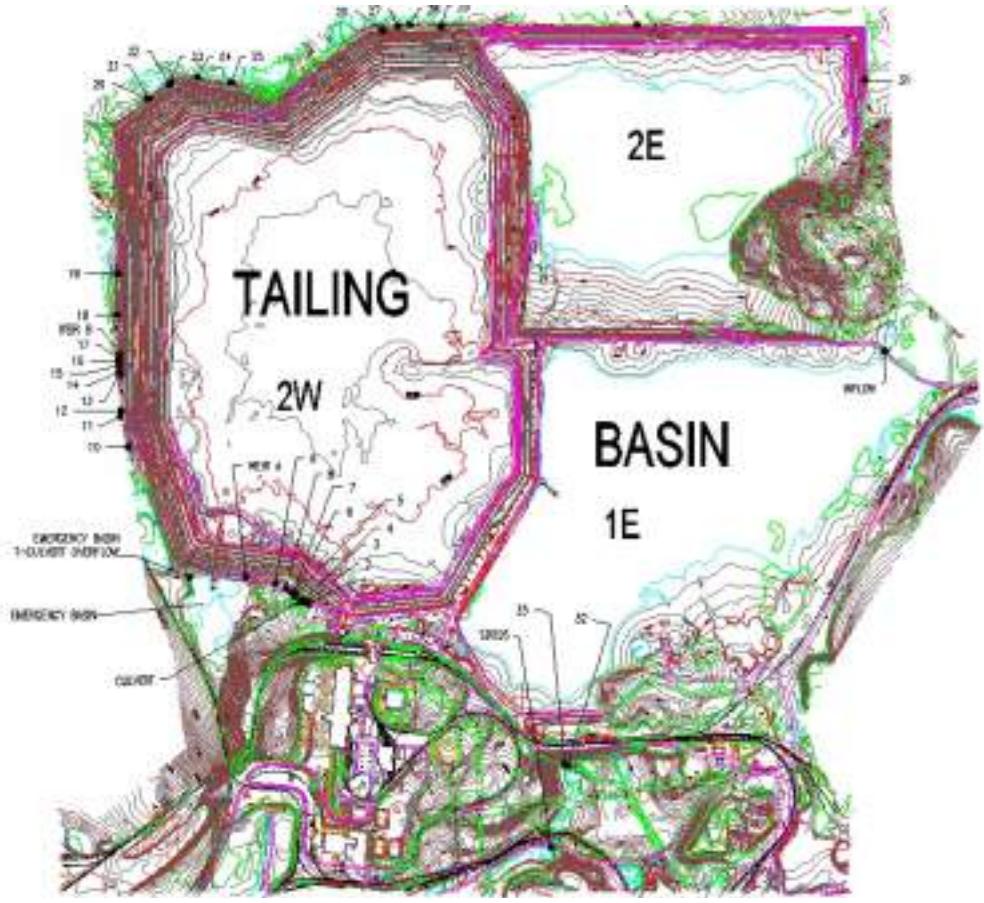

Figure 1: LTV Steel Mining Company Tailings Basins

2.4 Metallurgical Testing

Three ore composites were prepared under PolyMet's direction from diamond drill hole core bracketing head grades expected during mining. Pilot-scale metallurgical testwork was begun in July 2005 and continued into September. Flotation tailings testing was completed in August and included assessment of process alternatives. Addition of copper sulfate to improve sulfide concentrate recovery was evaluated for two ore composites.

A separate Environmental Sampling and Analysis Plan has been prepared to describe monitoring of air, water and solid emissions as the testwork proceeds. That document explains the rationale for preparation of the three ore composites representing copper grades of 0.3%, 0.35% and 0.4%. It also contains details of pilot plant monitoring designed to evaluate variations in tailings geochemical characteristics potentially produced by variations in ore characteristics and process performance. For example, the pilot testing program included frequent (every two hours) monitoring of tailings characteristic.

2.5 Design Basis

2.5.1 Flotation Tailings

Flotation tailings will consist primarily of silicate minerals with small amounts of residual sulfide minerals not recovered by flotation.

The number of variables expected to control reactivity is relatively small. Ore processing naturally results in a well-blended tailings product because the process requires a uniform feed to maximize recovery of commodities. Therefore, variables such as rock type, sulfide mineral type, silicate mineral type and source of ore within the layers of the Duluth Complex will not be significant. Preparation of the ore composite, which has been documented by PolyMet mimics mining at several faces and will result in composites each containing similar distributions of the main silicate and sulfide minerals. Variations will occur reflecting the distribution of commodity-containing minerals. Further, the requirement for grinding to optimize beneficiation of the commodity minerals limits the importance of variables such as mineral particle size and degree of liberation. The remaining variables are therefore expected to be:

- Sulfur content;
- Metal content; and
- Particle size (where separations occur at the deposition site as a result of hydraulic factors) resulting in differences in chemical and mineralogical composition.

The testing of tailings from the three ore composites will allow these variables to be evaluated. It is expected that recovery of sulfide minerals from ore will vary during testwork resulting in tailings containing variable concentrations of sulfur and metals. Results of two-hourly testing were provided in a memorandum to DNR dated January 6, 2006 (Appendix B). The composition of the four tailings samples initiated concurrently with preparation of this plan are shown in Table 1 along with the range of sulfur concentrations indicated by the two-hourly testing.

Table 1: Composition of Four Tailings Samples

Ore Composite	Total Sulphur Content of Ore	Copper Sulfate Used in Flotation	Range of Total Sulphur Concentrations in Tailings	Total Sulphur Content of Tailings Under Test
1	0.86	NO	0.19% to 0.28%	0.23%
		YES	0.09% to 0.13%	0.10%
2	0.90	NO	0.05% to 0.25%	0.20%
3	0.86	YES	0.09% to 0.25%	0.15%

2.5.2 Hydrometallurgical Residues

The residues are mainly chemical products in which the original concentrate components are oxidized to sulfates and hydroxides, and dissolved.

- Sulfur from the sulfide minerals is oxidized to sulfate and precipitated as calcium sulfate through the neutralization process;
- Iron released from sulfide and silicate minerals is precipitated as hydroxide; and
- Magnesium and aluminum released from silicate minerals are precipitated as hydroxides.

Copper is recovered in the process by electro-winning to produce copper anodes. Nickel, cobalt and zinc are recovered as a mixed hydroxide product.

Incomplete dissolution of the silicate and sulfide minerals results in generation of a fourth mineralogical leach residue.

Since these products represent near-complete dissolution, oxidation and neutralization of the concentrate, the chemistry of contact waters is expected to be controlled mainly by the simple dissolution of the compounds. Oxidation of residual sulfide minerals may occur from the leach residue but this effect is expected to minor.

Overall, the process of dissolution of neutralization products is unrelated to the composition of the ore and is not expected to vary in the long term except by depletion.

Pilot scale hydrometallurgical testing will be completed on two composite concentrate samples produced from processing of the three ore samples prepared with and without the use of copper sulfate in the flotation process.

3 Sample Handling and Analysis

3.1 Sample Shipping and Storage

SGS/Lakefield in Lakefield, Ontario, Canada is performing the metallurgical testing. Products will be shipped to Canadian Environmental and Metallurgical Inc. (Vancouver, British Columbia, Canada). Samples will be shipped and stored prior to testing as follows:

- Ore Feed Samples – Refrigerated;
- Sulfide Concentrate – Refrigerated;
- Flotation Tailings – Slurry in sealed pails with sufficient (6 cm) supernatant to ensure the samples are covered by water;
- Leach Residue – Cake, refrigerated; and
- Hydrometallurgical Residues – Cake, refrigerated.

Residual materials remaining after testing will be stored in the same condition as shipped.

3.2 Solids Characterization

3.2.1 Introduction

Table 1 summarizes the types of materials generated by testwork and the chemical testing procedures for each one. Physical testing of these products (including particle size determinations) is described in Environmental Sampling and Analysis Plan for the pilot plant test program. The DNR made several requests for multiple tests on some material types. In reality, the quantity of materials generated by metallurgical testing was limited and restricted the number of tests that could be performed. The 2-hourly testing showed that the sulfur content of the tailings did not vary widely and that the bulk samples under test will characterize the range of sulfur content of tailings.

Details of the test procedures are provided in the following sections.

Table 2: Procedures and Numbers of Samples for Testing

Material	Sulfur Forms	Neutralization Potential	Carbonate	Metals	TCLP	SPLP	Shake Flask	Optical Mineralogy	XRD	Sub-Optical	HCT	Sequential Shake Flask	Column	DNR Reactor Test	Layered Column
Ore feed	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore				
Sulfide Concentrate	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore						
Bulk Flotation Tailings – Without $CuSO_4$[1]	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore			1/Ore	
Bulk Flotation Tailings – With $CuSO_4$[1]	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore			1/ore	Multiple
Cyclone Sands	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore			1/ore	
Tailings Slimes	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		2	1/ore	
Tailings Beaches	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		1/Ore	1/Ore	1/Ore	1/Ore		2	1/ore	
Leach Residue	1	1	1	1	1	1	1	1	1	1	1	1		1	
Gypsum Residue	1	1	1	1	1	1	1		1	1	1	1		1	
Raffinate Neutralization Residue	1	1	1	1	1	1	1		1	1	1	1		1	
Fe/Al Residue	1	1	1	1	1	1	1		1	1	1	1		1	
Mg Residue	1	1	1	1	1	1	1		1	1	1	1		1	
Reactive Residues[2]	1	1	1	1	1	1	1		1	1	1	1		1	
Reactive Residue without Gypsum[3]	1	1	1	1	1	1	1				1	1		1	

Notes:

1. Flotation tailings samples were produced with and without the use of copper sulfate in the process to enhance recovery of sulfide minerals to the sulfide concentrate.

2. Combination of leach, gypsum, raffinate, Fe/Al and Mg residues.

3. Combination of leach, raffinate, Fe/Al and Mg residues.

3.2.2 Bulk Chemical Characterization

A split of each sample will be submitted for an extensive suite of analyses, as follows:

- Sulfur forms (total S, S as sulfate).
- Paste pH.
- Neutralization potential and carbonate.
- 50 elements (mostly metals by ICP scan following aqua regia (nitric and hydrochloric acids) digestion.
- Whole rock oxides. This is provides total concentrations of major elements.

Method detection limits are provided in Appendix C.

3.2.3 Regulatory Leachate Characterization

All samples will be submitted for regulatory leach tests (EPA 1311; EPA 1312) to provide data for waste classification purposes should this be needed. Testing of the ore samples will provide a baseline for comparison to effects from processing.

3.2.4 Shake Flask

A third leach leachate procedure was used as the first step of the sequential shake flask leach procedure described in Section 3.3.3 for hydrometallurgical residues. This procedure, developed by Price (1997) for the British Columbia Ministry of Energy Mines and Petroleum Resources, uses a lower leach ratio (1:3) to improve detection of low levels of metals. The lixiviant is deionized water which typically has a pH between 5 and 6. The leachate is not fixed as in the TCLP and SPLP methods. The extraction is performed by shaking in a glass or plastic container for 24 hours, after which the leachate is extracted and analyzed.

3.2.5 Mineralogical Characterization

Mineralogical characterization will include:

- Optical Analysis on feed, product and mineral wastes (Flotation Tailings, Leach Residue). Other residues will not be examined optically since they are precipitates;
- X-Ray Diffraction on all samples to determine crystalline compounds; and
- Sub-Optical Analysis on ore feed, mineral wastes, and if practical residues to determine the distribution of trace elements in individual minerals prior to and following processing.

3.3 Kinetic Test Methods

3.3.1 Humidity Cell

Humidity cell testing will be performed on ore feed (to characterize three types of ore stockpiles), flotation tailings and leach residues using ASTM Procedure D 5744 – 96 (Reapproved 2001). This procedure was selected for the following reasons:

- Similar procedures have been in use under different names since the late 1980s (e.g. MEND 1991). The results can therefore be evaluated in the context of more than a decade of experience using the procedure.
- It is a standard procedure approved by the ASTM and is therefore defensible as a method.

The ASTM procedure provides some options for varying the test procedure. Appendix D provides a detailed listing of the requirements of the ASTM procedure, options chosen and any variances from the ASTM procedure.

3.3.2 MDNR Reactor

To allow comparison with previous MDNR studies, bulk tailings samples will be tested using a procedure referred to as the "MDNR Reactor" experiment. An apparatus specifically designed by MDNR (Appendix E) contains 75 g of solids.

3.3.3 Sequential Shake Flask Test

All residues are being tested using a sequential leach procedure consisting of weekly repetition of the leach procedure developed by Price (1997) in which the solid to liquid ratio is 1:3 (Section 3.2.4). The procedure involves weekly leaching of roughly 300 g of solids in a plastic bottle. The leaching step consists of addition of deionized water and agitation for 24 hours. The leachate is then decanted for analysis. Between leach steps, the bottle remains open to the atmosphere.

This procedure was requested by the DNR to provide a more aggressive evaluation of hydrometallurgical residue dissolution than occurs in humidity cells.

3.3.4 Leach Columns

A procedure to evaluate the interaction between leachate from NorthMet tailings and LTV tailings was designed and presented to the DNR (Appendix F). The procedure provides for two subaerial columns to generate leachate from the NorthMet tailings.

3.3.5 Leachate Analysis

Leachates from kinetic tests will be analyzed for the parameters indicated in Table 2, which also shows reporting limits. These limits are higher than the detection limits for the analytical instruments. Reporting limits represent the level at which the analytical laboratory (CANTEST) is

confident that the concentrations are quantifiable to an acceptable level. The instrument is able to detect much lower levels but these concentrations carry a very high degree of uncertainty which includes “undetectable”.

Low level leachate analyses for dissolved elements as shown in Table 2 are performed every four weeks (weeks 0, 4, 8 etc). On the intervening even numbered weeks (2, 6, 10 etc.), an ICP scan is performed using a higher detection limit primarily to determine the trend in major ions. pH and conductivity are determined every week. Acidity, alkalinity, inorganic carbon, sulphate, fluoride and chloride are determined every other week.

It is expected that testing of flotation tailings will result in very dilute leachates containing low concentrations of the metals of interest. Back-calculation of metal concentrations from other testwork performed by DNR indicates that cobalt and nickel concentrations could be in the tens of nanograms per litre (ng/L) for nickel and near nanograms per litre for cobalt. Quantification of these low metal concentrations is needed to provide reasonably constrained estimates of metals concentrations in the tailings storage facility.

A number of different approaches are available to quantify low levels of nickel and cobalt:

- The routine leachate analysis will achieve a reporting level of 0.0001 mg/L (100 ng/L). Should concentrations be undetected, detection limits of 50 ng/L can be obtained with additional processing effort using the same routine method.
- Specialist methods can achieve lower detection limits. These are non-routine (for example, evaporation to increase concentrations) and will need to be developed as the need arises.
- Existing testwork demonstrates that good correlations exist between cobalt and nickel concentrations in leachates. Detectable nickel concentrations can be used to estimate cobalt concentrations if this relationship can be demonstrated.
- In the event of undetectable low levels, a scale-up methodology will be agreed upon with MDNR to translate non-detectable concentrations to tailings concentrations. Detection limit values will be used in modeling calculations.

Table 3: List of Parameters for Low Level Analysis of Humidity Cell Leachates

Parameter	Reporting Limit	Parameter	Reporting Limit
pH (standard units)	-	Acidity	1
Conductivity (µS/cm)	1	Alkalinity	1
Chloride	0.2	Sulfate	0.5
Fluoride	0.05	Total Inorganic Carbon	1
ORP (mV)	-		
Dissolved Elements (mg/L)			
Aluminum	0.001	Mercury	0.00005[2]
Antimony	0.0001	Molybdenum	0.00005
Arsenic	0.0001	Nickel	0.0001 (0.00005)[1]
Barium	0.0001	Potassium	0.02
Beryllium	0.0002	Selenium	0.0002
Bismuth	0.0002	Silicon	0.05
Boron	0.005	Silver	0.00005
Cadmium	0.00004	Sodium	0.01
Calcium	0.01	Strontium	0.0001
Chromium	0.0002	Tellurium	0.0002
Cobalt	0.0001 (0.00005)[1]	Thallium	0.00002
Copper	0.0001	Thorium	0.0001
Iron	0.01	Tin	0.0001
Lead	0.00005	Titanium	0.0002
Lithium	0.0002	Uranium	0.00005
Magnesium	0.005	Vanadium	0.0002
Manganese	0.00005	Zinc	0.001

Notes: 1. Low detection limits are available for cobalt and nickel as shown.
2. Lower level mercury analyses will be performed on selected samples.

3.4 Analysis of Remaining Sample Following Dissolution Tests

Analyses of the remaining sample following dissolution tests will be considered depending on the results obtained from the tests. Generally, these analyses can be of value if the test has undergone a major chemical change during the procedure (e.g. change from alkaline to acidic leachate) or if calculations indicate that a large quantity of one or more minerals or elements has been depleted. These is little value in post-test analysis if the depletion quantity is less than the uncertainty that can be expected from sampling of the test residue analysis. Experience also indicates that mineralogical analyses are also of little value unless weathering processes have had a detectable effect on the sample.

Therefore, residue analyses will be performed if:

- A large drop in pH has occurred (for example, from above 7 to below 5).
- Depletion calculations indicate that more than 10% of an important component was removed during the procedure.

Residue analyses will consist of the same procedures performed prior to the test.

3.5 Quality Assurance/Quality Control

A Quality Assurance Project Plan (QAPP) is being prepared for this project. To summarize, QA/QC includes the following components:

- Roughly 10% of all solids analyses will be performed in duplicate as sample availability permits.
- Roughly 10% of all cell and reactor tests will be run as duplicates if sufficient test material is available.
- A blank cell and reactor containing no sample will be operated to check for contamination of leachates by construction materials.
- Individual leachate results will be reviewed.
- Ion balances on leachate results will be reviewed. In general, imbalances of ±10% are considered acceptable. Re-analysis if requested depending on the nature of the imbalance.
- Data trends in kinetic test leachates will be analysed to check for anomalies.

4 Use of Data for Water Quality Predictions

4.1 Introduction

The data obtained from these programs will be used to estimate water quality during operation and closure. The following sections describe the application of the data to making water chemistry predictions.

4.2 Operational Water Quality

4.2.1 Flotation Tailings

Operational water quality in flotation tailings impoundments tends to be dominated by process water since this is the largest volume of water moving into and out of the impoundment by discharge and reclaim. Processes resulting from oxidation are not usually significant because continual placement of fresh tailings covers up older tailings before extensive weathering is initiated. The tailings pond water and trapped pore waters therefore reflect re-circulating process water. Seepage from the impoundments typically has a process water signature modified by anoxic conditions in the saturated tailings and interaction with LTVSMC taconite tailings.

Seepage chemistry will also be influenced by dam construction, particularly whether drains will be needed for stability. If drains are needed, water will be drained horizontally away from the dams, rather than the normal vertical seepage thru the coarse fraction of tailings near the dam.

The method used to estimate tailings pond water during operations is typically a coupled water and load balance that evaluates the effect of build-up of solutes in the tailings pond water due to interactions (e.g. mineral dissolution), reagent addition in the process, unintended additions in the process (e.g. Mo from lubricants), dilution due to rain and snowfall, dilution by run-in, and solute load loss due to encapsulation. If kinetic testing shows a short term leaching effect from beaches and dam faces, these loads are included.

The effect of interaction of saturated tailings with LTVSMC tailings will be evaluated directly by column tests (Appendix F). These tests will indicate whether any significant losses or additions occur as process water moves through the taconite tailings.

4.2.2 Leach Residues

Rinsed leach residues are expected to be relatively soluble and water chemistry associated with the residues will be dominated by equilibration of rinse. Water quality may be affected by the method of disposal (pumped slurry vs truck hauled solids). Since the residues will be continually accumulated, the operational water chemistry will be a result of mixing of rinsate water with precipitation and run-in. The chemistry of contact water during operation will be estimated directly from dissolution test results. Pore water chemistry indicated by testwork will be evaluated using MINTEQA2 or similar thermodynamic equilibrium models.

The effect of small amounts of sulfide minerals in the leach residues will be evaluated.

4.3 Water Quality at Closure and Post Closure

4.3.1 Flotation Tailings

At closure, the main effect is removal of inflows of process water and on-set of oxidation of tailings resulting in metal and possibly acidity loadings. The water and load balance developed for operational conditions is typically modified to evaluate these effects.

The long term closure modeling will need to consider evolution of the tailings profile in response to oxidation. This type of modelling uses humidity cell weathering rates, tailings physical characteristics, and moisture profiles predicted by modelling (HELP and HYDRUS-2D) as inputs. The propagation of the oxidation front and acidity front through the tailings is predicted and used to model the movement of solutes. The migration of the fronts can be used to estimate changes in solute loading in the future due to arrival of chemical fronts at the base of tailings. This type of modeling can be coupled with groundwater models to predict the chemistry of groundwater leaving the site.

4.3.2 Leach Residues

Water associated with the leach residues is expected to evolve as rinsate is displaced by dilute water from rainfall and snowmelt. Humidity cells will show whether this results in changing water quality. Concentrations indicated from humidity cells may be suitable for direct prediction of water chemistry or may be adjusted using MINTEQA2 or similar thermodynamic equilibrium models.

5 References

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Appendix A

Process Flow Sheets

BATEMAN METALS

ORE CAR TIPPER

COARSE CRUSHER DUMP POCKET

PRIMARY CRUSHER (1)

PRIMARY CRUSHER PRODUCT POCKET

SECONDARY CRUSHER (4)

APRON FEEDER (4)

CONVEYOR

CONVEYOR

COARSE ORE BIN

FEED CHUTE (18)

BELT FEEDER (6)

RETRACTABLE CONVEYOR (3)

TERTIARY CRUSHER (3)

VIBRATING SCREEN FEEDER (6)

VIBRATING SCREEN (6)

QUARTERNARY CRUSHER (6)

CONVEYOR

CONVEYOR

FINE ORE BIN

FEEDER (132)

RODS

BALLS

ROD MILL (12)

PRIMARY CYCLONE (36)

BALL MILL (12)

PAX

AIR

MIBC

ROUGHER FLOTATION (4)

ROUGHER CONDITIONING TANK (2)

AIR

MIBC

PAX

SCAVENGER FLOTATION (12)

REGRIND CYCLONE (9)

FLOCCULANT

FLOTATION TAILS THICKENER (2)

WW1572

PAX

CLEANER 1 CONDITIONING TANK (1)

MIBC

AIR

CLEANER 1 FLOTATION (8)

PAX

REGRIND HOPPER (2)

REGRIND MILL (3)

FLOCCULANT

CONCENTRATE THICKENER (1)

PROCESS WATER TANK (1)

FLOAT TAILINGS

AIR

MIBC

WW1572

CLEANER 2 FLOTATION (4)

PAX

CLEANER 3 CONDITIONING TANK (1)

CLEANER 3 FLOTATION (4)

(2)

(2)

CONCENTRATE GRINDING

FINAL CONCENTRATE SURGE TANK (2)

TO AUTOCLAVE FEED TANKS

BAB403-P-DX-003

FIGURE 1
POLYMET FEASIBILITY STUDY
COMMINUTION AND FLOTATION
SCHEMATIC FLOW DIAGRAM

BATEMAN METALS

1ST STAGE HYDROXIDE PRECIPITATION TANK (3)
AIR
MgO
1ST STAGE HYDROXIDE THICKENER (1)
FLOCCULANT
SERVICE WATER
HYDROXIDE FILTER (1)
MIXED HYDROXIDE PRODUCT
2ND STAGE HYDROXIDE PRECIPITATION TANK (3)
LIME
FLOCCULANT
2ND STAGE HYDROXIDE THICKENER (1)
TO AUTOCLAVE LET DOWN TANK
Mg REMOVAL TANK (4)
LIME
Mg REMOVAL THICKENER (1)
TO HM PROCESS WATER TANK
TO REACTIVE RESIDUES
STEAM
N_2
NaHS
Cu PRECIPITATION TANK (3)
CuS THICKENER (1)
FROM FINAL CONCENTRATE SURGE TANKS
HCl
AUTOCLAVE FEED TANK (3)
OXYGEN
AUTOCLAVE (3)
BLOWDOWN TO FLOTATION TAILINGS
SERVICE WATER
COOLING TOWER
SERVICE WATER
AUTOCLAVE GAS SCRUBBER (3)
HEAT EXCHANGER
TO ATMOSPHERE
SERVICE WATER
VENTS FROM ALL TANKS
FINAL GAS SCRUBBER
AUTOCLAVE FLASH VESSEL (3)
FLOCCULANT
LEACH RESIDUE THICKENER (1)
LIMESTONE
AIR
STEAM
BLEED
Fe REMOVAL TANK (4)
FLOCCULANT
Fe REMOVAL THICKENER (1)
HM PROCESS WATER
LIMESTONE
Fe/Al REMOVAL THICKENER FILTER
Fe/Al RESIDUE TO REACTIVE RESIDUES
Al REMOVAL TANK (2)
FLOCCULANT
Al REMOVAL THICKENER (1)
PRE NEUTRALISATION THICKENER (1)
FLOCCULANT
GYPSUM FILTER (1)
HM PROCESS WATER
GYPSUM RESIDUE TO REACTIVE RESIDUES
LIMESTONE
PRE NEUTRALISATION TANK (4)
FLOCCULANT
THICKENER (1)
SERVICE WATER
PGM FILTER (1)
NaHS
PGM PRECIPITATION (3)
HM PROCESS WATER
LEACH RESIDUE FILTER (3)
SERVICE WATER
HM PROCESS WATER
HM PROCESS WATER TANK
REACTIVE RESIDUES
ORGANIC MAKE-UP
LOADED ORGANIC TANK
COPPER EXTRACTION (3)
COPPER STRIPPING (2)
WASH (1)
SERVICE WATER
GUAR GUM
$CoSO_4$
H_2SO_4
COPPER ELECTROWINNING (160)
PRODUCT COPPER CATHODES
H_2SO_4
OXYGEN
RELEACH VESSEL (2)
TO ATMOSPHERE
SERVICE WATER
EW MIST SCRUBBER
DISCHARGE TO SOLVENT EXTRACTION
SERVICE WATER
PGM CONCENTRATE FILTER (1)
PRODUCT PGM CONCENTRATE
LIMESTONE
FLOCCULANT
RAFFINATE NEUTRALISATION THICKENER (1)
HM PROCESS WATER
RAFFINATE NEUTRALISATION (3)
RAFFINATE NEUTRALISATION FILTER (1)
GYPSUM RESIDUE
CRUD REMOVAL
SOLIDS TO REACTIVE RESIDUES
LIQUID TO LOADED ORGANIC TANK
BAB403-P-DX-004

FIGURE 2
POLYMET FEASIBILITY STUDY
HYDROMET PROCESS PLANT
OPTION 1 - MIXED Ni/Co
HYDROXIDE PRECIPITATION
SCHEMATIC FLOW DIAGRAM

Appendix B

Update on Sample Selection from 24 Hour Testwork

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Memo

To:	Jennifer Engstrom, MDNR	**Date:**	January 6, 2006
cc:	John Borovsky, Barr	**From:**	Stephen Day
	Jim Scott, PolyMet		
	Don Hunter, PolyMet		
Subject:	NorthMet Project Tailings and Hydromet Residue Testwork – Update on Sample Selection from 24 Hour Testwork	**Project #:**	1UP005.001

Jennifer

We have now received the 2-hourly tailings total sulfur analyses from the pilot plant testwork. These results allow final recommendations to be made for the selection of samples for tailings and hydrometallurgical testwork.

1 Results and Implications of 2-Hourly Sulfur Analyses

Results of the 2-hourly sulfur analyses are shown by the coloured solid lines in Figure 1. The broken horizontal lines are the concentration of sulfur in the composite tailings samples currently being tested in humidity cells. The sulfur content of the ore composites was very uniform (Parcel 1, 2 and 3, 0.86%, 0.9% and 0.86%, respectively) as shown by the solid black lines in Figure 1.

The trend in sulfur results in tailings is explained by the chronology of the testwork and evaluation of addition of copper sulfate as a reagent:

- Flotation testwork began on July 17 with Parcel 2 without the use of copper sulfate. Parcel 2 was processed entirely without using copper sulfate. As shown, sulfur concentrations varied from 0.05% to 0.25% reflecting adjustment of the process conditions early in the testwork. The average was 0.19%. The composite tailings sample has a sulfur content of 0.2% closely representing the average.

- Testwork continued with Parcel 1 without using copper sulphate. Processing was continuous so one point is shared between Parcel 2 and Parcel 1. The range of sulfur concentrations was 0.19% to 0.28% with an average of 0.24%. The composite sample was 0.23% and is close to the average.

- Pilot plant testwork was suspended on July 19 to allow for further bench scale testing on recovery of metals.

- The pilot plant resumed on August 8 using Parcel 1. Addition of copper sulfate was evaluated. This reagent causes activation of the sulfide mineral surfaces and improves bulk sulfide flotation. The effect of copper sulfate on tailings characteristics was immediately apparent for Parcel 1.

Total sulfur concentrations decreased to a range of 0.09% to 0.13% (average 0.1%) and the resulting tailings composite was 0.1%.

- Processing continued with Parcel 3 using the copper sulfate additive. Sulfur content of the tailings varied over a wider range (0.09% to 0.25%, average 0.18%) though the range was comparable to the total range indicated by processing of other ore packages. The resulting composite had a total sulfur content of 0.15%.

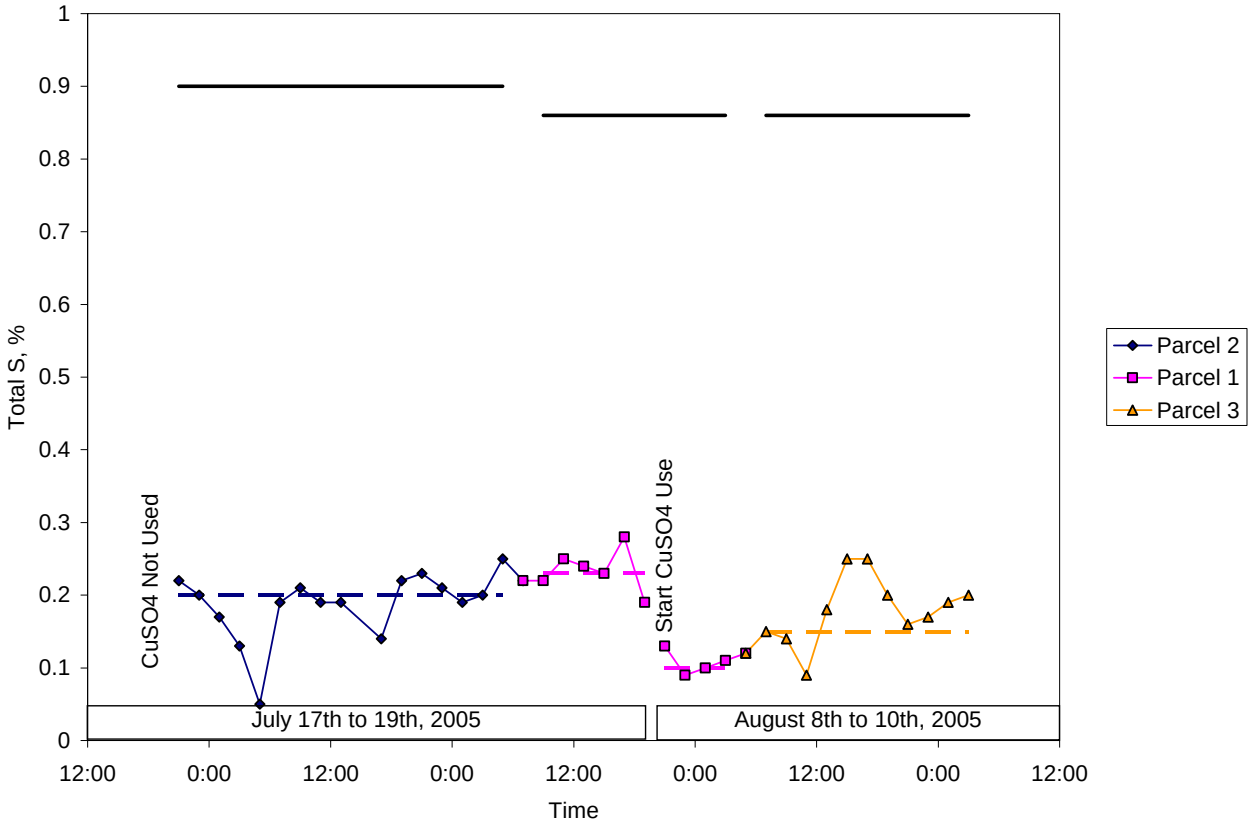


Figure 1. Results of 2-Hourly Total Sulfur Analyses. Solid lines and points connect 2-hourly results. Broken solid lines are sulfur concentrations in composite tailings samples representing each stage of testwork. Solid horizontal lines are the respective ore composite sulfur contents.

Based on the process testwork, Polymet has the made decision to advance the project with the use of copper sulfate to optimize overall sulfide mineral flotation. This decision is beneficial for the tailings since it is expected to lower the overall sulfide content.

The process testwork showed that sulfur concentrations in the tailings can be expected to vary in response to changes in process conditions including the use of copper sulfate. Parcel 3 showed that the use of copper sulfate may not always result in low sulfur content in tailings, and therefore there is need to capture sulfur concentrations approaching 0.25% in the kinetic testwork. The samples generated without copper sulfate provide the required range and can be tested to represent the potential for higher sulfur concentrations in the tailings. The lack of copper sulfate for the Parcel 2 and 1 samples is not expected to have significantly affected the reactivity of the residual sulfide minerals in the tailings:

It is therefore concluded that:

- Kinetic testing of all four tailings samples should be continued.
- No additional samples are needed to represent the range of sulfur content expected in tailings.

2 Testing of HydroMet Residues

2.1 Source of Sulfide Concentrate for Hydromet Process Evaluation

Evaluation of the hydromet process was performed using two bulk sulfide concentrates produced by processing of ore parcels 2 and 1 (without copper sulfate) and ore parcels 1 and 3 (with copper sulfate). The sulfide concentrates contained the following total sulfur concentrations:

- No Copper Sulfate
  - Parcel 2 – 23.6%
  - Parcel 1 – 21.3%
- With Copper Sulfate
  - Parcel 1 – 22.1%
  - Parcel 3 – 21.6%

It is apparent that the sulfur content of the concentrates does not vary significantly though the effect of copper sulfate on concentrate sulfur content for Parcel 1 is apparent and corresponds with the matching decrease in sulfur content of the tailings. Since the decision has been made to proceed with the use of copper sulfate, only the residues produced from sulfide concentrate generated using copper sulfide should be tested.

2.2 HydroMet Residues

All the expected HydroMet Residues were produced by processing of the sulfide concentrate generated using copper sulfate.

A difference exists between the way that the residues were recovered in the pilot test compared to actual operating conditions.

To summarize, the first step in the process is the leaching of the sulfide concentrate to produce a low pH pregnant solution containing all the commodity metals. Subsequent recovery of the metals involves a series of pH adjustments to the leach solution that results in precipitation of products and residues. The products are then refined to recover the contained metals (copper, nickel, cobalt, PGM, zinc). The residues contain entrained leach solutions that have to be recovered to optimize recovery of commodity metals. Under full-scale operating conditions, recovery of the leach solutions from the residues will occur by rinsing the residue cakes with pH-adjusted re-cycled final process water to displace the leach solutions. The pH adjustment is required to ensure that metals in the leach solution are not lost to the solids.

However, under pilot plant conditions, the recycled process water was not available because the processing of the leach solutions occurred in a stepwise rather than continuous fashion. Each metal recovery step was performed and completed before proceeding to the next. The final process solution that will be used for rinsing at full-scale was only generated at the end of the pilot plant and was therefore not available for the residue rinsing steps. The difference between full-scale and pilot plant conditions represents a practicality of metallurgical testing in that operation the pilot plant continuously is not an option with the available quantity feed concentrate.

Residues generated by the pilot plant were rinsed with locally obtained river water. No additional rinsing of the residues is proposed for the dissolution testwork. Rinsing with river water was less aggressive in displacing metal-laden leach solutions than can be expected with pH-adjusted process water. The residues can therefore be expected to contain higher metal content than under operating conditions and indicate greater leachable metals in dissolution tests. The testwork will tend to over-estimate rather than under-estimate water quality for water management planning and impact assessment.

3 Conclusions

The following actions are proposed:

- Kinetic testing of all four existing tailings samples in dissolution tests will continue.
- Testing of residues produced by hydromet testing of sulfide concentrate (with copper sulfate) will be started as described in the "Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan"

Appendix C

Parameter Lists and Detection Limits for Analysis of Solids

ALS/CHEMEX METHOD CODE	ITEM	UINTS	CHEMEX DETECTION LIMIT
ME-ICP61 (four acid)	CU%	%	0.001
ME-ICP61 (four acid)	NI%	%	0.001
S-IR08 (LECO SULFUR)	S%TOT	%	0.01
ME-ICP61 (four acid)	S%ICP	%	0.01
PGM-ICP23 (30 GRAM)	PT_PPB	PPB	5
PGM-ICP23 (30 GRAM)	PD_PPB	PPB	1
PGM-ICP23 (30 GRAM)	AU_PPB	PPB	1
ME-ICP61 (four acid)	CO_PPM	PPM	1
ME-ICP61 (four acid)	AG_PPM	PPM	0.5
ME-ICP61 (four acid)	ZN_PPM	PPM	2
ME-ICP61 (four acid)	CD_PPM	PPM	0.5
ME-ICP61 (four acid)	MO_PPM	PPM	1
ME-ICP61 (four acid)	PB_PPM	PPM	2
ME-ICP61 (four acid)	AS_PPM	PPM	5
ME-ICP61 (four acid)	CR_PPM	PPM	1
ME-ICP61 (four acid)	V_PPM	PPM	1
ME-ICP61 (four acid)	TI%	%	0.01
ME-ICP61 (four acid)	AL%	%	0.01
ME-ICP61 (four acid)	CA%	%	0.01
ME-ICP61 (four acid)	FE%	%	0.01
ME-ICP61 (four acid)	K%	%	0.01
ME-ICP61 (four acid)	NA%	%	0.01
ME-ICP61 (four acid)	MG%	%	0.01
ME-ICP61 (four acid)	MN_PPM	PPM	5
ME-ICP61 (four acid)	P_PPM	PPM	10
ME-ICP61 (four acid)	BA_PPM	PPM	10
ME-ICP61 (four acid)	BE_PPM	PPM	0.5
ME-ICP61 (four acid)	BI_PPM	PPM	2
ME-ICP61 (four acid)	SB_PPM	PPM	5
ME-ICP61 (four acid)	SR_PPM	PPM	1
ME-ICP61 (four acid)	W_PPM	PPM	10

ALS/CHEMEX METHOD CODE	ITEM	UINTS	CHEMEX DETECTION LIMIT
ME-ICP41 (Aqua regia digestion)	CU%	%	0.001
ME-ICP41 (Aqua regia digestion)	NI%	%	0.001
ME-ICP41 (Aqua regia digestion)	S%ICP	%	0.01
ME-ICP41 (Aqua regia digestion)	CO_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	AG_PPM	PPM	0.2
ME-ICP41 (Aqua regia digestion)	ZN_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	CD_PPM	PPM	0.5
ME-ICP41 (Aqua regia digestion)	MO_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	PB_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	AS_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	CR_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	V_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	TI%	%	0.01
ME-ICP41 (Aqua regia digestion)	AL%	%	0.01
ME-ICP41 (Aqua regia digestion)	CA%	%	0.01
ME-ICP41 (Aqua regia digestion)	FE%	%	0.01
ME-ICP41 (Aqua regia digestion)	K%	%	0.01
ME-ICP41 (Aqua regia digestion)	NA%	%	0.01
ME-ICP41 (Aqua regia digestion)	MG%	%	0.01
ME-ICP41 (Aqua regia digestion)	MN_PPM	PPM	5
ME-ICP41 (Aqua regia digestion)	P_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	B_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	BA_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	BE_PPM	PPM	0.5
ME-ICP41 (Aqua regia digestion)	BI_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	GA_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	HG_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	LA_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	SB_PPM	PPM	2
ME-ICP41 (Aqua regia digestion)	SC_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	SR_PPM	PPM	1
ME-ICP41 (Aqua regia digestion)	W_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	TL_PPM	PPM	10
ME-ICP41 (Aqua regia digestion)	U_PPM	PPM	10

ALS/CHEMEX METHOD CODE	ITEM	UINTS	CHEMEX DETECTION LIMIT
ME-ICP06--whole rock geochemisrty by ICP-AES	SIO2	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	AL203	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	TIO2	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	FE2O3	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	CAO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	MGO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	MNO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	NA2O	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	K2O	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	P2O5	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	BAO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	SRO	%	0.01
ME-ICP06--whole rock geochemisrty by ICP-AES	LOI	%	0.01

Appendix D

Options and Variance in ASTM Humidity Cell Procedure

9. Sample Preparation

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
9.1	Air dry as-received bulk samples of solid material to prevent the additional oxidation of reactive minerals or compounds. If air drying is not practicable, oven dry the solid material at a maximum temperature of 50 ± 2°C for 24 h, or until a constant weight is reached.	Samples were air-dried at room temperature (~ 20 °C).	
9.1.1	If exploration-generated or run-of-mine solid material samples are not readily available, archived dried and crushed samples from geological exploratory or development drilling programs may be used for preliminary evaluations of ore and waste rock from new operations; this is provided that the available solid material samples are not significantly finer than 95 % passing a No. 12 (1.7-mm) sieve. Document the sample drying and preparation procedures used during the drill sampling program in order to interpret the results properly. Evaluate the effects of drying temperature on metals volatilization (for example, mercury in cinnabar vaporizes at temperatures exceeding 80 to 90°C) and mineral morphology and chemistry modifications (for example, on heating at temperatures exceeding 100°C, chalcocite changes crystal form and is oxidized subsequently from Cu_2S to CuO, $CuSO_4$, and SO_2). Especially ensure that the effects of particle size distribution changes resulting from the more finely crushed sample are considered in the interpretation (this is, the potential for increased liberation of acid-producing and acid-consuming minerals with an attendant increase in mineral surface area).	NA	
9.1.2	In mining waste evaluations, the particle size for mill tailings will be significantly finer (commonly less than 150 µm/100 mesh) than the particle size distributions from ore and waste rock. Pilot plant tailings should be used if mill tailings are not available.	NA	
9.2	Screen the air-dried bulk samples through a 6.3-mm (¼-in.) screen in accordance with Test Method E 276. Crush any oversize material so that 100 % passes the screen.	NA	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
Note 7	Caution: Recent accelerated weathering studies of run-of mine waste rock from metal mines demonstrate that crushing a bulk sample so it passes a 6.3-mm (¼-in.) screen may change the character of the sample by artificially increasing liberation and consequent surface areas of acid-producing and acid-consuming minerals contained in the + 6.3-mm (¼-in.) material. A suggestion for avoiding this problem is to segregate the - 6.3-mm (¼-in.) fraction by screening rather than crushing, and to test that fraction according to the protocol and equipment described in this text. The + 6.3-mm (¼-in.) material can be tested separately (for example, Brodie, et al (10) describe a large-scale humidity cell test that would accommodate – 75-mm material). Samples from the drill core and cuttings also present material sizing problems, which must be considered when interpreting drill core and cuttings accelerated data. The drill core must be crushed to -6.3-mm (¼-in.) to fit the cell described in this test method. The resulting size distribution from crushing will differ from that of run-of-mine due to differences in fracture patterns inherent to blasting practices that produce run-of-mine material. By contrast, drill cuttings size fractions are commonly less than 6.3-mm (¼-in.) due to the rotary-percussive nature of obtaining the sample.	NA	
9.3	Mix and divide the bulk sample to obtain a representative test unit with a weight in the range of 8 to 10 kg, using a riffle splitter with 1-in. (2.54-cm) chutes. Divide the test unit into eight nominal 1-kg specimens. Seal each test specimen in a moisture-barrier bag.	NA	
Note 8	The dried sample should be mixed through the riffle splitter at least once before making any splits; recombine the splits resulting from the sample mixing exercise by pouring individual splits either over each other or through the splitter again. Once the actual split is made, it is wise to re-mix it (according to the above procedure) prior to making the next split.		Samples were mixed through the riffle splitter once.
9.4	Select one test specimen at random, and determine the moisture content by weighing and drying to constant weight at 80 ±5°C.		Determined at 20 °C
9.4.1	Crush the dried test specimen so that at least 95 % passes a 1.7-mm (10-mesh) screen, in accordance with Test Method E 276.	NA	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
9.4.2	Divide the crushed test specimen in half twice, using a riffle splitter with 6.35-mm (¼-in.) chutes, and select a ¼ subsample at random.	NA	
9.4.3	Transfer the selected subsample to a ring and puck grinding mill and grind to a nominal of 95 % passing a 150-µm (100-mesh) screen, in accordance with Test Method E 276. Use the subsample for chemical and mineralogical characterization of the test unit.	NA	
9.5	Select one test specimen at random, and determine the particle size distribution in accordance with Test Method E 276.	ASTM	
9.6	Select one test specimen at random for use in the accelerated test method. Divide the test specimen into four nominal 250-g subsamples using the riffle splitter with 25.4-mm (1-in.) chutes, and label and store in vapor-barrier bags until it is time to load the humidity cells.	NA	
9.7	Reserve the remaining test specimens for replicated testing or to resolve disputed results.	NA	

10. Apparatus Assembly

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
10.1	The humidity cells are table-mounted at a height sufficient to accommodate the placement of both the humidifier and one Erlenmeyer flask for effluent collection from the bottom of each cell. During the water-saturated and dry-air portions of each weekly cycle, feed air is metered to the bottom of each cell at the selected rate (1 to 10L/min). Feed air for the three-day dry–air portion is routed first through a desiccant column and then to each of the cells through a dry-air manifold. Feed air for the water-saturated air portion is routed through a water-filled humidifier by means of aeration stones or gas dispersion fritted cylinders/disks, and then to each humidity cell lid air exit port to prevent the short circuiting of air through cells containing more permeable solid material samples. A separatory funnel rack is mounted on the table that holds the cells if the weekly water leach is applied dropwise (drip trickle). Multiple separatory funnels (one for each cell) are held in the rack during the drip trickle leach that is performed on the seventh day of each weekly cycle. The separatory funnel can be used to meter the required water volume slowly down the sides of the cell wall until the sample is flooded if the weekly leach is to be a flooded leach.	Humidity cells are constructed of acrylic tubing with an inside diameter of four inches and an overall height of twelve inches, with an acrylic base plate. The base plate is glued to the tube and threaded with a nylon hose adapter to which a length of tubing is attached to allow for leachate drainage into a collection container. A perforated PVC support plate is positioned inside the cell, one inch above the base plate and covered with six layers of nylon mesh. A nylon adapter is threaded into the side of the cell between the support plate and the base plate and a length of tubing was connected from the side adapter to the humidifier to facilitate the inflow of humid air to the cell. A dry air line is also connected to each cell. Each cell is covered with a removable acrylic lid.	Approximately 16 cells per humidifier Flood leaching: peristaltic pump using a peristaltic pump Temperature: 20 ± 2°C. Feed air rate to be determined.

11. Procedure

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.1	Cell Loading:		
11.1.1	If more than one humidity cell is used at one time, label each with a sequential number, and use the same number for the matching collection vessel (Erlenmeyer flask).	ASTM	
11.1.2	Weigh each humidity cell (without its lid) and each collection vessel; record the tare weights of each to the nearest 0.1 g.	ASTM	
11.1.3	Cut the filter media (such as 12-oz/yd^2 polypropylene described in 6.11) to the humidity cell's inside diameter dimensions so that it fits snugly yet lies flat on the perforated support.		Shark Skin filter paper (320mm)
11.1.4	Re-weigh the humidity cell, and record the resulting tare to the nearest 0.1 g; the original cell tare (11.1.2) minus the new cell tare is the weight of the filter media.	ASTM	
11.1.5	Transfer the contents from each of the four bags containing the 250-g samples (9.6) into the humidity cell. Prior to the transfer, mix the contents of each bag by gentle rolling to eliminate possible stratification that may have occurred during sample storage.	ASTM	
11.1.6	Re-weigh the loaded cell, and record the weight to the nearest 0.1 g; the loaded cell weight minus the combined cell and filter-media tare weight is the weight of the sample charge.	ASTM	
11.2	First Leach:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.2.1	The first leach (whether drip trickle or flooded), designated as the Week 0 leach, initiates the 20-week long humidity cell test and establishes the starting or initial characteristics of the leachate. Either a 500-mL or 1-L volume of water may be used for the weekly leachates, depending on the weekly pore volume desired or the quantity of solution required for analytical purposes; however, once a weekly volume has been selected, that weekly volume must remain constant throughout the 20-week testing period. A centrifuged cell culture of *Thiobacillus ferrooxidans* may be used in the first leach in order to ensure that optimum conditions for accelerates weathering are present at the beginning of the test.	500 mL Flood Leach	
Note 9	In the testing of mining wastes, cation (including metals and trace metals) and anion loadings are commonly high in the Week 0 leachate due to the dissolution of pre-existing soluble oxidation salts present in the sample prior to sample collection. The average number of weekly accelerated weathering cycles required to flush these pre-existing salts ranges from 3 to 5 weeks. Oxidation products observed during these 3 to 5 weeks are principally from the pre-existing salts, while those products observed after this period are considered to be solely a function of the accelerated weathering procedure. A method for estimating the amount of pre-existing oxidation salts present in a solid material sample is described by Sobek, et al (6). A comparison of estimated salt storage data obtained using this method with the first thee weeks of humidity cell effluent loadings from three different samples is describes by White and Jeffers (7).	NA	
11.2.2	Fill a separatory funnel with for each cell with de-ionized water using a volumetric flask. If the leach is to be performed using the drip trickle method, set each separatory funnel above its corresponding cell, and adjust the drip rate (approximately 3 to 4 L/min) so that the solid material sample is wetted thoroughly but not flooded.	NA	
11.2.3	A minimum of 2 to 3 h is commonly required to complete the drip trickle leach.	NA	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.2.4	If the leach is to be performed by flooding, the separatory funnel can be used to meter the selected water volume slowly down the sides of the cell wall until the sample is flooded. This application method reduces hydraulic agitation of the sample surface commonly caused by pouring liquid from an open-mouthed vessel. Alternatively, flooding may be accomplished by any application apparatus (for example, a peristaltic pump) that supplies the selected volume of leachant at a reasonable rate without causing agitation and suspension of the finer fractions contained in the sample charge.	ASTM	
11.2.4.1	Allow the flooded cell to sit for a period of 1 h before draining the leachate into the Erlenmeyer collection flask. The 1-h leach time commences after all of the leachant has been placed in the cell. The solid material sample should be saturated and covered with leachant to a depth sufficient to maintain sample saturation. In testing mining wastes, the observed depth of leachant cover from a 500-mL flooded leach performed in 10.2-cm (4.0-in.) ID cells is approximately 2.5 cm (1.0 in.).	ASTM	
11.2.5	The following is performed once the leaching process has been completed: to reduce the effects of evaporation, and to prevent the contamination of each cell by airborne contaminants, place the lids on their corresponding cells and let the cells complete the leachate draining process for the remainder of the leaching day and overnight.	ASTM	
11.2.6	Disconnect the cells on the day following the leach, and weigh and record the weight of each cell and Erlenmeyer collection flask. Set each filled collection flask aside for leachate analyses. (Measurements of pH and Eh and sample preservation procedures must be performed as soon as possible after leachate collection.) Return each cell, replace the filled collection flasks with clean, tared Erlenmeyer flasks, hook up all connections, and begin the dry-air cycle.	ASTM	
11.3	Dry-Air Cycle:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.3.1	The commencement of the three-day dry-air period marks the beginning of each new weekly cycle of the accelerated weathering humidity cell test; the first full-week cycle after the first leaching is designated Week 1; subsequent weeks (commencing with the second dry-air period) are designated as Week 2, Week 3 … . Week n, etc.	ASTM	
11.3.2	To perform the dry-air cycle, feed air is metered to the humidity cell array with a flowmeter (see 6.3) set at a target rate in the range of 1 to 10 L/min per cell, depending on the objectives of the testing. The air flow rate must be checked daily and adjusted to the target value ± 0.5 L/min.	ASTM	
11.3.3	Feed air from the flowmeter is routed first through a desiccant column and then to each of the sells through a dry-air manifold. Air exiting the desiccant column should have a relative humidity of less than 10 % as measured with a hygrometer (see 6.23).	ASTM	
11.3.4	To maintain similar positive air pressure through the cells, attach a water-bubbling vessel to each humidity cell air exit port coming out of the humidity cell lid; a 50-mL Erlenmeyer flask with a rubber stopper containing a vent and air inlet tube serves as a simple and efficient bubbler.	ASTM	
11.3.5	The dry air is passed through each humidity cell for three days. Air flow rates from each of the cells should be checked each day, recorded, and adjusted, if necessary. See also Note 10.	ASTM	
11.4	Wet-Air Cycle:		
11.4.1	The three-day wet-air period commences on the fourth day of each weekly cycle.	ASTM	
11.4.2	To perform the wet-air cycle of the method, feed air is routed through a water-filled humidifier via aeration stones or gas dispersion fritted cylinders/disks and then to each humidity cell.	ASTM	
11.4.3	The water temperature in the humidifier is maintained at 30 ± 2°C to ensure that the sparged air maintains a relative humidity of approximately 95 % as measured with a hygrometer (see 6.23) from one of the humidifier exit lines. Air flow rates to each of the cells should be checked each day, recorded, and adjusted, if necessary.	ASTM	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
Note 10	It is good practice to measure the air flow rates and relative humidity of the air exiting each humidity cell during each day of the three-day dry- and wet-air periods; the measurements should be taken at the same time each day from the humidity cell air exit port; these measurements can be accomplished by installing a quick-disconnect fitting in the tubing that connects the air exit port to the bubbler.	NA	
Note 11	Coals spoils in eastern states are commonly saturated; Caruccio (10) has suggested the following geographic control alternative to the dry-air versus saturated-air scheduling: (1) Eastern States Samples – Six days of saturated air (versus three days dry/three days wet); and (2) Western States Samples – Three days dry/three days wet.	NA	
11.5	Subsequent Weekly Leaches:		
11.5.1	A second leach with water is performed on the day following the end of the three-day wet-air period (that is, day seven of the first weekly cycle). This leach marks the end of the first weekly cycle and is designated as the Week 1 leach.	ASTM	
11.5.2	Subsequent leaches are designates as Week2, Week 3 … Week n, and they mark the end of the weekly cycle for that numbered week. Perform each weekly leach as described in 11.2.2 – 11.2.5. Weekly weighing of the test cells is optional.	ASTM	No weekly weighing of the cells.
11.6	It is recommended that the weekly accelerated weathering cycles described in 11.2, 11.3, 11.4 and 11.5 be performed for a minimum of 20 weeks.	ASTM	
Note 12	Additional weeks of accelerated weathering may be required to demonstrate the nature of the material, depending on the chemical composition of the solid material. For some metal mining wastes, researchers have shown that as much as 60 to 120 weeks of accelerated weathering data may be required to demonstrate the complete weathering characteristics of a particular sample (7, 12). The criteria for ending the testing may be site specific and should be agreed before initiating the testing.	ASTM	
11.7	Leachate Analyses:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.7.1	Analyze the leachates for specific constituents or properties, or use them for biological testing procedures as desired, using (1) appropriate ASTM test methods or (2) methods accepted for the site where disposal will occur. Where no appropriate ASTM test method exists, other test methods may be used and recorded in the report, provided that they are sufficiently sensitive to assess potential water quality impacts at the proposed disposal site. Suggested minimum weekly analyses should include pH, Eh, conductivity, and selected metals could be analyzed less frequently (for example, at Weeks 0, 1, 2, 4, 8, 12, 16, and 20), especially if changes in leachate chemistry are slow. Whether visible phase separation during storage of the leachates occurs or not, appropriate mixing should be used to ensure the homogeneity of the leachates prior to their use in such analyses.	At the end of weekly cycle the volume of leachate collected is recorded. The leachate is filtered through a Gelman magnetic filter funnel fitted with a membrane filter with pore size of 0.45 microns and analyzed for the parameters listed in Table 2 of the RFP. Filtered leachate samples will be submitted to ALS Environmental/Cantest Ltd. for dissolved metals analysis as requested in Table 4 of the Waste Rock and Lean Ore Geochemical Characterization Plan. Conductivity, Eh, and pH are measured in the CEMI laboratory using standard procedures. An aliquot of filtered leachate is titrated with standardized sulphuric acid to pH 4.5 to calculate total alkalinity. Standardized sodium hydroxide is used to titrate an aliquot of leachate to pH 4.5 and to pH 8.3 to calculate total acidity. Analysis frequency: pH, cond, Eh every cycle; SO4, Cl, F, alkalinity, TIC, acidity cycle 0, 2, 4, 6 etc.; ICP-MS including Hg and Si cycle 0, 4, 8, 12, etc., ICP-ES including Si cycle 2, 6, 10, 14, etc.	
11.7.2	Table 1 is an example of a spreadsheet format used for recording 20 weeks of leachate analytical data.	ASTM	
11.7.3	Fig. 5 is an example of a method used to plot the temporal variation (by week) of leachate pH, sulfate load, and cumulative sulfate load from 21 weeks of accelerated load and release rates).	ASTM	
11.8	Weathered Solid Material Analyses:		

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.8.1	Weigh the humidity cell after collection of the final effluent and completion of a three-day dry-air period.	ASTM	
11.8.2	Transfer the weathered residue and filter media to a clean drying pan, and dry to constant weight at 50 ± 5°C. Record the final weight.	ASTM	
Note 13	Perform any gross sample examination (for example, sample texture and weathering product mineralogic characterization) desired for the weathered residues prior to pulverization. To facilitate such an examination, empty the humidity cell contains into a clean drying pan carefully by pushing gently on the bottom of the perforated plate with a wooden dowel until the sample exits the cell mouth. The perforate plate is accessed through the humidity cell drain port.	NA	
11.8.3	Identify and mark the top versus bottom portions of the sample for gross sampling purposes. Formations of cemented lumps of sample termed “ferricrete” that result from the accelerated weathering process arte common in iron-sulfide-mineral rich samples. Depending on the sample mineralogy, the degree of “ferricrete” cementation may vary vertically within the sample, and the investigator may wish to segregate the sample into upper, middle, and lower thirds to document and characterize such changes.	Procedure to be determined	
11.8.4	After drying to constant weight and prior to splitting, use an instrument such as a rolling pin to break up cemented lumps in the sample (if the cemented lumps cannot be sufficiently reduced to pass through the chutes of a riffle splitter, remove, record, and weigh separately):	ASTM	
11.8.4.1	Split the sample into halves using a riffle splitter with 2.54-cm (1-in.) chutes, and reserve one half to determine the particle size distribution in accordance with Test Method E 276.		Repeat same screen assay method as for pre-test characterization (s.9.5)
11.8.4.2	Split the remaining half sample into two quarters using a riffle splitter with 2.54-cm (1-in.) chutes, and submit one quarter for mineralogical characterization; pulverize the other quarter in either a ring-and-puck or disk-pulverizing machine to 95 % passing a 150-µm (100-mesh) screen in accordance with Test Method E 276.	Procedure to be determined	

Section	ASTM Procedure Description	Description of CEMI Procedure NA – Not applicable to this Project ASTM – ASTM Procedure Followed	CEMI Variance from ASTM
11.8.5	Mix the pulverized residue in a blender or on a rolling cloth. Use the prepared residue for chemical characterization and for comparison with the pre-weathered solid material sample.	Procedure to be determined	

Appendix E

Design of MDNR Reactor

Day, Stephen

From: Kim Lapakko [kim.lapakko@dnr.state.mn.us]
Sent: Tuesday, May 17, 2005 9:55 AM
To: Stephen Day
Cc: Dave Antonson; Jennifer Engstrom; Paul Eger
Subject: RE: Small reactor

Attachments: MN DNR psize methods 050517.doc

MN DNR psize methods 050517.do...

Steve,

Attached is a description of the reactors, masses, and rinse volumes used for various size fractions of Duluth Complex rock in our particle size experiment. As indicated in the attachment, I won't have access to the trace metal data from that experiment until tomorrow. I will need to examine this to help evaluate the expected metal concentrations in drainage relative to detection limits. I'm not sure it will give us as much as hoped because the sulfur contents of the samples typically were on the order of 0.9% to 1.3%. This may make extrapolation by more than an order of magnitude tenuous. It will be another pertinent piece of information.

Kim

>>> "Stephen Day" <sday@srk.com> 5/17/2005 11:18:50 AM >>>
Dave

A design drawing should be fine along with description of the procedure.

The main question is what do you do to scale-up the sample mass as the particle size increases? I want to copy your procedure exactly.

Thanks
Steve.

-----Original Message-----
From: Kim Lapakko [mailto:kim.lapakko@dnr.state.mn.us]
Sent: Tuesday, May 17, 2005 8:38 AM
To: Stephen Day
Cc: Dave Antonson
Subject: Small reactor

Steve,

Dave Antonson will email a figure depicting our small reactor, along with some design details (perforated plate, adehesive, filter). He could also send a reactor. Please contact him directly, with an address to send it, if you think that would be helpful.

Kim

17 May 2005

Steve,

In our particle size tests we used a small reactor and 75-g mass for particle sizes of –270, +270/-100, and +100/-35 mesh. We used the ASTM cell and 1000-g mass for +35/-10, +10/-0.25 inch, and +0.25/-0.75 inch particle sizes. For rinse volumes, we used 200 mL for the 75-g samples and 300 mL for the 1000-g samples. The 300-mL rinse volume was determined as the quantity of water, rounded up to the nearest 100 mL, required to submerge the solids.

I won't have access to the metal release data for the particle size experiment until tomorrow. As mentioned on the phone, sulfate release rates appear to vary linearly with surface area. It seems likely that nickel release rates will vary similarly, and I'll look into this further tomorrow. Hopefully this information will shed some light on the maximum particle size question.

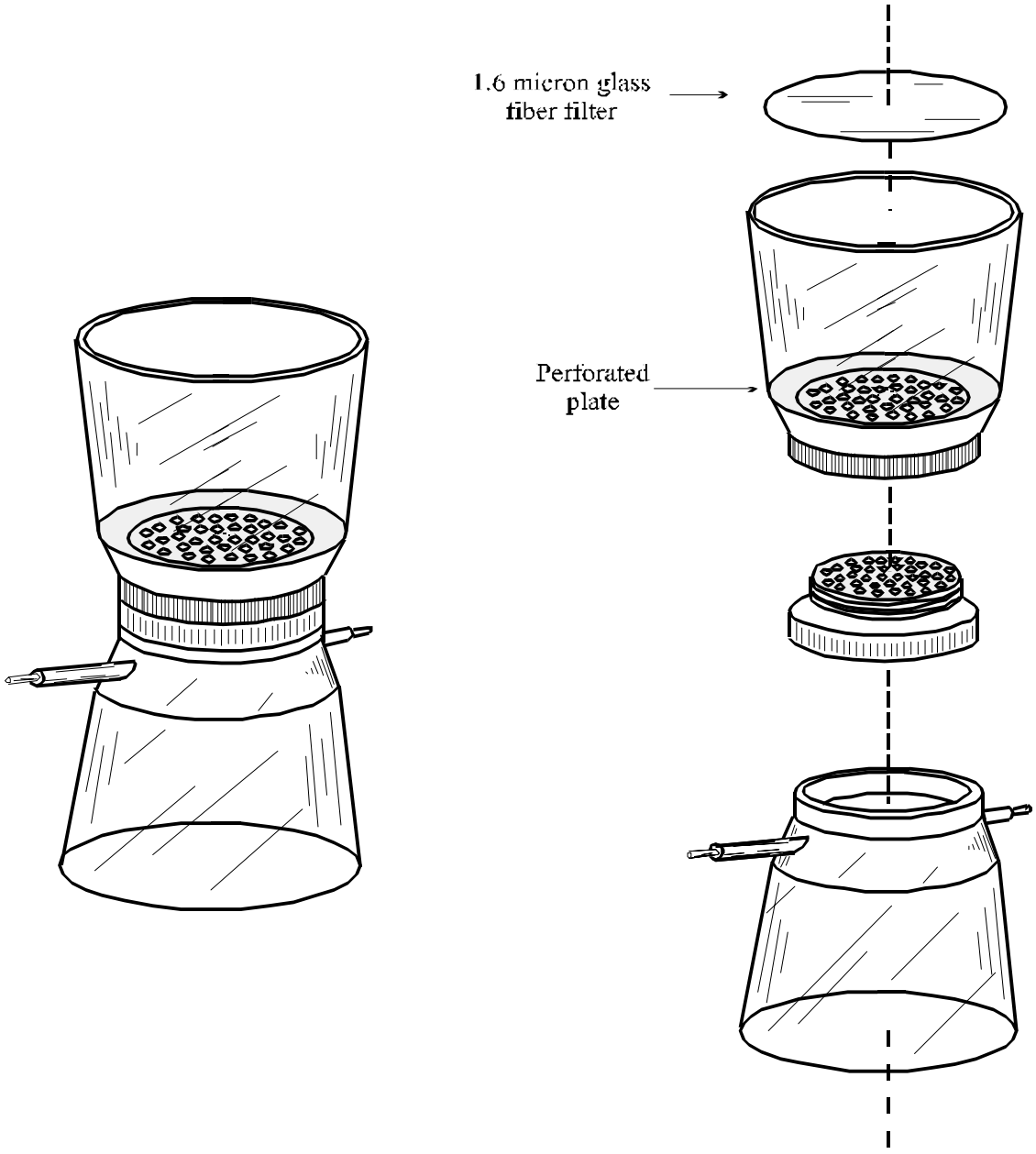
Day, Stephen

From: Dave Antonson [dave.antonson@dnr.state.mn.us]
Sent: Tuesday, May 17, 2005 11:53 AM
To: Kim Lapakko
Subject: reactor

Attachments: small reactor.doc

small reactor.doc
(271 KB)

see if this makes any sense. you can edit it if you want. if it seems adequate you can forward it to steve. maybe he doesn't need a sample of the base.


1.6 micron glass
fiber filter
Perforated
plate

The reactors were purchased from Millipore Corporation (1-800-645-5476). They are 47 mm Sterifil aseptic systems. You will need the 250 ml receiver flask, 250 ml funnel (top), silicone o-rings, and the filter holder base and support screen.

The perforated acrylic plastic base was purchased as flat stock and fabricated to fit the top funnel. The plates are 1/8" thick, 2 1/4" in diameter and tapered to fit into the reactor top. Approximately sixteen 1/16" holes were drilled in the plate. The plate was glued into the reactor using acrylic solvent cement purchased from United States Plastics (1-800-537-9724). Catalog # 44629 for 5 oz. tube. The acrylic flat stock was also purchased from United States Plastics.

After the plate is glued into the top of the reactor there should be approximatly a 3/8" gap between the bottom of the perforated plate and the top of the support screen of the filter unit.

The filter that rests on the perforated plate is a 55 mm Whatman GF/A glass microfibre filter (catalog # 1820 055).

Note: Before adding the solids to the filter you should wet the filter slightly with distilled water so no solids escape around the filter.

Appendix F

Design of Column Testing on Interaction between NorthMet and LTV Tailings

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Memo

To:	Jennifer Engstrom, MDNR	**Date:**	December 8, 2005
cc:	Kim Lapakko, MDNR Paul Eger, MDNR	**From:**	Stephen Day
Subject:	Design of Column Testing on Interaction Between NorthMet and LTV Tailings	**Project #:**	1UP005.001

Jennifer

During our conference call on Oct 25, 2005, I agreed to provide design details for column testing to evaluate the effect of contact of leach waters from the NorthMet Tailings with LTV tailings when we received characterization data for the samples obtained from the LTV impoundment in September. Complete results have now been received.

This memorandum describes:

- Characteristics of LTV tailings.
- Characteristics of available tailings sample material.
- Proposed testwork.

The original proposed program was described in a memorandum dated September 23, 2005. SRK is seeking comments from MDNR on this proposal. It is acknowledged, as we discussed during conference call that this column testwork may represent a preliminary assessment. Additional testing may be required as the design for the tailings basin progresses.

1 Characteristics of LTV Tailings

1.1 Sample Analysis

Seven holes were drilled in the tailings to a depth exceeding 60 feet using a geoprobe. Samples have been analyzed from five holes with tailings having the following textural characteristics:

- GP-1 – Mainly coarse sand.
- GP-2 – Interlayered fine sand and slimes.
- GP-3 – Coarse sand grading into fine sand and slimes.
- GP-4 – Interlayered coarse and fine sands.
- GP-5 – Interlayered fine sand and slimes.

Samples were obtained as core and shipped whole to Canadian Environmental and Metallurgical Inc. At the laboratory, all discrete textural layers were tested for rate of HCl reaction (ie a “fizz” test) and qualitative magnetism as an indicator of magnetite content. Samples were selected from each hole to to represent the surface material (ie potentially weathered) and two samples of each textural type from each hole. These samples were submitted for relative density, moisture content and particle size determinations, quantitative mineralogy by x-ray diffraction and chemical analysis.

1.2 Results

Table 1 shows selected data sorted by the main textural groups (coarse sand, fine sand and slimes).

The dominant mineral in all samples was quartz which varied from 58 to 79% (by weight) but was not different in the three textural groups. Hematite and magnetite were present as expected. Magnetite was lower in the slimes samples likely resulting from density segregation as the tailings were deposited. Carbonates were a significant mineralogical component varying from 5 to 14%. Total carbonate content was greater in the slimes fraction compared to the coarse sands. Ankerite and siderite dominated and occurred in about equal amounts. The calcite content was lower than either ankerite or siderite.

Pyrite was detected in most samples but at very low levels. The sulphur content of the samples varied from 0.02 to 0.04% equivalent to pyrite content of 0.04 to 0.08%.

Silicates occurring in all samples were hydrobioitite, kaolinite, amphibole (cummingtonite ± grunerite), diopside, ferripyrophyllite (possibly minnesotaite) and albite. Other minerals occurring in a few samples were pyrophyllite, muscovite and hydroxylapatite. There was no evidence that the mineral distribution was related to particle size.

Distribution of metals was also unrelated to particle size with the possible exception of manganese which appeared to be elevated in the slimes. This is consistent with the higher carbonate content and indicates that manganese is associated with the carbonates.

2 Characteristics of PolyMet Tailings

Four tailings samples have been tested. These samples originated from processing of three ore composites nominally containing 0.35%, 0.4% and 0.45% copper (Parcels 1, 2 and 3, respectively). Flotation testing also considered addition of copper sulphate as an activator of pyrite. For Parcel 1 and 3, copper sulphate was evaluated. Also for Parcel 1, and Parcel 2 processing without the addition of copper sulphate was evaluated. The addition of copper sulphate does not materially affect the copper and sulphur content of the tailings directly but changes the mineralogical composition due to the selective recovery of pyrite to the bulk sulfide concentrate.

The characteristics of the four samples are shown in Table 2. Very little difference between the tailings was apparent. As expected, addition of copper sulphate in the process resulted in lower sulphur content in the tailings (0.1% and 0.15%, Parcels 1 and 3, respectively) compared to 0.23% and 0.2% (Parcels 1 and 2, respectively) without copper sulphate. Copper showed the widest variation in concentration (223 to 527 mg/kg). Variations of cobalt, nickel and zinc were small.

Table 1. Characteristics of LTV Tailings Samples

Core	Sampled		Material	Q	Py	Cal	Ank	Sid	Hem	Mag	Bio	Kao	Fe-Pyr	Albite low	As	Cd	Co	Cr	Cu	Fe	Mn	Ni	P	Pb	S	Zn
	Start ft	Finish ft		%	%	%	%	%	%	%	%	%	%	%	%	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	ppm
Coarse Sand																										
GP-1	8	12	Coarse Sand	72	0.2	0.3	3	3	3	4	1	1	2	2	31	-0.5	12	100	14	15.85	5970	5	240	7	0.02	14
GP-1	20	40	Coarse Sand	71	0.2	0.2	5	8	2	2	2	1	2	2	36	-0.5	14	90	25	15.5	7110	8	250	5	0.04	13
GP-3	8	12	Coarse Sand	58	0	0.4	6	6	2	3	5	1	6	4	16	-0.5	7	42	7	13.7	3420	3	250	4	0.02	9
GP-4	4	16	Coarse Sand	59	0.1	0.7	2	4	2	3	10	2	2	3	19	-0.5	7	57	9	15.45	3890	1	240	7	0.02	9
GP-4	20	24	Coarse Sand	79	0	0.1	3	2	2	3	3	1	2	1	21	-0.5	10	45	8	12.85	4010	3	250	5	0.02	34
n				5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Min				58	0	0.1	2	2	2	2	1	1	2	1	16	-0.5	7	42	7	12.85	3420	1	240	4	0.02	9
Median				71	0.1	0.3	3	4	2	3	3	1	2	2	21	-0.5	10	57	9	15.45	4010	3	250	5	0.02	13
Max				79	0.2	0.7	6	8	3	4	10	2	6	4	36	-0.5	14	100	25	15.85	7110	8	250	7	0.04	34
Fine Sand																										
GP-1	60	72	Fine Sand	72	0.1	0.7	3	2	1	1	2	2	2	4	22	-0.5	11	77	20	12.05	7010	3	330	3	0.04	14
GP-2	0	1	Fine Sand	60	0	0.2	6	7	2	2	11	4	3	0	15	-0.5	9	27	7	14.4	4270	-1	490	-2	0.02	10
GP-2	24	28	Fine Sand	62	0.4	1	6	3	1	2	5	2	3	3	22	-0.5	11	61	13	14.55	5340	3	550	7	0.02	14
GP-3	44	60	Fine Sand	68	0.2	0.5	4	3	3	2	4	1	3	1	15	-0.5	12	45	17	13.4	8510	4	400	7	0.03	12
GP-4	18	20	Fine Sand	73	0	0.2	7	3	2	2	4	2	2	1	43	-0.5	10	41	12	13.3	4020	2	290	8	0.02	13
GP-5	8	20	Fine Sand	78	0.1	0.1	4	2	3	3	3	1	2	1	14	-0.5	7	42	7	13.45	3630	2	270	2	0.02	7
GP-5	36	48	Fine Sand	73	0.4	0.4	4	2	2	3	2	1	3	2	14	-0.5	9	53	14	13.55	5820	2	290	4	0.02	8
n				7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7
Min				60	0	0.1	3	2	1	1	2	1	2	0	14	-0.5	7	27	7	12.05	3630	-1	270	-2	0.02	7
Median				72	0.1	0.4	4	3	2	2	4	2	3	1	15	-0.5	10	45	13	13.45	5340	2	330	4	0.02	12
Max				78	0.4	1	7	7	3	3	11	4	3	4	43	-0.5	12	77	20	14.55	8510	4	550	8	0.04	14
Slimes																										
GP-2	28	32	Slimes	62	0	1	4	4	2	1	5	2	3	5	18	-0.5	14	31	10	14.2	7390	3	530	4	0.02	13
GP-3	60	72	Slimes	62	0	0.7	8	5	3	1	7	2	2	1	15	-0.5	14	33	20	12	10050	4	590	2	0.04	14
GP-5	20	24	Slimes	70	0.1	0.3	6	4	3	2	6	2	3	1	25	-0.5	9	38	7	13.75	4830	1	460	4	0.02	8
GP-5	48	52	Slimes	72	0	0.6	4	6	2	1	4	1	2	1	16	-0.5	16	40	19	13.1	12400	3	550	4	0.03	11
n				4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Min				62	0	0.3	4	4	2	1	4	1	2	1	15	-0.5	9	31	7	12	4830	1	460	2	0.02	8
Median				66	0	0.65	5	4.5	2.5	1	5.5	2	2.5	1	17	-0.5	14	35.5	14.5	13.425	8720	3	540	4	0.025	12
Max				72	0.1	1	8	6	3	2	7	2	3	5	25	-0.5	16	40	20	14.2	12400	4	590	4	0.04	14

Notes
Minerals Q = Quartz, Ank = Ankerite, Hem = Hematite, Mag = Magnetite, Sid = siderite, Bio = hydrobiotite, kao = kaolinite, Fe-Pyr = Ferriprophyllite, Cal = Calcite, py =pyrite

Table 2. Characteristics of NorthMet Tailings Samples

Sample Number	Source and Process	Total S %	As ppm	Cd ppm	Co ppm	Cr ppm	Cu ppm	Mn ppm	Ni ppm	P ppm	Zn ppm
PIS-ACS TO DSC COMP	Parcel 1, $CuSO_4$ Added	0.10	<5	<0.5	59	186	223	1130	319	790	95
PISA TO D COMP	Parcel 1, $CuSO_4$ Not Added	0.23	<5	<0.5	62	188	248	1140	329	800	97
P2SA TO D COMP	Parcel 2, $CuSO_4$ Not Added	0.20	<5	<0.5	61	199	527	1125	385	750	93
P3S-A TO D COMP	Parcel 3, $CuSO_4$ Added	0.15	<5	<0.5	60	175	418	1110	372	770	92

3 Proposed Column Testwork

3.1 Design Basis

Based on the characteristics of the NorthMet and LTV tailings, the following chemical processes can be expected to occur within the layered tailings basins:

- In NorthMet Tailings
  - Near surface oxidation of residual sulfide minerals resulting in release acidity, iron, sulphate and trace elements (copper and nickel).
  - Development and migration of an oxidation front due to consumption of oxygen near the surface.
  - Attenuation of metals as a result of interaction between pore fluids and mineral grains.
- LTV Tailings
  - Enhanced dissolution of ankerite and siderite under saturated conditions resulting in release of calcium, magnesium, ferrous iron, reduced manganese and bicarbonate alkalinity.
  - Localized re-precipitation of ferric hydroxides and manganese oxides due to variations in pH and oxidation-reduction potential.
- Interaction Between NorthMet Tailings Pore Water and LTV Tailings
  - Possible sorption of metals by ferric hydroxides and manganese oxides, particularly in the immediate contact zone where LTV tailing are probably partially oxidized.
  - Precipitation of metal carbonates due to alkaline conditions.

Testing of the LTV tailings indicates little significant variation in mineralogical and chemical content. The primary variable expected to influence the degree to metal attenuation occurs, if at all, is particle size which will control the availability of adsorption sites, oxidation-reduction conditions and contact time.

3.2 Proposed Testwork

The proposed leach column design includes the following main features (Figure 1):

- In series leach columns designed to generate NorthMet tailings pore water as feed into LTV tailings.

- First column containing 10 kg of unsaturated NorthMet Tailings and second receiving column containing 5 kg of LTV tailings. The larger volume of NorthMet tailings is intended to optimize development of near equilibrium pore water chemistry below an oxidation zone in the NorthMet tailings. The smaller volume of LTV tailings may allow breakthrough of NorthMet tailings water chemistry to be observed.
- NorthMet tailings open to atmosphere.
- Connecting pipe between columns and sampling "T" operated to exclude oxygen.
- Side sampling port in LTV tailings column to enable sampling of pore water just below the entry point.
- Sampling of final effluent.
- Application of 2 L of deionized water every week to allow withdrawal of up to 250 mL of water from each of the two intermediate location and 1.5 L of the final effluent. This application rate represents approximately one pore volume every 4 weeks.
- Analysis of intermediate sampling points for pH and Eh every week and composite sample for anions and cations every other week (including sulfur).
- Analysis of final effluent for same parameters, sulphate, alkalinity and anion scan.

Approximately 11 kg of each of the tailings samples shown in Table 2 are available for additional testing. The following matrix summarises six proposed tests (Table 3). It is preferred to test slimes and coarse sands as two extreme characteristics of LTV tailings. From experience, it is unlikely that the slimes will transmit sufficient water for the experiment. Therefore, a fine sand composite would be used instead.

Each "X" in Table 3 represents an in-series column pair. The two control experiments will operate without NortMet tailings in the first column to evaluate leachate chemistry from LTV tailings. This will allow comparison with seepage chemistry in the existing basin.

Table 3. Matrix of Proposed Tests

		NorthMet Tailings Samples		
		Control No NorthMet Tailings	**Lower S Composite (Parcels 1 and 3, $CuSO_4$ Added)**	**Higher S Composite (Parcels 1 and 2, $CuSO_4$ added)**
LTV Samples	Fine sand or Slimes Composite	X	X	X
	Sand Composite	X	X	X

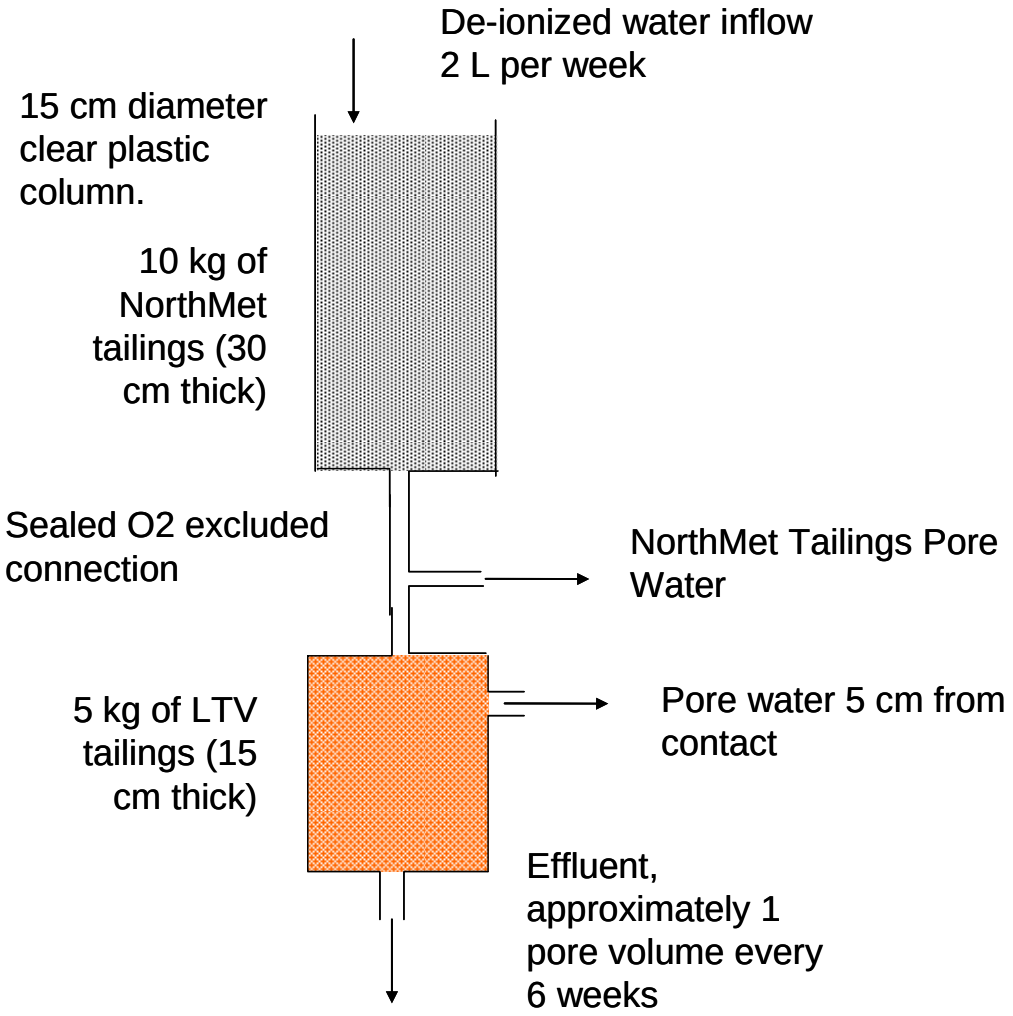


Figure 1. Schematic of In-Series Column Design

Appendix B

Leachate Extraction Test Results

			Residues							
Procedure	**Parameter**	**Unit**	**Leach, no CuSO4**	**Leach, with CuSO4**	**Gypsum**	**Raffinate Neutralization**	**Fe/Al**	**Mg**	**Combined**	**Combined no Gypsum**
SPLP Leachate Extraction Results	**pH**	**-**	2.79	3.38	4.12	4.91	5.03	9.72	9.18	9.42
	Hardness	**mgCaCO3/L**	1520	1480	1360	1290	1460	1880	1480	1650
	Al	**mg/L**	9.24	0.27	0.27	0.3	0.21	<0.005	<0.005	<0.005
	Sb	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	As	**mg/L**	<0.001	<0.001	0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Ba	**mg/L**	<0.001	<0.001	<0.001	0.009	<0.001	0.003	<0.001	<0.001
	Be	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Bi	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	B	**mg/L**	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
	Cd	**mg/L**	0.022	0.0058	<0.0002	0.0017	0.012	<0.0002	<0.0002	<0.0002
	Ca	**mg/L**	487	492	541	511	500	476	519	499
	Cr	**mg/L**	0.1	0.006	0.005	0.003	<0.001	<0.001	0.002	<0.001
	Co	**mg/L**	4.19	0.21	0.14	0.33	3.75	<0.001	<0.001	<0.001
	Cu	**mg/L**	241	6.67	5.38	0.5	13.9	0.006	0.009	0.005
	Fe	**mg/L**	59.6	0.26	0.5	0.69	<0.05	<0.05	<0.05	<0.05
	Pb	**mg/L**	0.004	0.002	0.009	0.093	0.001	<0.001	<0.001	<0.001
	Li	**mg/L**	0.023	<0.001	<0.001	<0.001	0.015	0.008	<0.001	<0.001
	Mg	**mg/L**	74.1	61.1	1.8	2.85	50.7	168	44.7	97
	Mn	**mg/L**	1.29	0.055	0.026	0.046	0.82	<0.001	<0.001	<0.001
	Hg	**ug/L**	0.13	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
	Mo	**mg/L**	<0.0005	<0.0005	0.0005	<0.0005	0.0014	<0.0005	0.0074	0.013
	Ni	**mg/L**	103	3.95	3.09	6.84	79	0.004	0.015	0.008
	PO4	**mg P/L**	<0.15	<0.15	<0.15	<0.15	<0.15	<0.15	<0.15	<0.15
	K	**mg/L**	<0.1	1.4	<0.1	<0.1	0.3	2.6	0.4	0.7
	Se	**mg/L**	0.005	0.003	0.002	<0.001	0.002	0.003	0.01	0.024
	Si	**mg SiO2/L**	0.8	0.8	<0.25	0.5	1.4	<0.25	15.8	10.5
	Ag	**mg/L**	0.0064	0.0077	<0.00025	<0.00025	0.0012	<0.00025	<0.00025	<0.00025
	Na	**mg/L**	47.2	147	2.8	4.63	41.9	238	51.4	97.6
	Sr	**mg/L**	0.26	0.32	0.16	0.3	0.19	0.24	0.18	0.26
	Te	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Tl	**mg/L**	0.0006	<0.0001	0.0002	0.0001	0.0012	<0.0001	<0.0001	<0.0001
	Th	**mg/L**	0.033	0.01	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
	Sn	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Ti	**mg/L**	0.044	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	U	**mg/L**	0.0017	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
	V	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Zn	**mg/L**	5.85	0.17	0.095	0.24	4.4	<0.005	<0.005	<0.005
	Zr	**mg/L**	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01

Notes
#N/A - result checking in progress

			Residues							
Procedure	**Parameter**	**Unit**	**Leach, no CuSO4**	**Leach, with CuSO4**	**Gypsum**	**Raffinate Neutralization**	**Fe/Al**	**Mg**	**Combined**	**Combined no Gypsum**
TCLP Leachate Extraction Results	**pH**	**-**	4.74	4.94	4.95	4.96	4.98	9.8	5.23	8.98
	Hardness	**mgCaCO3/L**	1500	1260	1490	1900	1610	3270	2400	3710
	Al	**mg/L**	2.6	0.008	0.22	0.28	1.51	<0.005	0.088	<0.005
	Sb	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	As	**mg/L**	<0.001	<0.001	0.001	0.001	<0.001	<0.001	<0.001	<0.001
	Ba	**mg/L**	<0.001	0.001	0.001	0.014	0.001	0.004	0.007	0.003
	Be	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Bi	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	B	**mg/L**	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
	Cd	**mg/L**	0.021	0.0055	<0.0002	0.0018	0.011	<0.0002	0.0025	<0.0002
	Ca	**mg/L**	451	402	594	754	562	586	733	718
	Cr	**mg/L**	0.031	0.002	0.006	0.007	0.13	0.002	0.022	0.004
	Co	**mg/L**	4.48	0.21	0.14	0.32	3.37	0.001	0.096	0.001
	Cu	**mg/L**	244	5.75	5.28	0.46	46.8	0.018	2.41	0.005
	Fe	**mg/L**	3.9	<0.05	0.42	0.55	0.06	<0.05	0.14	<0.05
	Pb	**mg/L**	0.001	<0.001	0.012	0.34	<0.001	<0.001	0.004	<0.001
	Li	**mg/L**	0.025	0.002	0.001	0.002	0.014	0.008	<0.001	0.002
	Mg	**mg/L**	91	62	2	2.74	50	438	137	464
	Mn	**mg/L**	1.39	0.057	0.029	0.046	0.76	<0.001	0.11	<0.001
	Hg	**ug/L**	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
	Mo	**mg/L**	<0.0005	<0.0005	0.0007	0.0006	0.0017	<0.0005	0.0006	0.027
	Ni	**mg/L**	107	4	3.15	6.35	73.4	0.018	3.02	0.093
	P	**mg PO4/L**	<0.15	<0.15	<0.15	1	<0.15	<0.15	<0.15	<0.15
	K	**mg/L**	0.6	2.2	0.5	0.6	0.7	3	1.2	1.7
	Se	**mg/L**	0.004	0.003	0.002	0.002	0.002	0.002	0.002	0.033
	Si	**mg SIO2/L**	3.5	4	1.1	1.5	2.7	0.8	14.9	14.1
	Ag	**mg/L**	0.0074	0.0093	<0.00025	<0.00025	0.0015	<0.00025	<0.00025	<0.00025
	Na	**mg/L**	1730	1680	1560	1600	1540	1710	1590	1710
	Sr	**mg/L**	0.29	0.35	0.22	0.46	0.29	0.41	0.34	0.47
	Te	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Tl	**mg/L**	0.0009	<0.0001	0.0002	0.0003	0.0016	<0.0001	0.0003	0.0001
	Th	**mg/L**	0.002	0.0013	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
	Sn	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	0.001	<0.001
	Ti	**mg/L**	0.009	<0.001	<0.001	<0.001	<0.001	<0.001	0.001	<0.001
	U	**mg/L**	0.0016	<0.0005	<0.0005	<0.0005	0.0069	<0.0005	0.0008	<0.0005
	V	**mg/L**	0.003	0.003	0.003	0.003	0.001	0.003	0.002	0.003
	Zn	**mg/L**	5.49	0.18	0.12	0.25	4.12	<0.005	0.22	<0.005
	Zr	**mg/L**	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01

Notes
#N/A - result checking in progress

			Residues							
Procedure	**Parameter**	**Unit**	**Leach, no CuSO4**	**Leach, with CuSO4**	**Gypsum**	**Raffinate Neutralization**	**Fe/Al**	**Mg**	**Combined**	**Combined no Gypsum**
3:1 Water Leach (First Week)	**pH**	**-**	2.12	2.64	3.30	4.22	5.06	9.33	8.65	8.95
	ORP	**mV**	625	601	568	530	490	277	390	262
	Cond.	**µS/cm**	12500	9720	13	2490	6290	14530	4480	7690
	Acidity to pH 4.5	**mg CaCO3/L**	1807.5	252.0	38.0	2.25	#N/A	#N/A	#N/A	#N/A
	Total Acidity	**mg CaCO3/L**	4362.5	365.0	119.0	58.5	400.0	#N/A	#N/A	#N/A
	Alkalinity	**mg CaCO3/L**	#N/A	#N/A	#N/A	#N/A	5.0	52.0	39.5	46.0
	Inorganic C	**mg CaCO3/L**	<1	<1	<1	<1	<1	7	9	8
	Hardness	**mg CaCO3/L**	2640	2810	1360	1300	3000	6020	2000	3730
	F	**mg/L**	<1	<0.5	<0.25	3.6	1.6	<1	<5	<0.5
	Cl	**mg/L**	717	717	21.7	23.5	473	1950	392	635
	SO4	**mg/L**	9485	4507	1600	1670	4490	7940	2430	4280
	Al	**mg/L**	47.9	1.34	1.35	1.69	1.06	<0.005	<0.005	<0.005
	Sb	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	As	**mg/L**	0.012	<0.001	0.004	0.003	0.002	<0.001	<0.001	<0.001
	Ba	**mg/L**	<0.001	<0.001	0.006	0.012	0.002	0.005	0.001	0.002
	Be	**mg/L**	0.003	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Bi	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	B	**mg/L**	0.07	<0.05	<0.05	<0.05	0.11	<0.05	<0.05	<0.05
	Cd	**mg/L**	0.102	0.031	0.0035	0.0082	0.064	<0.0002	<0.0002	<0.0002
	Ca	**mg/L**	322	434	522	494	463	458	450	453
	Cr	**mg/L**	0.54	0.03	0.02	0.013	0.001	0.001	0.006	0.002
	Co	**mg/L**	30.2	0.99	0.79	1.8	29.1	0.001	<0.001	<0.001
	Cu	**mg/L**	1300	51.1	31.4	2.81	52	0.025	0.002	0.006
	Fe	**mg/L**	511	12.6	2.72	3.94	0.12	<0.05	<0.05	<0.05
	Pb	**mg/L**	0.009	0.006	0.018	0.3	0.002	<0.001	<0.001	<0.001
	Li	**mg/L**	0.14	0.005	0.003	0.005	0.096	0.048	<0.001	0.001
	Mg	**mg/L**	446	419	10.1	15.9	446	955	213	630
	Mn	**mg/L**	6.69	0.28	0.13	0.28	4.83	<0.001	<0.001	<0.001
	Hg	**mg/L**	1.18	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
	Mo	**mg/L**	<0.0005	<0.0005	0.0013	<0.0005	0.0023	<0.0005	0.027	0.03
	Ni	**mg/L**	616	19	15.7	37.9	616	0.034	0.028	0.054
	P	**mg/L**	0.7	<0.15	<0.15	<0.15	<0.15	<0.15	<0.15	<0.15
	K	**mg/L**	0.4	7.6	0.1	0.5	2.2	20.3	1.8	3.9
	Se	**mg/L**	0.01	0.003	<0.001	0.001	0.003	<0.001	0.011	0.021
	Si	**mg/L**	3.3	3.2	1.2	2.7	4.5	<0.25	14.8	5.7
	Ag	**mg/L**	0.022	0.025	0.0003	<0.00025	0.013	<0.00025	<0.00025	<0.00025
	Na	**mg/L**	290	616	16	20.9	284	2020	255	717
	Sr	**mg/L**	0.17	0.38	0.2	0.46	0.22	0.3	0.2	0.27
	Te	**mg/L**	<0.001	<0.001	0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Tl	**mg/L**	0.0029	0.0005	0.0006	0.0005	0.0049	0.0004	<0.0001	0.0001
	Th	**mg/L**	0.145	0.109	0.0007	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
	Sn	**mg/L**	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Ti	**mg/L**	0.42	0.005	0.002	0.001	0.001	<0.001	<0.001	<0.001
	U	**mg/L**	0.0081	<0.0005	<0.0005	0.0009	0.001	<0.0005	<0.0005	<0.0005
	V	**mg/L**	0.12	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
	Zn	**mg/L**	48.6	0.92	0.15	1.34	34.7	<0.005	<0.005	<0.005
	Zr	**mg/L**	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01

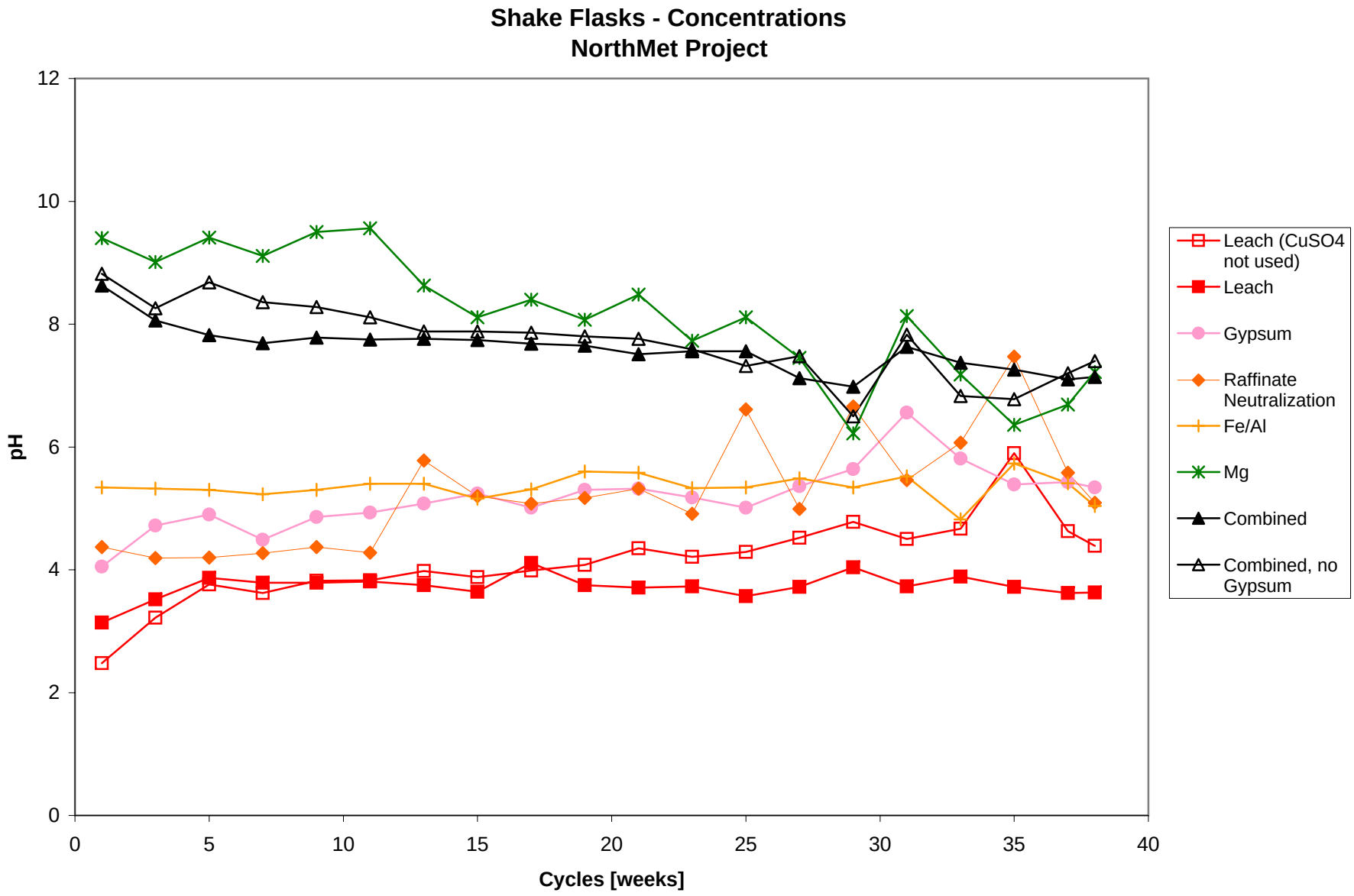
Notes
#N/A - result checking in progress

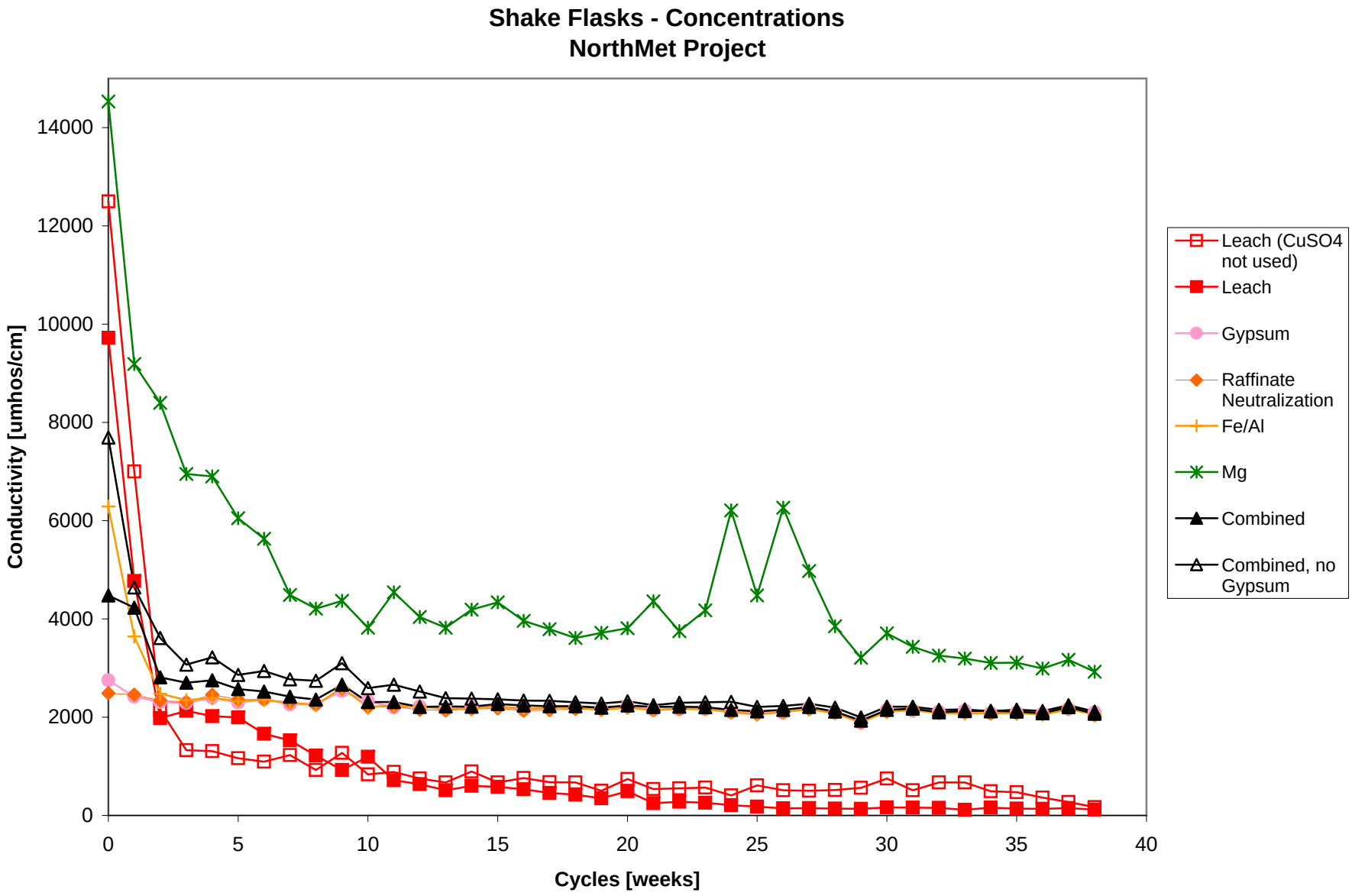
Appendix C

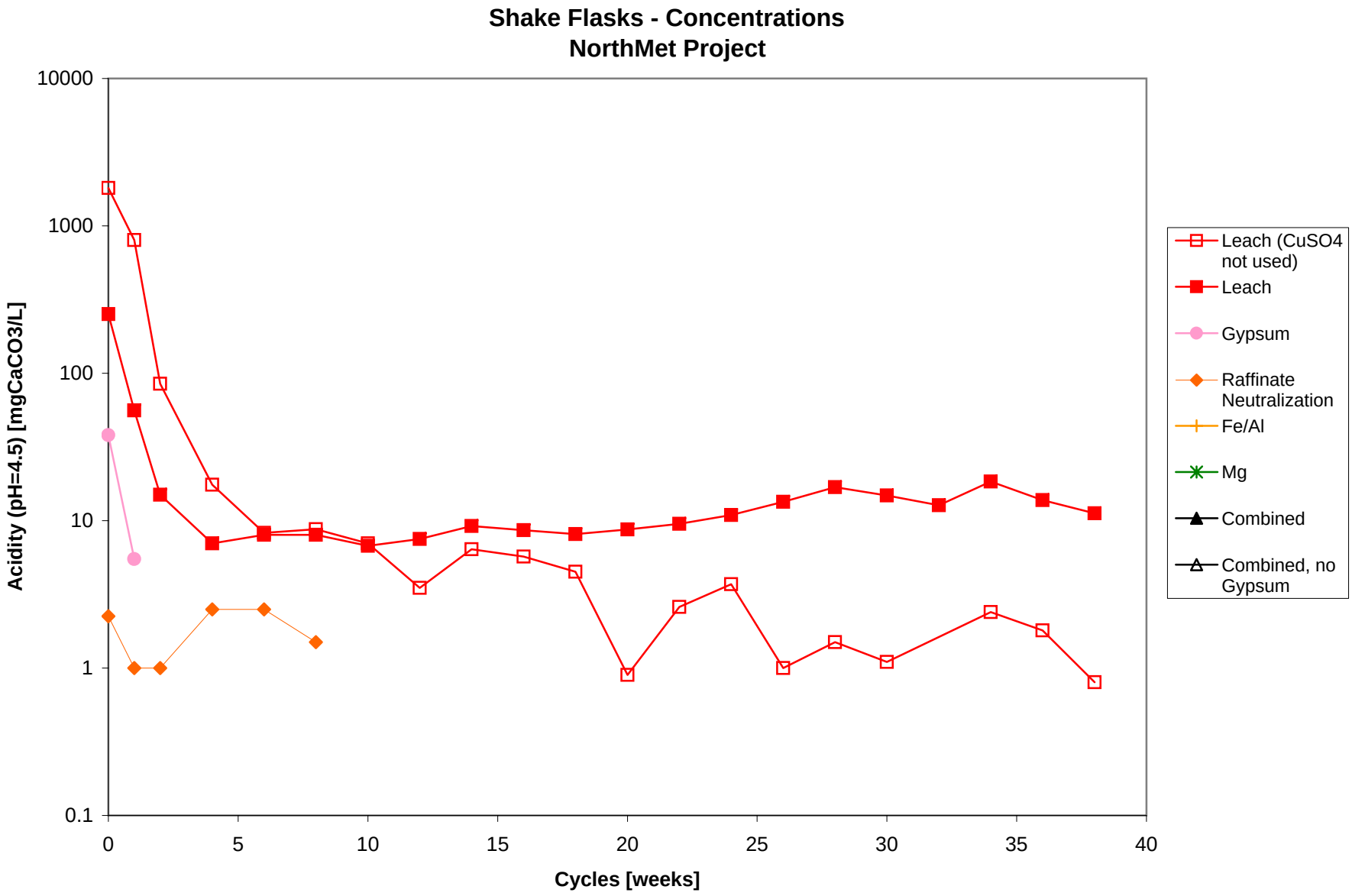
Leachate Chemistry Graphs

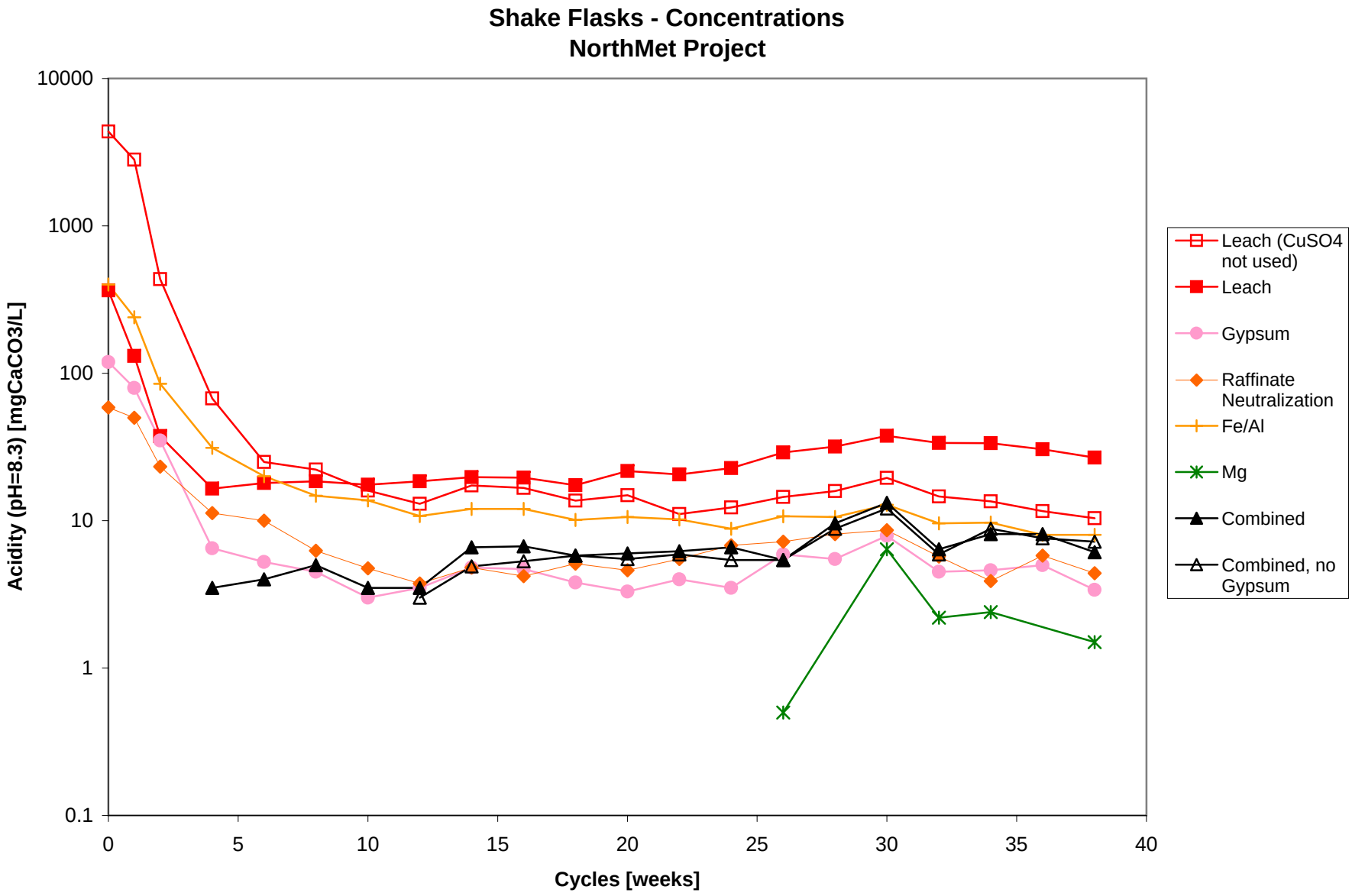
Appendix C.1

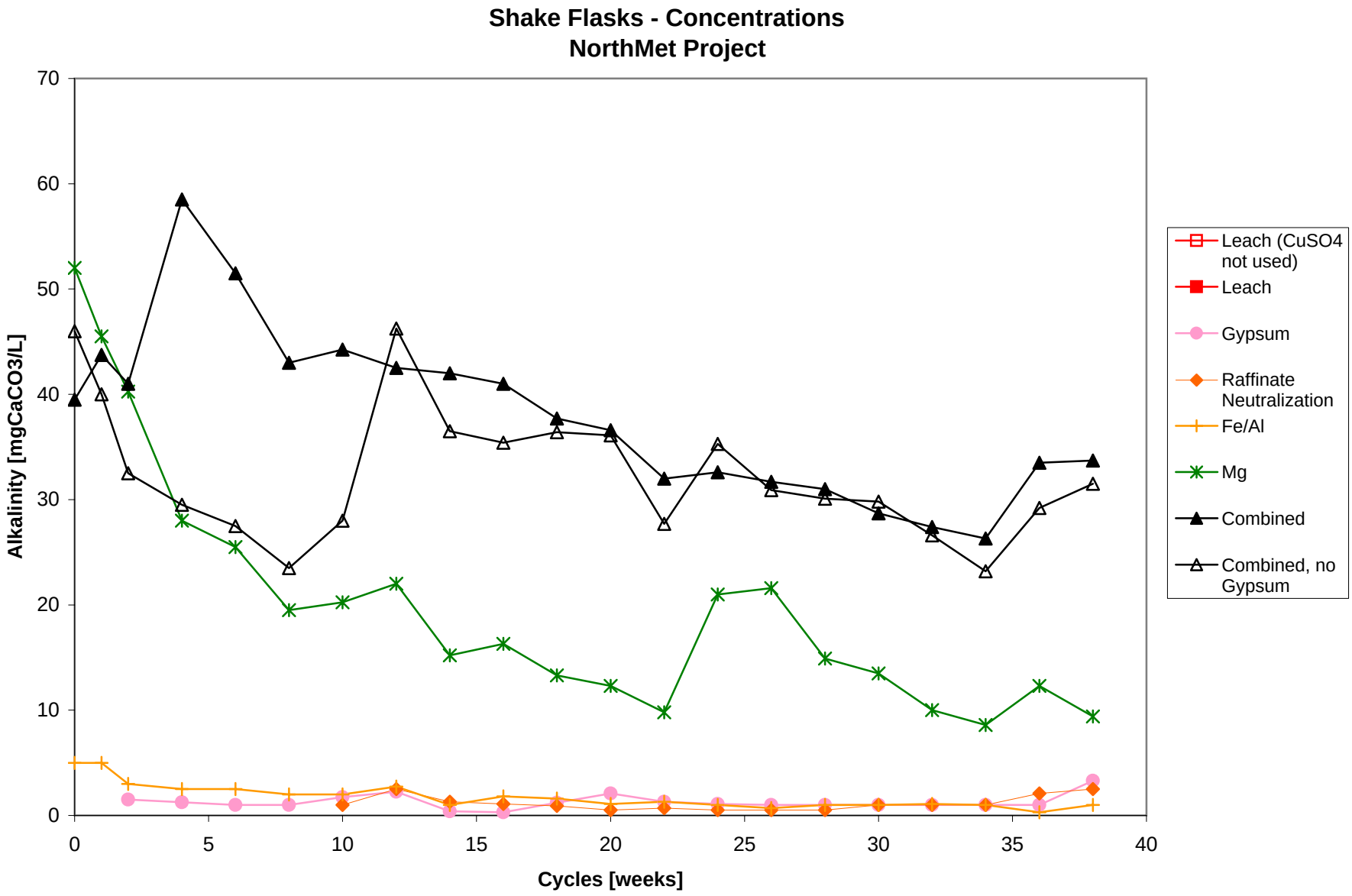
Sequential Shake Flasks

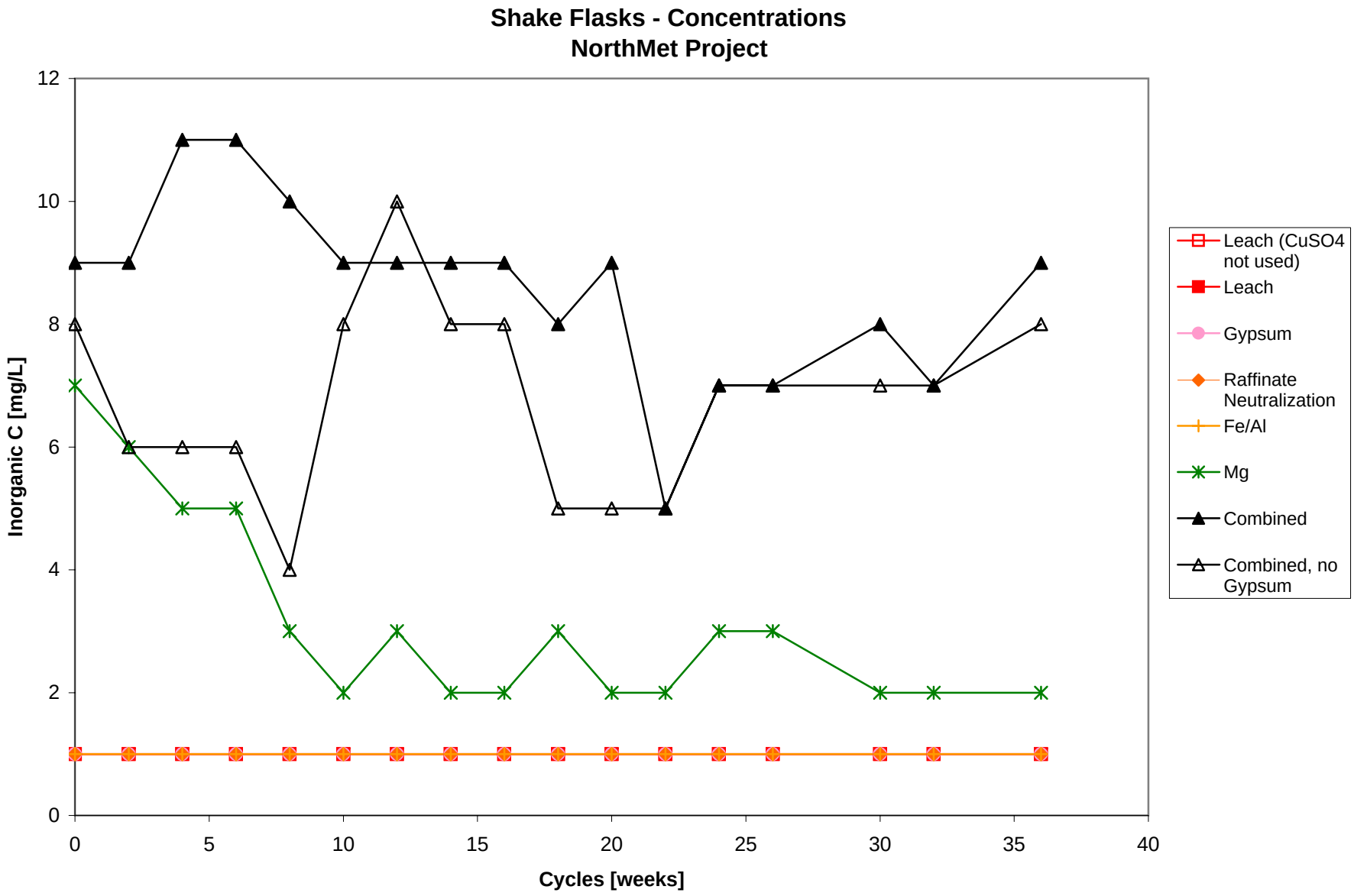

Shake Flasks - Concentrations
NorthMet Project
pH
Cycles [weeks]
0
2
4
6
8
10
12
0
5
10
15
20
25
30
35
40
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

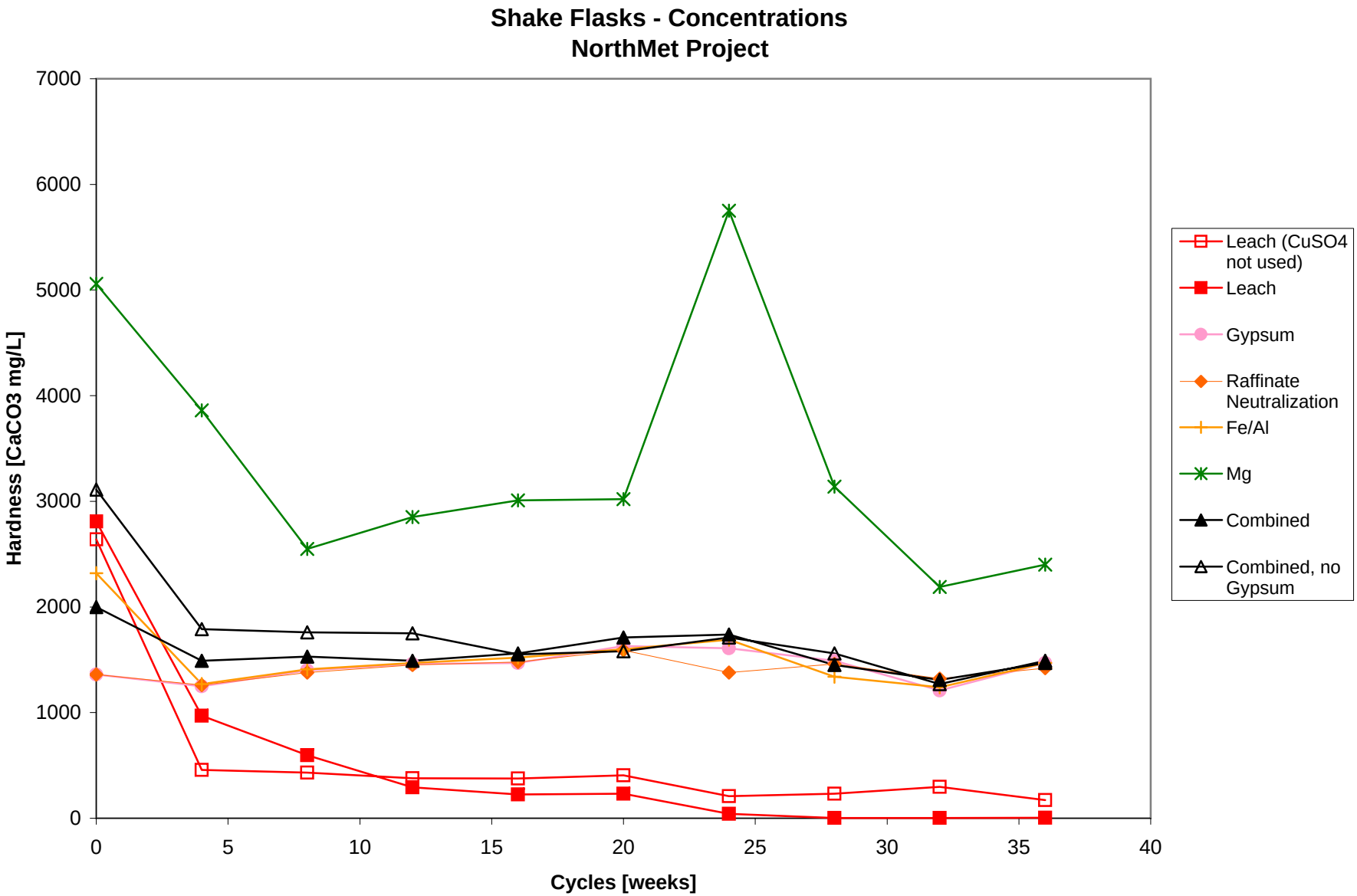

Shake Flasks - Concentrations
NorthMet Project
Conductivity [umhos/cm]
0
2000
4000
6000
8000
10000
12000
14000
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

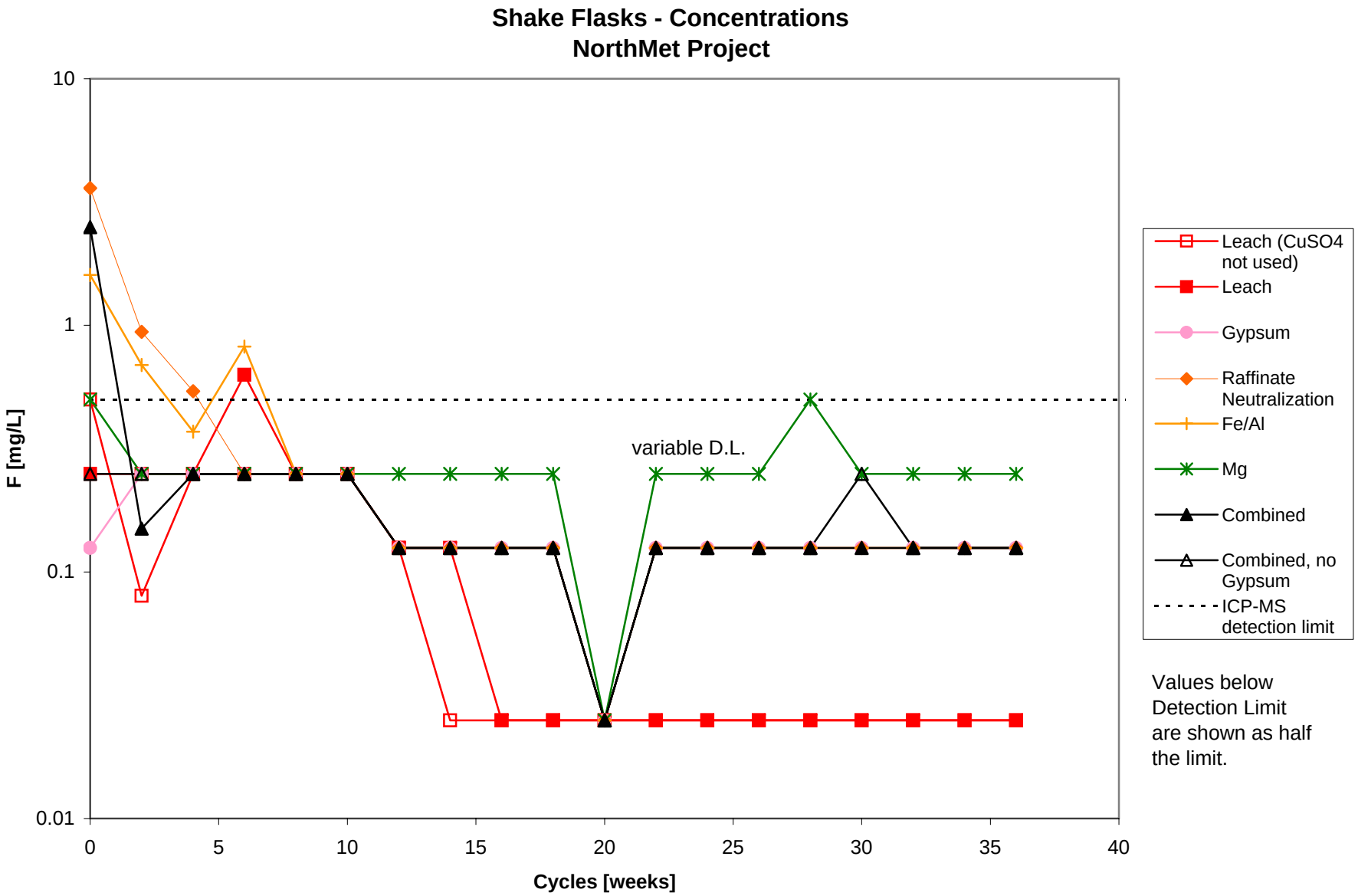

Shake Flasks - Concentrations
NorthMet Project
Acidity (pH=4.5) [mgCaCO3/L]
10000
1000
100
10
1
0.1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

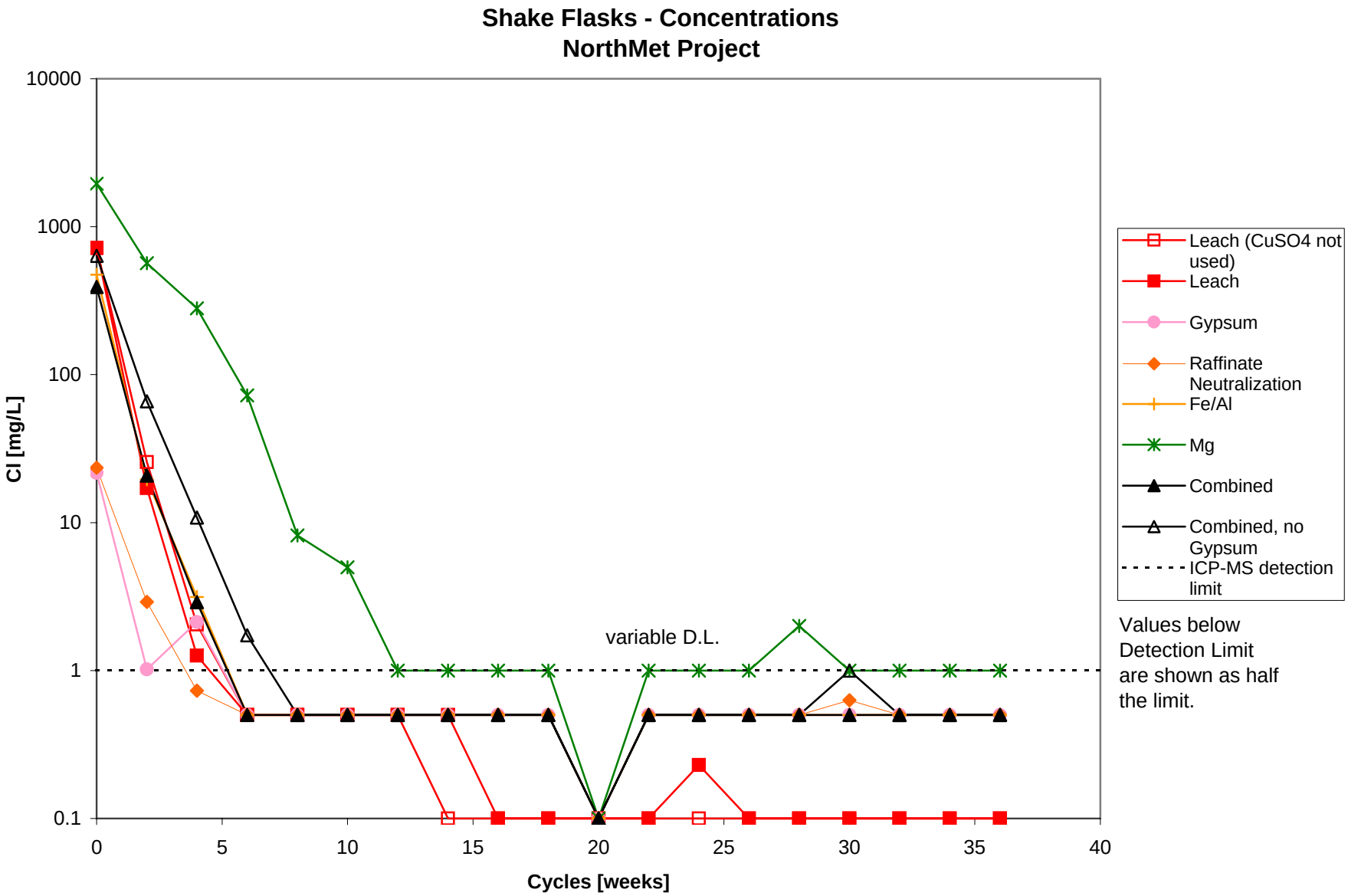

Shake Flasks - Concentrations
NorthMet Project
Acidity (pH=8.3) [mgCaCO3/L]
10000
1000
100
10
1
0.1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

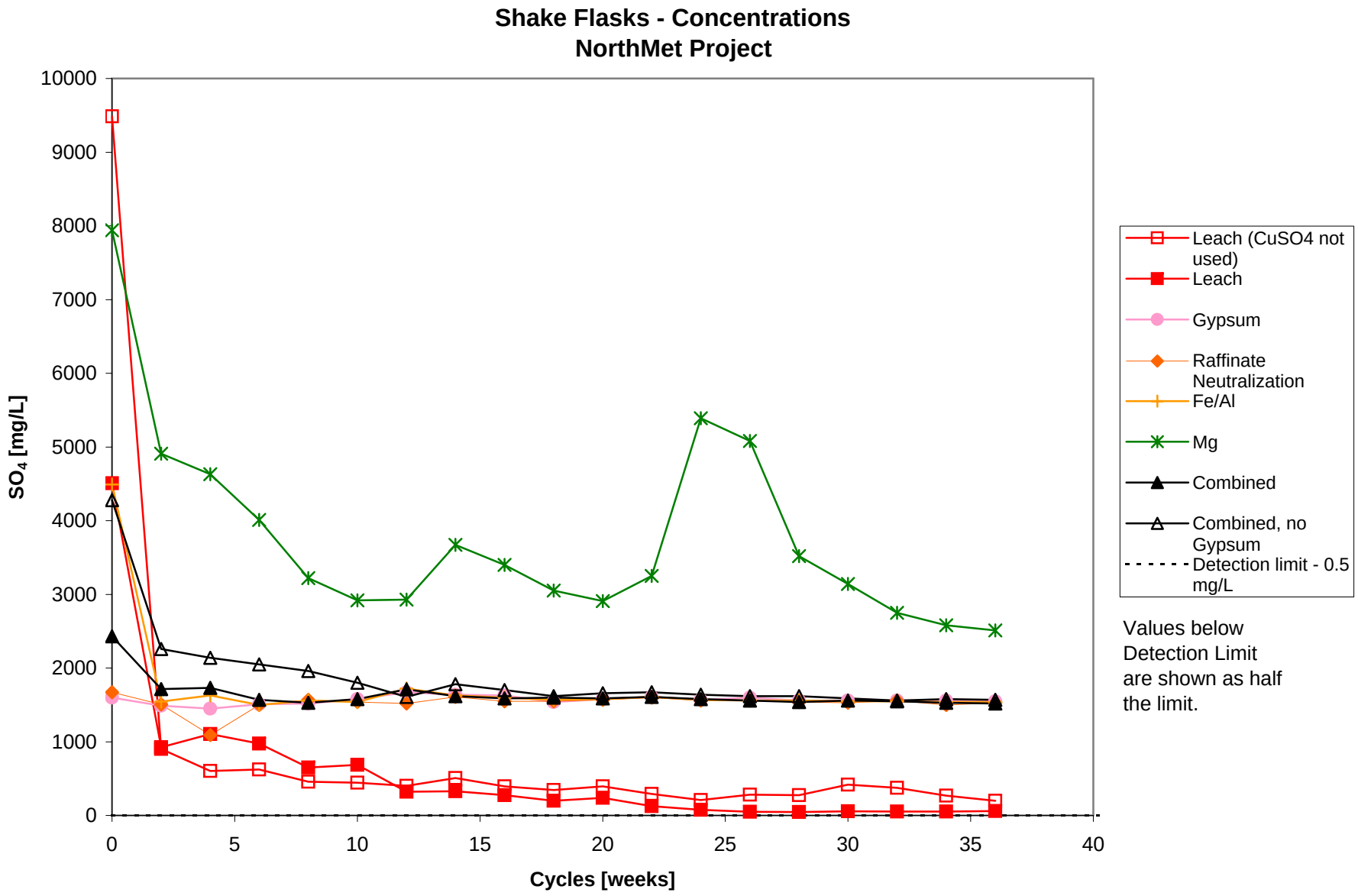

Shake Flasks - Concentrations
NorthMet Project
Alkalinity [mgCaCO3/L]
Cycles [weeks]
0
10
20
30
40
50
60
70
0
5
10
15
20
25
30
35
40
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

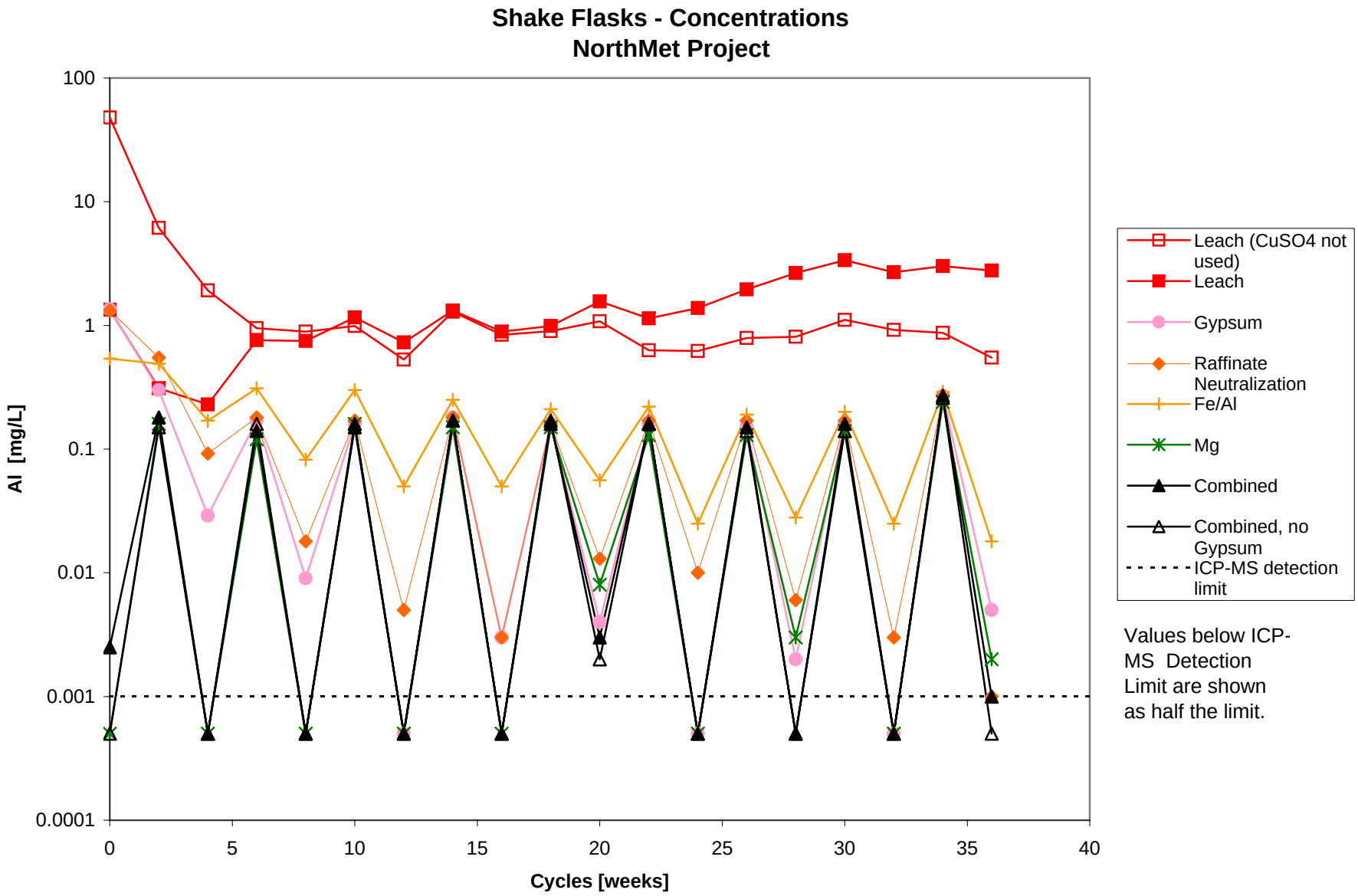

Shake Flasks - Concentrations
NorthMet Project
Inorganic C [mg/L]
Cycles [weeks]
0
2
4
6
8
10
12
0
5
10
15
20
25
30
35
40
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum


Shake Flasks - Concentrations
NorthMet Project
Hardness [CaCO3 mg/L]
7000
6000
5000
4000
3000
2000
1000
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

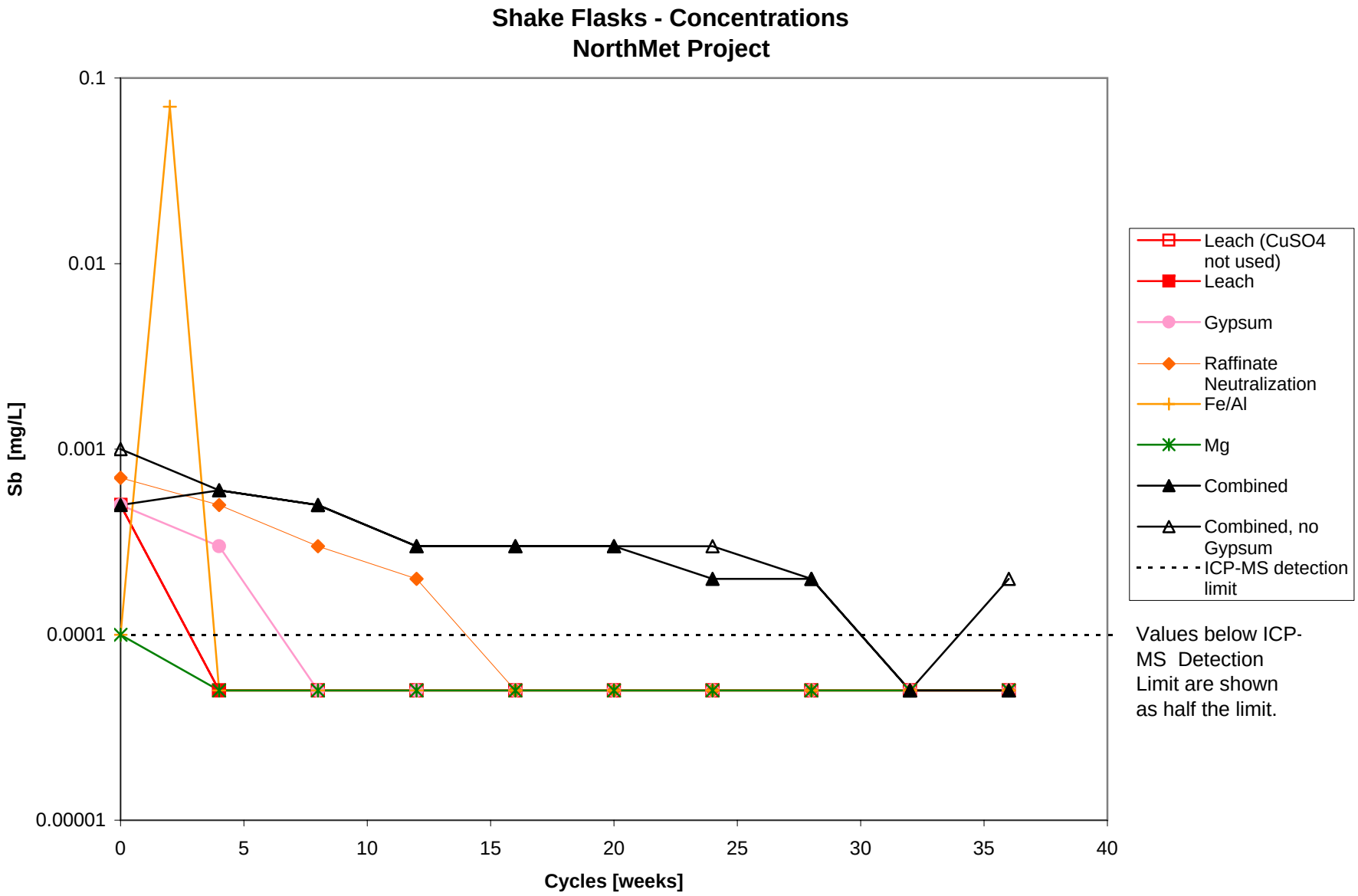

Shake Flasks - Concentrations
NorthMet Project
F [mg/L]
10
1
0.1
0.01
0
5
10
15
20
25
30
35
40
Cycles [weeks]
variable D.L.
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below Detection Limit are shown as half the limit.

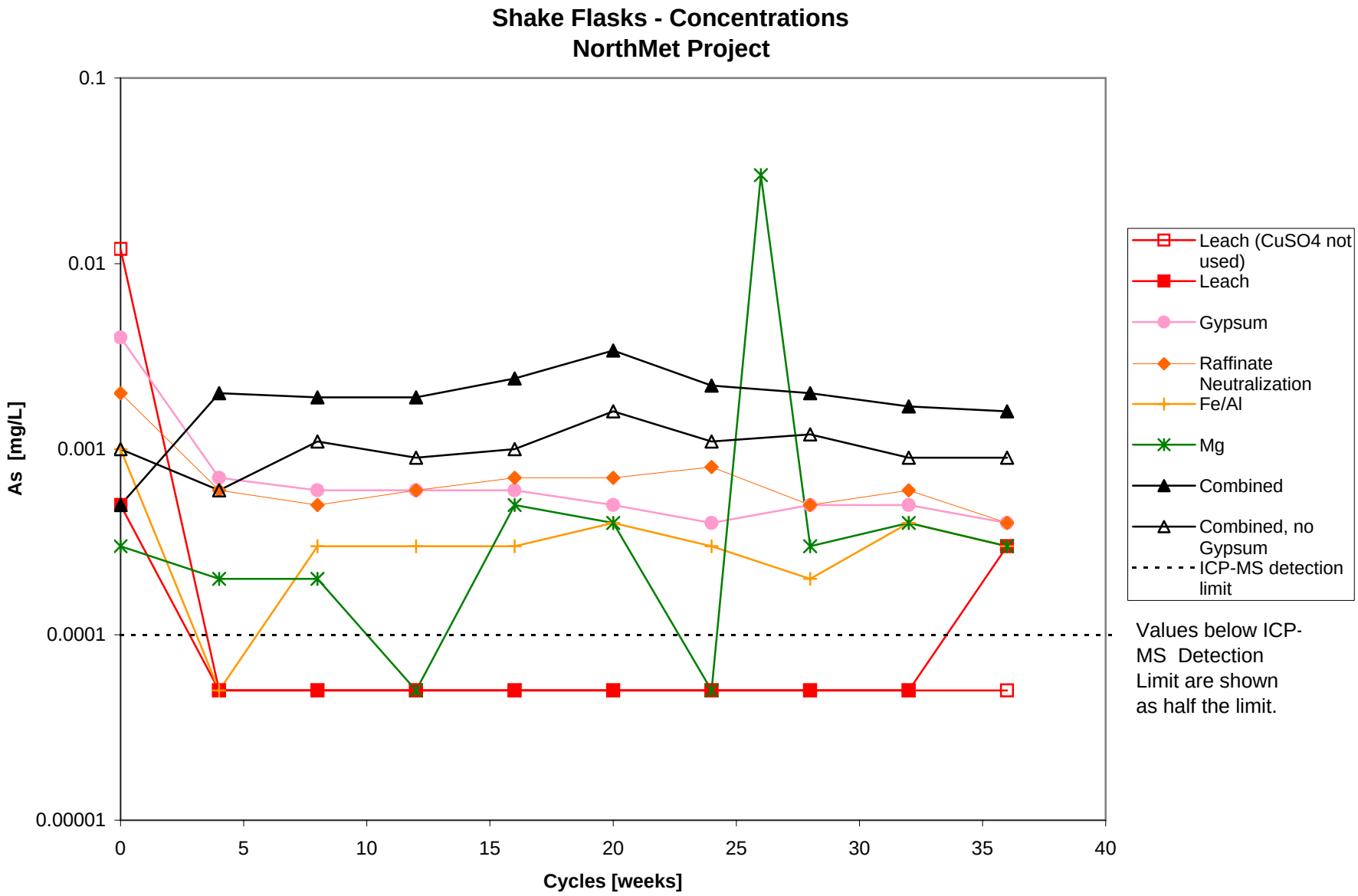

Shake Flasks - Concentrations
NorthMet Project
Cl [mg/L]
10000
1000
100
10
1
0.1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
variable D.L.
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below Detection Limit are shown as half the limit.

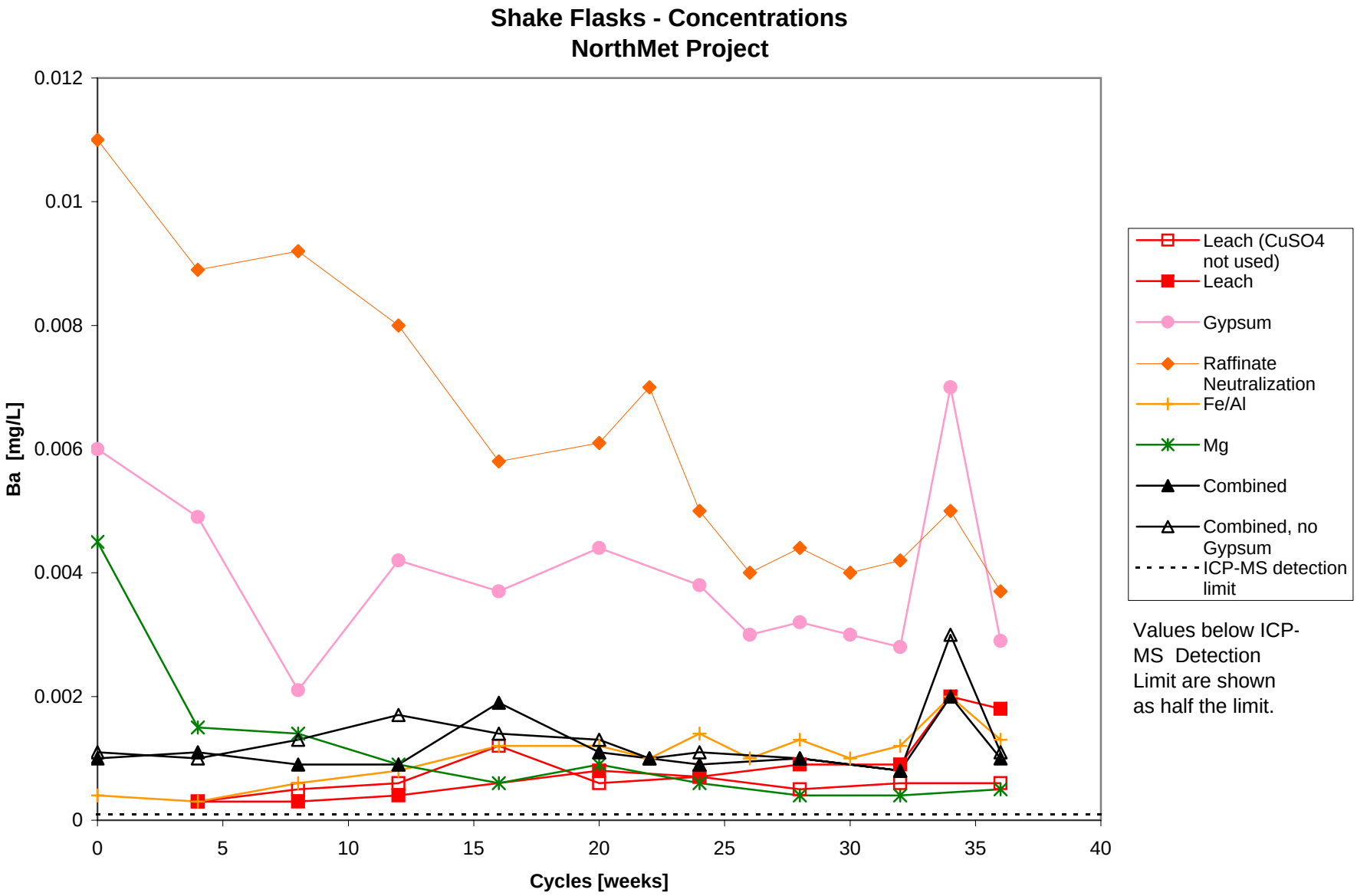

Shake Flasks - Concentrations
NorthMet Project
SO4 [mg/L]
Cycles [weeks]
0
1000
2000
3000
4000
5000
6000
7000
8000
9000
10000
0
5
10
15
20
25
30
35
40
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
Detection limit - 0.5 mg/L
Values below Detection Limit are shown as half the limit.



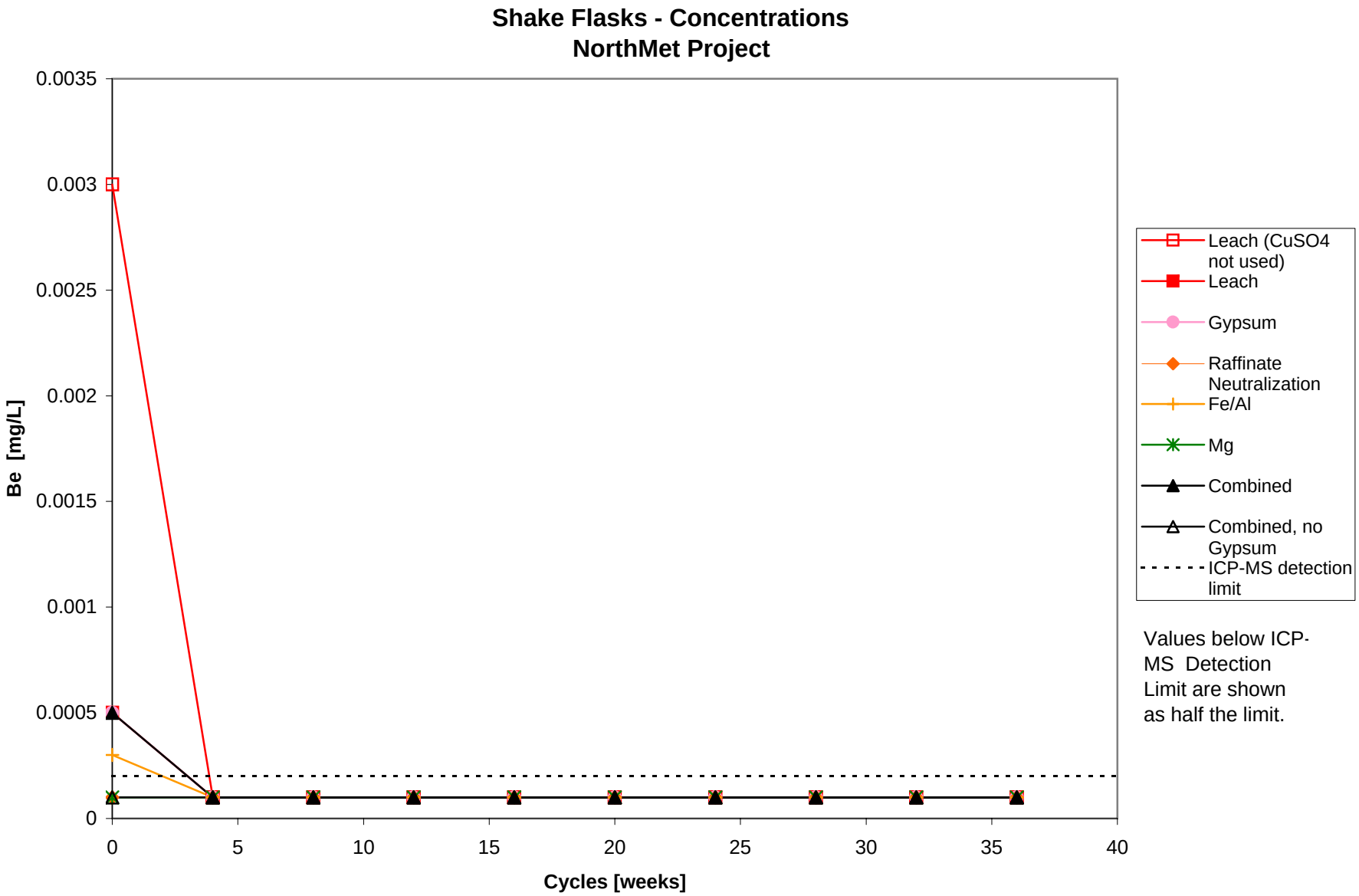
Values below ICP-MS Detection Limit are shown as half the limit.


Shake Flasks - Concentrations
NorthMet Project
Sb [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

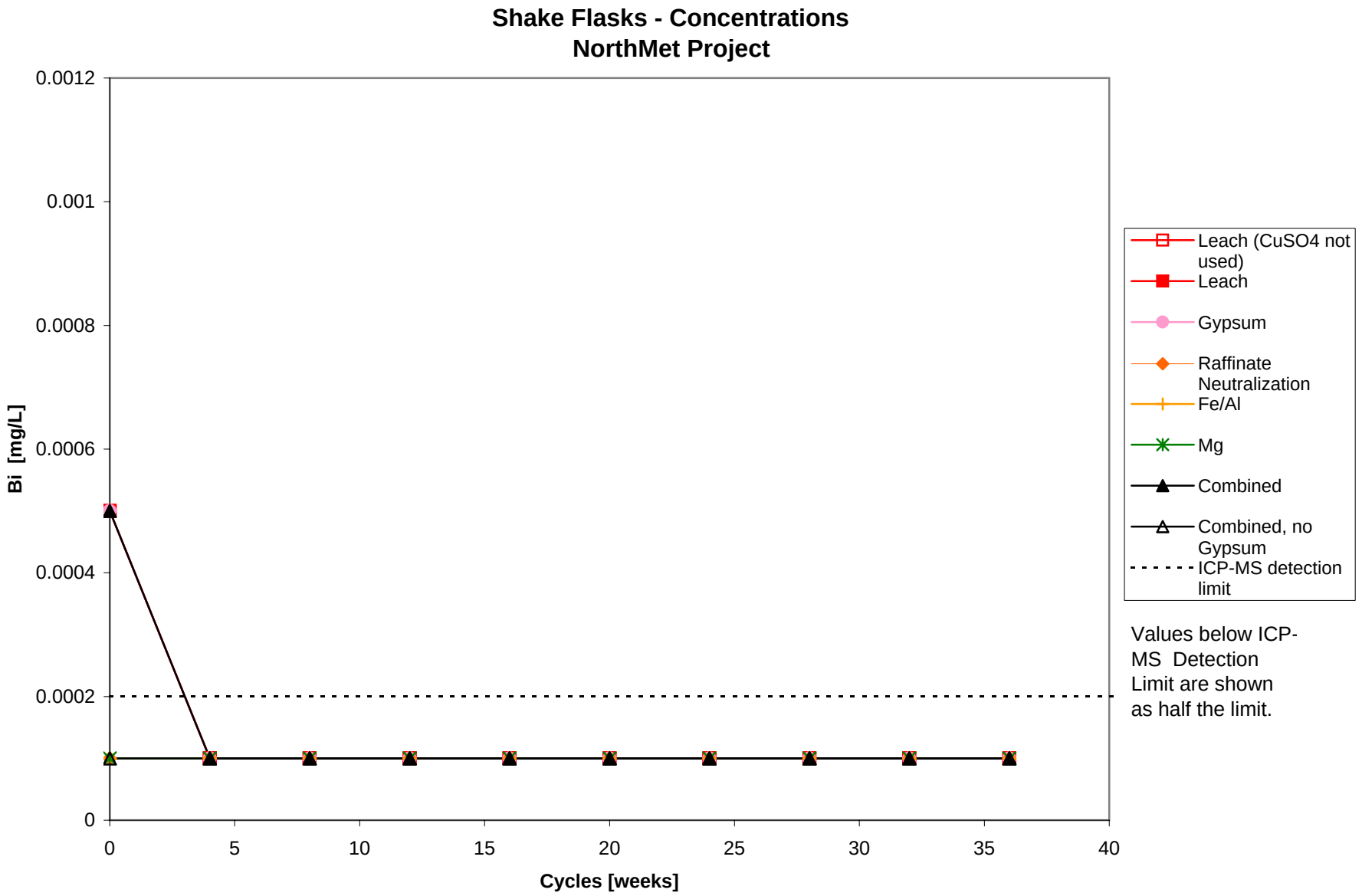

Shake Flasks - Concentrations
NorthMet Project
As [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

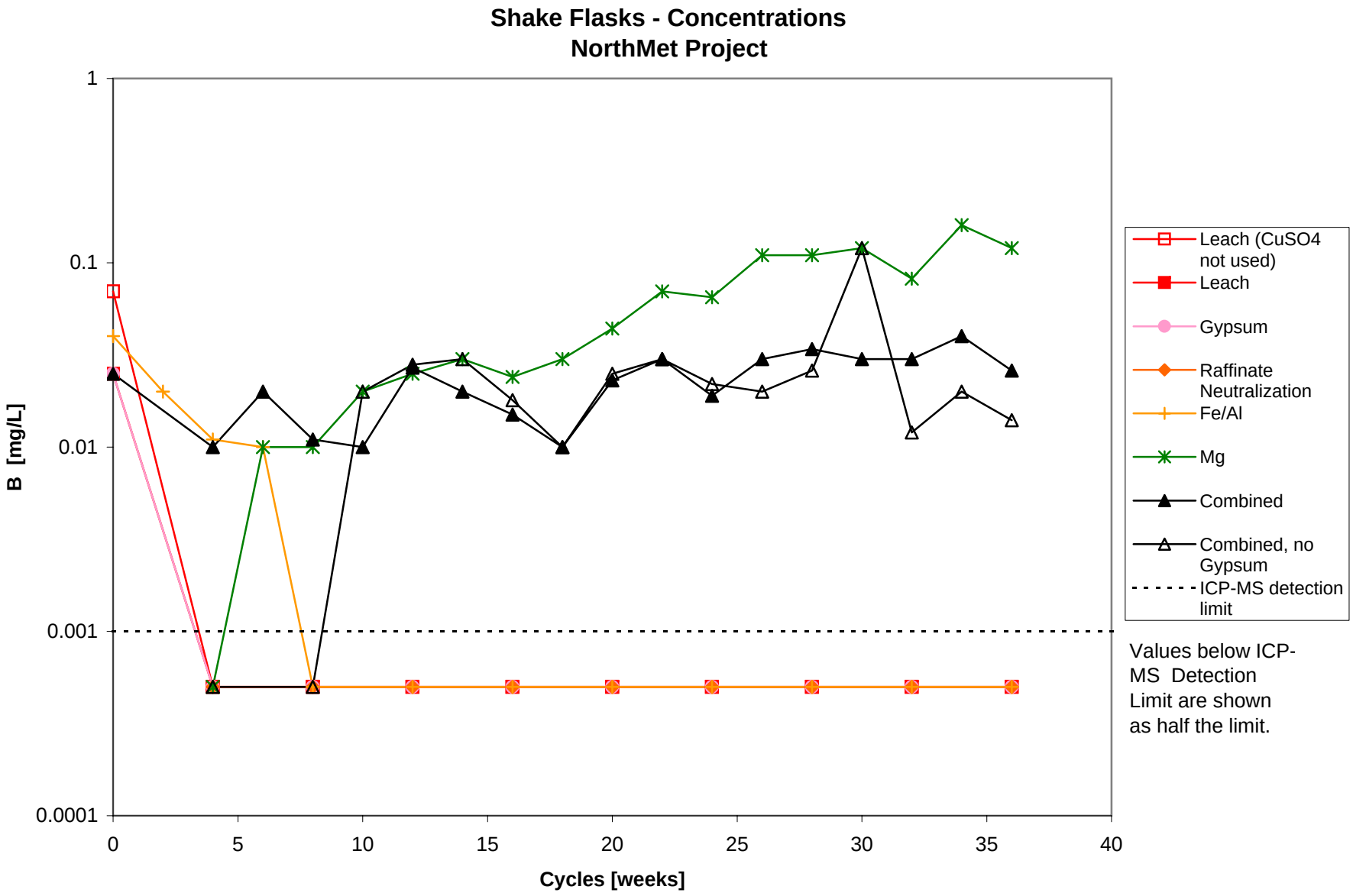


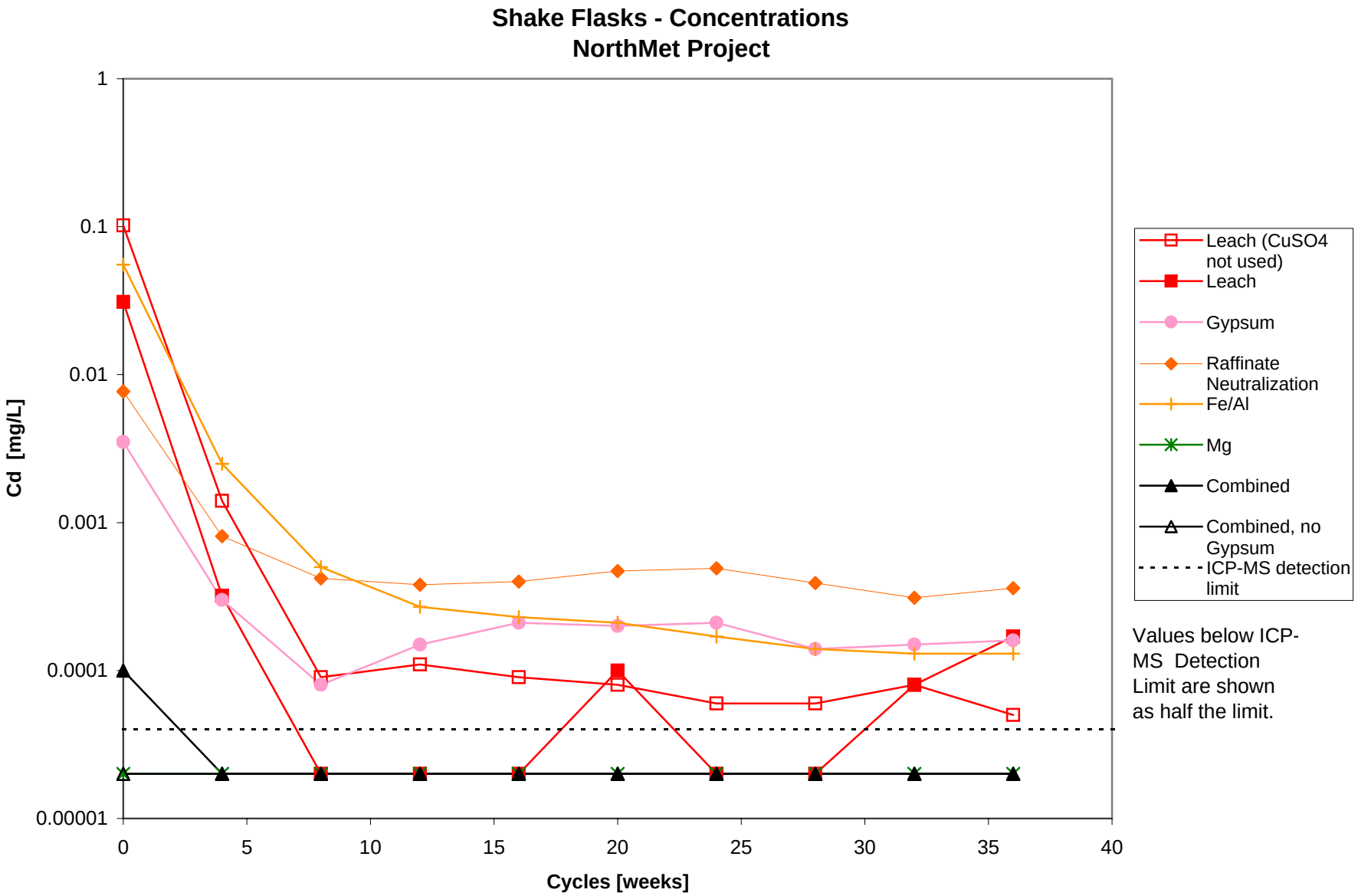
Values below ICP-MS Detection Limit are shown as half the limit.



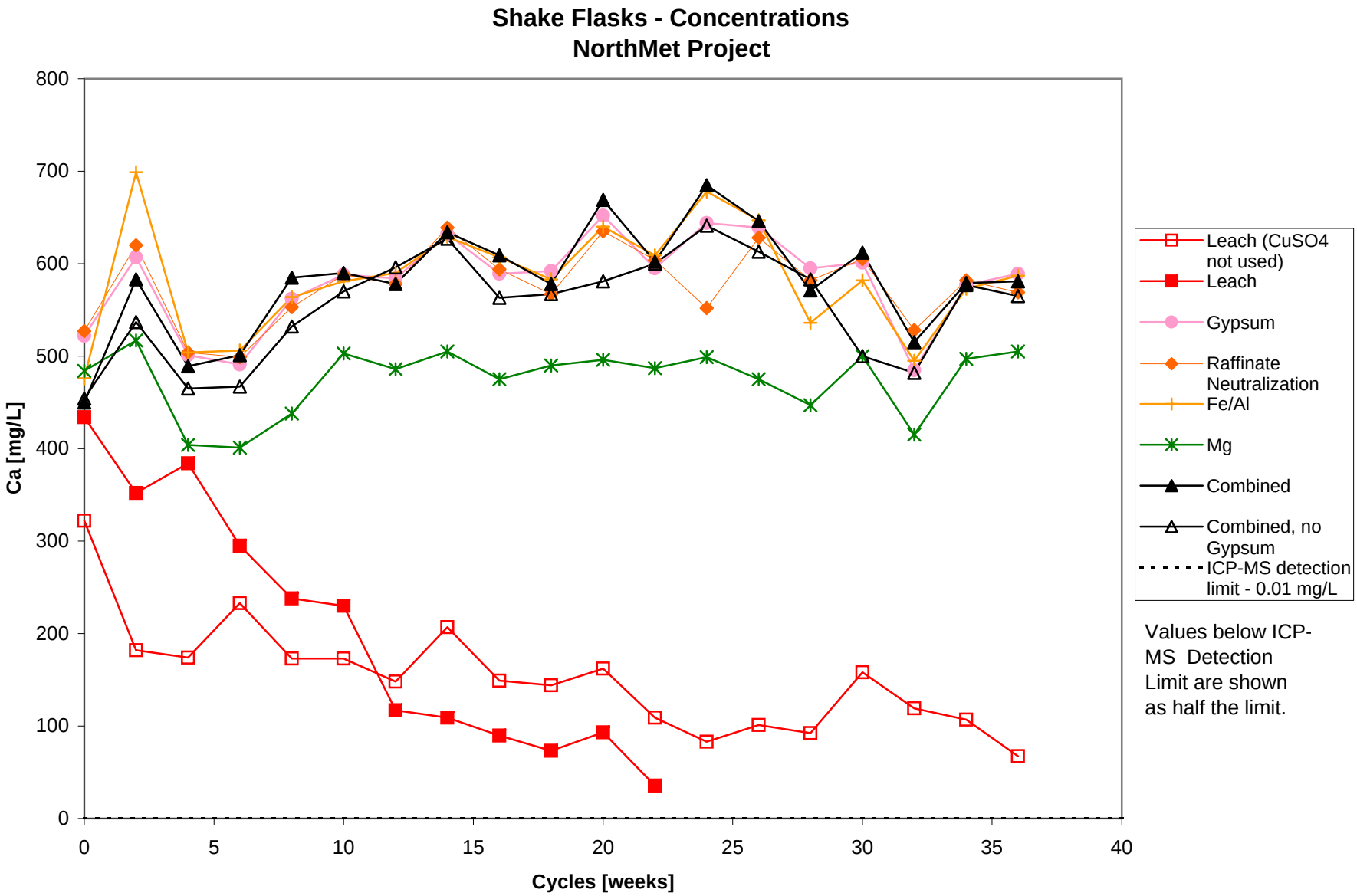
Values below ICP-MS Detection Limit are shown as half the limit.

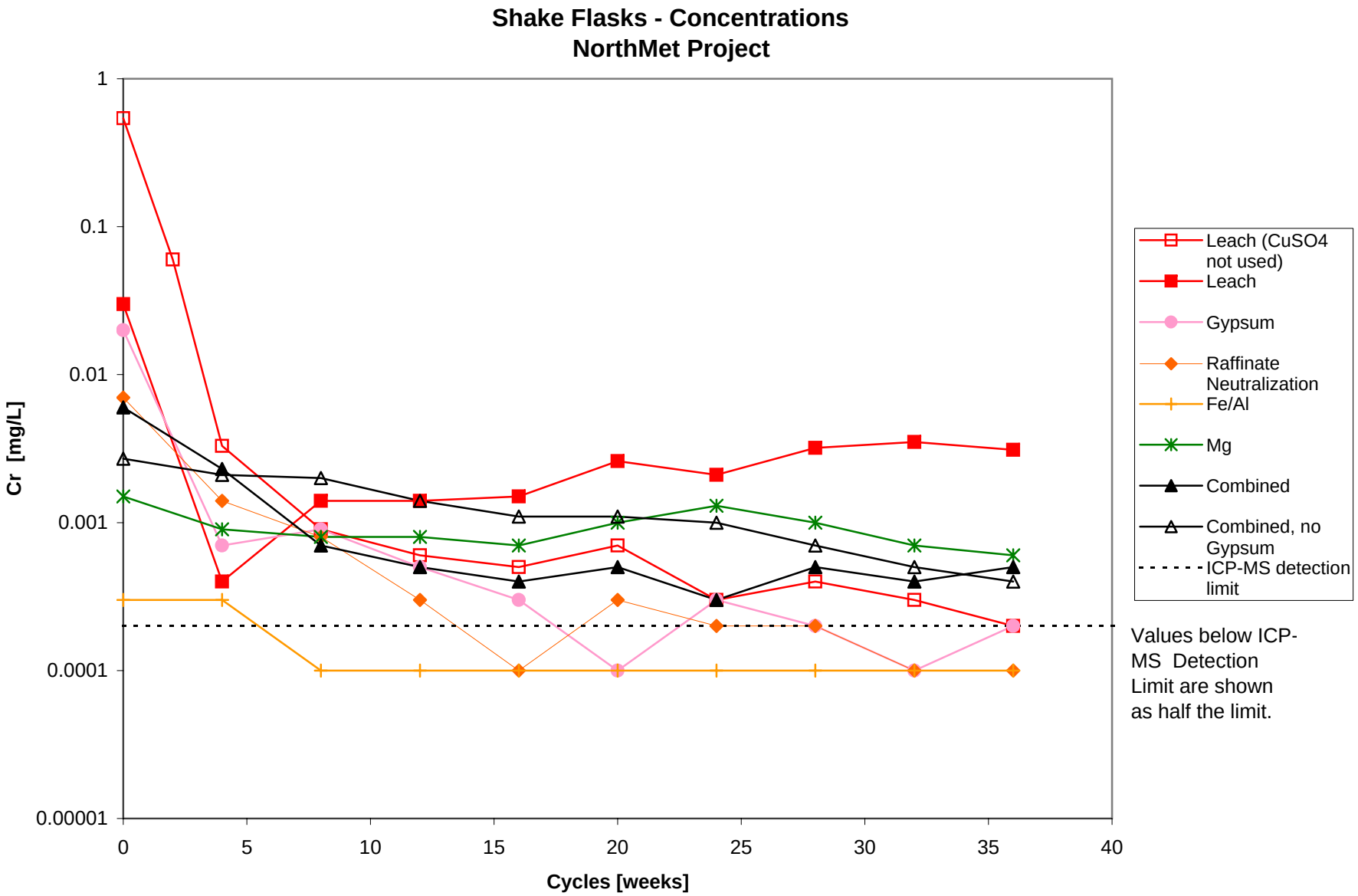

Shake Flasks - Concentrations
NorthMet Project
Bi [mg/L]
0.0012
0.001
0.0008
0.0006
0.0004
0.0002
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

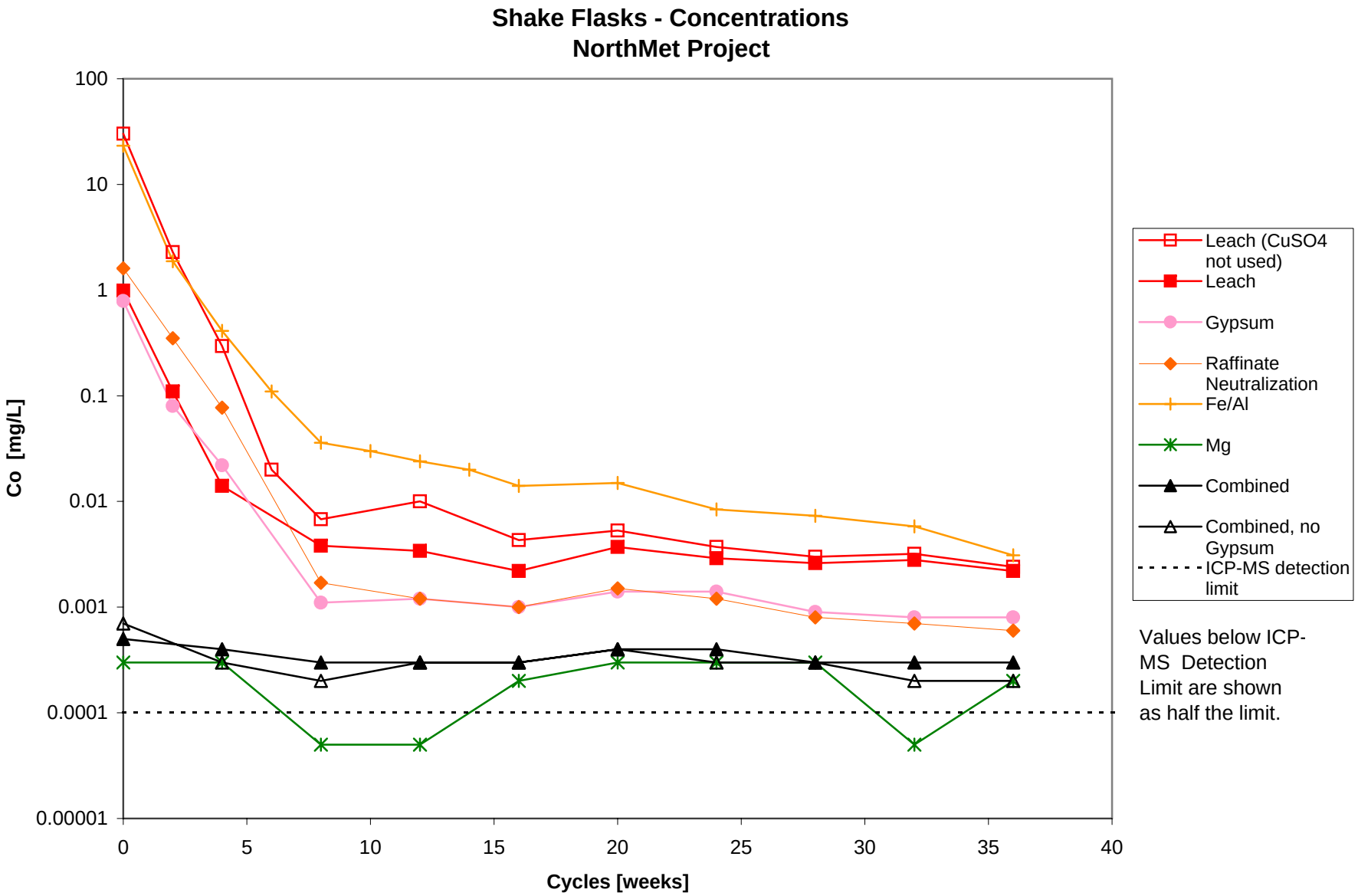

Shake Flasks - Concentrations
NorthMet Project
B [mg/L]
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.



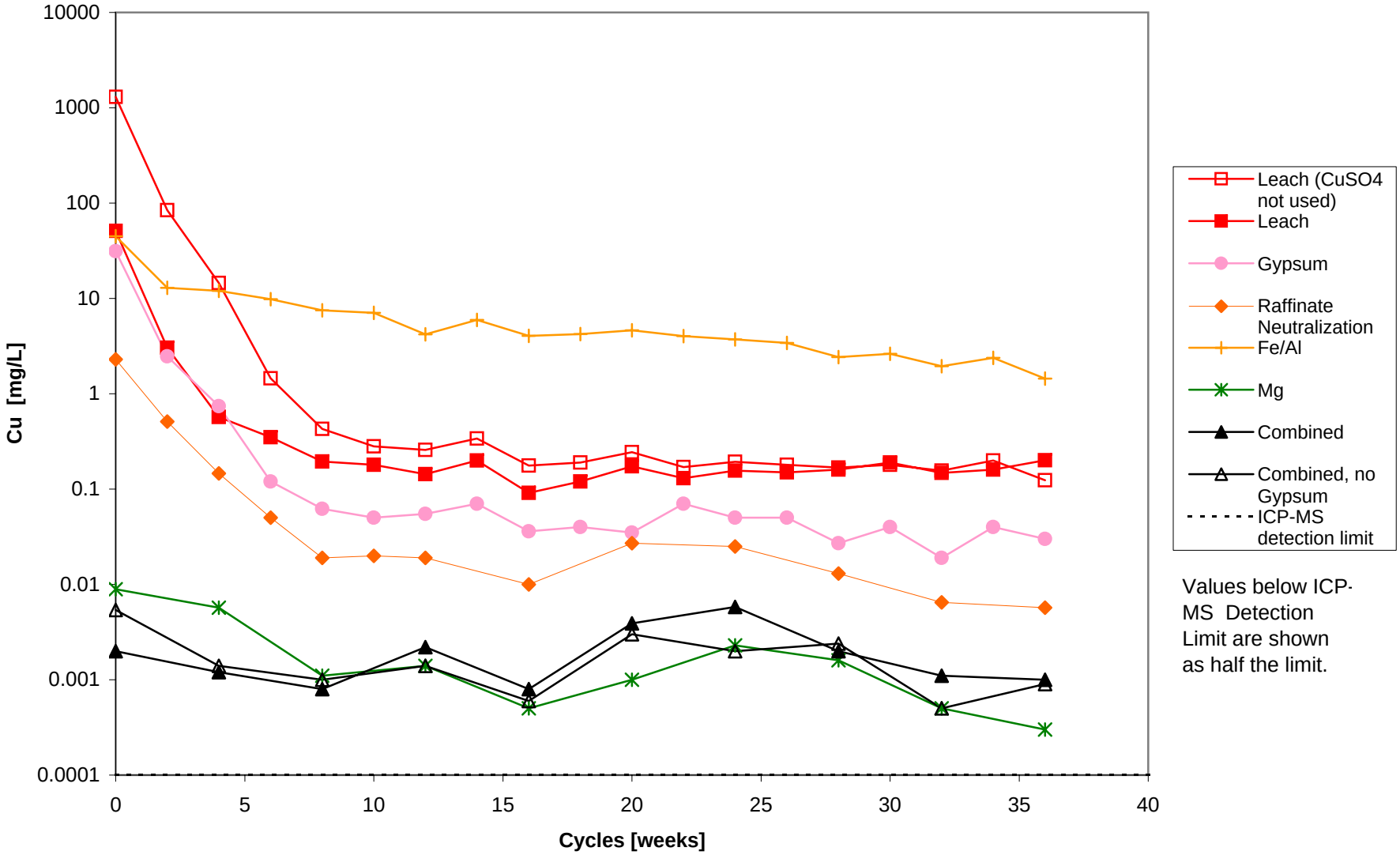
Values below ICP-MS Detection Limit are shown as half the limit.


Shake Flasks - Concentrations
NorthMet Project
Ca [mg/L]
Cycles [weeks]
0
100
200
300
400
500
600
700
800
0
5
10
15
20
25
30
35
40
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit - 0.01 mg/L
Values below ICP-MS Detection Limit are shown as half the limit.

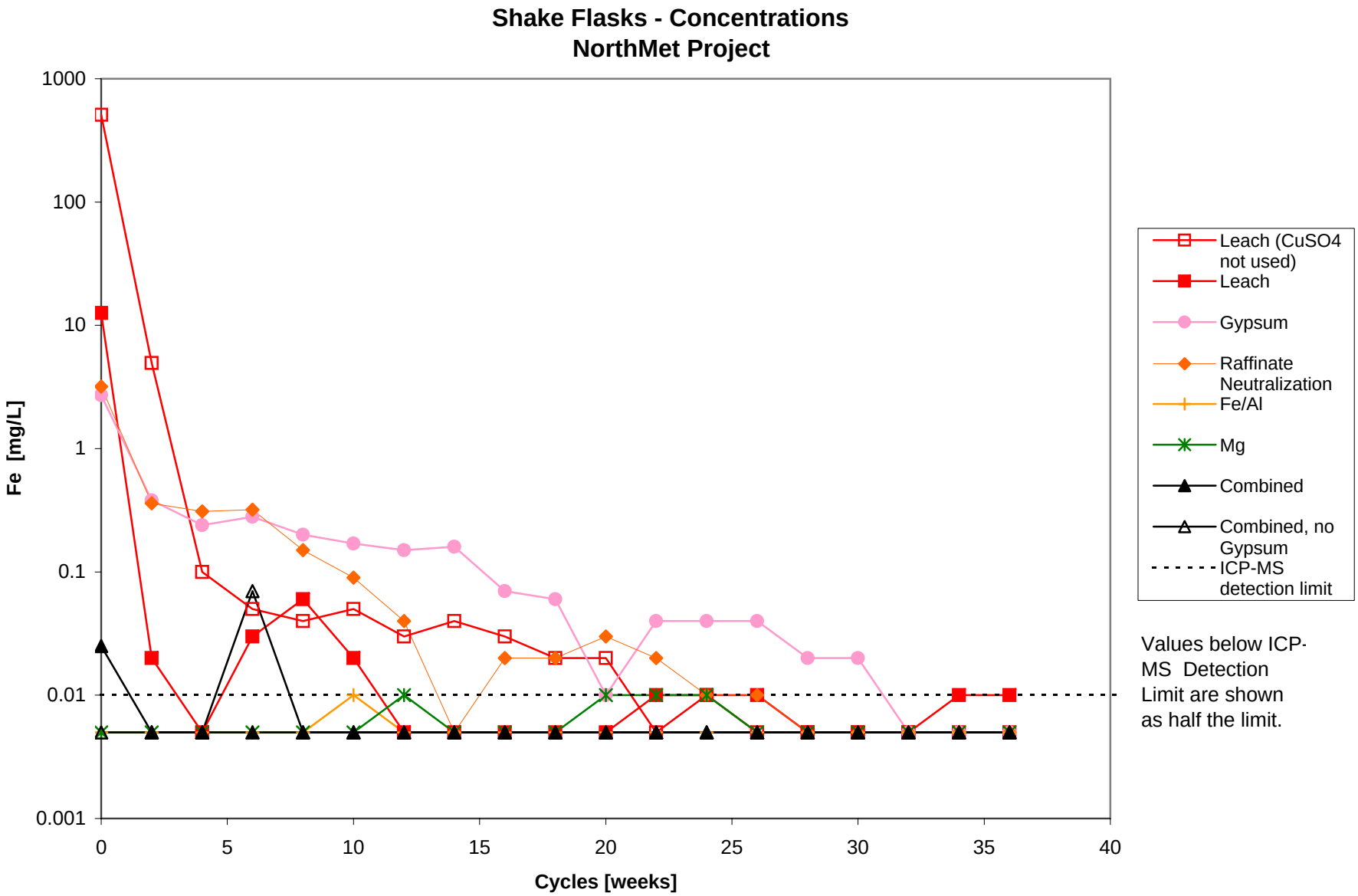

Shake Flasks - Concentrations
NorthMet Project
Cr [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

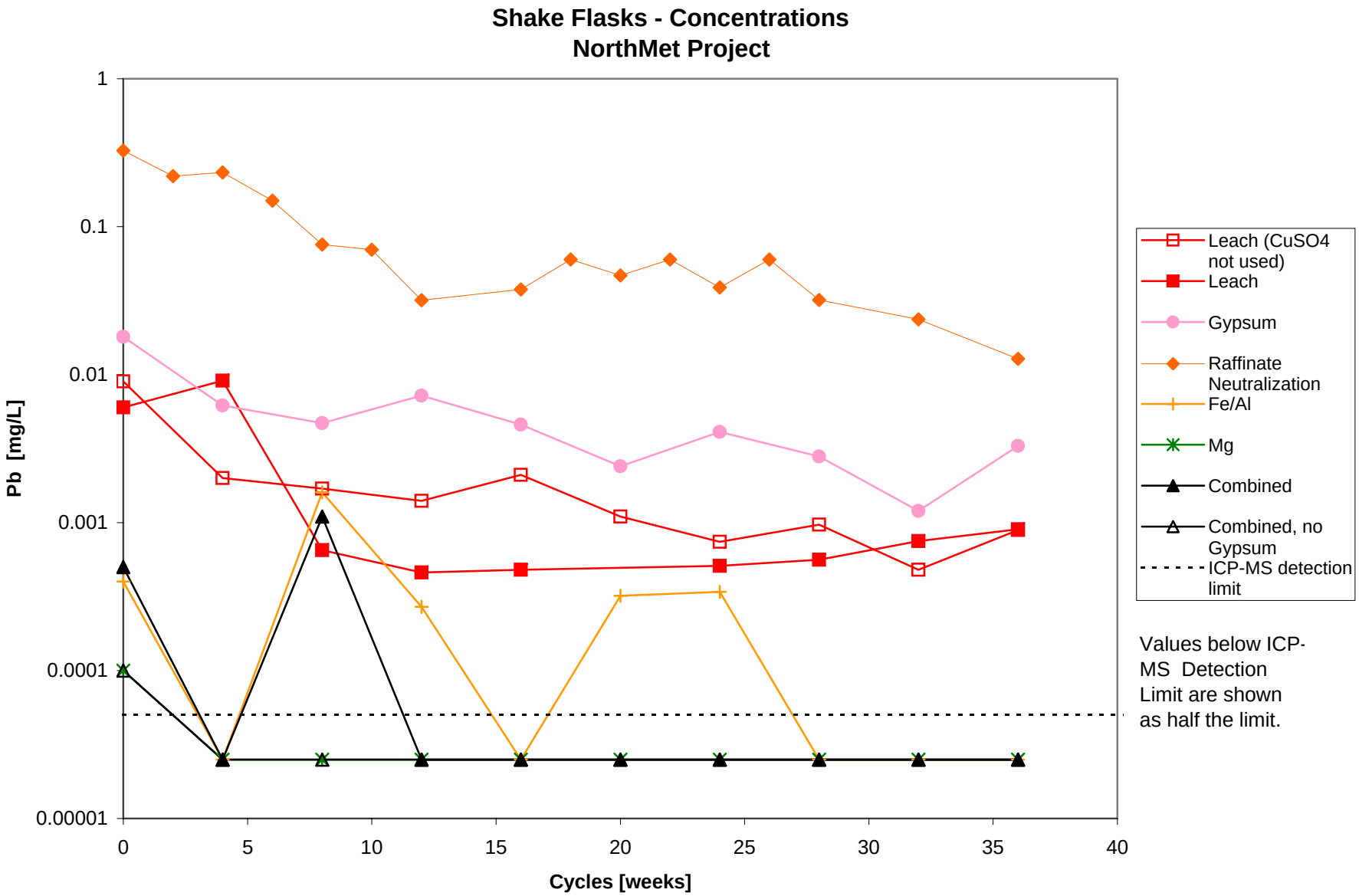

Shake Flasks - Concentrations
NorthMet Project
Co [mg/L]
100
10
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

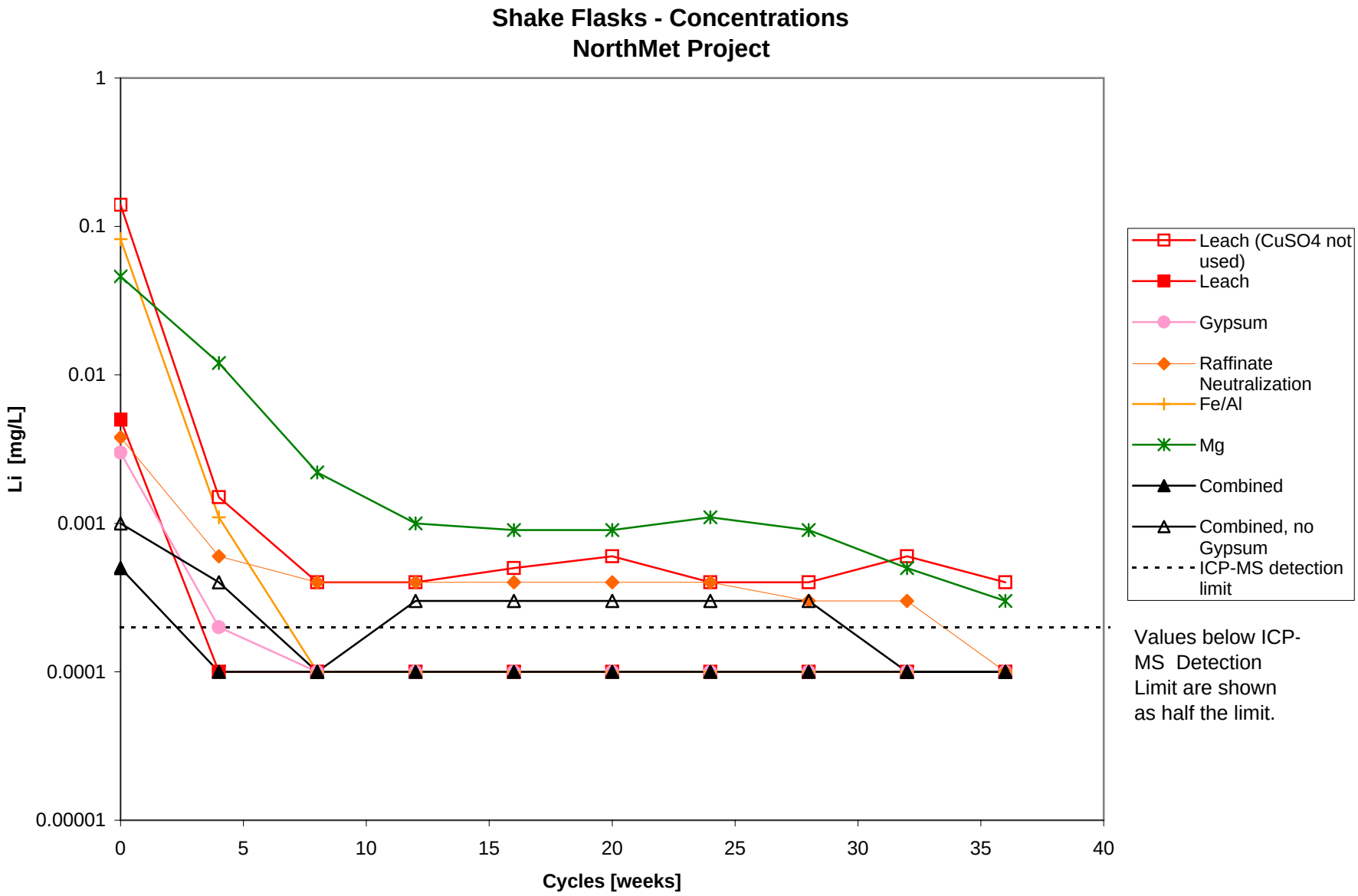
Shake Flasks - Concentrations
NorthMet Project

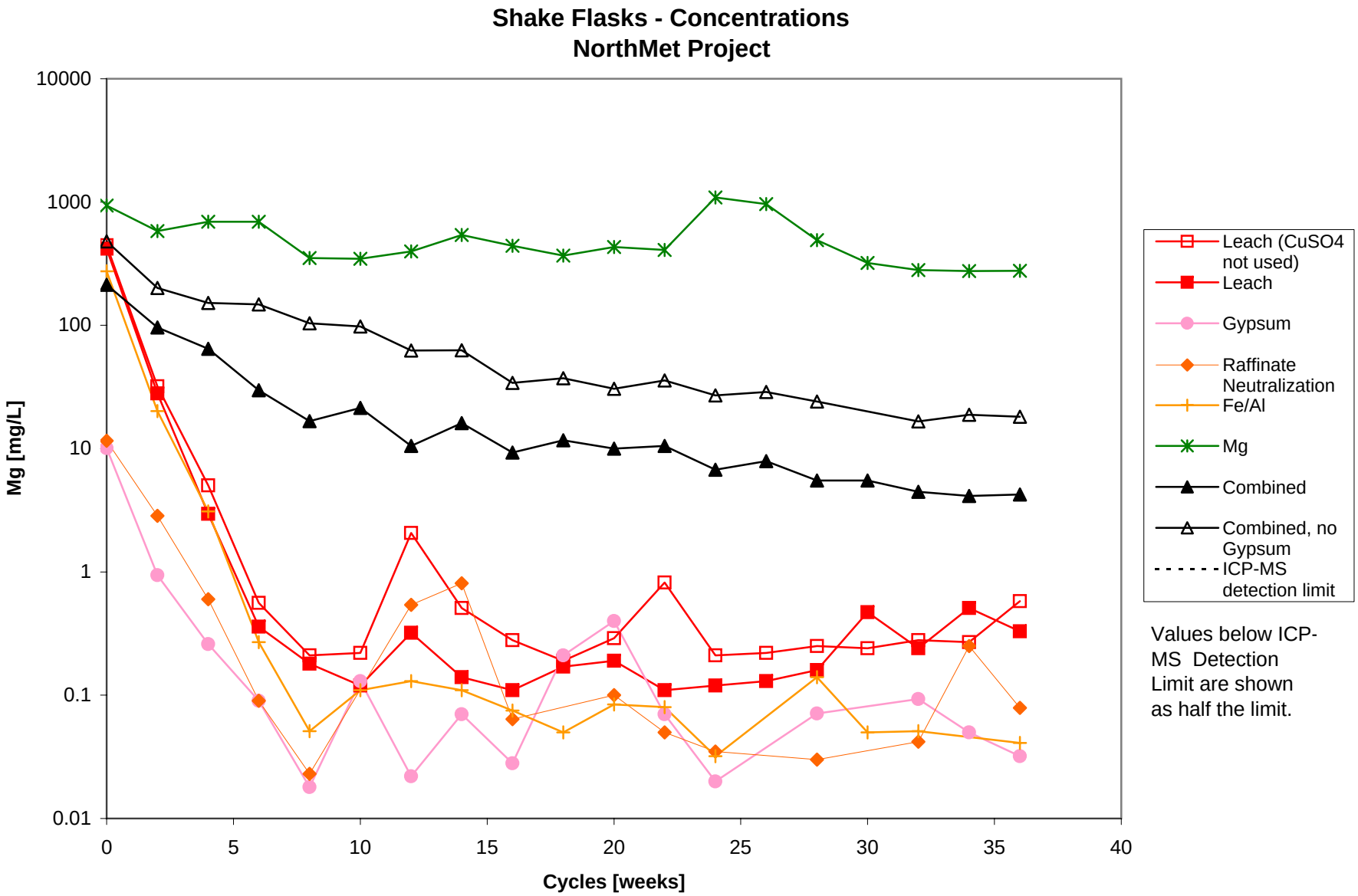


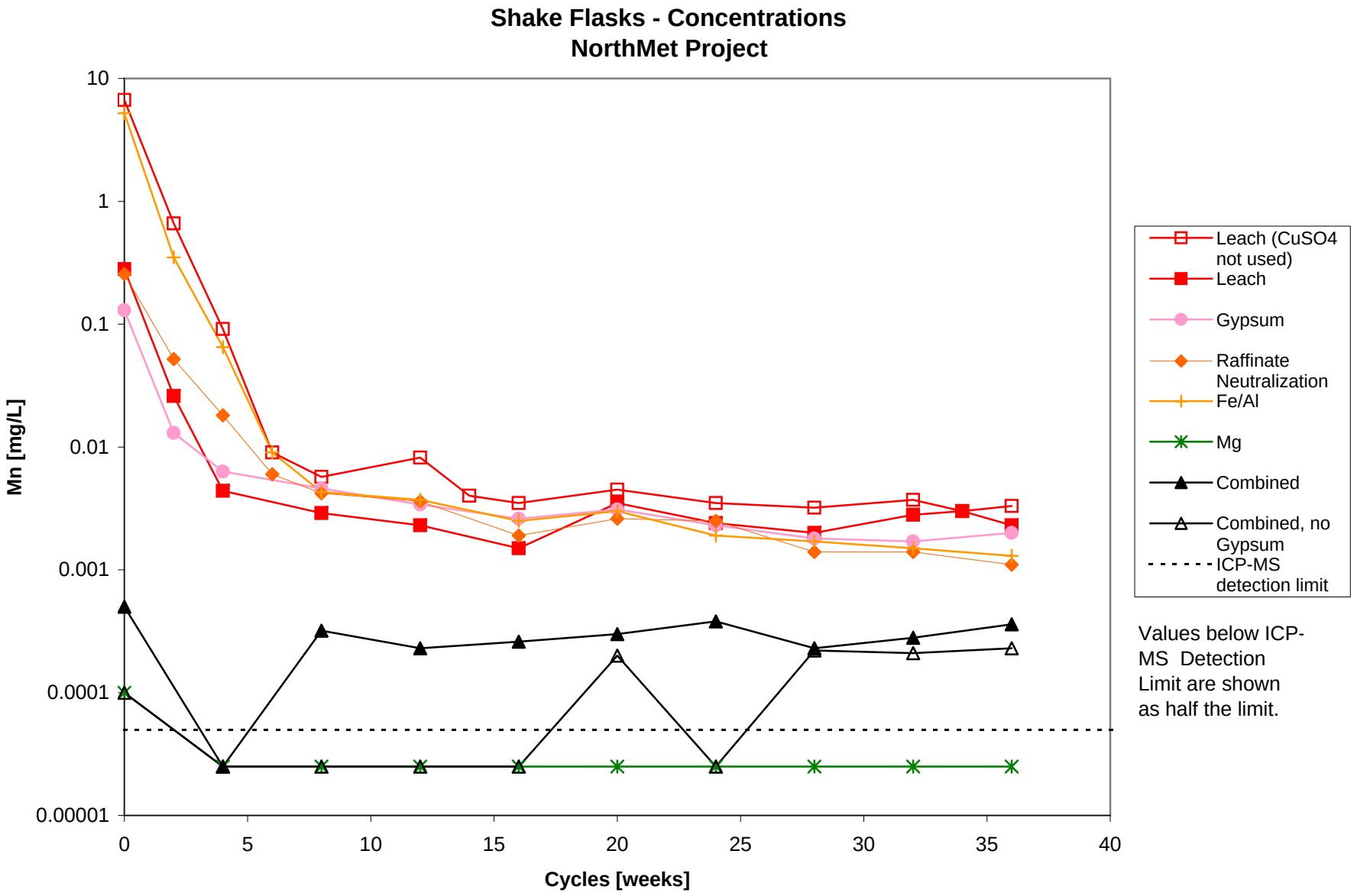
Values below ICP-MS Detection Limit are shown as half the limit.

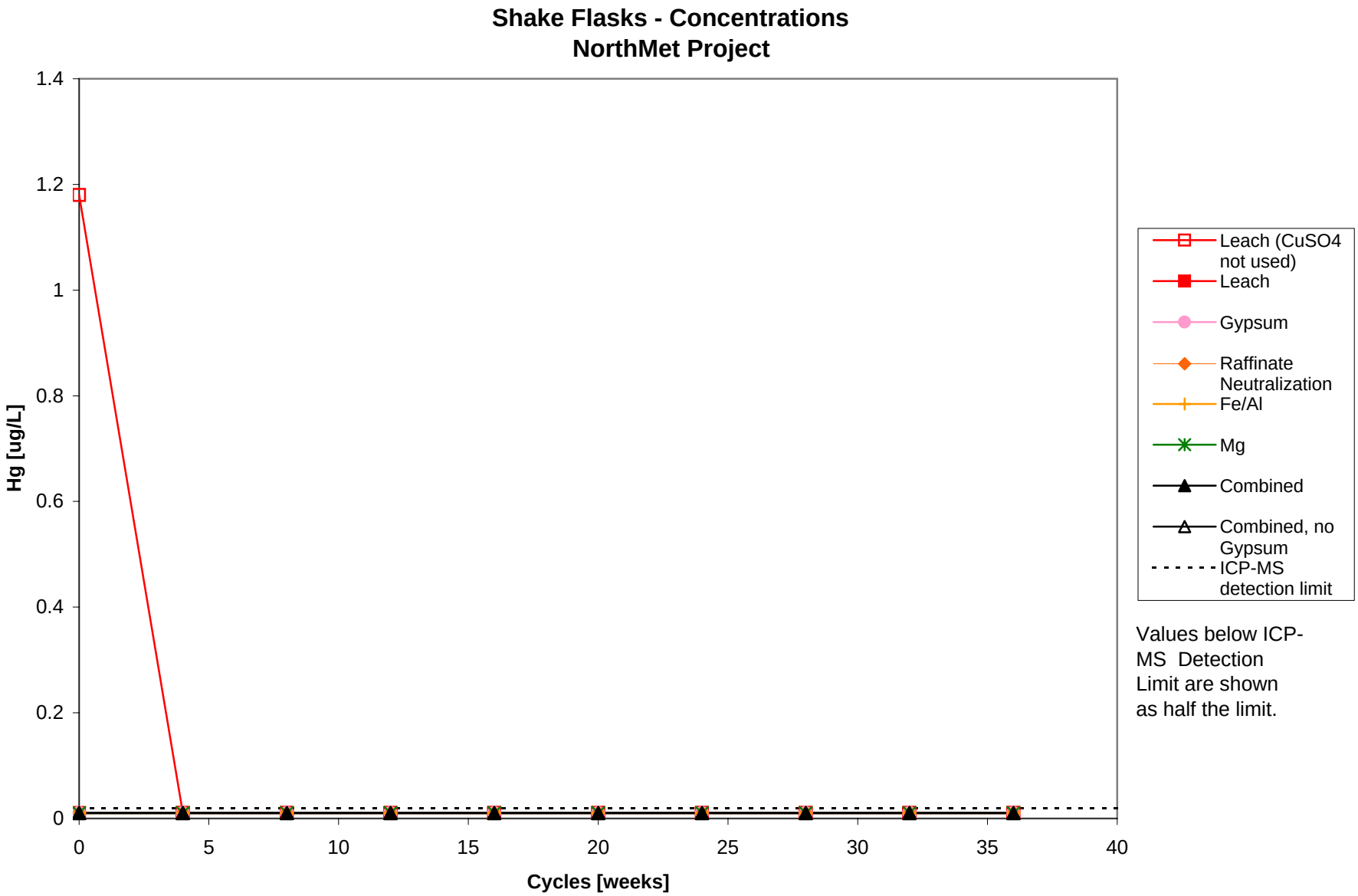

Shake Flasks - Concentrations
NorthMet Project
Fe [mg/L]
1000
100
10
1
0.1
0.01
0.001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.


Shake Flasks - Concentrations
NorthMet Project
Pb [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

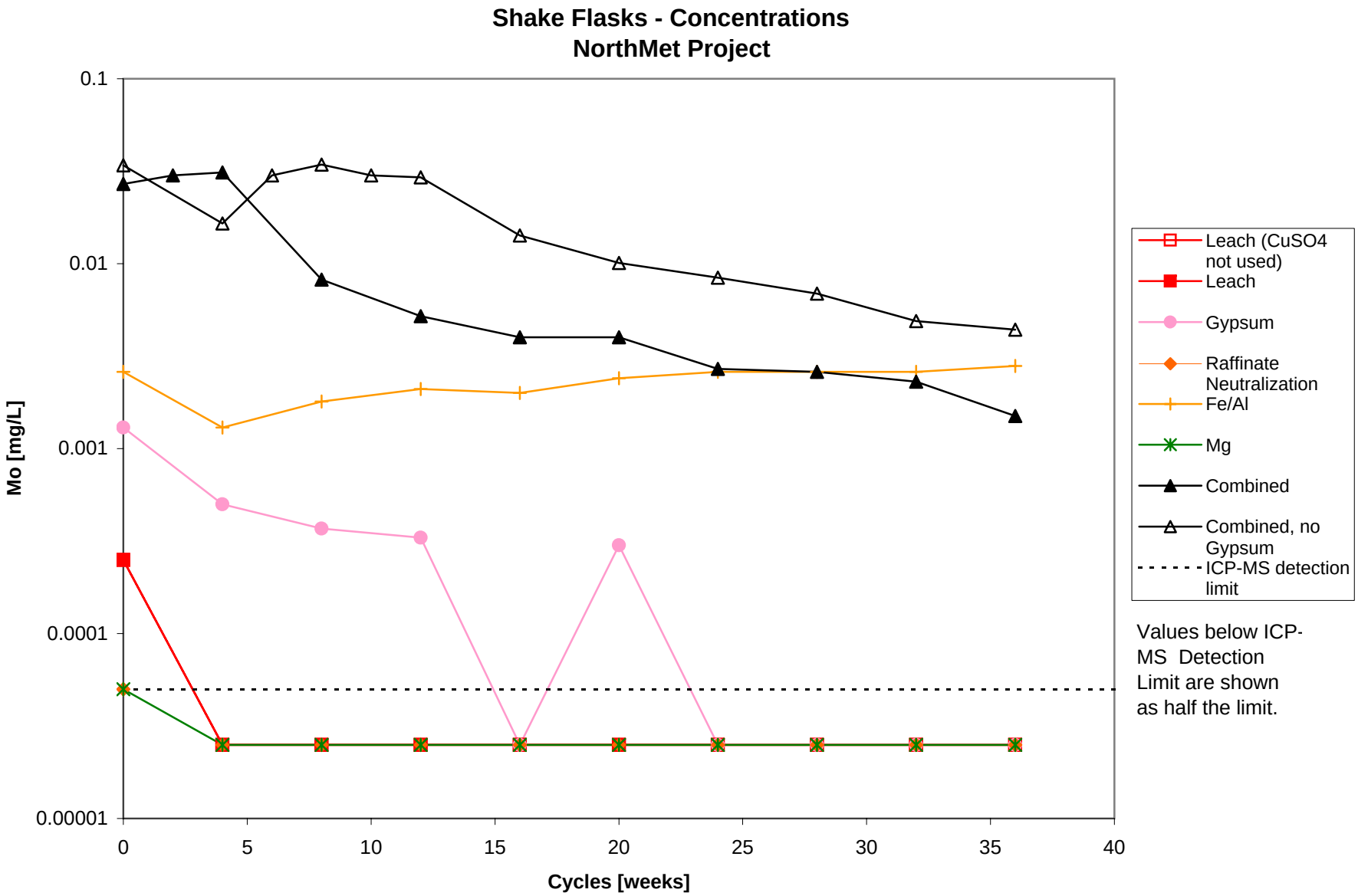

Shake Flasks - Concentrations
NorthMet Project
Li [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

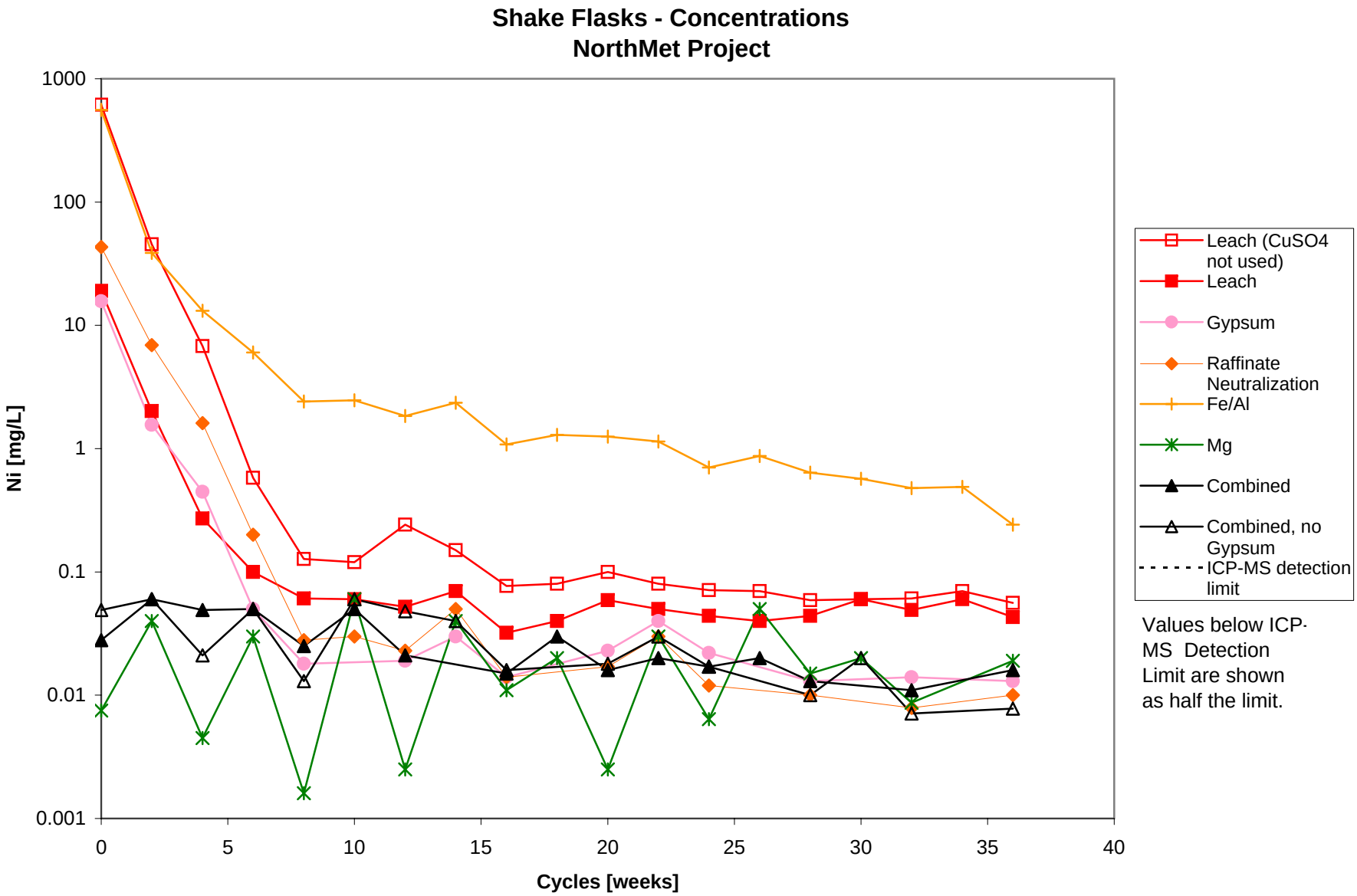

Shake Flasks - Concentrations
NorthMet Project
Mg [mg/L]
10000
1000
100
10
1
0.1
0.01
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

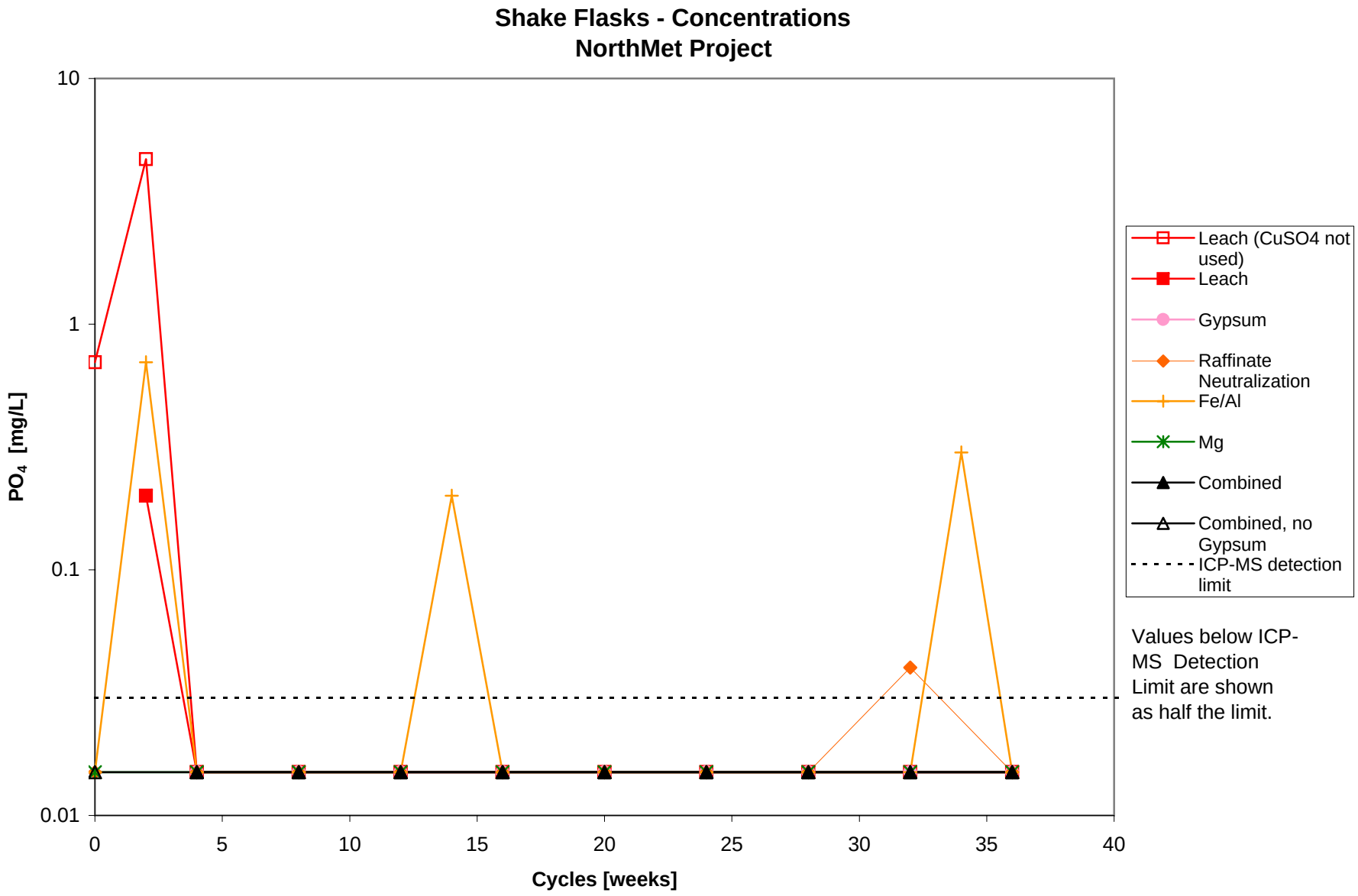

Shake Flasks - Concentrations
NorthMet Project
Mn [mg/L]
10
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.



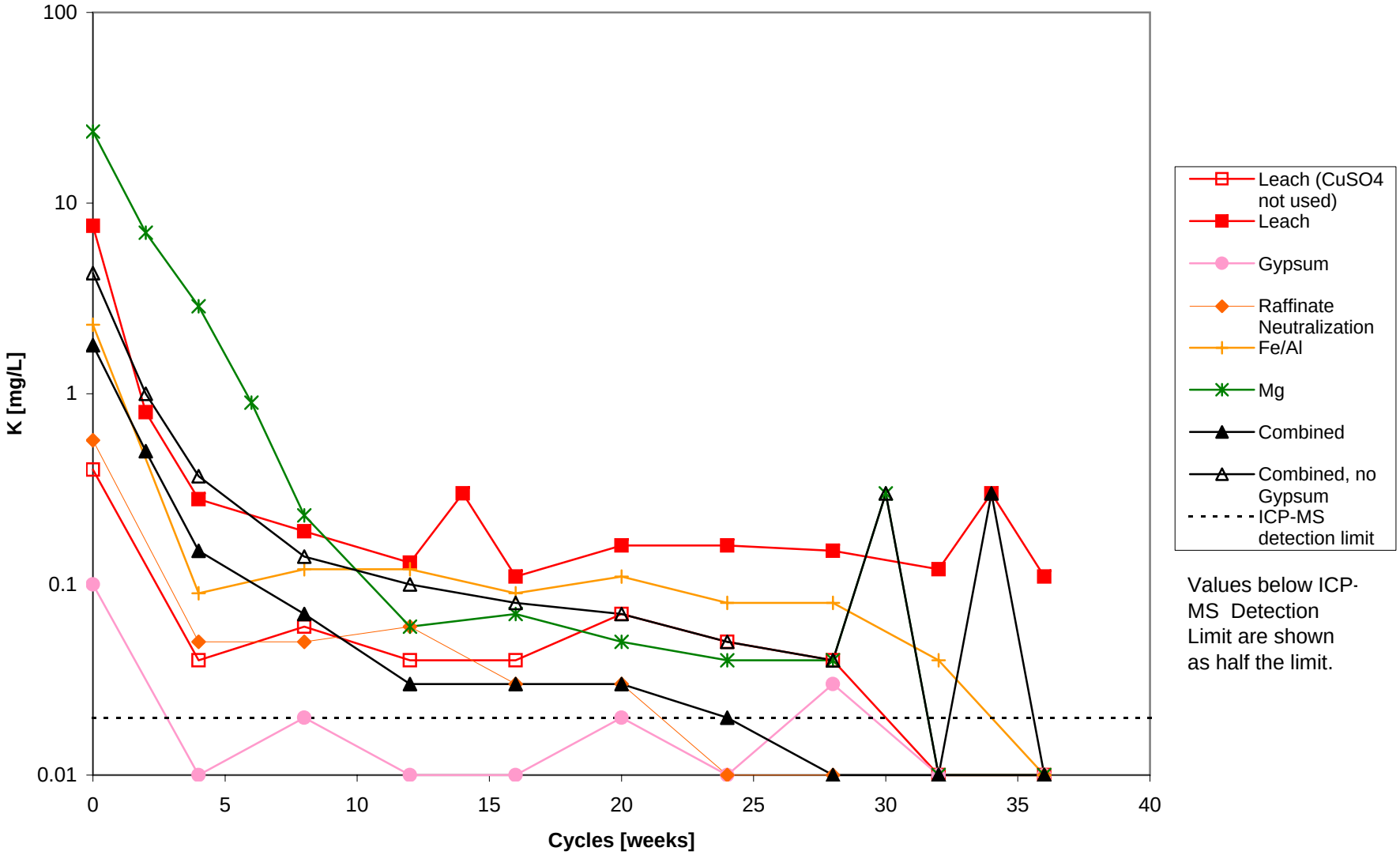
Values below ICP-MS Detection Limit are shown as half the limit.


Shake Flasks - Concentrations
NorthMet Project
Mo [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

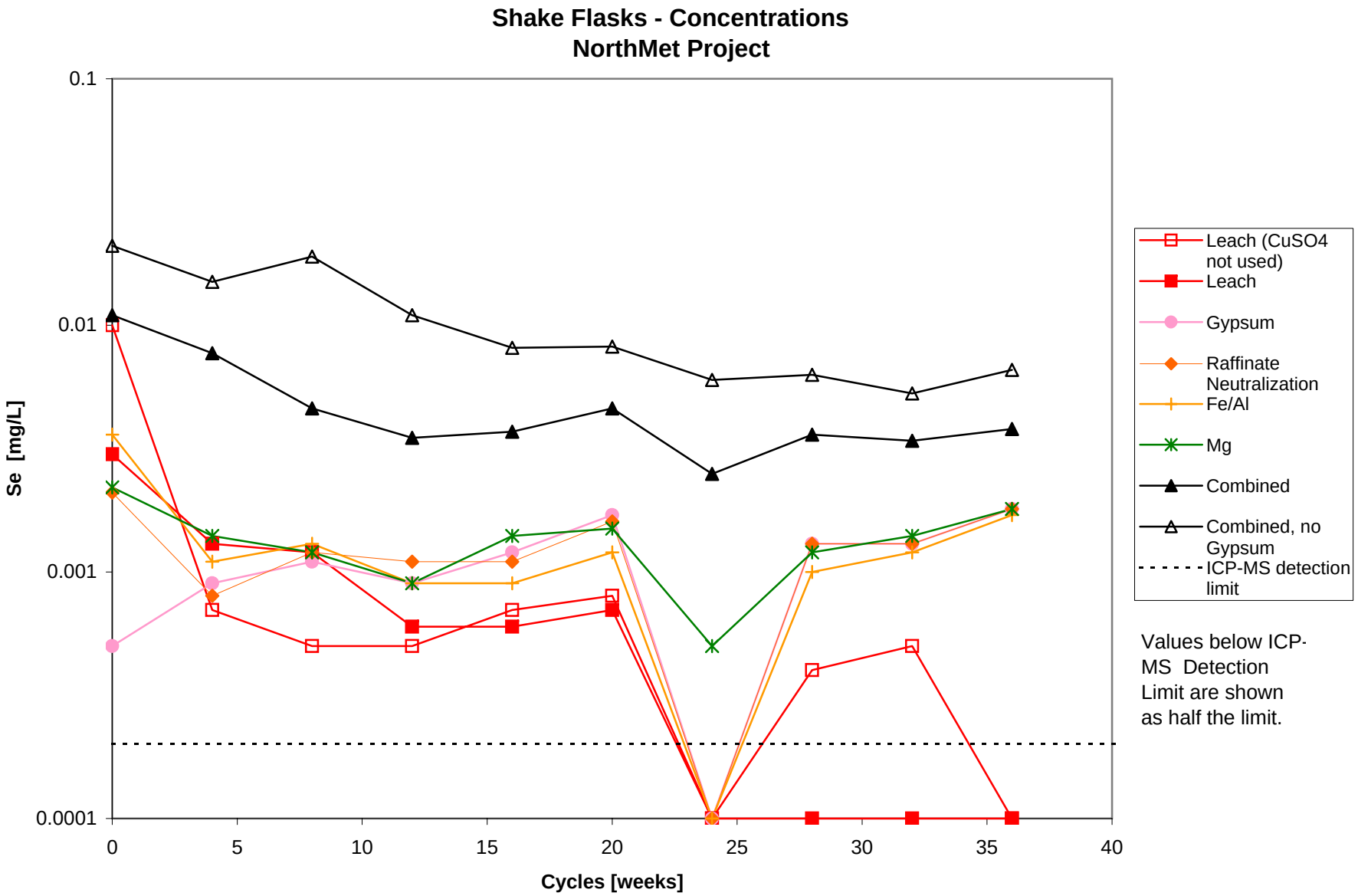

Shake Flasks - Concentrations
NorthMet Project
Ni [mg/L]
1000
100
10
1
0.1
0.01
0.001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

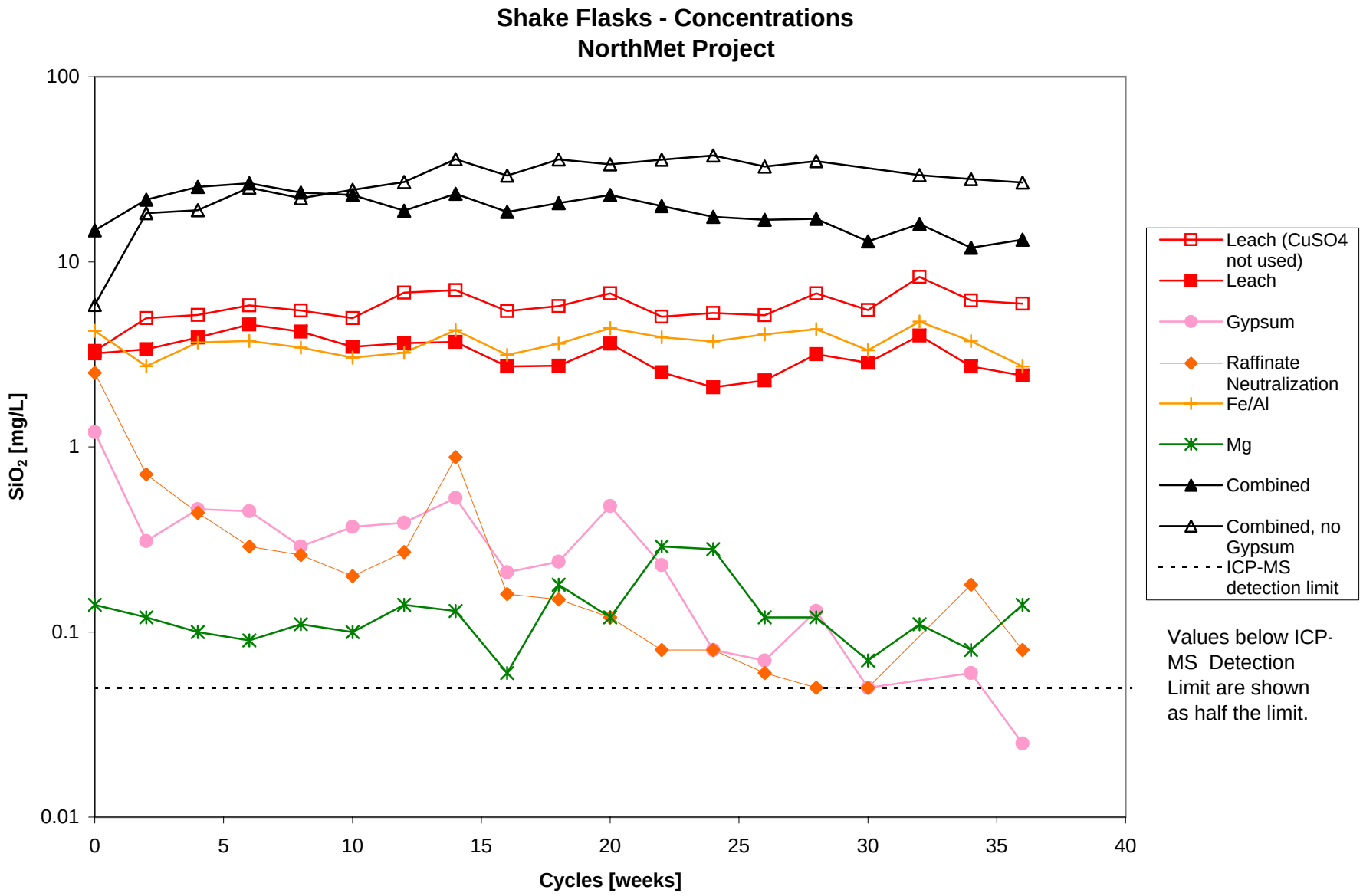

Shake Flasks - Concentrations
NorthMet Project
10
1
0.1
0.01
PO4 [mg/L]
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

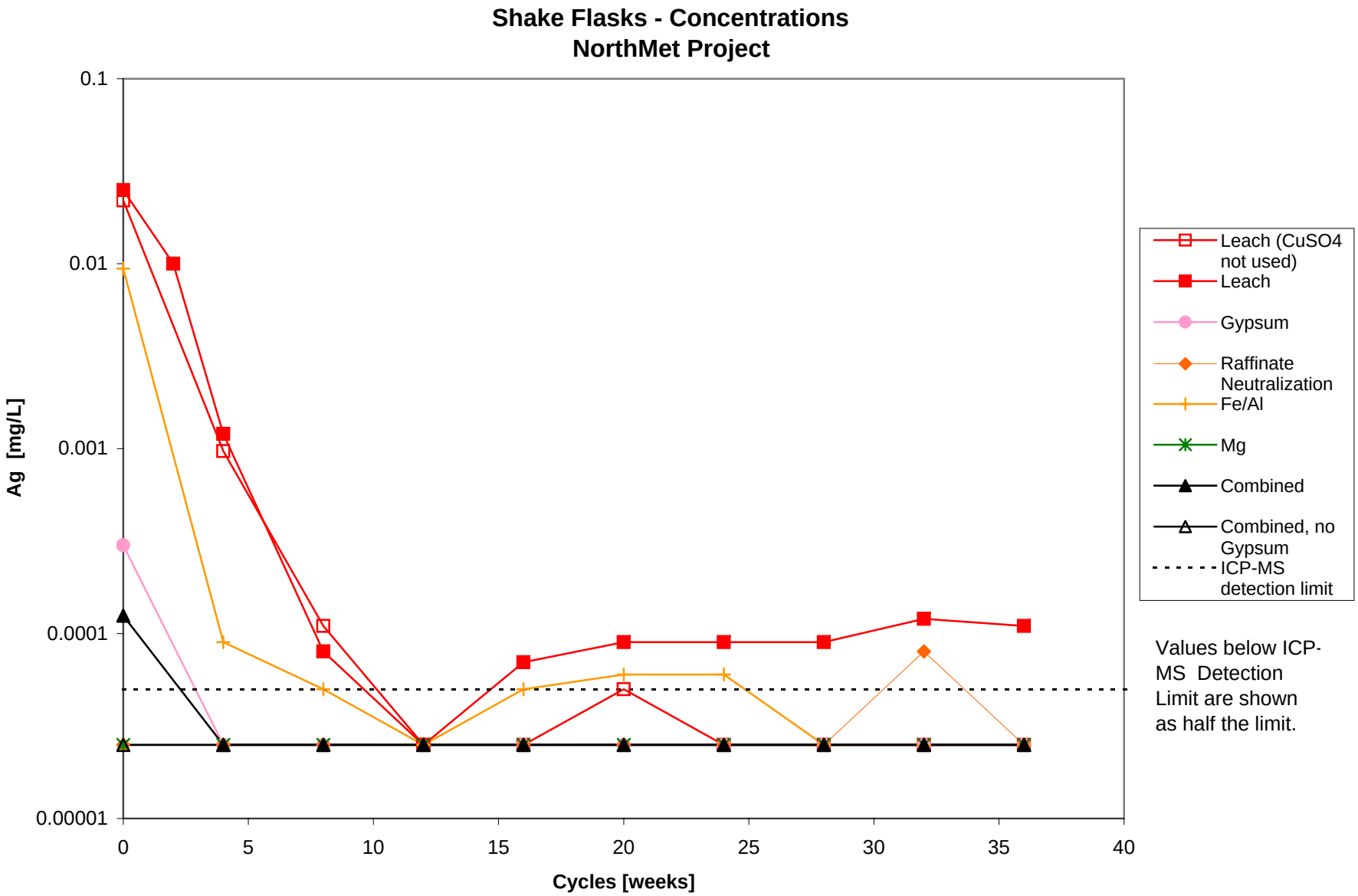
Shake Flasks - Concentrations
NorthMet Project



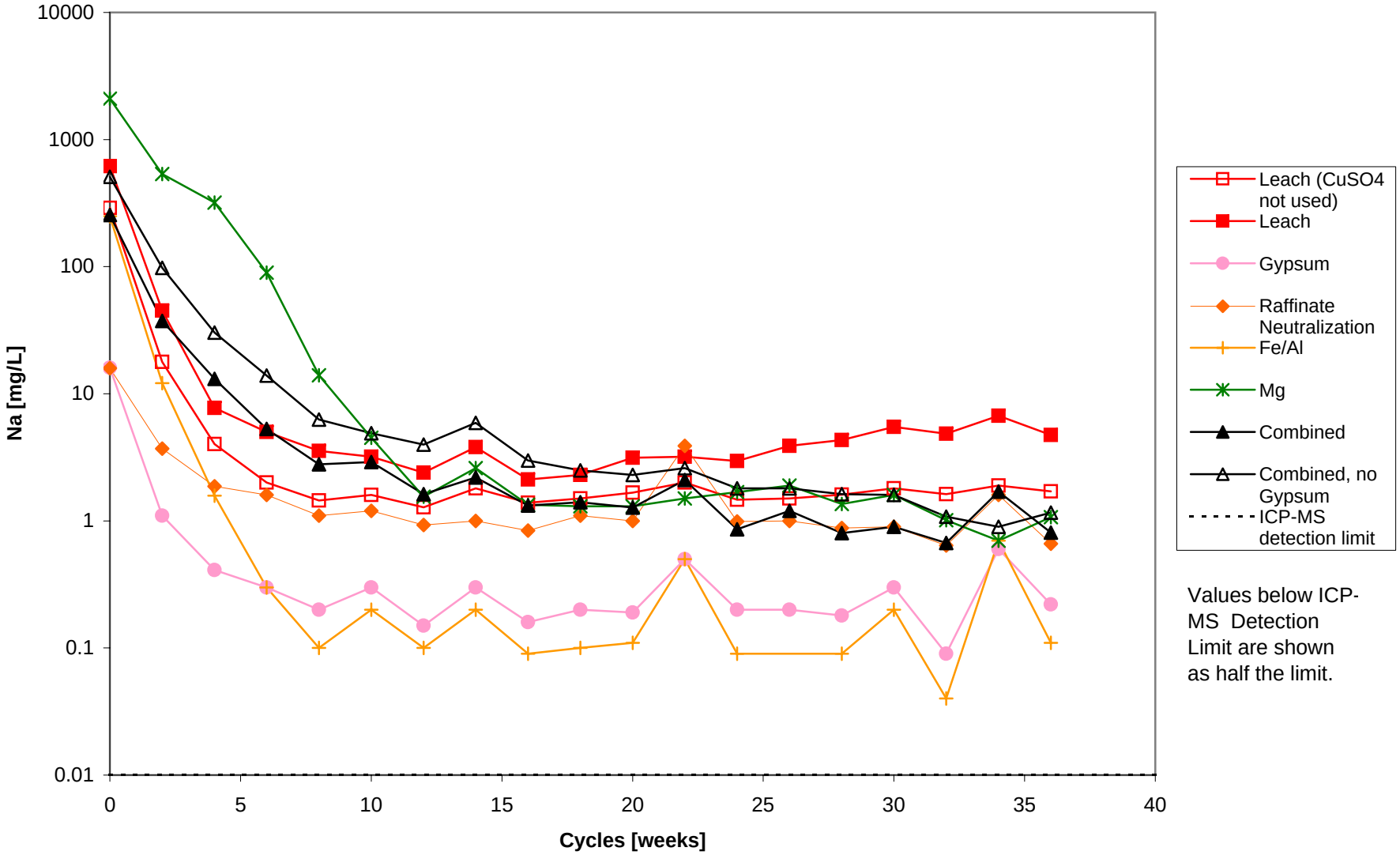
Values below ICP-MS Detection Limit are shown as half the limit.


Shake Flasks - Concentrations
NorthMet Project
Se [mg/L]
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

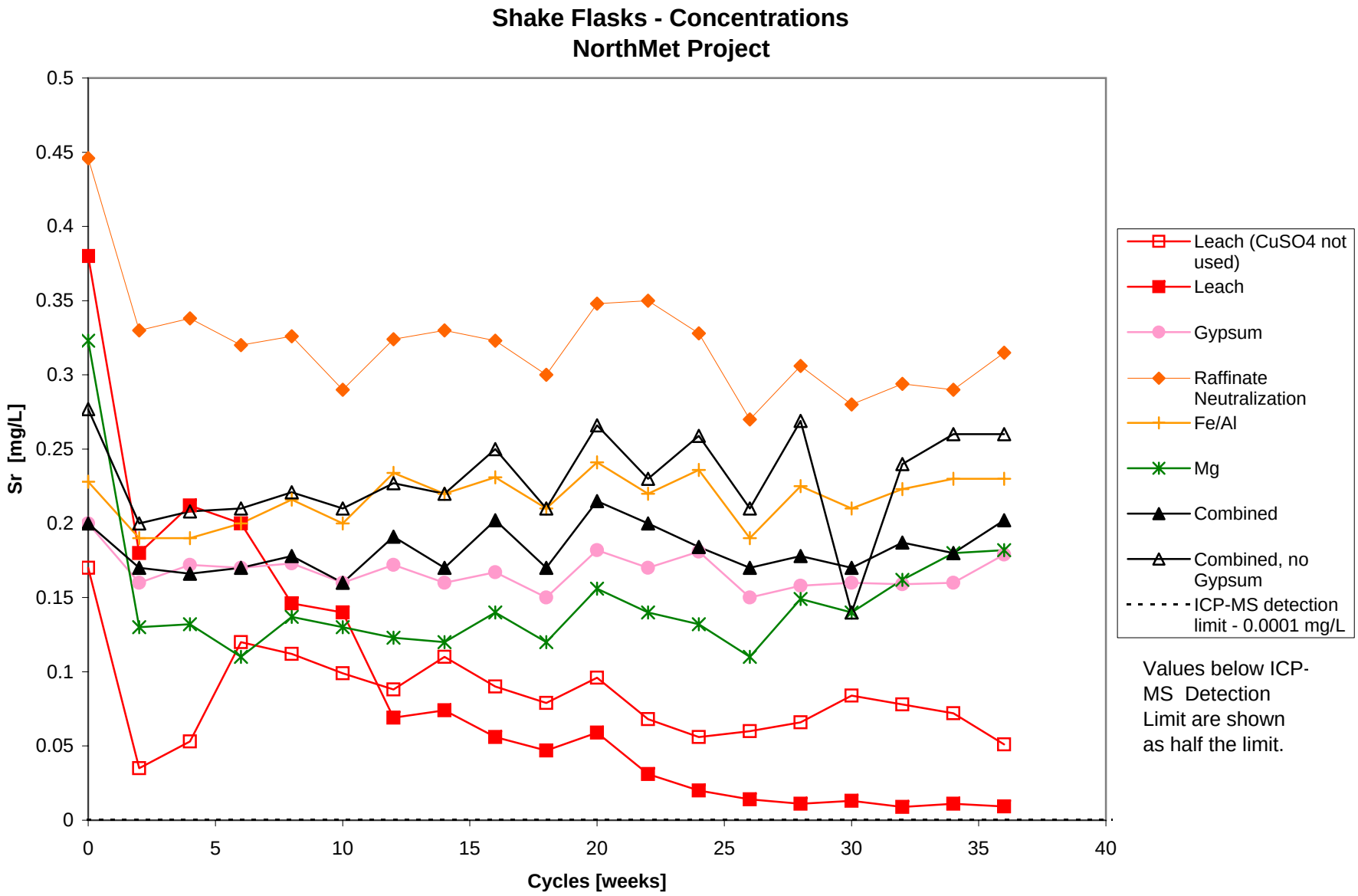

Shake Flasks - Concentrations
NorthMet Project
100
10
1
0.1
0.01
SiO2 [mg/L]
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.


Shake Flasks - Concentrations
NorthMet Project
Ag [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

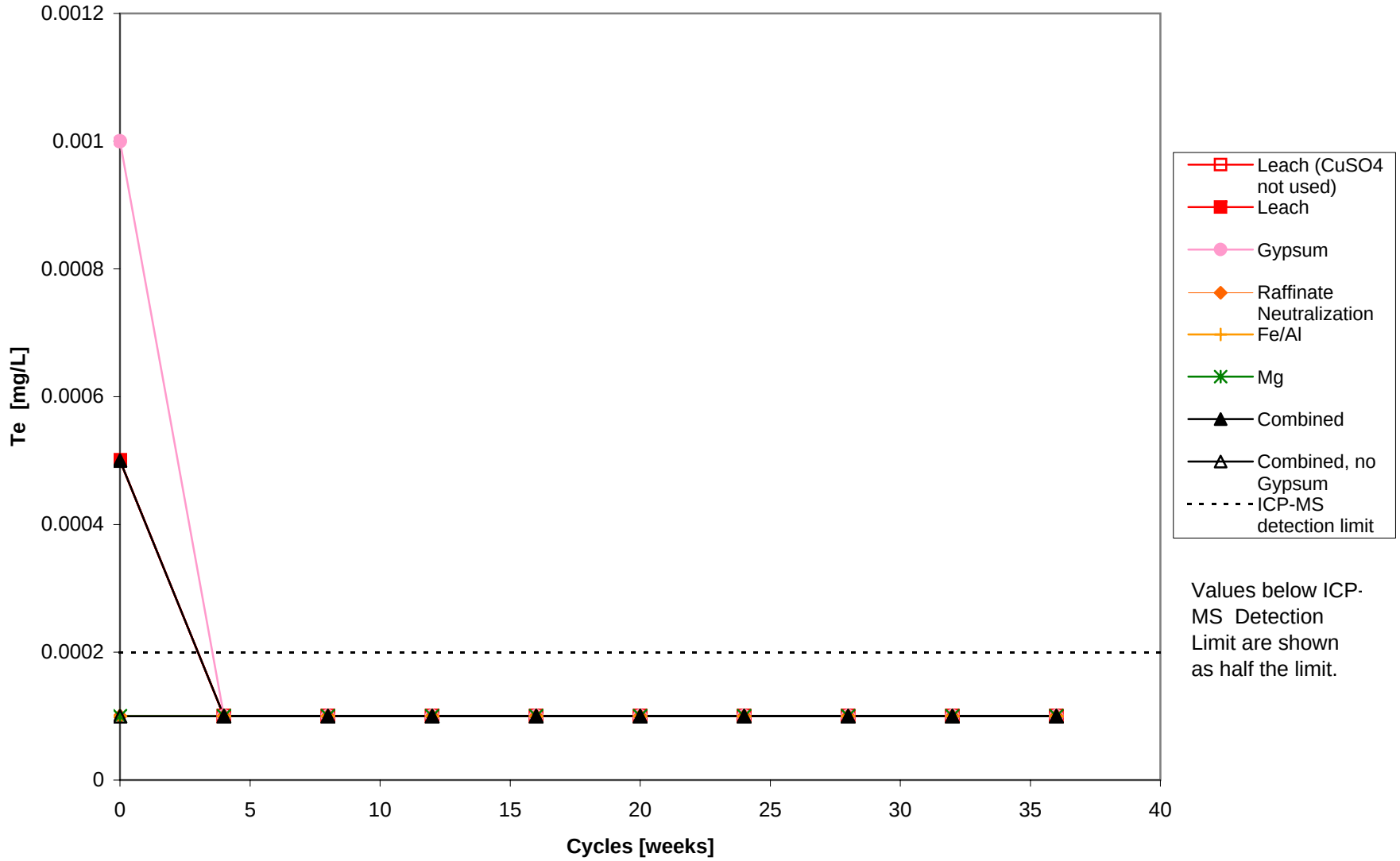
Shake Flasks - Concentrations
NorthMet Project



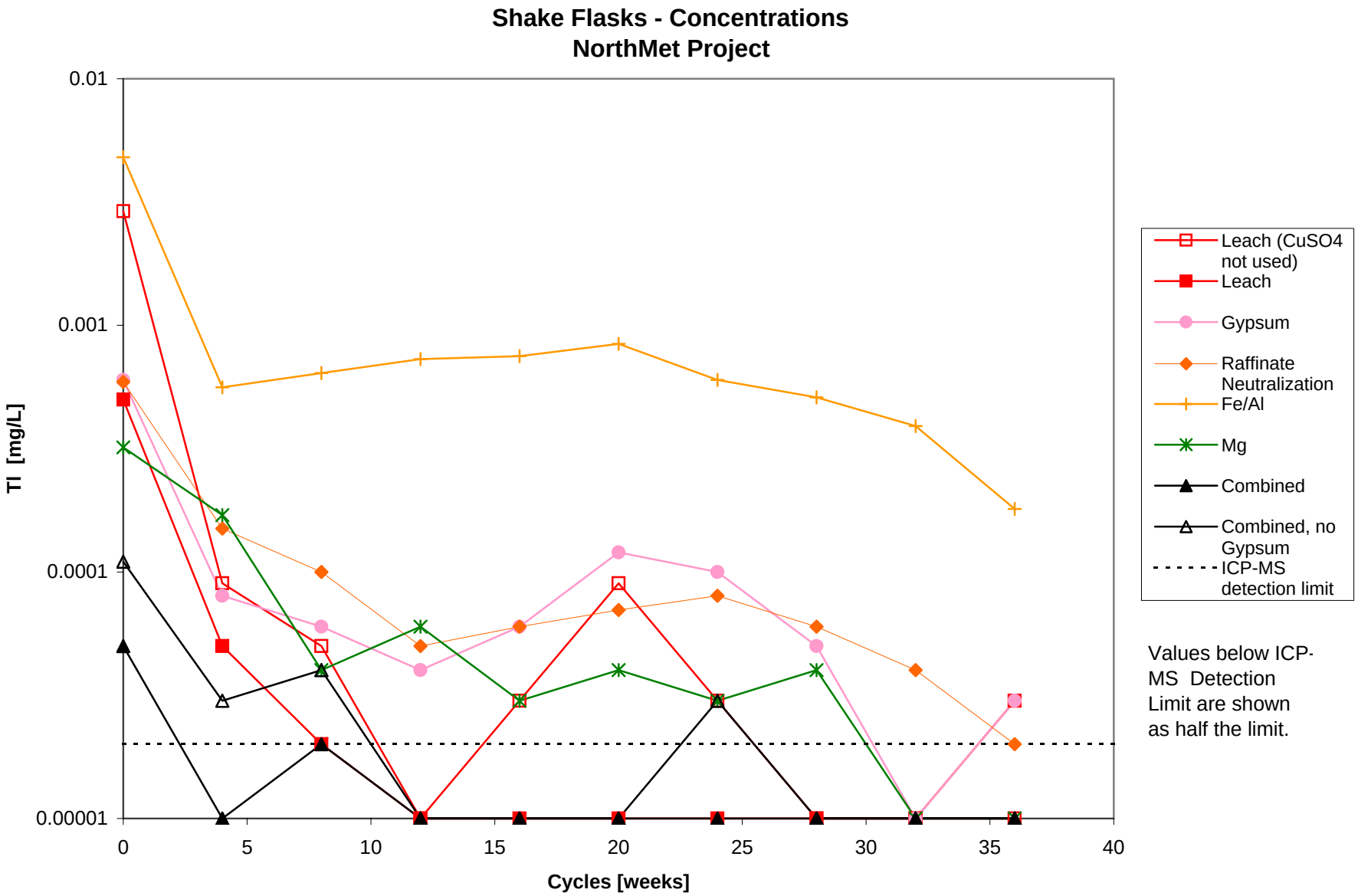
Values below ICP-MS Detection Limit are shown as half the limit.

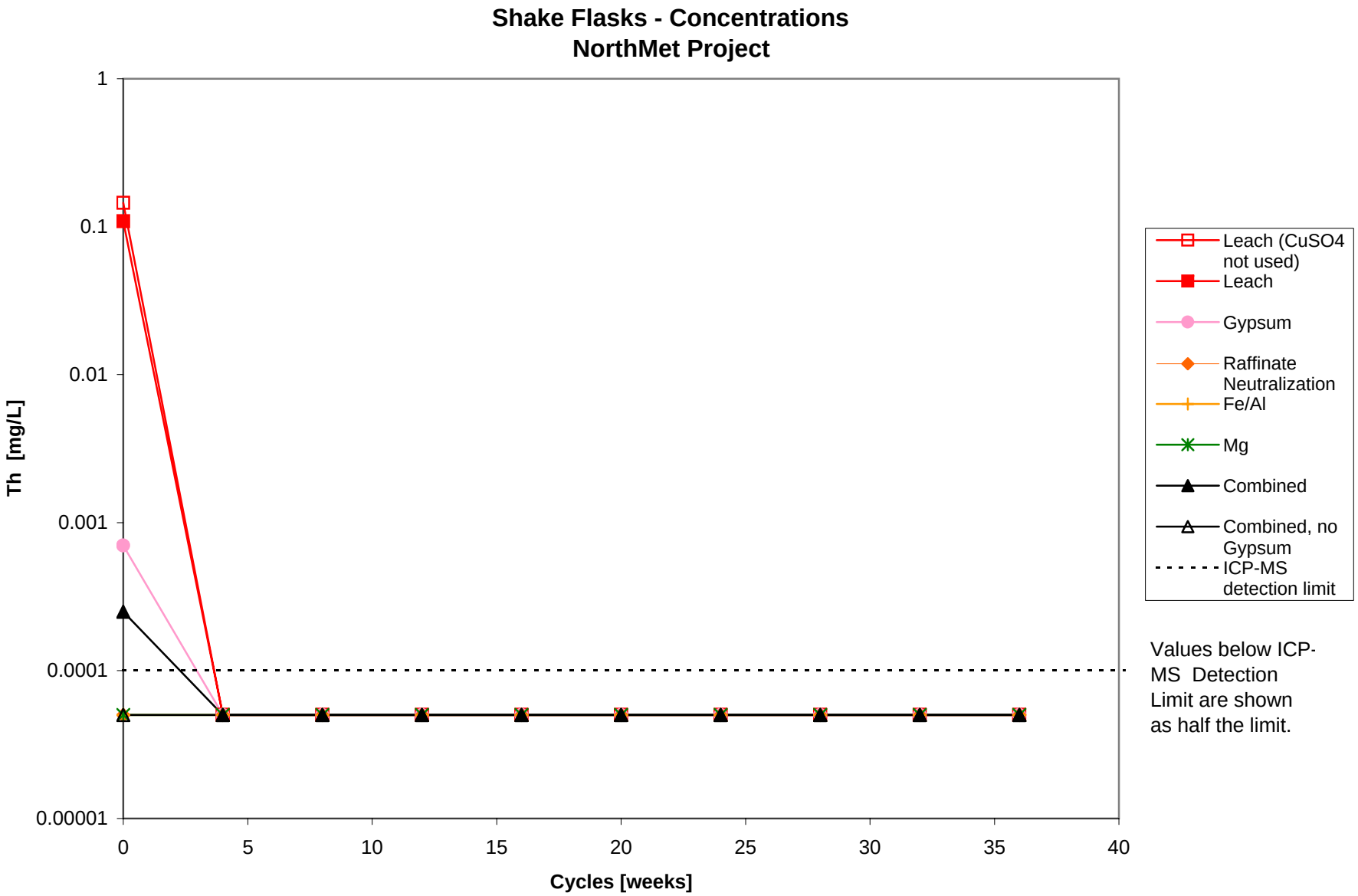

Shake Flasks - Concentrations
NorthMet Project
Sr [mg/L]
0.5
0.45
0.4
0.35
0.3
0.25
0.2
0.15
0.1
0.05
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit - 0.0001 mg/L
Values below ICP-MS Detection Limit are shown as half the limit.

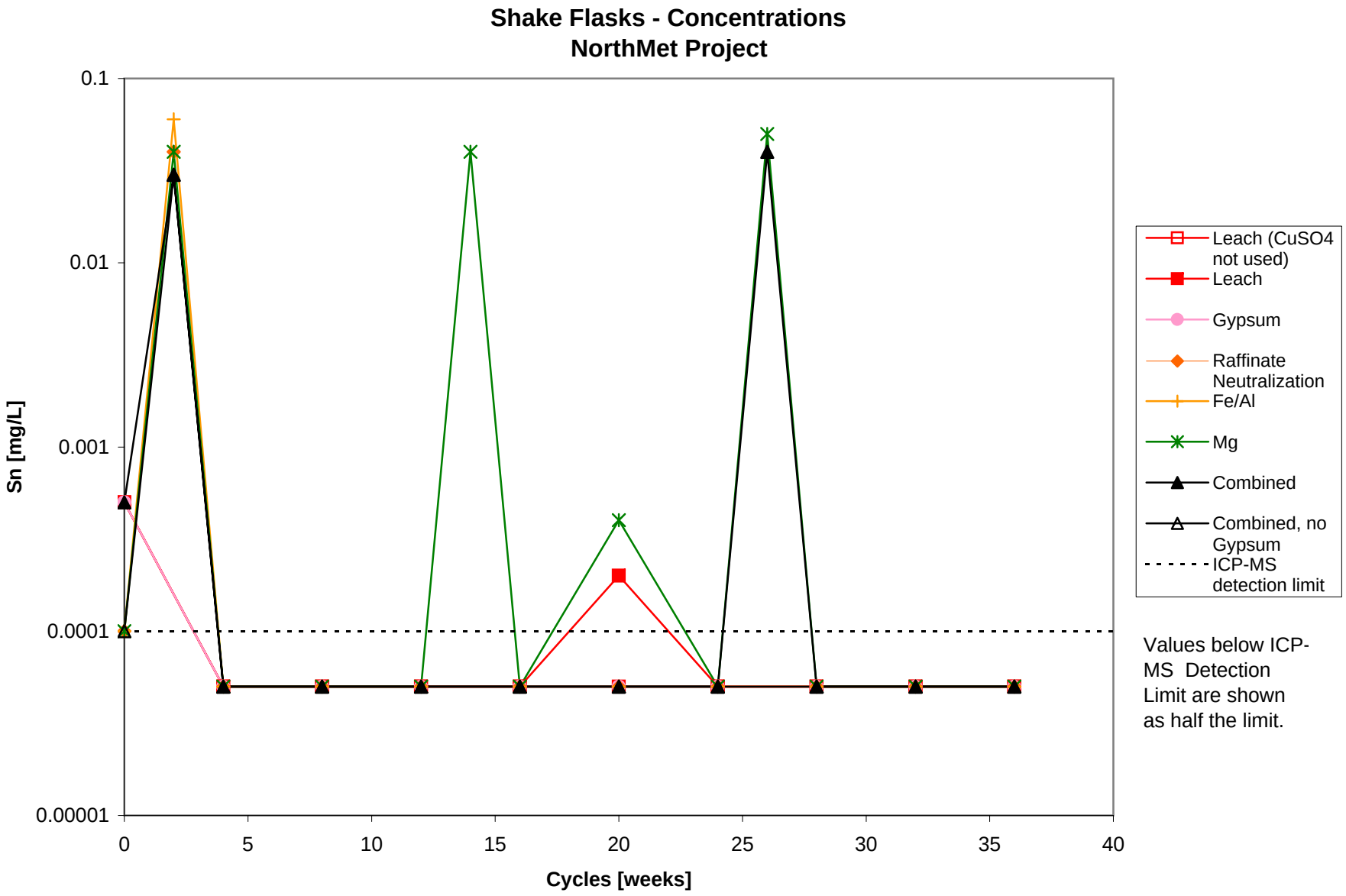
Shake Flasks - Concentrations
NorthMet Project

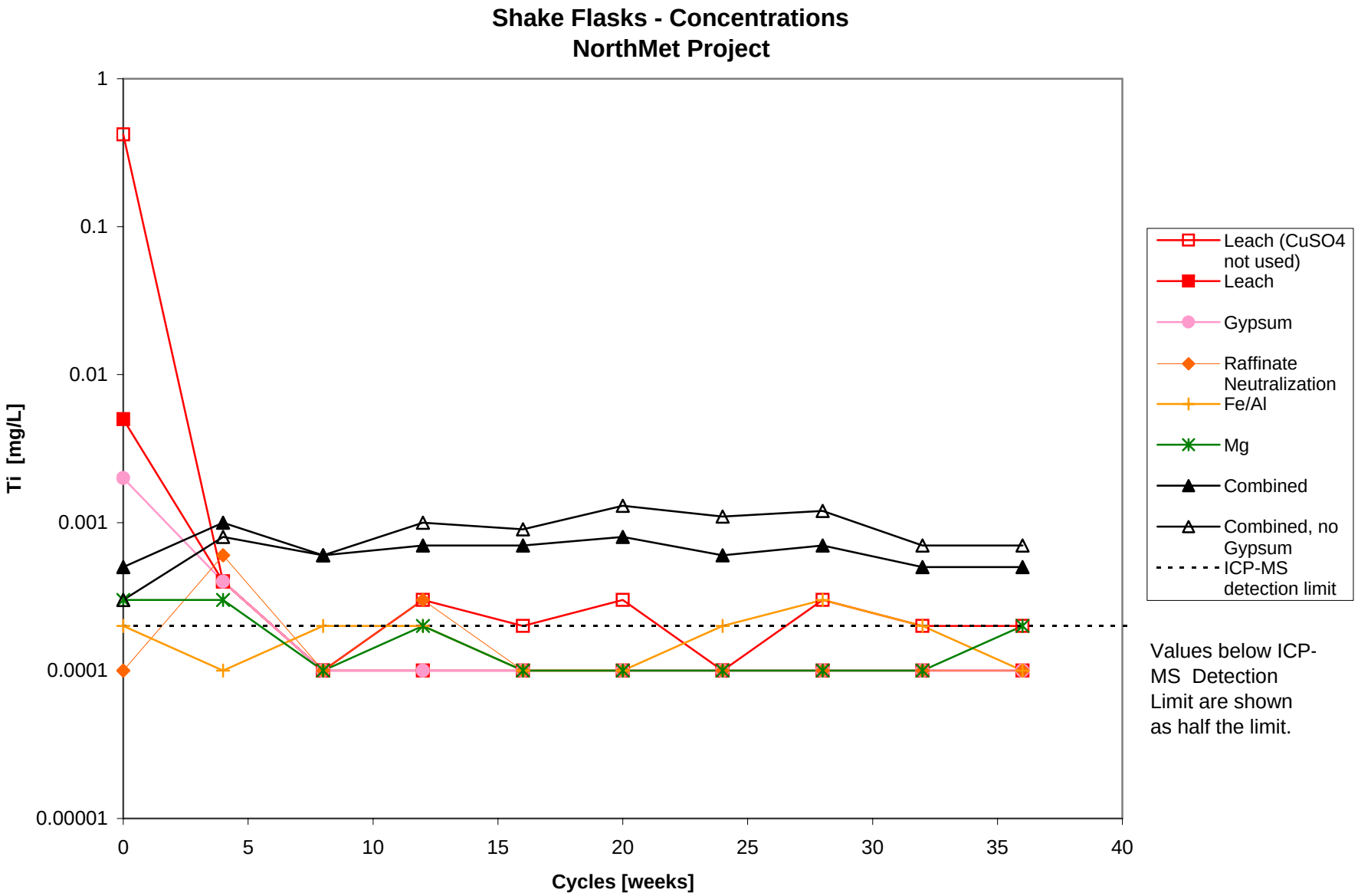


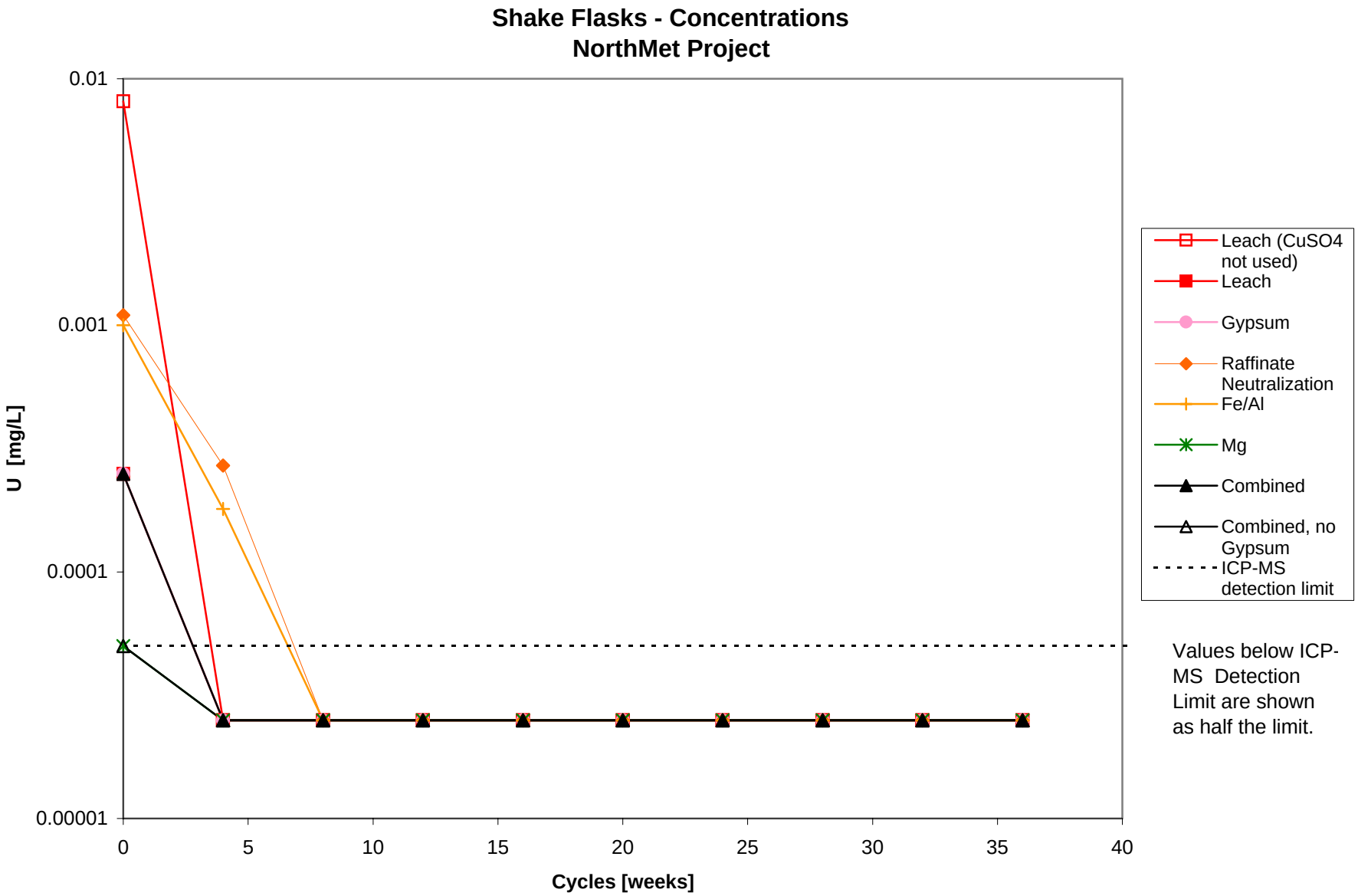
Values below ICP-MS Detection Limit are shown as half the limit.

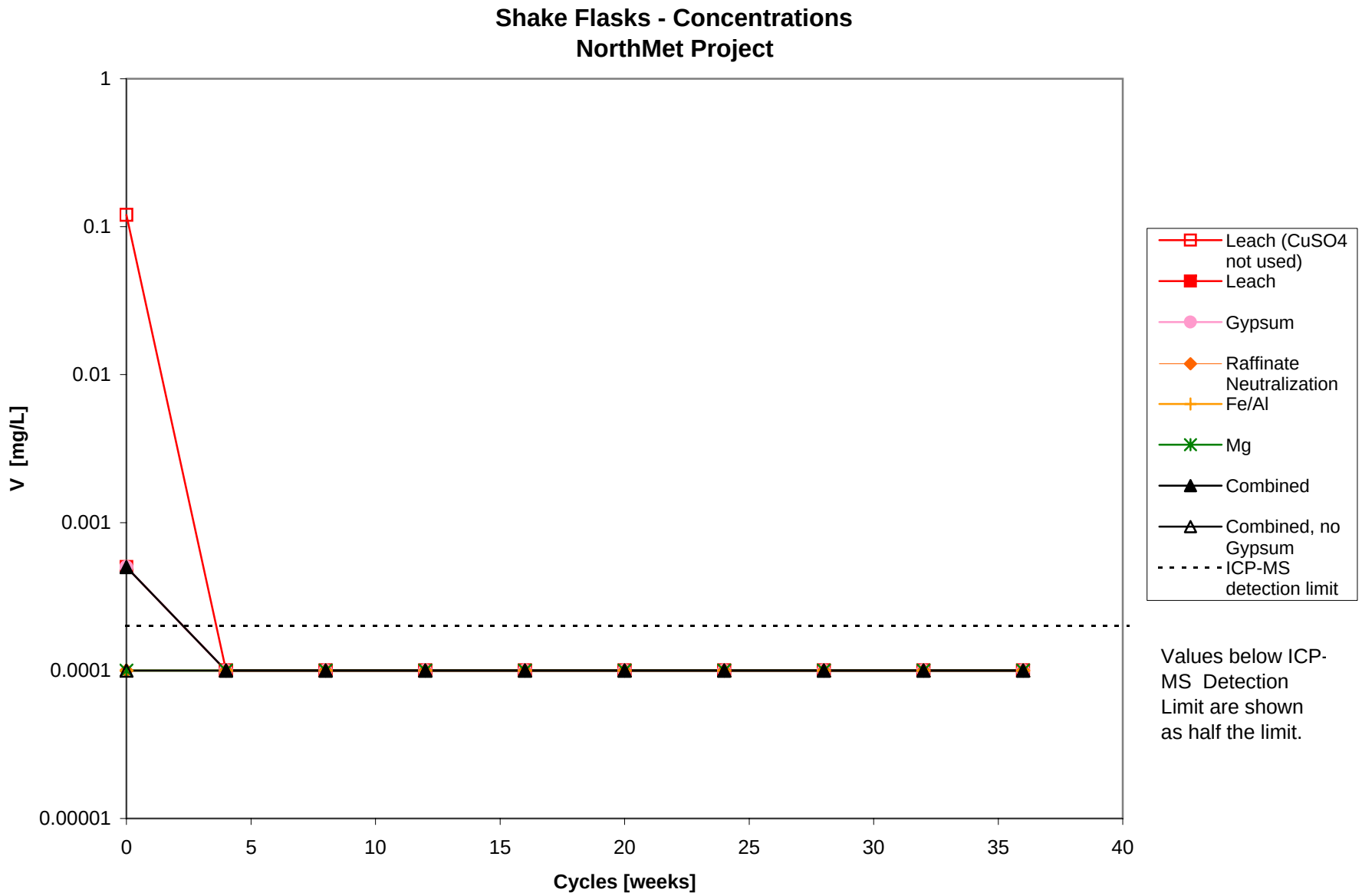

Shake Flasks - Concentrations
NorthMet Project
0.01
0.001
0.0001
0.00001
Tl [mg/L]
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.


Shake Flasks - Concentrations
NorthMet Project
Th [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

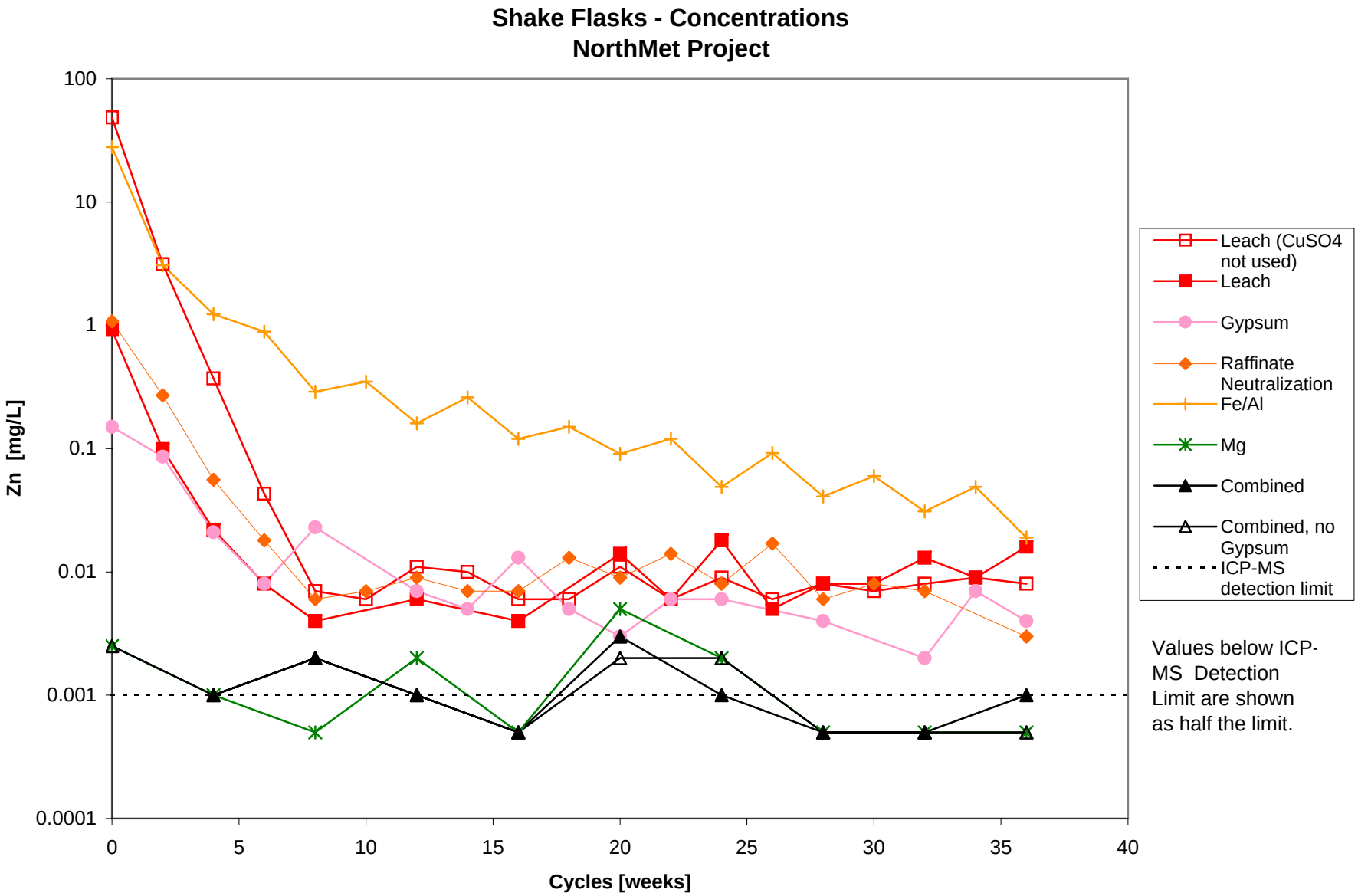

Shake Flasks - Concentrations
NorthMet Project
Sn [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

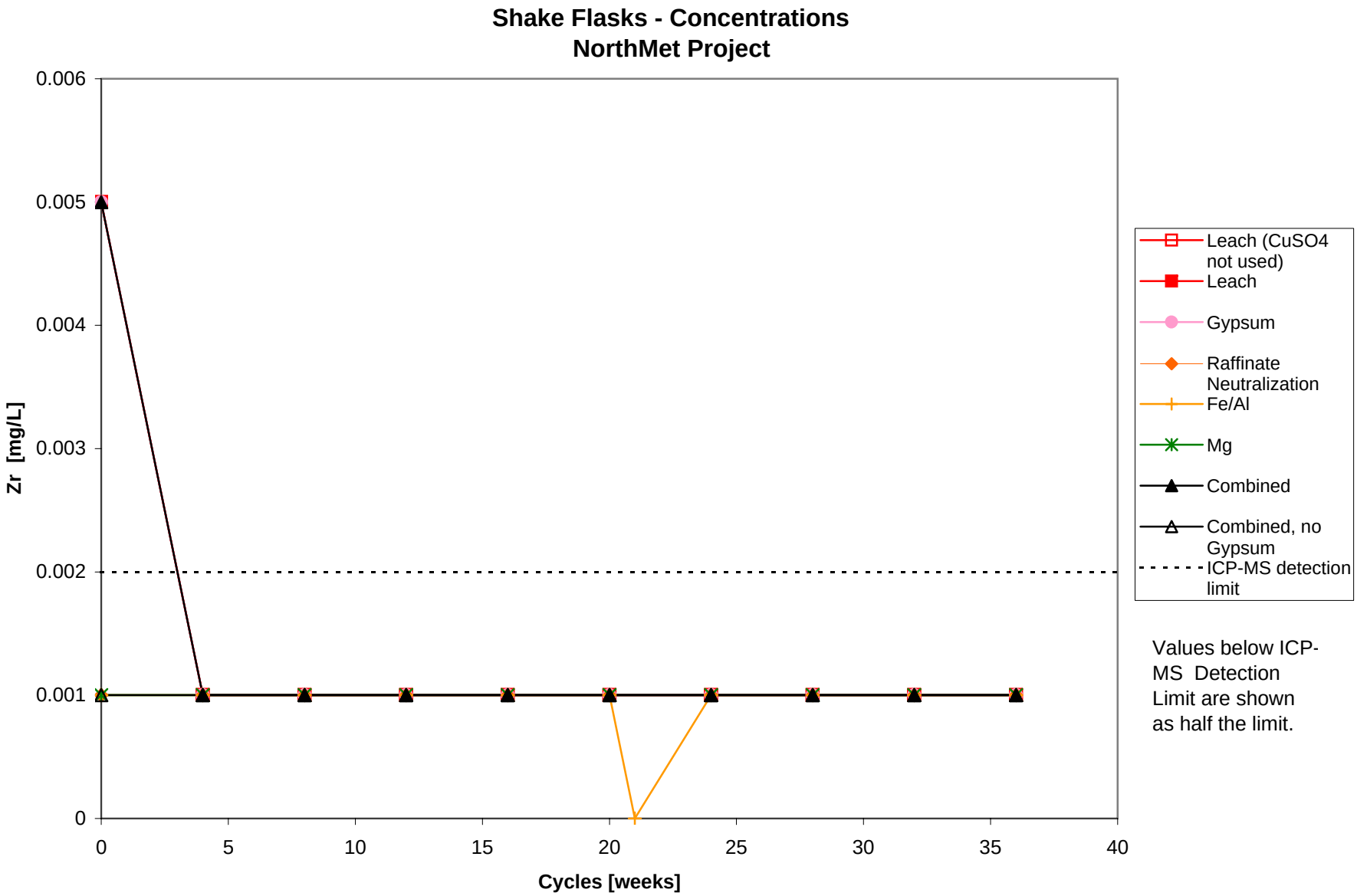

Shake Flasks - Concentrations
NorthMet Project
Ti [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.


Shake Flasks - Concentrations
NorthMet Project
U [mg/L]
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.



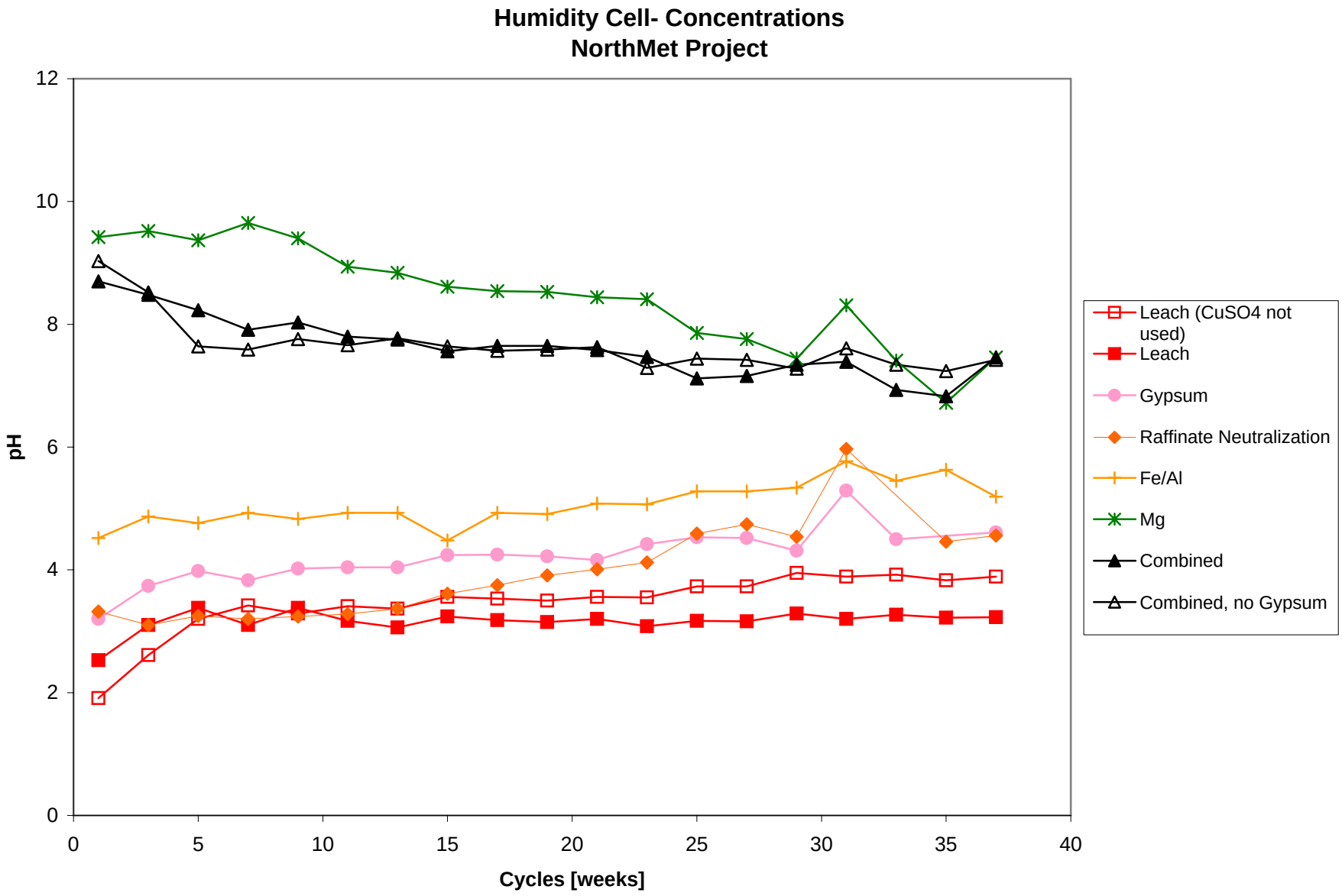
Values below ICP-MS Detection Limit are shown as half the limit.

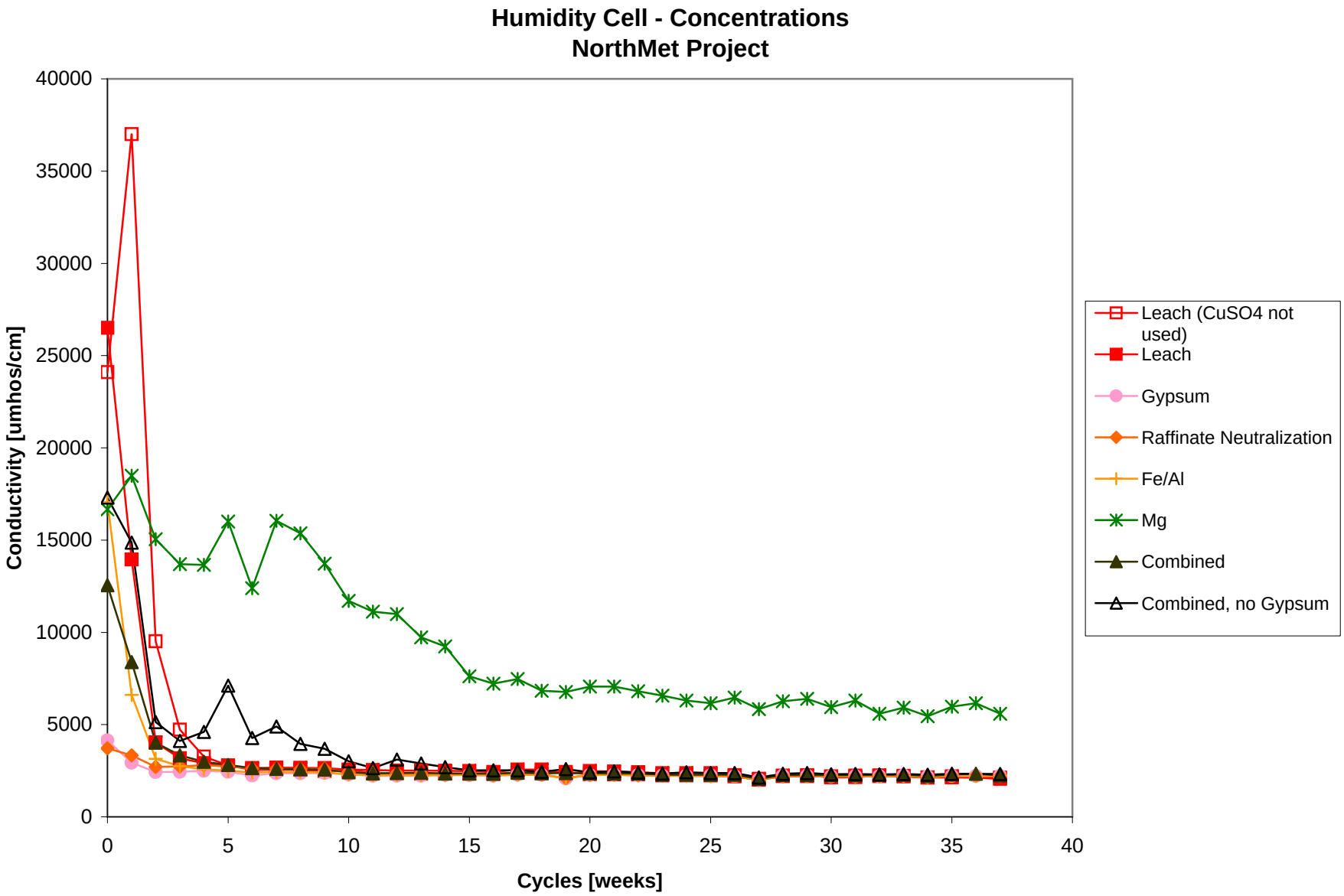

Shake Flasks - Concentrations
NorthMet Project
Zn [mg/L]
100
10
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

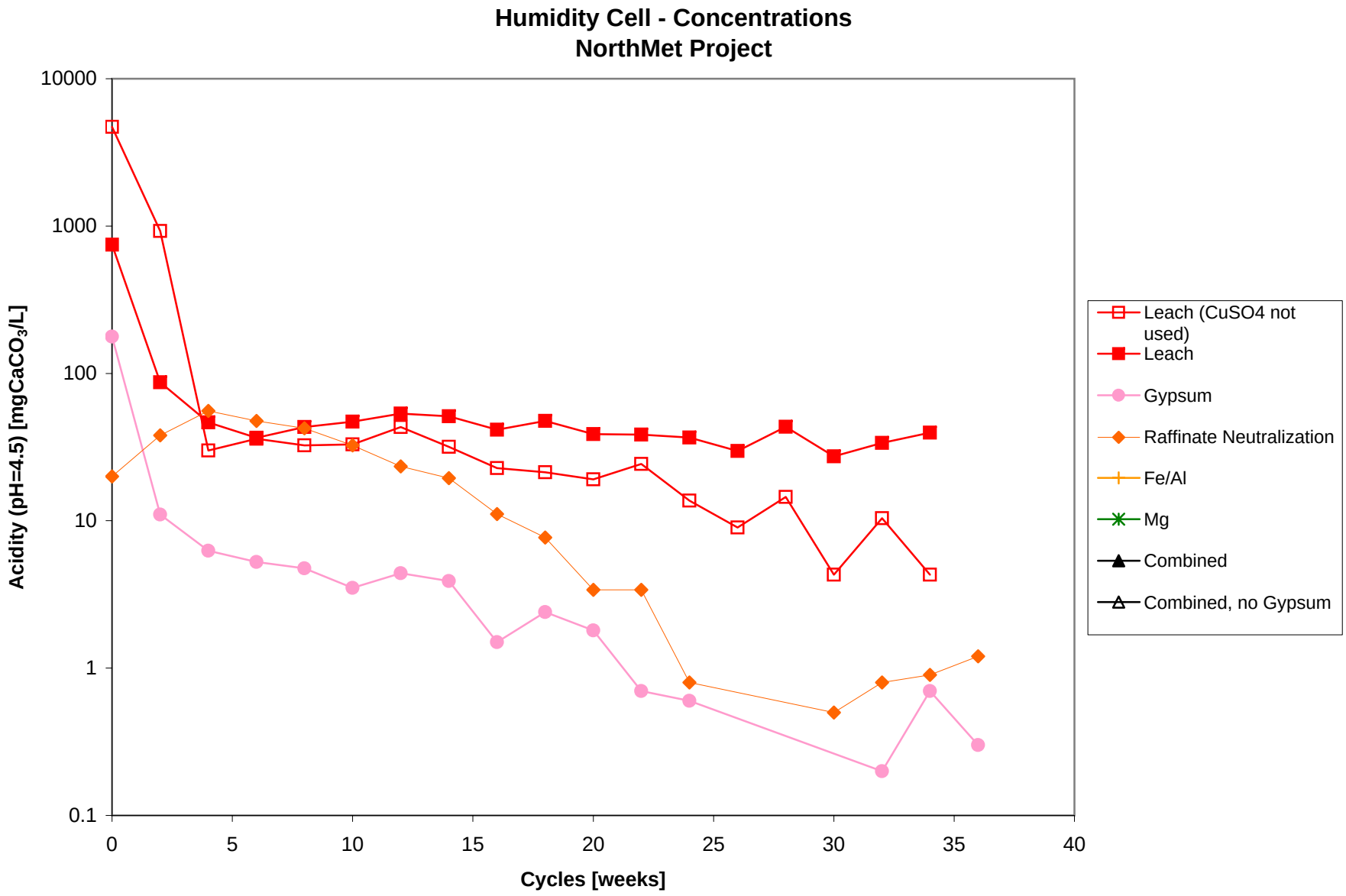

Shake Flasks - Concentrations
NorthMet Project
Zr [mg/L]
0.006
0.005
0.004
0.003
0.002
0.001
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

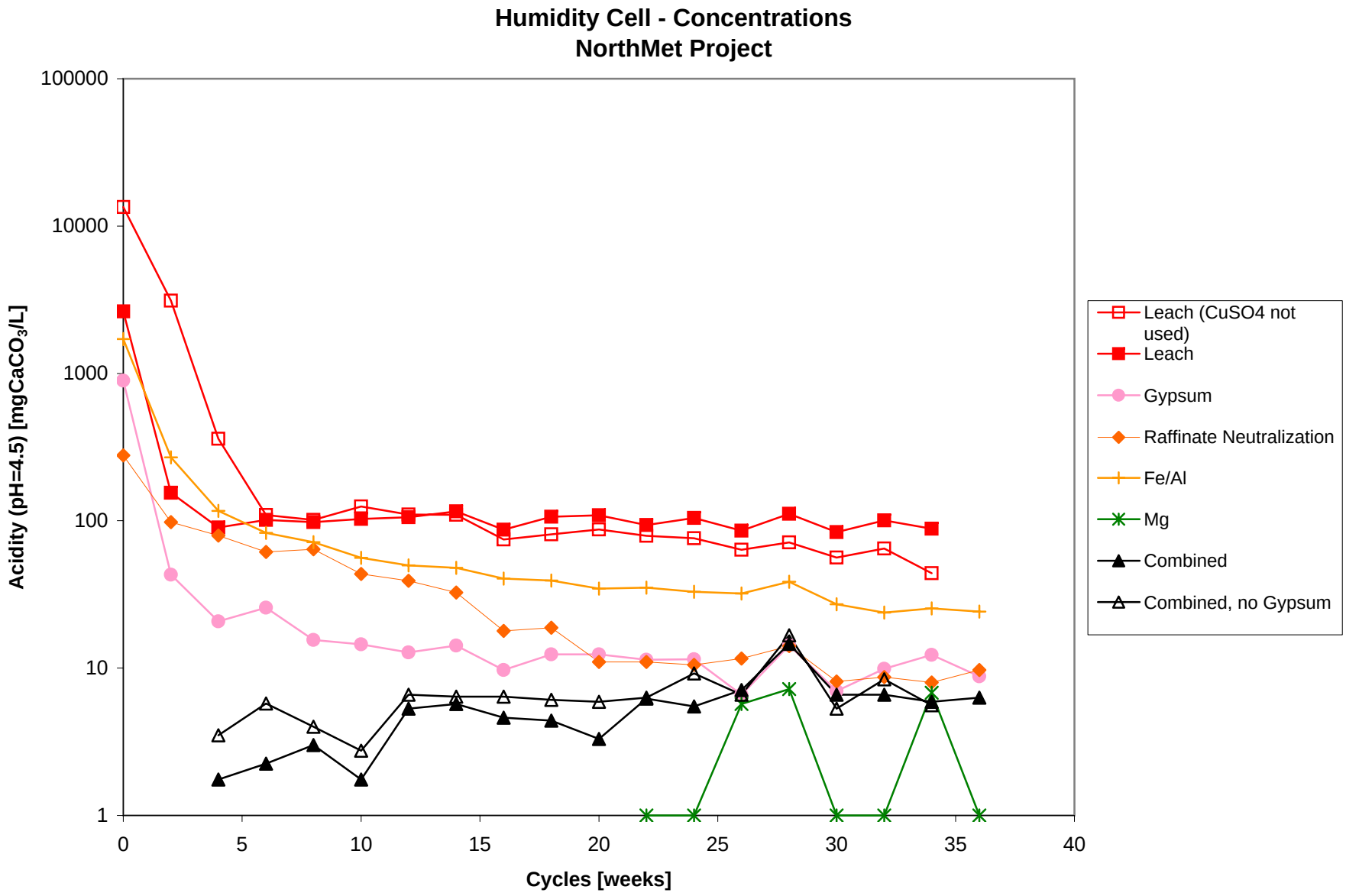
Appendix C.2

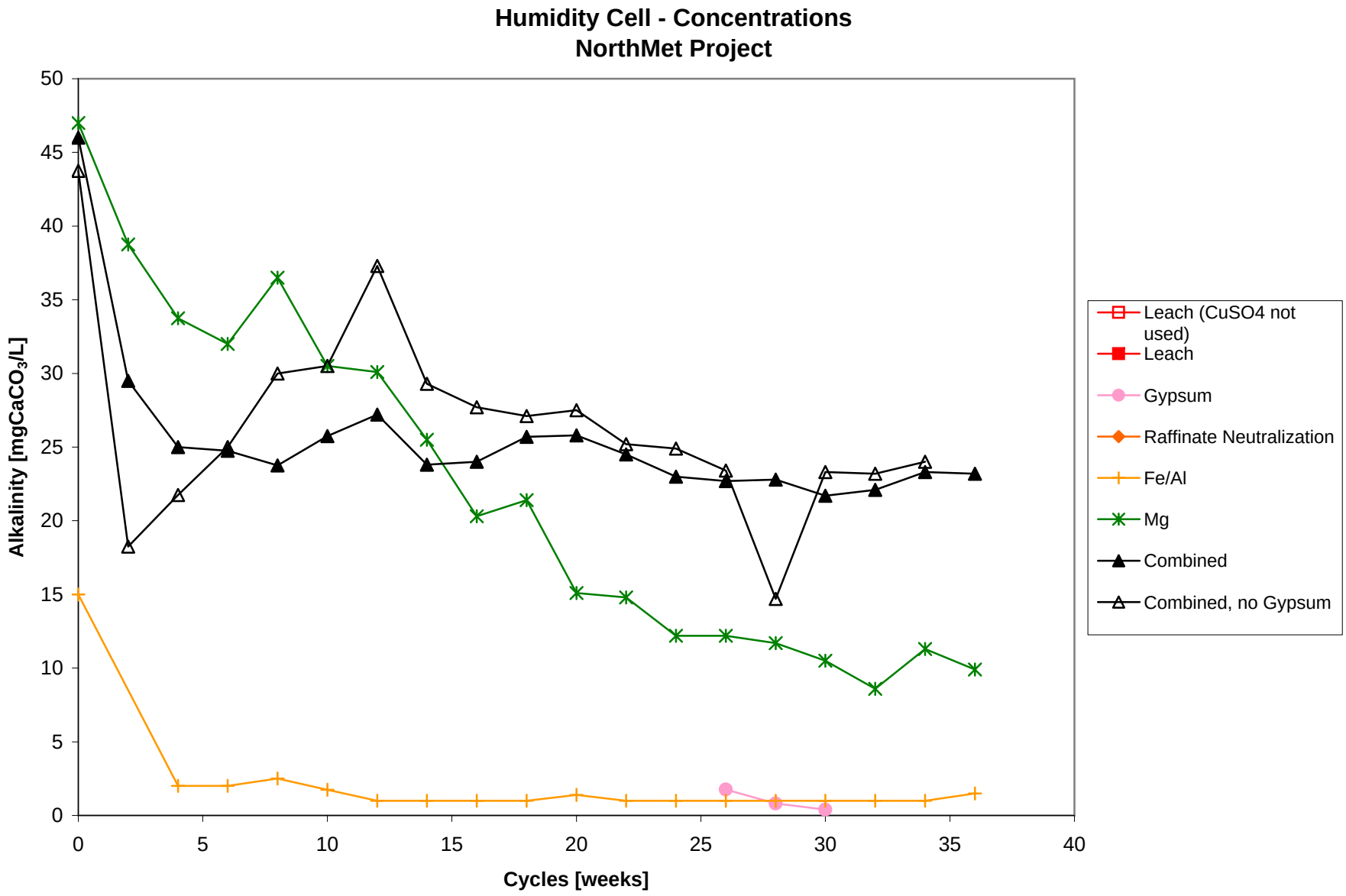
Humidity Cells

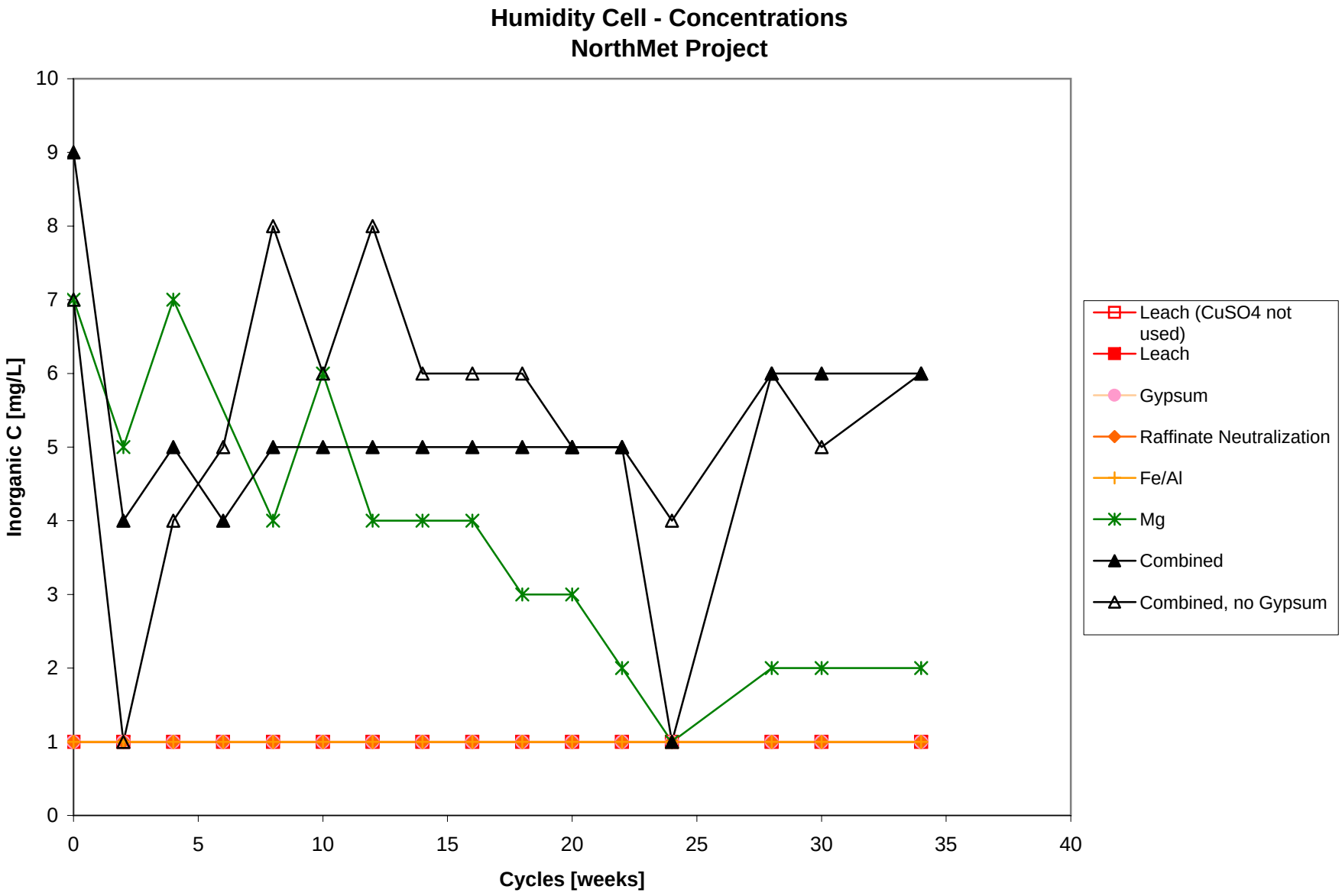

Humidity Cell- Concentrations
NorthMet Project
pH
Cycles [weeks]
0
2
4
6
8
10
12
0
5
10
15
20
25
30
35
40
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

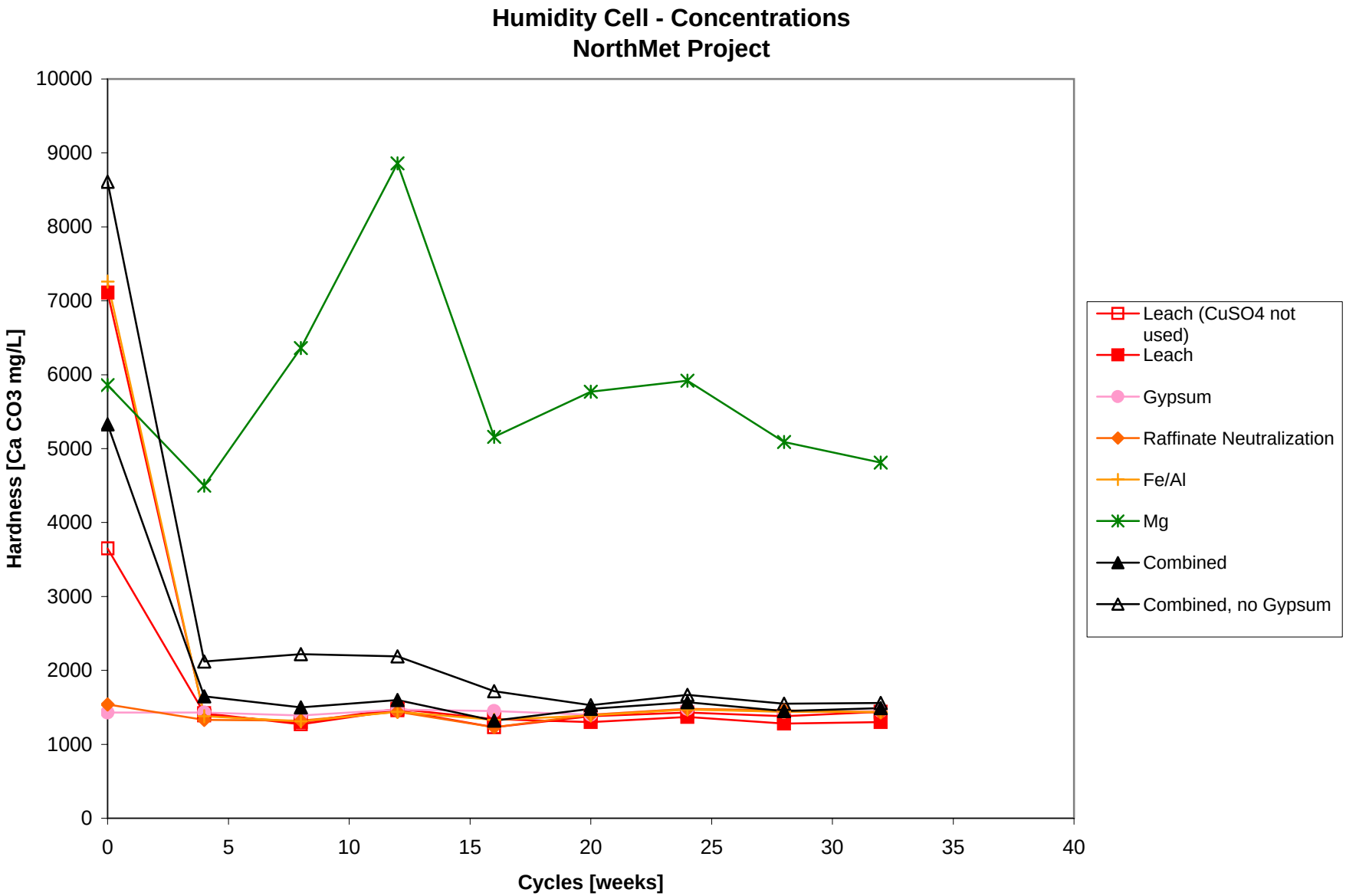

Humidity Cell - Concentrations
NorthMet Project
Conductivity [umhos/cm]
Cycles [weeks]
0
5000
10000
15000
20000
25000
30000
35000
40000
0
5
10
15
20
25
30
35
40
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

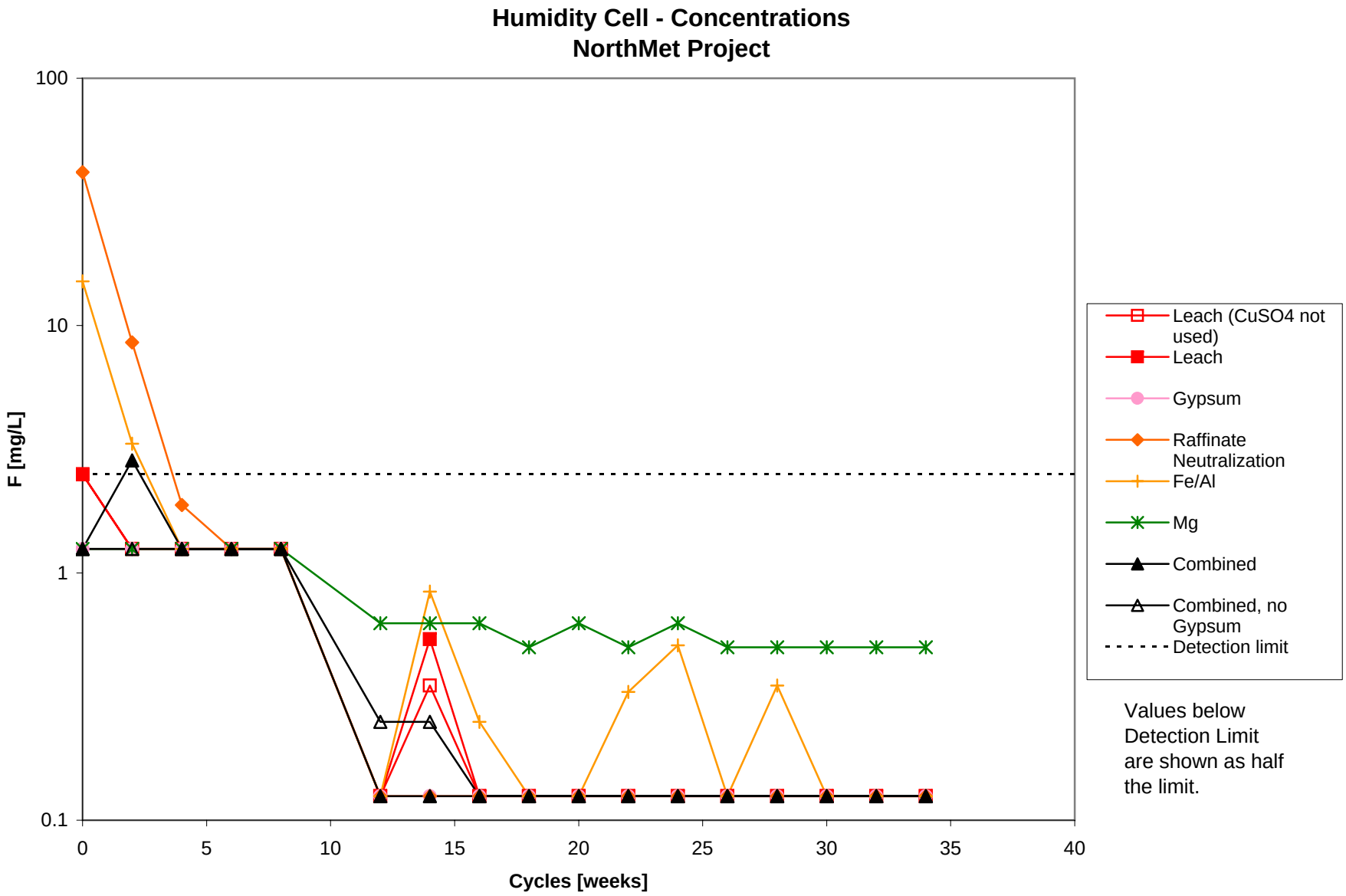

Humidity Cell - Concentrations
NorthMet Project
Acidity (pH=4.5) [mgCaCO3/L]
10000
1000
100
10
1
0.1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

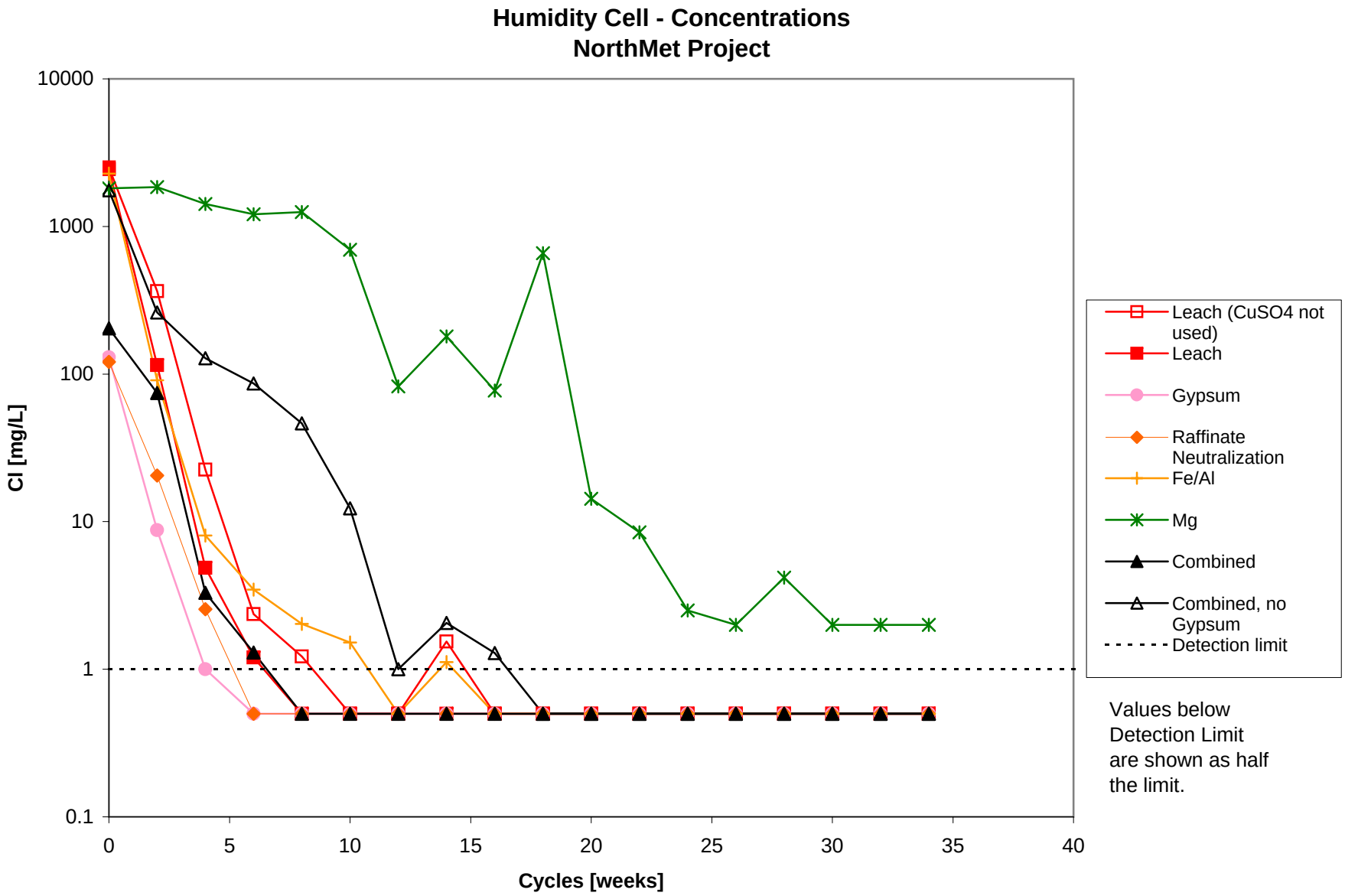

Humidity Cell - Concentrations
NorthMet Project
Acidity (pH=4.5) [mgCaCO3/L]
100000
10000
1000
100
10
1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

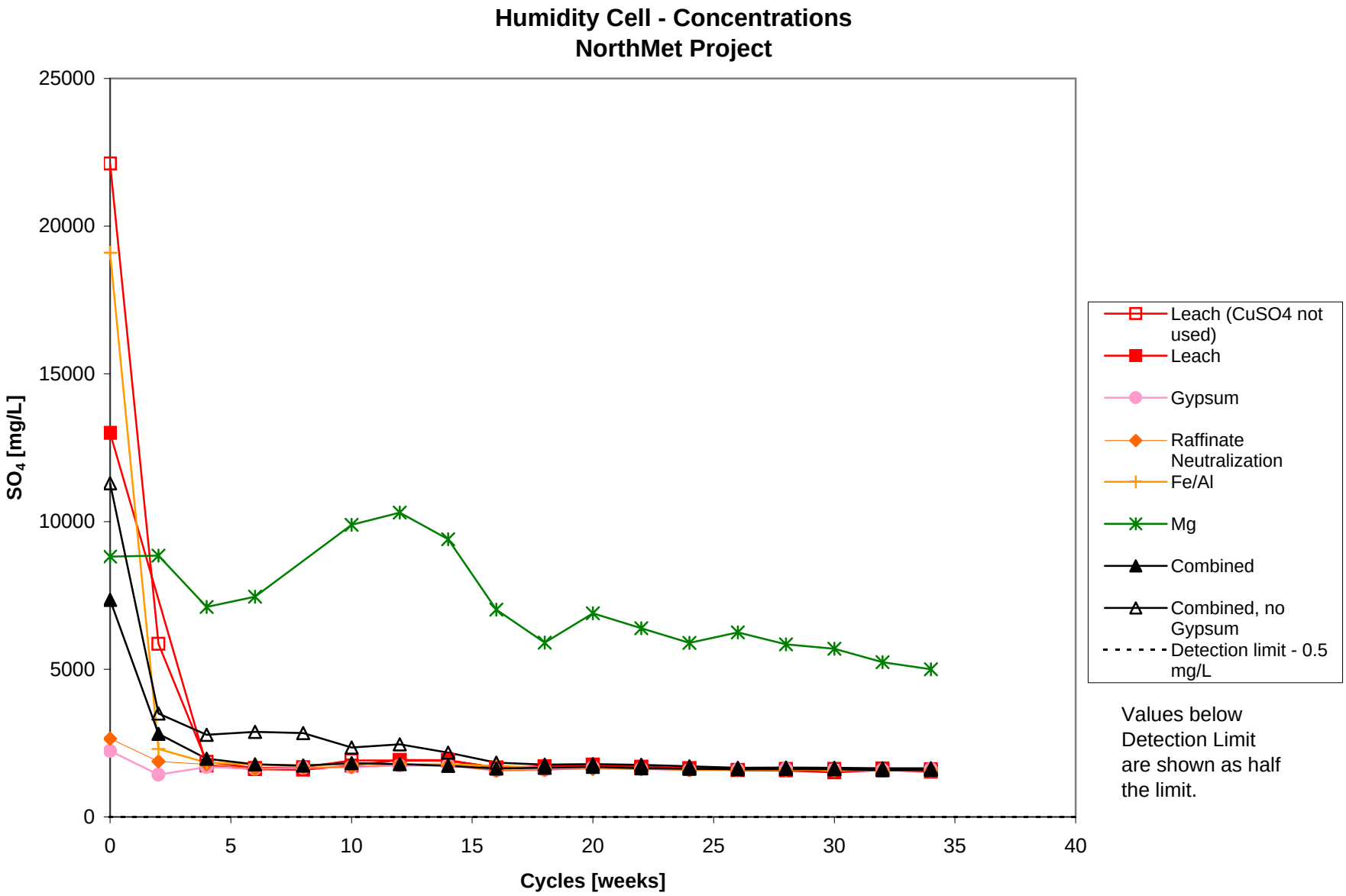

Humidity Cell - Concentrations
NorthMet Project
Alkalinity [mgCaCO3/L]
Cycles [weeks]
0
5
10
15
20
25
30
35
40
45
50
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

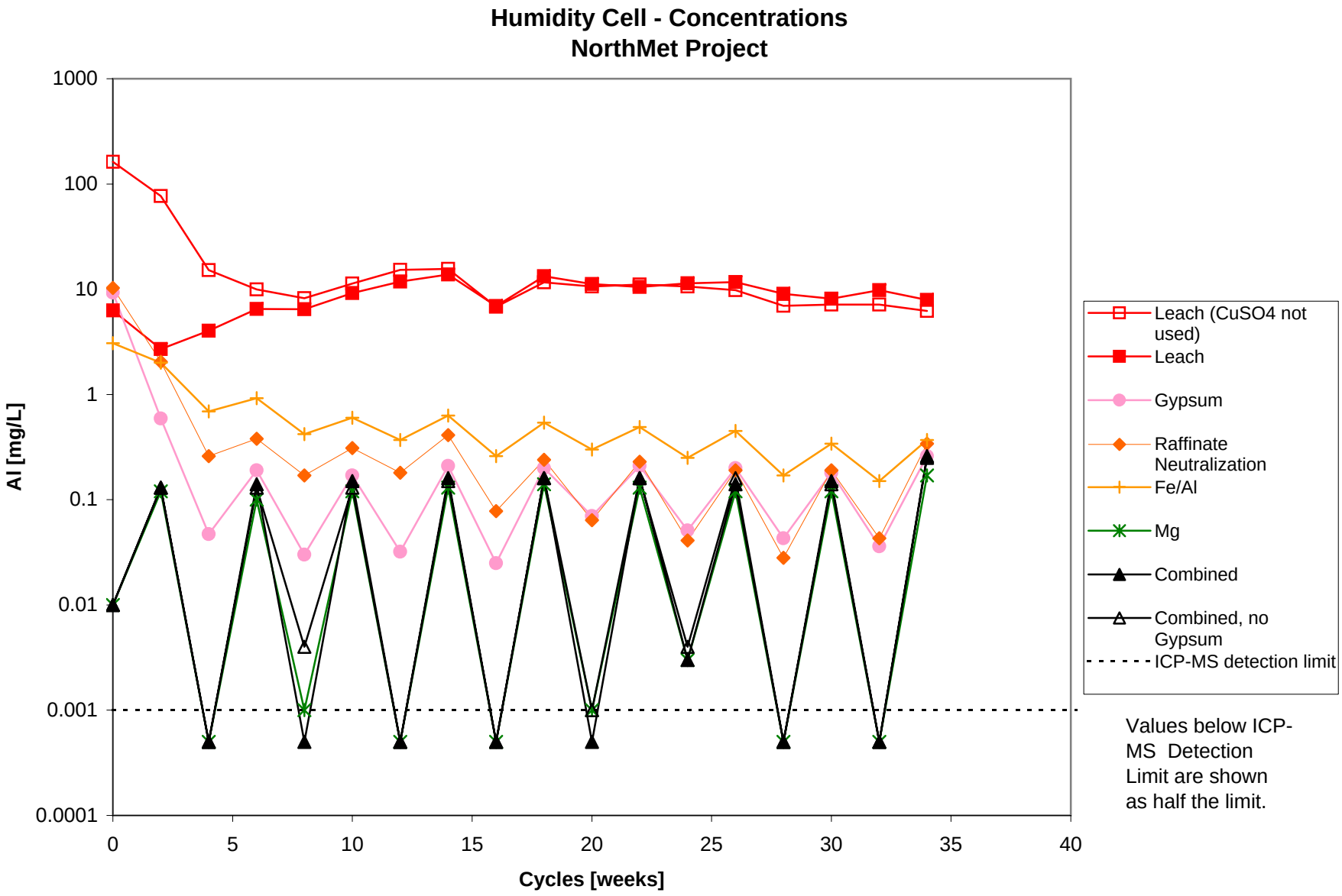

Humidity Cell - Concentrations
NorthMet Project
Inorganic C [mg/L]
Cycles [weeks]
0
1
2
3
4
5
6
7
8
9
10
0
5
10
15
20
25
30
35
40
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

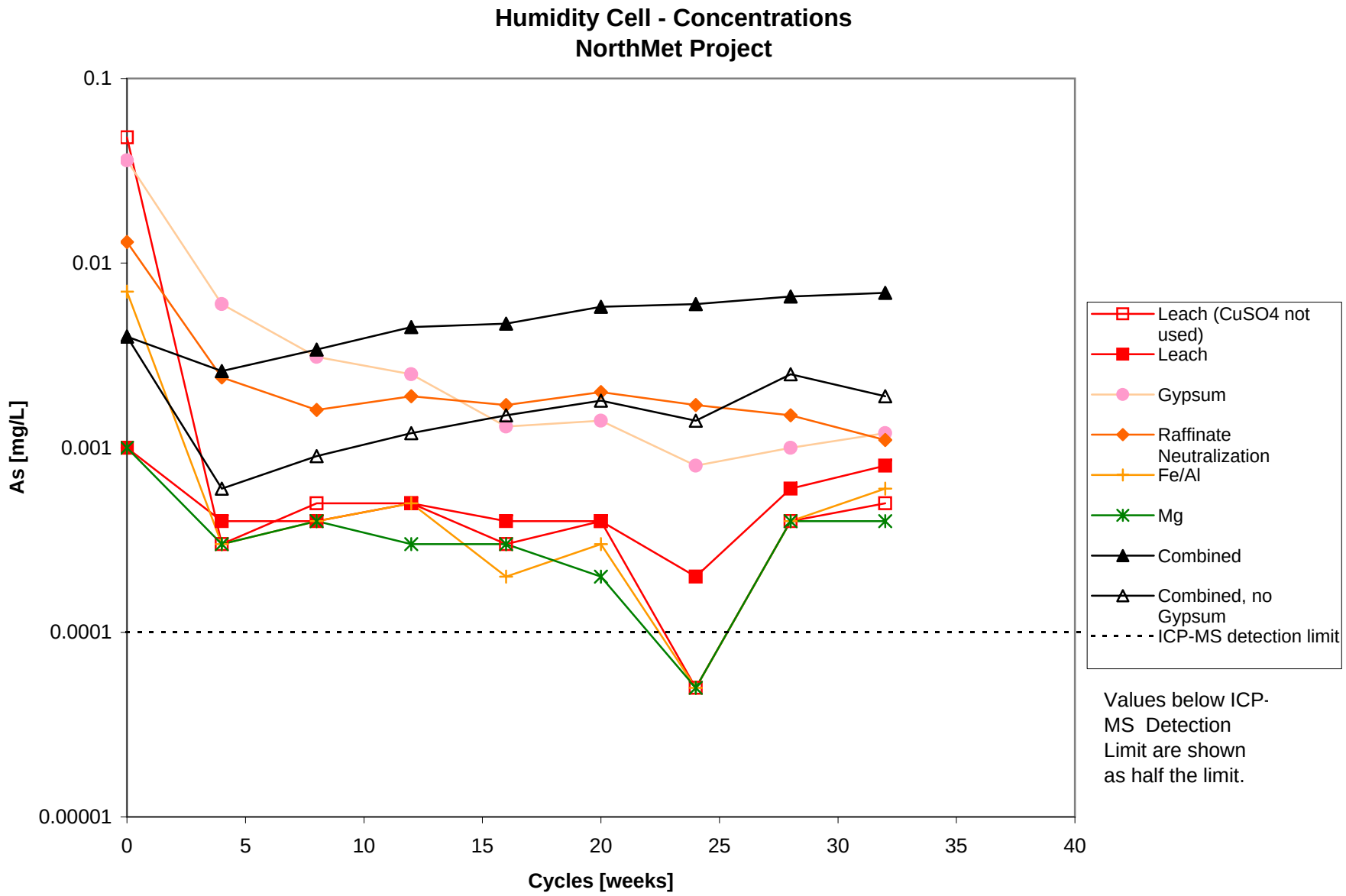

Humidity Cell - Concentrations
NorthMet Project
Hardness [Ca CO3 mg/L]
Cycles [weeks]
0
1000
2000
3000
4000
5000
6000
7000
8000
9000
10000
0
5
10
15
20
25
30
35
40
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

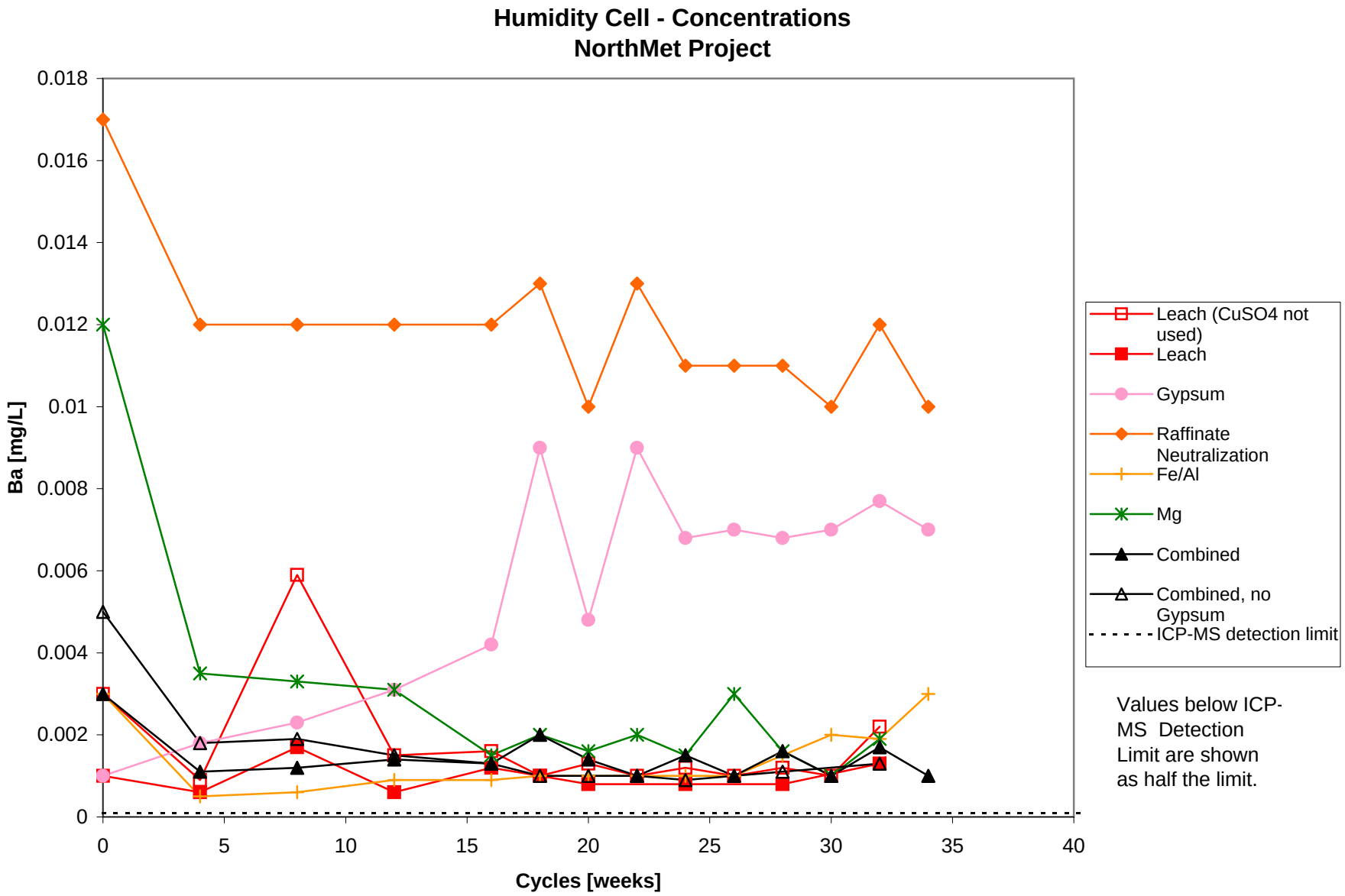

Humidity Cell - Concentrations
NorthMet Project
F [mg/L]
100
10
1
0.1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
Detection limit
Values below Detection Limit are shown as half the limit.

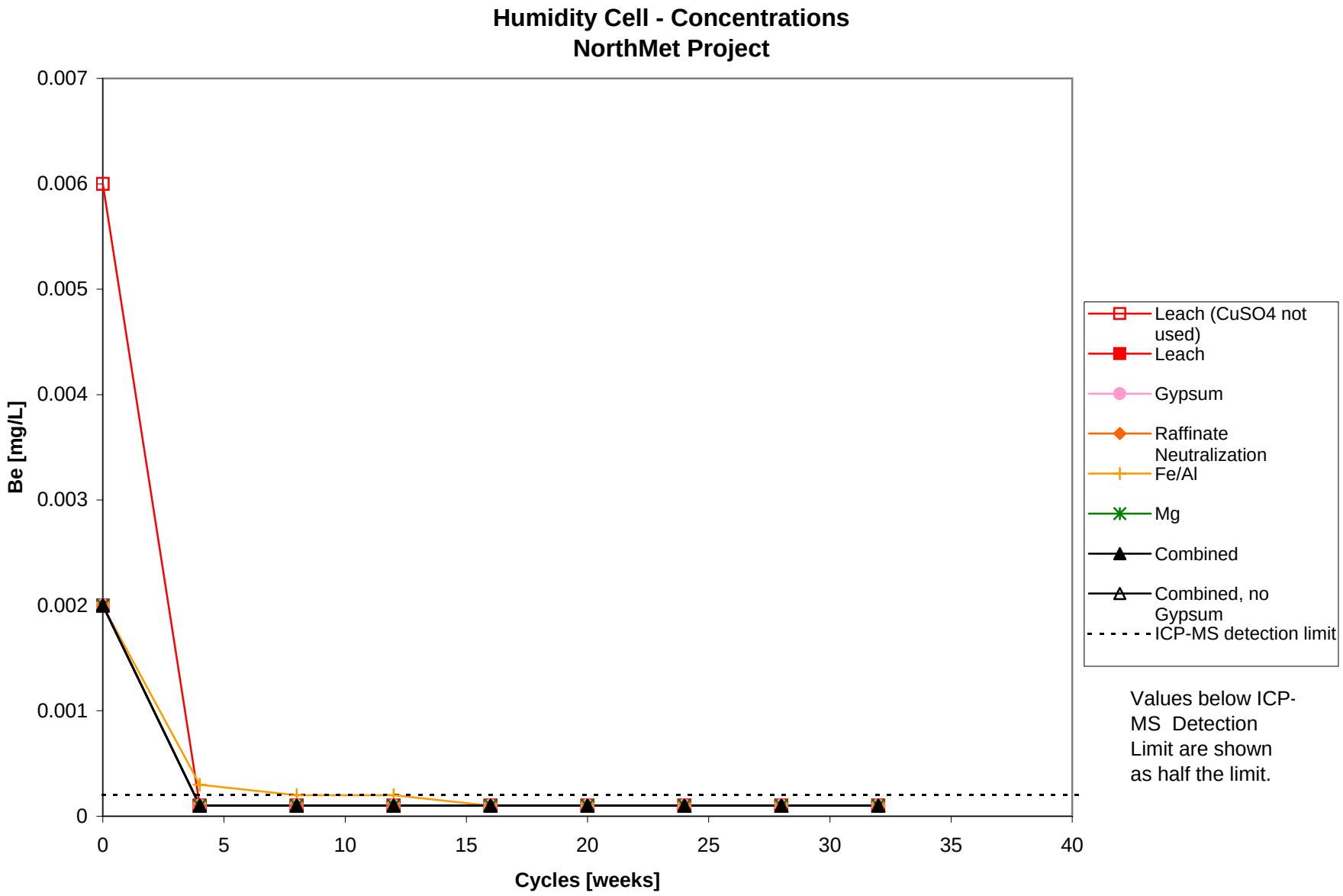

Humidity Cell - Concentrations
NorthMet Project
Cl [mg/L]
10000
1000
100
10
1
0.1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
Detection limit
Values below Detection Limit are shown as half the limit.

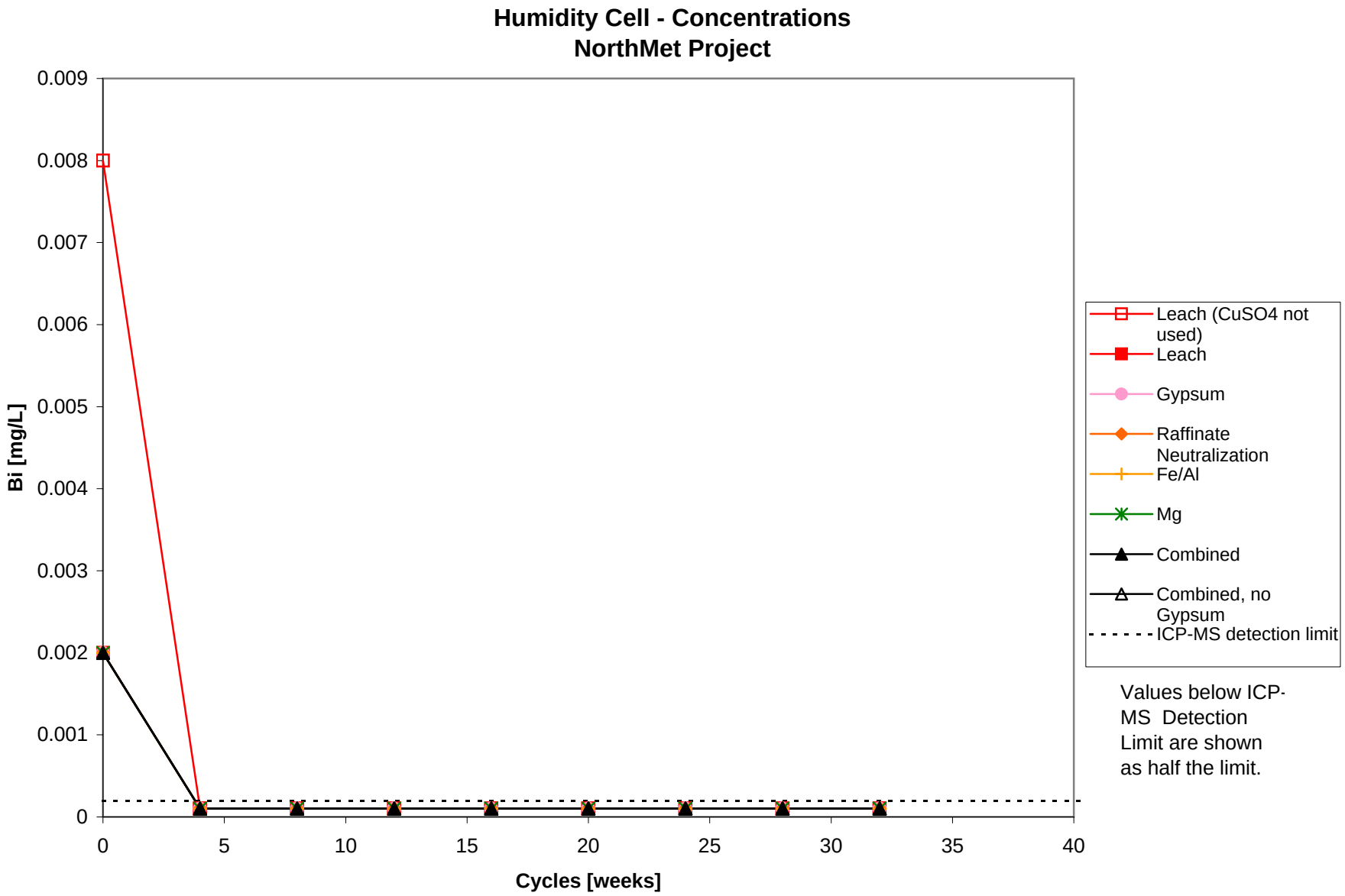

Humidity Cell - Concentrations
NorthMet Project
SO4 [mg/L]
25000
20000
15000
10000
5000
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
Detection limit - 0.5 mg/L
Values below Detection Limit are shown as half the limit.

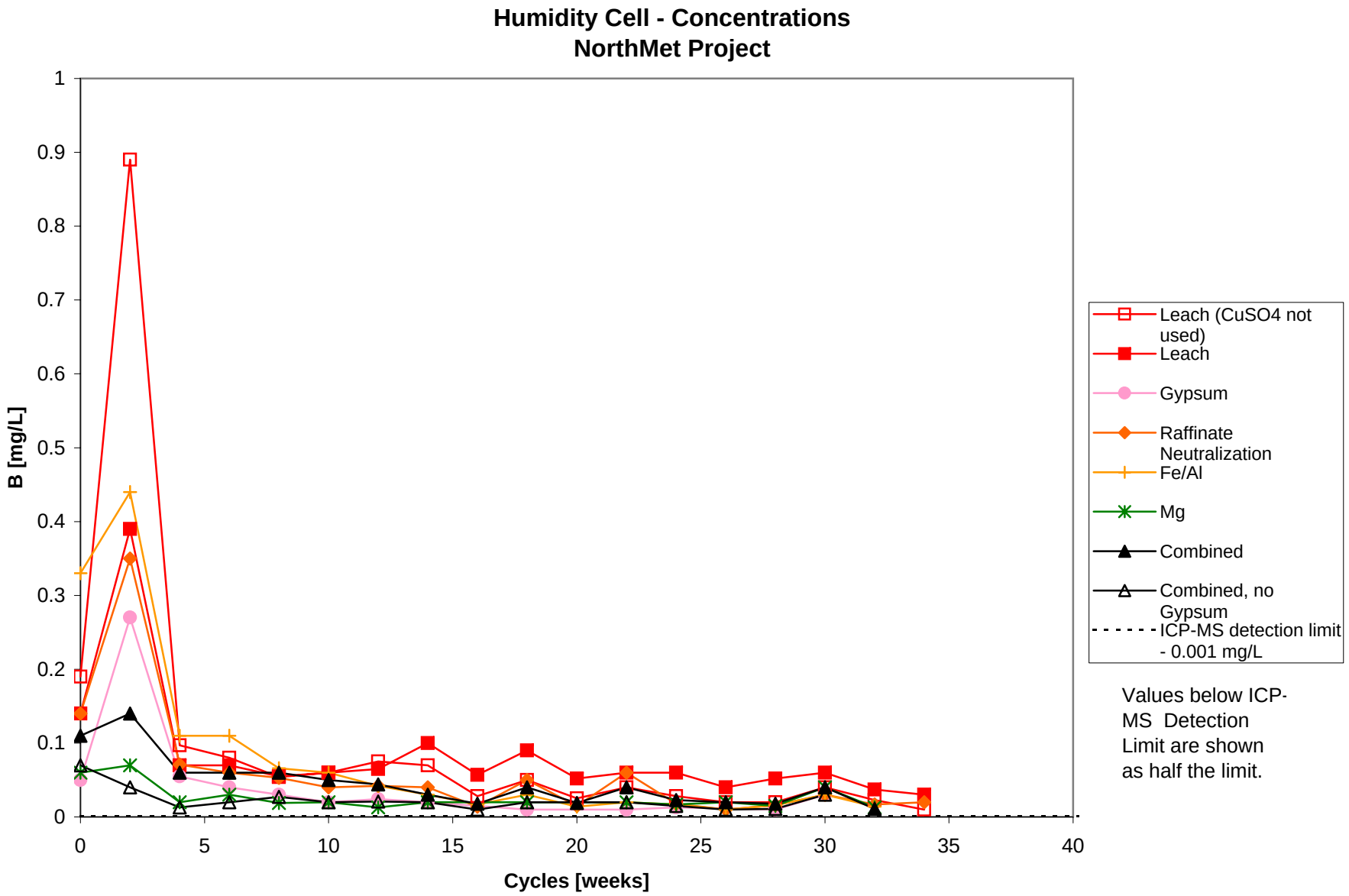

Humidity Cell - Concentrations
NorthMet Project
Al [mg/L]
1000
100
10
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

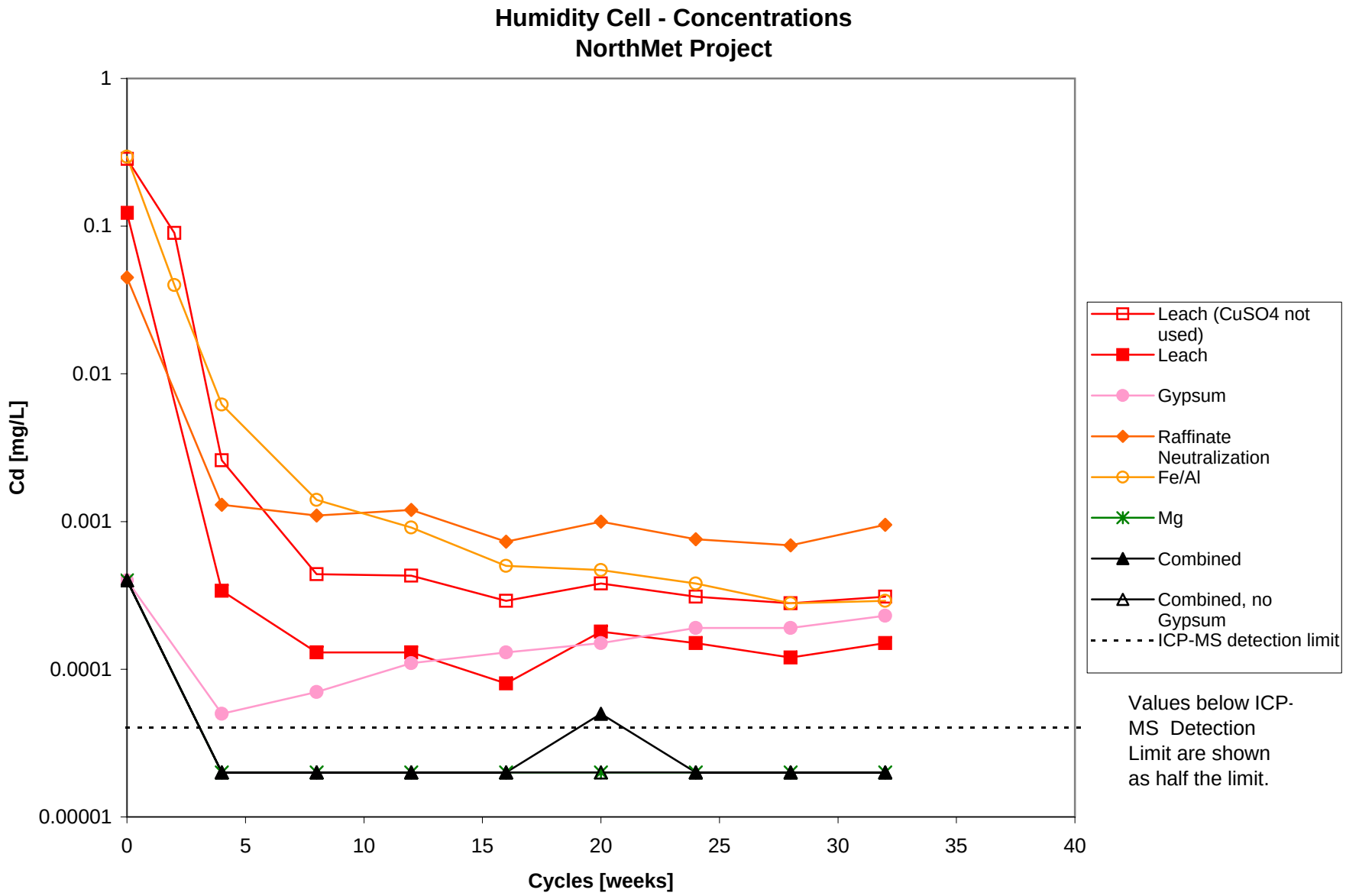

Humidity Cell - Concentrations
NorthMet Project
As [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

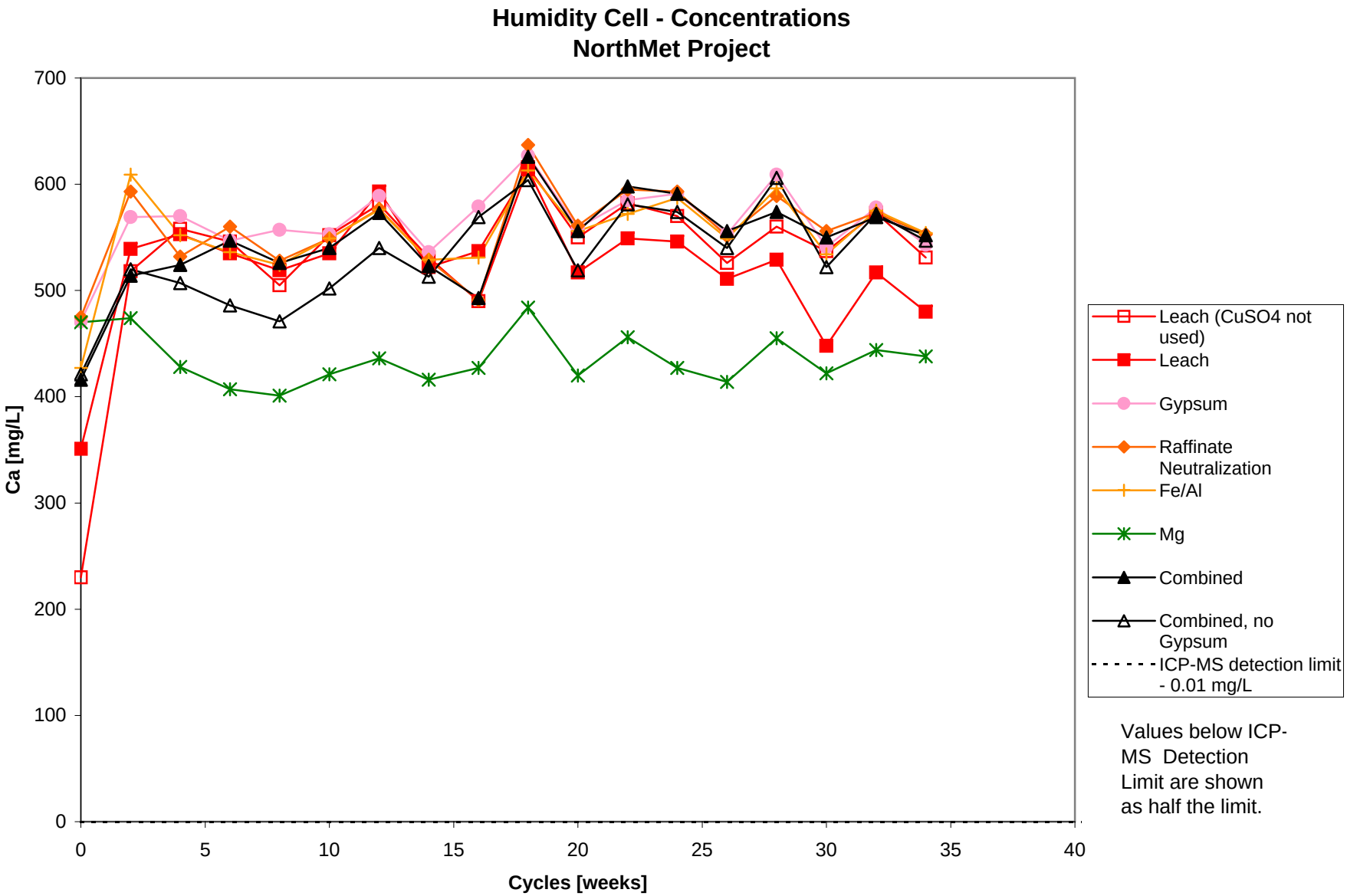

Humidity Cell - Concentrations
NorthMet Project
Ba [mg/L]
0.018
0.016
0.014
0.012
0.01
0.008
0.006
0.004
0.002
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

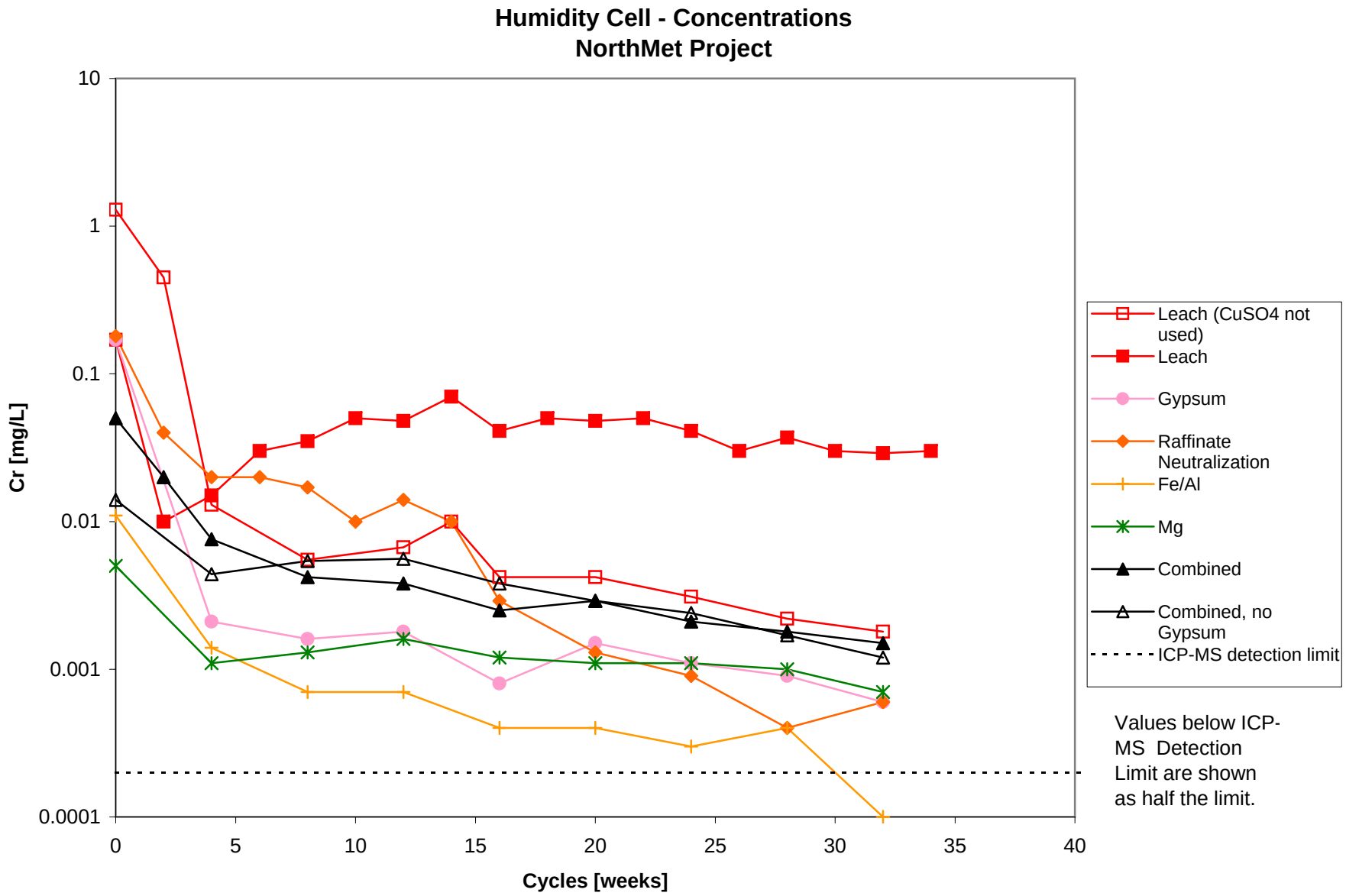

Humidity Cell - Concentrations
NorthMet Project
Be [mg/L]
0.007
0.006
0.005
0.004
0.003
0.002
0.001
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

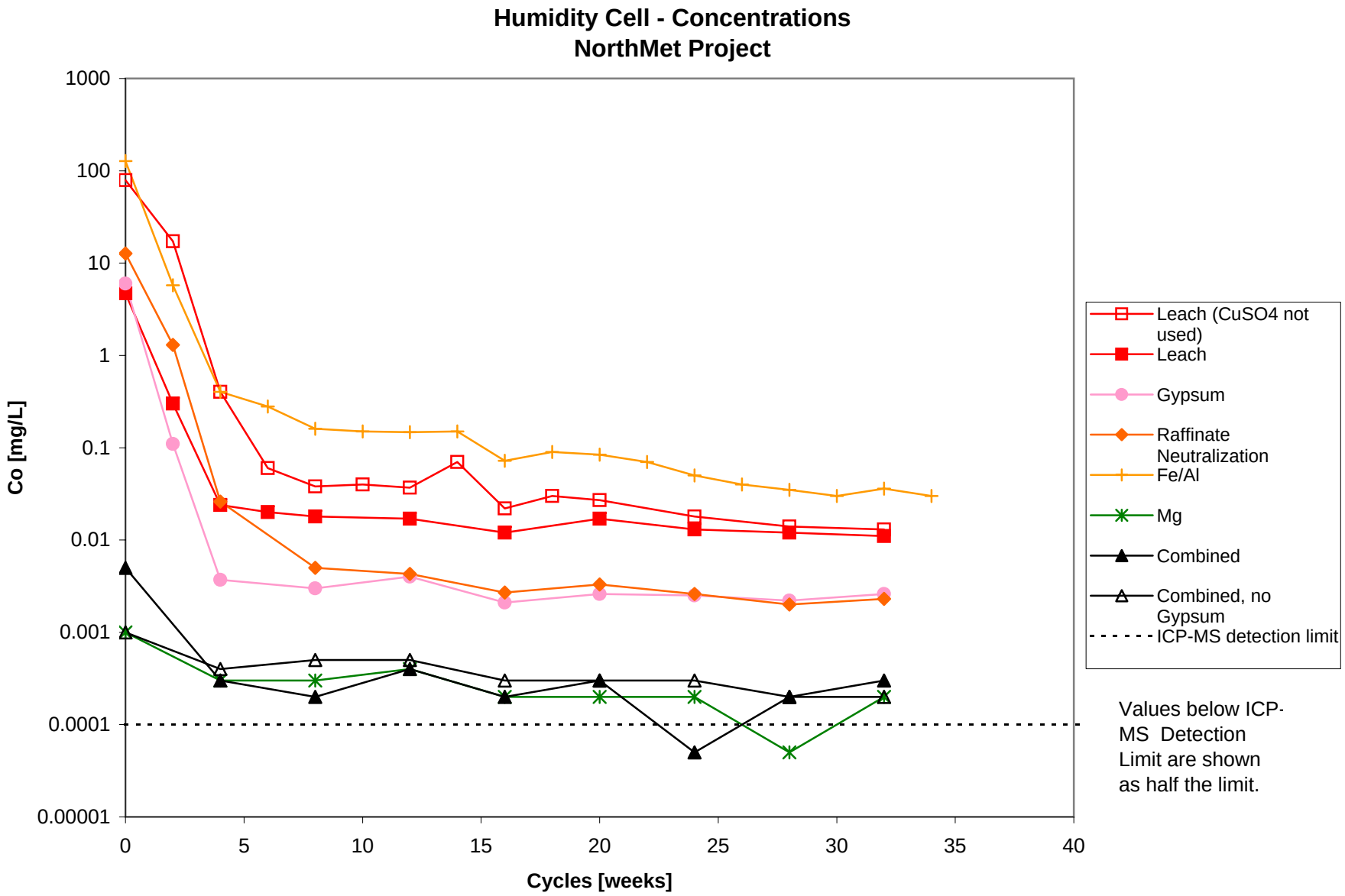

Humidity Cell - Concentrations
NorthMet Project
Bi [mg/L]
0.009
0.008
0.007
0.006
0.005
0.004
0.003
0.002
0.001
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

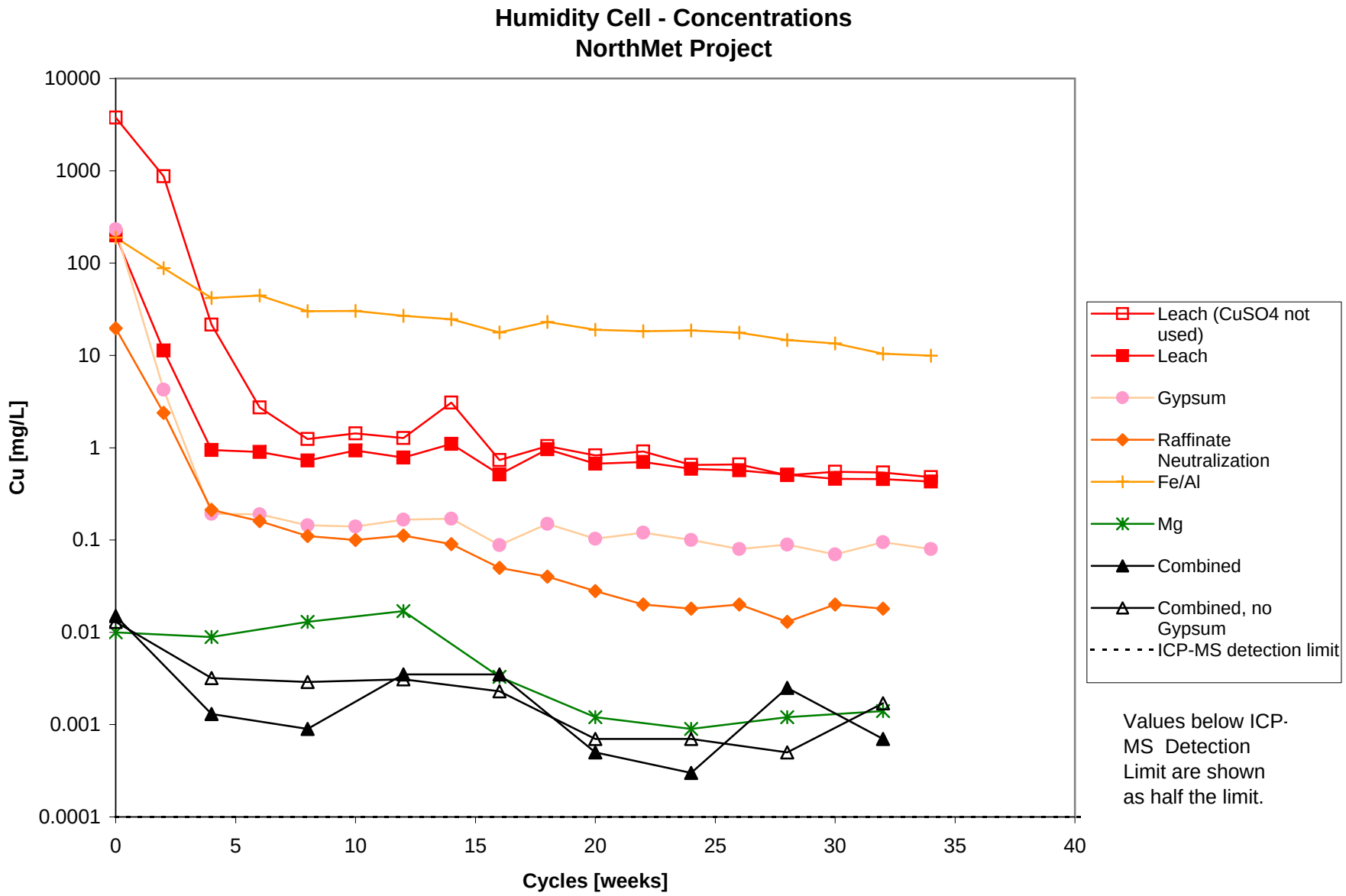

Humidity Cell - Concentrations
NorthMet Project
B [mg/L]
Cycles [weeks]
0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
0.9
1
0
5
10
15
20
25
30
35
40
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit - 0.001 mg/L
Values below ICP-MS Detection Limit are shown as half the limit.

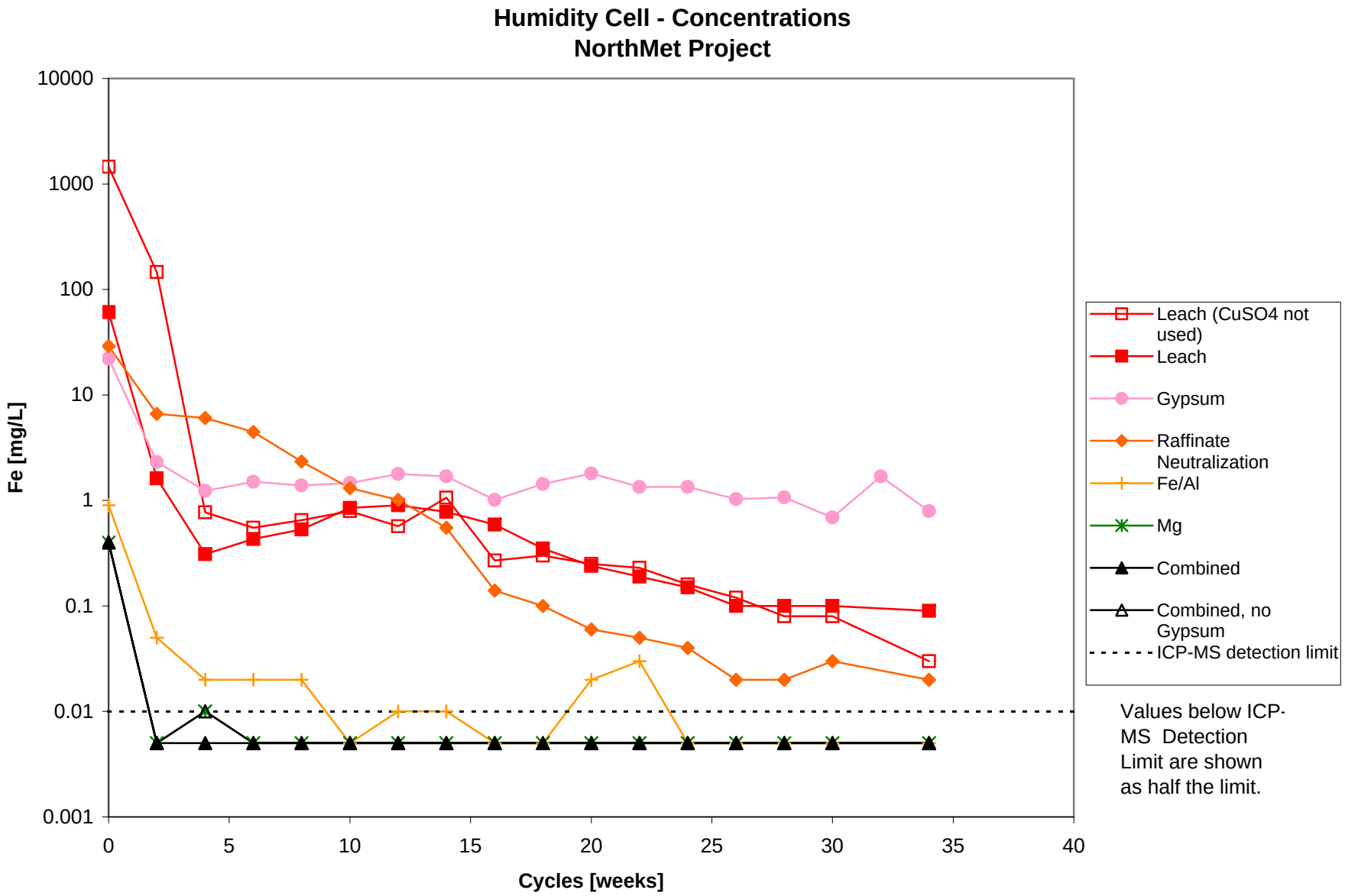

Humidity Cell - Concentrations
NorthMet Project
Cd [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

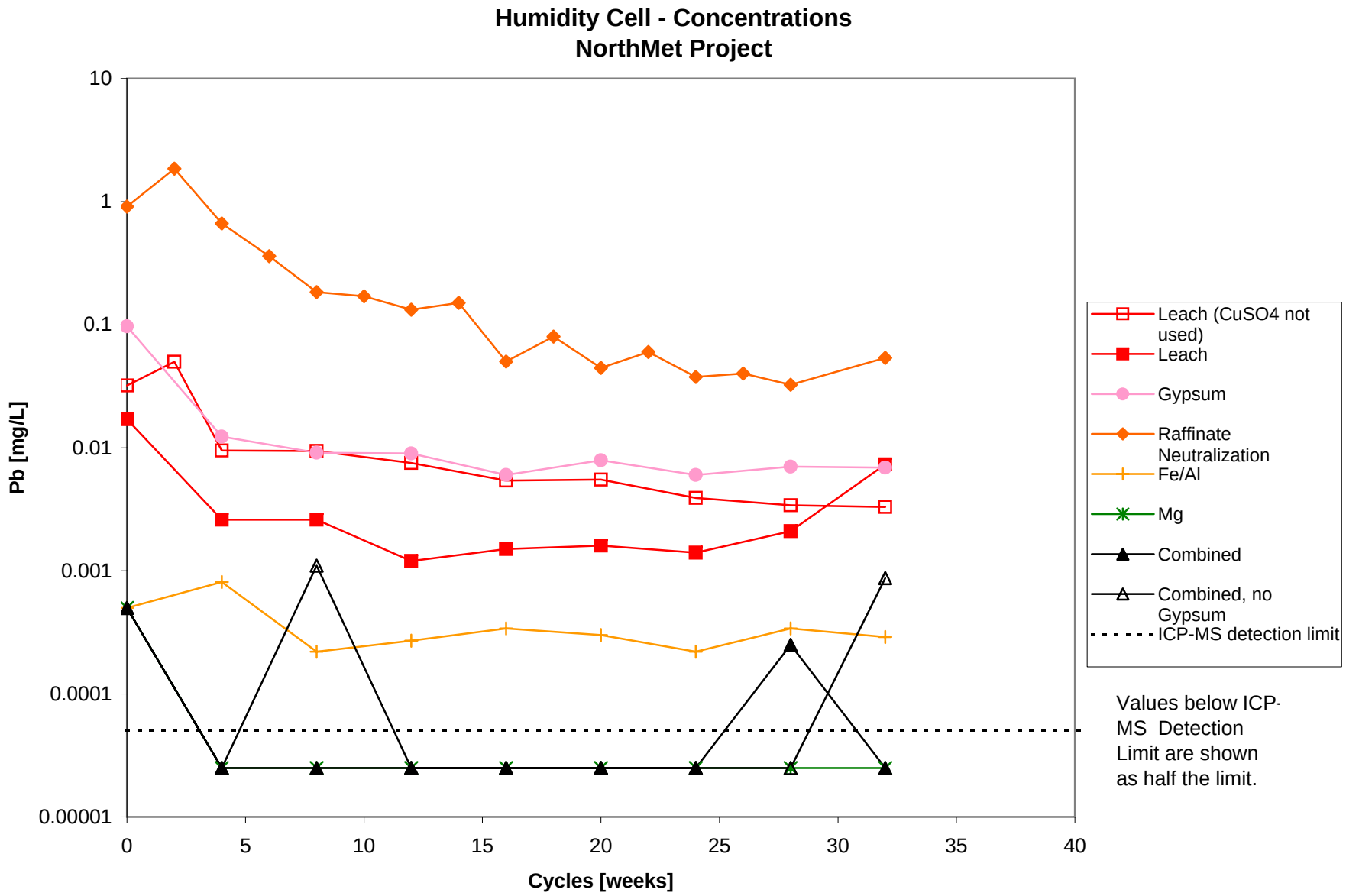

Humidity Cell - Concentrations
NorthMet Project
Ca [mg/L]
Cycles [weeks]
0
100
200
300
400
500
600
700
0
5
10
15
20
25
30
35
40
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit - 0.01 mg/L
Values below ICP-MS Detection Limit are shown as half the limit.

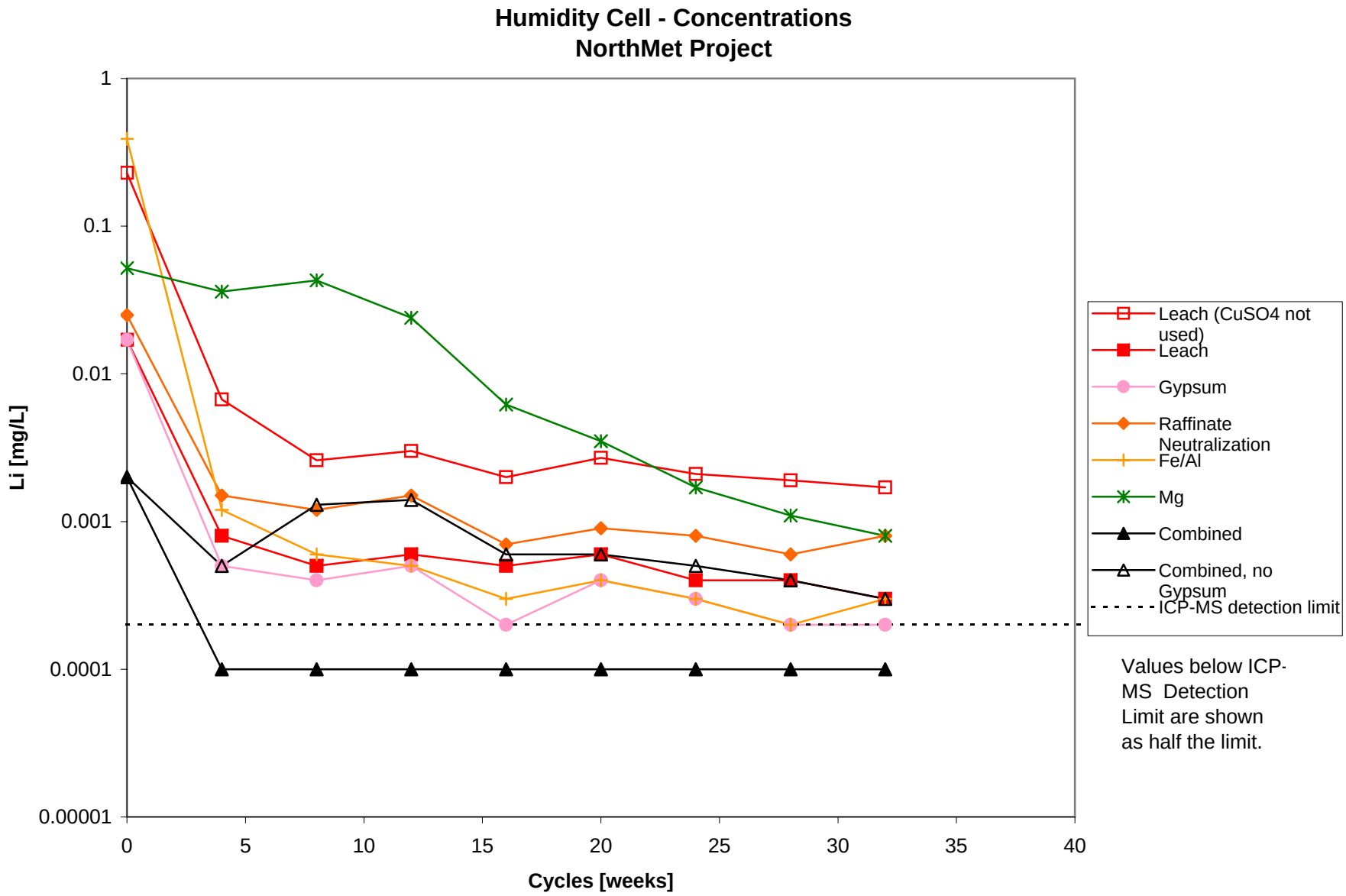

Humidity Cell - Concentrations
NorthMet Project
Cr [mg/L]
10
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

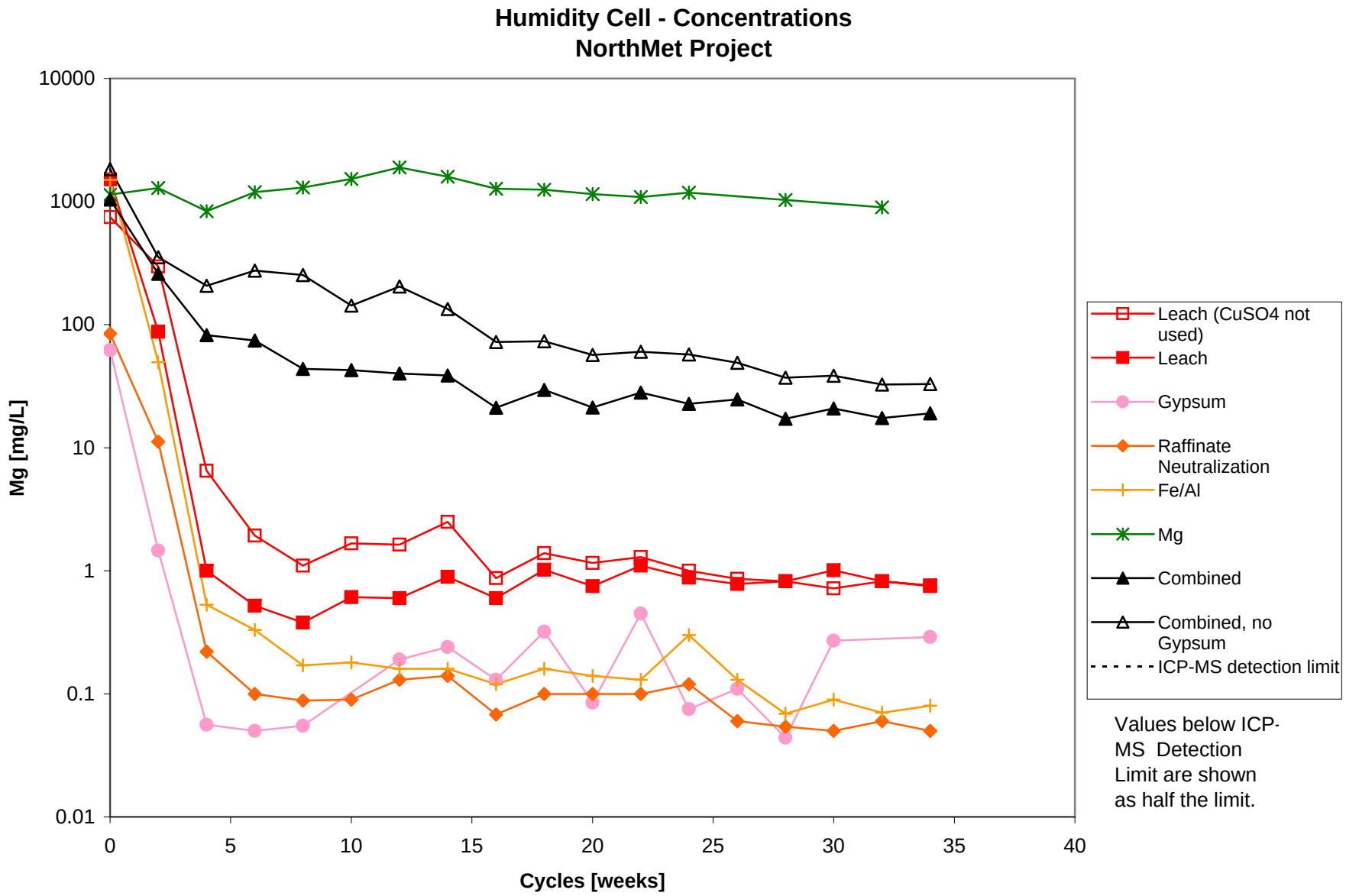

Humidity Cell - Concentrations
NorthMet Project
Co [mg/L]
1000
100
10
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

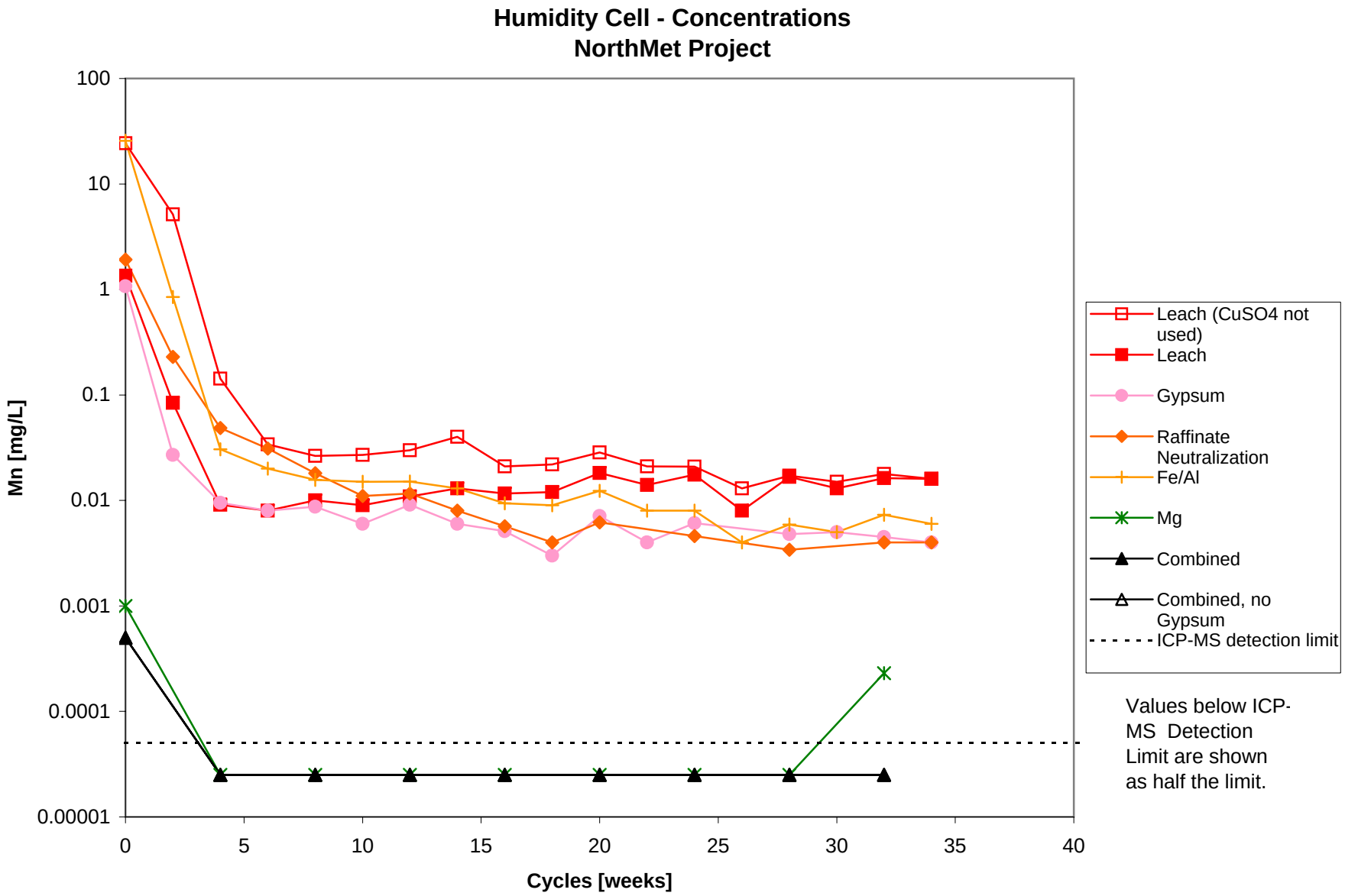

Humidity Cell - Concentrations
NorthMet Project
Cu [mg/L]
10000
1000
100
10
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

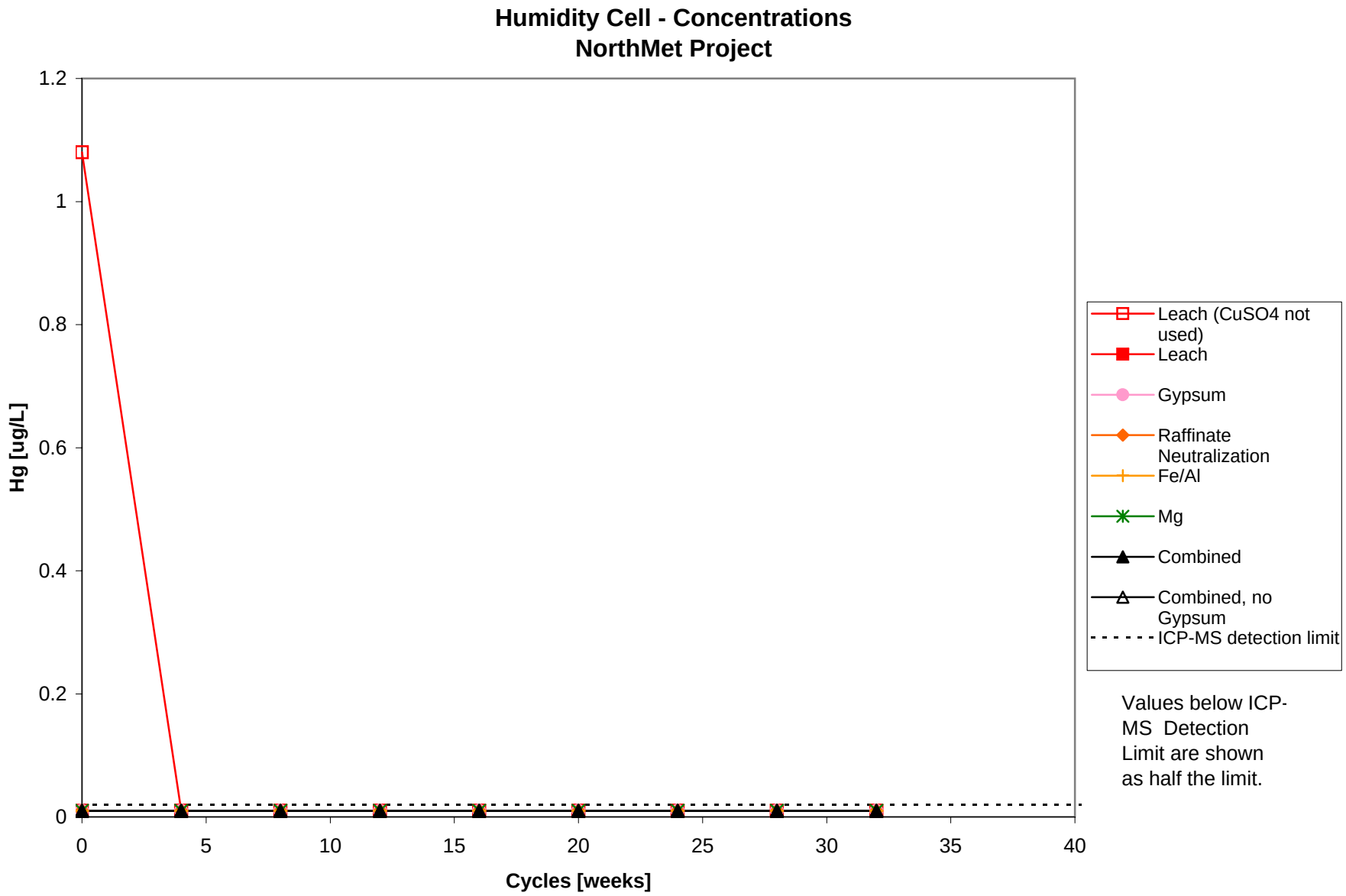

Humidity Cell - Concentrations
NorthMet Project
Fe [mg/L]
10000
1000
100
10
1
0.1
0.01
0.001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

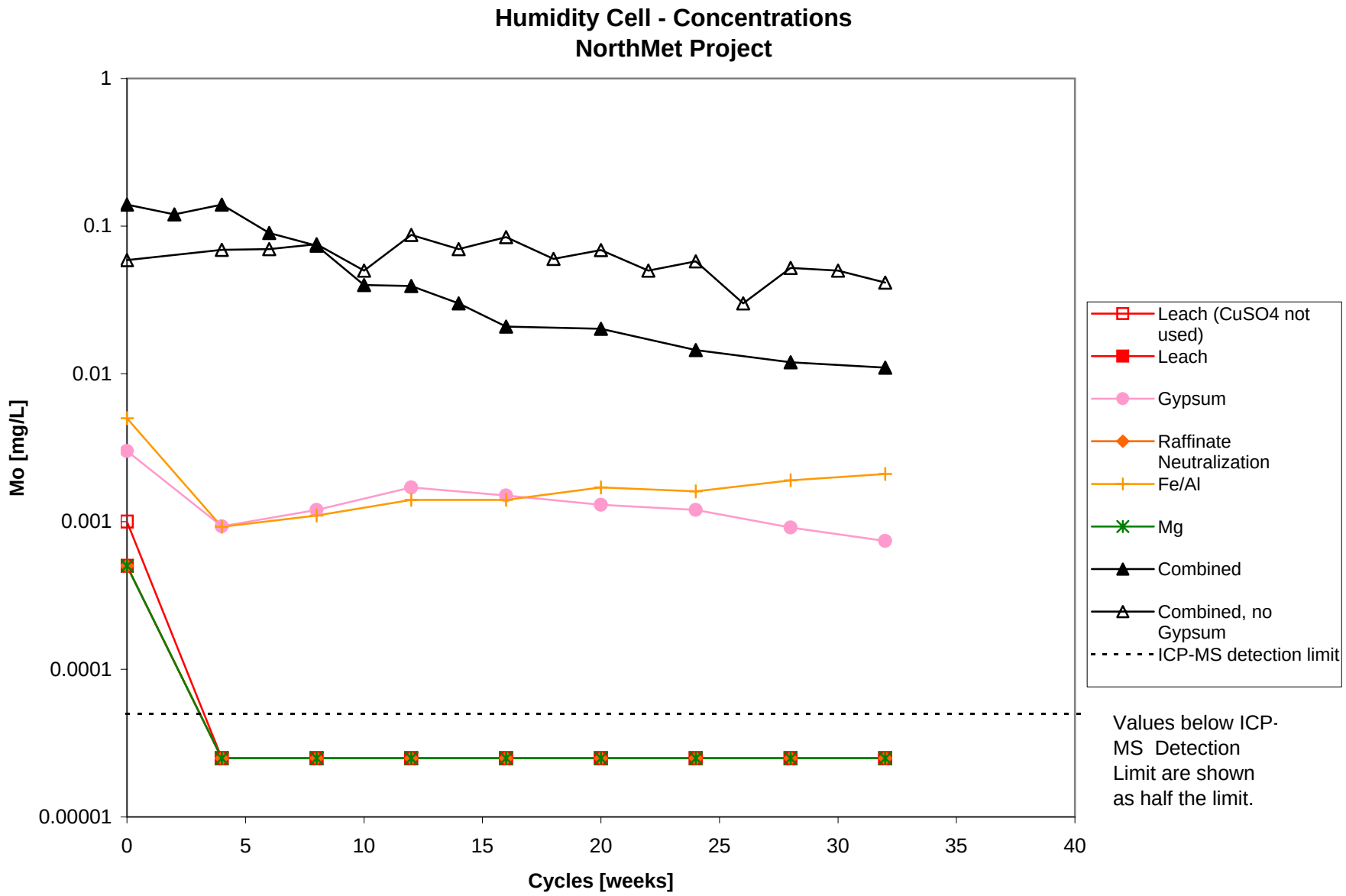

Humidity Cell - Concentrations
NorthMet Project
Pb [mg/L]
10
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

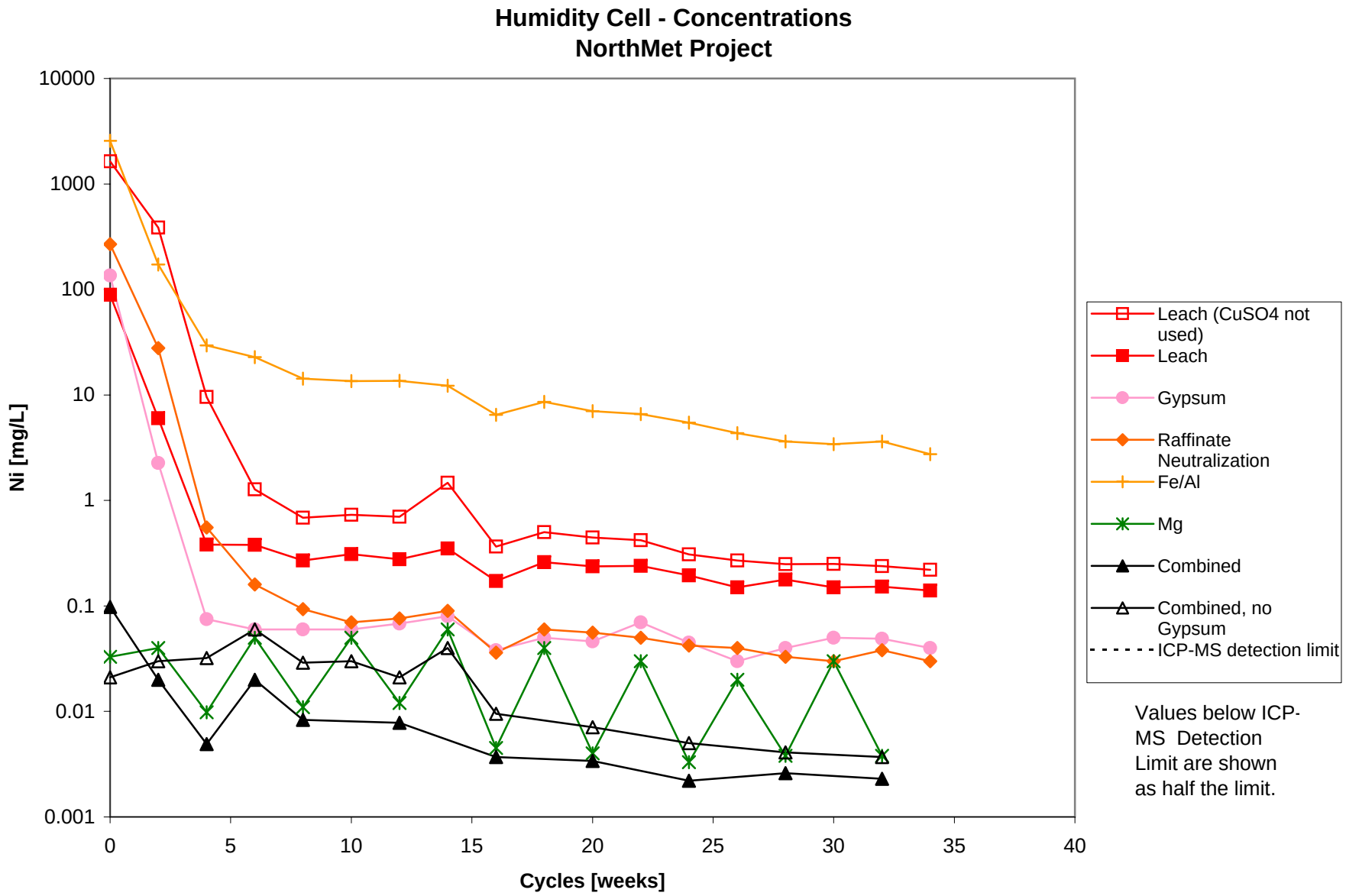

Humidity Cell - Concentrations
NorthMet Project
Li [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

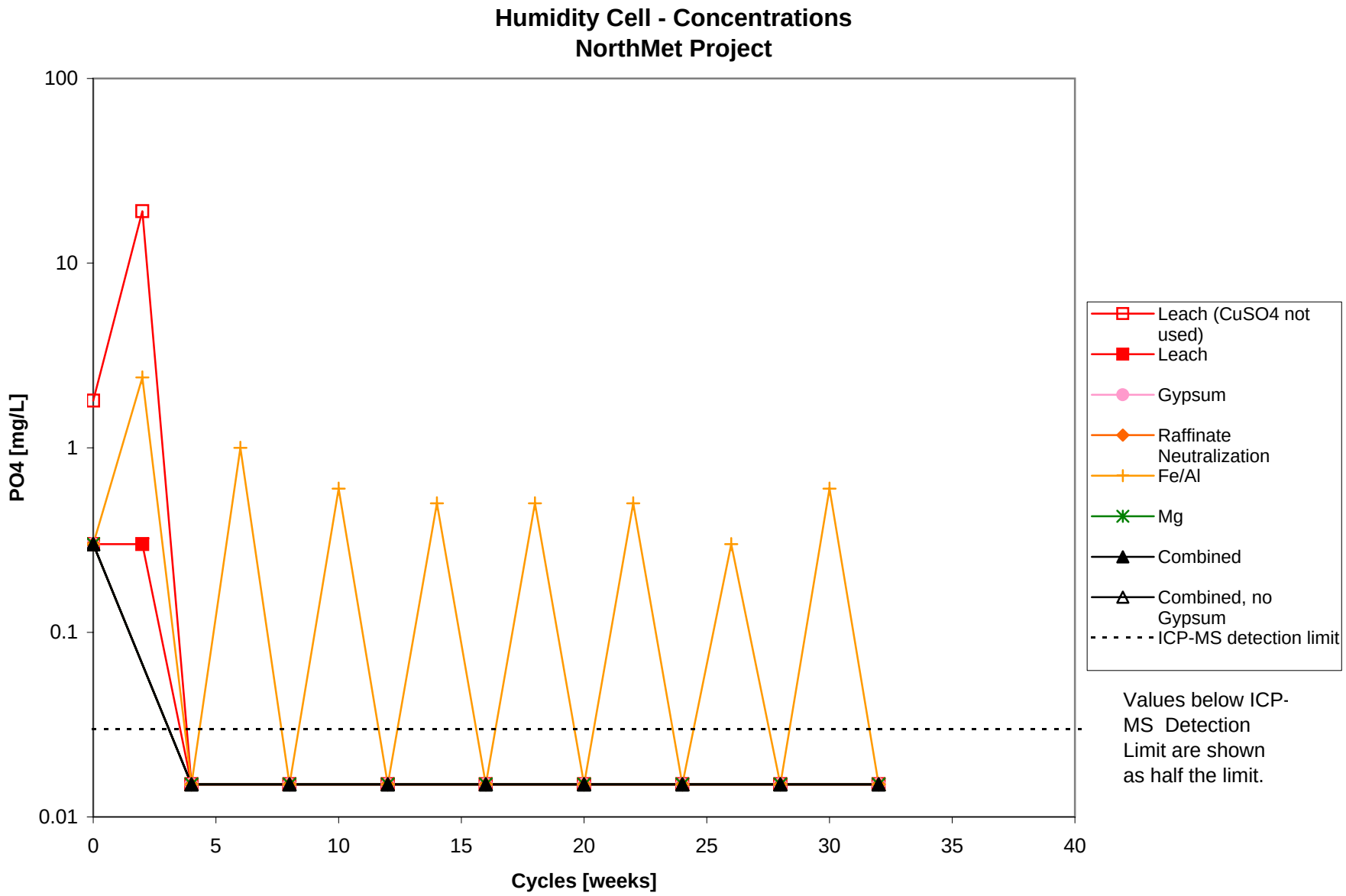

Humidity Cell - Concentrations
NorthMet Project
Mg [mg/L]
10000
1000
100
10
1
0.1
0.01
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

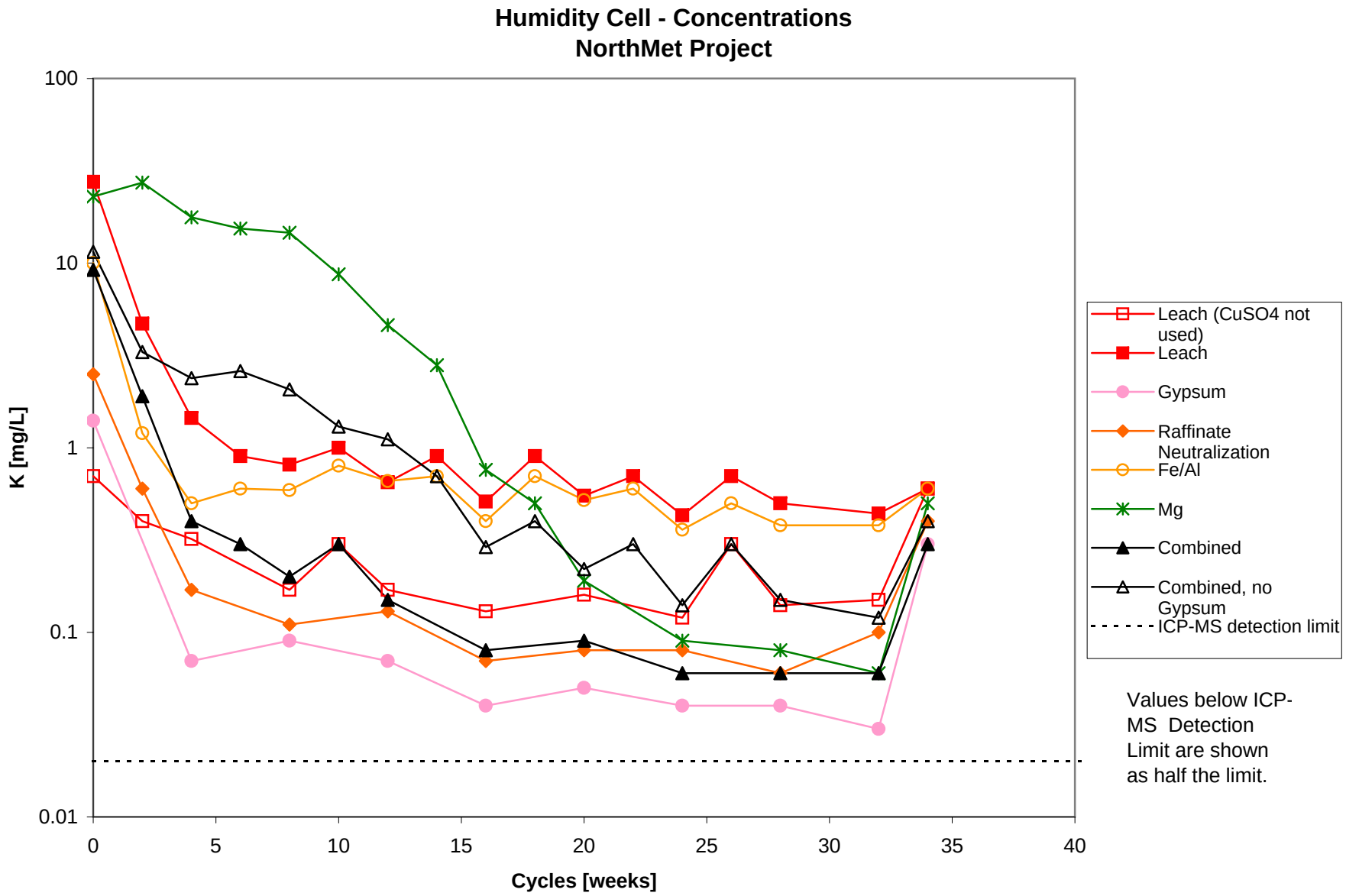

Humidity Cell - Concentrations
NorthMet Project
Mn [mg/L]
100
10
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

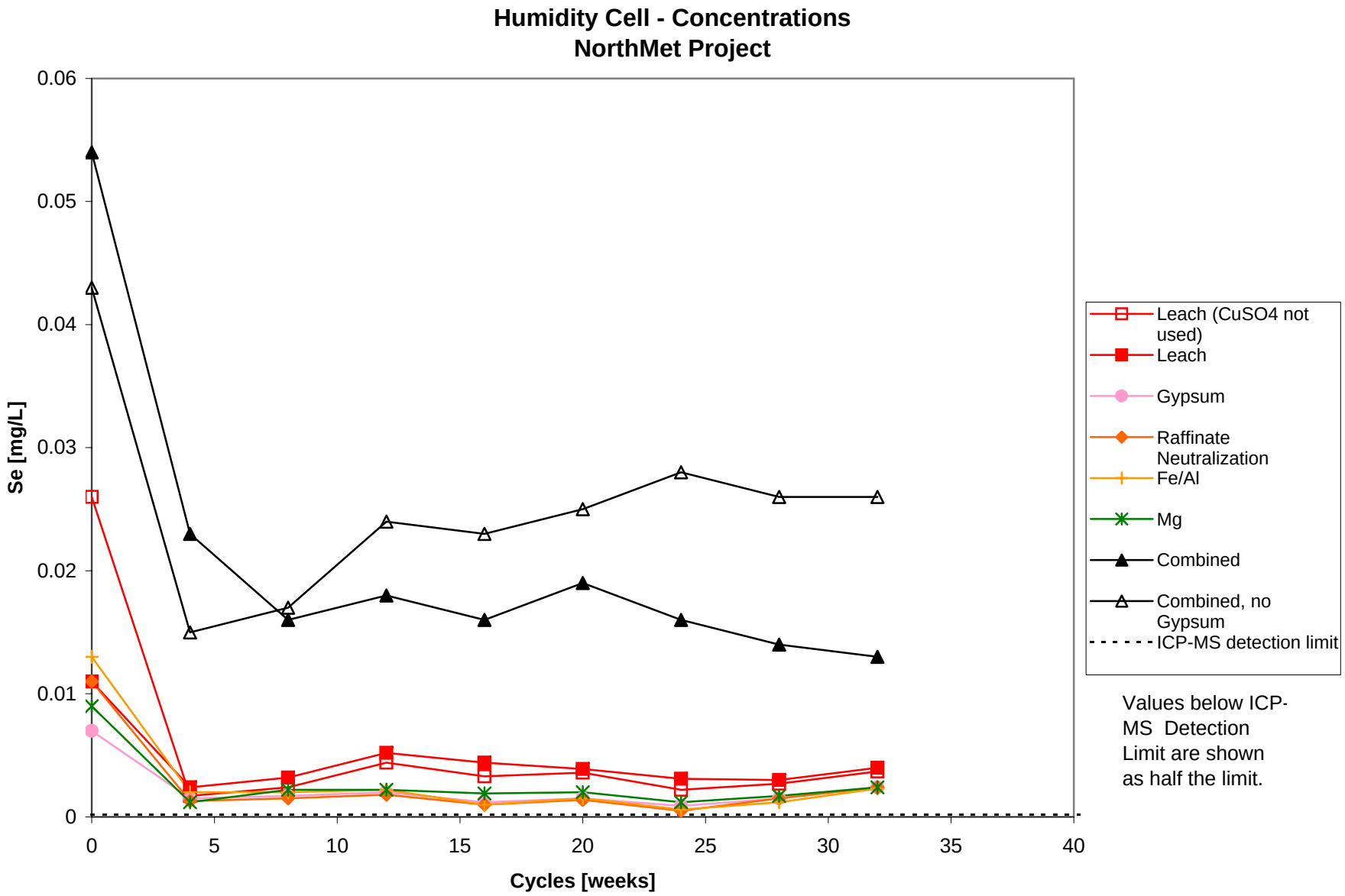

Humidity Cell - Concentrations
NorthMet Project
Hg [ug/L]
1.2
1
0.8
0.6
0.4
0.2
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

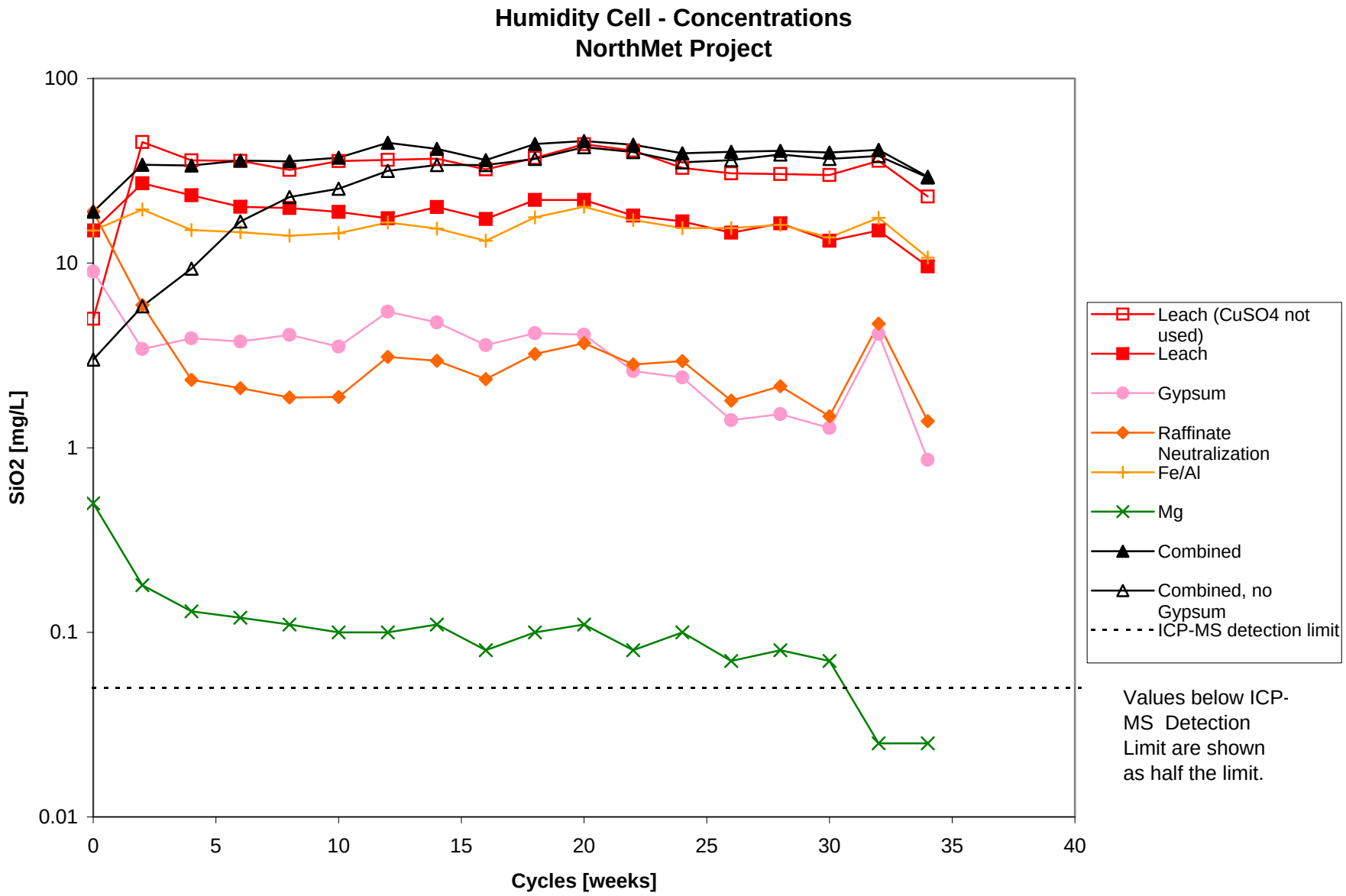

Humidity Cell - Concentrations
NorthMet Project
Mo [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

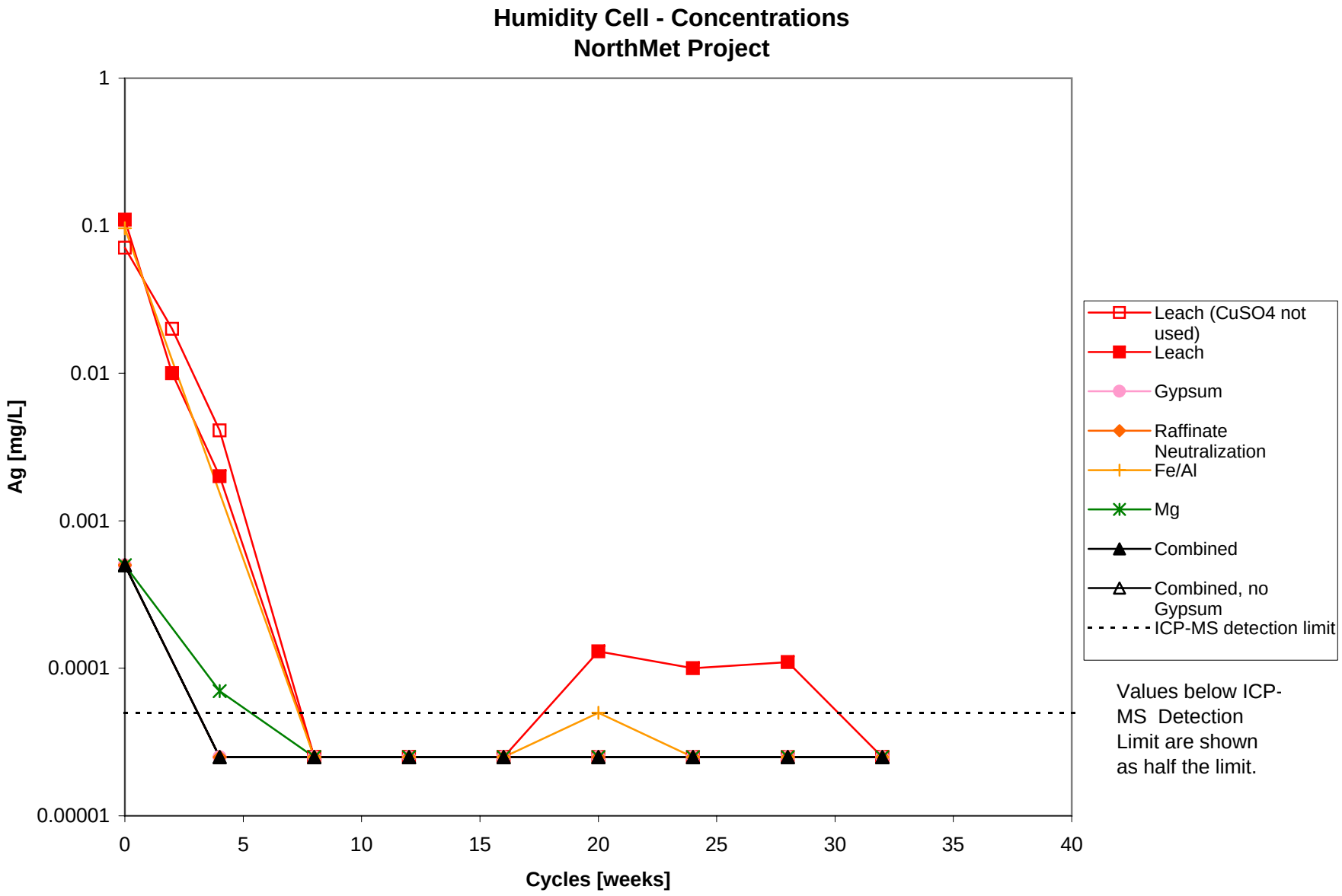

Humidity Cell - Concentrations
NorthMet Project
Ni [mg/L]
10000
1000
100
10
1
0.1
0.01
0.001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

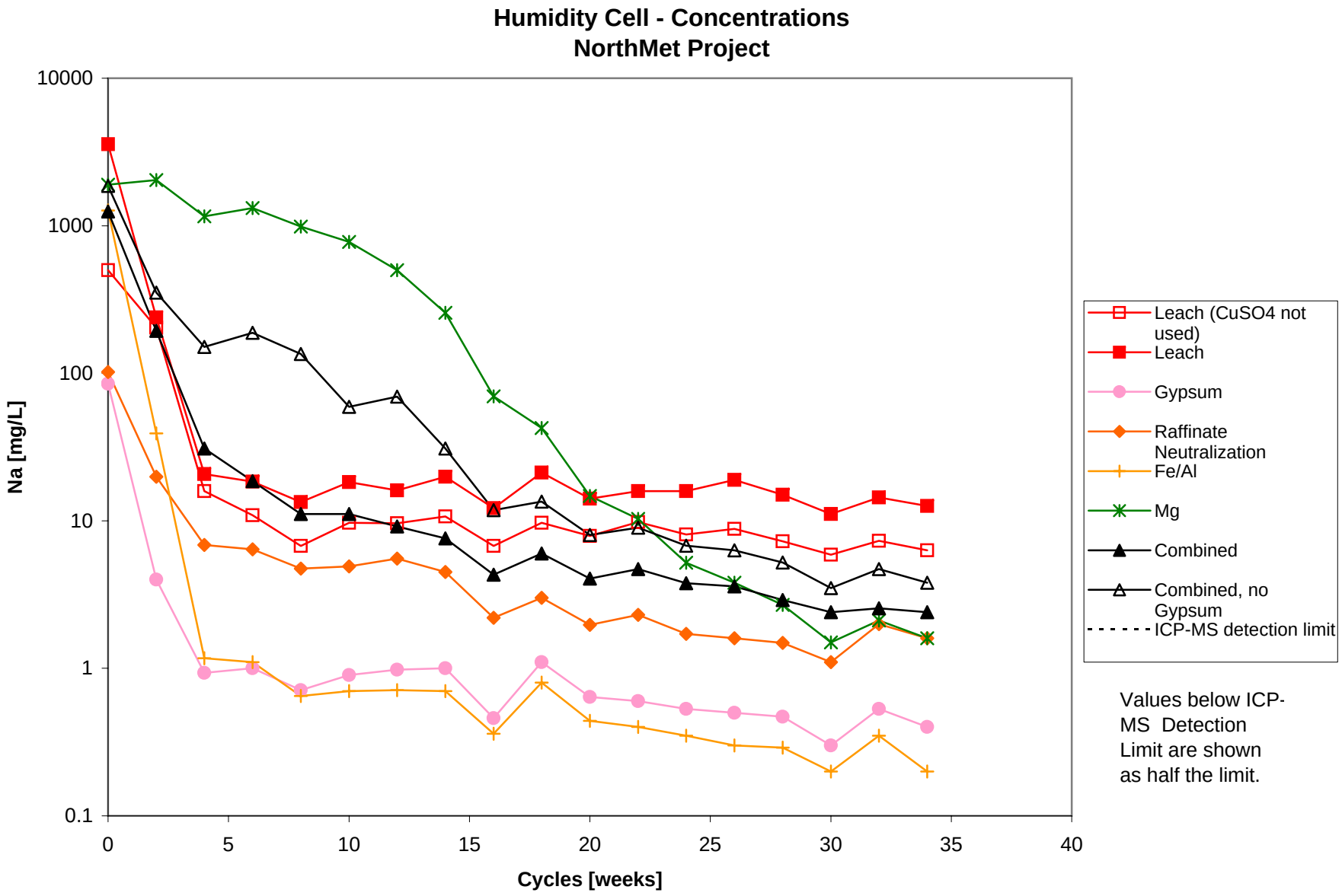

Humidity Cell - Concentrations
NorthMet Project
PO4 [mg/L]
100
10
1
0.1
0.01
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

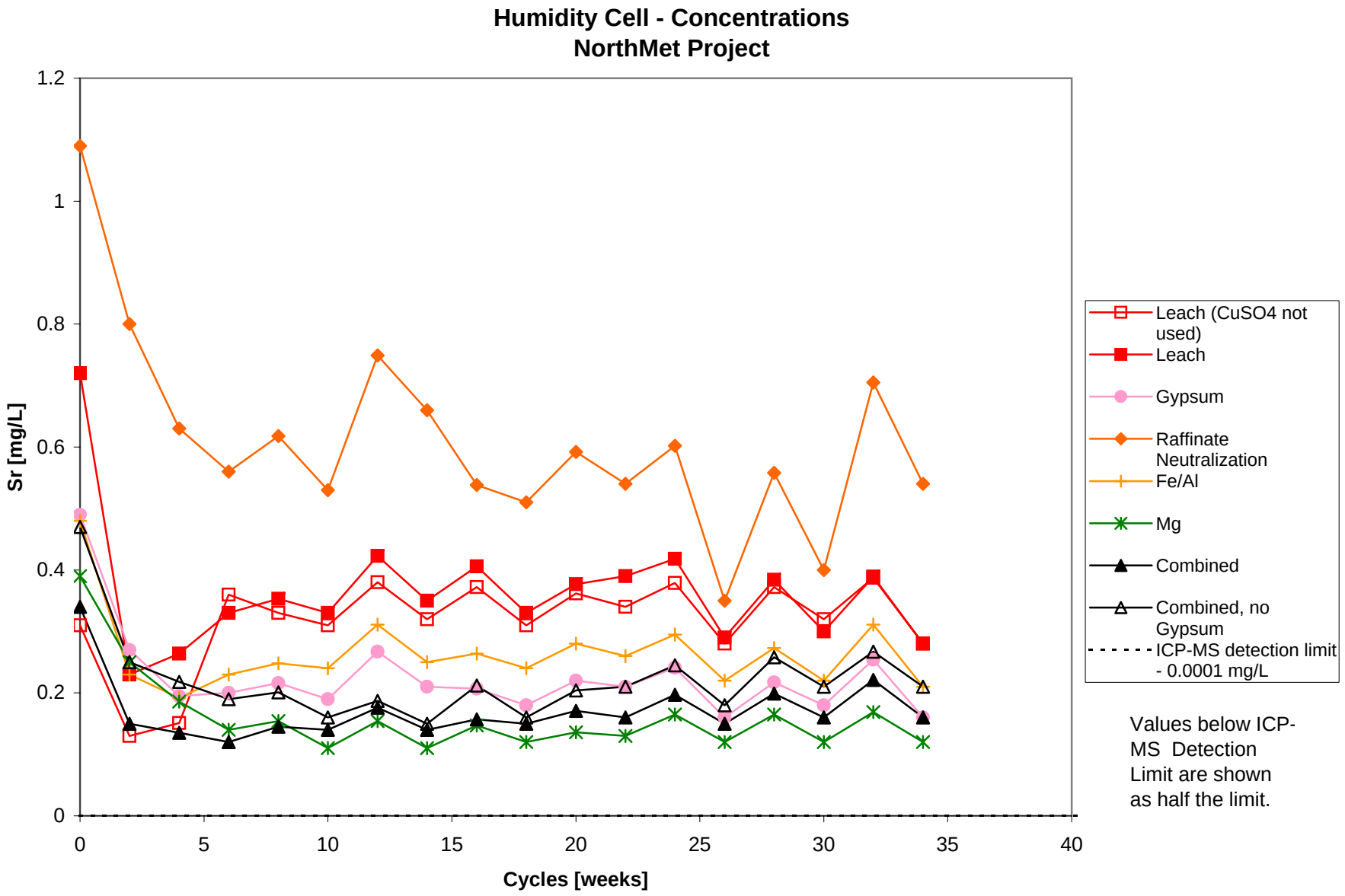

Humidity Cell - Concentrations
NorthMet Project
K [mg/L]
100
10
1
0.1
0.01
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

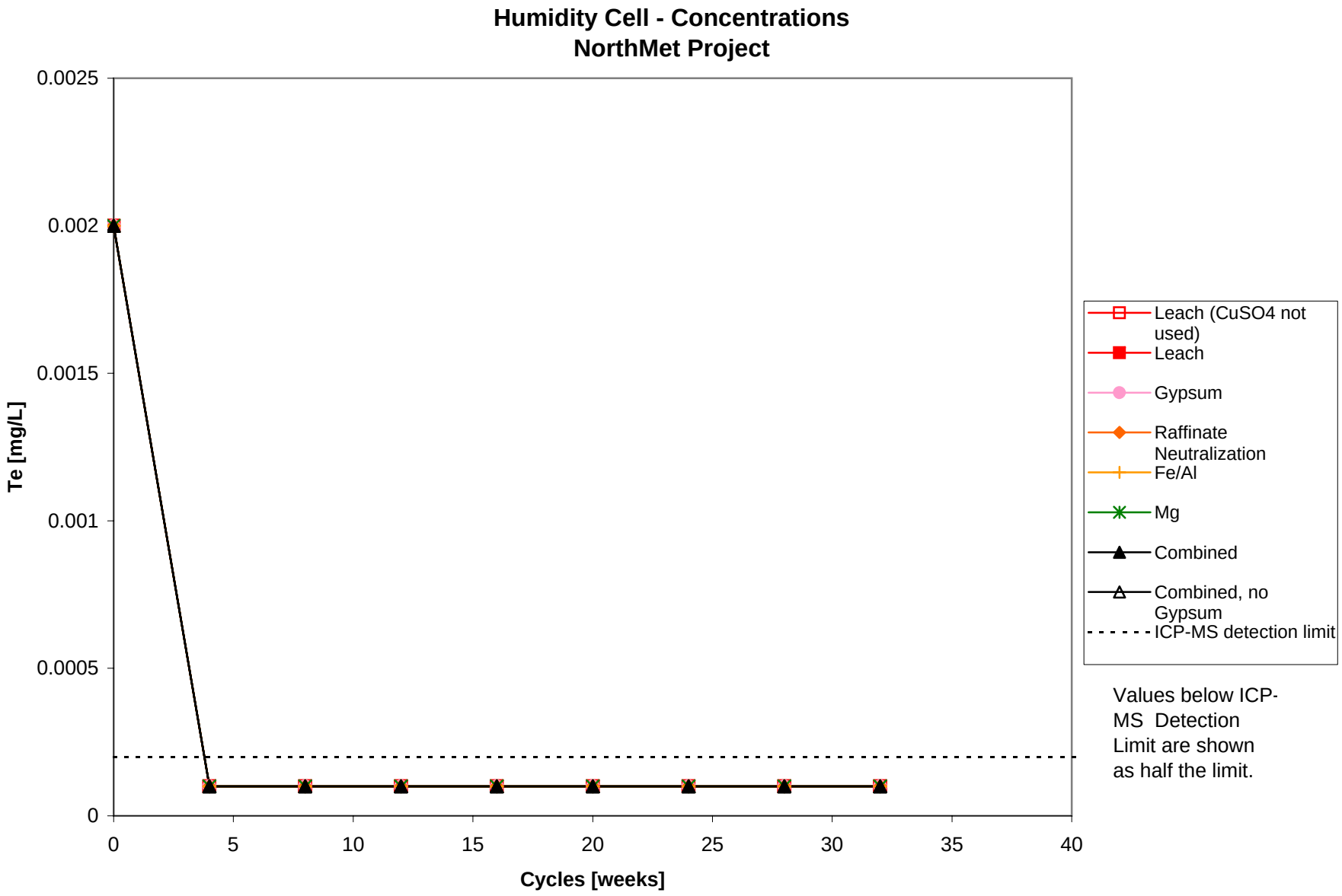

Humidity Cell - Concentrations
NorthMet Project
Se [mg/L]
0.06
0.05
0.04
0.03
0.02
0.01
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

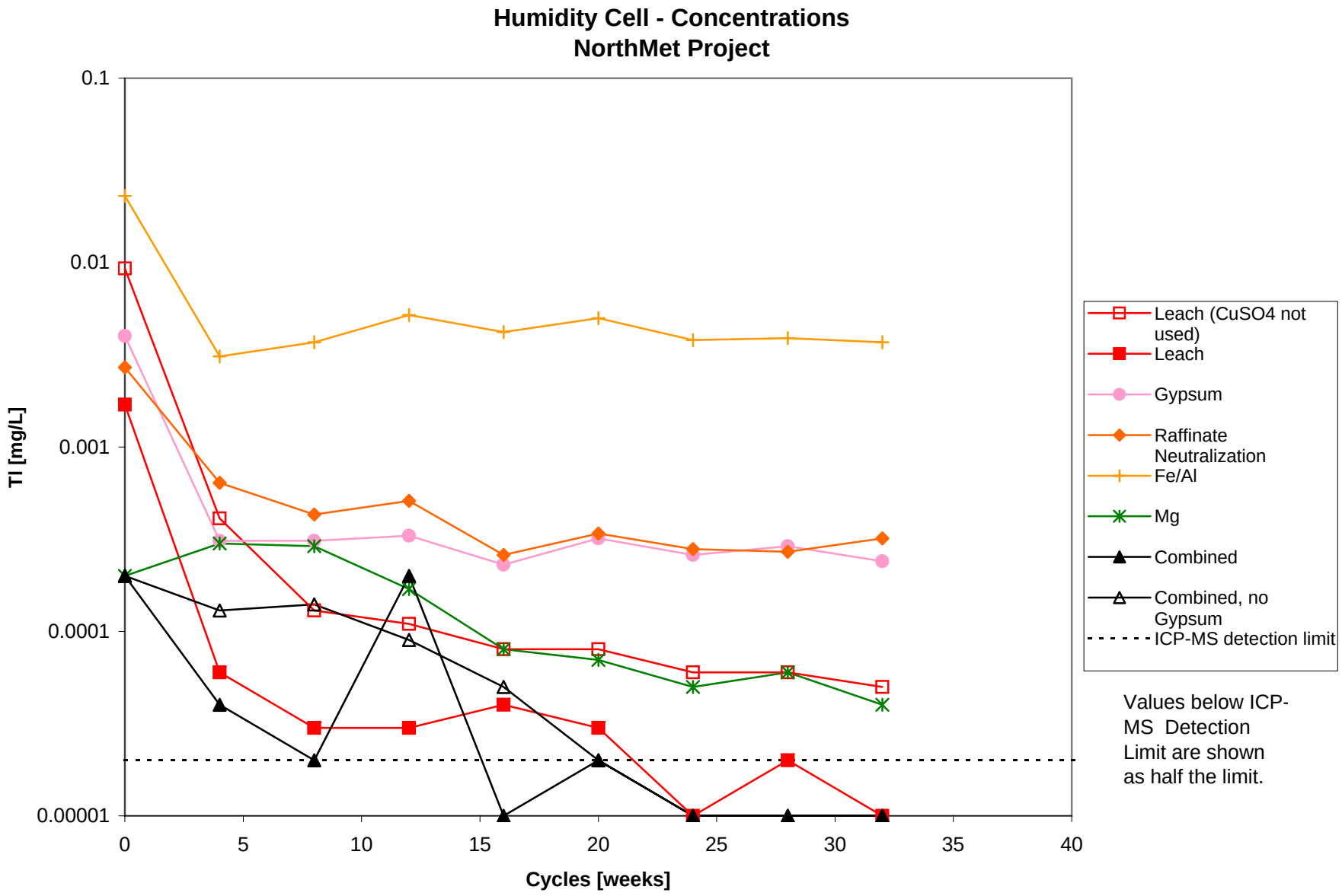

Humidity Cell - Concentrations
NorthMet Project
SiO2 [mg/L]
100
10
1
0.1
0.01
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

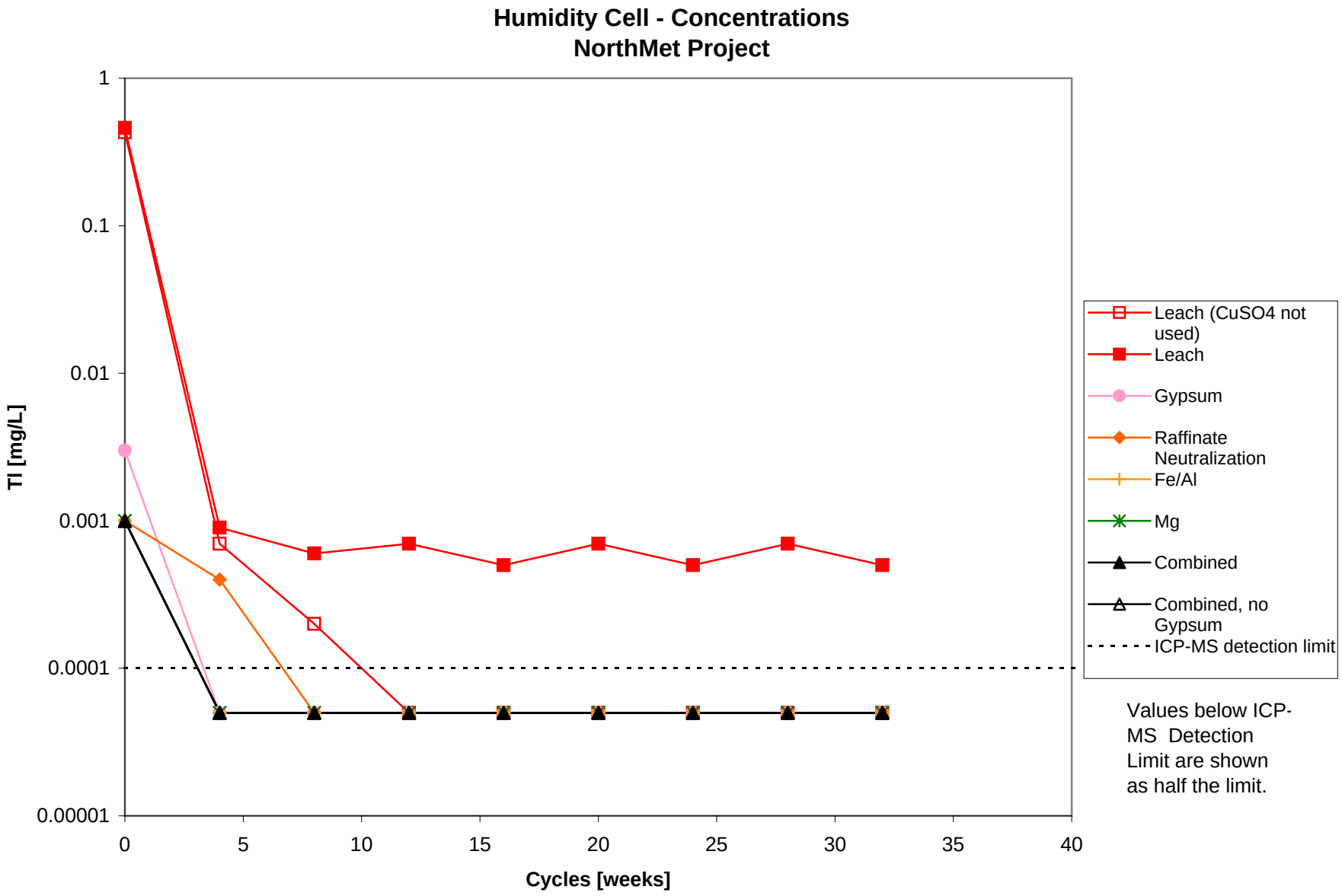

Humidity Cell - Concentrations
NorthMet Project
Ag [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

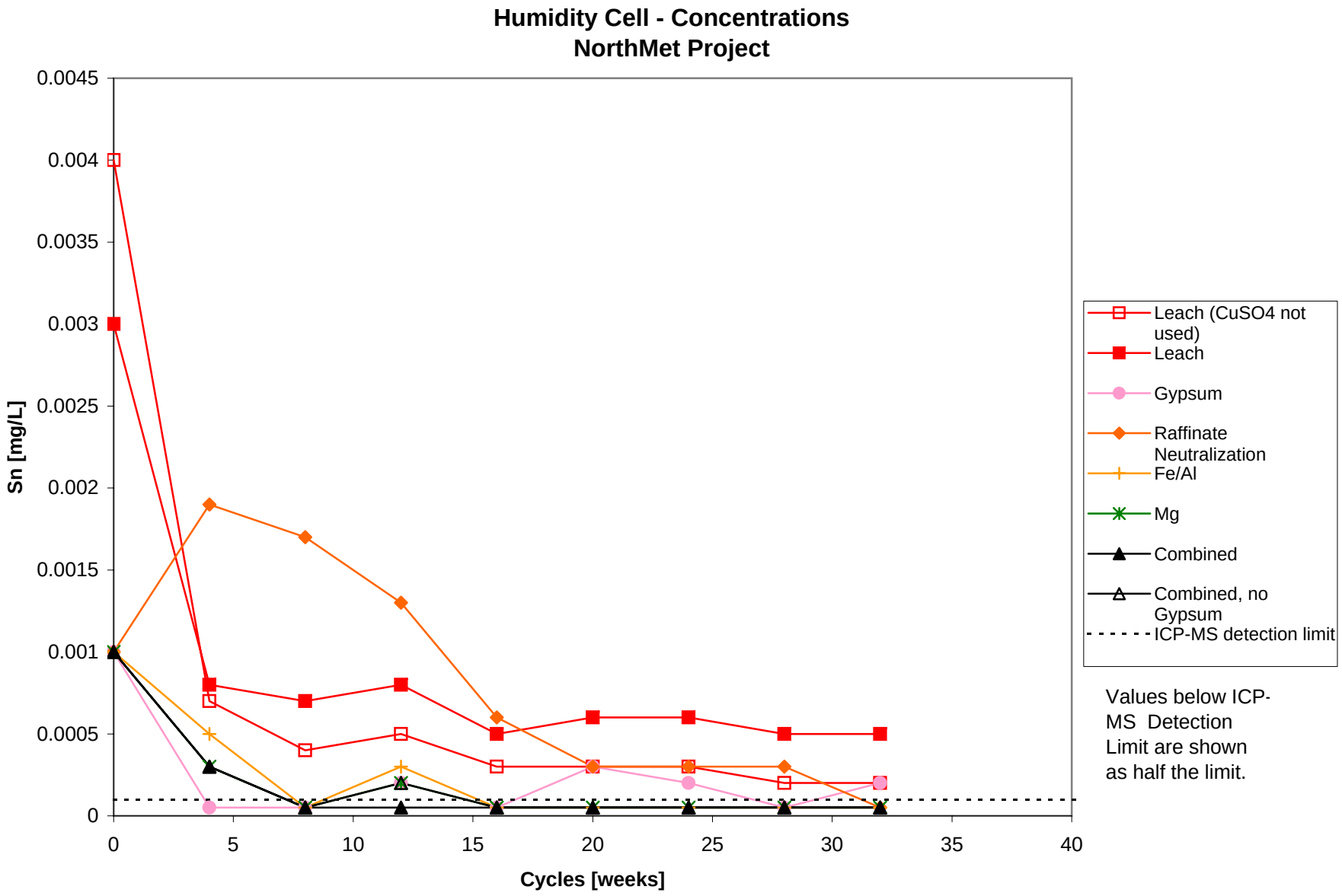

Humidity Cell - Concentrations
NorthMet Project
Na [mg/L]
10000
1000
100
10
1
0.1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

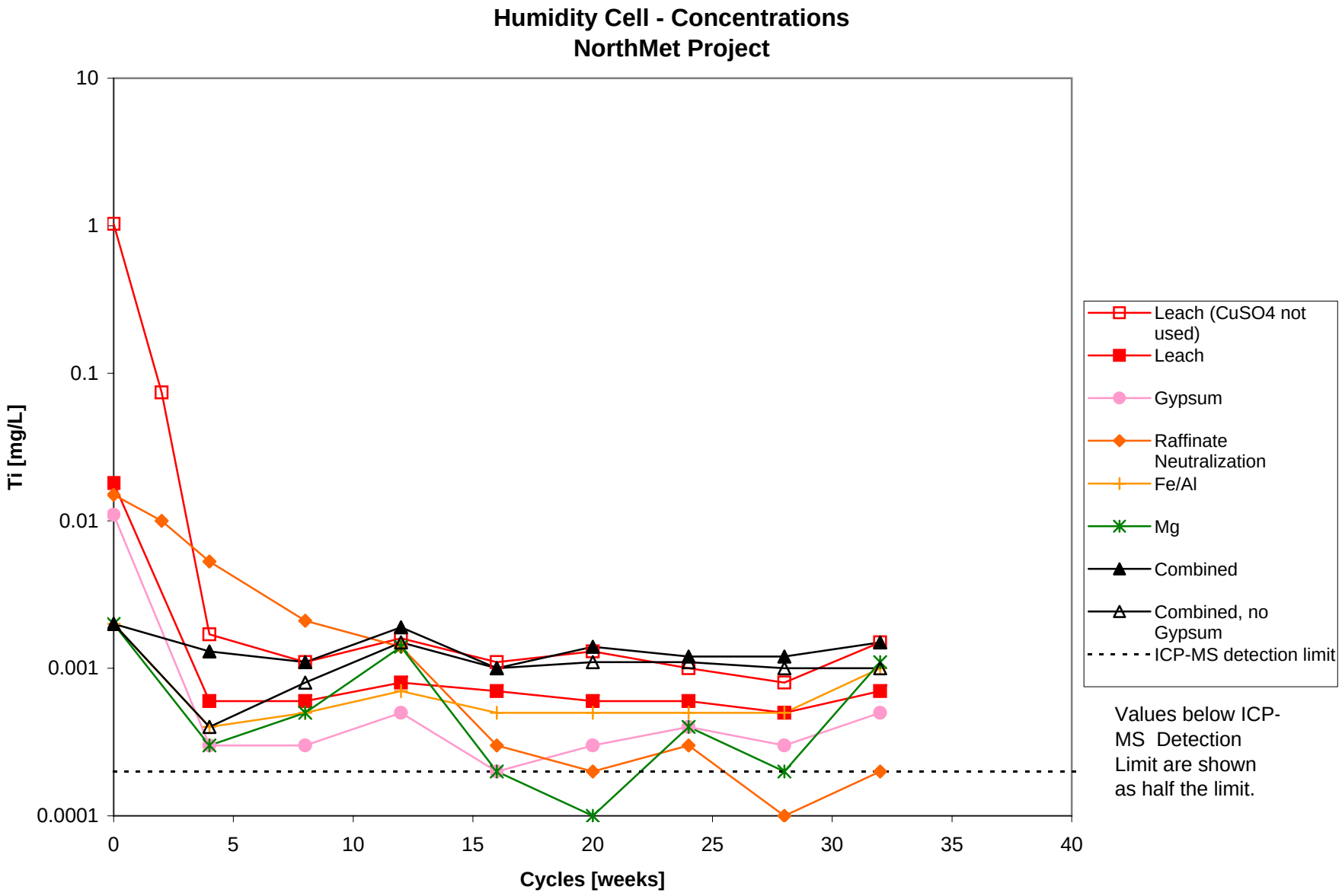

Humidity Cell - Concentrations
NorthMet Project
Sr [mg/L]
1.2
1
0.8
0.6
0.4
0.2
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit - 0.0001 mg/L
Values below ICP-MS Detection Limit are shown as half the limit.

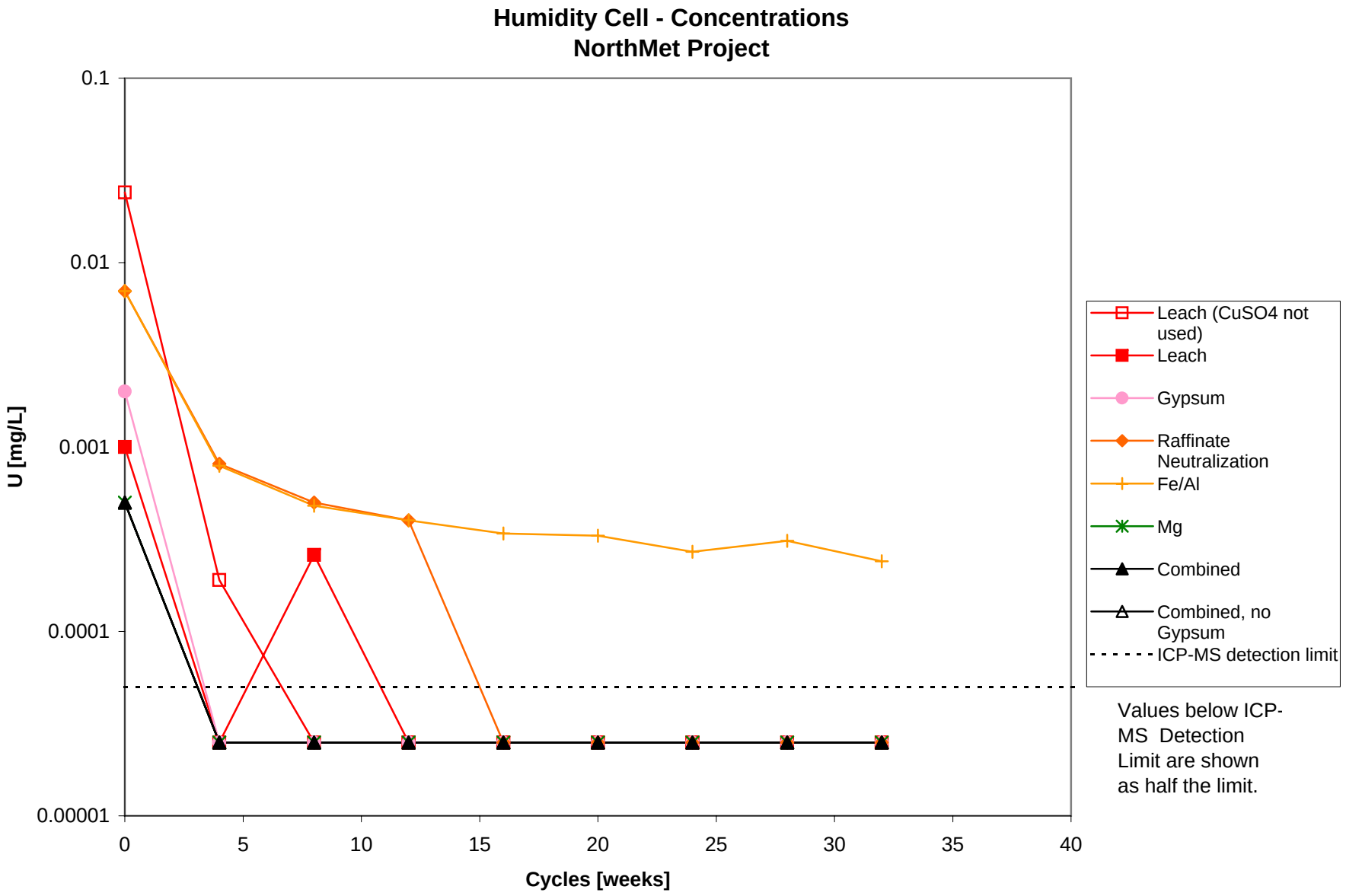

Humidity Cell - Concentrations
NorthMet Project
Te [mg/L]
0.0025
0.002
0.0015
0.001
0.0005
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

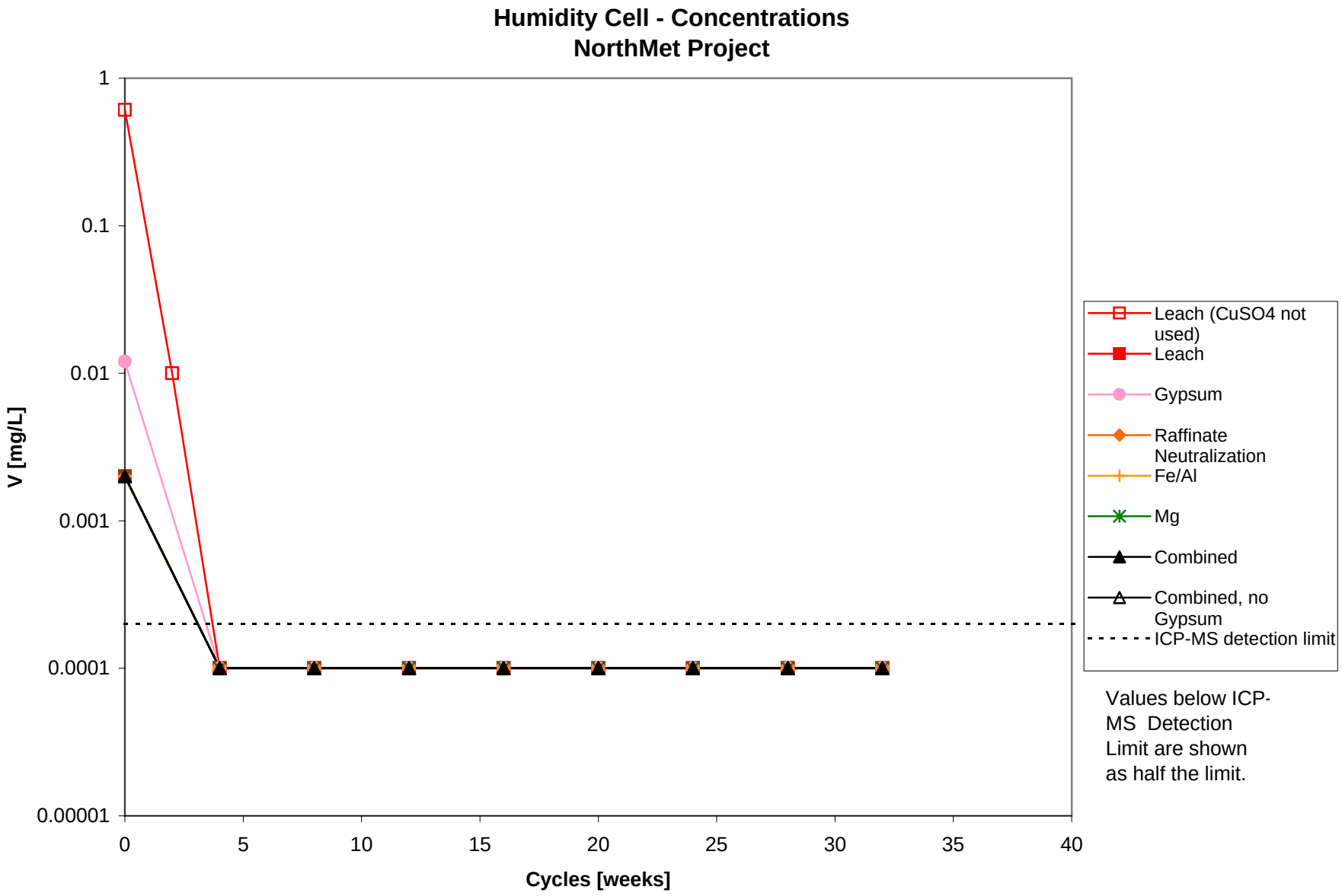

Humidity Cell - Concentrations
NorthMet Project
Tl [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

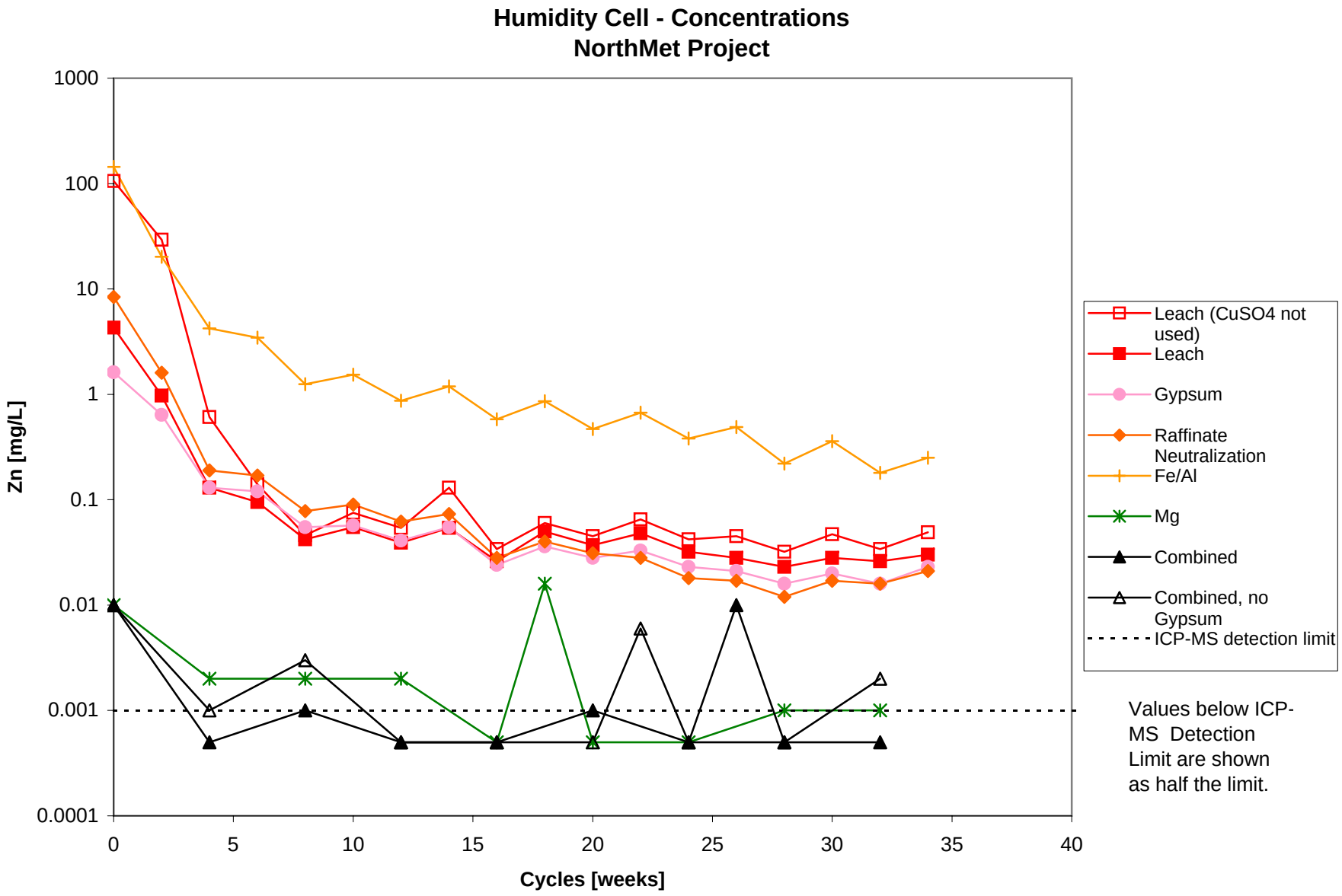

Humidity Cell - Concentrations
NorthMet Project
Tl [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

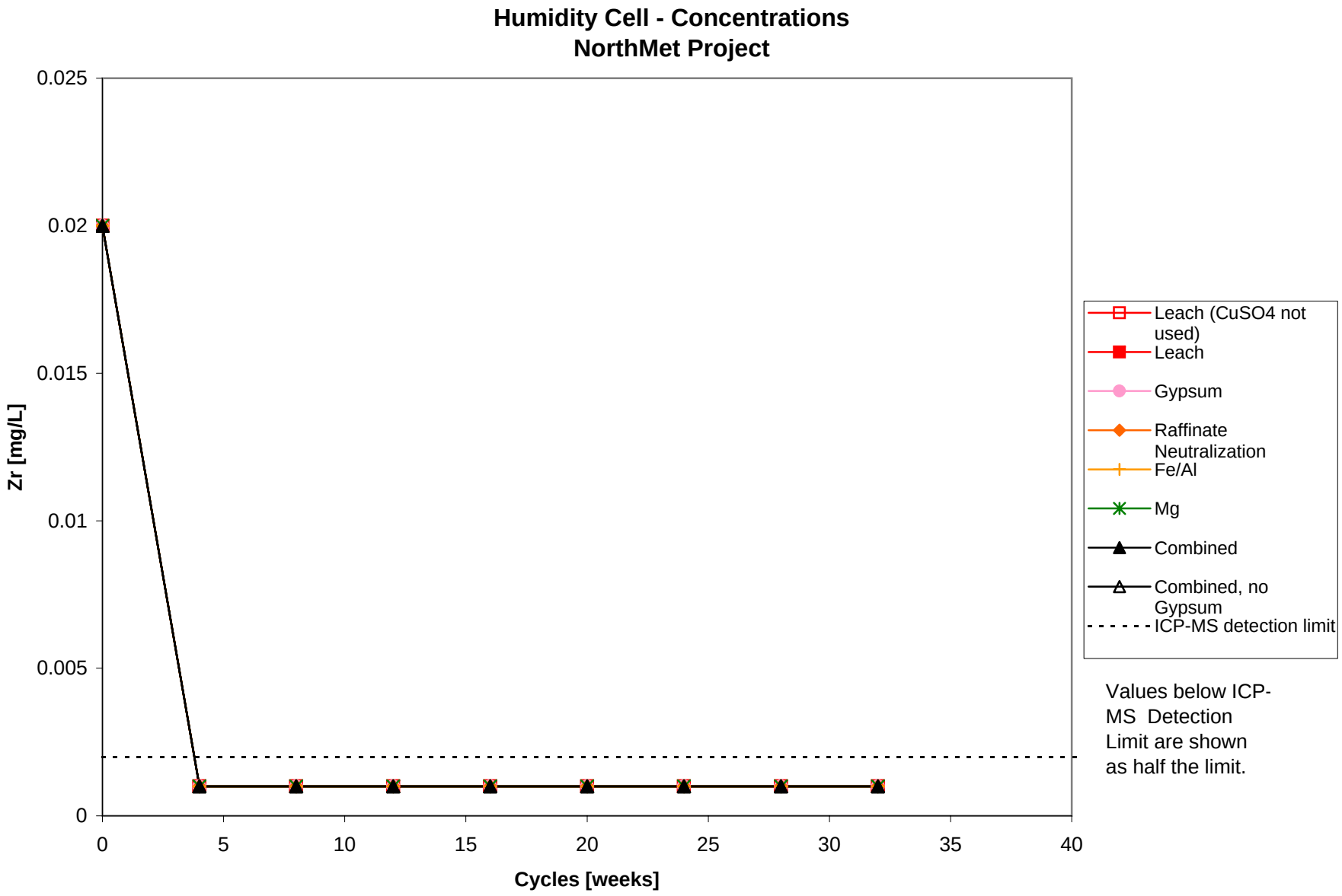

Humidity Cell - Concentrations
NorthMet Project
Sn [mg/L]
0.0045
0.004
0.0035
0.003
0.0025
0.002
0.0015
0.001
0.0005
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.


Humidity Cell - Concentrations
NorthMet Project
Ti [mg/L]
10
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.


Humidity Cell - Concentrations
NorthMet Project
U [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

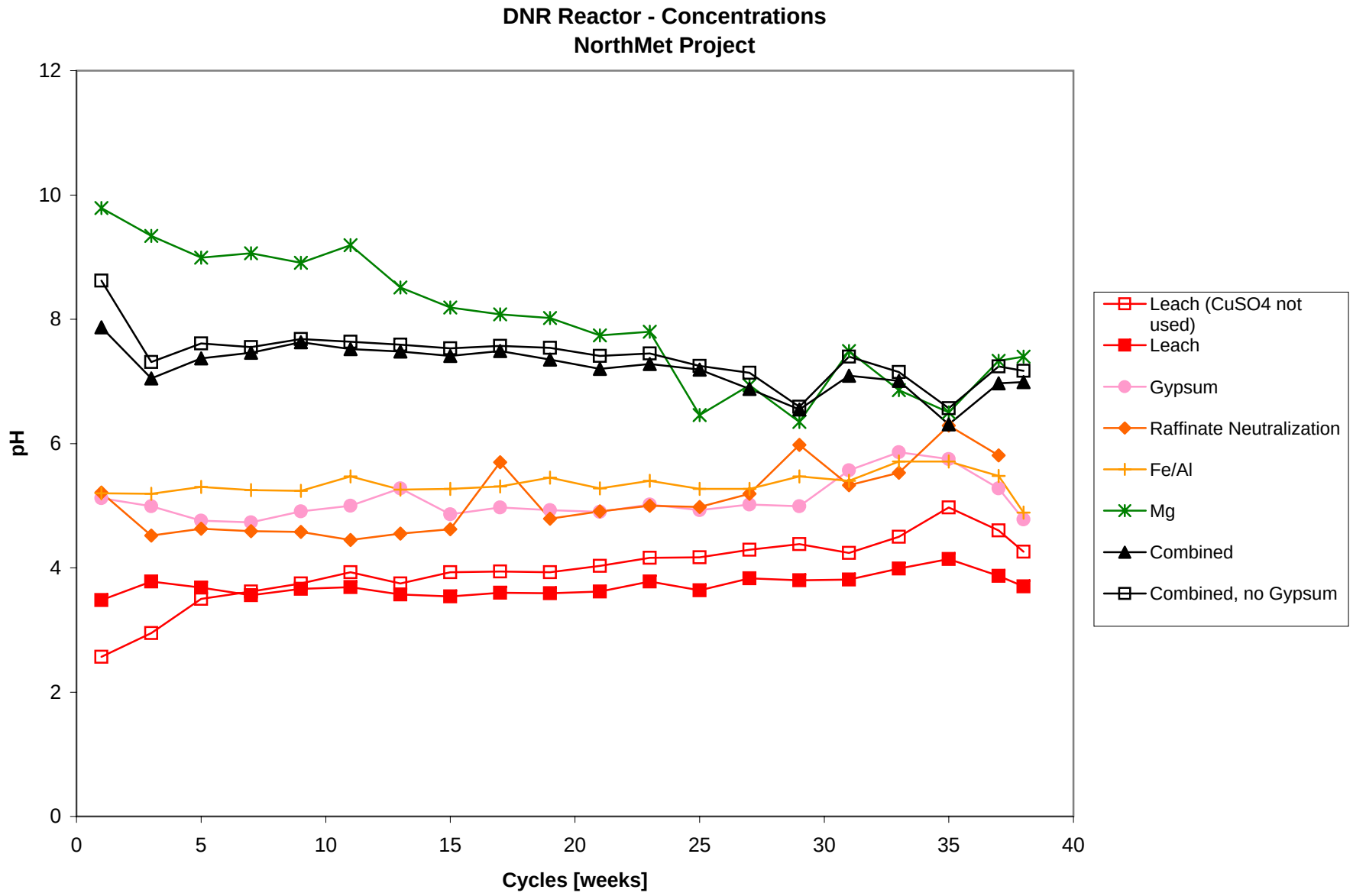

Humidity Cell - Concentrations
NorthMet Project
V [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

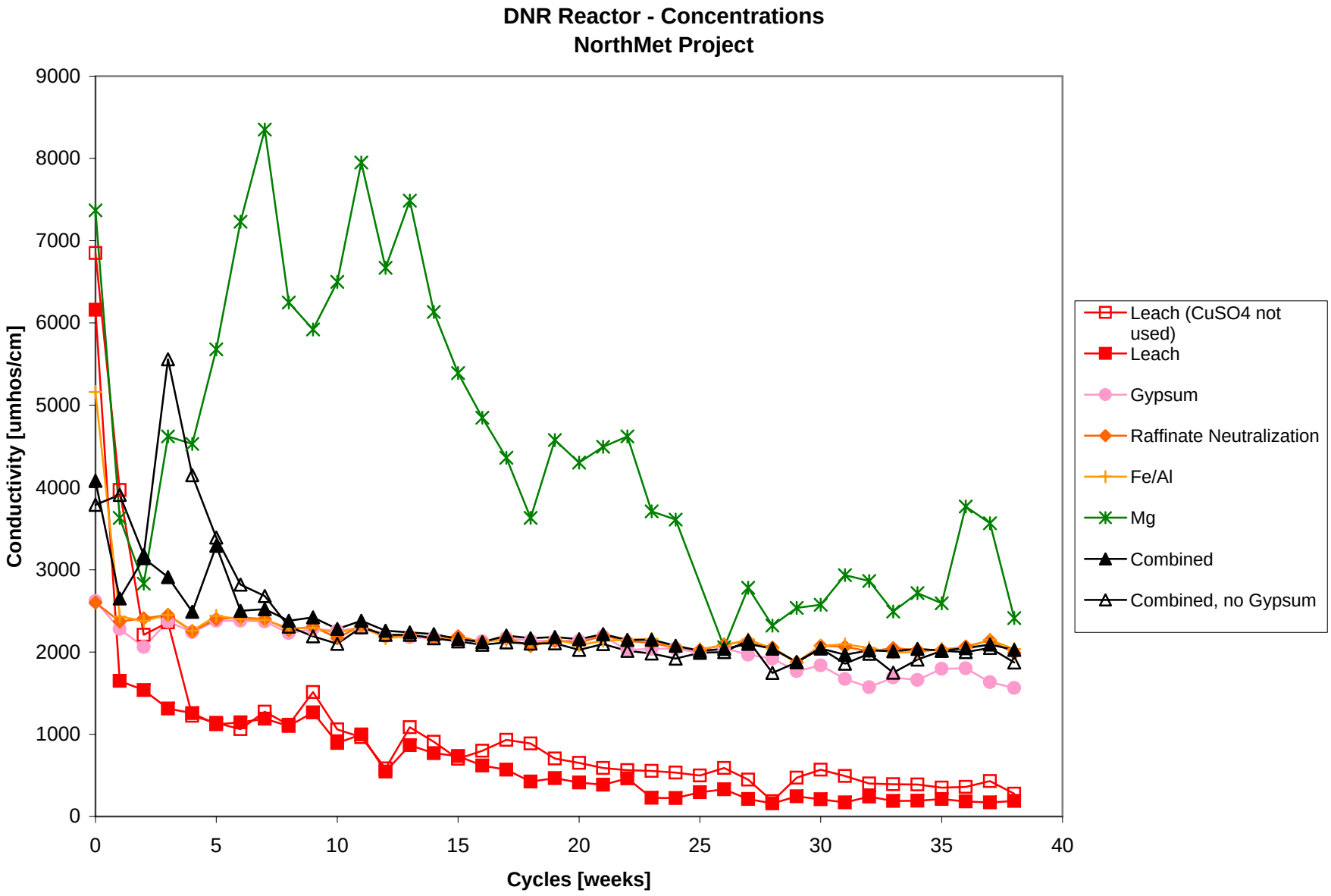

Humidity Cell - Concentrations
NorthMet Project
Zn [mg/L]
1000
100
10
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

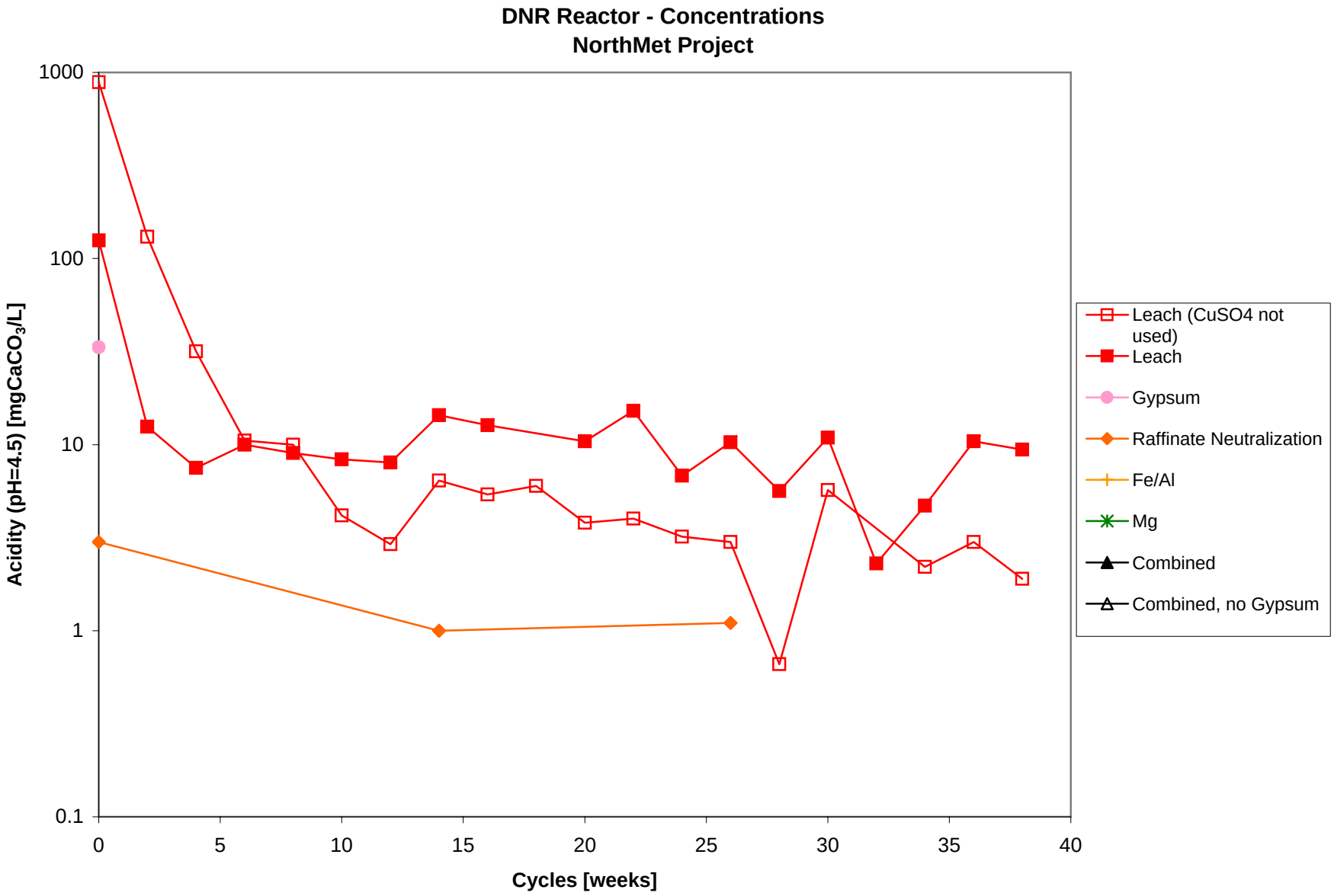

Humidity Cell - Concentrations
NorthMet Project
Zr [mg/L]
0.025
0.02
0.015
0.01
0.005
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

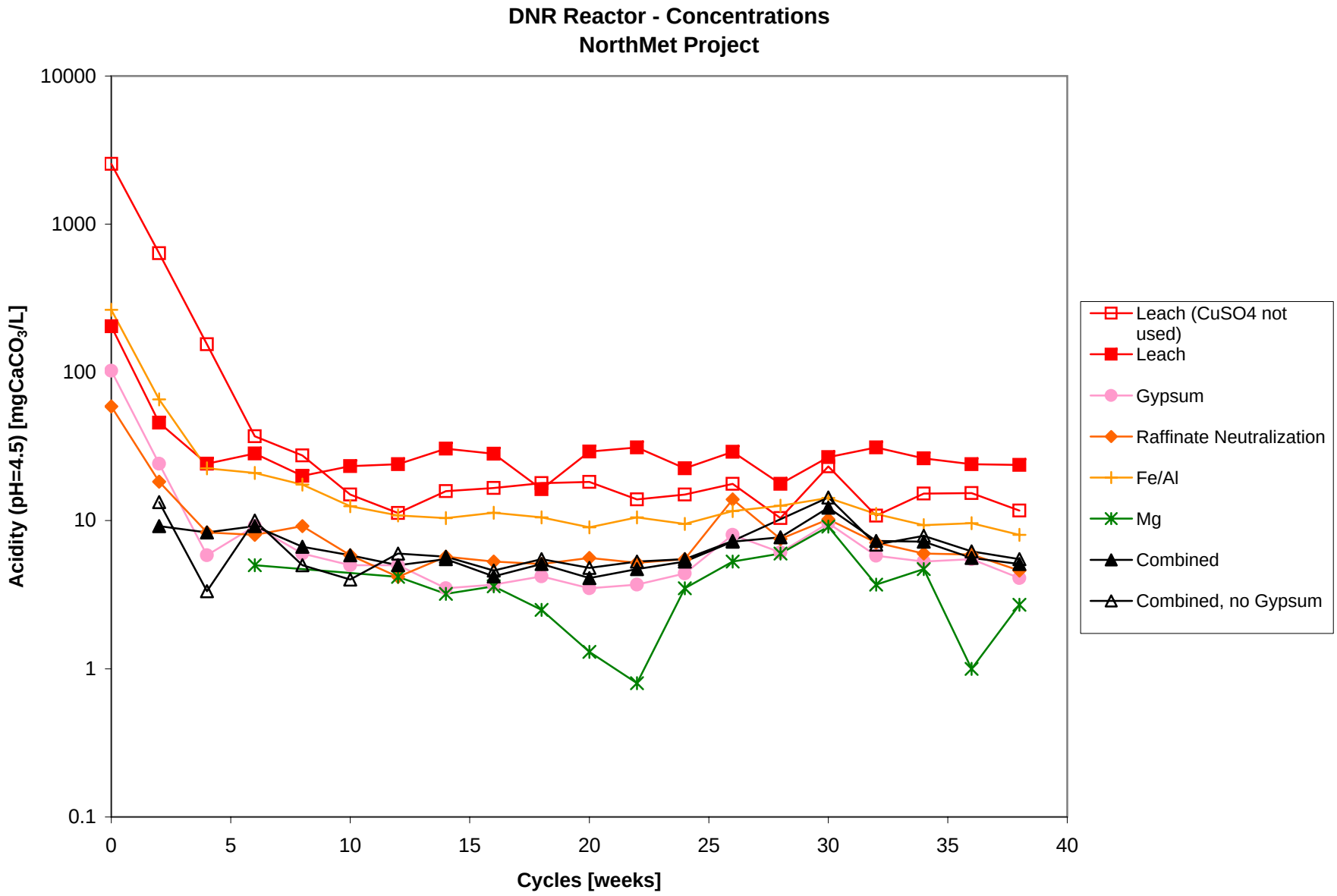
Appendix C.3

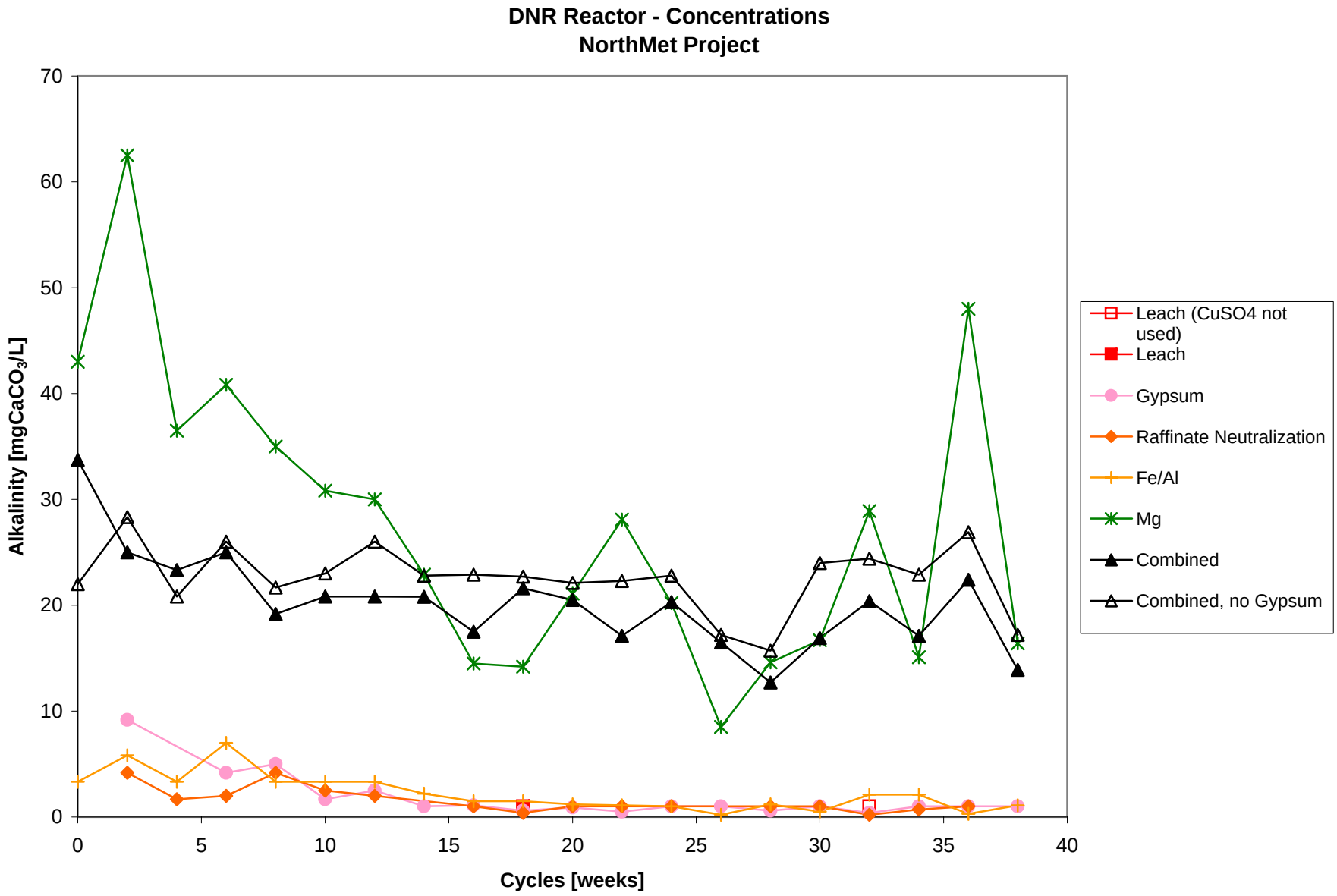
DNR Reactors

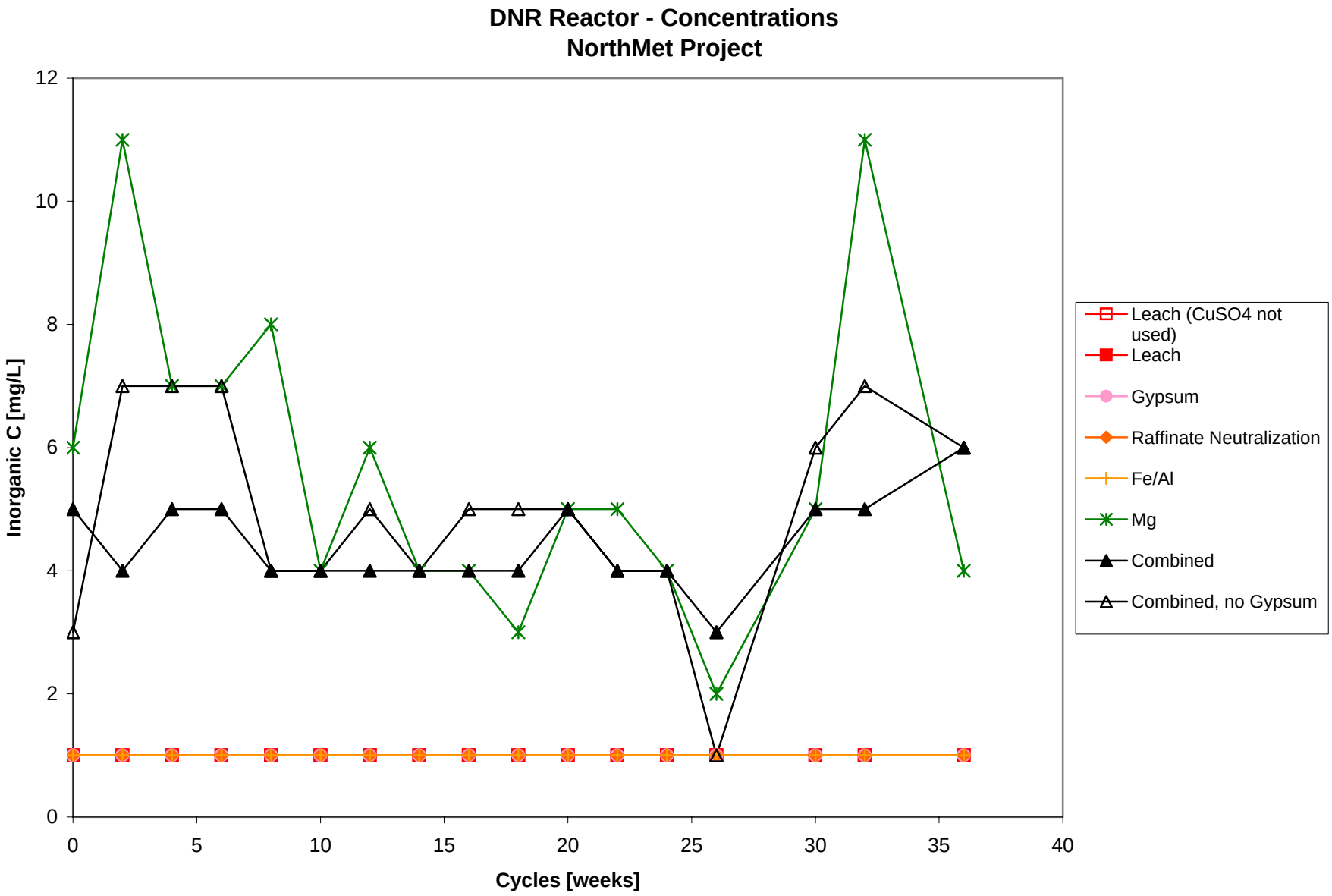

DNR Reactor - Concentrations
NorthMet Project
pH
Cycles [weeks]
0
2
4
6
8
10
12
0
5
10
15
20
25
30
35
40
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

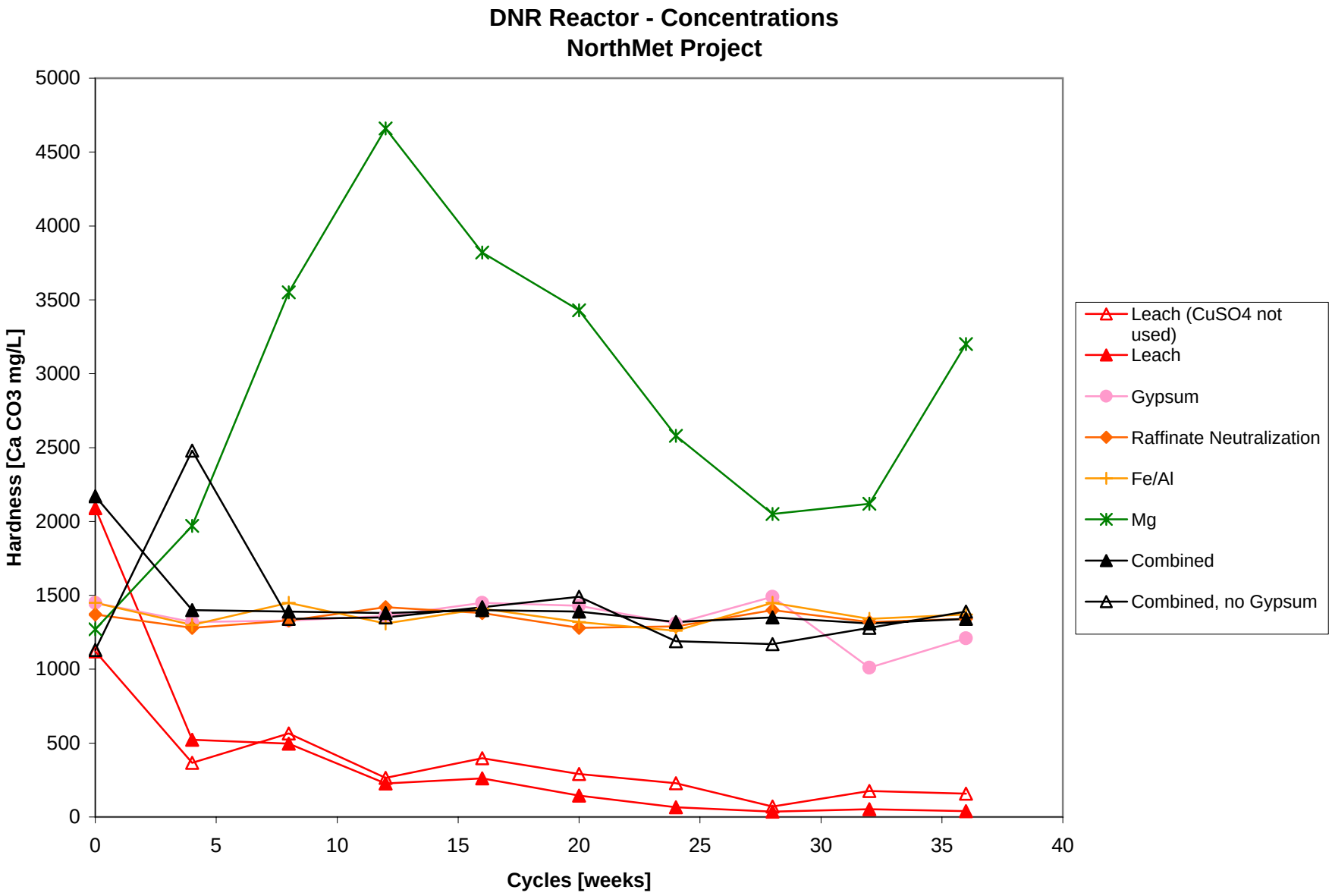

DNR Reactor - Concentrations
NorthMet Project
Conductivity [umhos/cm]
Cycles [weeks]
0
1000
2000
3000
4000
5000
6000
7000
8000
9000
0
5
10
15
20
25
30
35
40
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

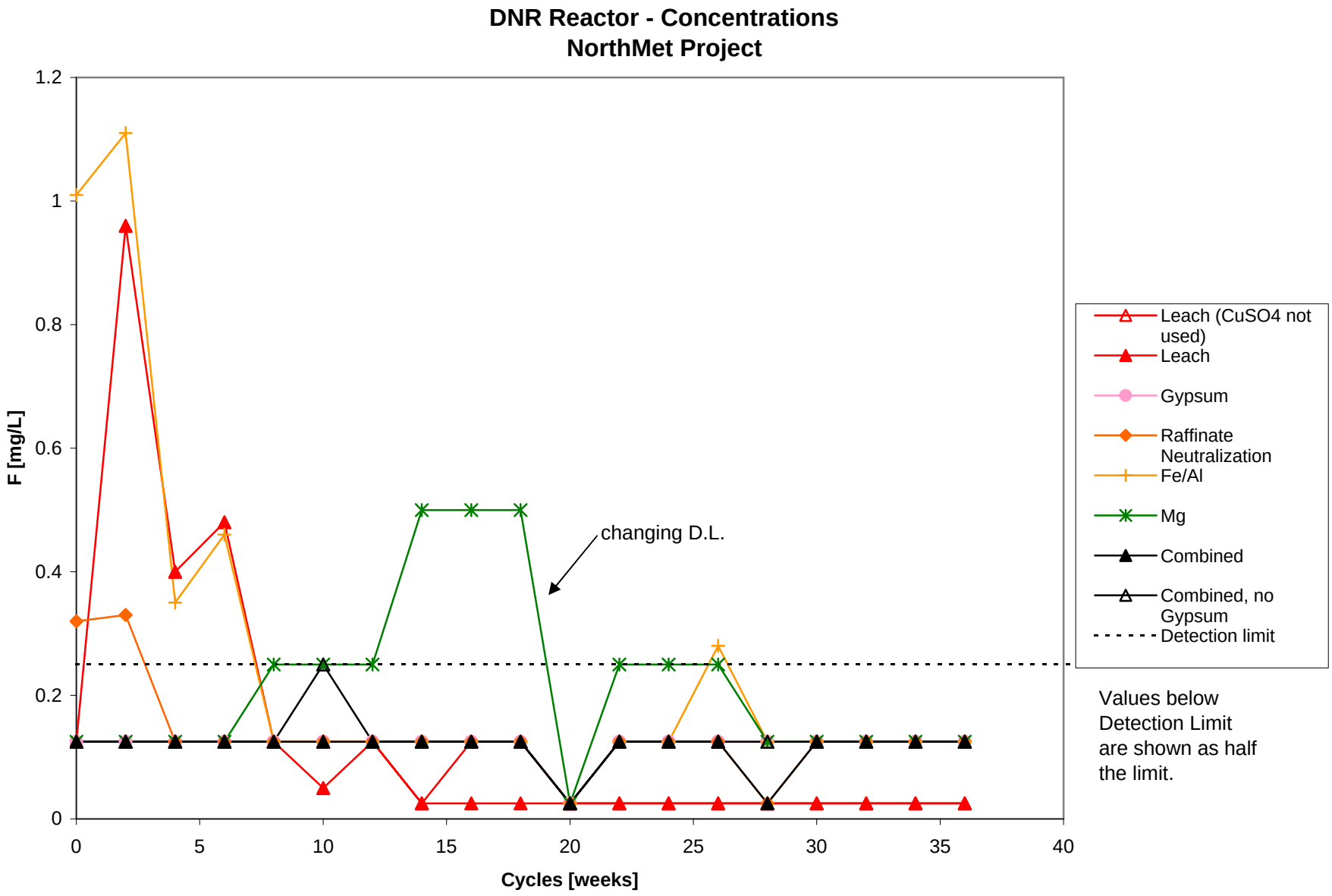

DNR Reactor - Concentrations
NorthMet Project
Acidity (pH=4.5) [mgCaCO3/L]
1000
100
10
1
0.1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

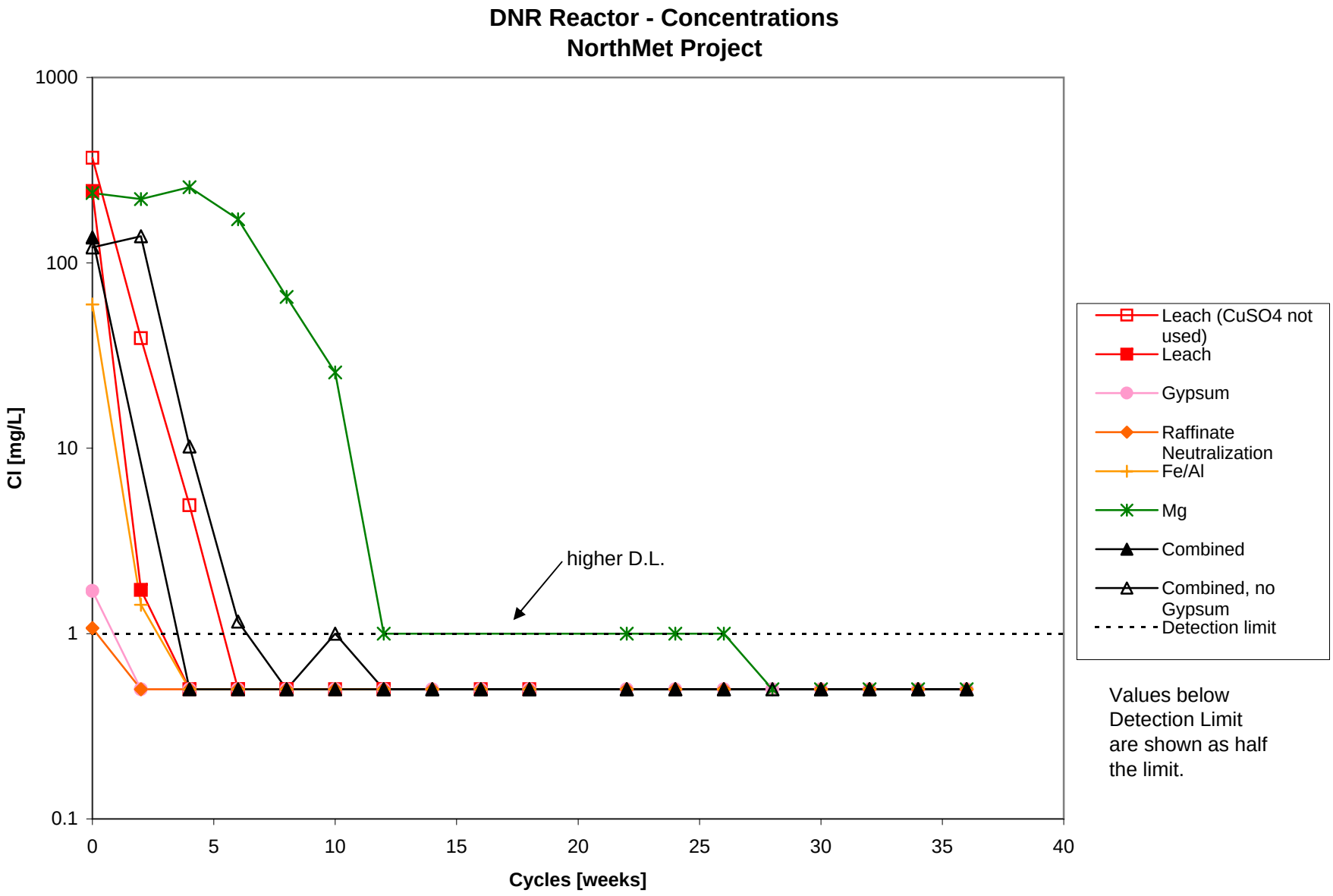

DNR Reactor - Concentrations
NorthMet Project
Acidity (pH=4.5) [mgCaCO3/L]
10000
1000
100
10
1
0.1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

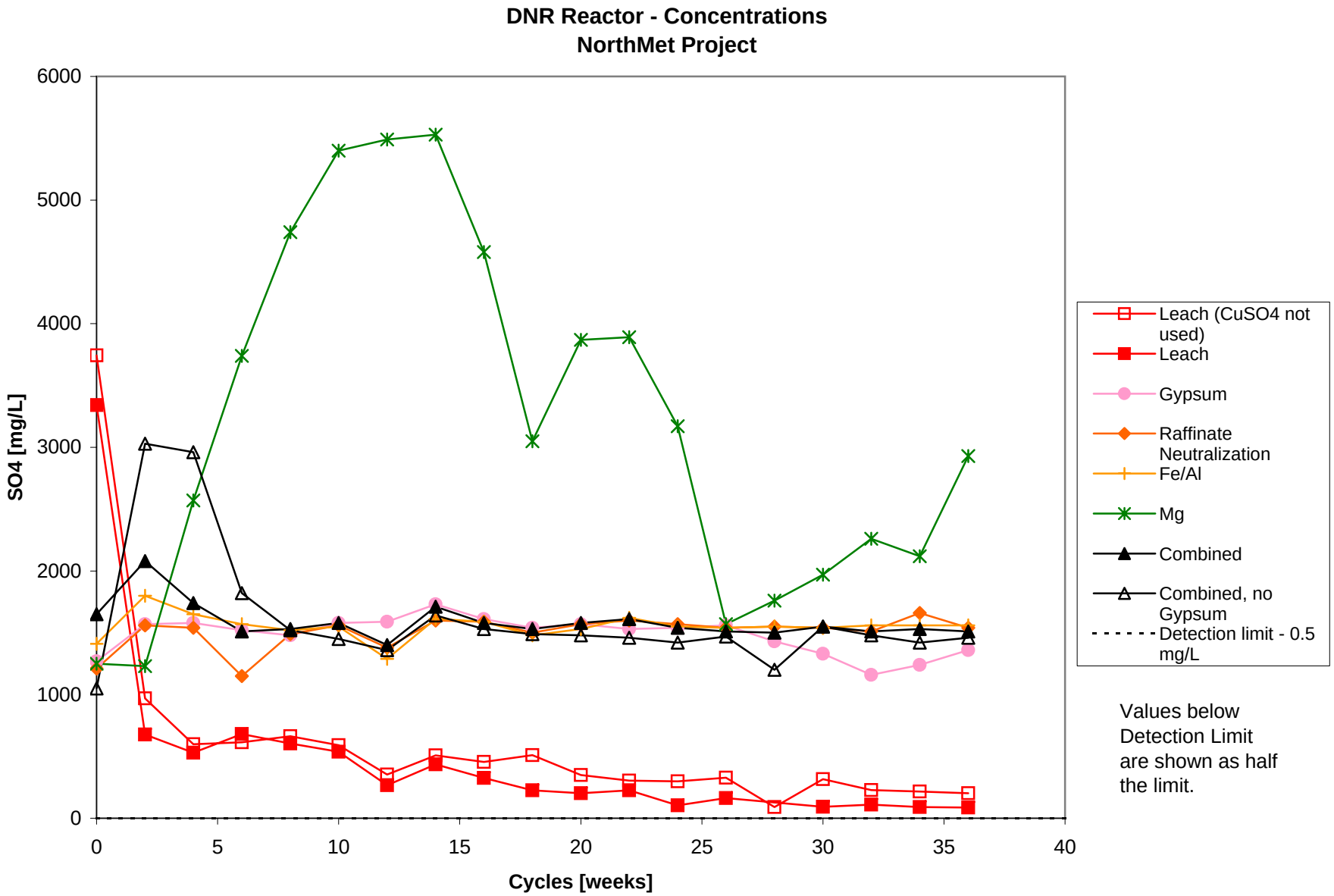

DNR Reactor - Concentrations
NorthMet Project
Alkalinity [mgCaCO3/L]
Cycles [weeks]
0
10
20
30
40
50
60
70
0
5
10
15
20
25
30
35
40
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum


DNR Reactor - Concentrations
NorthMet Project
Inorganic C [mg/L]
Cycles [weeks]
0
2
4
6
8
10
12
0
5
10
15
20
25
30
35
40
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

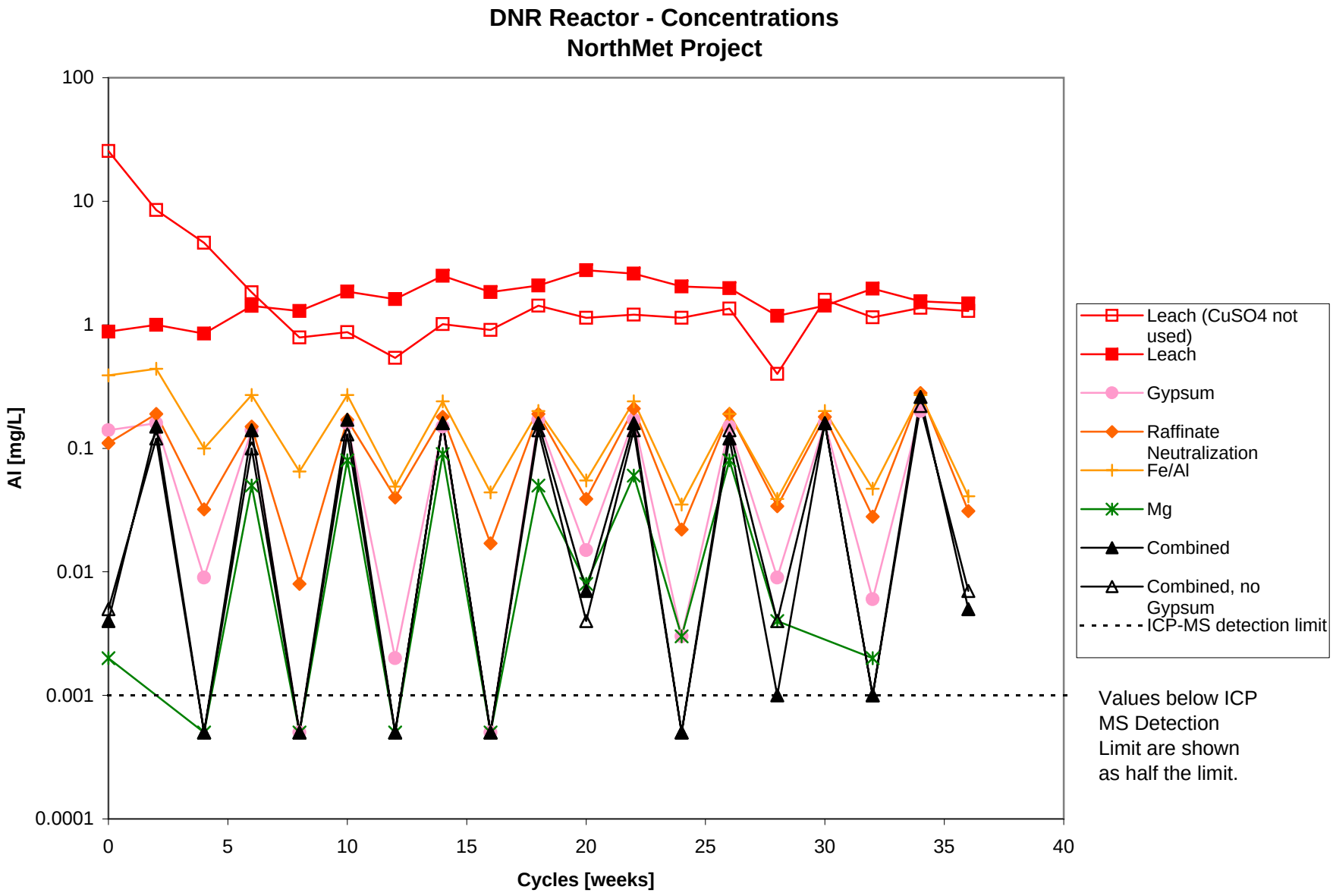

DNR Reactor - Concentrations
NorthMet Project
Hardness [Ca CO3 mg/L]
5000
4500
4000
3500
3000
2500
2000
1500
1000
500
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum

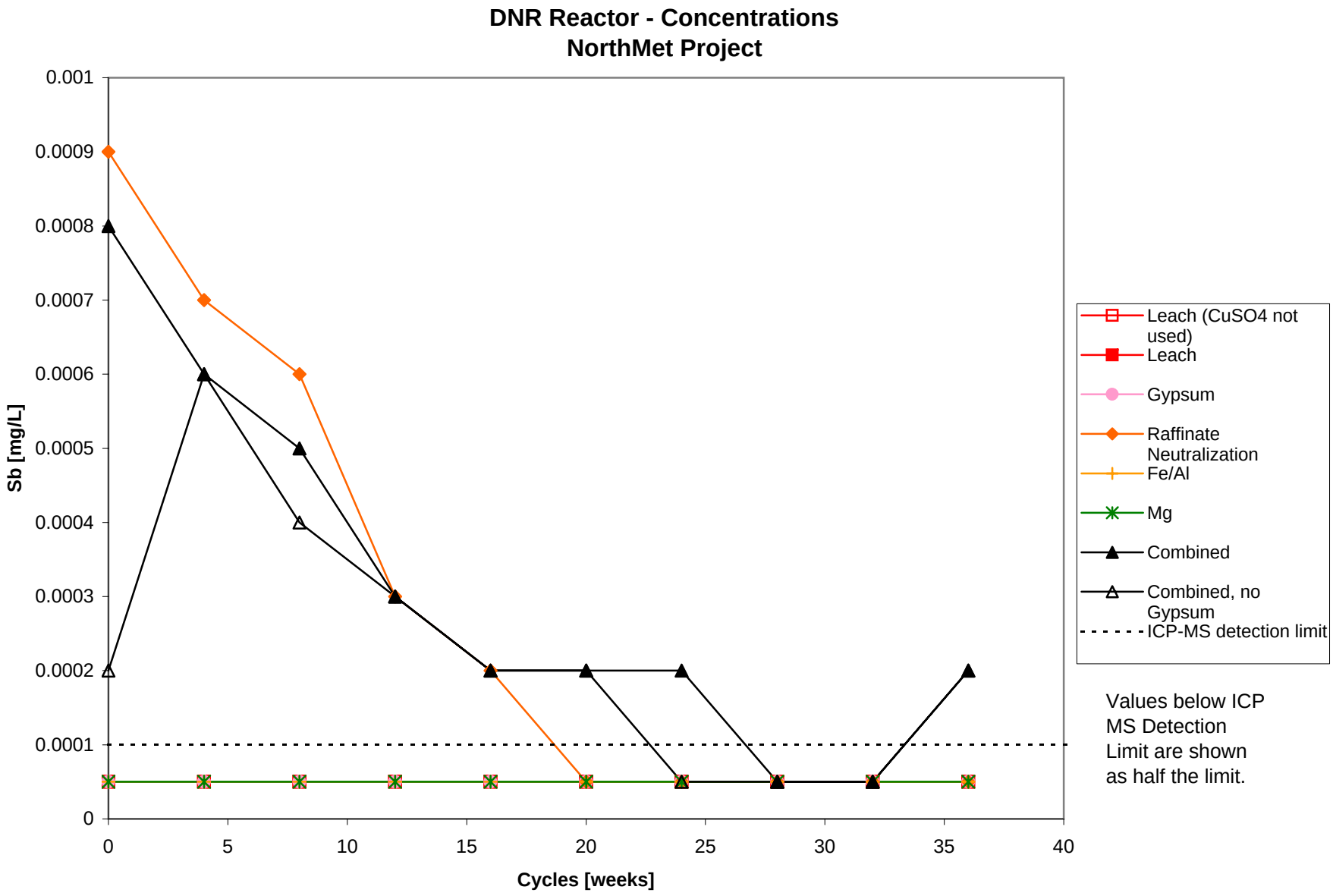

DNR Reactor - Concentrations
NorthMet Project
F [mg/L]
Cycles [weeks]
changing D.L.
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
Detection limit
Values below Detection Limit are shown as half the limit.

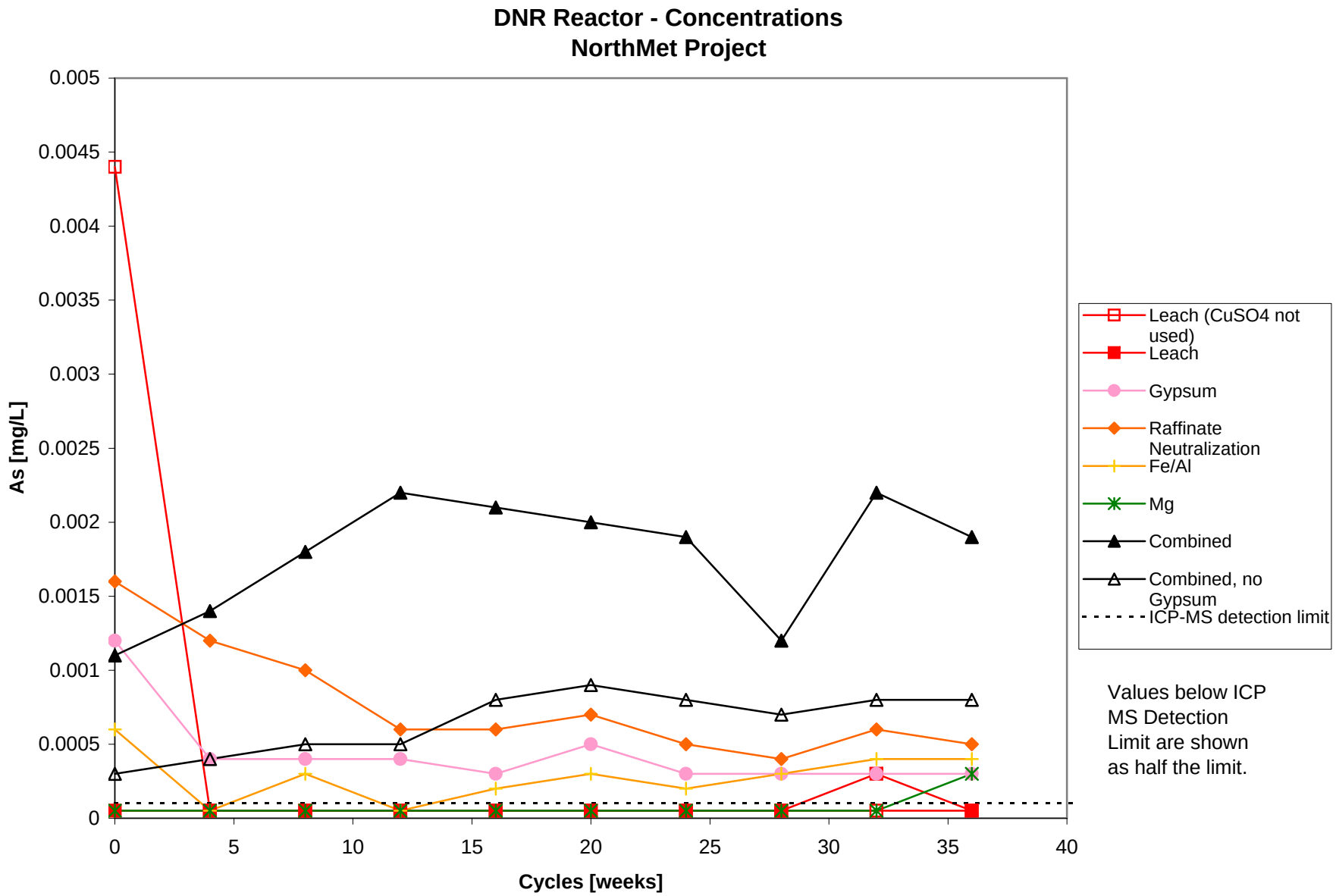

DNR Reactor - Concentrations
NorthMet Project
Cl [mg/L]
1000
100
10
1
0.1
0
5
10
15
20
25
30
35
40
Cycles [weeks]
higher D.L.
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
Detection limit
Values below Detection Limit are shown as half the limit.

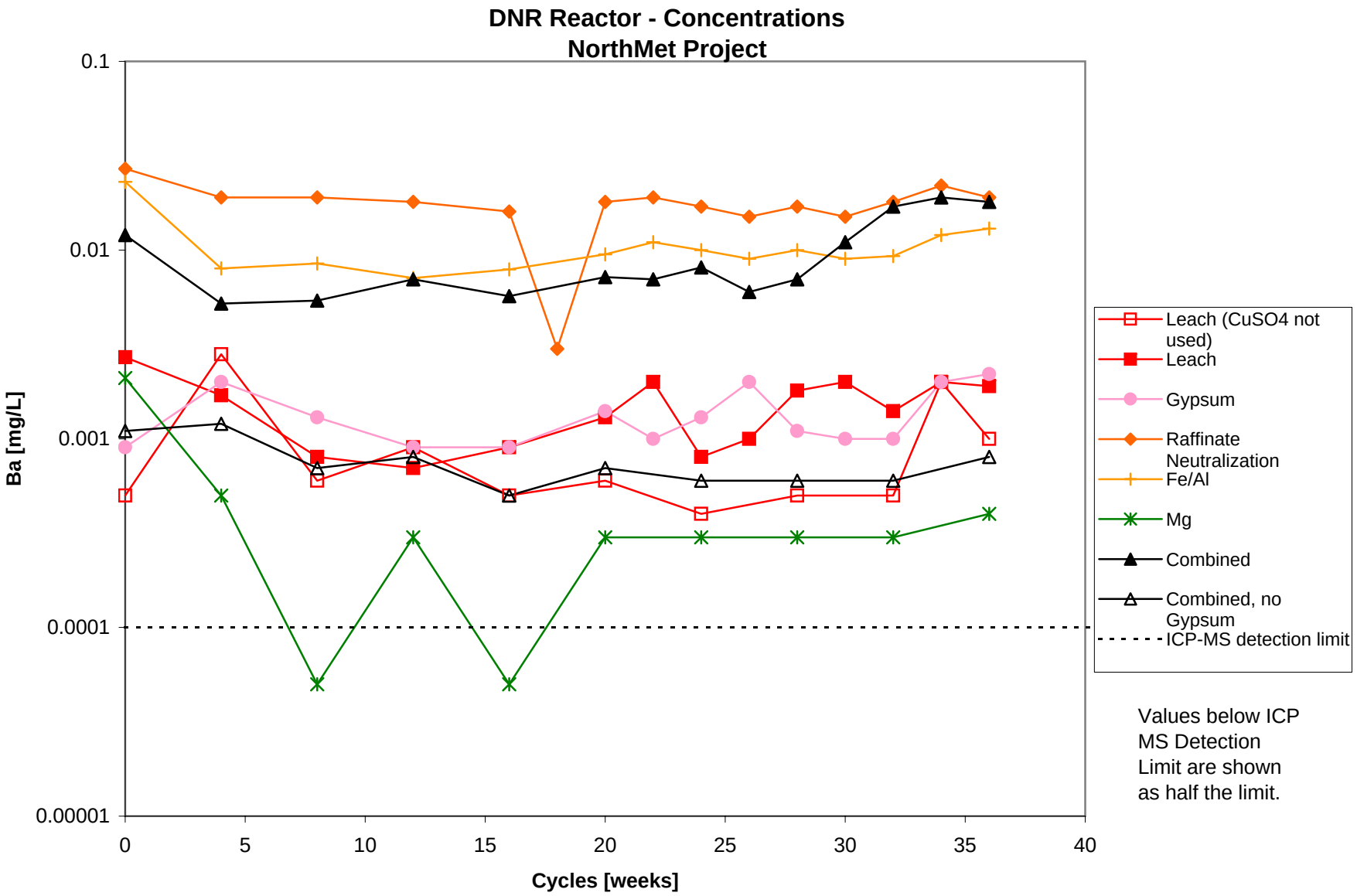


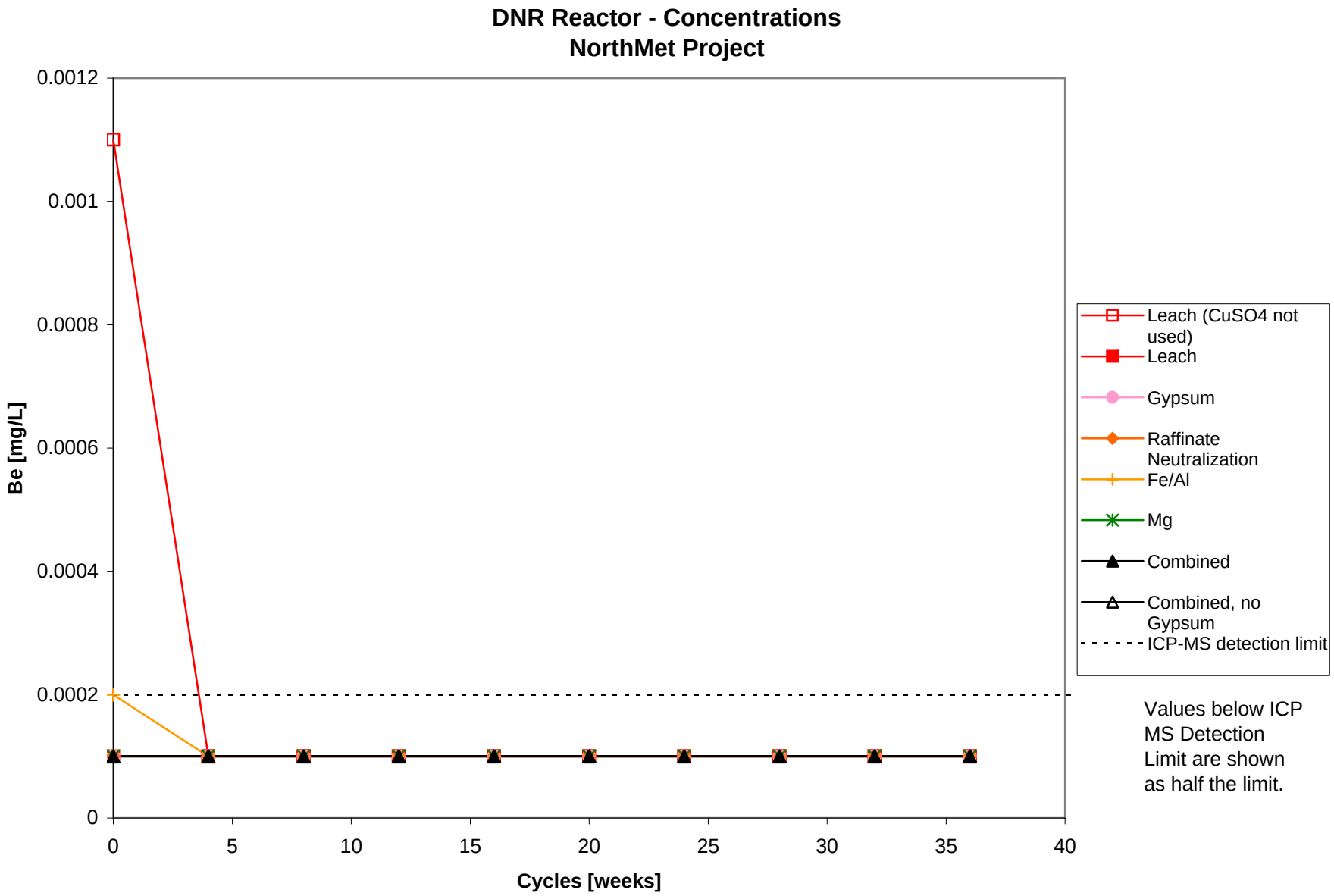
Values below Detection Limit are shown as half the limit.

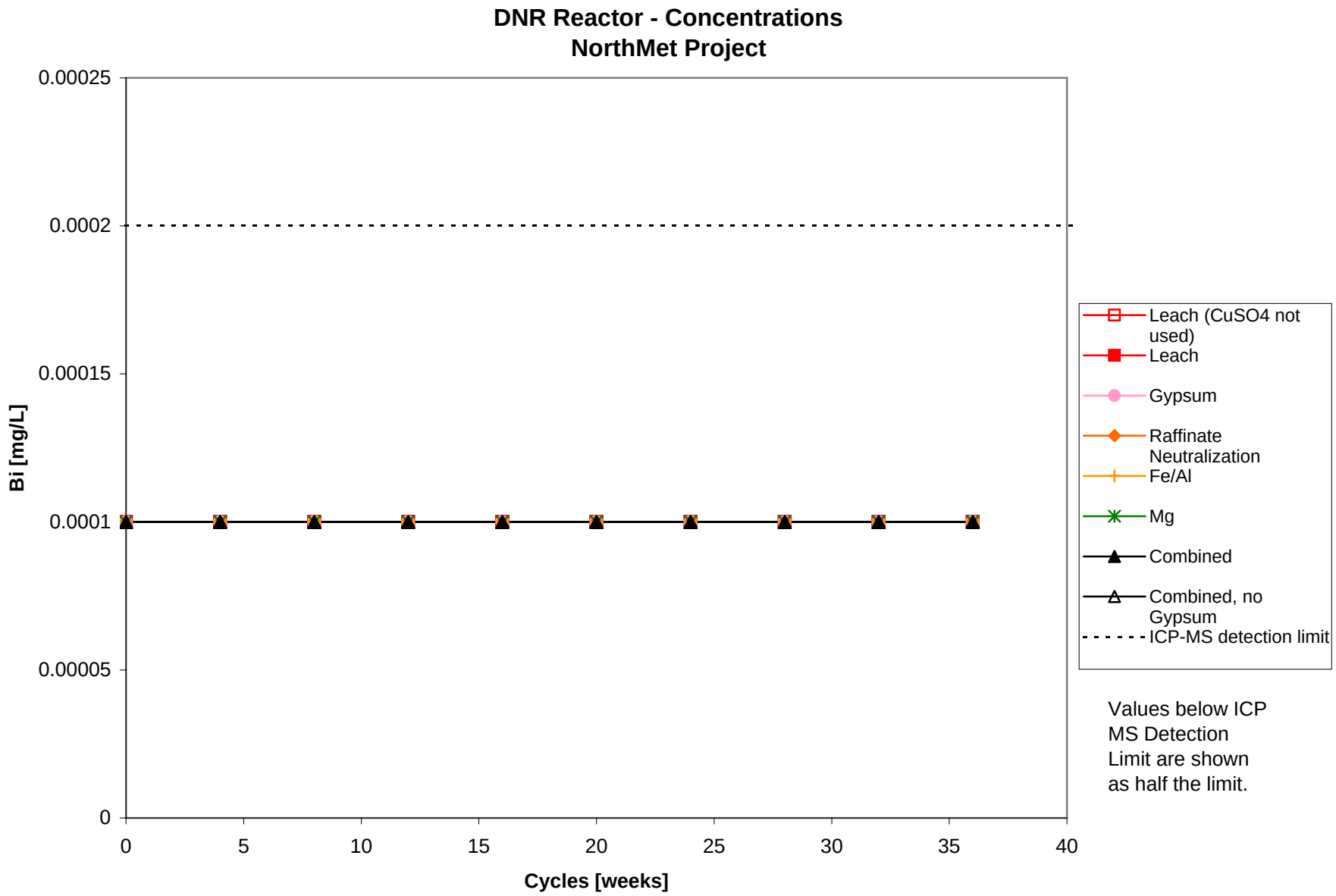

DNR Reactor - Concentrations
NorthMet Project
Al [mg/L]
100
10
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate
Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

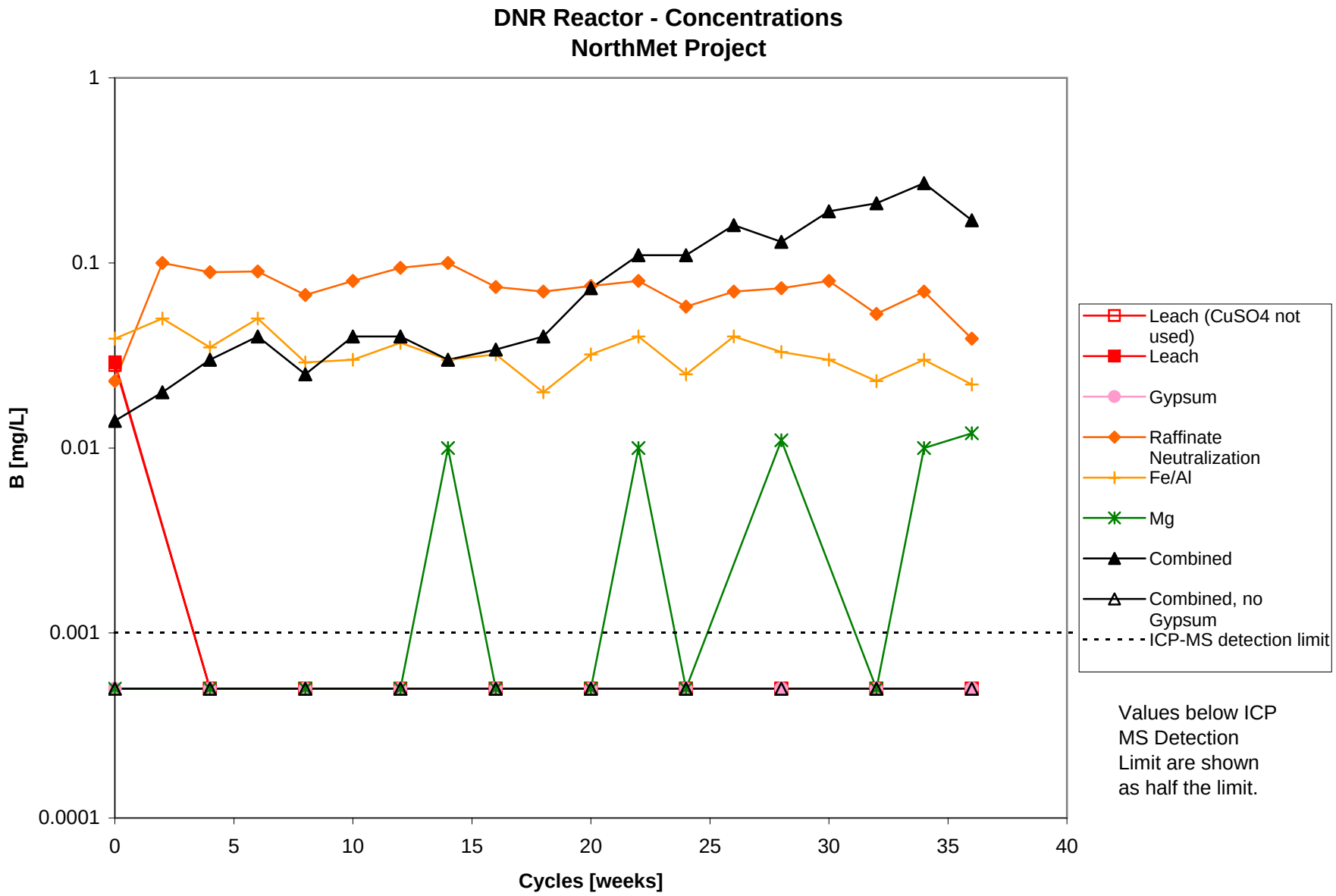

DNR Reactor - Concentrations
NorthMet Project
Sb [mg/L]
0.001
0.0009
0.0008
0.0007
0.0006
0.0005
0.0004
0.0003
0.0002
0.0001
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

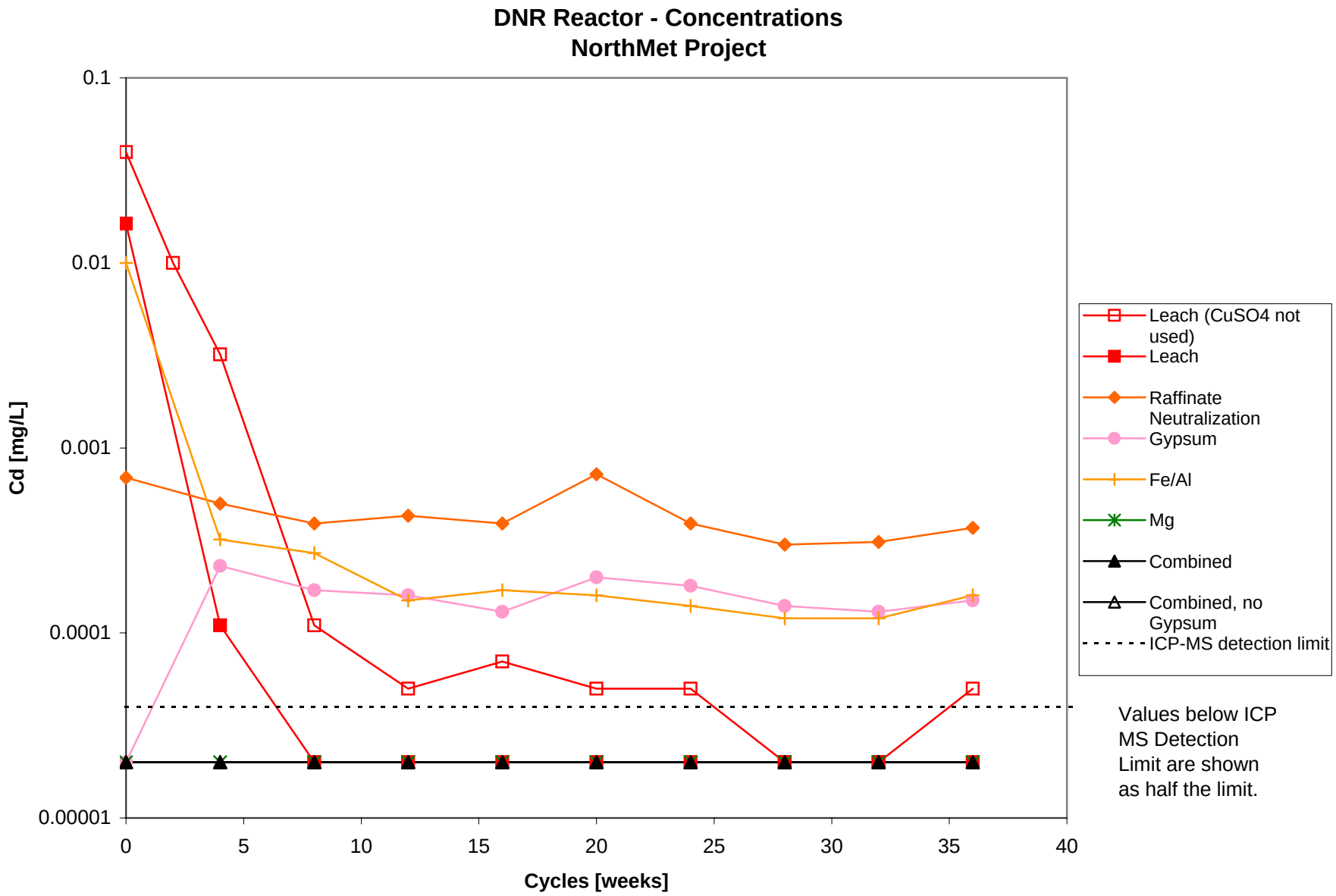

DNR Reactor - Concentrations
NorthMet Project
As [mg/L]
0.005
0.0045
0.004
0.0035
0.003
0.0025
0.002
0.0015
0.001
0.0005
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

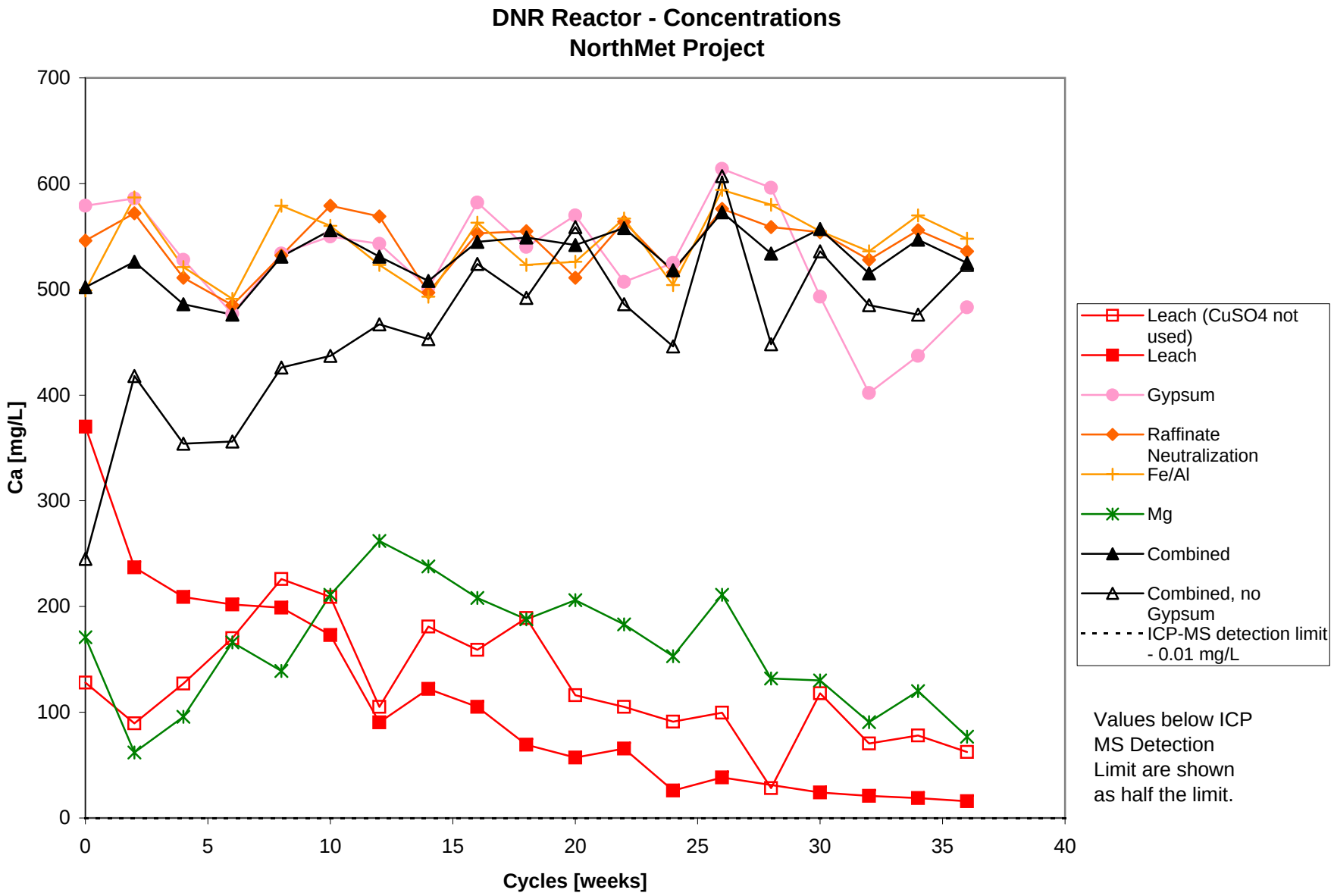

DNR Reactor - Concentrations
NorthMet Project
Ba [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.


DNR Reactor - Concentrations
NorthMet Project
Be [mg/L]
0.0012
0.001
0.0008
0.0006
0.0004
0.0002
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

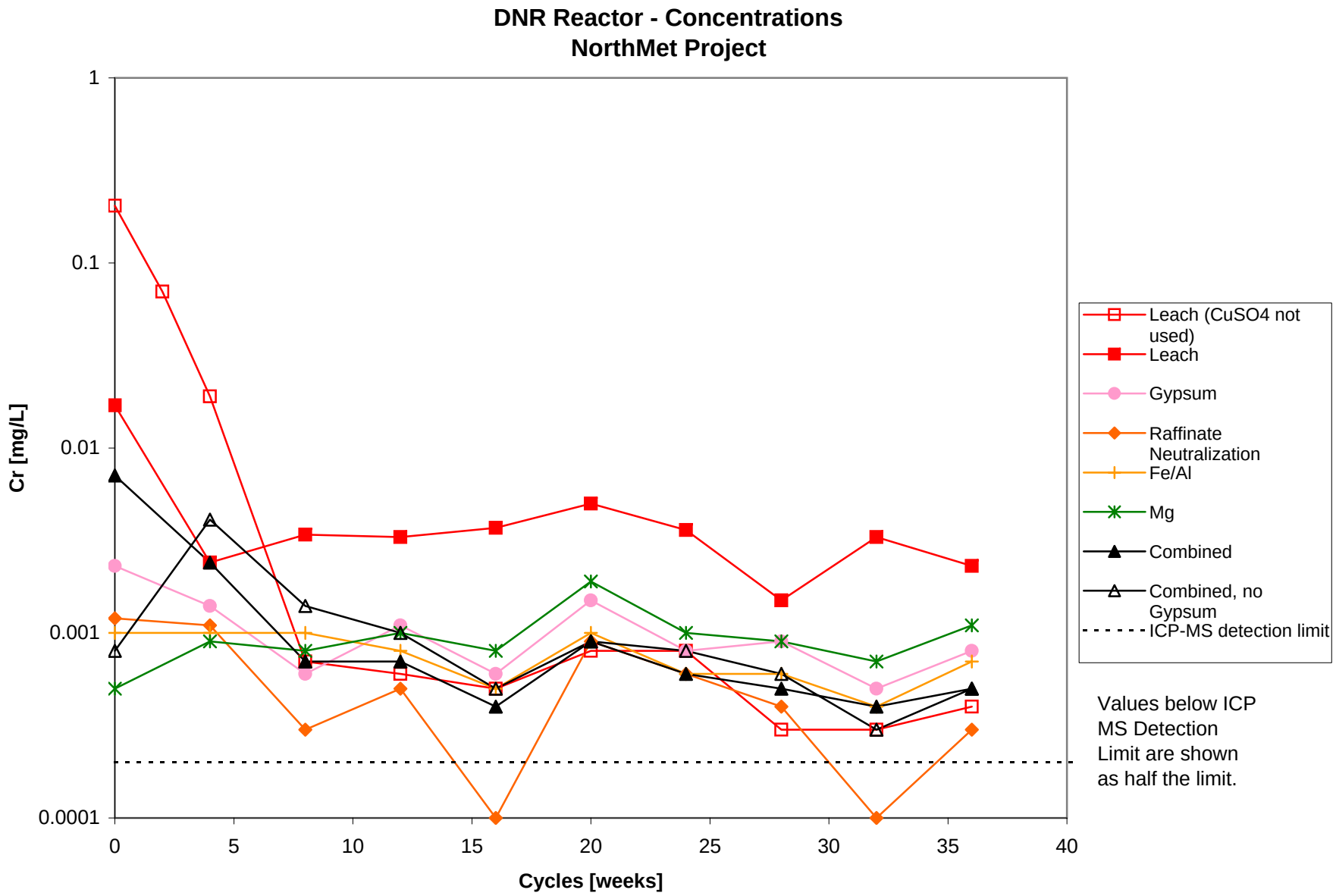

DNR Reactor - Concentrations
NorthMet Project
Bi [mg/L]
0.00025
0.0002
0.00015
0.0001
0.00005
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

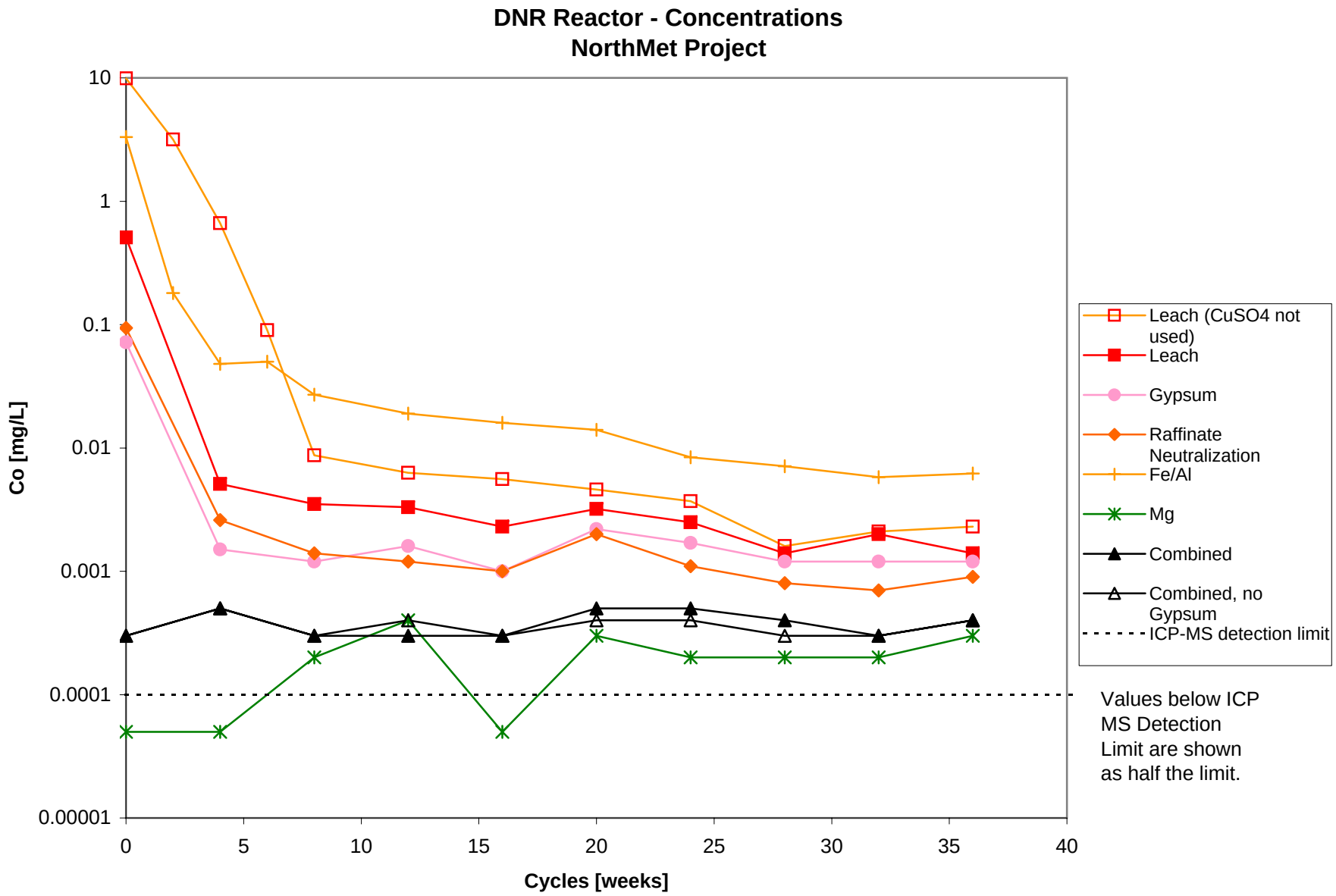

DNR Reactor - Concentrations
NorthMet Project
B [mg/L]
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

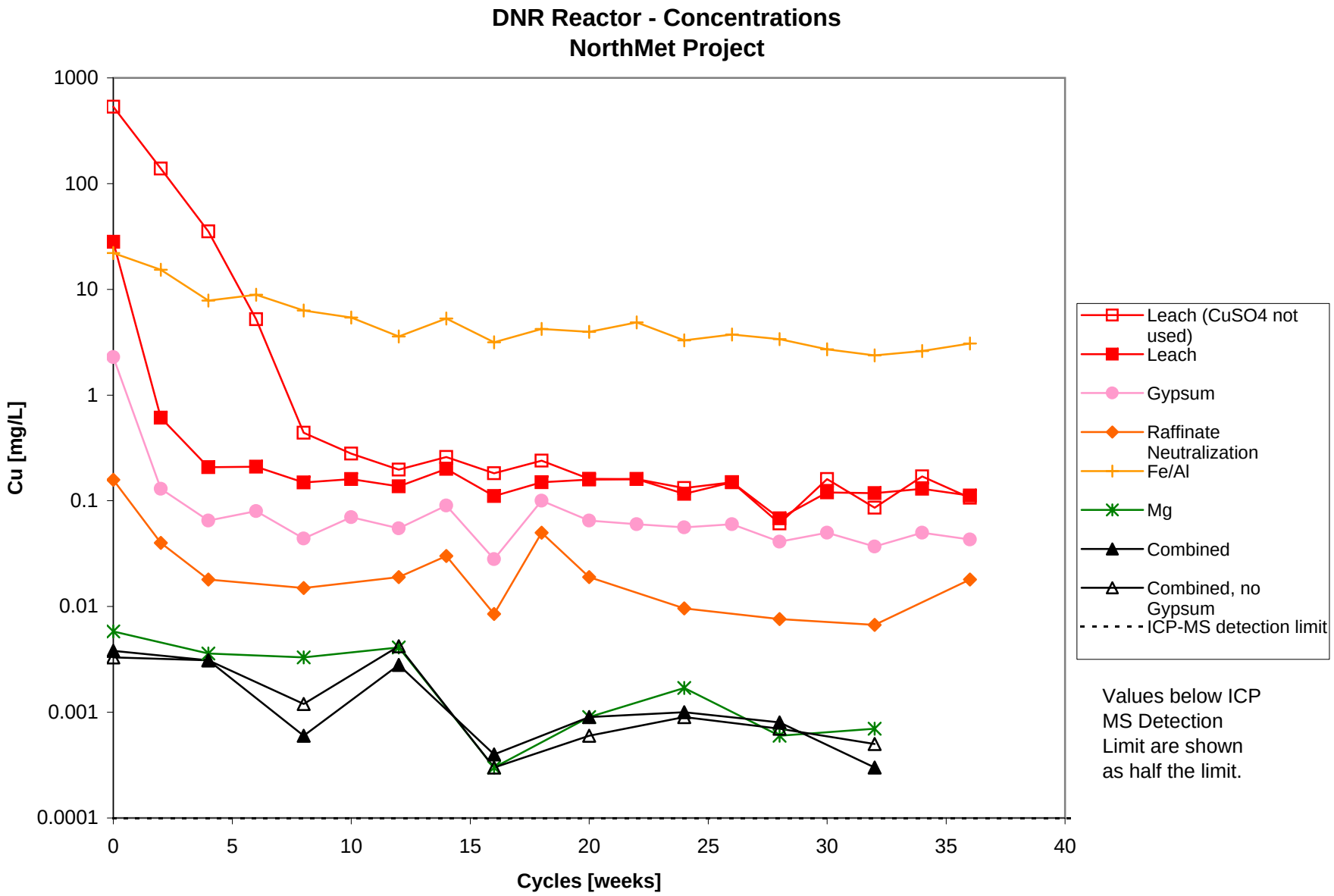

DNR Reactor - Concentrations
NorthMet Project
Cd [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Raffinate Neutralization
Gypsum
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

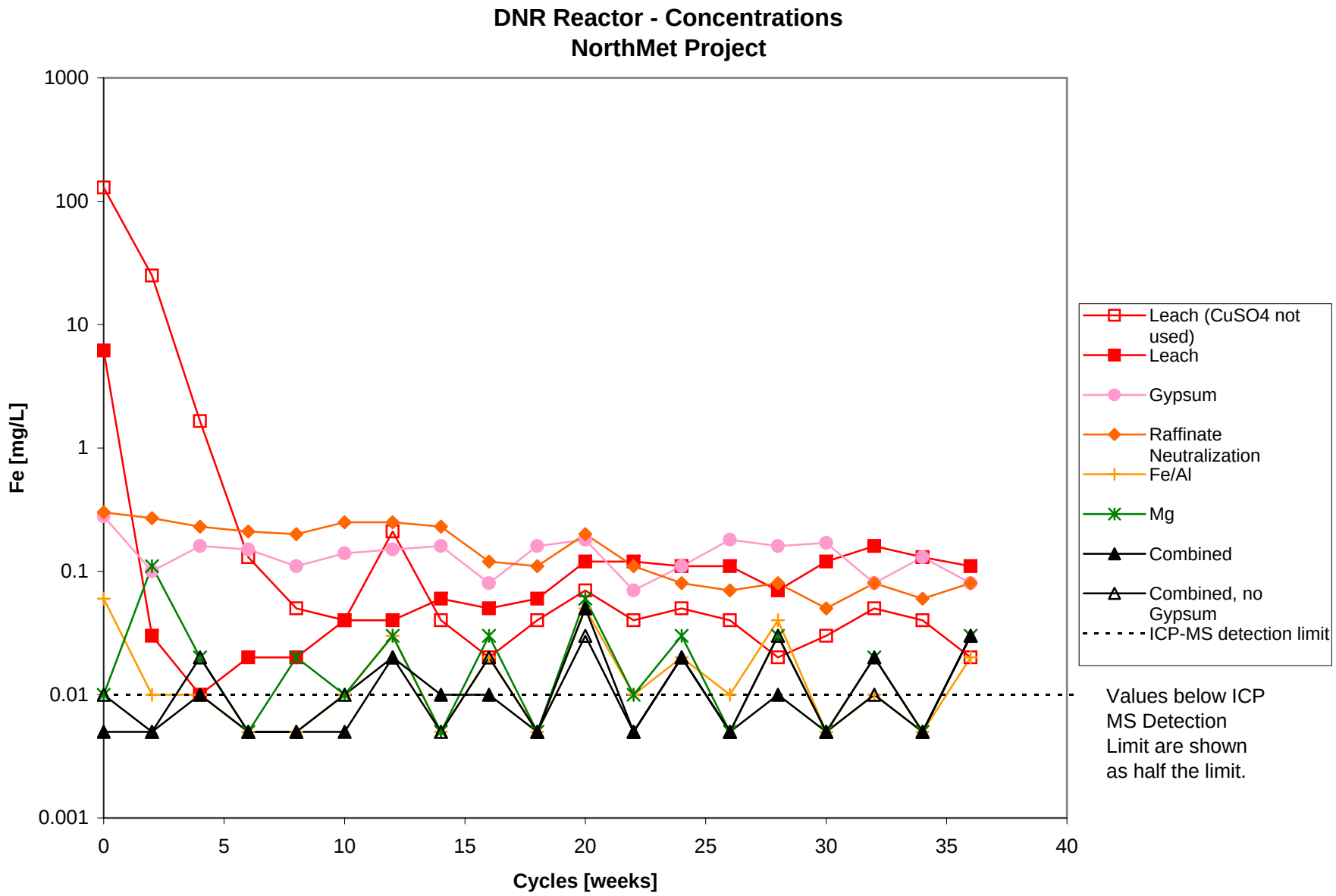


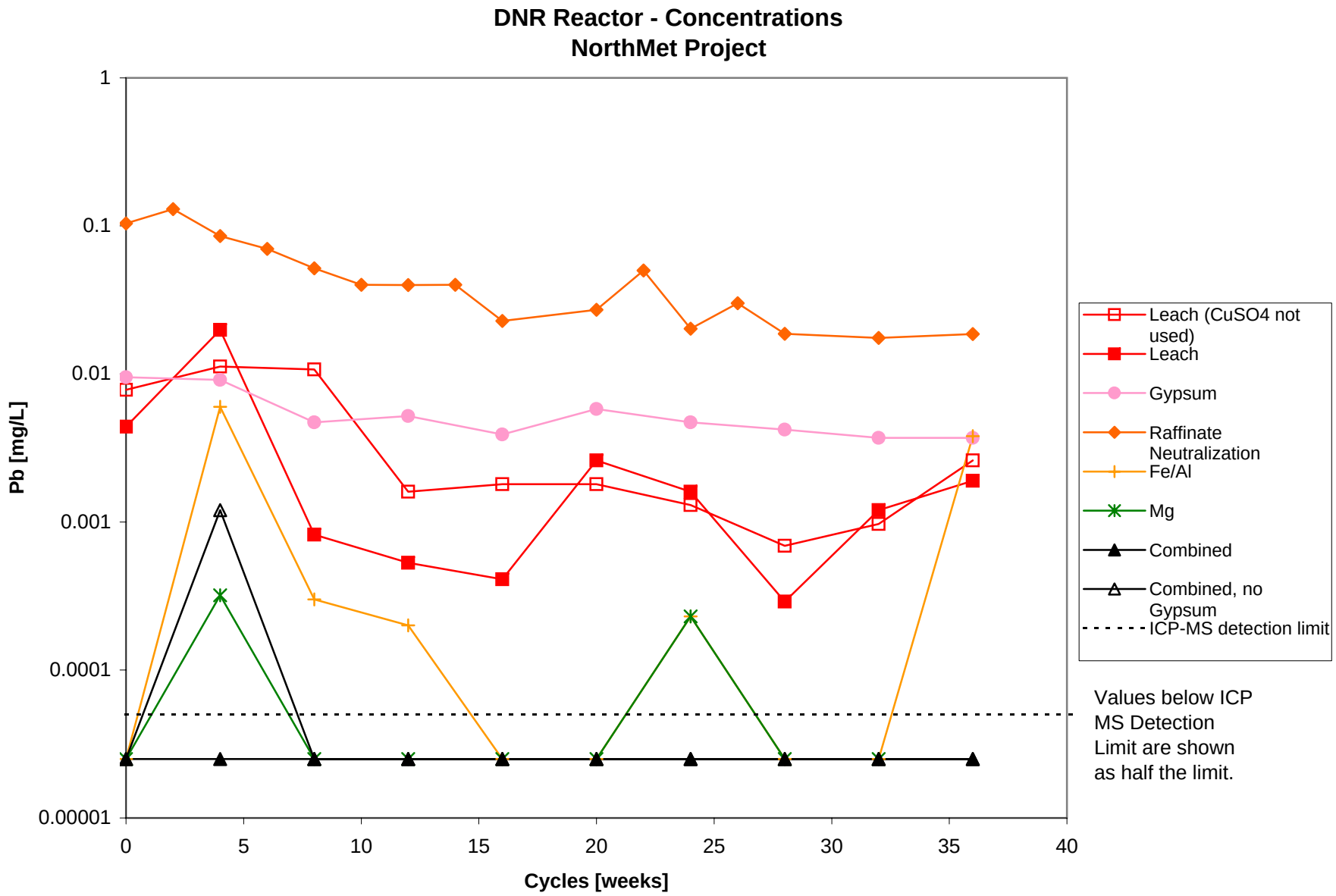
Values below ICP MS Detection Limit are shown as half the limit.

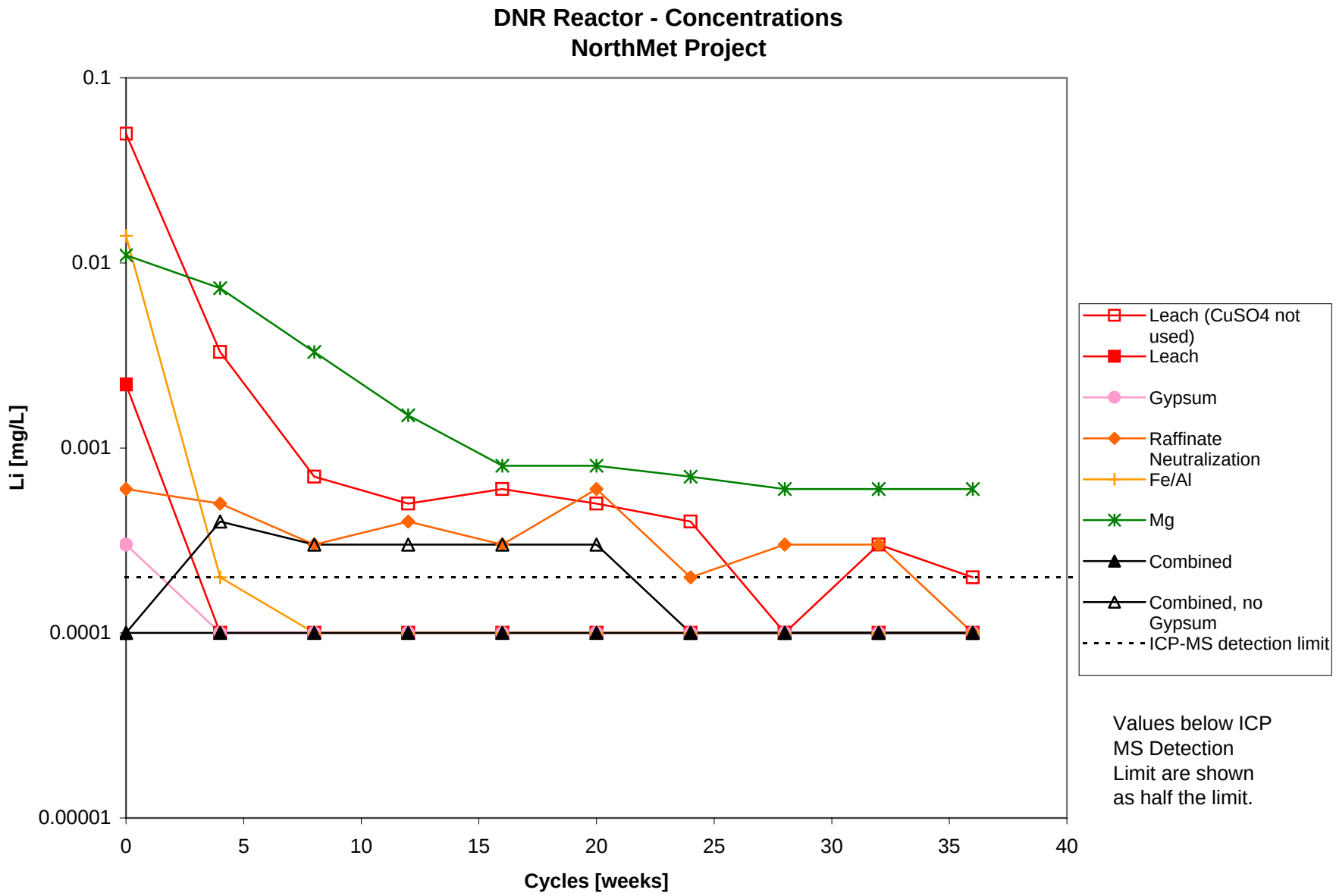

DNR Reactor - Concentrations
NorthMet Project
Cr [mg/L]
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

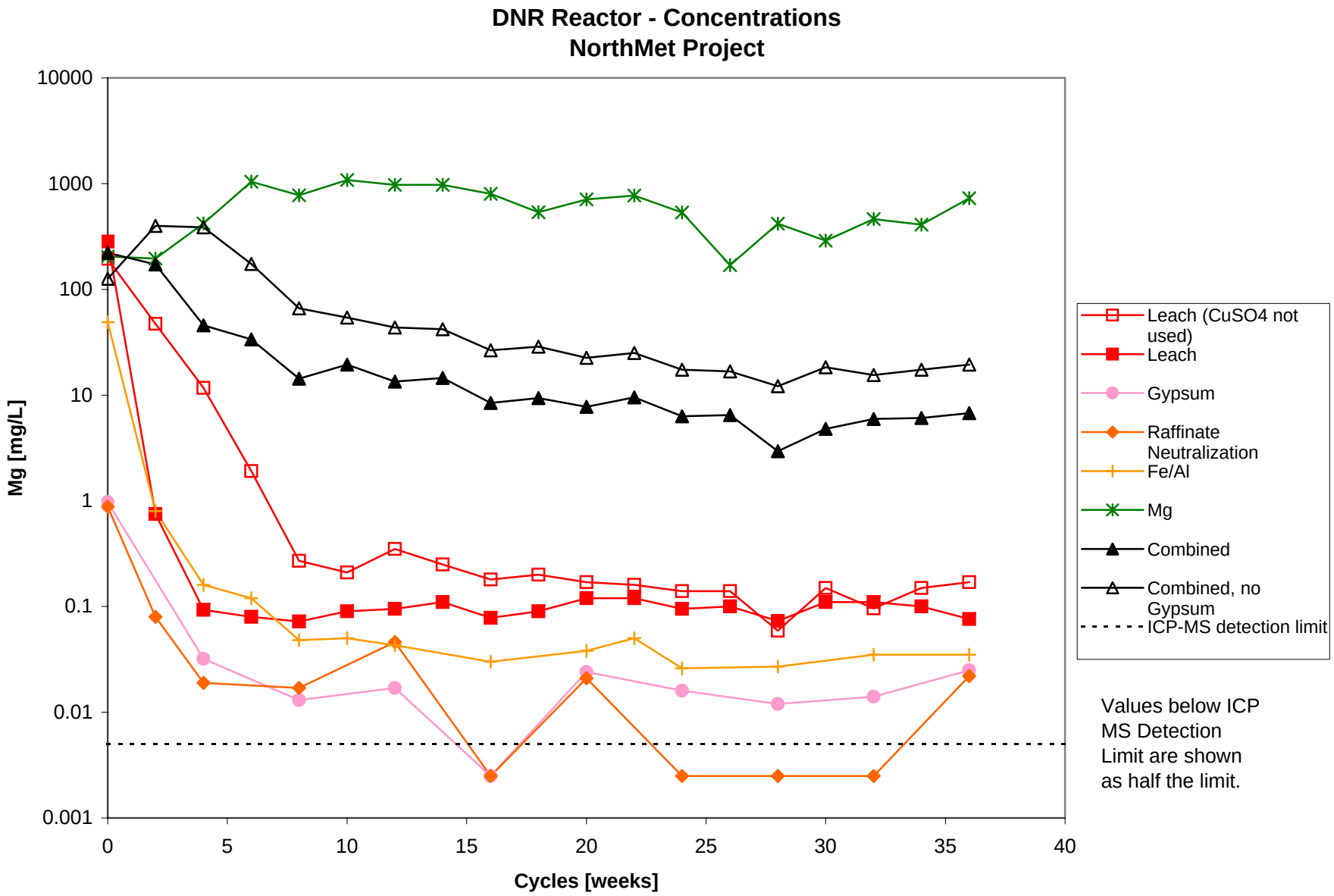

DNR Reactor - Concentrations
NorthMet Project
Co [mg/L]
10
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

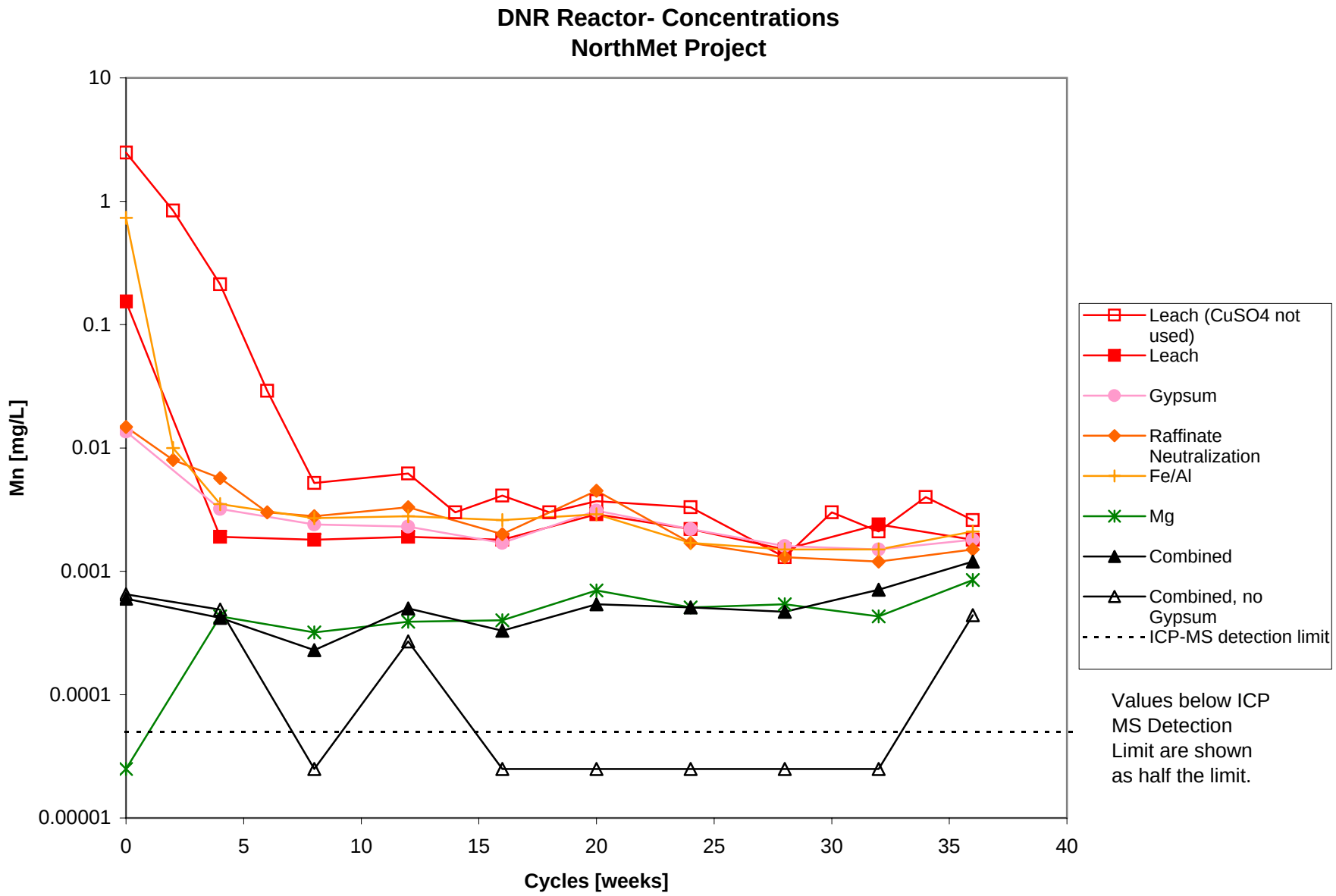

DNR Reactor - Concentrations
NorthMet Project
Cu [mg/L]
1000
100
10
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

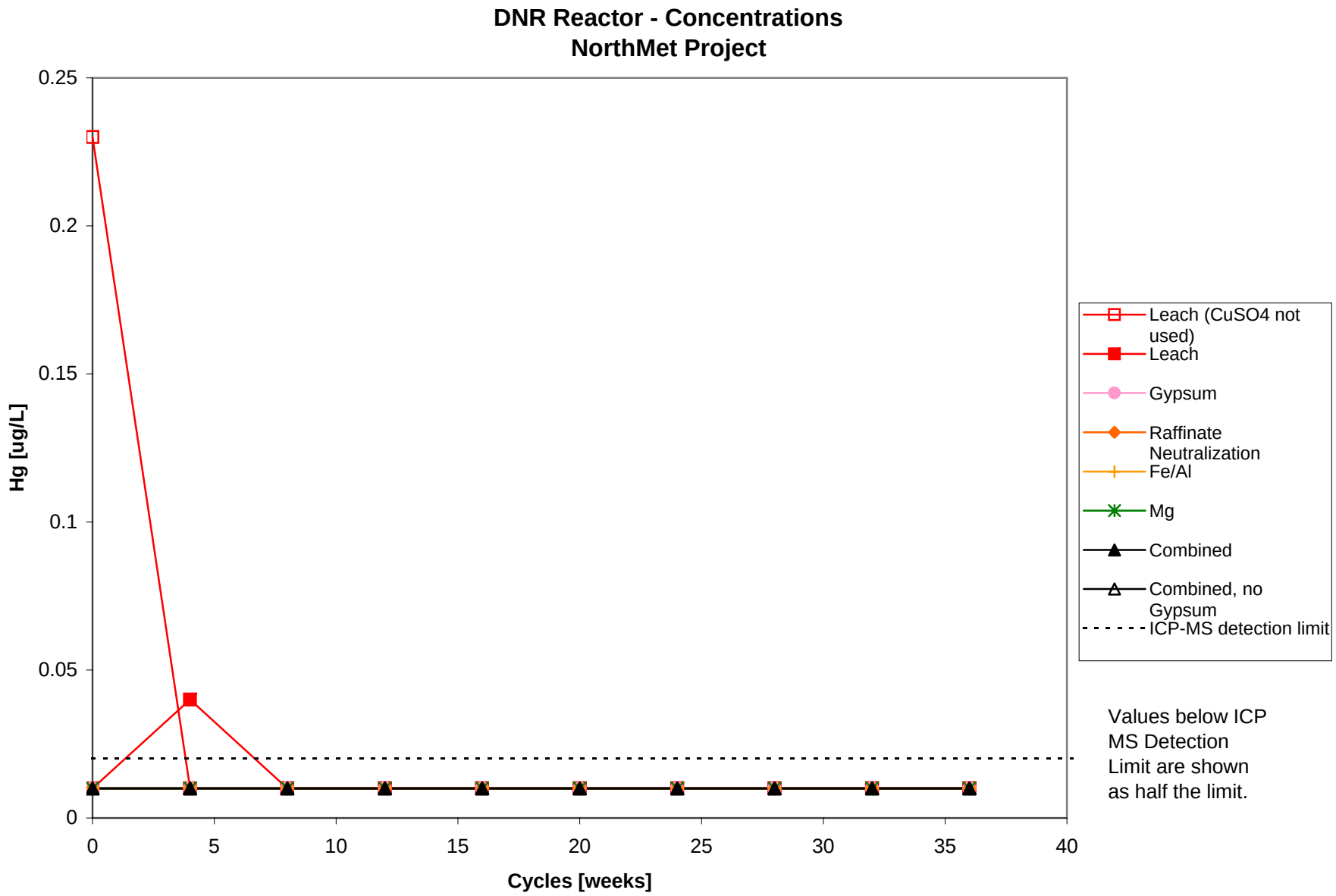

DNR Reactor - Concentrations
NorthMet Project
Fe [mg/L]
1000
100
10
1
0.1
0.01
0.001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

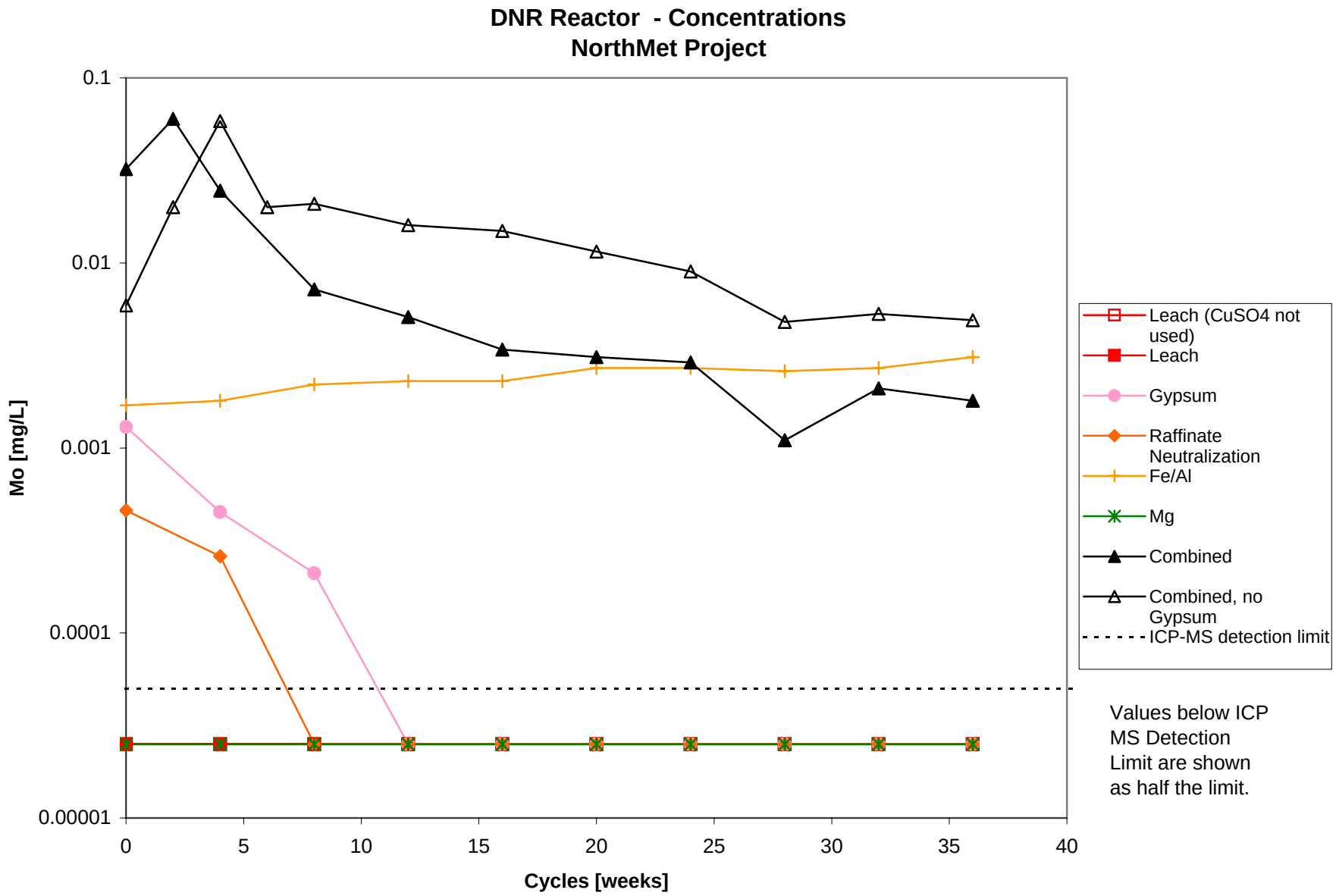

DNR Reactor - Concentrations
NorthMet Project
Pb [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

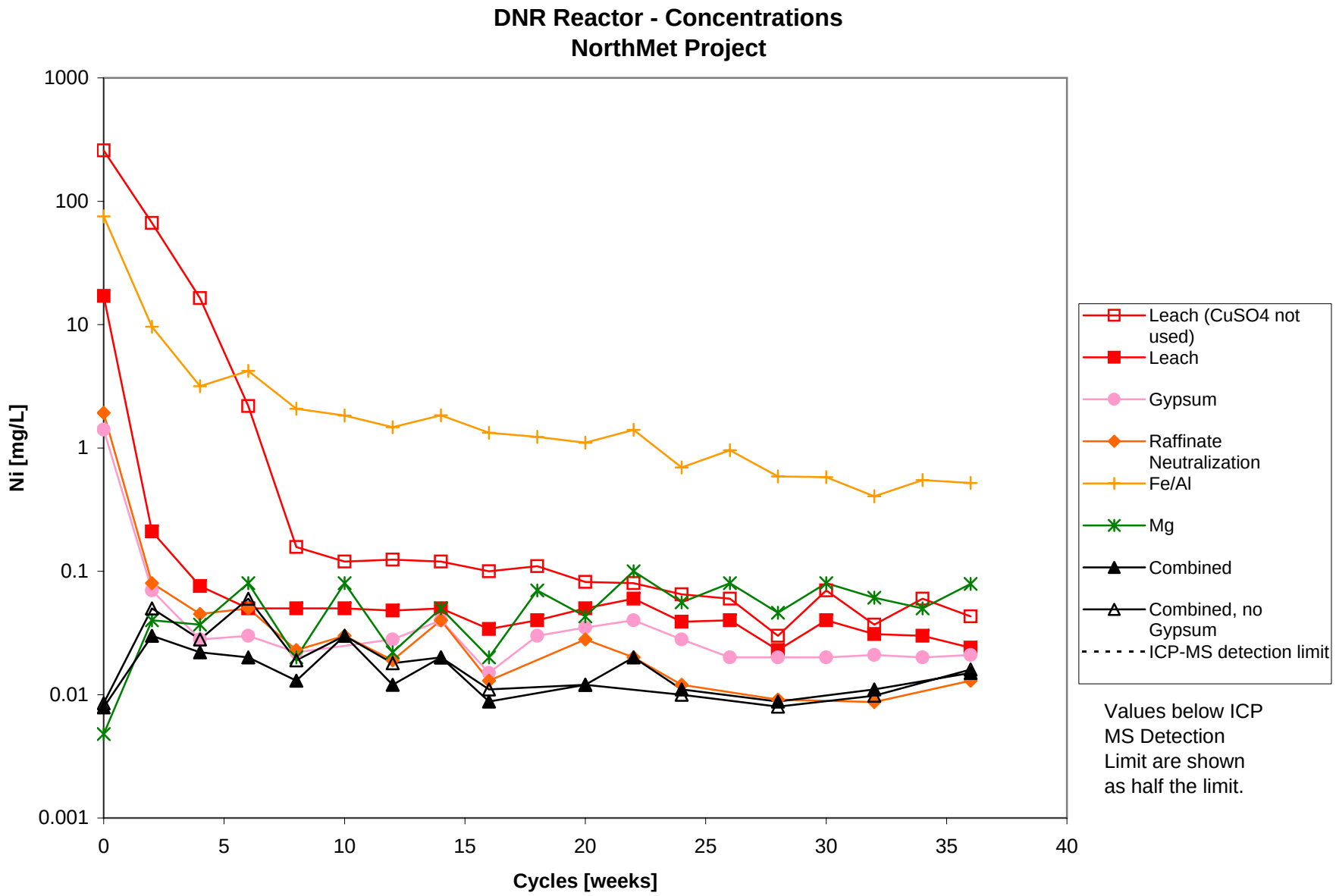

DNR Reactor - Concentrations
NorthMet Project
Li [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

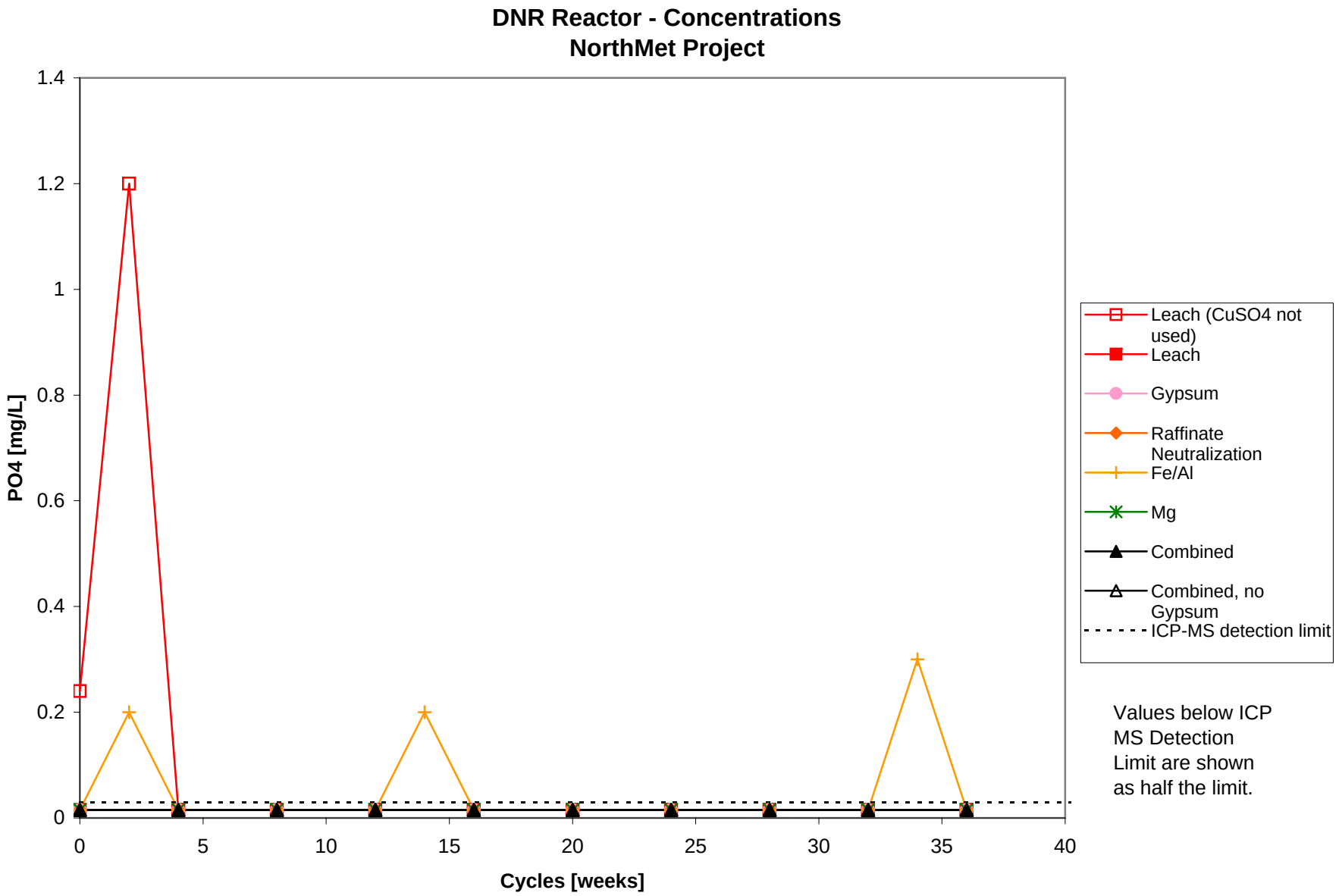

DNR Reactor - Concentrations
NorthMet Project
Mg [mg/L]
10000
1000
100
10
1
0.1
0.01
0.001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

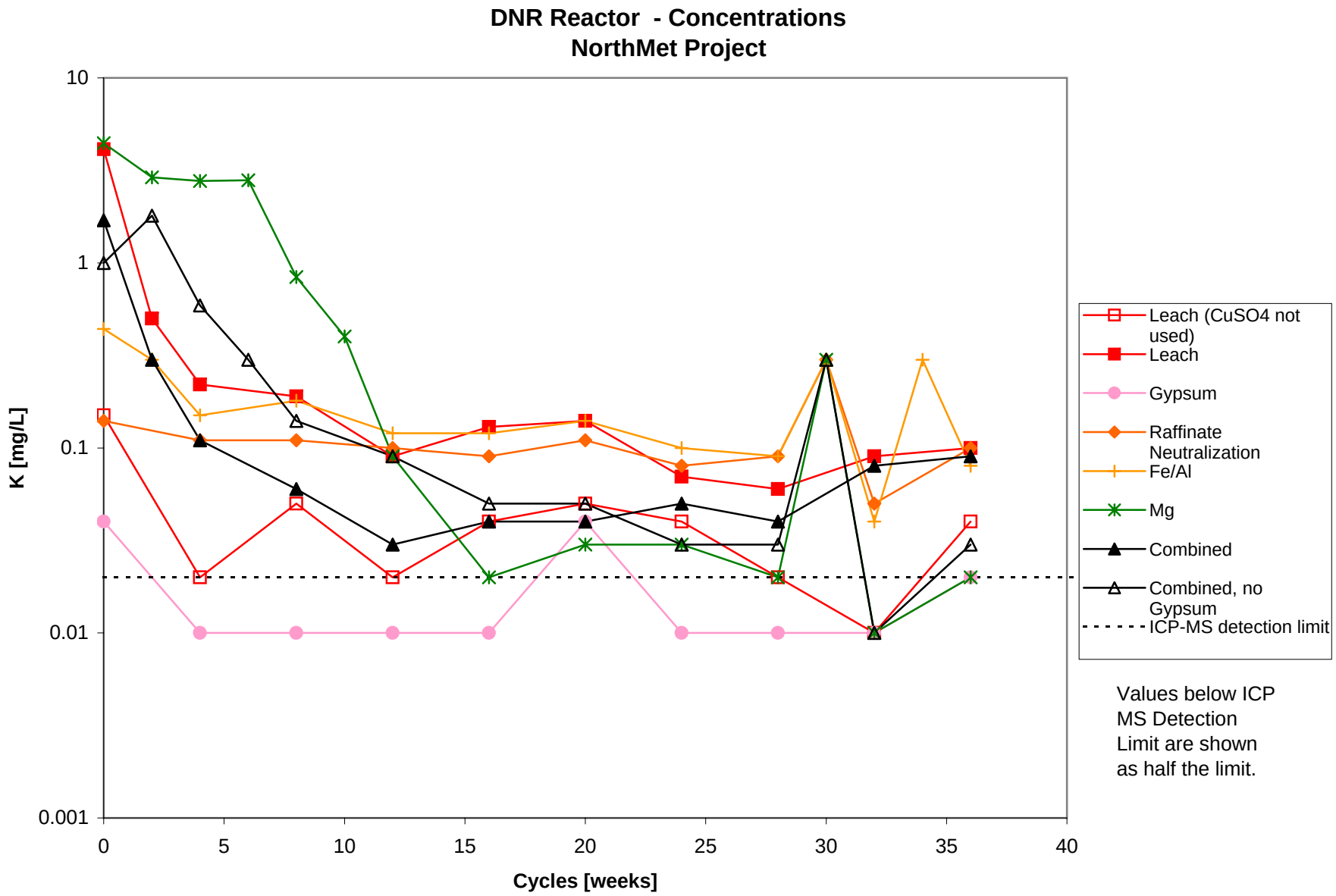

DNR Reactor- Concentrations
NorthMet Project
Mn [mg/L]
10
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

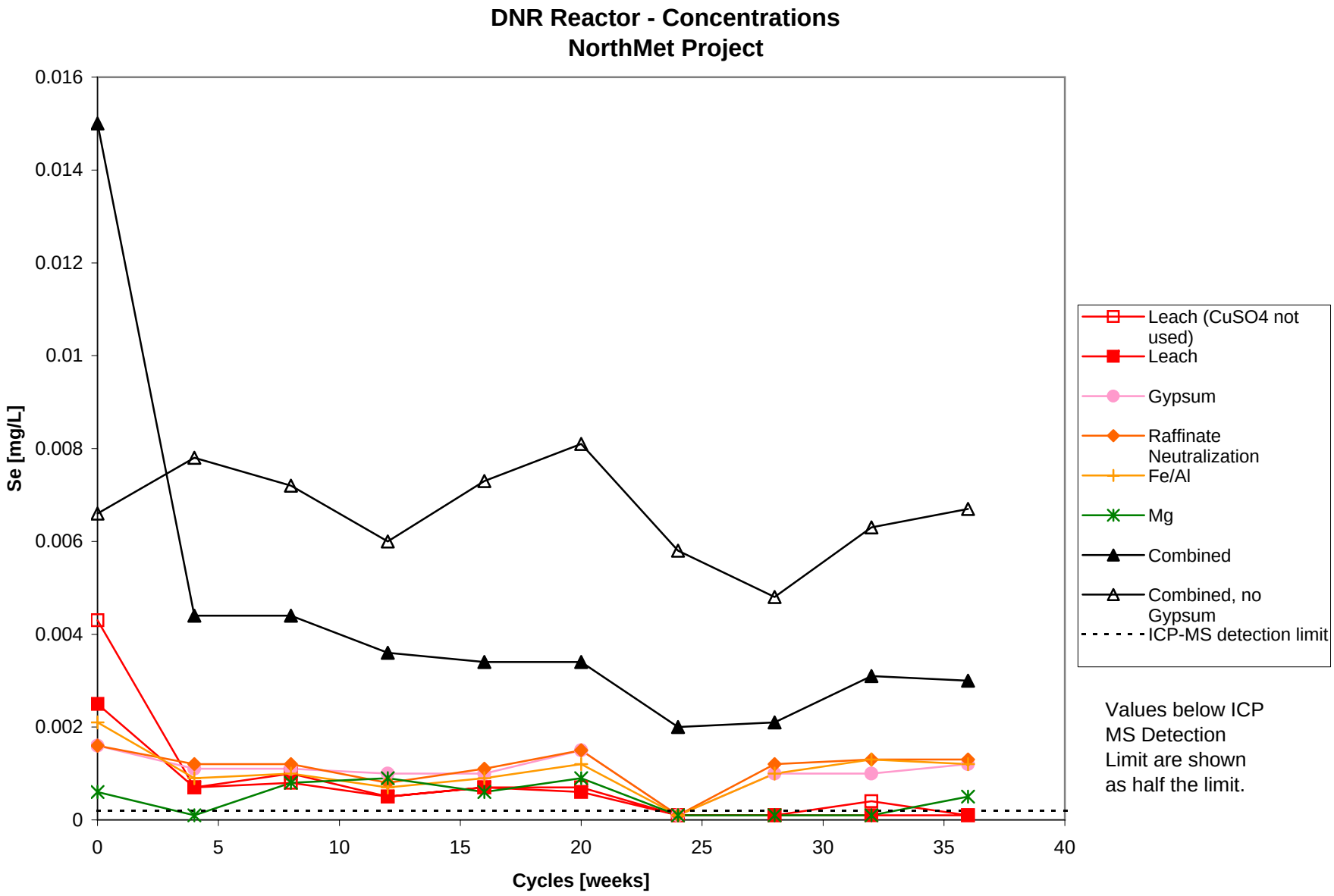

DNR Reactor - Concentrations
NorthMet Project
Hg [ug/L]
0.25
0.2
0.15
0.1
0.05
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.


DNR Reactor - Concentrations
NorthMet Project
Mo [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

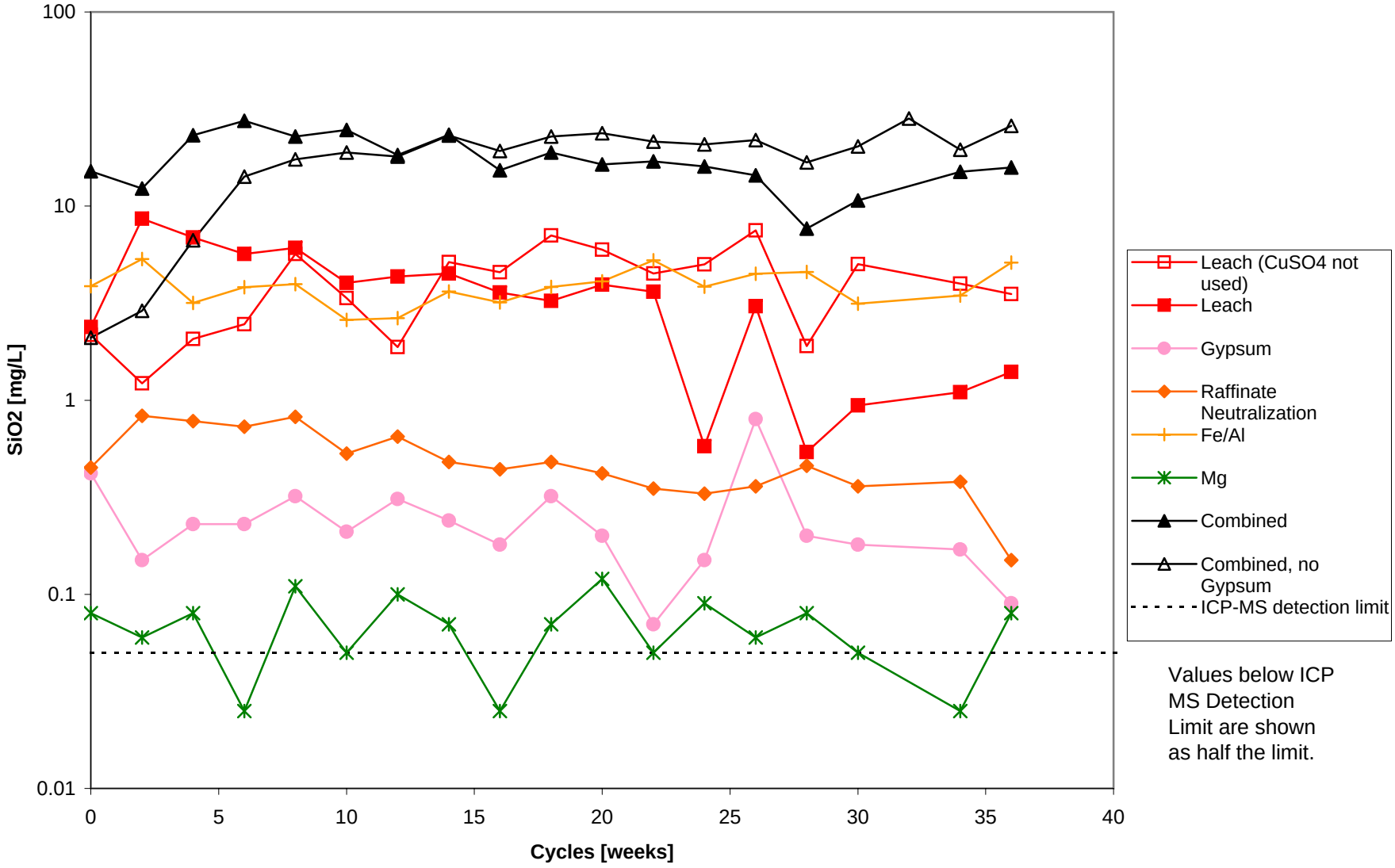

DNR Reactor - Concentrations
NorthMet Project
Ni [mg/L]
1000
100
10
1
0.1
0.01
0.001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.


DNR Reactor - Concentrations
NorthMet Project
PO4 [mg/L]
1.4
1.2
1
0.8
0.6
0.4
0.2
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

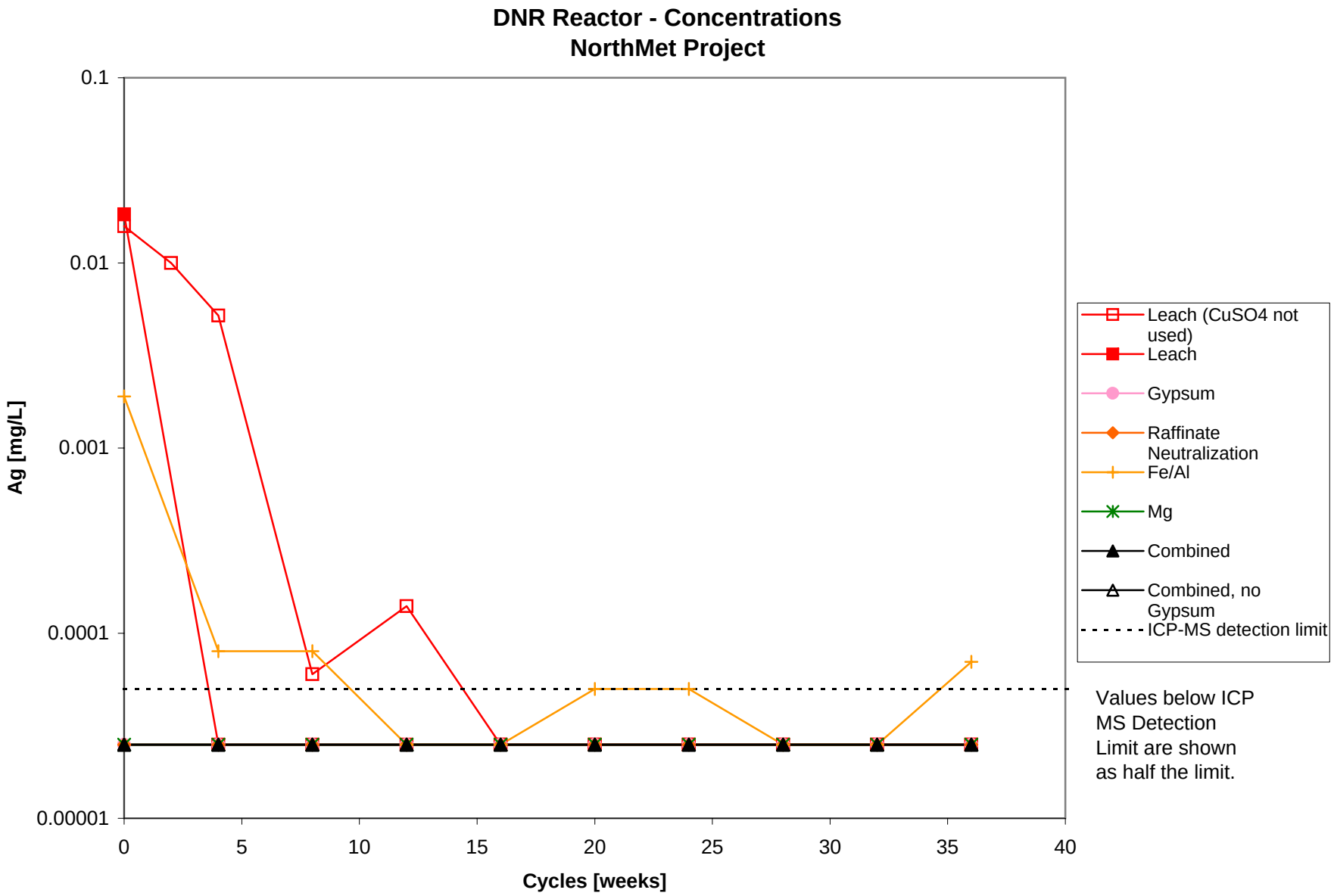

DNR Reactor - Concentrations
NorthMet Project
K [mg/L]
10
1
0.1
0.01
0.001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.


DNR Reactor - Concentrations
NorthMet Project
Se [mg/L]
0.016
0.014
0.012
0.01
0.008
0.006
0.004
0.002
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

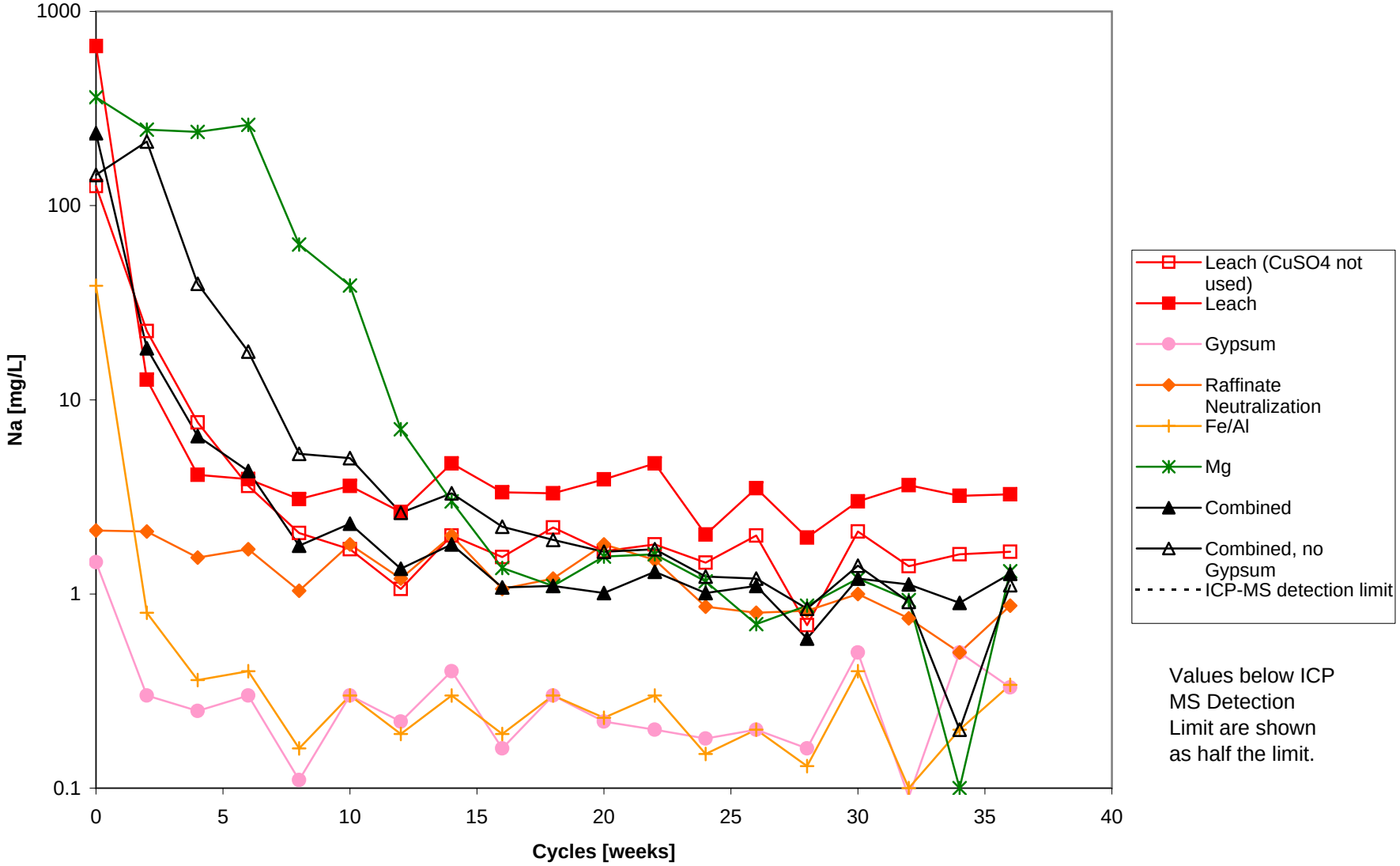
DNR Reactor - Concentrations
NorthMet Project



Values below ICP MS Detection Limit are shown as half the limit.


DNR Reactor - Concentrations
NorthMet Project
Ag [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

DNR Reactor - Concentrations
NorthMet Project



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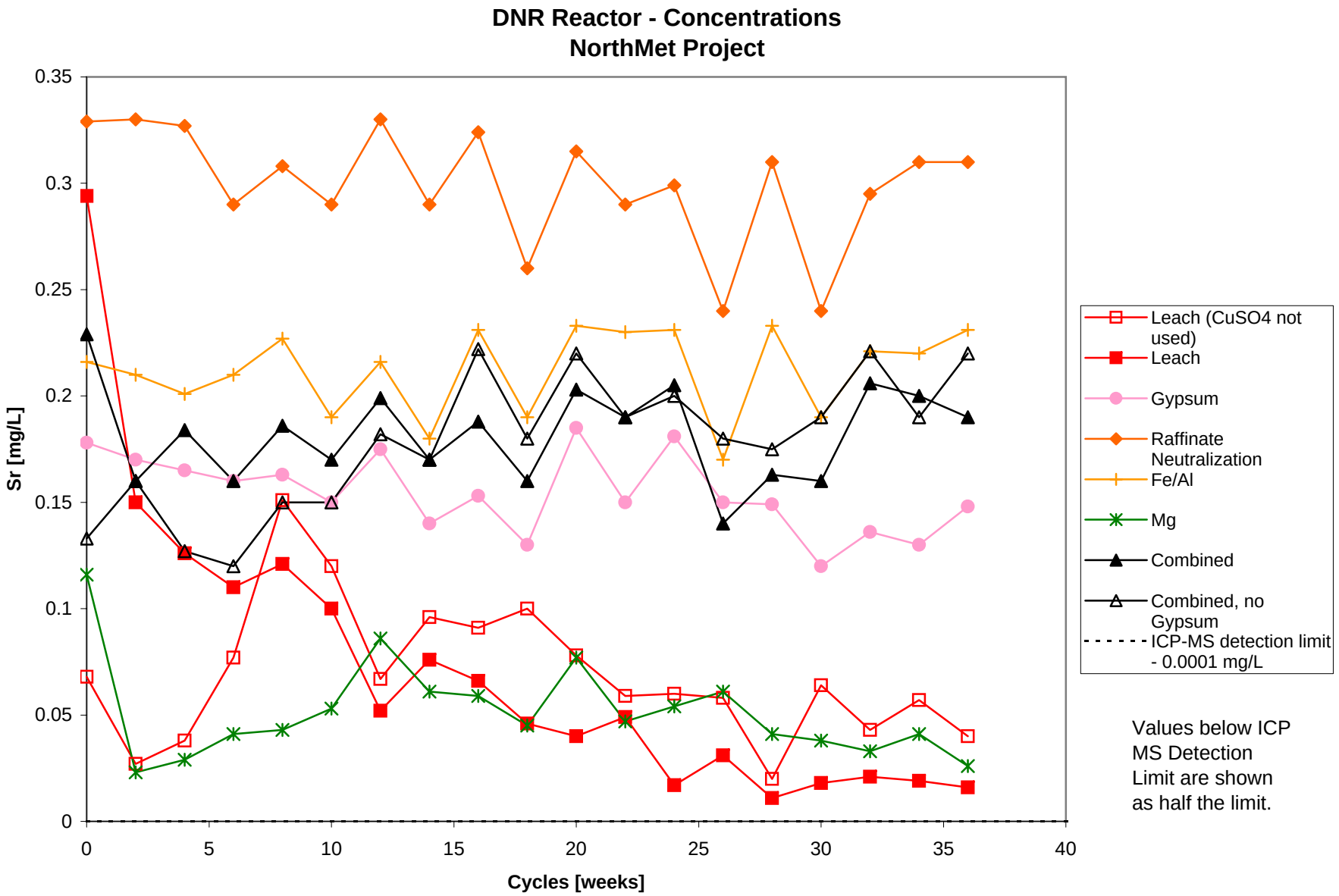
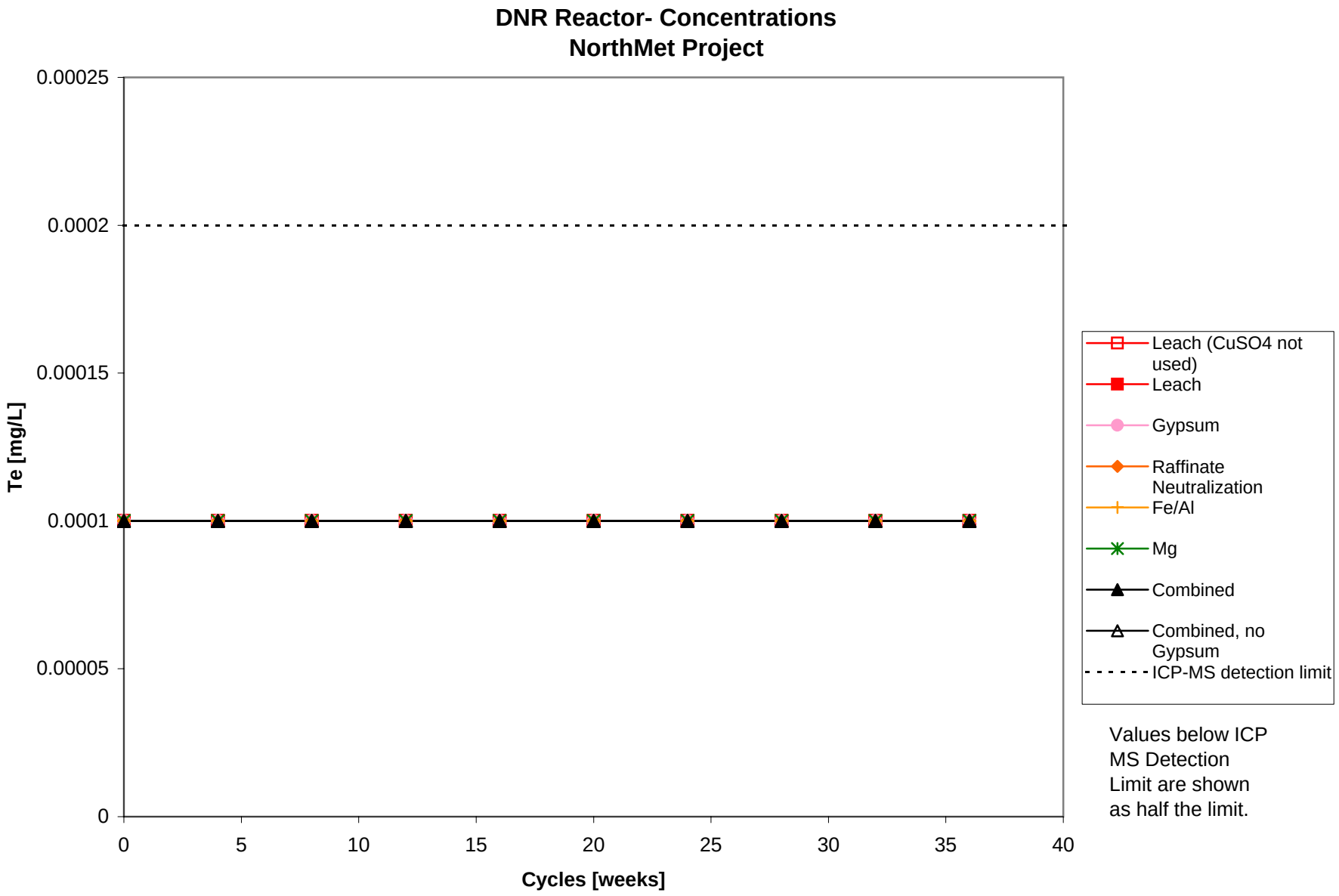
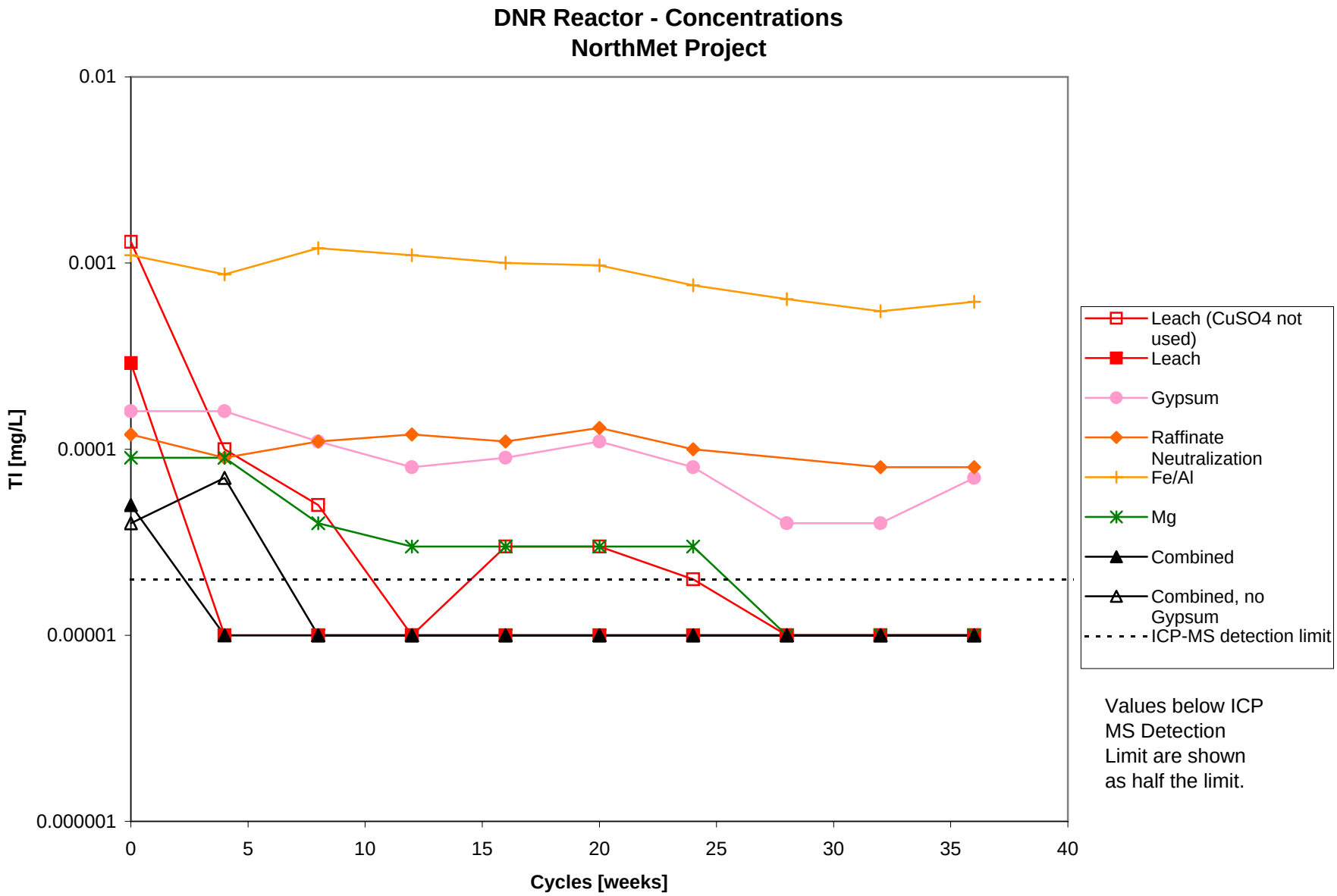
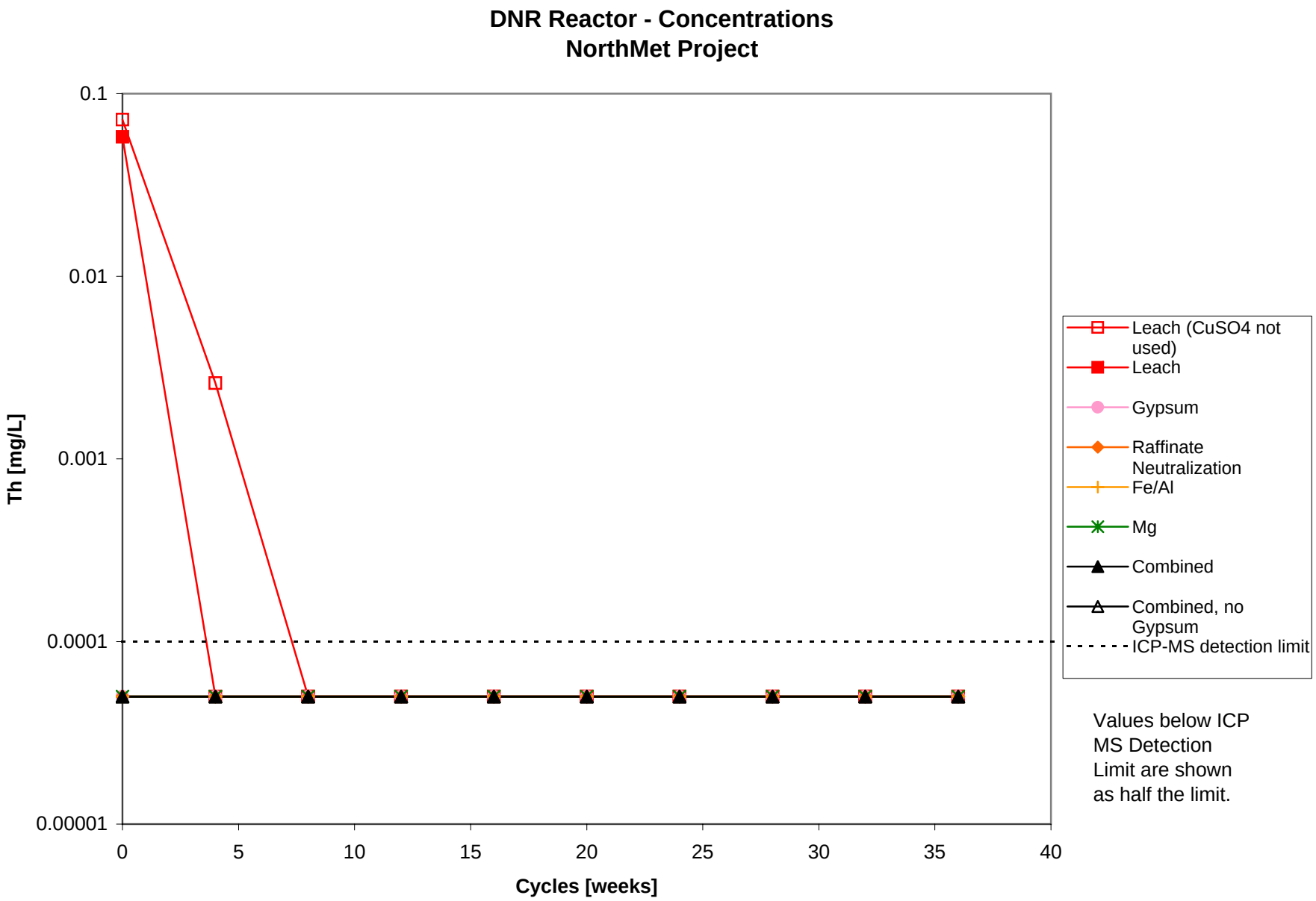
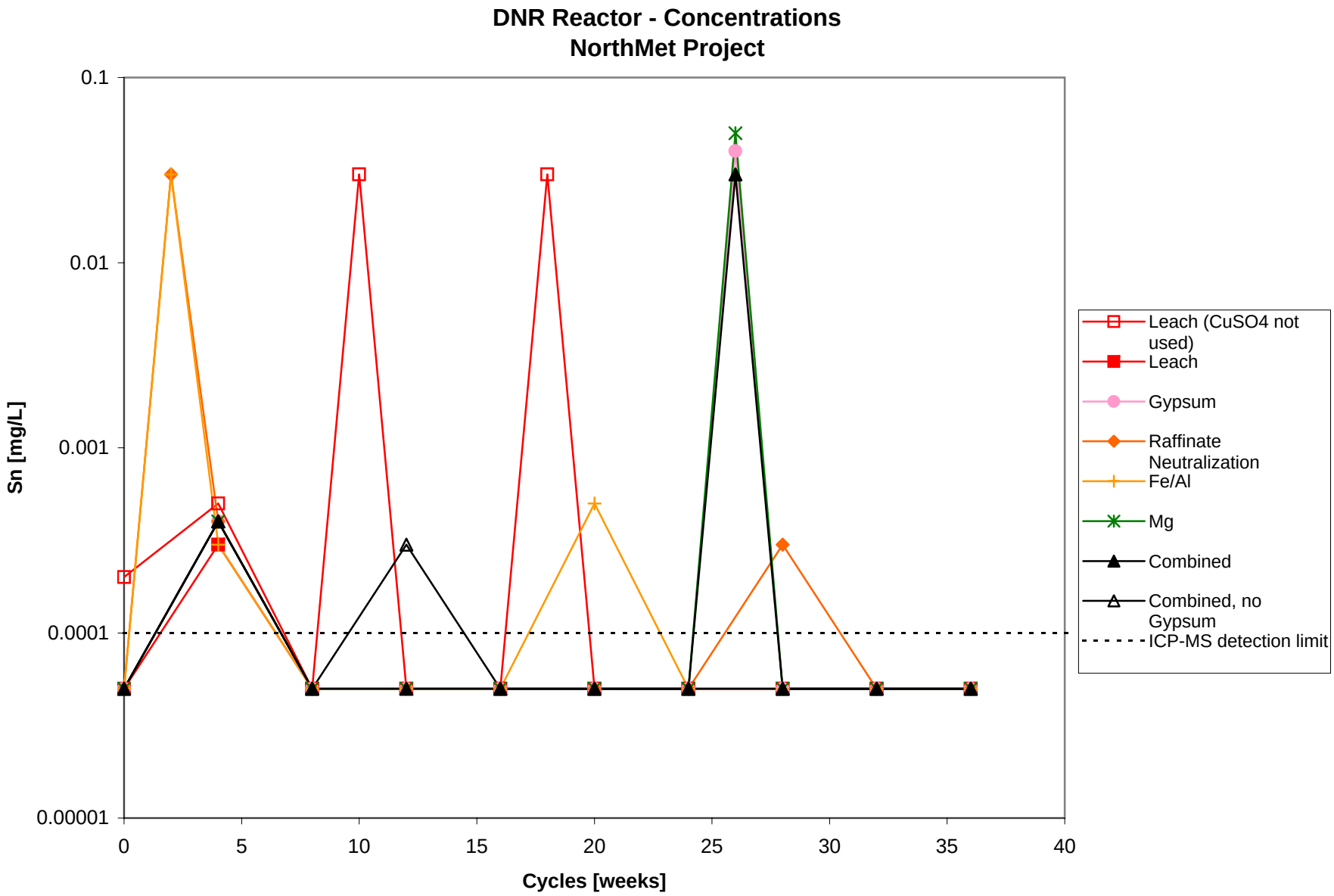


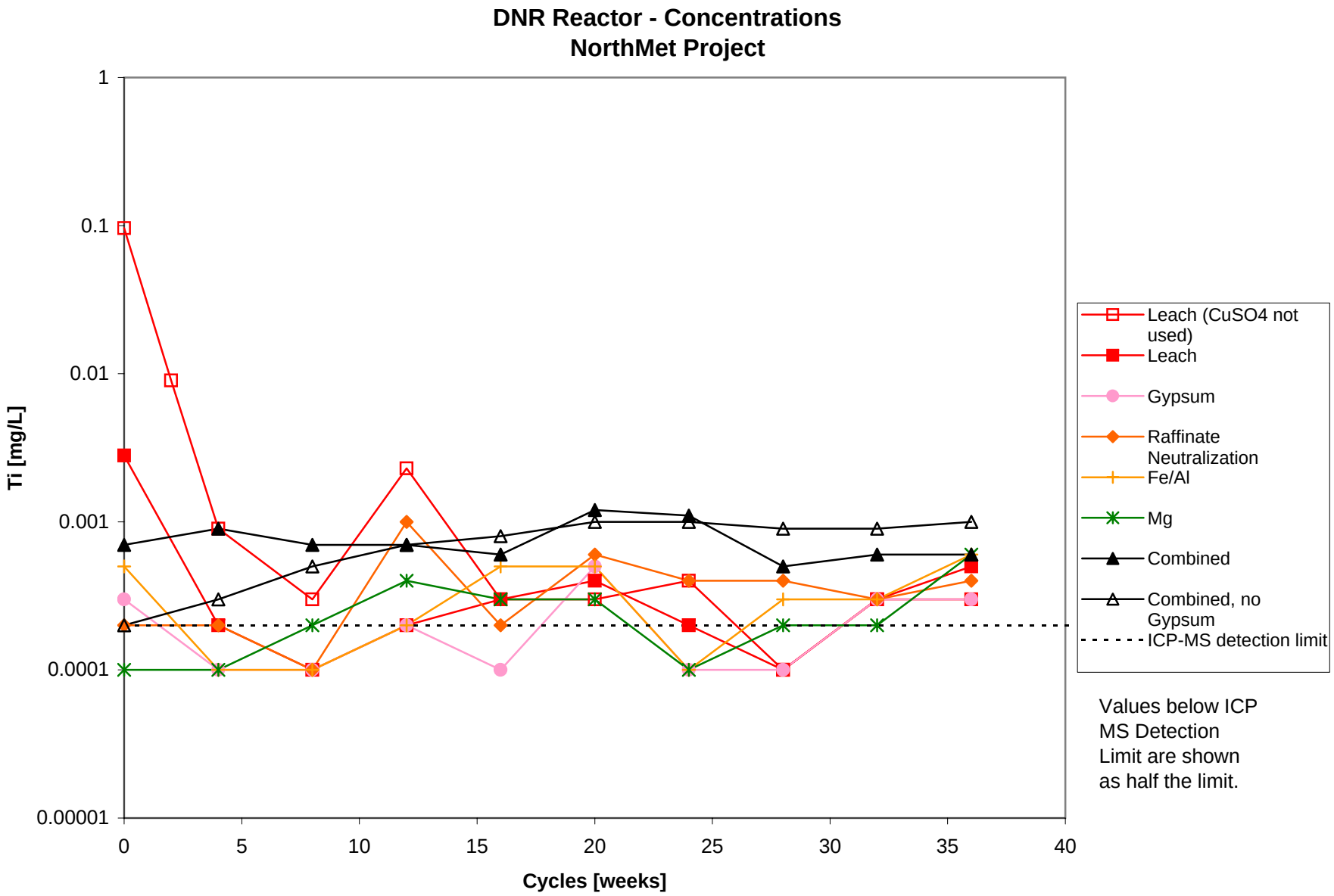
Values below ICP MS Detection Limit are shown as half the limit.

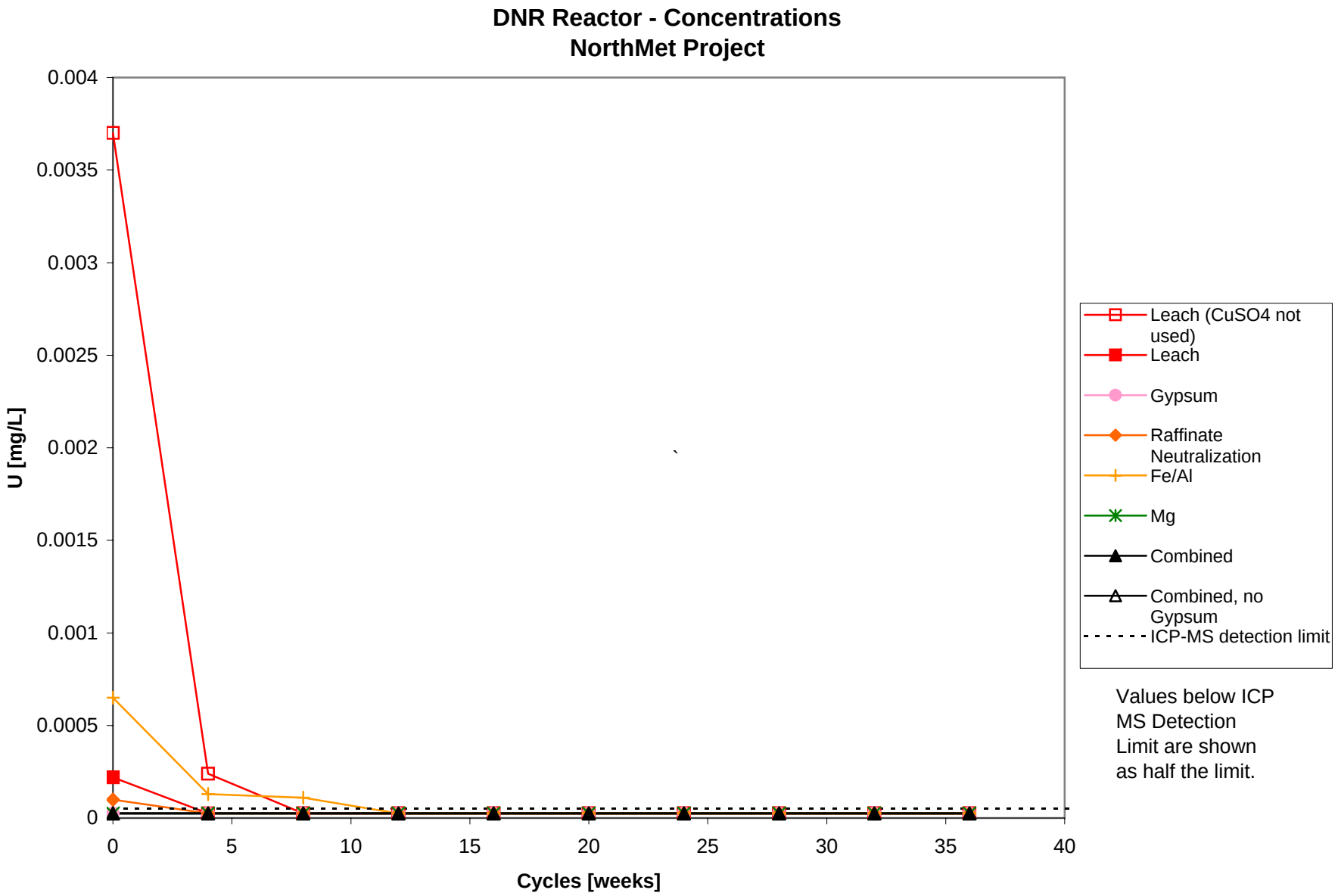

DNR Reactor- Concentrations
NorthMet Project
Te [mg/L]
0.00025
0.0002
0.00015
0.0001
0.00005
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

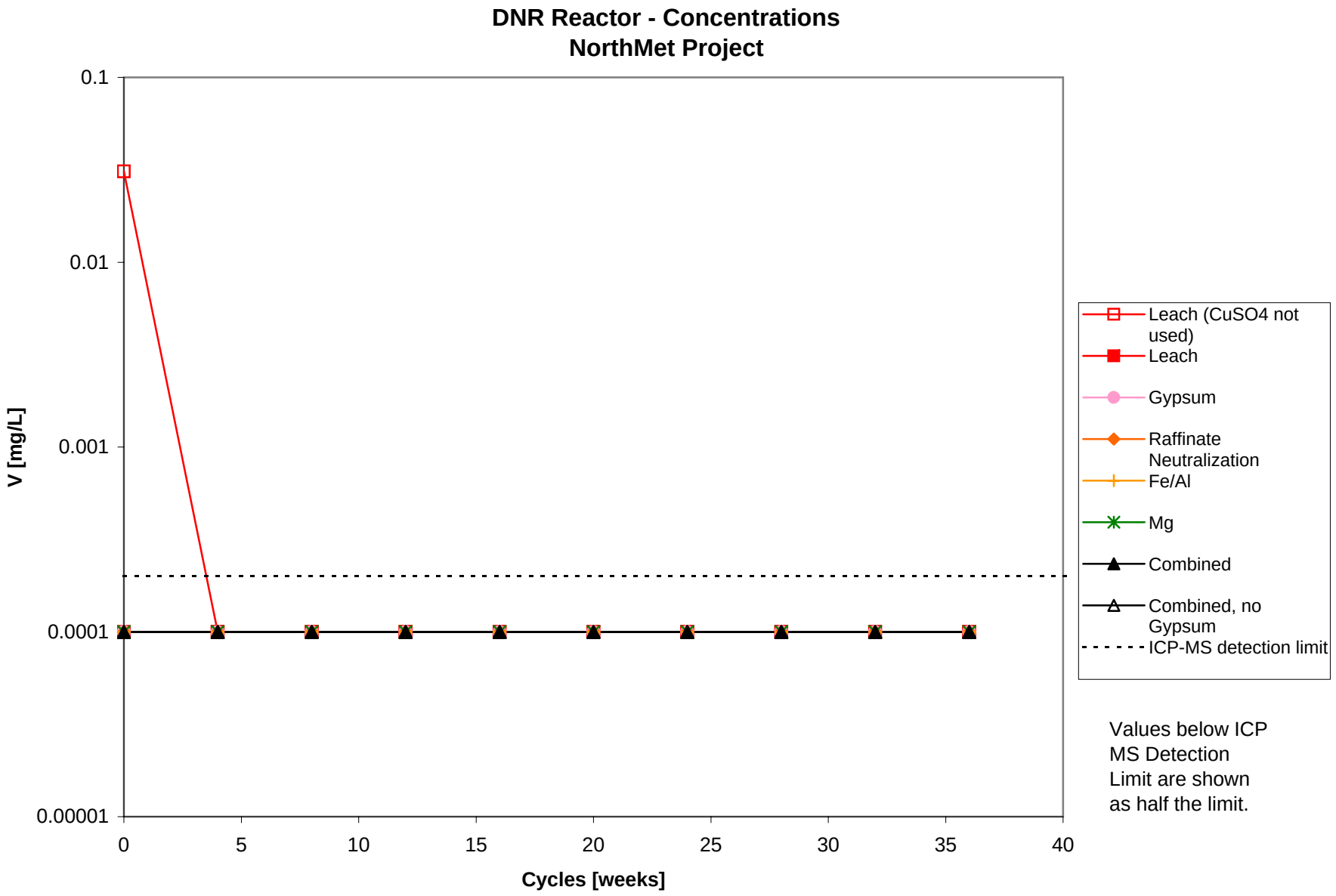

DNR Reactor - Concentrations
NorthMet Project
Tl [mg/L]
0.01
0.001
0.0001
0.00001
0.000001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.

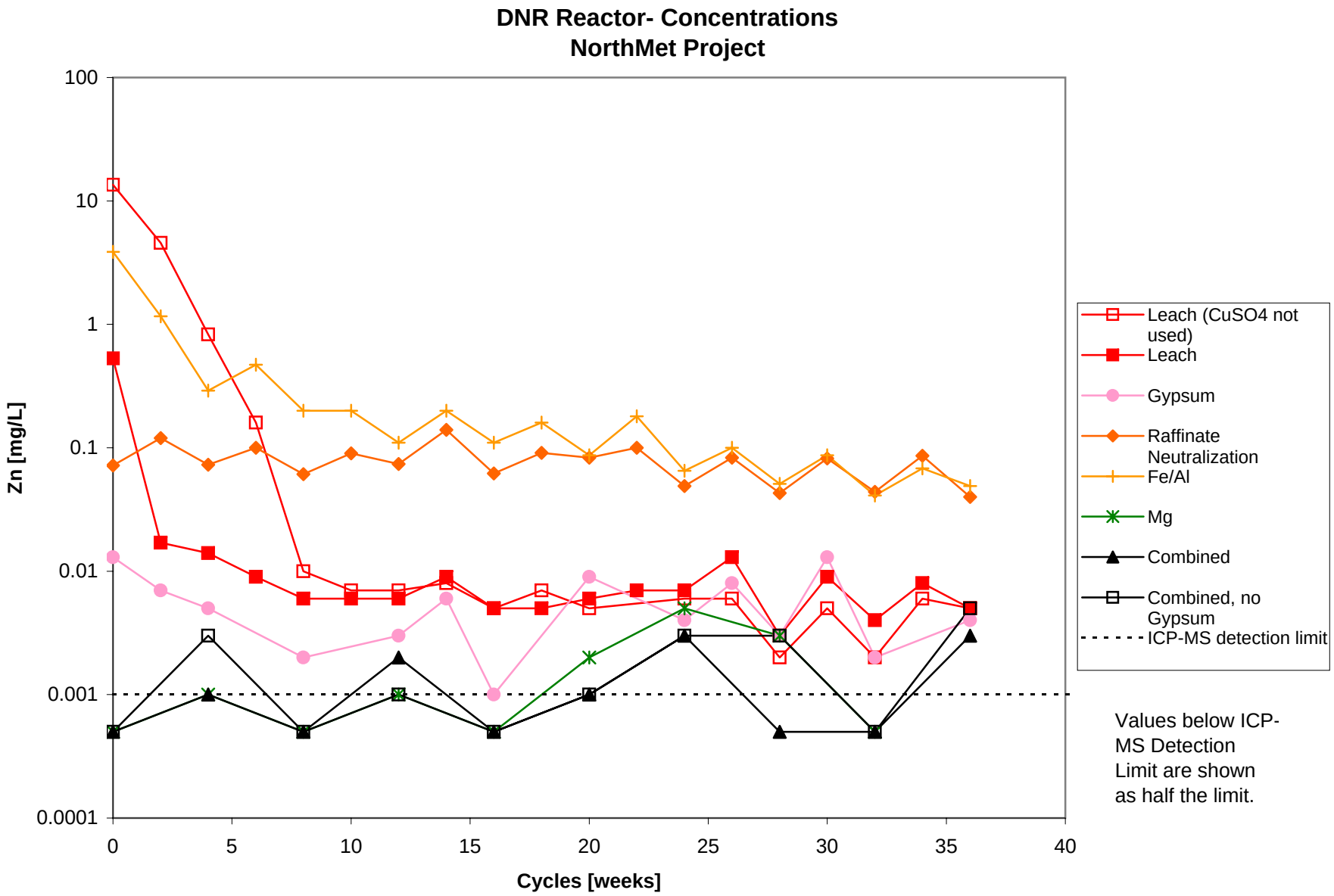

DNR Reactor - Concentrations
NorthMet Project
Th [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.


DNR Reactor - Concentrations
NorthMet Project
Sn [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit

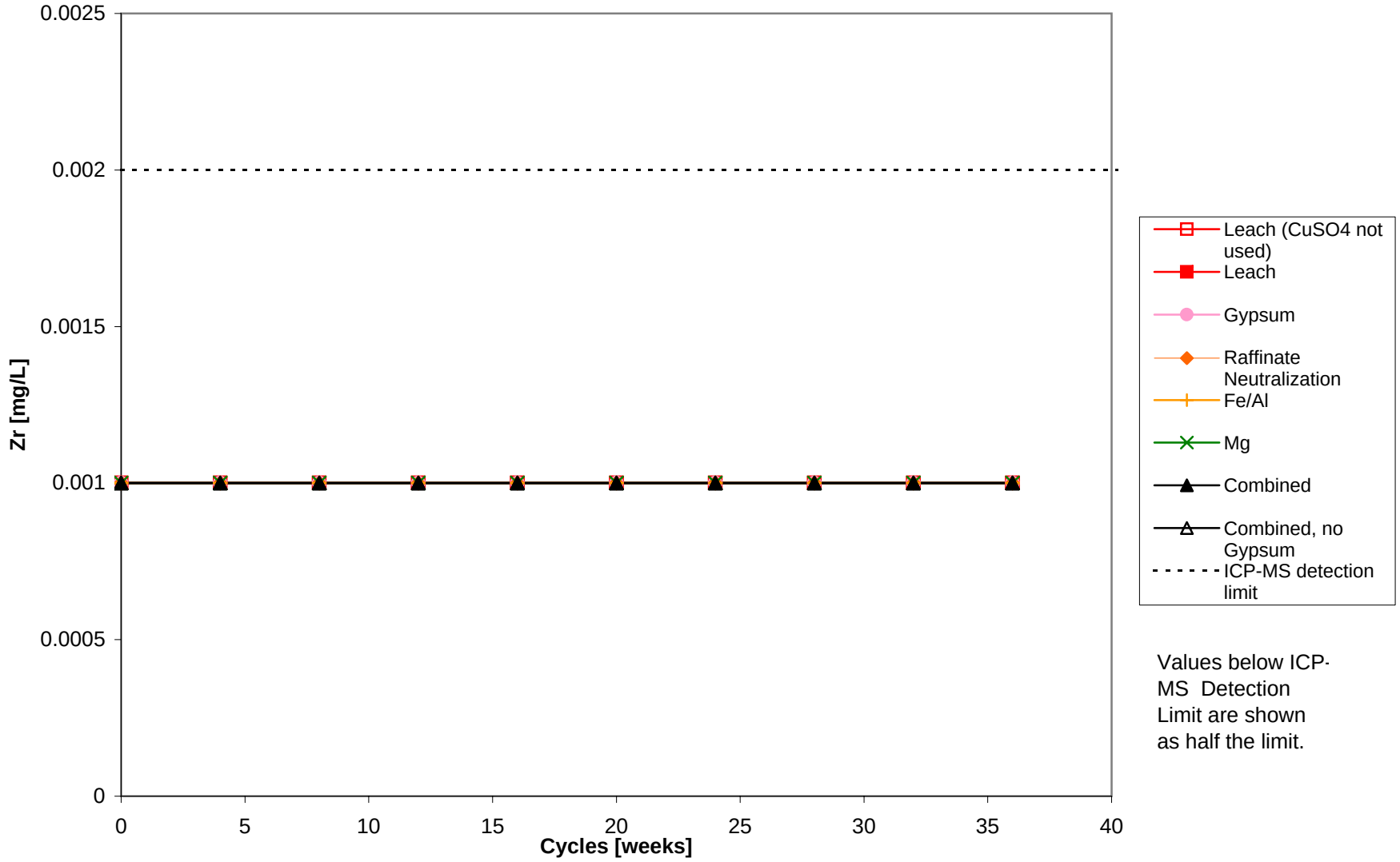

DNR Reactor - Concentrations
NorthMet Project
Ti [mg/L]
1
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.


DNR Reactor - Concentrations
NorthMet Project
U [mg/L]
0.004
0.0035
0.003
0.0025
0.002
0.0015
0.001
0.0005
0
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.


DNR Reactor - Concentrations
NorthMet Project
V [mg/L]
0.1
0.01
0.001
0.0001
0.00001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP MS Detection Limit are shown as half the limit.


DNR Reactor- Concentrations
NorthMet Project
Zn [mg/L]
100
10
1
0.1
0.01
0.001
0.0001
0
5
10
15
20
25
30
35
40
Cycles [weeks]
Leach (CuSO4 not used)
Leach
Gypsum
Raffinate Neutralization
Fe/Al
Mg
Combined
Combined, no Gypsum
ICP-MS detection limit
Values below ICP-MS Detection Limit are shown as half the limit.

DNR Reactor - Concentrations
NorthMet Project



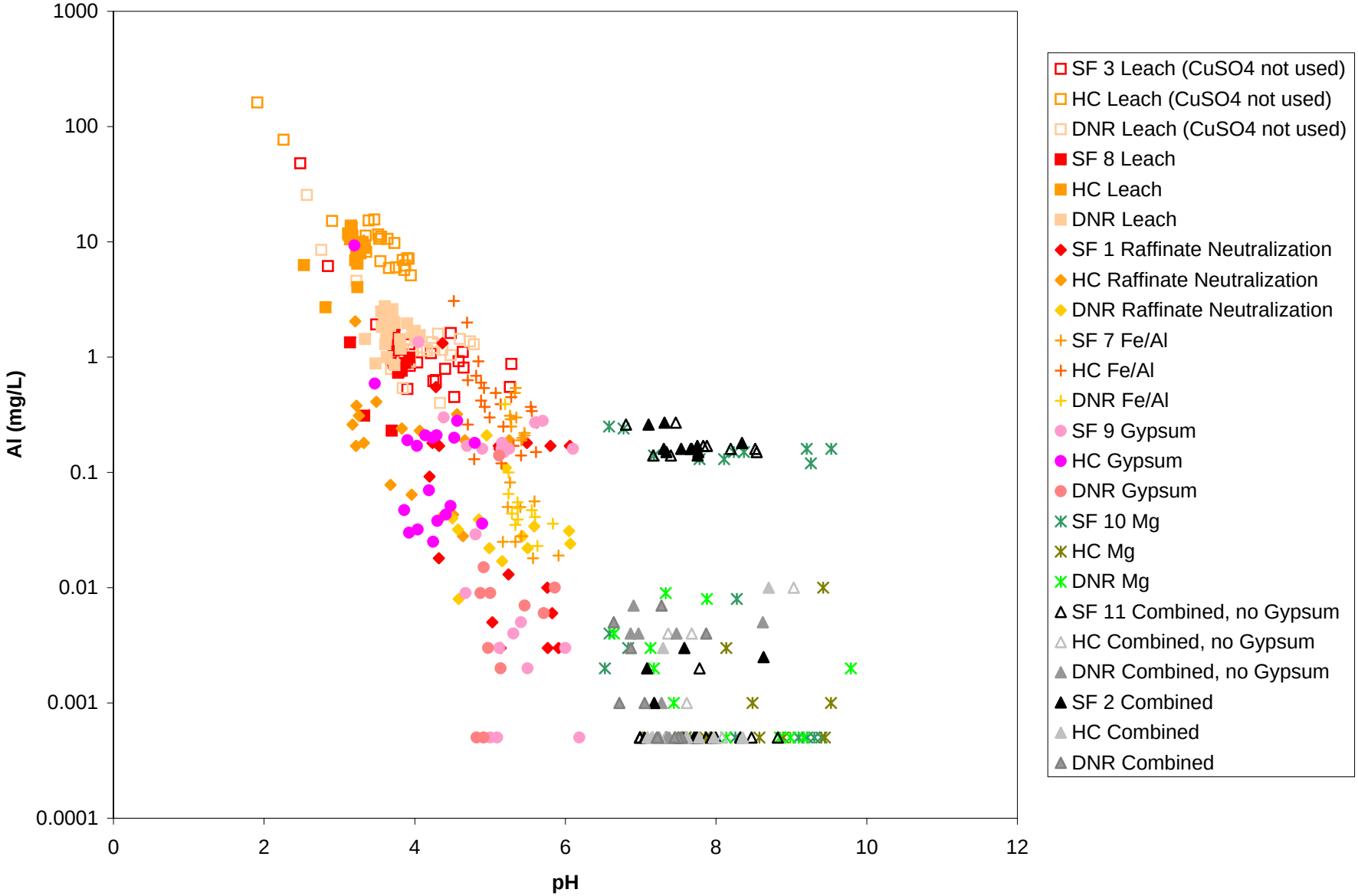
Values below ICP-MS Detection Limit are shown as half the limit.

Appendix D

Scatter Plots Showing Metal Concentrations Compared to pH

Appendix D
Scatter Plots Showing Metal Concentrations Compared to pH

Graph D.1



M(pH)

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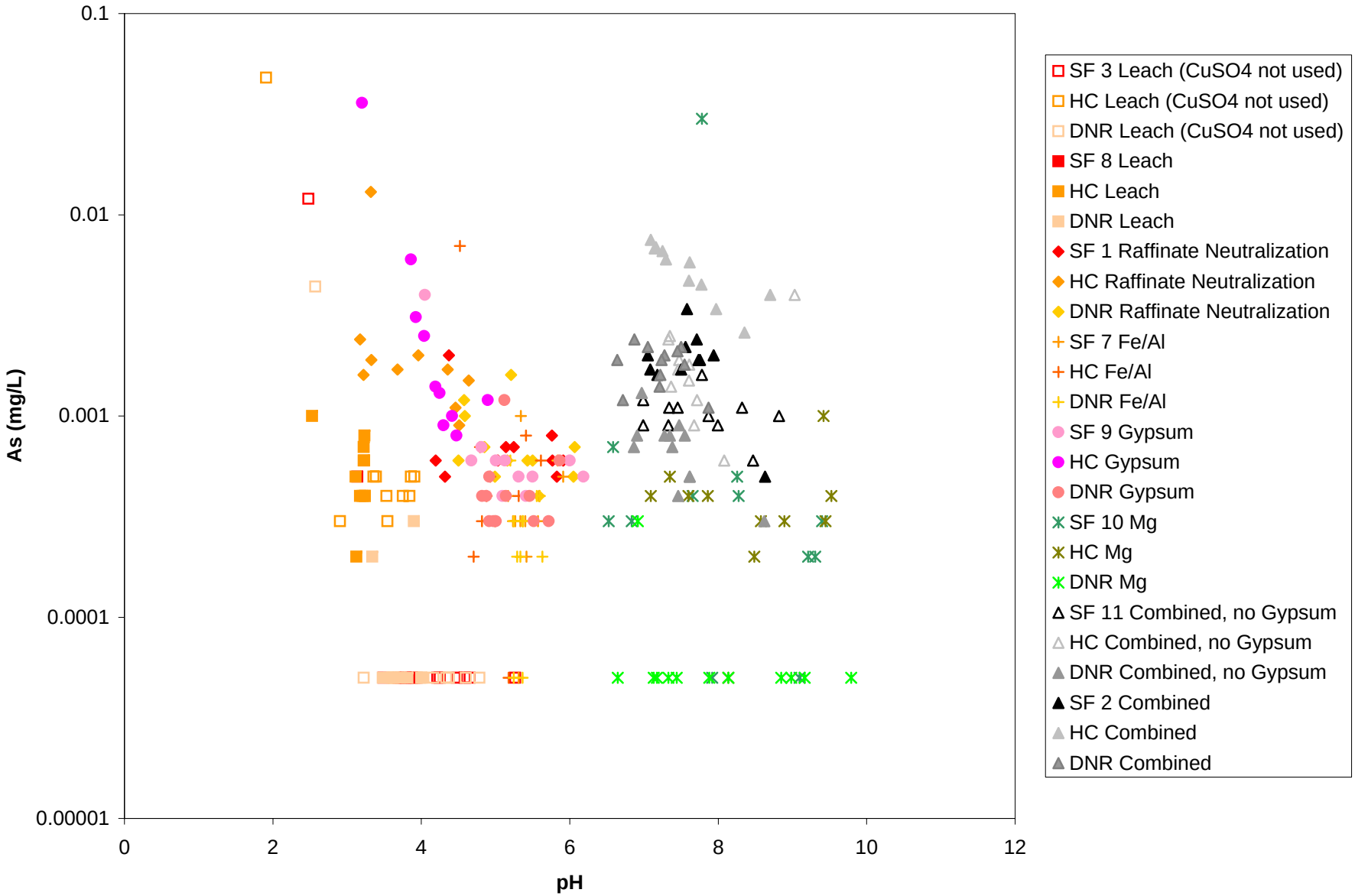
Appendix D
Scatter Plots Showing Metal Concentrations Compared to pH

Graph D.2



M(pH)

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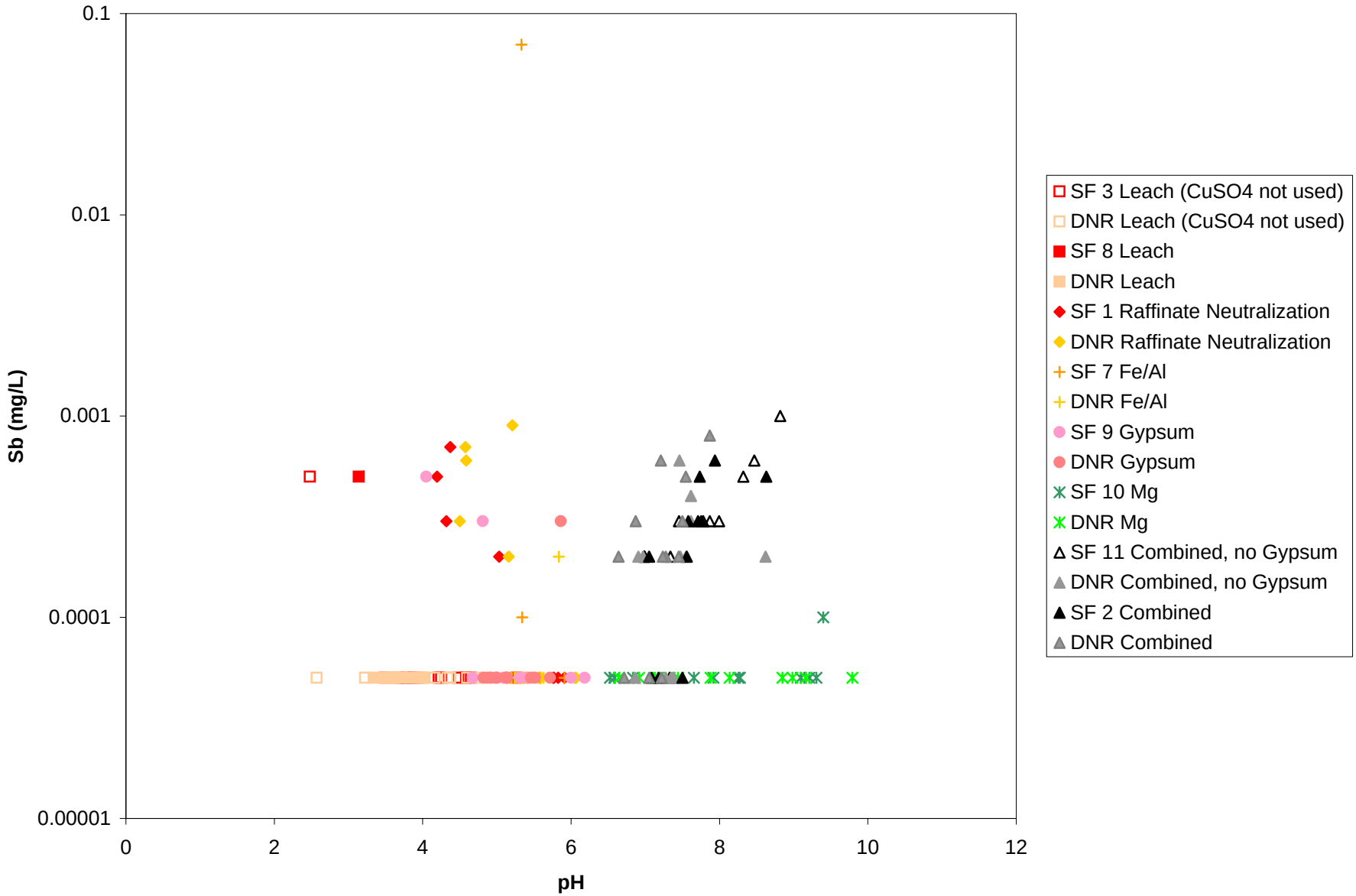
Appendix D
Scatter Plots Showing Metal Concentrations Compared to pH

Graph D.3



M(pH)

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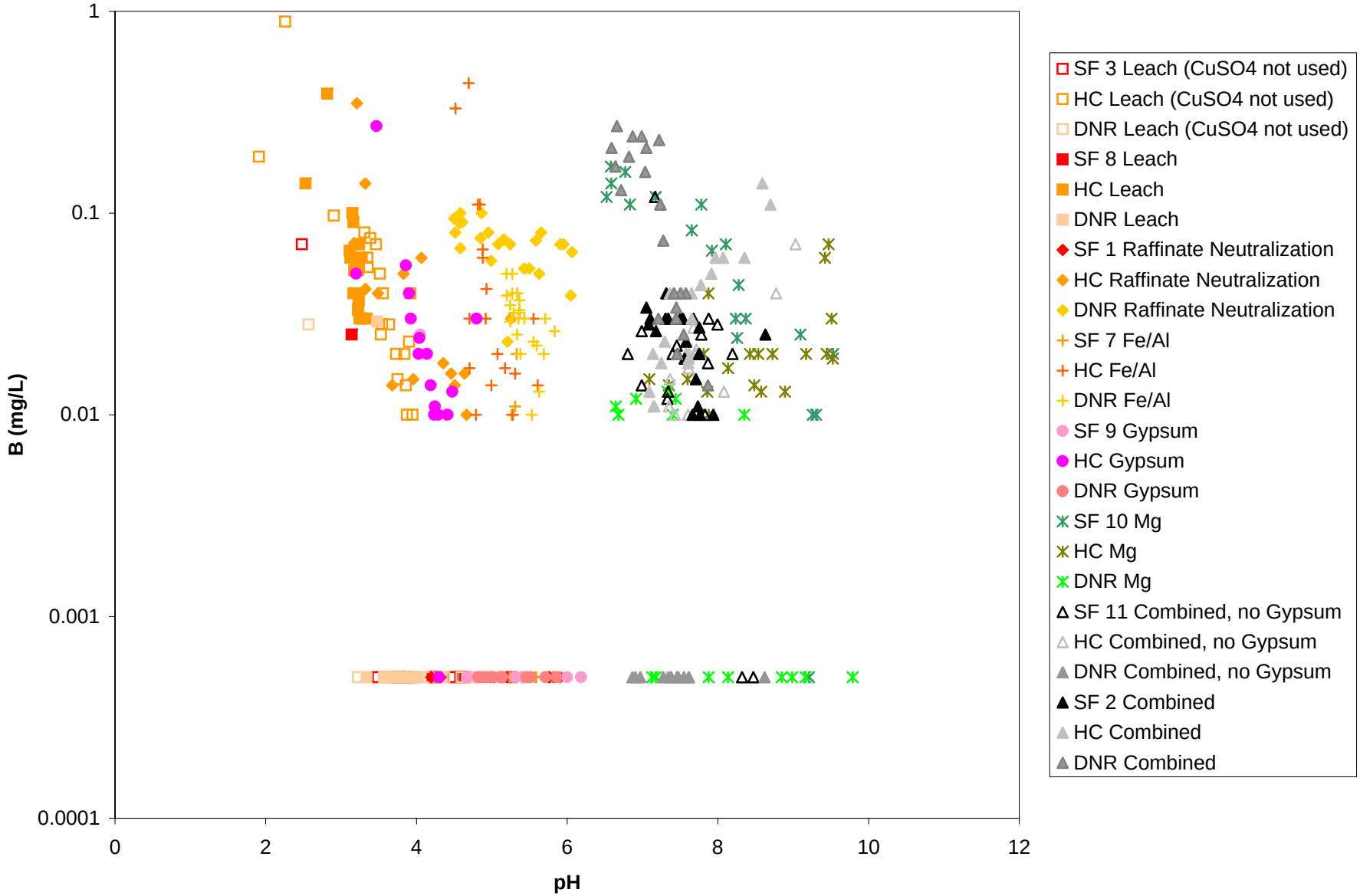
Appendix D
Scatter Plots Showing Metal Concentrations Compared to pH

Graph D.4



M(pH)

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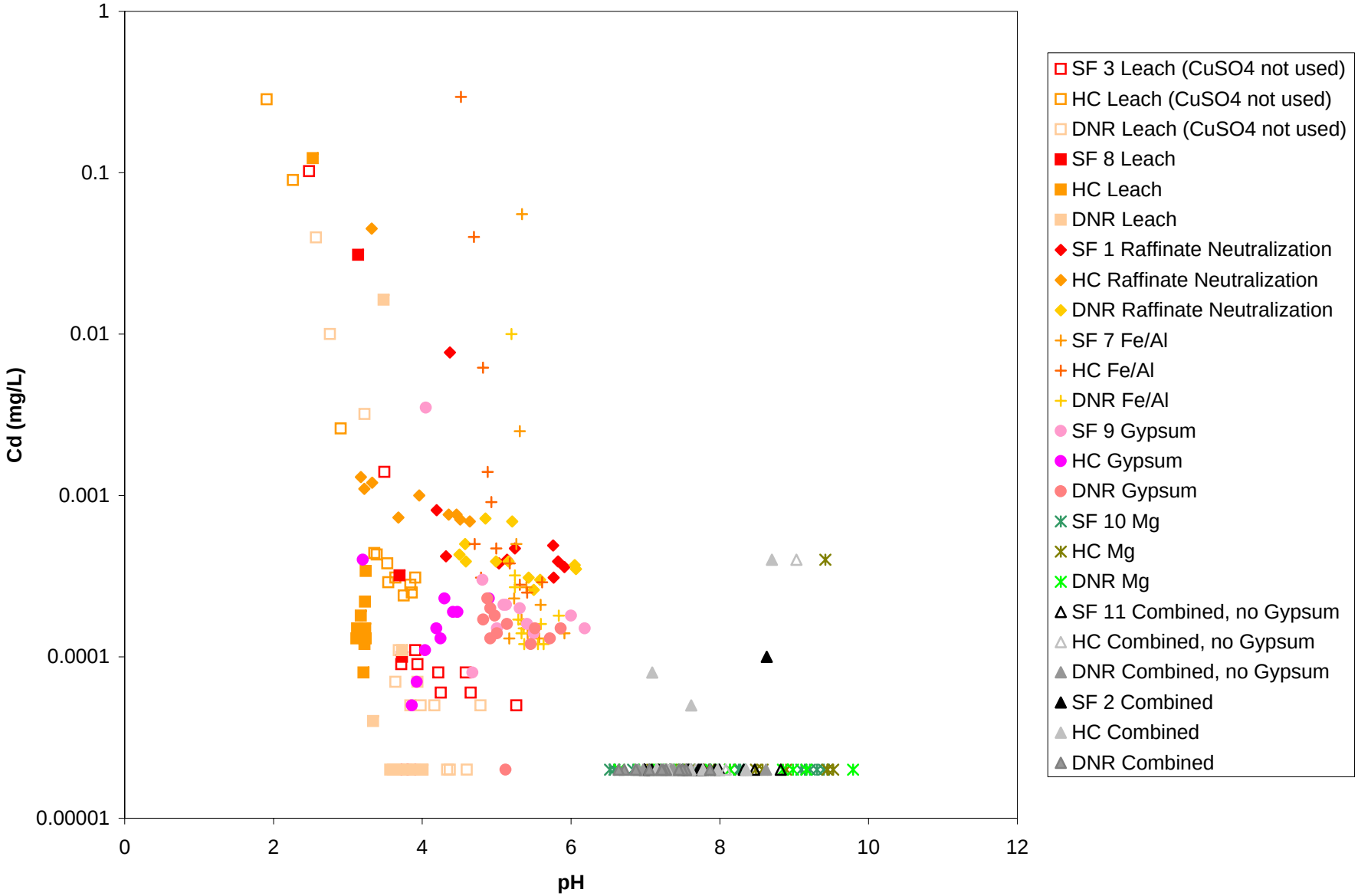
Appendix D
Scatter Plots Showing Metal Concentrations Compared to pH

Graph D.5



M(pH)

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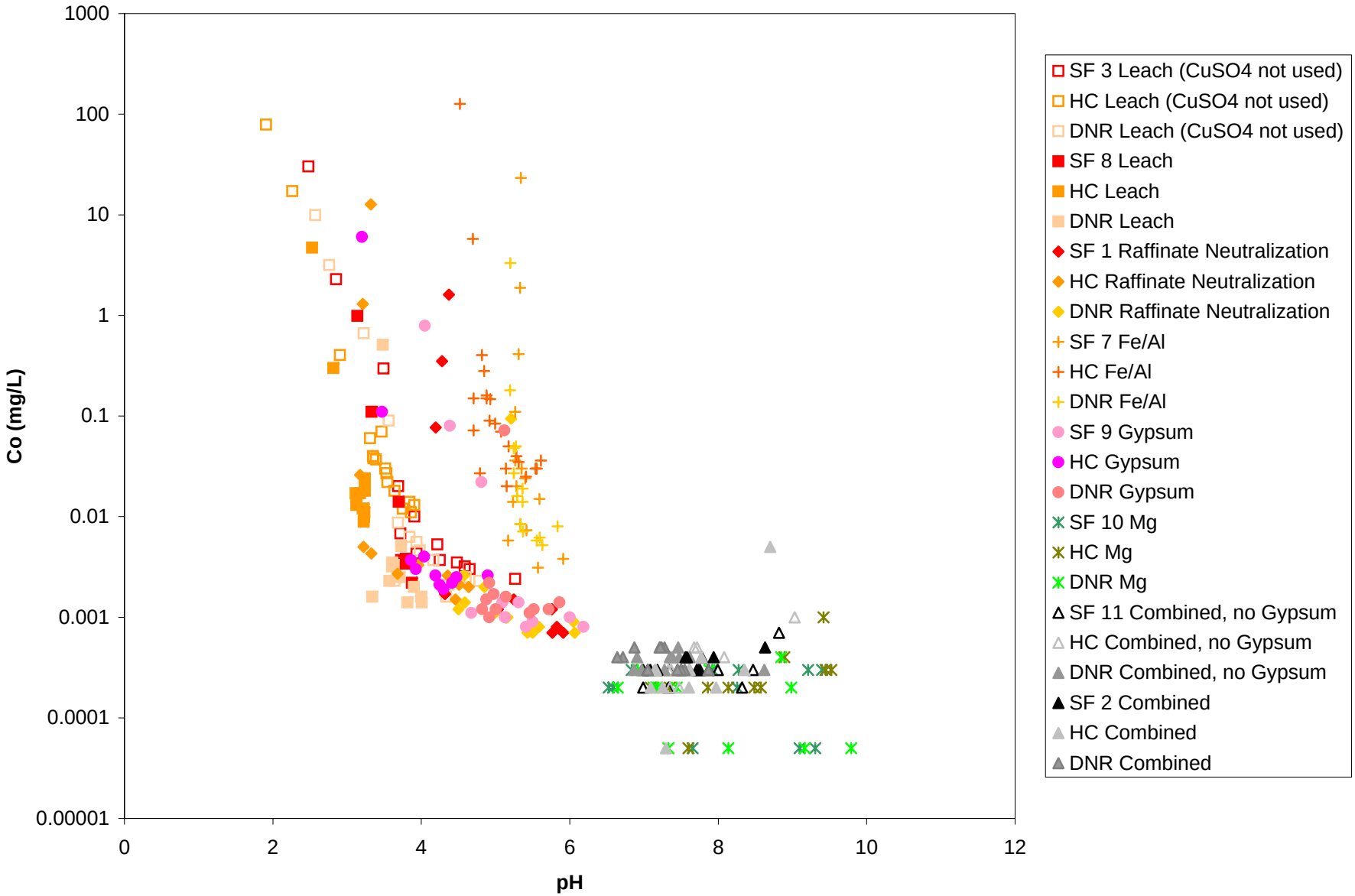
Appendix D
Scatter Plots Showing Metal Concentrations Compared to pH

Graph D.6



M(pH)

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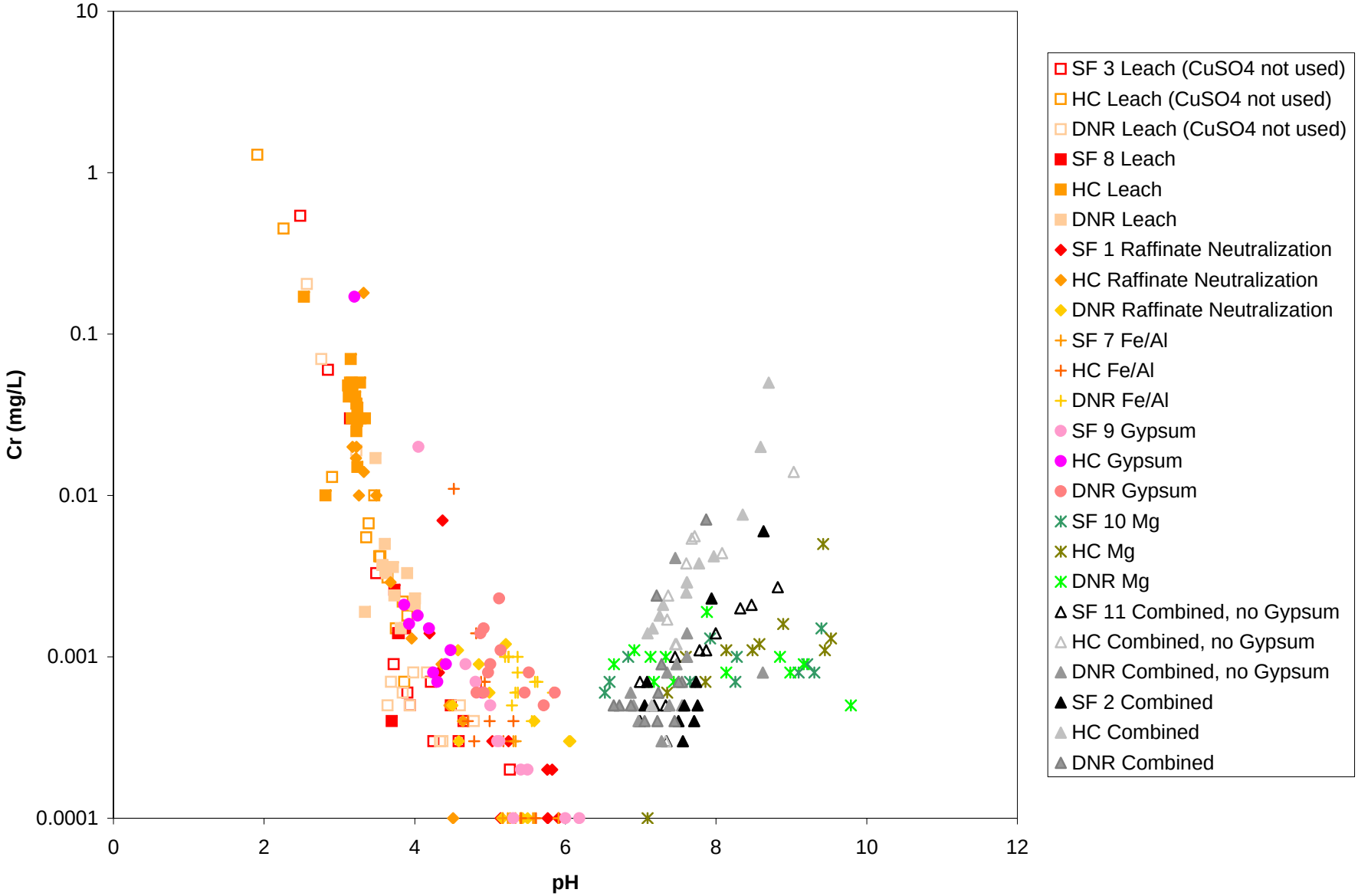
Appendix D
Scatter Plots Showing Metal Concentrations Compared to pH

Graph D.7

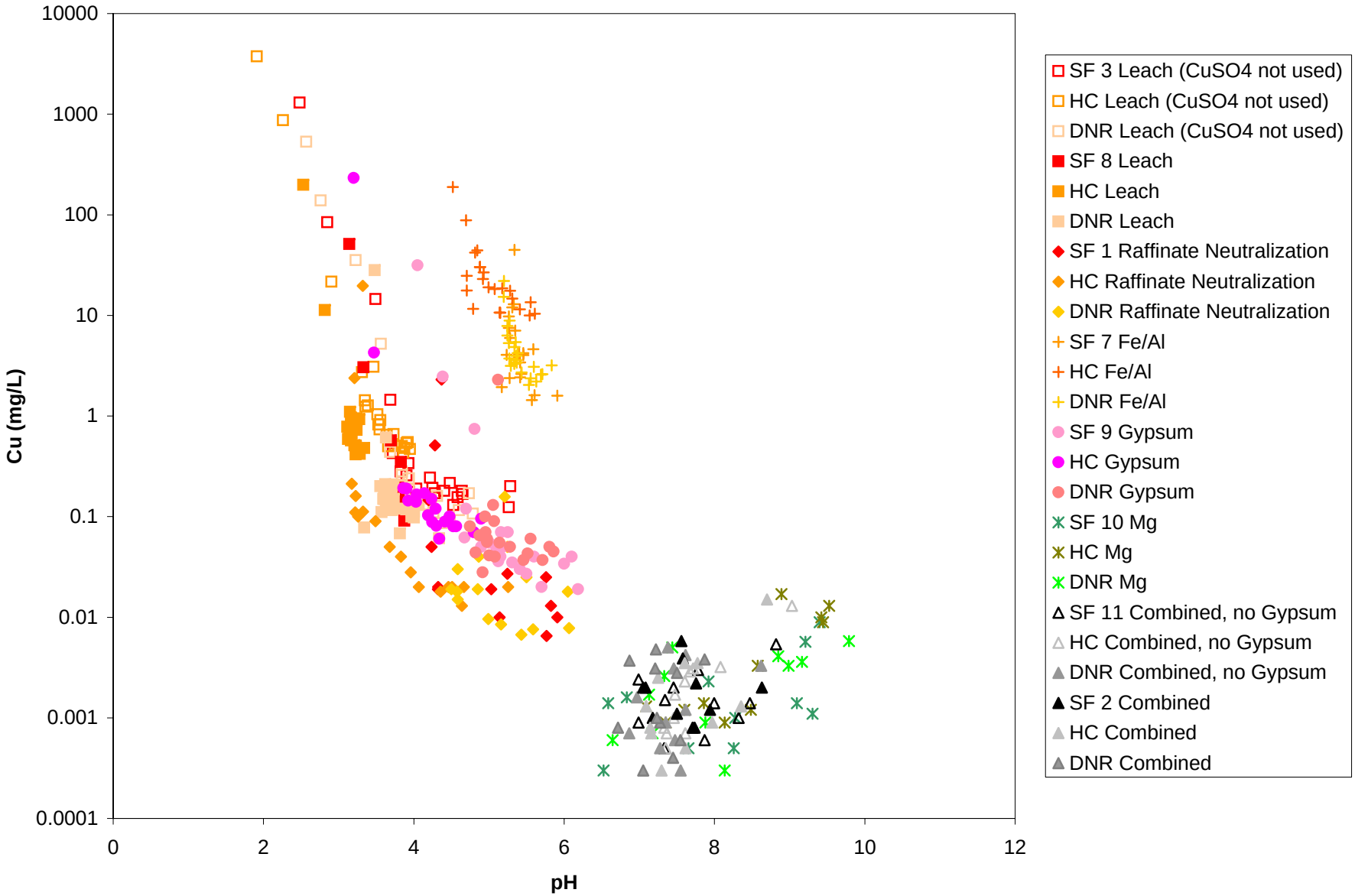


M(pH)

SRK Consulting
February 2007

Appendix D
Scatter Plots Showing Metal Concentrations Compared to pH

Graph D.8



M(pH)

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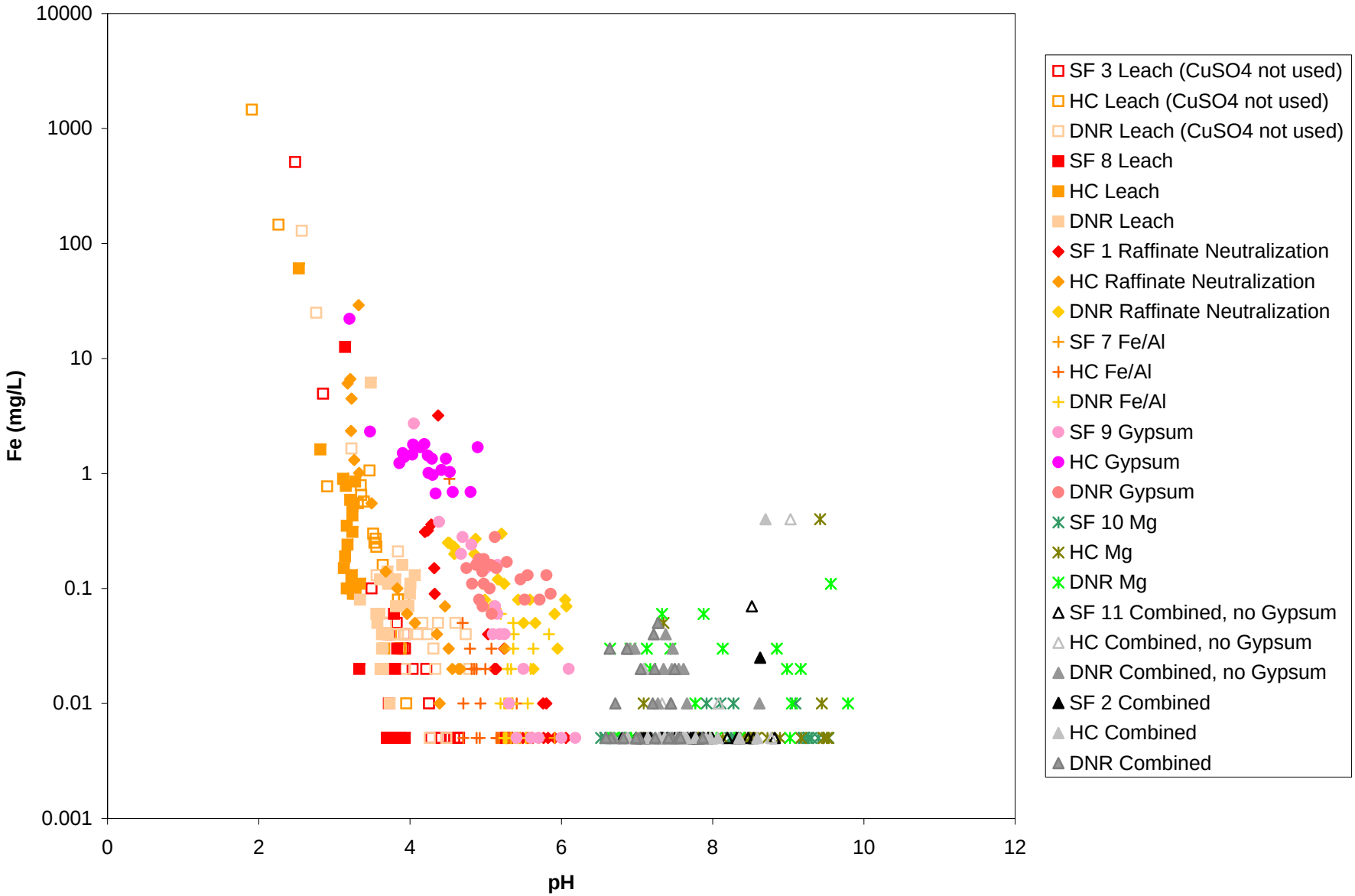
February 2007

Appendix D
Scatter Plots Showing Metal Concentrations Compared to pH

Graph D.9



M(pH)

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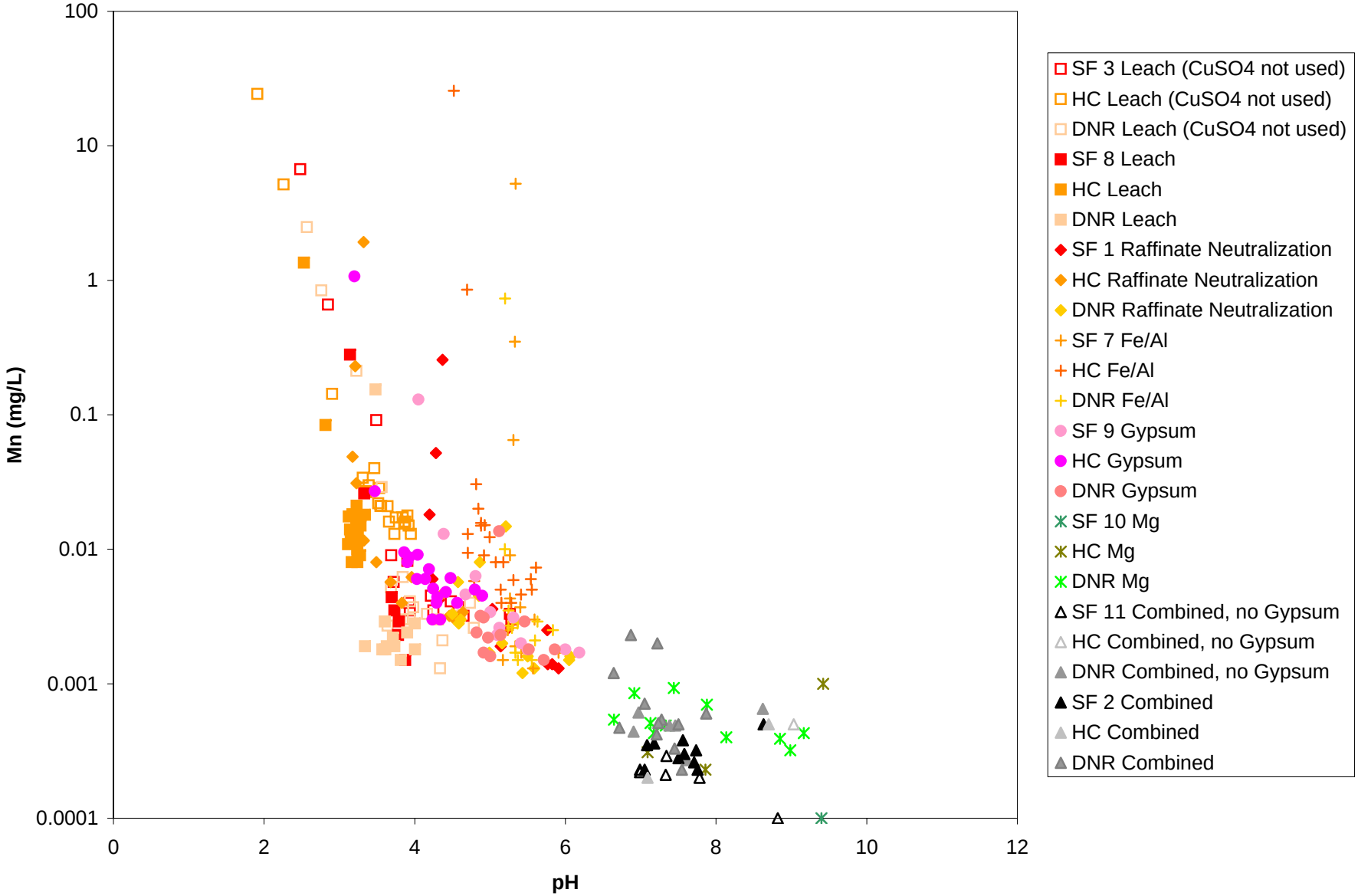
February 2007

Appendix D
Scatter Plots Showing Metal Concentrations Compared to pH

Graph D.10



M(pH)

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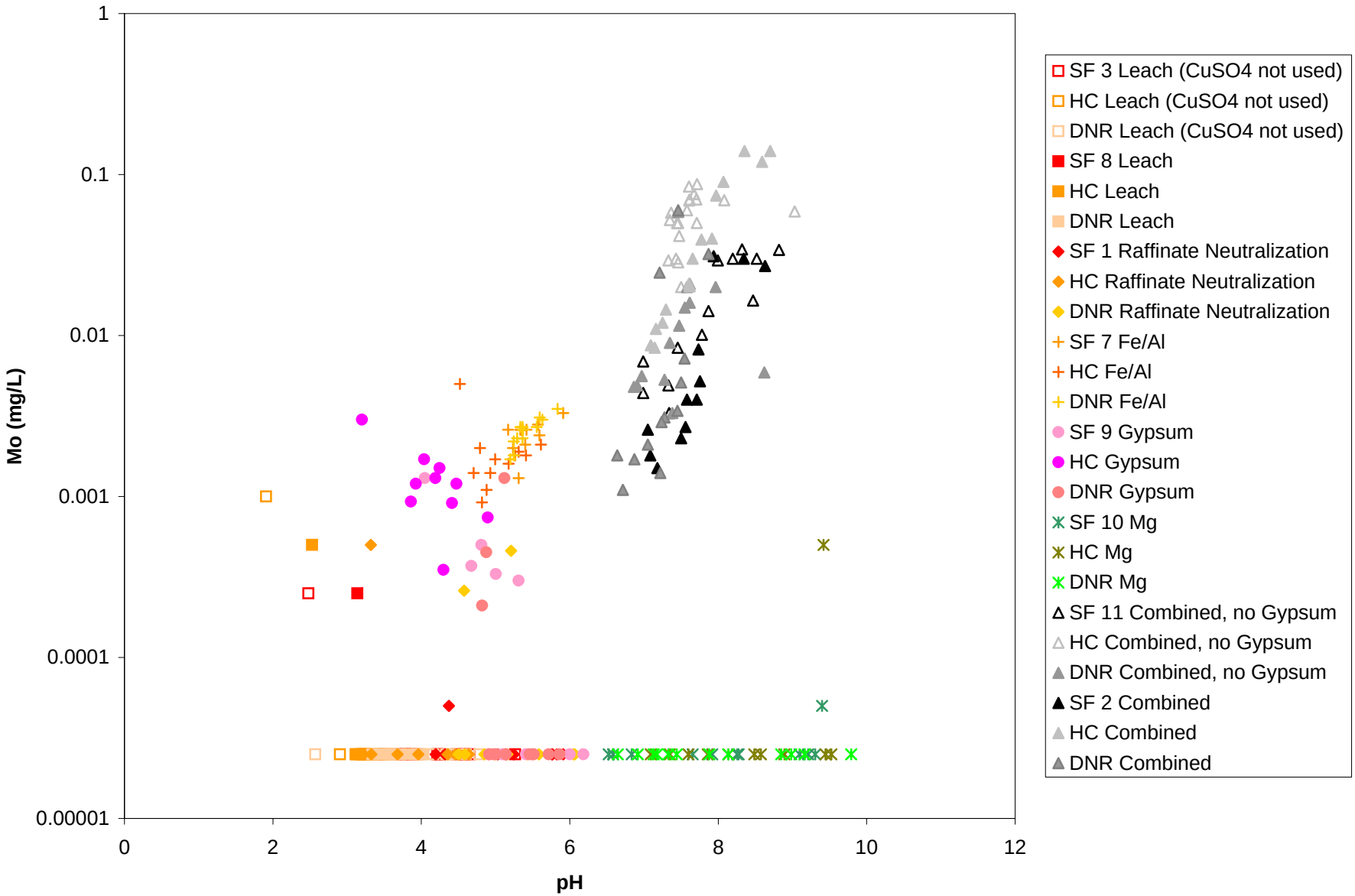
Appendix D
Scatter Plots Showing Metal Concentrations Compared to pH

Graph D.11



M(pH)

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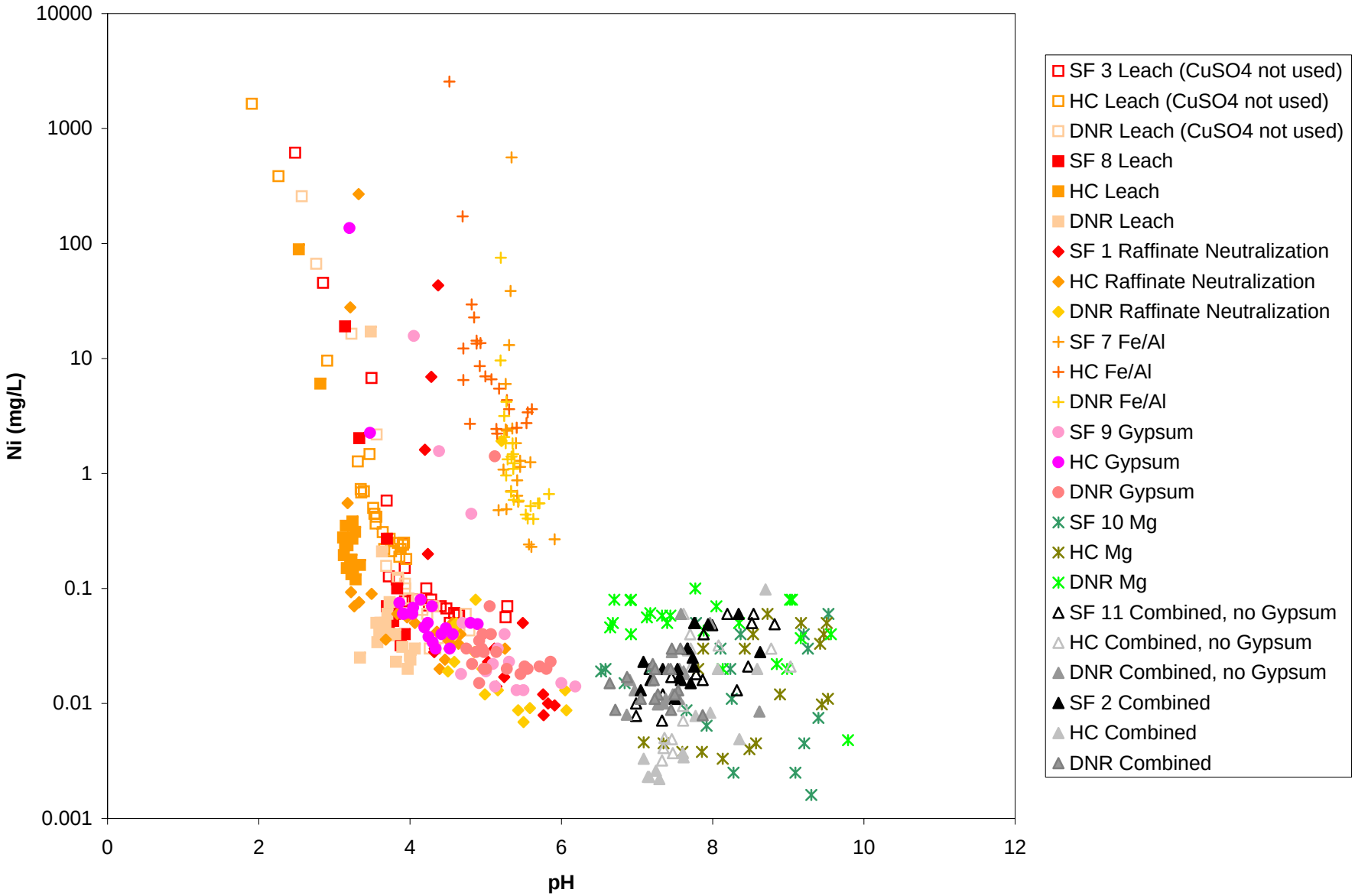
February 2007

Appendix D
Scatter Plots Showing Metal Concentrations Compared to pH

Graph D.12



M(pH)

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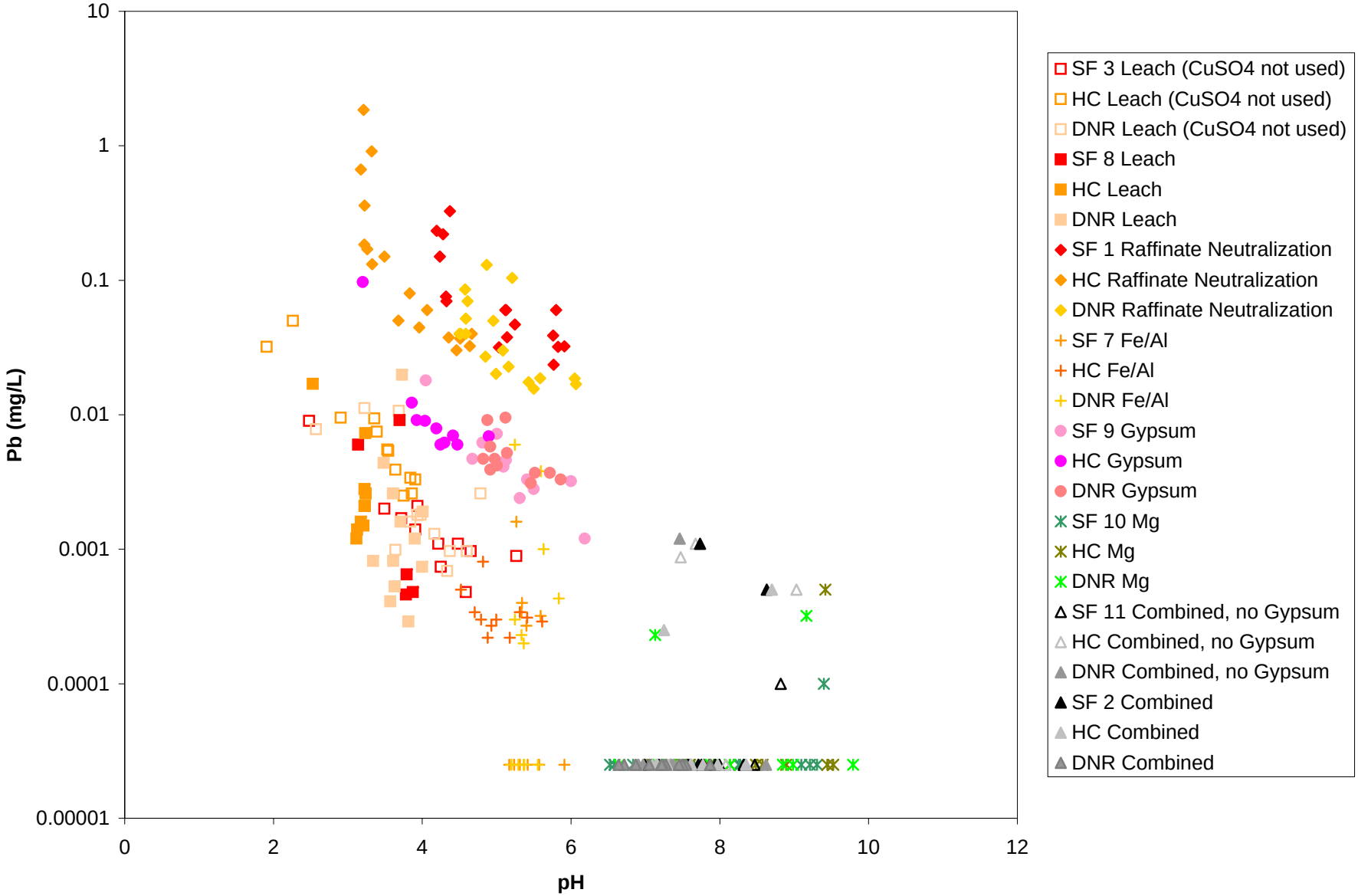
February 2007

Appendix D
Scatter Plots Showing Metal Concentrations Compared to pH

Graph D.13



M(pH)

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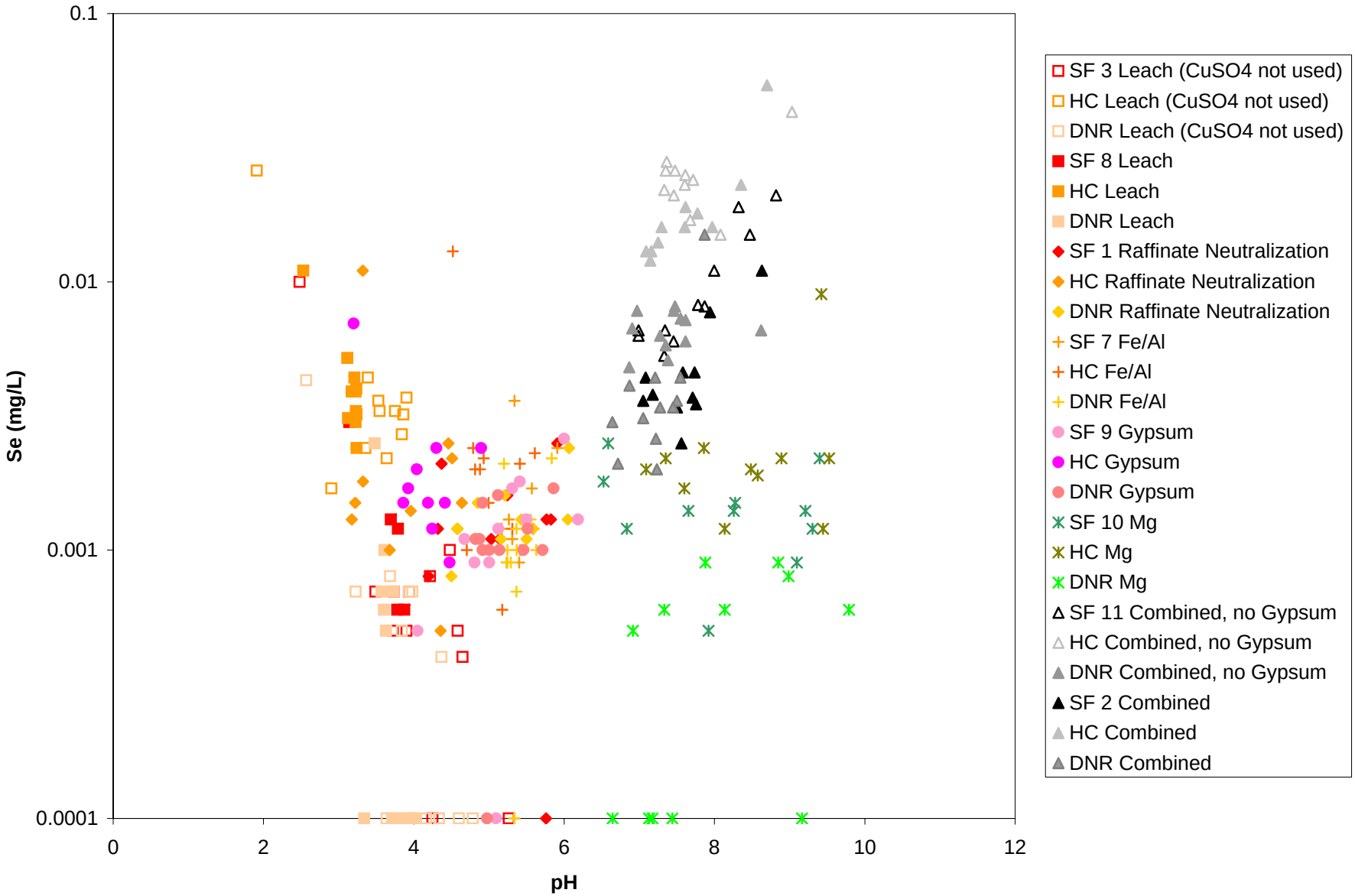
February 2007

Appendix D
Scatter Plots Showing Metal Concentrations Compared to pH

Graph D.14



M(pH)

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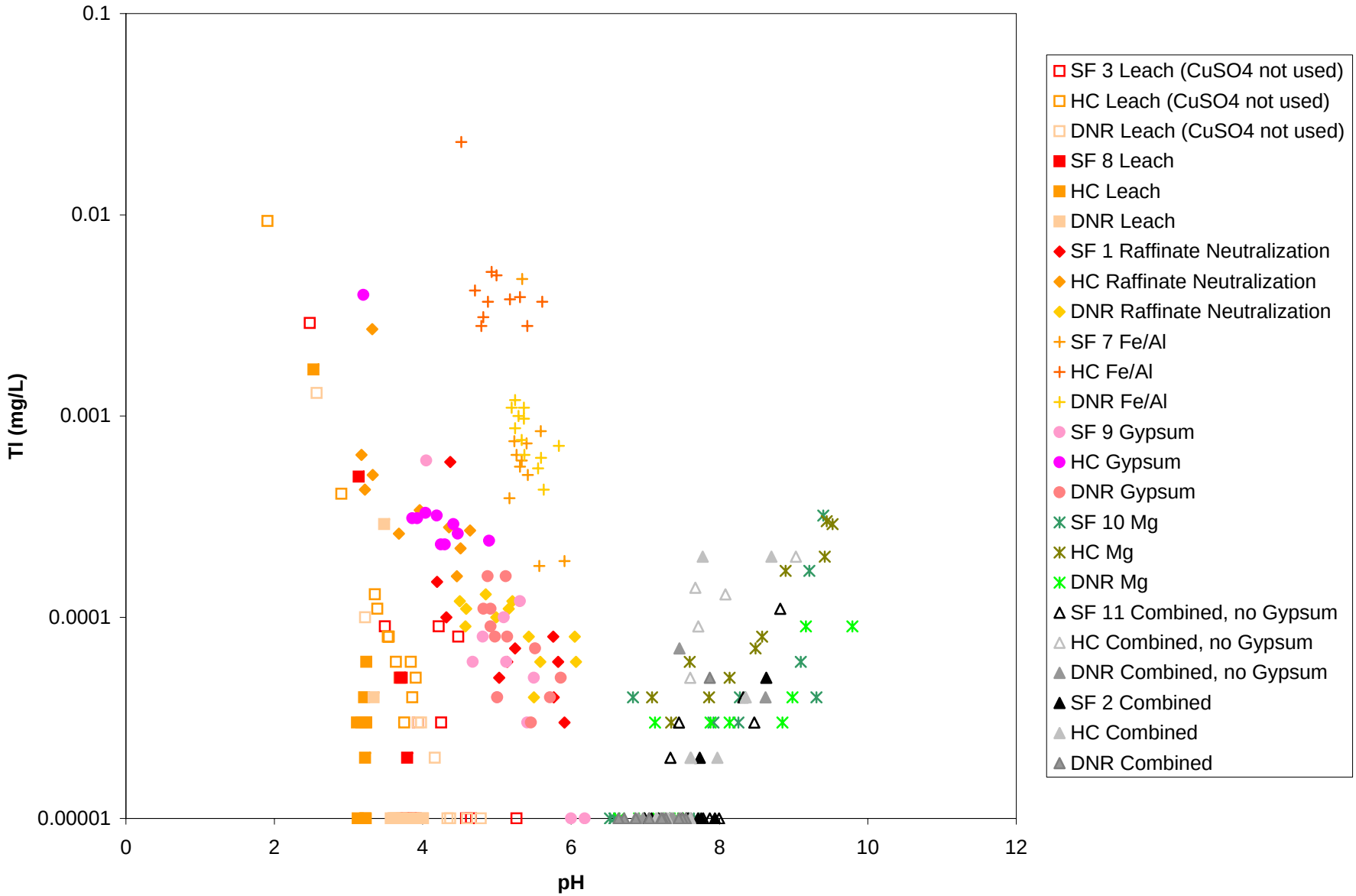
February 2007

Appendix D
Scatter Plots Showing Metal Concentrations Compared to pH

Graph D.15



M(pH)

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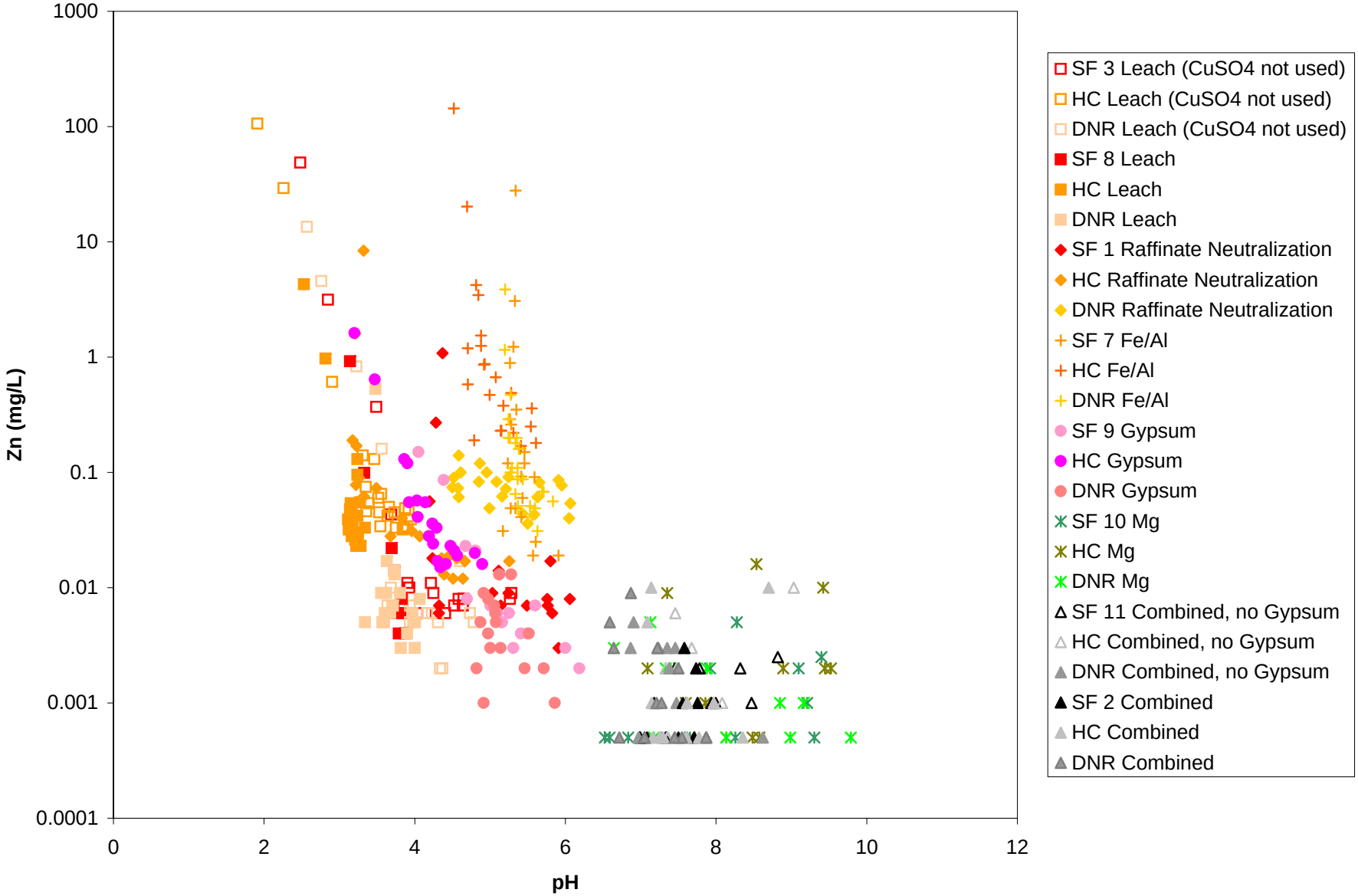
February 2007

Appendix D
Scatter Plots Showing Metal Concentrations Compared to pH

Graph D.16



M(pH)

SRK Consulting
February 2007

Attachment 2-13

SRK Consulting. RS53/RS42 - Waste Rock Characteristics/Waste Water Quality Modeling - Waste Rock and Lean Ore – NorthMet Project - DRAFT. March 9, 2007.

RS53/42
Draft 01
March 9, 2007

RS53/RS42 – Waste Rock Characteristics/Waste Water Quality Modeling - Waste Rock and Lean Ore – NorthMet Project - DRAFT

Report Prepared for

Polymet Mining Inc.

Report Prepared by

SRK Consulting
Engineers and Scientists

February 2007

RS53/42
Draft 01
March 9, 2007

RS53/RS42 – Waste Rock Characteristics/Waste Water Quality Modeling - Waste Rock and Lean Ore – NorthMet Project - DRAFT

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E-mail: vancouver@srk.com Web site: www.srk.com

SRK Project Number 1UP005.001

February 2007

Executive Summary

PolyMet Mining (PolyMet) is proposing to develop the NorthMet Project (former Dunka Road Project of US Steel) near Babbitt, Minnesota. As a part of the Minnesota Department of Natural Resources (MDNR) "Permit to Mine" process a complete "mine waste characterization" is required (Minnesota Rules Chapter 6132.1000). RS53/42 describes characterization of waste rock (including waste rock, lean ore and ore) and prediction of rock stockpile drainage chemistry.

The issues associated with waste rock at NorthMet are expected to include acid rock drainage (ARD) and leaching of some heavy metals. The latter in particular are expected to include nickel and cobalt both of which do not require acidic conditions to be mobilized at elevated concentrations.

The MDNR has been researching ARD in Duluth Complex and related host rocks (Virginia Formation) since the late 1970s. The research has focused on mineralized rocks associated with North Shore Mining's taconite Dunka Pit in the South Kawishiwi Intrusion and the Babbitt and NorthMet Deposits in the Partridge River Intrusion. The results of this research were the basis for design of PolyMet's geochemical testing program for waste rock, lean ore and ore at the NorthMet Project developed in the consultation with the MDNR. PolyMet's extensive test program and includes 92 ASTM humidity cells and 20 kinetic tests using MNDR methods. The samples being tested were characterized using a variety of chemical and mineralogical techniques.

Interpretation of MDNR's long term (nearly two decades) data, and about 1 year of PolyMet's data resulted in the following main findings for Duluth Complex rocks:

- Acidic leachate has not developed after 18 years of testing of samples containing less than 0.4% total sulfur despite the presence of sulfide minerals and near total absence of carbonate minerals.
- Sulfide oxidation rates are directly proportional to sulfur content with a secondary relationship related to sulfide mineralogy. Other variables (rock type, position in intrusive stratigraphy) do not appear to be important.
- If acidic conditions develop, the transition to acidic pHs takes many years. Following the onset of low pH, and depletion of sulfide minerals, pHs recover.
- Metal leaching is strongly linked to pH. Nickel in particular is sensitive to pH as it decreases below 7. Observed changes in pH and resulting enhanced metal leaching are linked to testwork conditions but provide useful information about metal leaching mechanisms.

The proposed explanation for the observations is that weathering of the main silicates (plagioclase and olivine) by weak carbonic acid (that is, atmospheric carbon dioxide dissolved in water) produces alkalinity. If the alkalinity generation rate from silicate weathering exceeds the acid generation rate from sulfide oxidation, it is predicted that leachate pHs will remain above neutral. Testwork results supported by geochemical modeling are internally consistent. It can be demonstrated that acidic

leachate is unlikely to be produced by rock containing 0.12% to 0.31% sulfur with the range due to differences in sulfide mineralogy.

The findings of the testwork program were used to develop a waste segregation scheme that includes three main categories (number 2 to 4):

1. Waste rock with negligible potential to produce ARD but likely to have drainage with component concentrations exceeding water quality objectives.
2. Waste rock with potential to produce ARD but with a delay of at least 5 years. Component concentrations are expected to exceed water quality objectives by a very wide margin.
3. Waste rock with potential to produce ARD immediately. Similarly, component concentrations are expected to exceed water quality objectives by a very wide margin.

All of these categories are defined as "reactive" because drainage would be unsuitable for direct discharge. The concept of a category for which drainage would be suitable for direct discharge was evaluated but not found to be achievable because hardness-based water quality discharges standards for copper may not be met.

Using tonnage information for waste rock and lean ore stockpiles provided by PolyMet Mining, and infiltration estimates provided by Barr Engineering, drainage chemistry was calculated on an annual basis through the operating life and closure period of the site. These predictions were adjusted to reflect water quality data for waters in contact with mineralized rock obtained by MDNR and PolyMet.

Final concentrations were provided to Barr Engineering for use in stockpile and water treatment design.

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1 Introduction

1.1 Background

PolyMet Mining (PolyMet) is proposing to develop the NorthMet Project (former Dunka Road Project of US Steel) near Babbitt, Minnesota. As a part of the Minnesota Department of Natural Resources (MDNR) "Permit to Mine" process a complete "mine waste characterization" is required (Minnesota Rules Chapter 6132.1000). RS53/42 describes characterization of waste rock (including waste rock, lean ore and ore) and prediction of rock stockpile drainage chemistry.

The issues associated with waste rock at NorthMet are expected to include acid rock drainage (ARD) and leaching of some heavy metals. The latter in particular are expected to include nickel and cobalt both of which do not require acidic conditions to be mobilized at elevated concentrations.

1.2 Objective

The specific objectives of this program include:

- Refinement of preliminary waste rock management criteria developed by PolyMet and MDNR.
- Development of water chemistry predictions for stockpile drainage water for input to water impact assessment and water treatment design.

1.3 Design and Consultation Process

The waste rock characterization program was developed in consultation with staff from the MDNR. The consultation included the following steps:

- December, 2004. PolyMet submitted a draft "Work Plan for Geochemical Characterization of Rock and Concentrator Flotation Tailings". The plan was presented to MDNR representatives.
- January 31 and February 1, 2005. Meetings were held by teleconference between SRK and MDNR representatives to further discuss the variables potentially affecting water chemistry from waste stockpiles.
- March 17, 2005. MDNR requested additional information on the tonnages of the major units and rock types, and the distribution of sulfur and minerals.
- March 28, 2005. PolyMet provided the requested information.
- April 12, 2005. MDNR provided a sample selection matrix. This matrix was accepted by PolyMet and is the basis for the selection of samples described in this document.
- May 15, 2005. MDNR provided a design for specific testwork.

- May 17, 2005. MDNR provided a design for specific testwork.
- June 6, 2005. A draft sampling plan was submitted to MDNR.
- June 15, 2005. MDNR provided comments on the draft plan.
- June 22, 2005. SRK provided responses and discussion of the MDNR comments in a letter to MDNR which were discussed during a teleconference on June 27, 2005.
- July 5, 2005. SRK provided results of candidate samples selected for kinetic testing in a memorandum to MDNR.
- July 13, 2005. MDNR provided comments on the July 5, 2005 SRK memorandum.
- July 15, 2005. SRK provided clarification on sample selection in a memorandum to MDNR.
- July 20, 2005. MDNR notified SRK and PolyMet that kinetic testing on the majority of waste rock samples could be initiated. It was recognized that analysis of a few candidate samples was ongoing.
- August 4, 2005. MDNR provided recommendations for lean ore characterization.
- August 29, 2005. As requested by SRK, MDNR provided additional rationale for the recommendations on lean ore sampling selection.
- September 14, 2005. Lean ore sample selection was further discussed during a conference call which provided the basis for completion of the characterization plan.

The resulting "Waste Rock and Lean Ore Geochemical Characterization Plan NorthMet Project, Minnesota" (Appendix A) was fully implemented in October 2005.

1.4 Structure of Report

This report combines results of two studies. RS53 provides results of characterization of waste rock, lean ore and ore and RS42 is the prediction of stockpile drainage water chemistry.

The structure of the RS42 report, which was a combination of RS53 and RS42, was agreed with the MDNR. The final version of the report outline was transmitted to the MDNR on April 26, 2006. The agreed outline has been followed, although in places this does not fit with the current thinking on the project. If any sections are redundant, the section heading is shown with a brief note to explain why the section is no longer relevant. A few minor sections have been added to help with clarity of the text.

1.5 Acknowledgements

SRK acknowledges the contribution of the following individuals to RS42:

- Barr Engineering – John Borovsky.
- Economic and Environmental Geochemistry, Inc. – Ron Schmiermund
- MDNR – Mike Berndt, Paul Eger, Jennifer Engstrom, Kim Lapakko.
- PolyMet Mining – Steve Geerts, Richard Patelke, Jim Scott.
- SRK Consulting – Stephen Day, Madeleine Corriveau, Ingrid Rozas, Kelly Sexsmith, Karen Wagner.

The SRK team recognizes in particular the contribution of our colleague Ingrid Rozas who died suddenly in July 2006.

2 Water Chemistry Prediction Methods

The method used to predict stockpile drainage chemistry dictates the inputs needed for the calculations. The following sections describe the prediction methods considered and how the approach chosen was factored into the design of the characterization program.

2.1 Theoretical Method

Several factors interact to influence the chemistry of waste rock stockpile drainage (MEND 2000). These include geology (chemical characteristics), construction method (including the effect of reclamation measures), climate (distribution of precipitation, availability of water, temperature, and air movement) and hydrogeology (origin of water flow).

Northwest Geochem (1991) comprehensively reviewed modeling methods to predict the chemistry of waste rock stockpile drainage and concluded that "no model exists which can even generally simulate the most critical physical, geochemical, and biological processes in waste-rock piles". Subsequently, MEND (2000) concluded that "If assessments of the behavior of waste rock stockpiles are required, it should be realized that no reliable modeling approaches are available. Advances have been made in understanding and modeling the various processes (e.g. flow in unsaturated materials, pyrite oxidation) but reliably coupling the models remains primarily a topic of research."

For the purpose of predicting drainage chemistry from waste rock stockpiles at NorthMet, theoretical modeling has been limited to consideration of well-established thermodynamic first principles.

2.2 Analog and Empirical Methods

2.2.1 Analog

The analog method uses comparisons with other sites in similar geological settings to predict water chemistry (Plumlee and Nash 1995).

For example, Caruccio and Ferm (1974) first proposed that paleo-environment is an important factor in determining water quality for coal mines because coal seams formed in salt water environments have higher initial sulfur content and are therefore more prone to the formation of pyrite during lithification and therefore generation of acid when these materials are exposed during mining.

Recently, Day and Rees (2006) compiled data for six porphyry copper mine sites in western Canada and found strong similarities between geographically scattered sites despite variations in host rock geology and climate. Porphyry deposits form by interaction of hot water with volcanic or plutonic rocks typically in a sub-volcanic environment. The similarities in drainage chemistry reflected the relatively simple sulfide mineralogy of these deposits and the formation of common alumino-silicate alteration minerals.

2.2.2 Empirical

The Empirical Method is also sometimes referred to as "scale-up calculations" because it involves translation of results from small laboratory or field tests to full-scale facilities. The attraction of this approach is that it involves the use of site-specific laboratory and field data, and does not rely on theoretical calculations. The results are transparent and easily explained. However, a significant issue is that the resulting concentrations are typically excessively conservative. This may be attractive for environmental assessment purposes but the resulting overly conservative predictions may unreasonably drive the need for mitigation measures to address potential water quality impacts which in turn may result in other impacts. A necessary component of the Empirical Method is the adjustment of resulting predictions to reflect basic geochemical controls as determined by geochemical modeling (Theoretical Method) and experience from other sites (the Analog Method).

There are three main steps in the method:

1. Design of a laboratory program to collect site specific chemical weathering rate information;
2. Calculation of component concentrations based on rock mixtures, scale-up factors, and hydrological considerations; and
3. Adjustment of calculated component concentrations to reflect geochemical constraints indicated by testwork, thermodynamic constraints and experience.

Additional description of these steps is provided in the following sections.

Laboratory Program

The laboratory program is designed to obtain weathering rates, typically expressed as mass of component released per mass of rock per time step (e.g. week). Rates are obtained for all rock types and a range of the characteristics for each rock type. Generally, the objective is to obtain rates that can be correlated with bulk characteristics of the rock so that overall rates can be calculated for mixtures. Examples of strong correlations of sulfur content with sulfate release are common and include humidity cell data obtained for this project. When good correlations are established, the rates can be interpolated between points.

Calculation of Concentrations Using Scale-Up Factors

The purpose of the scale-up calculation is to convert laboratory measured generation rates (for example in mg/kg/week) to drainage concentrations (in mg/L). The scale-up calculations need to consider the rock type mixture, temperature effects, grain size, rock mass, flow path development, and water volume. If a variety of different rock types are present, pore water concentrations are calculated for each rock type, and then mixed according to the proportion of rock types in the stockpile as provided by mine planners.

Temperature should be considered because oxidation rates decrease as temperatures decrease and vice versa. This correction is typically applied based on the average annual site temperature, and can be calculated using the Arrhenius equation. This equation provides a good approximation of actual rate decrease observed in laboratory experiments (SRK and Mehling Environmental 2006). The laboratory rate (R) is therefore adjusted using a constant factor (k_T) to obtain the adjusted rate (R_a):

$R_a = R.k_T$

This correction should be applied cautiously for reactive materials because the sulfide oxidation reaction is exothermic and will offset cooler site conditions.

The next step is to consider particle size effects. There are two issues to consider:

- Oxidation is a surface area phenomenon. A larger surface area provides a greater reactive surface area, and
- Reactive minerals encapsulated in large rock types do not oxidize at the same rate as exposed reactive particles because oxygen must diffuse through a solid rather than a gas.

Figure 2-1 illustrates the relationship between particle size and surface area for particles occurring as cubes. The graph shows that below a particle size of 0.1 cm, the available surface area increases rapidly. For larger particles, the area contribution is insignificant. Therefore, a standard humidity cell containing -¼" (0.6 cm) material provides a good representation of the surface area of a rock mixture containing much larger particles. For example, in a typical rock mixture containing 5% by weight finer than this size, the particles finer than 0.6 cm can account for 95% of the surface area.

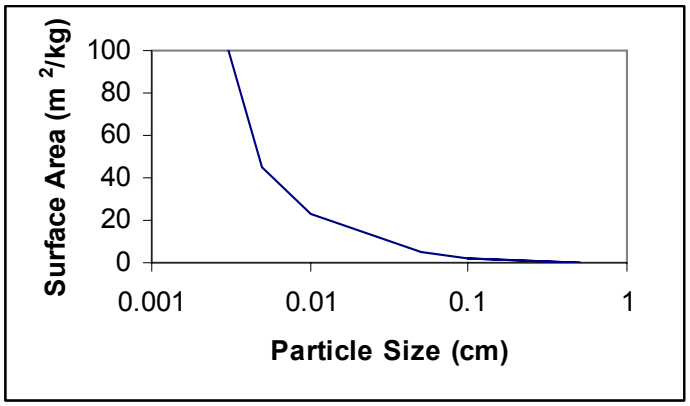


Figure 2-1: Particle Surface Area as a Function of Particle Size for Cubic Particles.

The correction for particle size then becomes a ratio of the fine-grained reactive mass (M_r) to the total mass (M):

$R_a = R.k_T.(M_r/M)$.

The scale up of rate to full scale is then obtained by multiplying by M to obtain:

$R_a' = R.k_T.M_r$ (in mg/week).

R_a' is the scale-up of laboratory rate to field rate for total mass; however, it represents production rather than release because humidity cells are designed to be fully flushed. Under field conditions, the entire rock mass is not flushed due to flow path development. For low waste rock stockpiles, flushing is likely to be relatively thorough but as a stockpile becomes higher, the flow path length increases and the degree of flushing decreases (e.g. Morin 1991; Morin and Hutt 1997). R_a' can therefore be converted to leached mass (L) by multiplying by a flushed proportion (k_f):

$L = R.k_T.M_r. k_f$ (mg/week).

This leached loading can then be converted to a concentration (C) by dividing by the volume of infiltrating water (Q):

$C = R.k_T.M_r.k_f/Q$ (mg/L).

The term with the greatest variability in this equation is usually R. Other terms have well-defined boundary conditions that do not vary over orders-of-magnitude. By varying R to reflect uncertainty in the correlation of weathering rates to bulk rock characteristics, the sensitivity of estimates of C can be developed.

Adjustment of Calculated Concentrations

As noted above, calculation of concentrations using the empirical method often results in unusually high concentrations and it is therefore necessary to evaluate the individual concentrations with consideration of chemical principals and experience from other sites.

The first step in evaluation of the calculated scale-up concentrations is to examine the major components (sulfate, calcium, magnesium, etc) and some minor components (iron, aluminum). An appropriate approach is to enter the data into a thermodynamic equilibrium model (such as MINTEQA2, PHREEQE). These models can assist with identifying concentrations that are not supportable thermodynamically. For example, when dissolving common salt in a container of water, only a finite amount can be dissolved after which any additional salt remains as solid in the bottom of the container. The water is said to be "saturated" with respect to salt, and the resulting sodium and chloride concentrations in solution can be no greater than when the salt stops dissolving. The reverse is not always true though. It is possible for a solution to be over-saturated with respect to a solid during evaporation. In this case, the energy required to start forming (or nucleating) the first crystals is not available. Thermodynamic models must therefore be used cautiously.

In summary, the result of the empirical calculation is a set of concentrations for major components that typically exceed expected values. This indicates that some products of the weathering reactions remain stored in the rock and are not leached by infiltrating processes. This fact can be applied to adjustment of minor and trace components.

Evaluation and Adjustment of Minor and Trace Components

Evaluation of minor and trace components is treated separately because they occur at concentrations that are not a major part of the ion balance and with some exceptions occur at concentrations below limits implied by saturation controls. In some cases, concentrations can be adjusted using well known thermodynamic controls. For example, copper concentrations are commonly controlled by the formation of copper carbonate (the mineral malachite) or oxide (tenorite).

Other components, for example, lead, zinc, nickel, cobalt, cadmium and selenium are often predicted by the scale-up calculation to be released at concentrations which seem to be "high" based on experience. To refine the predictions, calculated concentrations can be compared to measurements of chemistry for waters in contact with weathering rock. These waters can include natural springs, shallow groundwater, drainage from existing mine waste facilities and results of extraction tests on weathered rock. These data sources are compiled and evaluated on component versus pH scatter plots to determine if there are any correlations with pH, which is the main geochemical control on water quality. Possible solubility limits are implied by the maximum concentration of a dissolved component at any given pH. This approach can be used to adjust the concentrations calculated by the scale-up calculation provided that there is a reasonable expectation that waters characterized by the database are in equilibrium with weathering rock.

The explanation for the high prediction is probably that these components are being released as part of weathering processes but remain stored in the rock. The expected retention of iron released by iron sulfide oxidation represents a sink for these elements through sorption processes. At many sites, pyrite and/or pyrrhotite are the dominant sources of leachable iron, and are likely also the source for many trace elements. The ratio of iron to other metals is very high and represents a significant source of sorptive capacity. Because this process is pH dependent, it is expected that component concentrations would be correlated with pH (e.g. Day and Rees 2006).

2.2.3 Method Selected

Subaerial Disposal

The method selected involved a combination of the purely empirical approach with minor use of thermodynamic theory. Similar approaches have been used recently and accepted by regulators in other jurisdictions for new mine projects (e.g. Pogo Mine, Alaska (AKDNR, AKDEC, USEPA); Brule Coal Project; Trend Coal Project and Galore Creek Project, British Columbia, Canada (Provincial agencies).

Useful analog data is available for the Duluth Complex, but a search for drainage chemistry data for similar sites with comparable low sulfur content was unsuccessful (see Section 7.2 below). The calculated concentrations obtained from the empirical method were compared to data from the relevant Duluth Complex analog sites.

Figure 2-2 illustrates conceptually the integration of mine waste rock scheduling, mine designs, hydrological information and geochemical data to develop temporal water chemistry predictions as the mine is developed.

For each year (n) of the mine, PolyMet provided the quantity and chemical composition of waste rock (M_n) placed in each waste rock and lean ore stockpile from a block model. The empirical method is then used to estimate the load leached from this waste rock mass in subsequent year m ($L_{n,m}$). The load originating from all rock placed up to year n is totaled to obtain the total load produced by the waste rock stockpile in that year. The concentration is estimated by "dissolving" this load into the estimated water inflow to the waste rock stockpile determined by the area in year m (A_m) and average annual infiltration (I) calculated by Barr Engineering (2007a) in RS21.

Figure 2-2 shows how the calculation is performed for year 1 and Year 2. The mass of rock placed in the stockpile was M_1, and the load released is $L_{1,1}$. The infiltration is A_1 x I, which becomes the volume of water leaving the stockpile (V_1). The calculated concentration is:

$$\frac{L_{1,1}}{V_1}$$

In Year 2, a mass of rock M_2 is added and the area increases to A_2. The total load released is ($L_{1,2}$ + $L_{2,2}$). The concentration in drainage in Year 2 is:

$$\frac{(L_{1,2} + L_{2,2})}{V_2}$$

Subsequent sections of the report will show that the proposed mine plan includes segregation of waste rock based on potential to produce ARD. Some waste rock is expected to have delayed generation of ARD meaning that it may be several years after the rock is placed in a stockpile before that rock begins to produce ARD. This is handled in the prediction calculation by allowing two values of L depending on whether the rock is overall non-acidic or acidic. The switch between conditions is determined based on the age of the rock relative to its year of placement in the stockpile. If the rock has been in the stockpile longer than the time expected for acidic conditions to begin then the rock is assumed to be acidic and this higher load contributes to the total load generated by the stockpile.

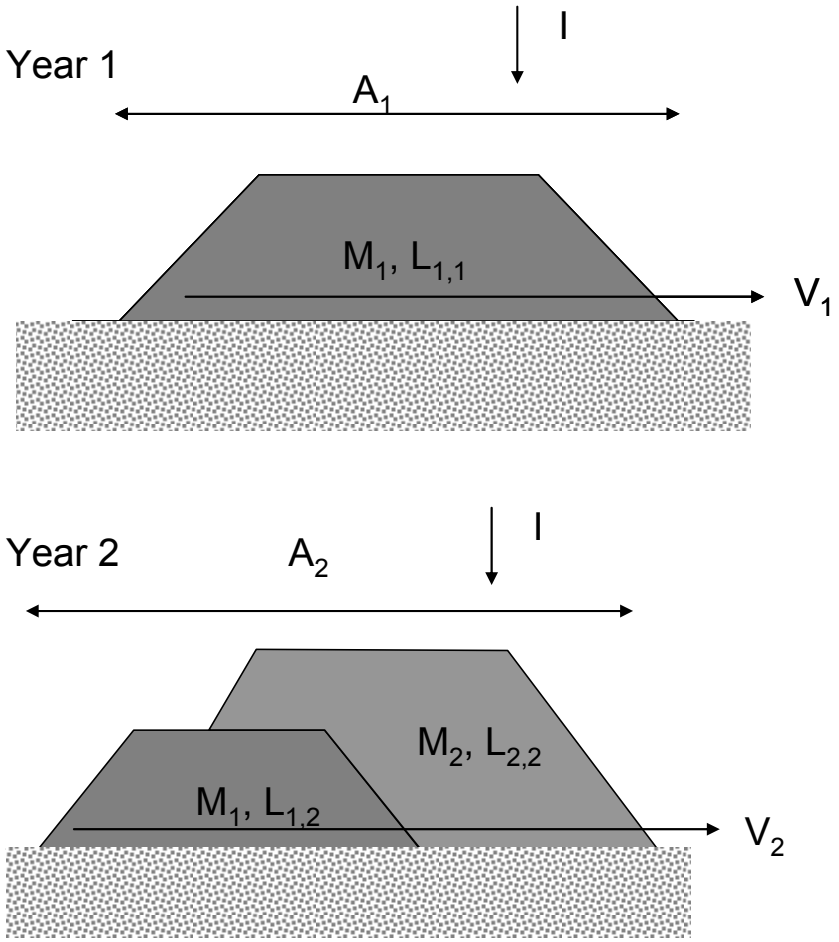


Figure 2-2: Water Quality Prediction Method

The calculated concentrations were adjusted as described for empirical method. The adjustments are described in more detail subsequently but include consideration of the formation of secondary minerals which are expected to include major minerals such as sulfates (gypsum), carbonates (calcite, dolomite), oxides (gibbsite) and silicate (e.g. kaolinite) minerals formed in response to weathering of silicate minerals, and secondary minerals formed by precipitation of metals released by oxidation of sulfide minerals.

Subaqueous Disposal

Subaqueous disposal will involve flooding of waste rock backfilled in the NorthMet East and Central Pits. The primary effect will be the leaching of accumulated weathering products as the waste rock becomes submerged. The prediction method selected involves a calculation of the rates of accumulation under subaerial conditions. The load leached during flooding is calculated as the residual of load not flushing under subaerial conditions. For the purpose of calculation, the entire oxidized load in the rock is conservatively assumed to be immediately flushed on contact with water. The effect of backfill on pit water chemistry is calculated in RS31 (in preparation by SRK).

3 Program Design

3.1 Geological Background

The NorthMet Deposit is located in the intrusive Duluth Complex of northern Minnesota. Disseminated copper-nickel-iron sulfides (chalcopyrite, cubanite, pentlandite and pyrrhotite) with associated platinum group element (PGE) mineralization will be extracted from several igneous stratigraphic horizons.

In the vicinity of the NorthMet Deposit, the Duluth Complex intruded and assimilated the Virginia Formation, which consists of argillite and greywacke with minor interbeds of siltstone, graphitic argillite, chert, and carbonate. This formation is the stratigraphic footwall of the NorthMet Deposit, but also occurs as xenoliths ("inclusions") within the deposit.

Detailed description of the geological setting of the deposit is provided in ER03 (PolyMet 2007b).

3.2 Geochemical Background

The MDNR has been involved in research on the weathering characteristics of Duluth Complex Rock in the Partridge River Intrusion (NorthMet and Babbitt deposits) and the South Kawishiwi Intrusion (Dunka Mine) since the late 1970s (e.g. Lapakko et al. 2001; MDNR 2004). The following sections provide an overview of these projects and the principal findings. The MDNR have prepared detailed reports on these projects.

3.2.1 AMAX Shaft

Test Piles

Six roughly 1000-ton test piles were constructed from rock removed from a test shaft sunk into the Babbitt Deposit in 1977 (Lapakko 1993b; Lapakko et al 2002; MDNR 2004) by AMAX. The rock contained sulfur concentrations varying from 0.64% to 1.41%, copper concentration of 0.3% to 0.4%, and nickel concentrations of 0.08% to 0.09%. The copper and nickel content was comparable to ore at NorthMet but sulfur concentrations were much higher than will be expected for most waste rock at NorthMet. The piles were constructed on lined pads, and the rock surfaces of some were reclaimed with soils, glacial tills and some were vegetated (Table 3-1). Drainage from the piles was monitored from 1977 to 1994 after which the piles were dismantled and the rock encapsulated in concrete.

The MDNR provided full leachate monitoring data for the piles and background information on the characteristics of the piles (Engstrom 2006b, d).

Table 3-1: Characteristics of AMAX Stockpiles (Lapakko et al 2002)

	FL1	FL2	FL3	FL4	FL6	FL5
Date completed	4/20/77	4/20/77	4/20/77	4/20/77	9/30/77	9/30/77
S (%)	0.64	0.64	0.64	0.64	0.79	1.41
Cu (%)	0.35	0.35	0.35	0.35	0.34	0.31
Ni (%)	0.083	0.083	0.083	0.083	0.09	0.09
Mass (T)	1100	1100	830	1700[1]	1300	815
Volume (m^3)	540	530	400	930	630	400
Collecting area (m^2)	290	340	300	330	320	340
Surface area (m^2)	330	450	430	350	390	370
Cover material	none	topsoil	coarse sand (glacial till)	none	none	sandy till over coarse sand
Avg. cover depth (cm)	0	23	34	0	0	54
Vegetated	no	yes	yes	no	no	yes
Drainage period	1978-93[2]	1978-93	1978-89[3]	1978-89	1978-93	1978-93

1 Approximately 40% of pile FL4 was removed in 1982.
2 FL1 was partially covered with plastic during 1990
3 FL3 was covered with plastic and Hypalon terminating flow at the end of the 1989 field season.

All six piles showed declining leachate pH over the monitoring period (Figure 3-1). In the first half of each year, pH tended to be higher followed by decrease into the late summer and fall. This appeared to occur regardless of whether the rock was bare or covered.

The four piles constructed from rock containing 0.64% sulfur reached lowest pHs of about 5. The large bare FL4 pile showed the lowest overall pH of this group apparently stabilizing between pH 4 and 5 before being stopped in 1989. The equivalent bare FL1 pile showed the highest pH though eventually pH declined to the same level as the same rock with a soil cover.

The drainage pH for the pile with 0.79% sulfur decreased to about 4 in 3 years and remained between 4 and 5 until the pile was dismantled. The pile containing 1.41% sulfur produced pHs between 3 and 4 in 1 year and remained at this level until 1994 though pH increased slightly.

Sulfate concentrations were greatest for the two piles containing higher sulfur concentrations. FL5 (1.4% S) showed the highest sulfate concentrations (approaching 10,000 mg/L) and a peak in 1981. FL6 (0.79% S) showed a weak peak in 1982. The other piles showed slowly increasing sulfate remaining generally below 3,000 mg/L. Seasonal variations were apparent. Sulfate concentrations tended to be greater later in the year when pH was seasonally lower. Alkalinity release tracked downwards for all tests but remained detectable for FL1 and FL2 at between 2 and 10 mg $CaCO_3$/L.

Initially for all tests, the dominant cation was sodium, then calcium. As low pH conditions developed, the ratio of Ca/Mg decreased so that tests with lowest pHs showed magnesium as the dominant cation. The downward trend in the ratio for FL1 and FL2 was continuing when the test was stopped in 1993 but had stabilized for the other continuing tests. Cations showed the same seasonal variations as sulfate.

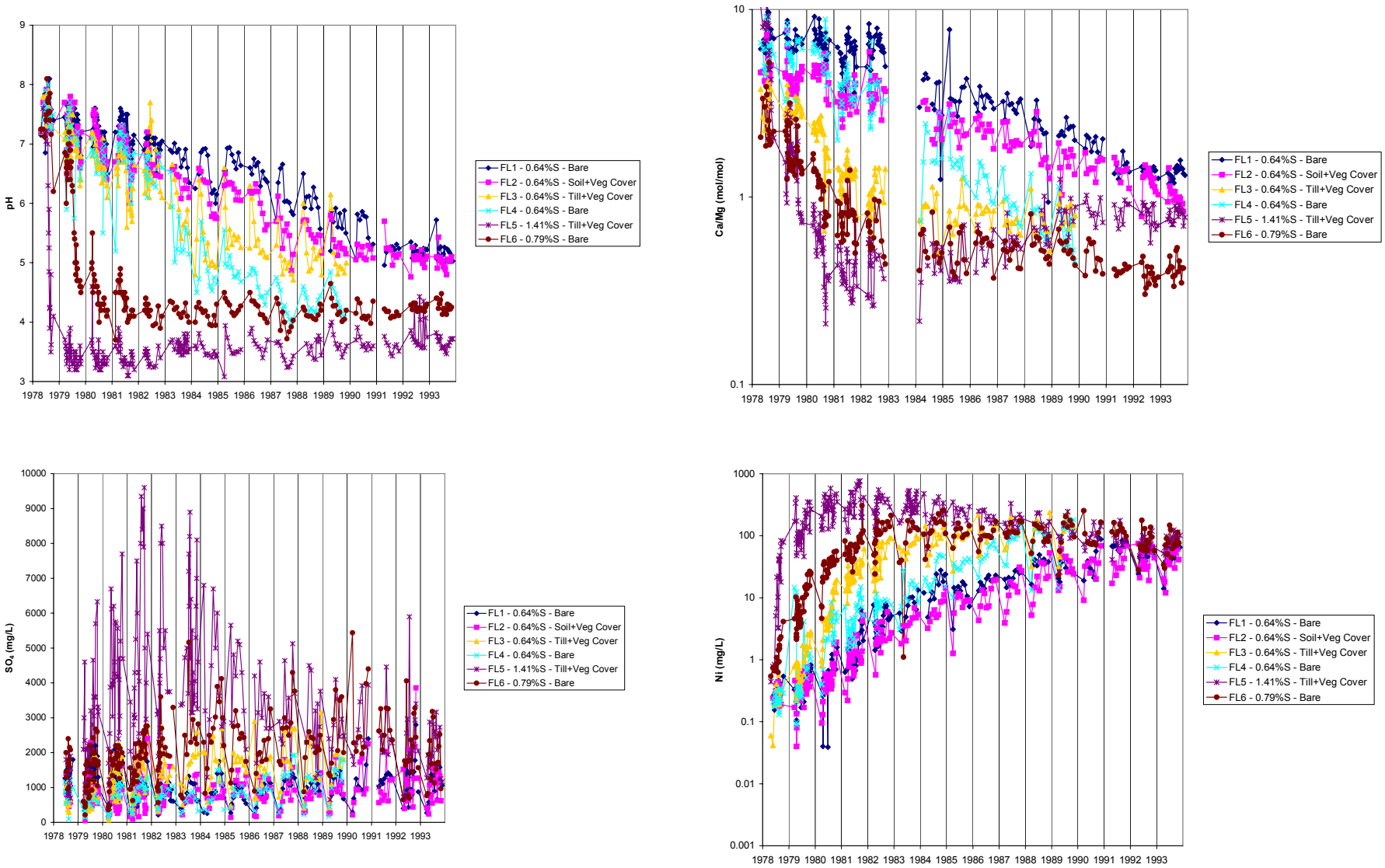


Figure 3-1: Leachate Results for AMAX Test Piles.

Nickel, copper and cobalt leaching all showed the same general trends of increasing metal concentrations as pH decreased then reaching relatively stable concentrations with seasonal variations paralleling sulfate. Nickel release for FL5 (1.4% S) initially peaked (with sulfate at concentrations approaching 800 mg/L) but then declined. Cobalt showed the same trend with peaks at 50 mg/L. Copper showed a weak long term peak approaching 200 mg/L. Zinc showed a similar trend with peaks approaching 30 mg/L.

Metal concentrations were negatively correlated with pH. The most leachable metal was nickel (concentrations were greater than 100 mg/L at lowest pHs), followed by copper (near 100 mg/L), cobalt (near 10 mg/L) and zinc (between 1 and 10 mg/L).

When the piles were dismantled, the MDNR found elemental sulfur and copper, gypsum, green precipitates and extensive cementation. The MDNR concluded that pyrrhotite was the main source of acid, and that plagioclase contributed to acid consumption for pH above 5. Below this pH, olivine became more important.

SRK used the monthly flow and sulfate concentration data provided by the MDNR to calculate the average annual release of sulfate from the piles. Where sulfate concentration data were occasionally missing, the next available data point was used for the missing value. Results of the calculation are provided in Figure 3-2.

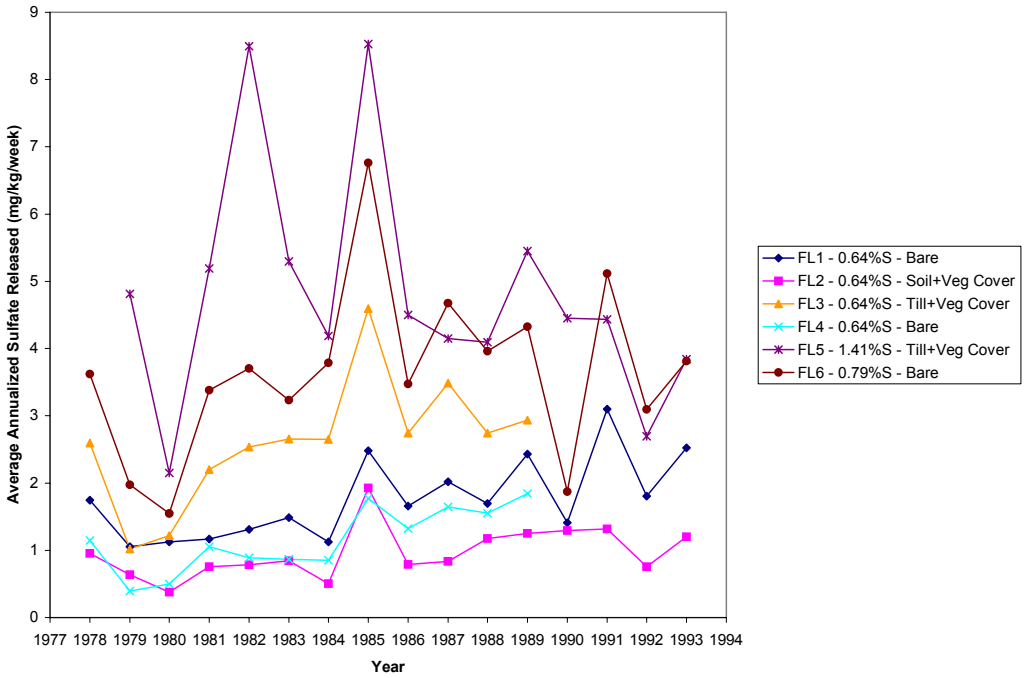


Figure 3-2: Average Sulfate Release from the AMAX Stockpiles.

The averages showed a range from 0.5 to 8.5 mgSO_4/kg/week. The lowest rates were for the 0.64% sulfur piles (uncovered and soil cover). The till covered pile containing 0.64% sulfur showed higher rates. The two higher sulfur piles indicated higher still sulfate release rates. FL5 with 1.41% sulfur showed the highest rates. The four lower sulfur piles all showed slight long term increases in sulfate release though the increase was not substantial.

Sulfate release may be partly limited by the presence of gypsum which was observed during dismantling of the piles. Thermodynamic modeling of the leachates generally indicated that sulfate concentrations were below levels required to form gypsum due to the high magnesium and sodium concentrations in the leachates indicating that sulfate release was also partly indicative of sulfide mineral oxidation and not just leaching of sulfate controlled by gypsum precipitation.

In summary, the monitoring data showed that rock with sulfur content at the tested levels (0.64% to 1.41%) produced acid rock drainage (ARD) but the sulfur content was a controlling factor in the severity and delay to onset of ARD.

3.2.2 AMAX Drill Core Tests

In a parallel experiment to the AMAX test piles, twenty-four 75-g samples crushed to -100+270 mesh containing sulfur concentrations between 0.47% and 2.57% were tested in MDNR's kinetic reactor (Lapakko 1993a; 1994b). The data were provided to SRK (Engstrom 2006d). The tests were operated for 30 to 49 weeks. Over this testing period, the minimum pH of the drainage varied from near 7 for the samples containing lowest sulfur to below 4 for samples containing the highest sulfur concentrations (Figure 3-3).

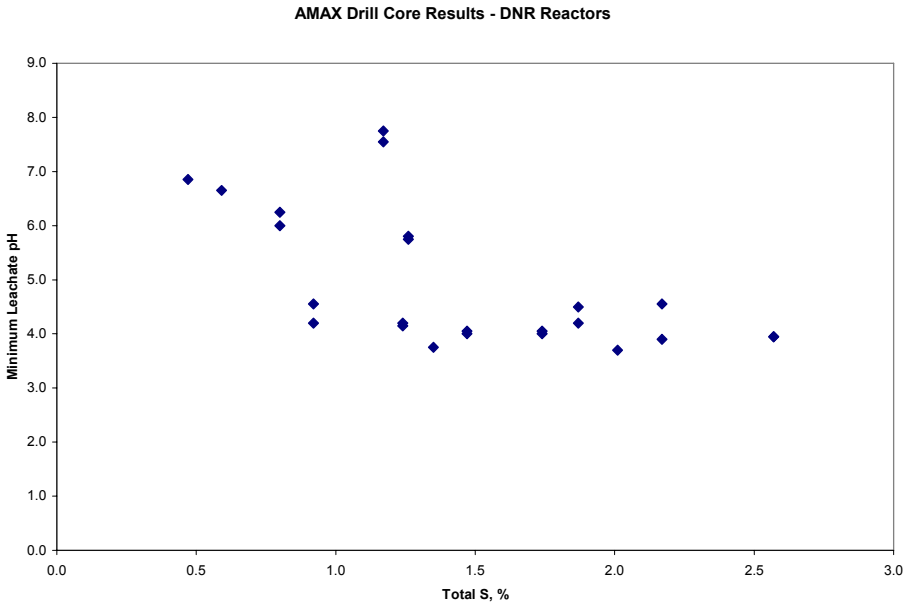


Figure 3-3: Minimum pH of AMAX Samples in MDNR Reactors as a Function of Sulfur Content.

3.2.3 AMAX Particle Size Experiment

The MDNR evaluated weathering of six particle sizes (from -0.75+0.25" to -270 mesh) of rock obtained from the pile containing 0.79% sulfur (Lapakko 1994b). SRK received pH data as a chart for 10 years.

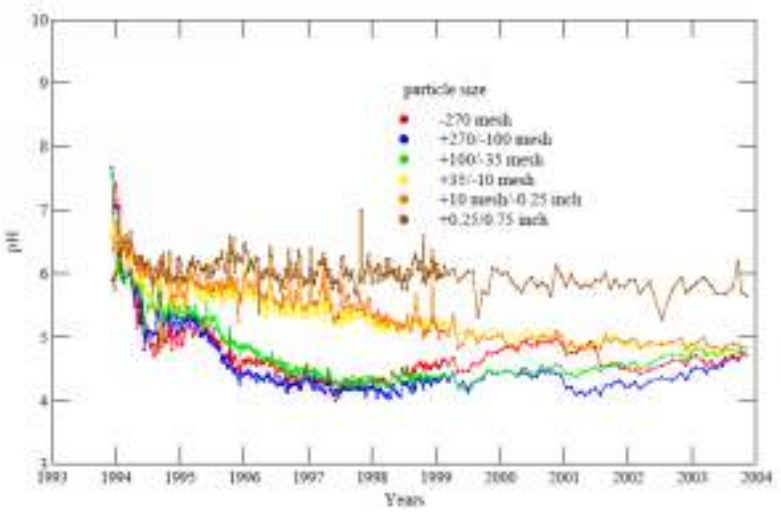


Figure 3-4: pH of Leachates from Particle Size Experiment on 0.79% Sulfur Rock from AMAX test shaft. Source: MDNR.

The coarsest size fraction generated near neutral pH leachate throughout the experiment. For the next two fractions (-0.25"+10 mesh and -10+35 mesh), pH gradually declined to just below 5 in ten years. The three finest fractions showed parallel trends with pH decreasing to near 4 for three to four years than gradually recovering to near 5.

3.2.4 Dunka Pit Stockpiles

Beginning in the 1960's, Duluth Complex rock was removed to access underlying iron formation at the Dunka Pit (MDNR 1994). Eight stockpiles varying in quantity from 0.1 to 21 million tons were constructed of which five contained mixed iron formation and Duluth Complex and three contained mainly Duluth Complex rock. Sulfur, nickel and copper concentrations were determined on the rock. Monitoring of stockpile drainage water quality is required as part of the NPDES permit for the site. Treatment of the drainage from the stockpiles using wetlands has been investigated (Eger and Lapakko 1980, 1988; Eger et al. 2000).

While monitoring of the site has produced a detailed long term record of stockpile drainage chemistry (MDNR 1994, 1996), the uncertain composition of the rock (including the possible presence of reactive components of the Virginia Formation beneath the Duluth Complex) has been confirmed by the MDNR (Eger 2006) and uncertain drainage pathways led to uncertainty about the relevance of the data to understanding weathering and leaching of Duluth Complex rock. The drainage chemistry for the stockpiles was therefore not considered to be a reliable analogue for stockpiles proposed at NorthMet.

3.2.5 Dunka Blast Holes Samples

In 1989 and 1990, MDNR began testing of 20 samples of Duluth Complex and Virginia Formation rock obtained from blast holes at the Dunka Pit in MDNR-style reactors (Table 3-2) (Lapakko 1988a,b; 1993a) for comparison with drainage from the site stockpiles. All but one sample was initially tested in duplicate. As of October 2006, three tests containing 0.18%, 0.22% and 0.82% sulfur had been stopped. None of the duplicate tests are continuing. The database of leachate chemistry was provided by MDNR (Engstrom 2006a, c). Leachates were analyzed frequently for pH and specific conductivity and sporadically for alkalinity, acidity, sulfate, silicon, calcium, magnesium, sodium, potassium, iron, aluminum, cobalt, copper, nickel and zinc. Leachate recovery from the cells was also recorded though not every week. Graphs illustrating the data are provided in Appendix B.1.

Table 3-2: Dunka Pit Samples Tested in MDNR-Style Reactors by MDNR

Sample Number	% S	Reactor #	Date of operation		Weeks as of 10/24/06
			Begin	End	
1519	0.18	1	2/14/89	---	923
		2	2/14/89	5/30/95	328
1517	0.22	3	2/14/89	---	923
		4	2/14/89	5/30/95	328
1515	0.40	5	2/14/89	7/18/06	909
		6	2/14/89	5/30/95	328
1503	0.41	7	2/14/89	12/31/02	724
		8	2/14/89	5/30/95	328
1520	0.51	9	2/14/89	12/31/02	724
		10	2/14/89	5/30/95	328
1507	0.54	11	2/14/89	7/29/97	441
		12	2/14/89	5/30/95	328
1522	0.57	13	2/14/89	7/29/97	441
		14	2/14/89	5/30/95	328
1518	0.58	15	2/14/89	12/31/02	724
		16	2/14/89	5/30/95	328
1521	0.71	17	2/14/89	12/31/02	724
		18	2/14/89	5/30/95	328
1537	0.82	43	8/12/97	---	480
1532	1.12	35	9/4/90	7/29/97	360
		36	9/4/90	5/30/95	247
1542	1.16	29	9/4/90	12/31/02	643
		30	9/4/90	5/30/95	247
1543	1.40	37	9/4/90	7/29/97	360
		38	9/4/90	5/30/95	247
1545	1.44	33	9/4/90	12/31/02	643
		34	9/4/90	5/30/95	247
1508	1.63	19	2/14/89	12/31/02	724
		20	2/14/89	8/29/94	289
1531	1.64	31	9/4/90	7/29/97	360
		32	9/4/90	5/30/95	247
1506	2.06	21	2/14/89	8/14/90	78
		22	2/14/89	8/14/90	78
1514	3.12	23	2/14/89	8/14/90	78
		24	2/14/89	8/14/90	78
1509	3.72(VF)	25	2/14/89	8/14/90	78
		26	2/14/89	8/14/90	78
1513	5.44(VF)	27	2/14/89	8/14/90	78
		28	2/14/89	8/14/90	78
Notes: Table provided by MDNR.					

The tests have shown that:

- Samples with 0.18% and 0.22% sulfur have not produced acidic leachate after 18 years. After declining for about 3 years from initial levels between 7 and 8, pH remained between 6 and 7 with no obvious upward or downward trend. At the same, sulfate concentrations declined. Calcium and magnesium concentrations also declined as pH declined then stabilized. Alkalinity declined during the first two or three years and became undetectable (<2 mg $CaCO_3$/L) in 1998. Nickel concentrations were erratic at first reaching concentrations of up to 0.07 mg/L then declined over the long term. The trend in copper concentrations was similar.
- A sample with 0.4% sulfur (Reactor 5) showed a slow decline in pH then a sharp decrease after about 14 years of weathering. The lowest pH reached was 3.8 in August 2004, after which pH recovered to 5 by June 2006. The trend in sulfate was downward to near 1 mg/L prior to the decrease in pH which resulted in a peak sulfate concentration of 37 mg/L coincident with the lowest pH. This was followed by a steep decline in sulfate concentrations to 5.8 mg/L. Calcium showed a similar peak along with aluminum and iron. Magnesium showed a subdued peak. Nickel and copper concentrations increased as pH declined reaching maximum concentrations of 0.3 mg/L and 0.4 mg/L, respectively. Peaks in metal concentrations were coincident with the pH minimum and sulfate maximum.
- Samples with 0.41% to 0.58% sulfur showed decreasing pH (to about 4) for 7 years but then increasing pH for the following 7 years. The decline in pH was also matched by increasing sulfate (peak concentrations less than 30 mg/L), calcium (though not all tests), magnesium, nickel, cobalt and copper. A distinctive feature of nickel release was the presence of double concentration peaks for long term reactors 7 (0.41% S), 9 (0.51% S) and 13 (0.57% S) and possibly also reactor 15 (0.58% S). Reactor 11 (0.54% S) did not show the same feature though there were wide gaps between analysis events and the peaks may not have been detected by the sampling schedule. The first peaks were greatest and coincident with an initial decrease in pH below 6, though sulfate release was declining at the same time. The second peaks were approximately coincident with a second sharp drop in pH below 5 which occurred at different times for different tests between 1992 and 1995. These nickel leaching peaks were coincident with sulfate peaks. Cobalt leaching showed similar trends. Copper leaching also showed double peaks, or a rise in copper leaching followed by a plateau. Where two peaks were observed, the second peaks were higher than the first peaks.
- Samples with 0.71 to 1.64% sulfur showed quickly decreasing pH (to between 3 and 4) for 5 to 7 years then pH recovery. The trend for decreasing pH was similar to the previous lower sulfur set. pHs declined rapidly then flattened between 4 and 5 before declining rapidly to the low point. This pattern appeared to be repeated during pH recovery with a short term flattening and/or decrease in pH as part of the general increase. The decrease in pH again resulted in increasing sulfate concentrations though not until pH dropped below 5. The greatest acceleration in sulfate increase occurred as pH dropped below 4. As pH recovered, sulfate concentrations dropped rapidly except for a slight flattening or increase coincident with the short term decrease

in pH noted above. Calcium and magnesium followed the same trend as sulfate. The double peak effect for nickel release was also observed though the difference in peaks varied. Reactors 20 (1.63% S), 31 (1.64% S) and 35 (1.12% S) had higher first peaks. Reactors 33 (1.44% S) and 37 (1.4% S) had higher second peaks. These two samples had the lowest pH minimums. Following the second peak nickel concentrations decreased to below 0.01 mg/L. The cobalt trends were similar to nickel but copper showed three peaks. The first minor peak was associated with the initial drop in pH. This was followed by the largest peak when pH were lowest, and a second minor peak that appeared to be associated with the pH flattening as part of the overall pH increase. Except for Reactor 33 (1.44% S), copper concentrations eventually decreased to less than 0.3 mg/L. Reactor 33 continued to have concentrations between 0.7 and 1.1 mg/L. This reactor showed the slowest pH recovery remaining below 5 until it was terminated. It also had the highest stable long term sulfate concentrations. Zinc concentrations showed some elements of the nickel and copper trends.

- MDNR also tested two Duluth complex samples that contained 2.06% and 3.12% sulfur and two Virginia Formation samples containing 3.7 and 5.4% sulfur. The tests proceeded for less than 2 years. All tests showed a rapid drop in pH to between 4 and 5 in a few weeks for the Virginia Formation samples, and a few months for the Duluth Complex samples. Declining pHs continued and when the test was stopped, the Virginia Formation sample with the highest sulfur content had a pH of 3.1. At the same time sulfate concentrations rose. Nickel and cobalt concentrations peaked early in the test period for the Duluth complex sample containing 2.1% sulfur when pH initially dropped but then declined despite the ongoing decrease in pH. Copper in the same sample peaked slightly as pH dropped at first but then showed a large peak as pH decreased. When the test was stopped, copper concentrations appeared to be decreasing.

SRK further analyzed the data provided by MDNR to evaluate the trend in sulfur depletion from the tests. Sulfur depletion was calculated by interpolating sulfate release and leachate recovered between results. The proportion of sulfur depleted varied from 4% to 101% though only one sample showed more than 84% depletion. Low depletions were indicated for tests that ran for a few years. The anomalous 101% value was for Reactor 5 (0.4% sulfur). The result was unexpected because the sample continued to leach sulfate despite the apparent depletion of more than the available sulfur. Because the calculation indicated that more than the original sulfur content was depleted, sulfur depletion was also calculated by correlating electrical conductivity with sulfate and using the resulting calculated sulfate values to calculate depletion. The calculation indicated that 107% of the original sulfur content had been depleted, confirming the original calculation. These calculations show that the original sulfur value was probably incorrect.

Figure 3-5 compares pH of leachates with the proportion of original sulfur consumed as a means of tracking how much sulfur was consumed as pH decreased and then recovered. It is apparent that for all but Reactor 5, the decrease in pH occurred before 40% of sulfur had been consumed and that pH recovered before 65% of sulfur was consumed.

Reactors 1 and 3 contained low sulfur concentrations oxidizing at low rates. However, 53% of sulfur has been removed from Reactor 1 (0.18% S) and 42% of sulfur has been removed from Reactor 3 (0.22% S). By comparison with the other tests, these samples would no longer be expected to generate acidic leachate at any time because insufficient sulfur remains. The absolute amount of sulfur remaining in these samples is 0.09% and 0.1% but no other sample showed a decrease in pH with this level of sulfur remaining.

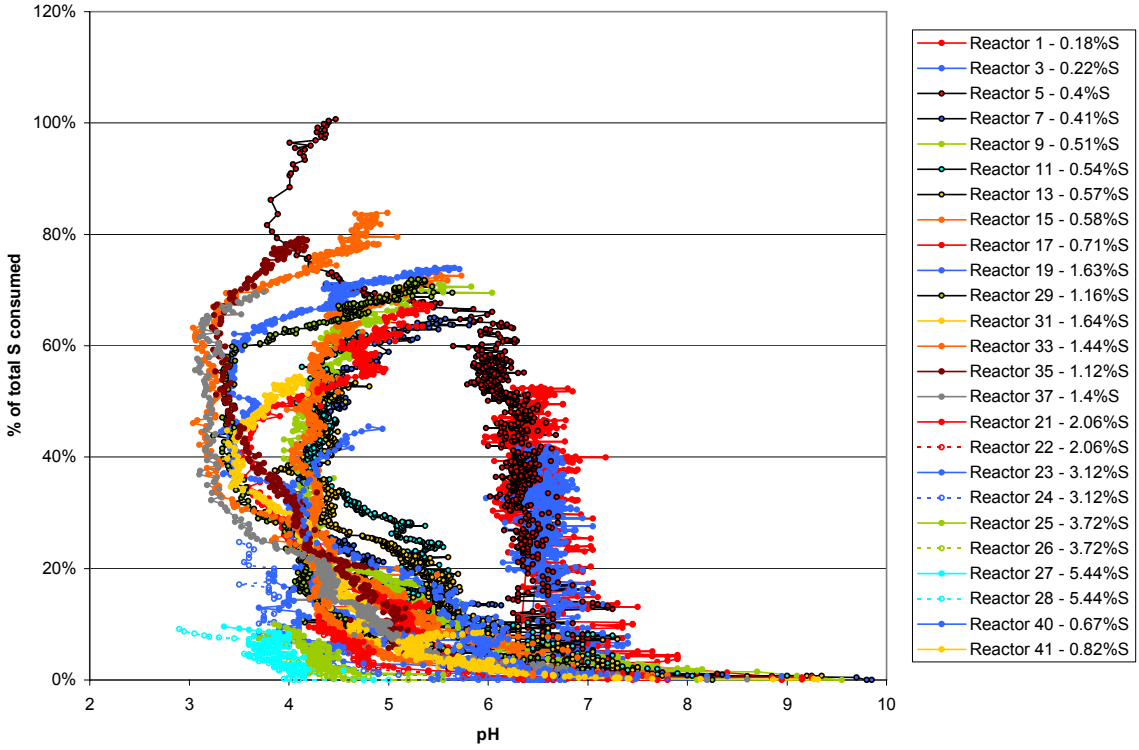


Figure 3-5: pH of Leachates Compared to Cumulative Depletion of Sulfur.

3.2.6 Babbitt and Dunka Road (NorthMet) Deposit Samples

In August 2003, MDNR began laboratory testing of ten Duluth Complex samples from the Babbitt (seven samples) and Dunka Road (three samples) deposits. Lapakko and Antonson (2006) reported data for 130 weeks of testing and the MDNR provided SRK with 170 weeks of data (Folman 2006a, b).

Sulfur and total inorganic carbon content of the samples tested are shown in Table 3-3. No other chemical analyses were available due to funding limitations (Lapakko, personal communication).

Leachates were analyzed for pH, conductivity, alkalinity, sulfate, calcium, magnesium, potassium, sodium, copper, nickel, cobalt and zinc.

Table 3-3: Characteristics of MDNR Samples from the Babbitt and Dunka Road (NorthMet) Deposits

Deposit	MDNR Reactor	%S	$\%CO_2$
Babbitt	1	0.07	0.08
Babbitt	2	0.11	0.13
Babbitt	3	0.2	0.12
Babbitt	4	0.31	0.2
Babbitt	5 (dup. of 4)	0.31	0.2
Babbitt	6	0.63	0.07
Babbitt	No MDNR Reactor		
Babbitt	7	0.94	0.08
Dunka Road	No test	No test	No test
Dunka Road	8 (D)	0.31	0.211
Dunka Road	No test	No test	No test

Deposit	ASTM Humidity Cell	%S	$\%CO_2$
Babbitt	9	0.13	0.05
Babbitt	10	0.21	0.08
Babbitt	11	0.33	0.08
Babbitt	12	0.55	0.08
Babbitt	13 (dup. of 12)	0.55	0.08
Babbitt	14	0.72	0.06
Babbitt	15	1.03	0.1
Babbitt	16	1.36	0.08
Dunka Road	17 (D)	0.23	0.07
Dunka Road	18 (D)	0.31	0.211
Dunka Road	19 (D)	0.61	0.161

Babbit Deposit

The Babbitt samples were from a bulk sample collected in 2001. The rock was processed to produce suitable charge material for MDNR Reactors (-100+270 mesh) and ASTM Humidity Cells (-1/4"). Six samples were tested in MDNR style reactors with one test performed in duplicate. An additional -1/4" sample was also tested. The sulfur contents of the two fractions are not the same because they were prepared by screening rather than crushing. As shown in Table 3-3, the coarser samples used in the ASTM tests had higher sulfur contents than the finer samples used for the MDNR reactor.

Leachate results are graphed in Appendix B.2.

The overall pH trend for the samples was downward to varying degrees with the degree of pH depression observed being related to sulfur content. Three tests produced leachate with pH below 5 (ASTM tests containing 1.03 and 1.36% sulfur, and MDNR Reactor containing 0.94% S). The 0.63% S sulfur sample in a MDNR reactor generated leachates mostly with pH below 5.5. For samples containing lower concentrations of sulfur, pH depression occurred sooner and to a greater degree in the MDNR reactors. The range of pH readings for the MDNR reactors at about 30 weeks was 5.9 to 6.6 compared to 6.7 to 7.3 for the humidity cells. The difference between tests was smallest for the samples containing the lowest sulfur concentrations. As the test proceeded, pH appeared to recover for the MDNR reactors containing the lowest sulfur concentrations but remained low for samples containing more than 0.2% sulfur. The pattern of sulfate release was also different for the two types of tests. The decrease in pH observed for the MDNR reactors coincided with sulfate release peak for all samples with the exception of the reactor containing 0.07% sulfur. The magnitude of the peak was correlated with sulfur content of the sample. In contrast, even the two ASTM cells which produced acidic leachate showed only weak increases in sulfate production as pH dropped. Sulfate generation for the ASTM humidity cells was generally uniform or decreasing. The pattern of calcium and magnesium release was comparable to sulfate for both types of test.

Nickel release appeared to be related to pH of the leachate. The MDNR reactors containing more than 0.07% sulfur indicated rapid increases in nickel release as pH dropped. The effect was weakest for the sample containing 0.11% sulfur. This was followed by slowly declining nickel release. At weeks 118 and 122, nickel concentrations decreased very sharply (by two orders-of-magnitude) and then recovered by three orders-of-magnitude. The dataset beyond 120 weeks seems suspect and has not been considered further. The MDNR is evaluating the data to determine if they are erroneous.

A similar overall nickel release trend was shown by the ASTM cells containing 1.03% and 1.36% sulfur though nickel release increased more rapidly at first. The other ASTM cells and MDNR reactor containing 0.07% sulfur showed slowly increasing nickel release. The relative degree of release was correlated with sulfur content. The sample containing the greatest sulfur content in this group (0.72%) showed the greatest nickel release (0.4 mg/kg/week). The highest nickel release for the sample containing 0.13% sulfur was 0.01 mg/kg/week.

Release of copper, nickel and zinc showed similar trends.

Dunka Road (NorthMet) Deposit

The Dunka Road samples were obtained from PolyMet's drill core. Three samples were tested in humidity cells while one sample was tested in a MDNR reactor. The data were provided to SRK (Folman 2006a, b). The MDNR reactor and ASTM humidity cell samples contained the same sulfur concentration (0.31%). The sulfur content of the other samples was 0.23% and 0.61%.

Leachate chemistry data charts are provided in Appendix B.3.

The pH trend for the samples containing more than 0.23% sulfur was generally downward at first with the MDNR reactor sample showing the greatest degree of depression, but leachate pH subsequently recovered to the same level as the equivalent humidity cell. The sample containing 0.23% sulfur showed an initial small decrease in pH, but leachate pH remained generally near 7. The differences in pH resulted in differences in sulfate release which were comparable to the Babbitt tests. The MDNR reactor showed the highest sulfate release which then declined whereas the three humidity cells showed relatively stable sulfate generation after 40 weeks. The level of stable sulfate generation was correlated with sulfur content of the sample.

Calcium and magnesium showed similar trends.

The nickel trends were also similar to the Babbitt samples. The MDNR reactor showed an initial increase in nickel as pH decreased, then nickel release trended downward. The two humidity cells containing higher sulfur content showed slowly increasing nickel release whereas the sample containing 0.23% sulfur showed nickel release below the detection limit. Cobalt release showed the same trend as nickel though the early trend was obscured by sub-detection limit results. Similarly copper release trends were affected by the detection limit of 0.0005 mg/L. The MDNR reactor showed a brief peak at 0.02 mg/kg/week then copper release dropped back to near the detection limit. Only the 0.61% S sample indicated significant detection of copper leaching after about 100 weeks. This contrasted with the Babbitt samples which generally appeared to release copper at a higher rate.

3.3 Data Requirements

As indicated in Section 1.2, the objectives of characterization of waste rock were:

- Refinement of preliminary waste rock management criteria developed by PolyMet and MDNR.
- Development of water chemistry predictions for stockpile drainage water for input to water impact assessment and water treatment design.

The MDNR's testwork has shown that sulfur content is the primary variable controlling pH of leachate, oxidation rate and metal release rates. These variables are not independent but sulfur content at least appears to be the underlying control. As a result, the MDNR and PolyMet agreed on initial waste classification criteria for "reactive" and "non-reactive" waste rock as defined in the Minnesota Rules. "Non-reactive" waste rock was defined as rock with less than or equal to 0.05% sulfur. This low level of sulfur was considered to eliminate the possibility of acidic drainage (based on MDNR's findings of lack of acidity from samples containing 0.22% sulfur) and also reduce the likelihood of significant metal leaching. The "reactive" category was expected to include rock that may or may not generate ARD but regardless would leach metals at a level that would not meet discharge water quality standards.

In addition to sulfur content, the MDNR and PolyMet also discussed other potentially important variables that could affect the chemistry of drainage from stockpiles:

- Sulfide mineral type.
- Rock type.
- Fragment particle size.

Other important variables include mineral content, mineral grain size, mineral chemistry, and mode of mineral occurrence.

Sulfide mineral type can be important in terms of rate of reaction and metal release. For example, it is well know that pyrrhotite is more reactive than pyrite. Chalcopyrite and pentlandite are sources of copper and nickel, respectively in drainage. However, because the dominant sulfide mineral in the waste rock appeared to be pyrrhotite (based on distribution of metal content) and the commodity sulfide minerals (chalcopyrite, pentlandite and cubanite) were expected to be present at low concentrations, sulfide mineral type was not considered as a primary variable for sample selection. As described below, concentrations of copper, nickel, cobalt and zinc were used as secondary factors for sample selection which is expected to capture variations in sulfide mineralogy. Lean ore characterization is considered separately. All samples were characterized to evaluate assumptions about the mineralogical occurrence of the important metals.

All rock types in the Duluth Complex are variants of troctolite and to a lesser extent ultramafic rocks. Carbonate minerals are absent or occur at very low concentrations in this rock type.

Therefore, the variation in silicate content of these rocks was considered to be an important variable controlling drainage pH.

MDNR also considered that igneous layers in the intrusive complex may be a significant variable because the reactivity of the minerals may be different in each of the layers. Finally fragment particle size is an important factor because it controls exposure of the reactive minerals and the overall surface area available for reaction.

The objective of refinement of the initial waste classification criteria was therefore addressed by evaluating numerous variables that could conceivably affect water chemistry. The data necessary for water chemistry predictions was also obtained by the same program.

4 Sampling and Analytical Methods

4.1 Exploration Drilling and Chemical Analysis Database

The drilling and sampling methods used to accumulate the database of bulk rock characteristics are described in ER03 (PolyMet 2007b). The following headings were included in the outline for the RS42 report but are covered in ER03 and RS78:

- Historical (Pre-PolyMet) Data Collection Methods
- PolyMet Data Collection Methods
- Database QA
- Block Modeling
- Model Limitations

4.2 Dissolution Testwork

4.2.1 Sample Selection

Waste Rock

A sampling matrix for waste rock characterization (Table 4-1) was developed by MDNR, SRK and PolyMet through a series of discussions and exchange of relevant data. Table 4-1 shows how the main variables were translated to a sampling design. The table also provides estimates of the tonnages of each major rock type within each unit. Reading from left to right, the columns in the table show the following:

- Unit. This refers to the stratigraphic igneous layers in the complex (numbered 1 to 7). Unit 20 refers to the footwall of the deposit composed of Virginia Formation and localized igneous intrusions
- Rock Type. This refers to a generalized rock description in the associated unit.
- Estimated Rock Tonnages. These tonnages indicate the estimated amounts of each rock type within each layer and therefore their relative importance. The provisional categories were developed by MDNR and PolyMet to indicate rock with sulfur less than 0.05%[1] ("non-reactive"), sulfur greater than 0.05% but not likely ore grade ("reactive"), and rock with marginal ore grade (lean ore).

[1] Note that the non-reactive classification was a temporary criterion agreed between MDNR and PolyMet.

Selection of samples for each unit and rock type combination was based on sulfur concentrations in order to develop correlations between reactivity and bulk characteristics such as sulfur and metal content. This provides a basis for water chemistry predictions using bulk characteristics, prediction of waste management criteria (based on sulfur content and metal content) and ultimately for the selection of easily-measured parameters that can be used for waste management during mining.

Separate sulfur ranges were defined for the "non-reactive" and "reactive" categories. Within the non-reactive category, three sulfur concentrations were selected to represent the lowest possible sulfur concentration in the rock type (typically 0.01%), the upper limit to this category (0.05%) and an intermediate (0.03%). The need for samples was identified for relatively abundant rock types contributing more than 1,000,000 tons (i.e. more than 1% of the rock mass). For the reactive category, sample selections were based on sulfur concentration percentiles calculated by Polymet. Again, rock types contributing more than 1,000,000 tons were identified for testing.

Sulfide mineral variability was considered by preferring samples with higher concentrations of Ni, Co, Cu and Zn.

The search for suitable samples included all candidate sulfur values indicated in grey shading (Table 4-1).

Characterization of Lean Ore

Lean ore is defined as rock containing grades of commodity minerals below that at which processing can currently be justified, but may eventually be processed if project economics improve. In terms of sulfur content, lean ore mainly overlaps the "reactive" waste rock category and also to some degree the "non-reactive" category but contains higher nickel and copper concentrations than waste rock. Therefore, the main difference between lean ore and waste rock is expected to be in the mineralogical occurrence of sulfur. In waste rock, sulfur occurs mainly as iron sulfide but in lean ore the commodity minerals pentlandite, chalcopyrite and cubanite are expected to be more important. This has important implications for drainage chemistry. In particular, oxidation and leaching of pentlandite is expected to release more nickel than pyrrhotite due to the higher Ni/Fe ratio in pentlandite. This limits the co-precipitation of nickel with iron oxyhydroxides during oxidation. The overall approach to selection of samples was similar to that of waste rock.

Characterization of Ore

Three ore composites (referred to as "Parcels") were prepared for Pilot Plant testing described in ER03 (PolyMet 2007b). These samples were used to characterize the leaching performance of ore.

Interval Selection, Sample Shipping and Preparation

Details of these aspects of the project are provided in Appendix A.

Table 4-1: Matrix for Sample Selection in Waste Rock Types

Unit	Rock Type	Non-Reactive	Reactive	Approximate sulfur contents																			Reactive Rock
				Non-reactive[1]			Reactive[1]									Lean Ore[1]							P size
		M. tons	M. tons	NR1	NR2	NR3	P10	P25	P50	P75	P80	P85	P90	P95	P100	P10	P25	P50	P75	P80	P85	P90	P95	
1	Anorthositic	0.57	0.99				0.08	***0.1***	***0.15***	***0.29***				***1.09***	1.09	0.11	0.26	***0.36***	***0.93***				***1.95***	***4***
1	Gabbroic	0	0.68				0.06	0.07	0.07	0.08				0.19	0.5	0.09	0.1	0.18	0.37				0.5	
1	Sedimentary hornfels	0	1.6				***0.08***	***0.35***	***0.69***	***2.2***	2.32	***2.81***	3.38	3.5	3.78	0.34	1.37	***1.58***	1.76				***4.91***	
1	Troctolitic	17.2	40.1	***0.01***	***0.03***	***0.05***	0.06	**0.07**	***0.1***	0.18	***0.21***	0.26	***0.34***	***0.62***	***1.97***	0.08	***0.15***	***0.24***	0.42	0.49	***0.55***	0.68	***0.98***	***4***
1	Ultramafic	0.21	1.1				0.07	***0.08***	0.1	0.13	***0.2****	***0.3****	***0.5****	***0.8****	1.35	0.07	0.09	***0.14***	***0.33***	0.55	***0.56***	0.65	***0.81***	
1	Vein	0.055	0.022				0	0	0	0				0										
2	Anorthositic	2.4	0.56	***0.01***	***0.03***	***0.05***	0.06	0.06	0.08	0.11				0.19		0.09	0.12	0.17	0.21				0.25	
2	Basalt inclusions	0.28	0													0	0	0	0				0	
2	Gabbroic	0	0.082				0	0	0	0				0										
2	Troctolitic	16.9	9.7	***0.01***	***0.03***	***0.05***	0.06	0.06	0.07	0.09	0.09	0.1	0.11	0.12	0.25	0.05	0.07	0.12	0.18	0.2	***0.22***	0.26	***0.32***	
2	Ultramafic	0.38	0.25	***0.03***	***0.04***	***0.05***	0	0	0	0				0		0.04	***0.05***	0.07	0.12	***0.14***	0.16	0.19	***0.23***	
3	Anorthositic	9.4	1.2	***0.01***	***0.03***	***0.05***	0.07	0.08	0.08	0.12				0.14		0.06	0.08	0.12	0.27				0.38	
3	Fault-Breccia	0	0.055				0	0	0	0				0										
3	Gabbroic	0.2	0.72				0.09	0.15	0.18	0.27				0.29		0	0	0	0				0	
3	Noritic	0.11	0																					
3	Sedimentary hornfels	0.38	0.46				0.12	1.42	1.67	1.97				2.22		1.66	1.77	1.85	2.43				3.26	
3	Troctolitic (augite)	41.2	12.5	***0.01***	***0.03***	***0.05***	0.06	0.06	***0.08***	0.1	0.11	***0.12***	0.14	***0.19***	0.36	0.05	0.13	0.19	***0.32***	0.35	0.45	0.48	***0.52***	***4***
3	Ultramafic	0.24	0.071				0	0	0	0				0		0	0	0	0				0	
4	Anorthositic	0.16	0.055				0	0	0	0				0										
4	Sedimentary hornfels	0	0.055				0	0	0	0				0										
4	Troctolitic	7	2.2	***0.01***	***0.03***	***0.05***	0.06	***0.07***	0.09	***0.18***	0.19	0.19	***0.48***	***0.92***	1.53	0.09	0.15	0.22	0.47				***1.52***	***4***
4	Vein	0.055	0																					
5	Troctolitic	2.5	2	***0.01***		***0.05***	0.07	0.09	0.11	0.16				0.22	0.22	0.12	0.17	0.26	0.37				***0.45***	
6	Chlorite	0.16	0													0	0	0	0				0	
6	Fault-Breccia	0.055	0.055				0	0	0	0				0										
6	Troctolitic	6.8	0.59	***0.02***	***0.04***		0	0	0	0				0		0.01	0.01	0.03	0.07				***0.19***	
6	Ultramafic	0	0.11				0	0	0	0				0										
7	Ultramafic	0.082	0													0	0	0	0				0	
20	Troctolitic	0	0.27				3.18	3.45	3.76	4.3				4.31										
20	Virginia	0	10.4				0.59	***1.25***	2.98	***4.15***	4.49	4.85	5.07	***6.06***	7.45	0	0	0	0				0	***4***

Notes:
1. Non-reactive rock categories (lower, medium and higher sulfur contents). The designation as "non-reactive" was provisional for the purpose of sampling design.
2. Sulfur percentiles for reactive rock types calculated by PolyMet are shown. "*" indicates approximate percentiles.
3. Grey – sampling plan.
4. Bold and italic – samples obtained.
5. Bold border – duplicate cell in operation.

4.2.2 Physical Characterization

The specific gravity and particle size distribution of all samples was determined. Particle size distributions were determined by screening with five sieves at ¼", 10 mesh, 35 mesh, 100 mesh and 270 mesh.

4.2.3 Mineralogy

Optical

Two pieces of typical core from each interval sampled were taken for preparation of polished thin sections to confirm the rock type and quantify reactive minerals. Optical mineralogy reports provided mineral types, mineral abundance, grain sizes and mineral occurrence. Results are provided in Appendix D.1. Photomicrographs are provided in Appendix D.2.

Sub-Optical (Microprobe)

Sub-optical analysis included determination of the trace element content of major minerals on selected samples using microprobe analyses.

Results are provided in Appendix D.3.

4.2.4 Analytical Methods

Solids Characterization

A split of each sample was submitted for an extensive suite of analysis, as follows:

- Total sulfur and carbonate. Sulfur as sulfate was not determined because previous work shows that sulfur occurs exclusively as sulfide. Carbonate rather than neutralization potential was determined because neutralization potential determinations on rocks containing reactive silicates are ambiguous (Lapakko 1994a) and do not reflect field capacity to neutralize acid. If carbonate is present, it indicates the known field reactive component of acid neutralization potential provided that occurs in non-ferrous mineral forms.
- 27 elements by ICP scan following four-acid (nitric-hydrochloric-perchloric-hydrofluoric) digestion (near total).
- 34 elements by ICP scan following aqua regia (nitiric-hydrochloric acid) digestion.
- Whole rock oxides.

These methods were selected to provide continuity with the earlier work.

Results are provided in Appendix D.4.

In addition, 200 g of five size fractions (-20 mesh, -100+270 mesh, -35+100 mesh, -10+35 mesh and -0.25”+10 mesh) were submitted for analysis of total S and 27 elements by four acid digestion.

Results are provided in Appendix D.5.

Kinetic Testing

ASTM Humidity Cell

Humidity cell testing was performed using ASTM Procedure D 5744 – 96 (Reapproved 2001) (ASTM 2001). This procedure was selected for the following reasons:

- Similar procedures have been in use under different names since the late 1980s (e.g. MEND 1991). The results can therefore be evaluated in the context of more than a decade of experience using the procedure.
- It is a standard procedure approved by the ASTM and is therefore defensible as a method (White and Lapakko 2000).

The ASTM procedure provides some options for varying the test procedure. Appendix A provides a detailed listing of the requirement of the ASTM procedure, options chosen and any variances from the ASTM procedure.

MS Excel ® spreadsheets of the results are provided on the compact disk included with the report. Calculated release rates are charted in Appendices E.1, E.2, E.3 and E.4.

Minnesota MDNR Cell and Particle Size Experiments

The MDNR’s research has shown that particle size is an important consideration for understanding the reactivity of waste rock (e.g. Lapakko 1987). To evaluate size fraction effects, four size fractions (-100 mesh, -35+100 mesh, -10+35 mesh and -0.25”+10 mesh) from five samples are being tested using a procedure referred to as the “MDNR Reactor” experiment. These experiments also allow comparison of the ASTM method with the MDNR reactor which has been used for much of the MDNR’s prior experimentation. Similar comparisons have been made by the MDNR (Lapakko and White 2000).

The two smallest size fractions were tested in an apparatus designed by MDNR to contain 75 g (Lapakko 1988b; Appendix A; SRK 2006). The two coarser fractions are being tested in cells with the same configuration as ASTM Procedure D 5744–96.

For the small reactors, a weekly leachate volume of 200 mL was used. For the larger samples, the leachate volume was 300 mL.

MS Excel ® spreadsheets of the results are provided on the compact disk included with the report. Calculated release rates are provided in Appendix F.

Leachate Analysis

Leachates from kinetic tests were analyzed for the parameters indicated in Table 4-2. Conductivity, pH and ORP were analyzed every week. Sulfate, alkalinity, acidity, chloride and fluoride were analyzed on even numbered weeks. Every four weeks beginning on the first rinsing cycle (week 0) the metals indicated in Table 4-2 were analyzed using an ICP-MS scan in filtered samples. On other even numbered weeks (i.e. weeks 2, 6, 10 etc.), the leachates were analyzed for a higher level scan (ICP-OES) to evaluate trends in major elements. This scan also provided trace metal concentrations but at higher detection limits.

Based on experience, testing of non-reactive rock samples with very low sulfur concentrations was expected to result in very dilute leachates containing low concentrations of the metals of interest. Back-calculation of metal concentrations from other testwork performed by MDNR indicated that nickel and cobalt concentrations could be as low as 0.0002 mg/L (200 ng/L) and 0.00001 (10 ng/L), respectively. Quantification of these low metal concentrations was needed to provide reasonably constrained estimates of metals concentrations in waste rock drainage.

A number of different approaches were considered to quantify low levels of metals:

- The routine leachate analysis achieved a detection level of 0.0001 mg/L (100 ng/L). A detection limits of 50 ng/L could be obtained with additional processing effort using the same routine method.
- Specialist methods can achieve lower detection limits. These are non-routine (for example, evaporation to increase concentrations) and would need to be developed as the need arises. In order to generate a 10 times decrease in detection limit, the samples would need to be concentrated at 1east 10 times. A composite leachate sample would be prepared from several cycles.
- The MDNR's testwork (Folman 2006a) demonstrated that good correlations exist between cobalt and nickel concentrations in leachates. Detectable nickel concentrations could be used to estimate cobalt concentrations if this relationship could be demonstrated.
- The particle size experiments provide a larger surface area and provide greater likelihood that lower concentrations will be detected.
- In the event of undetectable low levels, detection limit values could be used in subsequent calculations.

The mercury detection limit of 0.02 µg/L (20 ng/L) was supplemented for selected leachates by using a method to achieve a detection limit of 2 ng/L.

Table 4-2: List of Parameters for Humidity Cell Leachate Analyses. Concentrations in mg/L except where indicated

Parameter	Limit	Parameter	Limit
pH (standard units)	-	Acidity	1
Conductivity (µS/cm)	1	Alkalinity	1
Chloride	0.2	Sulfate	0.5
Fluoride	0.05	Total Inorganic Carbon	1
ORP (mV)	-		
Dissolved Elements			
Aluminum	0.001	Mercury	0.02 µg/L
Antimony	0.0001	Molybdenum	0.00005
Arsenic	0.0001	Nickel	0.0001 (0.00005)[1]
Barium	0.0001	Potassium	0.02
Beryllium	0.0002	Selenium	0.0002
Bismuth	0.0002	Silicon	0.05
Boron	0.005	Silver	0.00005
Cadmium	0.00004	Sodium	0.01
Calcium	0.01	Strontium	0.0001
Chromium	0.0002	Tellurium	0.0002
Cobalt	0.0001 (0.00005)[1]	Thallium	0.00002
Copper	0.0001	Thorium	0.0001
Iron	0.01	Tin	0.0001
Lead	0.00005	Titanium	0.0002
Lithium	0.0002	Uranium	0.00005
Magnesium	0.005	Vanadium	0.0002
Manganese	0.00005	Zinc	0.001

Notes:

1. Low detection limits are available for cobalt and nickel as shown.

QA/QC

To summarize, QA/QC included the following components:

- Roughly 10% of all solids analyses were performed in duplicate.
- Roughly 10% of all cell and reactor tests were run as duplicates.
- A blank cell and reactor containing no sample were operated to check for contamination of leachates by construction materials.
- Individual leachate results were reviewed.
- Ion balances on leachate results were reviewed. In general, imbalances of ±10% were considered acceptable. Re-analysis if requested depending on the nature of the imbalance.
- Data trends in kinetic test leachates were analyzed to check for anomalies.

4.2.5 Interpretation Methods for Kinetic Tests

Trend Analysis

All results from kinetic plots were plotted as time series which were continually updated as the waste characterization program progressed to allow trends to be assessed. Results were plotted as raw concentrations and as loadings, or release rates calculated from:

Loading (mg/kg/week) = Concentration (mg/L) x Leachate Recovered (L) / Mass of Sample (kg)

As indicated above, metal concentrations were determined by two different methods on alternate even-numbered weeks. For the purpose of plotting and loading calculations, the following rules were used:

- If the result was determined by ICP-MS and was below the reporting limit, the value on the graph is at the reporting limit. If the value is at or above the reporting limit, the value is plotted.
- If the result was determined by ICP-ES and was determined to be below the reporting limit, no value is plotted.
- If the result was determined by ICP-ES and was determined to be above the reporting limit, the value is plotted.

These rules can result in four cycles between plotted results if the parameter is not detected by ICP-ES.

Occasionally, "saw tooth" trends are apparent in which values alternate between high and low for the ICP-ES and ICP-MS analyses. This results from analytical "noise" around the ICP-ES reporting limit when reported values are slightly above the reporting limit. Aluminum is a particular example that commonly shows reported values above the ICP-ES reporting limit of 0.05 mg/L.

Many graphs are plotted on logarithmic axes to allow data spanning a wide range of concentrations to be compared.

Average Rate Calculations

Average rates (in mg/kg/week) were calculated to evaluate correlations between bulk characteristics (e.g. metal and sulfur content, mineralogical characteristics). The following method was used to calculate average rates:

- The loading trends for sulfate were examined as an indicator of sulfide oxidation rates and the expected main factor driving other parameters such as release of metals and the products of acid neutralization.
- The loading trends typically showed relatively rapid initial release of sulfate followed by decrease, then a longer term trend (stable, increasing, or slow decrease). The initial trend is

usually a result of leaching of weathering products produced by oxidation of the sample in storage prior to testing. The trend following the short term effect reflects dissolution of weathering products produced each week. For trends showing relatively stable release, the trend was examined to find the first week when the release rate was below the highest point in the stable trend. If a decreasing or increasing trend, the trend was visually assessed to estimate when the initial flush ended. The release rates following the development of the stable trend are then used to calculate average release for the entire trend. In the event that the trend showed much more variability than other tests, the average was not calculated.

- Loading trends for other parameters were calculated using the same time period as sulfate so that comparisons between parameters could be made on a consistent basis.
- Some dissolved ions were not determined on a weekly basis, and in some cases have variable analytical frequency depending on detection by ICP-ES or ICP-MS. The average rates for individual weeks were pro-rated between analyses by summing the load leached rather than just averaging weekly rates.

Depletion Calculations

Rates were also used to evaluate depletion of rock components by totaling the load leached over the entire period of the test.

5 Results

5.1 Exploration Drilling and Chemical Analysis Database

The bulk rock characteristics are in ER03 (2007b). The following headings were included in the outline for the RS42 report but are covered in ER03:

- Unit Characteristics
- Rock Type Chemical Characteristics
- Metal Distribution
- Sulfur distribution
- Location of samples (spatial distribution within mine)

Appendix C.1 summarizes PolyMet's calculations of the distribution of sulfur, copper, nickel and zinc by rock type and unit.

5.2 Dissolution Testwork

5.2.1 Reconciliation of Sample Selections with Target Characteristics

Discussion of the sulfur composition of the 79 rock samples selected for kinetic testing compared to the target characteristics shown in Table 4-1 was provided in a memorandum dated November 24, 2005 to the MDNR.

Some deviation from the target sulfur concentrations shown in Table 4-1 was expected because (a) samples were selected using weighted average analyses in PolyMet's drill core sample database, (b) suitable samples were not always available; and (c) the actual analyses combine compositing and analytical errors for both the original database values and composite analytical results. Deviations from target values were calculated from:

$$Deviation(\%) = \frac{(S_{Actual} - S_{Target})}{S_{Target}} \times 100$$

The following summarizes resulting deviations compared to target sulfur concentrations.

- For samples with S≤0.05%:
  - Deviations of 100% are common but they usually represent absolute differences of less than or equal to 0.02% on sulfur concentrations below 0.05% and close to the method detection limit of 0.01%.
  - Two samples (8% of non-reactive samples) with target concentrations of 0.01% had deviations exceeding 100%.

- For samples with S>0.05%:
  - 84% of samples selected had deviations below 50%.
  - For four samples (13%), the deviation was between 50 and 100% and only one sample had a deviation of 200%. This latter sample was classified as "optional" in the selections.
- For lean ore samples, only one sample (less than 5% of samples) showed a deviation of 50%.

Based on the low overall incidence of deviation from target values and the large size of the dataset, it was concluded that the sample distribution for waste rock, lean ore and ore would provide a more than adequate database for characterization of these materials. No further sampling was proposed for these rock categories or deferred ore.

5.2.2 Mineralogical Characteristics of Samples

Appendix D.1 provides mineralogical data for samples in kinetic tests. Mineralogy is described below for each of the main sulfur groupings.

S≤0.05% Rock

The dominant silicate phases in all samples were either plagioclase or olivine. One sample of ultramafic (00-368C-460-465) showed alteration to chlorite (20%) and serpentine (30%). One sample of troctolite (00-367C-290-310) also showed alteration to chlorite (40%). The only other significant mineral group was pyroxenes which occurred in concentrations varying from 0 to 25%. Clinopyroxene was dominant.

Sulfide mineral content was described as not detected (0% sulfide), rare and trace, which was consistent with the low sulfur content of the samples. The sulfide mineral content was further described in terms of minerals and proportion of the overall sulfur content. Ten sulfide minerals were identified, but five were present at proportions exceeding 15% in at least some samples with the approximate order of importance of these minerals represented by:

Chalcopyrite > Pyrrhotite > Pentlandite > Cubanite > Bornite.

In a few samples, pyrrhotite exceeded chalcopyrite. Other sulfide minerals identified included digenite, covellite, violarite, mackinawite/valleriite and sphalerite. Pyrite was not recognized in any of the samples. The distribution of sulfide minerals in the non-reactive waste rock showed the importance of chalcopyrite rather than pyrrhotite as the main host for sulfur.

PolyMet have indicated from their mineralogical interpretation that approximately 95% to 98% of sulfide mineralization occurs interstitial to the silicate minerals. The remaining 2 to 5% of sulfide occur as inclusions in silicates, microveinlets in microfractures and secondary remobilized blebs in altered silicates.

S>0.05%

This waste rock group includes Duluth Complex igneous rocks, xenoliths and footwall Virginia Formation rocks.

With the exception of one sample (00-357C-335-340) of the Duluth Complex rocks, the non-sulfide mineralogy of the samples was dominated by plagioclase, followed primarily by olivine. As in the S<0.05% rocks, chlorite and serpentine alteration was present in some samples along with both orthopyroxene and clinopyroxene. Sample 00-357C-335-340 is an unusual example of highly to intensely altered rock, with possible contamination of partially assimilated footwall rocks. It is identified by PolyMet as an ultramafic rock from Unit 1, and is dominated by potassium feldspar, chlorite, clinopyroxene, and epidote.

Sulfide content in the Duluth Complex samples was dominated by similar minerals as the S<0.05% rocks, but pyrrhotite was the dominant sulfide, and bornite was a small proportion so that the overall order of importance was:

Pyrrhotite > Chalcopyrite > Pentlandite = Cubanite[2]

Other sulfide minerals identified were chalcocite, covellite, violarite, mackinawite/valleriite, sphalerite, galena, enargite and talnakhite.

The dominant non-sulfide minerals in the sedimentary hornfels xenoliths and Virginia Formation were potassium feldspar and cordierite. Other minerals included quartz, clinopyroxene, biotite, white mica, graphite, carbonate and idocrase. Carbonate and idocrase were identified in one sample of sedimentary hornfels (26030-1047-1052). The dominant sulfide mineral was pyrrhotite with mostly minor sphalerite, galena and chalcopyrite. One sample (26030-1047-1052) of sedimentary hornfels contained sphalerite as the dominant sulfide.

Lean Ore

The lean ore category consists mainly of Duluth Complex rock types with some xenoliths (sedimentary hornfels).

The dominant silicate minerals in the Duluth Complex samples were plagioclase and olivine, with the proportion shifting depending on the rock type. Other important silicates were orthopyroxene and clinopyroxene. Serpentine and chlorite alteration was apparent in several samples. One sample of anorthositic rock from unit 1 (00-331C-255-260) had epidote and carbonate alteration. This sample also contained no visible olivine.

The order of abundance of sulfide minerals was found to be:

[2] The “equals” sign means that cubanite and pentlandite occur in about equal quantities.

Chalcopyrite > Pyrrhotite > Pentlandite = Cubanite

The proportions of pentlandite and cubanite were variable with pentlandite sometimes exceeding cubanite and vice versa. The bornite content was variable reaching 10 to 15% of sulfide in some samples and exceeding cubanite. Other sulfide minerals included covellite, violarite, mackinawite/valleriite and sphalerite.

One sedimentary hornfels sample was dominated by clinopyroxene, potassium feldspar, white mica and pyrrhotite. Other sulfides (chalcopyrite, sphalerite and galena) were present in low quantities.

Analysis of Mineral Grains

Microprobe was used to estimate the concentrations of Fe, Cu, S, Ni, Co, Zn, Ti and As in 268 sulfide mineral grains and SiO_2, Al_2O_3, TiO_2, FeO, MnO, MgO, K_2O, CaO, Na_2O, NiO, CuO and CoO in 236 silicate mineral grains. The grains were from non-reactive and reactive waste rock, and lean ore.

Complete analytical results are provided in Appendix D.3.

Microprobe analyses of mineral grains indicated the average compositions of the two major minerals (plagioclase and olivine) in the Partridge River Intrusion. Plagioclase averages $An_{59}Ab_{41}$ (i.e. 59% anorthite and 41% albite) and olivine averages $Fo_{57}Fa_{43}$ (i.e. 57% forsterite and 43% fayalite).

Table 5-1 provides summary statistics for nickel concentrations in the major sulfide minerals. The detection limit was approximately 0.02%. The main nickel-bearing mineral was pentlandite. In terms of overall significance as a source of nickel, pyrrhotite was the next most significant (median 0.1%). Other minerals containing nickel as a major component were cobaltite and maucherite ($Ni_{11}As_8$) but both were rare minerals. Pyrite when detected contained more nickel than pyrrhotite but it was uncommon.

Table 5-2 shows nickel concentrations in silicates minerals. Concentrations expressed as nickel oxide were converted to nickel. The detection limit was 0.016% (0.02% NiO). These results show that olivine and biotite contained comparable levels of nickel followed by the pyroxenes. Plagioclase contained low levels of nickel.

Table 5-1: Nickel Concentrations (%) in Sulfide Minerals

	Bornite	Chalcopyrite	Cobaltite	Cubanite	Maucherite	Pentlandite	Pyrite	Pyrrhotite
n	3	75	3	41	3	70	8	64
Min.	0.021	0.021	11	0.02	46	26	0.02	0.02
P5	0.029	0.021	12	0.02	47	29	0.04	0.02
Median	0.106	0.022	18	0.02	52	32	0.44	0.10
Mean	0.085	0.057	16	0.94	50	32	0.51	0.22
P95	0.126	0.131	20	1.87	53	37	1.15	0.80
Max.	0.128	0.639	20	30.75	53	43	1.22	1.48

Table 5-2: Nickel Concentrations (%) in Silicate Minerals

	Olivine	Plagioclase	Clinopyroxene	Orthopyroxene	Biotite
n	61	71	57	20	71
Min.	0.016	0.016	0.016	0.016	0.016
P5	0.016	0.016	0.016	0.016	0.016
Median	0.082	0.016	0.016	0.016	0.058
Mean	0.077	0.017	0.023	0.022	0.057
P95	0.130	0.022	0.048	0.047	0.095
Max.	0.143	0.031	0.059	0.059	0.128

Table 5-3 shows copper concentrations in sulfide minerals. Chalcopyrite and cubanite are the main copper-bearing minerals. The copper content of pentlandite and pyrrhotite was low. Table 5-4 shows that in contrast to nickel, silicate minerals contain low copper concentrations with no apparent enrichment in any particular mineral.

Table 5-3: Copper Concentrations (%) in Sulfide Minerals

	Bornite	Chalcopyrite	Cobaltite	Cubanite	Maucherite	Pentlandite	Pyrite	Pyrrhotite
n	3	75	3	41	3	70	8	64
Min.	59	33	0.3	0	0.04	0.02	0.03	0.02
P5	59	33	0.8	22	0.04	0.03	0.04	0.02
Median	60	34	5.5	23	0.05	0.13	0.07	0.04
Mean	62	34	4.5	22	0.06	0.34	0.16	0.08
P95	67	35	7.4	23	0.07	1.58	0.45	0.35
Max.	68	35	7.7	34	0.08	4.38	0.53	0.42

Table 5-4: Copper Concentrations in Silicates

	Olivine	Plagioclase	Clinopyroxene	Orthopyroxene	Biotite
N	13	71	57	20	71
Min.	0.025	0.025	0.025	0.025	0.025
P5	0.025	0.025	0.025	0.025	0.025
Median	0.025	0.025	0.025	0.025	0.025
Mean	0.025	0.025	0.025	0.025	0.025
P95	0.025	0.025	0.027	0.025	0.025
Max.	0.028	0.039	0.038	0.036	0.037

Table 5-5 shows that pentlandite is the only major sulfide mineral enriched in cobalt. Cobaltite ((Co,Fe)AsS) is present but in low quantities. Cobalt is primarily present in olivine (Table 5-6).

Table 5-5: Cobalt Concentrations in Sulfide Minerals

	Bornite	Chalcopyrite	Cobaltite	Cubanite	Maucherite	Pentlandite	Pyrite	Pyrrhotite
N	3	75	3	41	3	70	8	64
Min.	0.015	0.019	21	0.02	0.4	0.6	0.04	0.03
P5	0.016	0.021	21	0.03	0.5	0.8	0.04	0.05
Median	0.023	0.034	22	0.04	1.4	1.8	0.05	0.06
Mean	0.023	0.038	22	0.11	1.2	2.3	0.06	0.06
P95	0.031	0.064	23	0.08	1.6	6.0	0.10	0.09
Max.	0.032	0.116	23	1.77	1.7	9.4	0.11	0.20

Table 5-6: Cobalt Concentrations in Silicates

	Olivine	Plagioclase	Clinopyroxene	Orthopyroxene	Biotite
n	13	71	57	20	71
Min.	0.021	0.021	0.021	0.021	0.021
P5	0.032	0.021	0.021	0.021	0.021
Median	0.051	0.021	0.021	0.022	0.021
Mean	0.050	0.021	0.022	0.027	0.024
P95	0.073	0.021	0.025	0.044	0.033
Max.	0.080	0.025	0.040	0.048	0.041

5.2.3 Particle Size Distribution

All samples considered for dissolution testing were crushed, and screened through four sieves to obtain four size fractions (+10 mesh, -10+35 mesh, -35+100 mesh, -100+270 mesh) which were analyzed. Results for selected parameters are shown in Figure 5-1.

All fractions showed similar results, with generally increasing concentrations into the fine fractions. For sulfur, the effect was most apparent for sulfur concentrations above about 0.1%. For nickel, the finer fractions clearly contained greater concentrations though again the effect was stronger for the samples containing higher nickel concentrations. Cobalt showed a similar effect. For copper, the majority of samples did not show clear enrichment in the fine fractions though a few samples with higher concentrations showed a wide spread of concentrations with the finest fraction containing more than 150% of the copper concentration of the finest fraction.

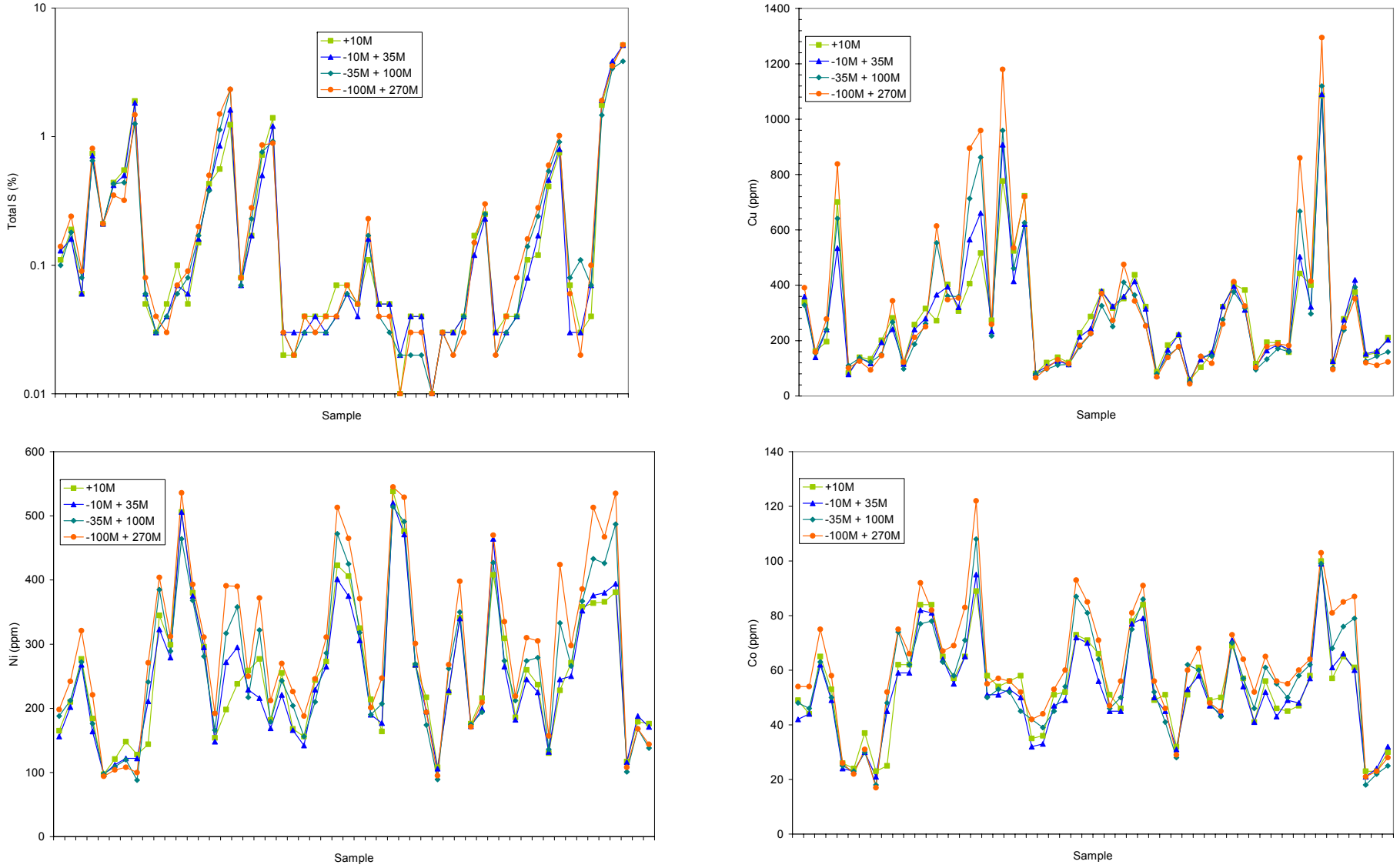


Figure 5-1: Results of Analysis of Size Fractions. x-axes are individual samples.

5.2.4 Description of Leachate Chemistry

Waste Rock Humidity Cells

Data Appendices

Trend graphs for loadings in each general group (waste rock S≤0.05%, waste rock S>0.05%, lean ore and ore) are provided in Appendices E.1, E.2, E.3 and E.4. An overall discussion of trends is provided in the following sections for each of these groups.

In the graphs, the labels indicate the drill hole (e.g. 26029), core interval in feet (815-825), intrusive layer in the Duluth Complex (e.g. U1 indicates Unit 1, U2 indicates Unit 2, etc.), rock type (e.g. TR – troctolitic; AN – Anorthositic; UM – Ultramafic; SH – Sedimentary Hornfels; VI – Virginia Formation) and sulfur content. For samples with S≤0.05%, the sulfur content is shown by the target sulfur level (0.01, 0.03, 0.05%). The waste rock and lean ore, the sulfur content is shown by the sulfur percentile (Table 4.1).

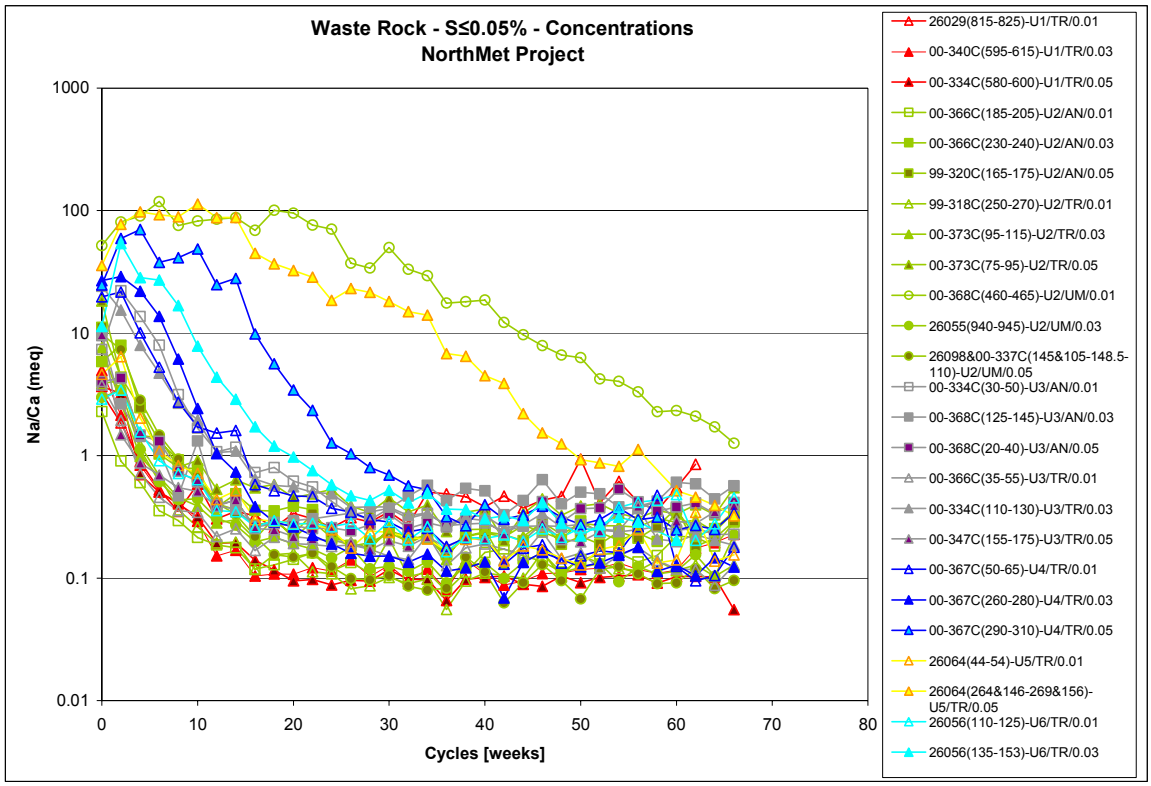
Data listings are provided on the CD attached to this report.

Waste Rock S≤0.05%

All tests showed a general decreasing pH trend starting mostly between 8 and 9.5 then decreasing to 6.7 to 7.5. Major ions in leachates were initially alkalinity and sodium. The trend in inorganic carbon in leachates was comparable to alkalinity which indicated that alkalinity was composed mainly of bicarbonate. Both alkalinity and sodium decreased as the tests proceeded, but the decrease in sodium was matched by increasing calcium. Calcium concentrations then decreased but became the major cation. Magnesium concentrations in some cases decreased steadily while in others increased and then decreased. Sulfate was less significant as an ion compared to alkalinity. It generally decreased than stabilized. No indications of increasing trends were apparent.

The ratios of Na/Ca and Mg/(Na+Ca) were examined because these relate to the leaching of major minerals in the rock (plagioclase and olivine). The trend for Na/Ca was sharply downward and then stabilizing for all but two tests (Figure 5-2). The ratio stabilized between about 0.1 and 0.9 (in meq terms). The ratio of Mg/(Na+Ca) increased sharply then stabilized between 0.1 and 0.5 indicating preferred leaching of plagioclase compared to olivine.

Trace element concentrations showed similar trends to the major ions but due to the low levels were commonly erratic. For example nickel concentrations were generally below 0.0015 mg/L and were flat or decreasing. Copper concentrations showed some erratic spikes but were generally below 0.004 mg/L and showed stable trends. Cobalt was mostly undetected at 0.0001 mg/L. Zinc concentrations were erratic between 0.001 and 0.01 mg/L and no trends were apparent.



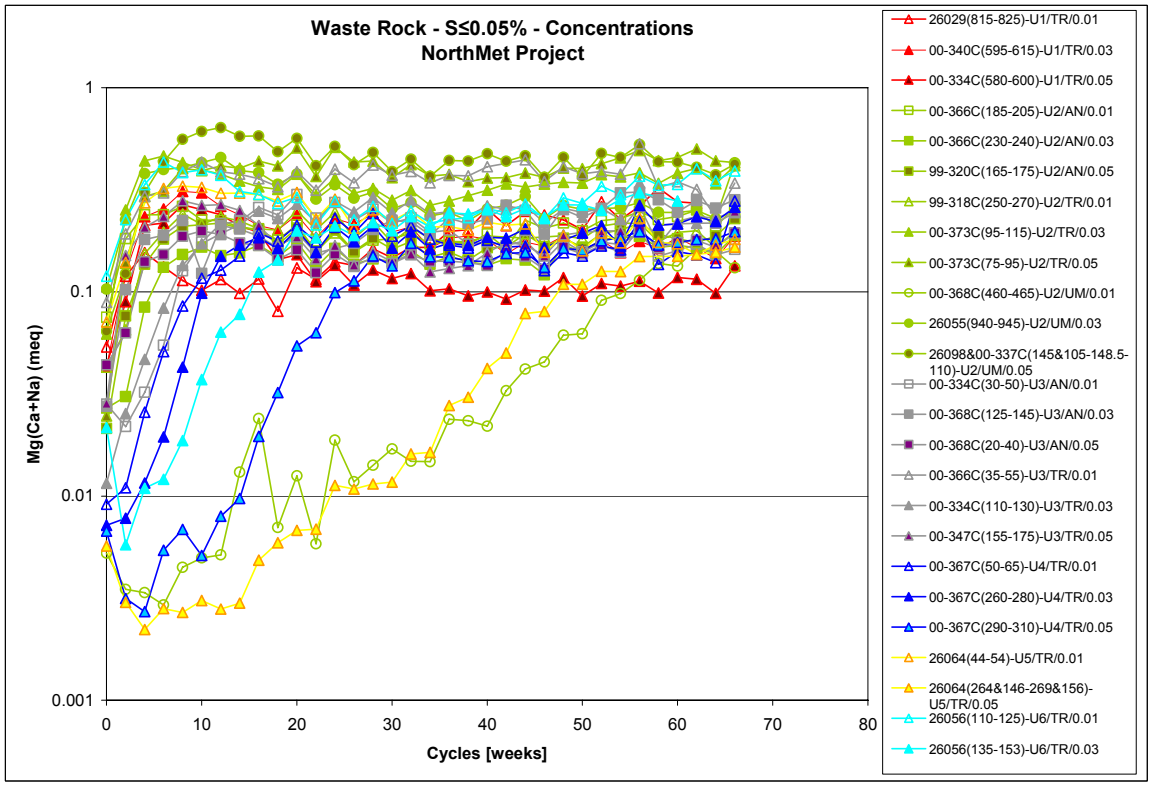


Figure 5-2: Na/Ca and Mg/(Na+Ca) Ratios for Waste Rock with S≤0.05%.

Waste rock S>0.05%

This class of waste rock includes Duluth Complex (Partridge River Intrusion) Rock, Virginia Formation in the footwall of the Complex and xenoliths of the footwall rocks included in the Complex. The latter are referred to as Sedimentary hornfels.

- **Duluth Complex Rock**. The sulfur concentration range for these samples was 0.05% to 1.7%. Leachate pH from the Duluth Complex rocks generally declined as the test proceeded, with initial pHs mostly above 8 and more recent results above 7. A few tests showed pHs between 6 and 7. These samples with slightly lower pHs were typically ultramafic rock types. Although pH decreased for a period, more recent results showed increases. Initial leachate chemistry was dominated by alkalinity and sodium with a gradual transition to comparable alkalinity and sulfate, balanced primarily by calcium. These changes mainly occurred as a result of decreasing alkalinity and sodium. Sulfate loadings were generally stable though some tests showed increasing sulfate to reach a higher stable rate. In some cases, sulfate increased then decreased (for example, troctolite from Unit 4 with 95th percentile sulfur concentrations). Potassium leaching occurred at comparable levels to sodium in the later stages of testing.

  Trends in Na/Ca and Mg/(Na+Ca) were similar to the S≤0.05% samples. Na/Ca decreased then stabilized for most samples at ratios between 0.03 and 0.6 (meq terms). Mg/(Na+Ca) generally increases than stabilized between 0.08 and 0.3.

  Four samples from Unit 1 showed increasing nickel leaching trends distinct from other samples that showed generally stable nickel release at low concentrations. Three samples were ultramafic rock and one sample was troctolitic rock. The same samples also showed increasing cobalt leaching at concentrations about an order-of-magnitude below nickel. There was also some evidence of copper leaching showing the same trend at comparable concentrations to cobalt. The samples showing increasing nickel, cobalt and copper release were also the same samples showing lowest pH.

- **Virginia Formation**. Sulfur concentrations in these samples ranged from 2.0 to 5.7%. Two samples with 75th and 90th percentile sulfur concentrations (3.8 and 5.7% sulfur, respectively) yielded acidic leachates (pH<5) after 5 weeks of testing. The sample with 25th percentile sulfur (2% sulfur) showed declining pH but lowest pHs were generally near 7. Acidic conditions were marked by rapidly increasing sulfate concentrations as the tests started which peaked and then declined. Iron was the dominant balancing cation as pH initially decreased. As sulfate generation stabilized, iron and magnesium were important cations. Trace elements showing similar trends to iron were cadmium (peak concentration 0.01 mg/L), cobalt (1 mg/L), copper (0.03 mg/L), manganese (0.6 mg/L), nickel (7.5 mg/L) and zinc (1.1 mg/L).

  The 75th percentile sample yielded a lower pH (<4) than the 90th percentile sample and also showed increasing cadmium and zinc concentrations as pH decreased further.

- **Sedimentary Hornfels**. Sulfur concentrations in these samples ranged from 0.24% to 2.5%. Two samples of hornfels representing the 75th and 85th percentile sulfur concentrations (1.7 and 2.5%) showed declining pH and eventually generated leachates with pH<5 between 40 and 50

weeks into the test. At the same time, release of sulfate and a number of ions and trace elements also increased. The group of elements was similar to the Virginia Formation. The other three samples representing 10th, 25th and 50th percentile sulfur concentrations (0.24%, 0.44% and 0.55%) all produced slightly alkaline leachates. These samples all contained detectable carbonate ranging from 4 to 22 kg $CaCO_3$/t with the highest carbonate content in the lowest sulfur sample which was described as granoblastic calc-silicate. The main feature of leachate chemistry for these samples was relatively elevated leaching of arsenic for the 10th and 25th percentile samples (up to 0.09 mg/L later in the test), antimony (0.01 mg/L) and molybdenum (0.001 mg/L) compared to the Duluth Complex igneous rocks.

Lean Ore

Lean ore samples are a variant of the S>0.05% sample set in which copper and nickel constitute a relatively greater proportion of the sulfide minerals than waste rock. This group includes both Duluth Complex Rock and mineralized footwall xenoliths.

- **Duluth Complex Rock**. Sulfur concentrations in these samples ranged from 0.06% to 1.8%. All samples showed declining leachate pH beginning at between 8 and 9.3 then falling to between 5.9 and 7.6. The lowest pHs were indicated for samples containing between 0.4% and 1.4% sulfur. Overall leachate trends were comparable to S>0.05% rock. Initial leachates were dominated by alkalinity and sodium shifting to mostly calcium with alkalinity and/or sulfate. Magnesium leaching from some samples dominated compared to calcium and sodium.

  Upward trending nickel leaching was apparent for at least five samples containing troctolitic or anorthositic rock with at least 75th percentile sulfur concentrations in Unit 1 (S>0.4%) reaching maximum concentrations of 1.3 mg/L. The highest nickel concentrations were associated with the lowest pHs. Likewise, cobalt concentrations showed the same trend reaching 0.1 mg/L. Maximum copper concentrations were 0.08 mg/L. Zinc concentrations increased to 0.06 mg/L.

- **Sedimentary Hornfels**. Two the samples were tested containing 1.5% and 4.5%. The sample containing 4.5% sulfur (95th percentile) showed initially rapidly decreasing pH which subsequently stabilized to between 4.2 and 4.7. The decrease in pH was accompanied by elevated but steady sulfate release. The main cations were iron and calcium. Iron concentrations increased then stabilized. Other metals showing similar trends were cadmium, cobalt, manganese, nickel and zinc. A second sample contained 1.5% sulfur (50th percentile) with minor detectable carbonate equivalent to 6 kg $CaCO_3$/t. This sample did not produce acidic leachate but sulfate release was equivalent to the other sample. Calcium was the dominant cation as leachate chemistry stabilized. Nickel and cobalt concentrations were increasing in the latter stages of the test. This sample released elevated arsenic concentrations (0.8 mg/L) in the first 10 weeks.

Ore

Sulfur concentrations in all three samples were 0.9%. Leachates from the samples showed slowly declining pH starting between 7 and 8 and ending between 6 and 7. The trend in major parameters was very similar to the other tests. Sodium, alkalinity and sulfate were initially dominant, but as sodium and alkalinity decreased, calcium and sulfate became dominant. Sulfate release was variable in the P3 sample. About week 30, sulfate increased than decreased. All three samples showed increasing copper, cobalt, manganese and nickel concentrations as pH decreased. Other parameters, including arsenic and antimony decreased.

Particle Size Experiments

Five samples were selected for dissolution testing on four size fractions. Characteristics of the samples are provided in Table 5-7.

Table 5-7: Characteristics of Size Fraction Samples

Sample ID (Drill Hole, Footage)	Rock Type Descriptor	Geological Unit	HC or MDNR Cell	Fraction	S	Cu	Ni	Co
					%	%	%	mg/kg
00-361C(310-320)	Anorthositic	1	HC 81	-1/4"+10#	0.19	0.016	0.021	44
			HC 82	-10+35#	0.16	0.014	0.020	44
			L1	-35+100#	0.18	0.016	0.021	46
			L2	-100#	0.24	0.016	0.024	54
00-334C(640-660)	Troctolitic	1	HC 83	-1/4"+10#	0.10	0.032	0.051	84
			HC 84	-10+35#	0.07	0.028	0.051	82
			L3	-35+100#	0.06	0.026	0.046	77
			L4	-100#	0.07	0.025	0.054	92
00-369C(305-325)	Troctolitic	3	HC 85	-1/4"+10#	0.25	0.038	0.031	57
			HC 86	-10+35#	0.23	0.031	0.027	54
			L5	-35+100#	0.25	0.032	0.027	57
			L6	-100#	0.30	0.033	0.034	64
00-367C(290-310)	Troctolitic	4	HC 87	-1/4"+10#	0.04	0.019	0.024	46
			HC 88	-10+35#	0.04	0.019	0.023	43
			L7	-35+100#	0.04	0.017	0.028	55
			L8	-100#	0.08	0.019	0.031	56
00-364C(210-229)	Virginia	20	HC 89	-1/4"+10#	3.64	0.016	0.018	23
			HC 90	-10+35#	3.87	0.016	0.019	24
			L9	-35+100#	3.39	0.014	0.017	22
			L10	-100#	3.55	0.011	0.017	23

Four of these samples were various types of Duluth Complex rock containing bulk sulfur concentrations varying from 0.04% to 0.25%. One sample was Virginia Formation rock containing bulk sulfur content of 3.8%. Results are described below in order of increasing bulk sulfur content of the sample.

Direct comparison of leachate chemistry for the four samples is not appropriate because the two finest samples are being leached using 2.7 L/kg of rock, whereas the two coarse fractions are leached at the rate of 0.3 L/kg. Therefore, release rates must be compared to see differences between fractions. Graphs showing selected results are provided in Appendix F.

- Sample 00-367C (290-310) (Unit 1 Troctolite – 0.04% Sulfur). Leachate pHs showed an initial downward trend but then stabilized between 6.5 and 7.6. Sulfate release was stable and greatest for finest size fractions. Likewise, the finest fraction also produced the highest loads of rock weathering products. Except for the initial phases of the tests, nickel and cobalt releases were barely detectable. Copper release in contrast was detected and was greater for the two finest size fractions.
- Sample 00-224C (640-660) (Unit 1 Troctolite – 0.08% Sulfur). Leachate pHs for the four fractions were between 6.5 and 7.5 with no particular trend. Sulfate release was downward trending. Sulfate release was inversely correlated with particle size. The smallest particles released the highest sulfate load on a mg/kg/week basis. Similarly, leaching of the main rock components and trace metals were greatest for the finest particles. Metal release was stable with no clear increasing or decreasing trends.
- Sample 00-361C (310-320) (Unit 1 Anorthosite – 0.18% Sulfur). Leachate pHs again decreased then stabilized except for the coarsest fraction which showed erratic pH toward the latter part of the test. At this time, lowest pHs were apparent for the two finest fractions operated in a MDNR reactor configuration. Over the duration of the test, the -10+35 mesh sample consistently had the highest pH. Sulfate release was stable and greatest for the two finest fractions. Likewise, main rock components were leached at the greatest rate for the two finest fractions. Nickel was low but appeared to be increasing for the coarsest and two finest fractions while the -10+35 mesh fraction had low stable nickel release. Cobalt showed similar results. Copper release was greatest for the two finest fractions but appeared to be stable. Zinc release was increasing for the finest fraction.
- Sample 00-369C (305-325) ((Unit 3 Troctolite – 0.25% Sulfur). Leachate pH values were consistently greater for the -10+35 mesh sample (7.3 to 7.7) compared to the coarsest and two finest fractions (6.4 to 7.1). Sulfate production was generally greatest for the two finest fractions and lowest for the coarsest fraction. Major rock weathering components showed the same trend. Nickel release was barely detectable except toward the later stages when it began to increase very slightly. Cobalt was undetected except for the first leachate. Copper was released at a greater rate for the two finest fractions and there were no clear trends.
- Sample 00-364C (210-229) (Virginia Formation – 3.79% Sulfur). All four fractions have generated acidic leachate (pH below 5). The two coarsest fractions generated the lowest pH

water (3.7) compared to a minimum pH of 4.2 for the two finer fractions. The two finest fractions showed higher sulfate release than the two coarse fractions. A similar effect was apparent for major rock weathering components. Nickel and to some extent cobalt, zinc and cadmium leaching initially peaked in the -100+270 mesh and -10+35 mesh samples, and to a lesser degree for the -35+100 mesh sample but not for the coarsest sample. Copper did not show a comparable peak but instead showed an upward trend in the two finest fractions. Iron trends showed a combination of the two effects. An initial peak occurred early in the test followed by increasing iron release in the later stages.

Comparison of Results for Different Tests

Two comparisons have been made:

- Comparison of ASTM humidity cell and MDNR Reactor results for size fractions obtained from the same sample; and
- Calculation of the weighted sum of weathering rates indicated by the particle size experiments compared to the equivalent ASTM Humidity cell.

These comparisons can be made for the same five samples tested as four size fractions.

The comparison of ASTM humidity cells and MDNR reactors indicated some consistent features. For the four samples generating leachate chemistry with pH greater than that of the deionized water used to leach the sample, the MDNR reactor consistently produced lower pH leachate. The difference was variable but was typically about 0.5 pH units with the MDNR reactors producing leachate pHs near or below 7. Sulfate concentrations tended to be greater for the ASTM humidity cells. For two samples indicating detectable nickel release (00-334C(640-660) and 00-361C(31-320)), nickel concentrations in the leachate from the MDNR reactors was greater than for the equivalent ASTM humidity cell. Also, the MDNR reactors showed an upward trend in nickel release compared to the humidity cell. Cobalt concentrations were not frequently detectable and copper concentrations showed stable release.

The Virginia Formation sample produced acidic leachate. In this case, pH was lower for the ASTM humidity cell (less than 4) and on a decreasing trend.

Low Level Analysis for Cobalt and Nickel

During planning of the program, it was expected based on MDNR experience that cobalt and to a lesser extent nickel concentrations would be below the reporting limit of the analytical method (0.0001 mg/L or 0.1 µg/L) (Folman 2006a). As described in Section 4.2.4, several approaches were considered to address this issue, one of which was to use analytical methods with lower reporting limits. The laboratory indicated that it could report concentrations of 0.05 µg/L by concentrating composite leachate samples.

Ultra low level analyses was completed for a total of 243 leachate samples collected from testwork on waste rock (Appendix G.1). Leachates were analyzed primarily because cobalt was below the reporting limit. All samples were analyzed for cobalt and about half were analyzed for nickel using the lower reporting limit of 0.05 µg/L. A summary of detections with respect to the two reporting limits is shown in Table 5-8. The results show that 201 (83%) of the cobalt results were below the original reporting limit of 0.1 µg/L. The majority of these samples (144) were below the ultra low level reporting limit. Forty-eight results were between the two reporting limits.

Table 5-8: Results of Low Level (0.05µg/L) Analysis for Cobalt and Nickel

Concentration Range (µg/L)	Number of Co Results	Number of Ni Results
Total Determinations	243	133
Less Than 0.05	144	53
Detected at 0.05	9	0
Between 0.05 and 0.1	48	7
Greater than 0.1	42	73

The results obtained have been considered in two ways:

- Do the short term release rates indicated by the results improve understanding of the correlation between the cobalt and nickel content of the rock and the release of these elements?
- Does the improved detection of nickel and cobalt allow a regression relationship to be developed between these parameters that will allow cobalt to be predicted from nickel?

The above comparison demonstrated that the majority of the cobalt concentrations below the routine reporting limits were also below the ultra low level reporting limits but Figure 5-3 shows that the low level reporting limit cobalt concentrations did not result in improved understanding of the relationship between cobalt in the rock and cobalt release.

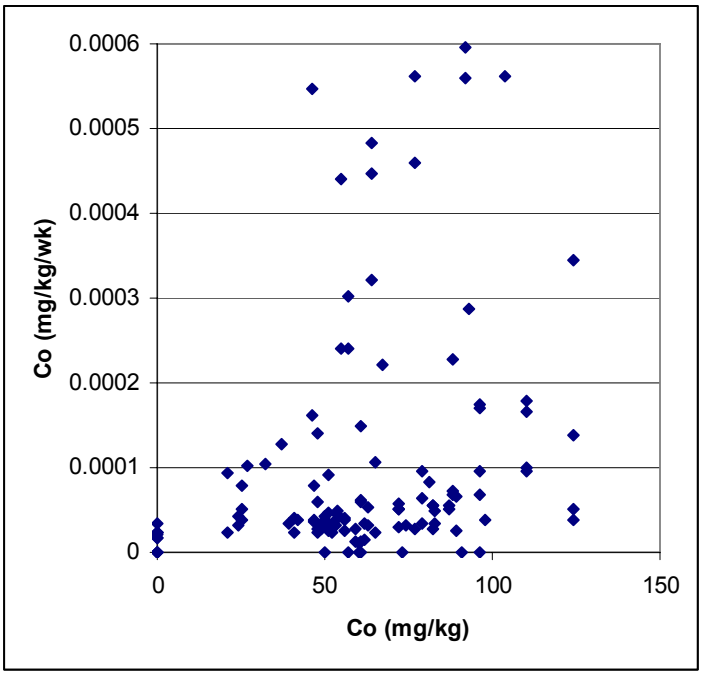


Figure 5-3: Short Term Cobalt Release Using Ultra Low Level Results Compared to Total Cobalt Content of Waste Rock.

Finally, detectable cobalt and nickel concentrations were compared to determine if cobalt could be predicted from nickel. Figure 5-4 shows that nickel and cobalt concentrations are correlated and the correlation coefficient is low (0.56) but statistically significant. However, this level of correlation is not reliable to predict cobalt release from nickel release.

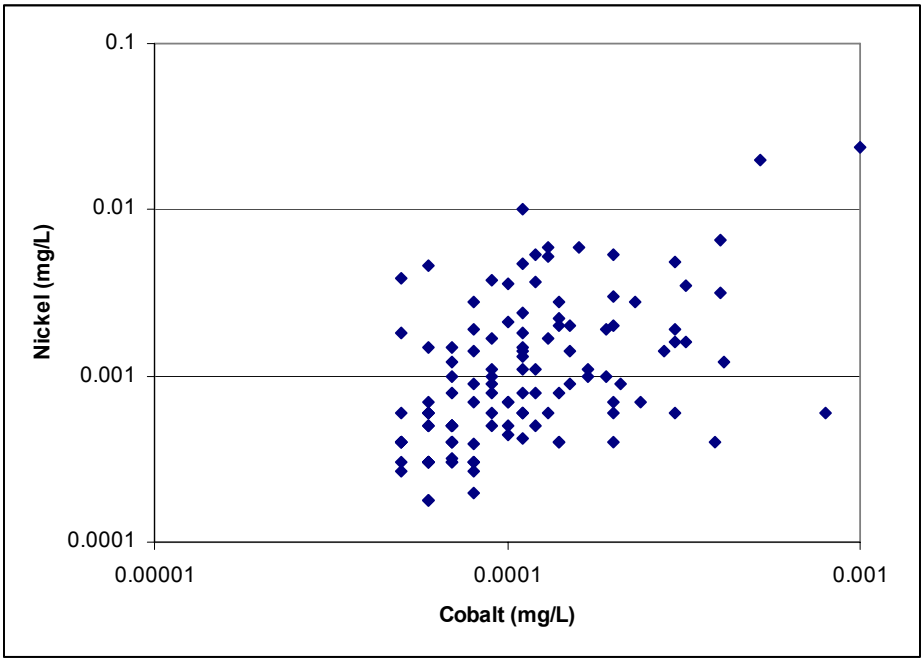


Figure 5-4: Comparison of Low Level Cobalt and Nickel Concentrations.

Low Level Analysis for Mercury

The routine ICP-MS detection limit for mercury is 0.02 µg/L (20 ng/L). In order to better understand mercury leaching from the rock, one set of water samples from all humidity cell tests was analyzed using a specific method to achieve a detection limit of 2 ng/L. A total of 93 analyses were completed (Appendix G.2).

All except one leachate contained detectable mercury concentrations, and only one sample contained a mercury concentration exceeding 20 ng/L (30.1 ng/L). The method blank contained 2.4 ng/L indicating traces of detectable dissolved mercury. Summary statistics by sulfur content and rock type are provided in Table 5-9. As shown, average mercury concentrations were between 5 and 7 ng/L and there was no statistical difference for the various groupings of the data. Because the majority of leachates contained mercury above the blank concentration, the rock leaches low levels of mercury which are unrelated to rock type or sulfur content.

Table 5-9: Summary Statistics for Mercury Analyses (in ng/L) on Humidity Cell Leachates

Group	n	Mean	Standard Deviation	95% Confidence Limit	
				Lower	Upper
S≤0.05%	27	5	2.4	4	6
S>0.05%	37	7	5.2	5	9
Lean Ore	24	6	3.9	5	8
Ore	3	7	4.0	-3	17
Anorthositic	15	6	6.9	2	9
Troctolitic	47	6	3.6	5	7
Ultramafic	15	6	3.2	4	8
Sediments	11	6	4.1	4	9

5.2.5 QA/QC

The following sections describe QA/QC results. The emphasis in these sections is on major element chemistry and parameters that can be expected to be regulated at the site. A number of trace parameters are reported in the ICP scans are not discussed below because they do not fall into either category.

Method Blanks

Method blanks have been run for both the ASTM humidity cell and MDNR reactor style tests. The ASTM method blank cell reported detectable SO_4, Al, As, Ni, K, and Si in only one to three analyses. All of these components had concentrations were less than 5 times the method detection limits (MDL) of 0.5, 0.001, 0.0001, 0.0001, 0.25, and 0.05 mg/L, respectively.

Several waters showed measurable Zn concentrations but they were all within 4 times the ICP-MS MDL of 0.001mg/L or 2 times the ICP-OES MDL of 0.005 mg/L. Two to three analyses for Pb, Mn, and Na reported concentrations up to 10 times the MDL (0.00005, 0.00005, and 0.01 mg/L, respectively). Most Ca and Cu concentrations were greater than the MDL (up to 8 times greater than the MDL of 0.01 mg/L for Ca and up to 60 times greater than the MDL of 0.0001 mg/L for Cu). In the case of copper, concentrations have been stable between about 0.001 and 0.002 mg/L. It is believed that these detections of calcium and copper represent the effect of leaching of the plastics by the weak acidity in the deionized water because there are no matching detections in the travel blank. Calcium was undetected in the travel blanks and copper concentrations were mostly undetected.

Elevated copper concentrations (up to 0.006 mg/L) were reported in the blank humidity cell in first 10 weeks prior to re-location of the laboratory and installation of new equipment to prepare the deionized water.

The MDNR reactor method blank cell reported detectable SO_4, Cr, Fe, Ni, K, Si and Na in one to four analyses. All of these concentrations were less than 5 times the MDLs of 0.5, 0.0002, 0.01, 0.0001, 0.25, 0.01, 0.0001, and 0.03 mg/L, respectively. Several analyses reported measurable Zn and Al concentrations. Most were within 5 times the MDL of 0.001 mg/L for both parameters, except one Zn analysis that was 22 times the MDL. Detectable concentrations of Ca and Cu were consistently reported. Calcium concentrations were within 6 times the MDL of 0.01 mg/L while Cu concentrations were up to 22 times the MDL of 0.0001 mg/L.

Review of testwork data for cells and reactors containing rock samples showed substantial differences in antimony leaching between the humidity cells and MDNR Reactors. A similar effect was noted for tailings samples (RS46, SRK 2007). The effect was traced to leaching of some polyvinyl chloride (PVC) components of the humidity cell apparatus leached antimony. These components were present in the waste rock humidity cells but were inadvertently not included in the waste rock humidity cell blank test started at the beginning of the program. A new blank test was started and confirmed that antimony leaching from waste rock cells was attributable to the PVC components. Leachate antimony data for the waste rock humidity cell were therefore rejected. MDNR reactor tests did not contain components that leached antimony and the antimony data have therefore been retained for subsequent data interpretation.

Ion Balances

The laboratory uses a criterion of 15% for ion balance differences. When the difference is greater than 15%, additional ions may be included in the calculation to improve the balance or there may be an analytical error that would cause the imbalance. Detection level limitations make determination of charge balance on low-strength solutions (e.g. EC<50 µS/cm) difficult, and higher ion imbalances must be accepted for these samples. *Standard Methods for the Examination of Water and*

Wastewater[3] recommends an acceptable ion balance difference of +/- 0.2 meq/L for an anion sum of 0 – 3 meq/L. Therefore, at a meq/L of 0.5 for both the anions and cations the acceptable error is 20% while at 1 meq/L for both anions and cations the acceptable error is 10%.

Approximately 1.8% of all of the ASTM analyses had ion balance differences greater than 15% (0.5% had greater than a 20% difference). These differences were largely acceptable based on *Standard Methods* criteria indicating that charge balances were excellent.

Nearly 16% of MDNR reactor analyses had ion balance differences greater than 15% (7.5% greater than 20%) due to the generally lower concentrations compared to the humidity cell leachates. Due to the low conductivities of these samples they were largely acceptable according the *Standard Methods* criteria.

Duplicate Tests

Nine duplicate tests have been run for ASTM tests, and two for the MDNR reactor tests. In general, duplicate samples demonstrated similar concentrations and trends for all parameters. One duplicate sample of waste rock (00-364C (210-229)) had Al, Cd, and Zn concentrations that were occasionally nearly an order of magnitude different, although the trends were similar. Other ions for this sample had comparable concentrations and trends.

pH

Trend analysis of pH measurements indicated a "saw tooth" trend in which values alternated between higher and lower values every other week. The reason for the pattern was that pH measurements were performed on filtered and unfiltered leachates on alternate weeks depending on whether samples were being collected for metals analysis. This was consistent with the analytical method. Because vacuum filtration potentially causes weakly buffered leachates to respond to changes in pressure by taking up or releasing carbon dioxide, determination of pH of filtered leachates was discontinued. For results prior to this point in the test program, pH measurements on filtered leachates were not reported. Where pH results were needed for interpretation of other chemical parameters, the two nearby results were averaged.

Conclusions

QA/QC measures indicated that the data generated are of very high quality. QA/QC identified a few minor concerns (antimony and pH) that were evaluated and corrected in consultation with the laboratory as the program proceeded.

3 20th Ed. American Public Health Association , American Water Works Association Water Environmental Federation AWWA, WEF, 1998.

6 Interpretation of Dissolution Testwork

6.1 General Interpretation of Leachate Chemistry

6.1.1 Observed Effects

Understanding of waste rock stockpile drainage water quality for Duluth Complex waste rock was facilitated by developing a conceptual geochemical model that could explain both the leachate chemistry observed in a variety of small laboratory experiments and field tests conducted by the MDNR over nearly two decades and more recently laboratory testing by PolyMet specifically for the NorthMet deposit.

The salient features that have been observed in kinetic testwork and need to be explained by the geochemical model are as follows:

- Overall Observations
    - The variable long term delay in development of acidic conditions in the absence of abundant carbonate mineral buffering capacity. Acidic conditions in this context are defined as leachate pH below the deionized water pH of about 5.5.
    - The absence of acidic conditions for samples containing sulfur concentrations less than 0.4%.
    - Strong correlation of oxidation rates with sulfur content.
- Leachate pH trends
    - Initially strongly basic alkaline leachates (pH>8) followed by a steep decline in pH to less than 8 within a few months of initiation of kinetic tests.
    - Stable leachate pH for samples that have not generated acidic leachate after 18 years of testing.
    - The generally slow decline in leachate pH in kinetic tests that eventually produce acidic leachate. The initial decline is not necessarily accompanied by increasing sulfate release.
    - At times stepwise sharp decreases in pH under acidic conditions.
    - Steady (at times with steps) recovery of leachate pH following a short term pH minimum.
- Difference between test procedures
    - Generally lower leachate pHs for MDNR reactor tests compared to ASTM humidity cells on the same material when pHs are above 5.5.

- Generally lower leachate pHs for ASTM humidity cells compared to MDNR reactor tests on the same material when pHs are below 5.5.

- Metal Leaching Trends
  - Increasing nickel and cobalt concentrations as pH decreases below 7 but not necessarily in the presence of increasing sulfate release.
  - Subsequent decrease in nickel and cobalt release even as pH remains below 7 and continues to decrease.
  - Coincident sulfate, nickel, cobalt and copper concentration peaks as pH drops below 5.
  - Long term declining metal release as pH recovers following lowest pH.

The following explanation for these observations considers in turn superimposed effects of bulk rock weathering, oxidation of sulfide minerals, and storage and dissolution of secondary minerals.

6.1.2 Bulk Rock Weathering

Rock weathering under atmospheric conditions occurs in response to reaction of water and carbon dioxide dissolved in water (carbonic acid) with carbonate and silicate minerals (Drever 1982). This is the primary process responsible for chemical breakdown of silicate minerals and formation of soils (Birkeland 1984). The products of this process are dissolved alkali and alkali earth ions, dissolved alkalinity as bicarbonate, and secondary weathering products. The secondary products are usually clays, which coat the primary mineral with a weathering rind.

Initial weathering occurs when the fresh silicate mineral surfaces produced by crushing or blasting react with water. This process is analogous to generation of abrasion pH (Price 1997) and results in exchange of base cations on the surface of silicates with H^+ in the water causing the water to become enriched in hydroxyl ions and forming an initial layer of silicate clay on the surface of the primary mineral. This explains the initial high pHs observed for most kinetic tests.

The decline in pH occurs as the silicates become coated and the generation of hydroxyl ions decreases. As the rind thickens, the weathering of the primary minerals must occur by diffusion of reaction products through the rind (Drever 1982).

Troctolites of the Duluth Complex at NorthMet are composed of some of the most reactive common silicates (plagioclase and olivine) (Birkeland 1984). Weathering of the components of these minerals (anothorite, albite and forsterite) by reaction with carbonic acid can be written as:

$$CaAl_2Si_2O_8 + H_2O + 2H_2CO_3^o \rightarrow Ca^{2+} + Al_2Si_2O_5(OH)_4 + 2HCO_3^-$$

$$2NaAlSi_3O_8 + H_2O + 2H_2CO_3^o \rightarrow 2Na^+ + Al_2Si_2O_5(OH)_4 + 2HCO_3^- + 4SiO_2$$

$$Mg_2SiO_4 + 4H_2CO_3^0 + 2H_2O \rightarrow 2Mg^{2+} + SiO_2 + 4HCO_3^- + 2H_2O$$

Weathering of the fayalite component of olivine releases ferrous iron which then oxidizes and precipitates as ferric hydroxide:

$Fe_2SiO_4 + 4H_2CO_3^0 + 2H_2O \rightarrow 2Fe^{2+} + H_4SiO_4^o + 4HCO_3^- + 2H_2O$

$2Fe^{2+} + 5H_2O + 0.5O_2 \rightarrow 2Fe(OH)_3 + 4H^+$.

While the weathering of fayalite produces alkalinity in the first step, it also releases acidity (H^+) in the second step which offsets the alkalinity. Overall, dissolution of fayalite does not contribute alkalinity or acidity.

The long term MDNR testwork did not include testing of samples containing sufficiently low sulfur that would allow these effects to be seen directly. Dissolution testwork on six samples containing 0.02% sulfur was conducted by PolyMet. These samples were found to leach slowly declining levels of alkalinity which was accompanied by steady or declining levels of calcium, declining then stable sodium, and stable silica concentrations. Alkalinity leaching for four samples was distinctly lower (4 to 8 mg $CaCO_3$/L) than two samples (15 to 20 mg $CaCO_3$/L) (Figure 6-1). The ratio (Ca/2 + Na/3 + 2Mg)/Si was used to evaluate the relationship between silicon leaching and the expected cations leached according to the formulae of anorthite, albite and forsterite (Figure 6-1). Leaching of the fayalite component of olivine could not be considered because iron was not detected. This does not mean that fayalite was not leaching. Precipitation of ferric hydroxide typically controls iron to sub-detectable levels.

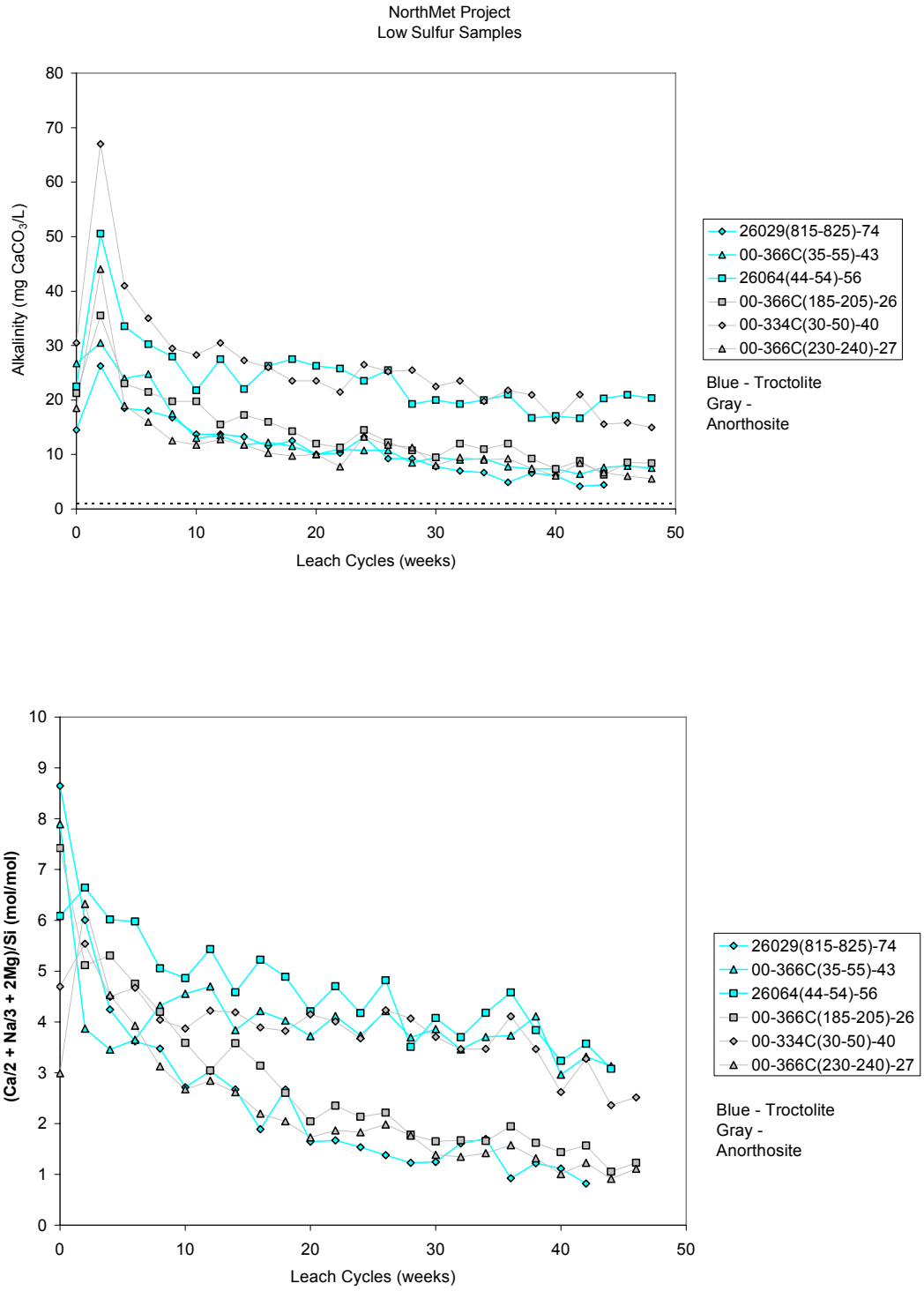


Figure 6-1: Evaluation of Silicate Mineral Weathering for Samples Containing Low Sulfur Concentrations. Top – Alkalinity Concentrations. Bottom – Silicate Ratio.

The ratio for three samples leaching low levels of alkalinity was decreasing and approached the expected value of 1 if none of the elements (including silica) were precipitating as secondary minerals. As shown by the above equations for dissolution of the silicates, calcium, sodium and magnesium would not be expected to precipitate as secondary minerals but silica might be retained as silicate clays. Three other samples showed decreasing values of the ratios. The possibility exists that alkalinity released is due to leaching of small quantities of carbonate minerals. Carbonate was detected in sample 26064(44-54) at 0.06% C just above the detection limit of 0.05% and in sample 00-334C (30-50) at the detection limit therefore proving a plausible explanation for its higher ratio. All other samples contained no detectable carbonate and the ratio was consistent with silicate weathering.

Leachate data shows that alkalinity generation occurs at low rates because silicates react relatively slowly compared to dissolution of carbonate minerals. Because kinetic testing is performed using a high liquid to solid ratio (0.5 L/kg) in the ASTM procedure, the acidity and dilution provided by the deionized water is a significant consideration. The effect of liquid to solid ratio was evaluated using Geochemist's Workbench (Bethke 2005) and assuming constant weathering rates for the silicate minerals (Figure 6-2).

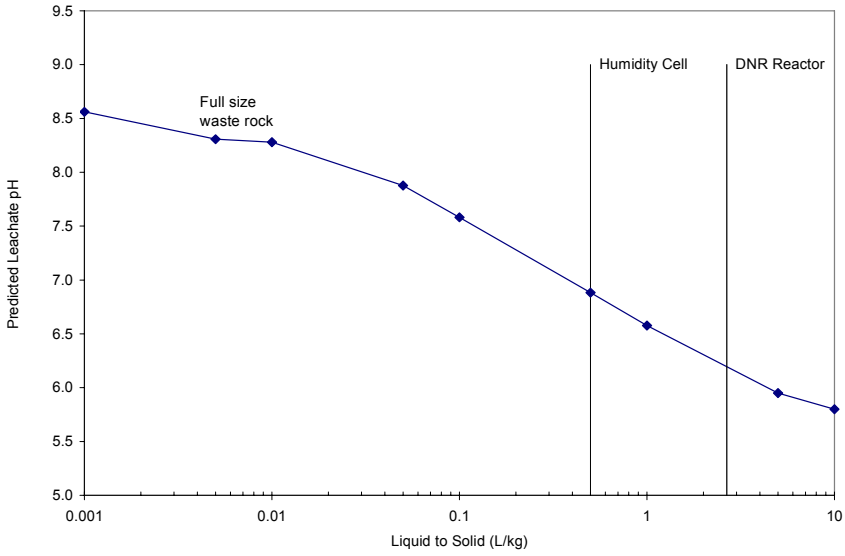


Figure 6-2: Predicted Effect of Liquid to Solid Ratio on Leachate pH with Alkalinity from Silicate Weathering.

The results indicated that at high liquid to solid ratio used for MDNR reactors and ASTM humidity cells, pHs can be expected to be in the 6 to 7 range with pHs 0.5 units higher for the ASTM test. At lower liquid to solid ratios that would be expected in a waste rock stockpile, pHs would be expected to be between 8 and 8.5. At these higher pHs, secondary carbonate minerals are predicted to form by weathering to produce a stored form of carbonate alkalinity. These findings represent important considerations for sulfide mineral oxidation and the assessment of metal mobility as described below.

6.1.3 Oxidation of Sulfide Minerals

Sulfide minerals are a trace to minor component of the rocks at NorthMet but oxidation of them results in generation of acidity by well-understood processes. The products are sulfate ions, protons, iron in some form such as dissolved ferric or ferrous iron, other dissolved metals and/or various types of precipitates.

Testing by the MDNR and for NorthMet has shown that sulfate release (which indicates sulfide mineral oxidation) is well-correlated with sulfur (i.e. sulfide mineral) content of the rock (e.g. Lapakko and Antonson 2006). This indicates that acid generation is expected to become an increasingly important process.

The following sections describe the proposed geochemical mechanisms for interaction of acidity from oxidation of sulfide minerals with alkalinity produced by weathering of silicate minerals.

At very low sulfur content, the acid produced by oxidation of sulfide minerals is easily neutralized by reaction with the bicarbonate alkalinity produced by weathering of silicates as described above. In fact, because alkalinity generation by silicate mineral weathering is independent of sulfide mineral oxidation, excess alkalinity is available to neutralize the acidity. As sulfide content and acid generation rate increase, the acid generation rate will eventually exceed the alkalinity generation rate from weathering of silicates, and there will be excess acidity. This will not immediately cause a decrease in pH because the acidity can also react directly with the silicate minerals. However, the neutralization process is less efficient because direct reaction causes formation of alumino-silicate clay weathering products where the acid contacts the silicate minerals limiting further reactions. As this process proceeds, pH depression can be expected to occur due to incomplete neutralization of acid. Depression of pH causes bacterial activity to accelerate which in turn further accelerates sulfide mineral oxidation. Downward trending pH continues as long as sulfide minerals are readily available.

Testwork results have shown that at times pH decrease occurs in steps. This reflects dissolution of secondary minerals which act as pH buffers. For example, as pH decreases, dissolution of secondary aluminum minerals formed at higher pHs will buffer pH at between 4 and 5. As this buffer is exhausted, and sulfide oxidation accelerates further, pH drops again. Buffering at pH below 4 occurs as ferric hydroxide minerals are dissolved.

A feature of the MDNR's long term testwork is the sustained and strong recovery of pH following a minimum pH. This a unique feature of these experiments because kinetic tests from rock dominated by carbonate minerals show negligible or slow pH recovery once pH has decreased below 4 (for example, Day 1994). For the MDNR's experiments, the minimum pH appears to represent the point when readily available reactive sulfide minerals are consumed and acid generation decelerates. As this occurs, reaction of acidity with silicate minerals improves and dissolved alkalinity from silicate weathering can again contribute to acid neutralization. These observations show that the onset of acidic conditions in these rocks is not the result of depletion of acid buffering minerals as occurs when carbonate minerals are involved but rather the acceleration of oxidation. As the source of

acidity is depleted, the minerals which contribute buffering capacity through reaction with acid and generation of alkalinity by weathering remain and are the reason for the steady increase in pH.

Metal Leaching

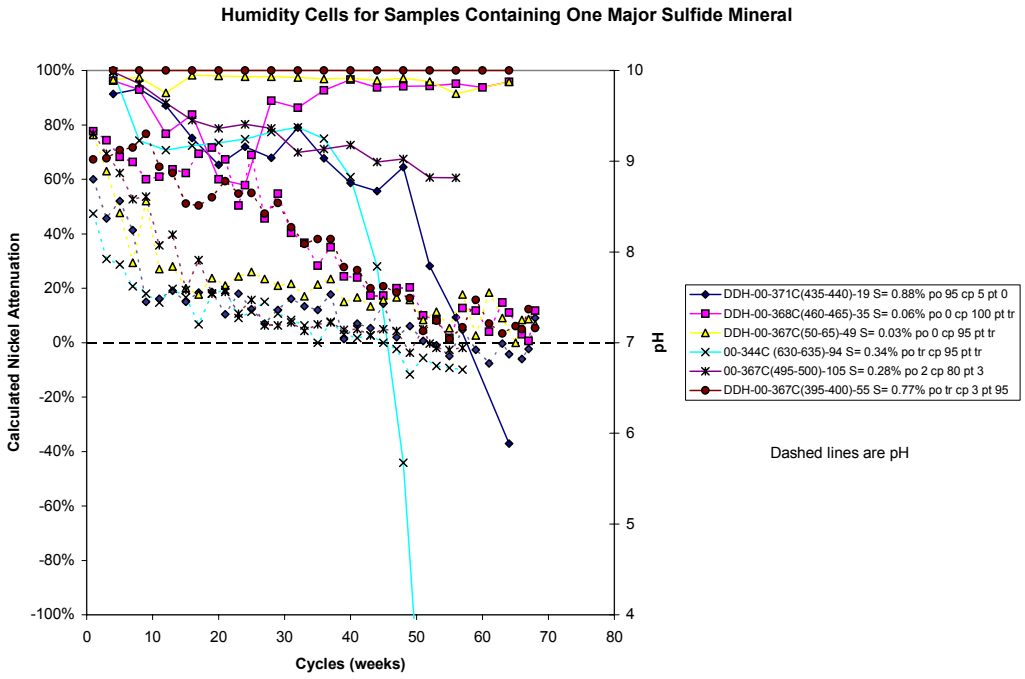
Metals are expected to leach from both silicates and sulfides. Silicates are not normally a significant source of metal leaching but the relatively elevated nickel and cobalt content of olivine and olivine's relatively high reactivity indicate that it may be important for NorthMet at lower sulfur concentrations.

In an effort to evaluate the contribution of silicate weathering to metal leaching, microprobe data and leachate data were used to construct mass balances for metals leaching from samples containing low sulfur contents. Actual nickel concentrations in leachates from these samples were very low (0.0002 mg/L after 40 weeks) but using magnesium to calculate nickel from olivine, nickel concentrations released by olivine would be between 0.001 and 0.004 mg/L. Based on these calculations, nickel was attenuated to a significant degree possibly by formation of a nickel silicate, by co-precipitation by iron released by weathering of the iron component of olivine and sulfides minerals, and adsorption to aluminum hydroxides.

A similar assessment was performed on samples containing relatively pure sulfide mineral assemblages and higher sulfide contents. Metal attenuation was estimated using magnesium to indicate olivine weathering and sulfate to indicate oxidation of the sulfide mineral (Figure 6-3). The calculation showed that nickel attenuation was initially significant but as pH decreased to between 6 and 7 nickel release increased and became greater than the nickel generated by weathering of olivine and oxidation of sulfide minerals. Copper did not show the same effect. Although pH decreased, copper continued to be attenuated in the sample.

These assessments have shown that copper and nickel are both released by weathering and oxidation but even at low concentrations are attenuated possibly as silicates, co-precipitated oxides or adsorbed to hydroxide surfaces. These products appear to be sensitive to pH in the case of nickel. As pH drops below 7, the weathering products are re-dissolved releasing nickel. Dissolution of stored products as pH drops explains the first of two peaks observed for nickel (and to a lesser degree copper) release from the MDNR's reactor experiments (Appendix B.1). As shown, nickel release increased sharply when pH dropped below 7 but then decreased despite further decreases in pH. The second peak is explained by accelerating oxidation (shown by increasing sulfate) which released nickel and copper at even lower pH. Because the oxidation products are not stored, metal release follows the sulfate peak.

As discussed in Section 6.1.3, the initial pH depression in the MDNR's reactor tests is probably an artificial effect caused by the excess acidity introduced by deionized water as the initial "abrasion pH" effect diminishes. As shown in Figure 6-2, leachate pHs at full scale are expected to be greater than in testwork due to the much lower liquid to solid ratio. This implies that provided the alkalinity produced by weathering of silicates does not exceed the acidity produced by oxidation of sulfides, pHs will remain above the critical level of around 7 below which nickel secondary minerals appear to become more soluble.



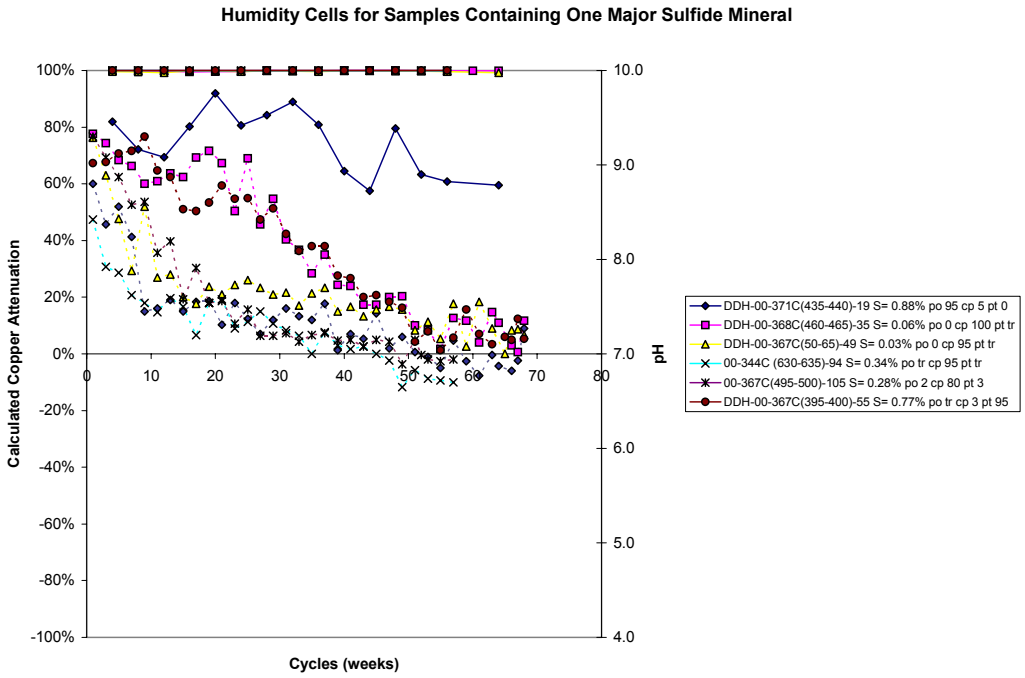


Figure 6-3: Results of Nickel and Copper Attenuation Calculation for Samples Containing One Major Sulfide Mineral (as indicated in legend). Solid lines in the figures refer to attenuation (left axis), dashed lines are pH (right axis).

Interpretation Conclusions

Table 6-1 provides a summary of the features of the data described in Section 6.1.1 and the qualitative explanation as described in Sections 6.1.2 and 6.1.3.

Table 6-1: Interpretation of Dissolution Test Observations

OBSERVATION	MDNR Testwork	NorthMet Testwork	INTERPRETATION
General			
The variable long term delay in development of acidic conditions in the absence of carbonate mineral buffering capacity.	X		Silicate minerals provide buffering capacity through carbonic acid weathering and direct reaction of sulfuric acid with silicate minerals.
The absence of acidic conditions for samples containing sulfur concentrations less than 0.41%.	X	X	Bicarbonate alkalinity produced by carbonic acid weathering of silicates permanently offsets acidity from sulfide oxidation. Sulfide oxidation is strongly correlated with sulfur content.
Leachate pH Trends			
Initially strongly basic alkaline leachates (pH>8) followed by a steep decline in pH to below 8 within a few months of initiation of kinetic tests.	X	X	Reaction of fresh silicate mineral surfaces with water ("abrasion pH").
Stable leachate pH for samples that have not generated acidic leachate after 18 years of testing	X		Steady generation of bicarbonate alkalinity by weathering of silicates.
The generally slow decline in leachate pH in kinetic tests that eventually produce acidic leachate. The initial decline is not necessarily accompanied by increasing sulfate release.	X		Bicarbonate from silicate weathering is less than acidity from sulfide oxidation and excess acid must react directly with silicate minerals. Blinding of silicate minerals limits this process.
At times stepwise sharp decreases in pH under acidic conditions.	X		Consumption of weathering mineral buffers leachate acidity. When exhausted, the pH drops sharply until the next buffer is reached.
Steady (at times with steps) recovery of leachate pH following a short term pH minimum.	X		As sulfide oxidation decelerates due to sulfide mineral depletion, silicate minerals become more effective again and pH recovers.
Difference between test procedures			
Generally lower leachate pHs for MDNR reactor tests compared to ASTM humidity cells on the same material when pHs are above about 5.5.	X	X	Higher liquid to solid ratios result in greater effect of deionized water when reacting with silicate minerals.
Increase in sulfate in some MDNR Reactor tests when pH decreases early in test	X		Lower pH enhances bacterial activity and accelerates oxidation.
Generally lower leachate pHs for ASTM humidity cells compared to MDNR reactor tests on the same material when pHs are below 5.5.	X	X	Higher liquid to solid ratios dilute acidity produced by sulfide oxidation.
Metal Leaching Trends			
Increasing nickel and cobalt concentrations as pH decreases below 7 but not necessarily in the presence of increasing sulfate release.	X	X	Secondary minerals formed above pH 7. As pH drops below 7, the secondary minerals dissolve resulting in release of metals.
Decrease in nickel and cobalt release even as pH remains below 7 and continues to increase.	X		Depletion of secondary minerals formed above pH 7.
Coincident sulfate, nickel, cobalt and copper concentration peaks as pH drops below 5.	X		Metals released by oxidation of sulfides remain in solution due to lower pH
Long term declining metal release as pH recovers.	X		Depletion of sulfide minerals

6.2 Trend Evaluation

This section was included in the agreed outline for RS42. Interpretation of trends was provided in Section 5.2.4.

6.3 Comparison of Results with Other Testwork Programs

It is apparent from the foregoing discussion that broad data trends in kinetic tests are very similar for the two programs, but direct comparison of NorthMet test program data with the MDNR's similar programs is not possible because identical samples have not been tested by the same laboratories using the same method to calibrate any differences in laboratory operating procedures.

A useful semi-quantitative comparison can be made for results of testing by MDNR and PolyMet using ASTM humidity cells on NorthMet (former Dunka Road) Project samples. Figure 6-4 shows average rates for sulfate release compared to total sulfur with release rates calculated using the same approach as the PolyMet data for the same data period (approximately 40 weeks). The large blue triangles are for the three ASTM humidity cell tests performed by the MDNR. The figure shows that despite probable differences in sample preparation methods, laboratory operating procedures and leachate analysis methods, oxidation rates indicated by the MDNR's testwork on NorthMet were similar to PolyMet's testwork. The relationship between sulfur content and oxidation rates are comparable.

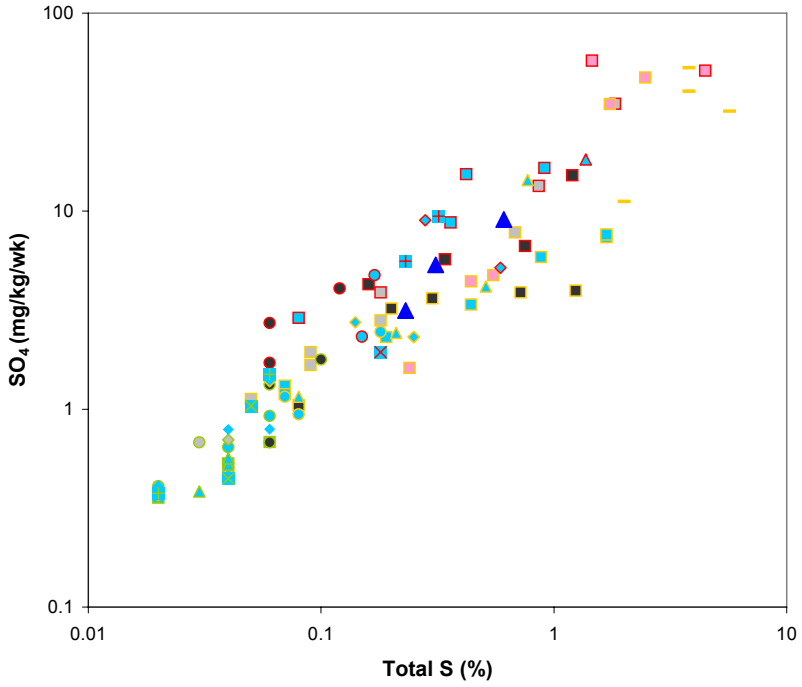


Figure 6-4: Comparison of MDNR ASTM Tests with PolyMet ASTM tests. Blue triangles are MDNR tests on NorthMet Deposit rock. Other symbols are PolyMet's tests (see Figure 6-5 for legend).

6.4 Evaluation of Waste Classification Factors

Waste classification criteria to replace the provisional criteria described in Section 4.2.1 were evaluated in order to identify the following categories of waste:

1. "Non-reactive" waste rock suitable for construction with negligible potential for ARD and metal leaching expected to produce component concentrations in drainage below water quality objectives.

2. Waste rock with negligible potential to produce ARD but likely to have drainage with component concentrations exceeding water quality objectives.

3. Waste rock with potential to produce ARD but with significant delay. Component concentrations are expected to exceed water quality objectives by a very wide margin.

4. Waste rock with potential to produce ARD immediately. Similarly, component concentrations are expected to exceed water quality objectives by a very wide margin.

The objective was to develop bulk chemical tests (or analyses) that can be linked to water chemistry and can be practically implemented during mining. The chemical tests must be easily performed at the mine laboratory and produce results rapidly so that management decisions can be made in the mine.

The approach used was to graphically compare observed average weathering rates with bulk chemical characteristics. Primary factors considered were sulfide oxidation rates as they relate to potential for ARD, and release rates for copper, nickel and cobalt. These metals have been identified as the critical parameters with respect to meeting water quality objectives.

6.4.1 Silicate Mineralogy and Content

The dominant silicate minerals are plagioclase and olivine. The relative proportions of these minerals determined the major rock sub-divisions into anorthositic, troctolitic, and ultramafic gabbro variants. The proportion of plagioclase relative to olivine decreases from anorthositic to ultramafic gabbro, therefore evaluation of weathering effects with respect to rock type provides an indication of mineralogical effects. Rocks of the Virginia Formation are lithologically distinct and show different weathering behavior.

The following was observed for the Duluth Complex rocks:

- Silicate mineral content had no effect on sulfide oxidation rates or copper release rates.

- Ultramafic (olivine-rich) rocks leach more nickel. As described below, this is partly related to the higher nickel content of ultramafic rocks.

6.4.2 Rock Type

Rock type and silicate mineralogy are effectively the same factor as described above.

6.4.3 Sulfur Mineralogy and Content

Correlation of Sulfur Content with Oxidation Rates

The result of this evaluation was that a strong relationship was found between average sulfate release and sulfide content of the rock (Figure 6-5). The relationship was well constrained at lower sulfur concentrations (<0.2%) but became more scattered above this level. In general the extension of the relationship at lower sulfur concentrations was continued by samples classified mainly as lean ore (red lines in symbols) whereas samples showing somewhat lower oxidation rates for the same sulfur content were classified mainly as waste rock (amber lines in symbols). The difference in oxidation behavior appeared to be related to mineralogical assemblage. Waste rock samples with lower sulfur content and lean ore have a copper sulfide (mainly chalcopyrite) dominated assemblage whereas waste rock with higher sulfur content has both pyrrhotite and chalcopyrite. This difference in sulfide assemblage was quantified using the ratio of Cu/S (by weight) as a surrogate for mineralogy. The separation between samples showing somewhat higher and lower oxidation rates for a given sulfur content was indicated by Cu/S=0.3. Generally, the higher oxidation rates were for samples with Cu/S>0.3 (chalcopyrite dominated).

The finding that a sulfide mineral assemblage containing more chalcopyrite is more reactive than an assemblage containing pyrrhotite as the dominant mineral was unexpected because pyrrhotite is widely considered the most reactive of the common sulfide minerals. The explanation remains empirical at this point as none of the mineralogical (crystal formation, particle size or liberation of grains, contact between grains) or crystal chemistry (trace element content) factors commonly linked to differences in mineral reactivity can be quantified.

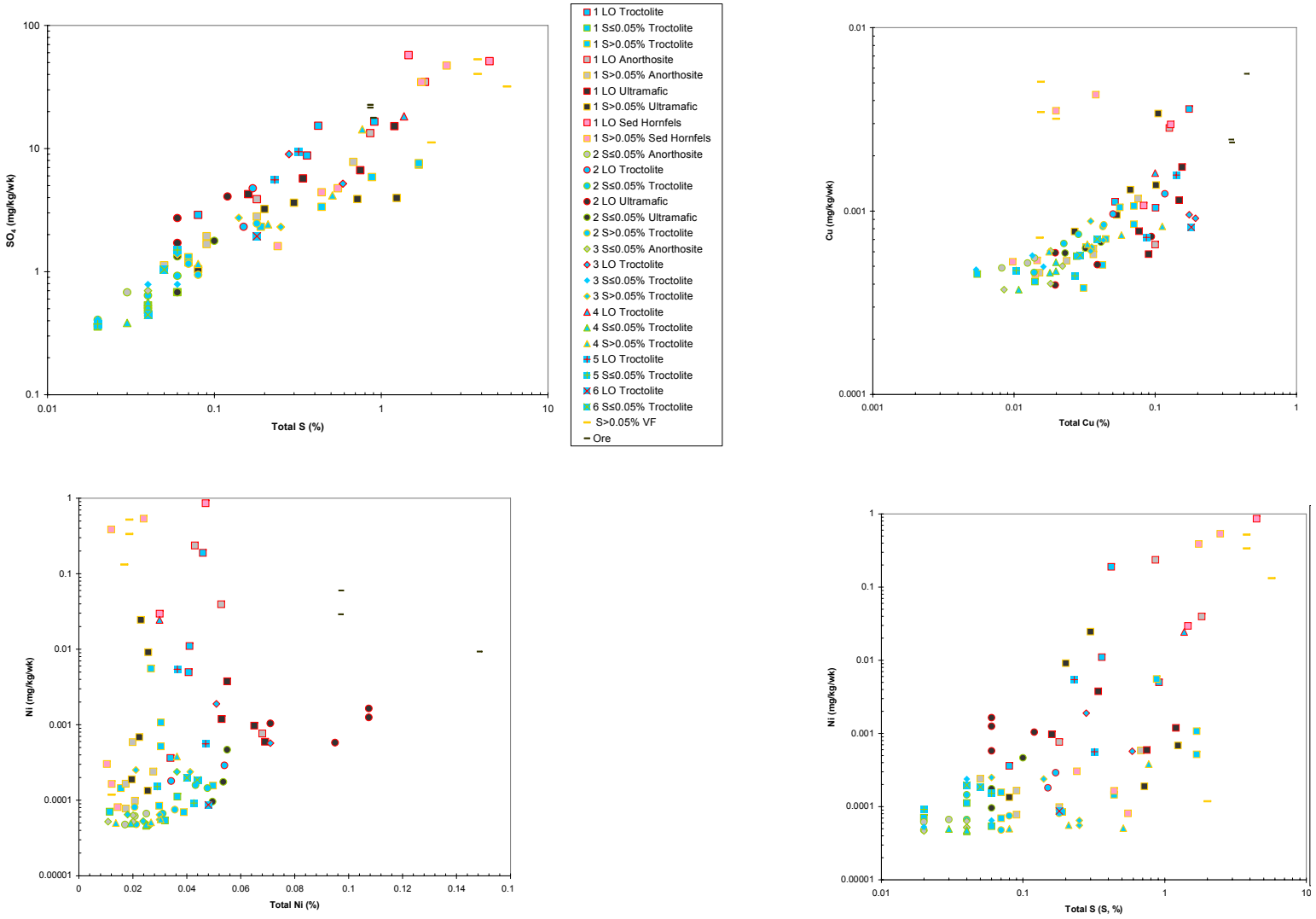


Figure 6-5: Comparison of Average Release Rates of Sulfate, Copper and Nickel Compared to Bulk Content of the Rock. Individual labels indicate intrusive layer (e.g. 1), S content, lean ore (LO) and rock type.

Development of Sulfur Content Criteria for ARD Potential

Based on the above findings, a method was used to estimate criteria that would allow an initial level of classification into low ARD potential and high ARD potential (Categories 1 and 2; and Categories 3 and 4). The secondary level of classification was then evaluated.

As described in Section 6.1.3, interpretation of leachate chemistry indicated that low level generation of alkalinity by silicate weathering should offset similarly low levels of acid generation by oxidation of sulfide minerals. Because oxidation rates are correlated with sulfide content, the critical level of oxidation at which alkalinity from silicate mineral weathering exceeds acid generation can be related to sulfide content, a measurable parameter during mine operation. The method used to develop conservative criteria were:

- Estimate bicarbonate generation rate for silicate minerals using samples containing low levels of sulfur (Figure 6-1) and exhibiting lowest levels of alkalinity. The rate used in the calculation is the lower 95% confidence limit on the average of measurements toward the latter part of the test.
- Convert alkalinity generation rate to equivalent acidity generation rate expressed as sulfate generation.
- Determine regression relationships between oxidation rate (sulfate generation rate) and sulfur content. One regression relationship was calculated for the straight line trend represented by the chalcopyrite-dominated waste rock and lean ore (Cu/S>0.3). The second relationship was calculated for the pyrrhotite-dominated waste rock (Cu/S<0.3). For each regression relationship, the lower regression envelope was calculated. The predicted sulfur content equivalent to the sulfate generation rate determined above was calculated using the lower regression envelope (95% confidence limit). This resulted in the lowest equivalent sulfur for the oxidation rate that would balance alkalinity generation.

The resulting classifications of waste rock according to sulfur and Cu/S content are provided in Table 6-2. These classifications are consistent with the MDNR's long term findings on the generation of ARD from samples under laboratory conditions. The MDNR's results have shown acid leachate from one sample with sulfur content of 0.41% and no ARD from samples with sulfur concentrations of 0.22% and 0.18%. The proposed classifications do not exceed the observed sulfur level that has resulted in acidic leachate in MDNR tests. Figure 6-2 shows that the humidity cell tests are conservative when compared to expected pH in a stockpile. A further conservative factor is that the proposed sulfur criteria are thresholds rather than average targets. The average characteristics of the rock in a stockpile will be much lower (about 0.08% sulfur) indicating that excess alkalinity will be generated by silicate weathering.

Table 6-2: Matrix of ARD Criteria

Cu/S Criteria	S Criteria		
	S<0.12%	0.12≤S<0.31	S≥0.31
Cu/S<0.3	Category 1/2	Category 1/2	Category 3 or 4
Cu/S≥0.3	Category 1/2	Category 3 or 4	Category 3 or 4

Criterion for Severity of ARD and Onset of ARD (Categories 3 and 4)

The purpose of this criterion is to separate rock that is expected to produce severe ARD rapidly from rock that will likely produce ARD at some in the future. NorthMet kinetic test database provided some information to address this aspect. Lowest pHs have been observed for samples of Virginia Formation and Sedimentary Hornfels with more than 1.7% sulfur (Figure 6-6). Leachates from other tests were not below the pH of the deionized water used to leach the samples and therefore are not yet acidic. The results showed at this stage of testing that rapid (<1 year) onset to ARD can be expected for rock containing more than 1.7% sulfur.

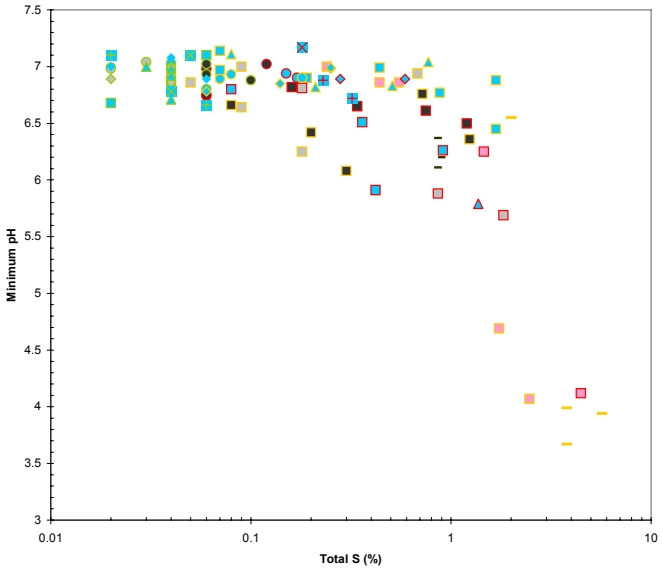


Figure 6-6: Minimum pH as a Function of Sulfur Content. Refer to Figure 6-5 for explanation of legend.

The MDNR's studies at other sites have demonstrated that sulfur is an important control on the timing of ARD and minimum pH of leachates. The AMAX test piles constructed from shaft development rock at the Babbitt Deposit showed that rock with sulfur content of 0.79% and 1.4% developed ARD with pH less than 5 in about 1 year whereas rock containing 0.64% sulfur took 6 to 15 years. The rock containing 1.4% sulfur developed the most severe ARD with the highest metal content and lowest pH.

These findings have been used to propose a sulfur criterion of 0.6% to separate Category 3 and 4 rock.

6.4.4 Metal Mineralogy and Content

Average release rates for copper and nickel were compared to absolute quantities of sulfide minerals and ratios of sulfide minerals indicated by optical mineralogy. No useful relationships were found. Copper release was related to copper content of the rock though the relationship generally defined an upper bound to copper release (Figure 6-5). Samples of sedimentary hornfels and Virginia Formation that produced acidic leachate during the test period showed higher copper release due to the greater mobility of copper at lower pHs. Nickel release was generally uncorrelated with nickel content of the rock though ultramafic samples containing higher nickel concentrations showed higher nickel release rates. Upper limits to nickel release however were related to the sulfur content of the rock. As sulfur content increased, maximum observed nickel concentrations also increased (Figure 6-5) though some samples with elevated sulfur content also showed negligible nickel leaching. The scatter in nickel release data is probably partly related to the strong sensitivity to pH of leachates as discussed in Section 6.1.3.

6.4.5 Particle Size Distribution

Particle size distribution was considered as a potential classification factor by size screening of waste rock.

As described in Section 5.2.4, sulfur and metal concentrations are enriched in the finer size fractions. Figure 6-7 also shows that average release rates for sulfate, metal release and rock weathering components increased as particle size decreased. This relationship showed that rock weathering processes are driven by surface area which increases as particle size decreases.

The weighted sum of release weathering rates was calculated using the size fraction analyses for the five samples. This calculated release rate was then compared to observed release rate for the equivalent ASTM Humidity cell (Figure 6-8). The comparison serves to determine if the individual components are performing the same separately as they do in the mixture, and whether there are effects due to the differences in the volume of water applied on a per weight basis for each fraction.

For the four Duluth Complex samples, common features were apparent. Sulfate release indicated by the humidity cell was close to the calculated rate shown by the size fractions. Calcium release was greater for the calculated rate compared to the measured rate typically for 10 to more than 30 weeks. After this period, calculated calcium release became equivalent or greater than the ASTM cell. Magnesium showed a similar trend and typically rates became equivalent toward the latter part of the tests. Evaluation of cobalt and nickel release was complicated by concentrations near the detection limit for leachate from the coarsest fraction. This fraction comprised the majority of the mass and therefore the use of at detection limit concentrations resulted in release rates for the fraction which skewed the calculated release rate. Typically, the calculated release rate was greater than humidity cell rate for this reason. Sample 00-361C(310-320) (anorthosite with 0.18% S) showed increasing calculated nickel concentrations compared to the ASTM cell which were a result of decreasing pH and increasing nickel release from the coarsest fraction. Calculated copper and zinc release rates were slightly greater than the ASTM cell.

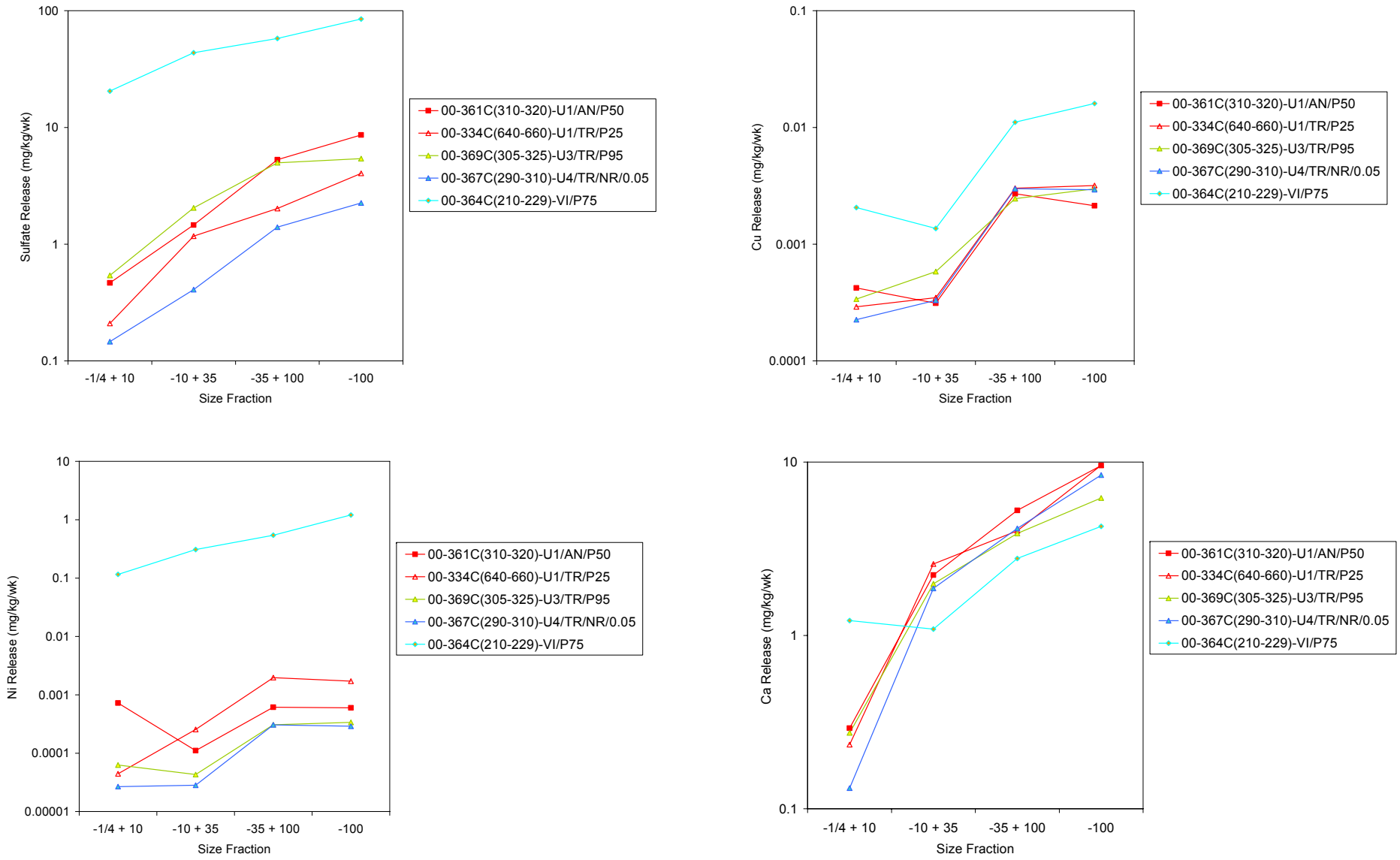


Figure 6-7: Relationships Between Average Release Rates and Particle Sizes

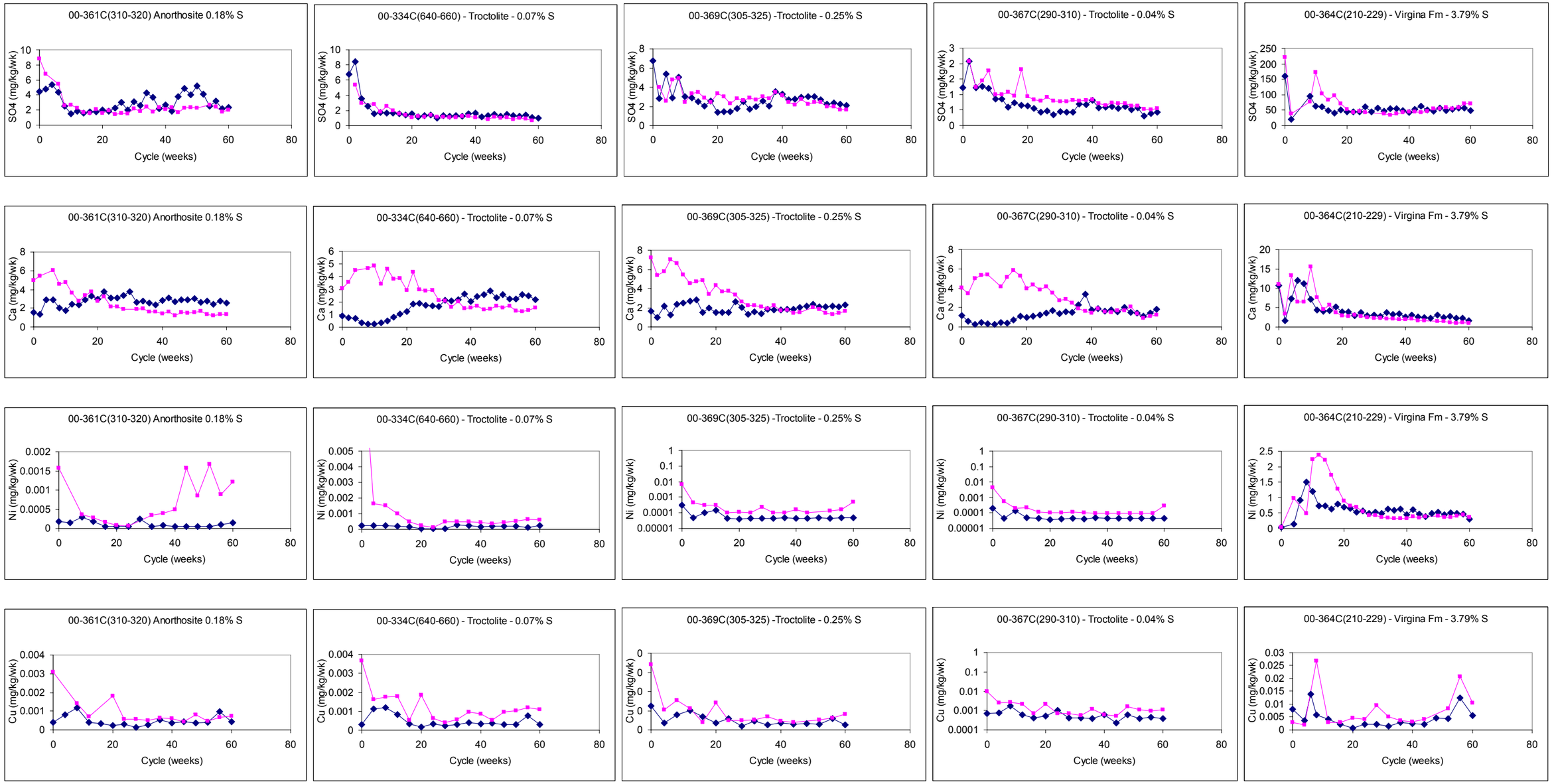


Figure 6-8: Comparison of Rates Observed in ASTM Humidity Cells (squares) with Rates Indicated by Combined Individual Size Fractions (diamonds).

The Virginia Formation sample for which leachates from the ASTM cell and size fractions were all acidic showed strong similarity for calculated and humidity cell release rates after 20 weeks. All rock weathering components also tracked closely. The calculated nickel release peaked at a higher level (2.4 mg/kg/week) than the humidity cell (1.5 mg/kg/week) but after 26 weeks nickel leaching tracked closely. Cobalt showed a similar effect. Copper release possibly showed a subdued initial peak after which both calculated and humidity cell copper release increased. Calculated zinc release peaked initially then decreased. The ASTM cell zinc release for this sample increased and peaked at 56 weeks.

Based on review of the size fraction data, it is concluded that the weathering behavior of the whole samples can be accounted for by the performance of the individual fractions and that the size fractions react mainly in response to the decrease in particle size and corresponding increase in surface area. Segregation based on particle size would result in concentration of leachable surface area into the finer particle sizes. Coarse material would be expected to leach to a lesser degree (on a mass basis).

6.4.6 Waste Classification Criteria

Potential Classification Criteria

The following potential classification criteria were indicated by the testwork:

- Sulfur content to separate rock based on ARD potential and severity of ARD potential.
- Copper/Sulfur Ratio to refine the sulfur criteria.
- Ultramafic rock as a potential means to isolate rock with greater potential for nickel leaching.
- Particle size segregation.

Operational Classification Criteria

Waste management planning as described in RS43 (PolyMet 2007c) is proceeding on the basis of using sulfur criteria linked to copper content for primary segregation. Segregation based on rock type and particle size was not considered practical under operational conditions.

6.5 Application of Results to Subaqueous Disposal

6.5.1 Implications of Results to Solute Build-Up and Dissolution

There are three considerations for subaqueous disposal:

- The simple dissolution of primary minerals and secondary (weathering) minerals. The latter are formed subaerially prior to subaqueous disposal and may affect water chemistry if the waste

rocks are highly reactive and the time between exposure and disposal resulted in extensive build-up of weathering products.

- Oxidation of sulfide minerals by oxidants contained in secondary minerals.
- Oxidation of sulfide minerals by dissolved oxygen.

6.5.2 Reaction Rates

Dissolution of Secondary Minerals

Dissolution of secondary minerals formed by weathering prior to subaqueous disposal can be assumed to occur at the ideal solubility of the secondary mineral under the disposal conditions. The disposal conditions may be substantially different from the conditions of formation resulting in higher or lower solubility. Backfill in a flooded open pit, for example, may see the development of chemically reducing conditions and contact with inflowing groundwater. The latter may have partial pressures of carbon dioxide above atmospheric conditions which will affect the dissolution of minerals occurring as carbonates. The development of reducing conditions can affect the mobility of elements stored as sorbed components of ferric hydroxides and manganese oxides. For reduction to occur, a reducing agent (such as organic matter) needs to be present.

The rate at which the secondary mineral dissolves can be estimated from the ideal solubility and the rate at which contact water is replaced. In other words, the contact water assumes the ideal solubility of the mineral as long as the mineral has not completely dissolved, and the loading is the concentration multiplied by the flow.

Oxidation of Sulfide Minerals from Oxidants Contained in Secondary Minerals

Dissolution of ferric iron salts (for example, ferric sulfate) become a source of dissolved ferric iron:

$$Fe_2(SO_4)_3 \rightarrow 2Fe^{3+} + 3SO_4^{2-}$$

The ferric iron produced can then oxidize sulfide minerals contained in the waste:

$$FeS_2 + 8H_2O + 14Fe^{3+} \rightarrow SO_4^{2-} + 15Fe^{2+} + 8H^+$$

The rate at which this process occurs is controlled by the rate at which ferric iron can oxidize the sulfide mineral. This process indicates that it is inadvisable to place highly oxidized waste underwater because it can result in acidification of the water, release of metals in salts and additional oxidation of sulfide minerals contained in the waste. The effect can be mitigated by adding a buffering agent before flooding to increase pH and precipitate ferric iron as hydroxide:

$$Fe^{3+} + 3OH^- \rightarrow Fe(OH)_3$$

Oxidation of Sulfide Minerals by Dissolved Oxygen

Dissolved oxygen in water can oxidize sulfide minerals by the same reaction as gaseous oxygen:

$FeS_2 + 7/2H_2O + 15/4O_2(aq) \rightarrow 2SO_4^{2-} + Fe(OH)_3 + 4H^+$

The rate at which the reaction occurs is limited by the low solubility of oxygen in water and the rate at which oxygenated water contacts the sulfide minerals. Lapakko (1994c) demonstrated in laboratory tests that highly reactive Virginia Formation (5% sulfur as pyrrhotite) in contact with a small volume of water reacted as rapidly as exposed rock probably because oxygen was not limited and the rock did not contain any buffering capacity.

In a situation where flowing water saturated with oxygen at 10 mg/L contacts pyrite, assuming that all the oxygen is consumed, results in a sulfate concentration of 16 mg/L and an alkalinity demand of 20 mg HCO_3/L. This low sulfate and low alkalinity demand demonstrates why acidification of subaqueous potentially acid generating waste does not occur (MEND 2001). Disposal of reactive wastes typically occurs in stagnant facilities where oxygen supply is limited, and the wastes often contain some carbonate minerals that can consume the small amount of acid produced by oxidation of sulfides.

6.5.3 Conclusions

The main factor when considering subaqueous disposal of sulfide waste rocks is the degree to which the wastes are already oxidized. Dissolution of weathering products is the primary factor affecting pore water chemistry which can be important if the waste is acidic (SENES 1996). If the waste rock is not oxidized, oxidation under saturated conditions is very slow and will not result in acidic conditions. Pore water chemistry will be controlled by the dissolution of primary minerals and release of metals from slowly oxidizing sulfide minerals controlled by the pH of the water in contact with the waste rock.

If acidic oxidation products are present, they can be neutralized by addition of lime prior to flooding.

7 Water Chemistry Database

7.1 Objective

As described in Section 2.2.2, one of the fundamental limitations of the empirical method is that direct scale-up of laboratory test results to full-scale conditions often results in prediction of elevated dissolved metal concentrations because the calculation takes loads indicated by well-flushed laboratory tests and dissolves them in small volumes of water. This does not account for the solubility of secondary minerals which form as a result of weathering of primary minerals or the solubility of the primary minerals themselves. This effect can be evaluated for known major primary and secondary minerals by considering their ideal equilibrium solubility using thermodynamic data. However, for many trace elements, solubility is controlled by small amounts of secondary minerals that cannot be identified directly and must be inferred by evaluation of water chemistry data, or by sorption processes (co-precipitation and adsorption).

Water chemistry databases make use of the concept of Geo-Environmental Models or Analogs (Plumlee and Nash 1995) because it is reasonable to assume that contaminant solubilities are controlled by secondary minerals that form in comparable geological and environmental settings. A large database containing data from testwork and full scale monitoring allows solubility relationships to be developed and applied to water quality predictions for the current project. Day and Rees (2006) provide an example of the application of this approach to water chemistry prediction to a porphyry copper mine.

The primary need for the database was to identify data from comparable mineralogical settings because mineral solubility is controlled by the availability of ions to form secondary minerals. Also, because it is expected that the majority of NorthMet waste rock will leach under non-acidic conditions, the search was targeted to mines where ARD is not being released. The solubility of metals under acidic conditions is known to be high so the search for sites with data on ARD composition was limited.

Data was sought from a number of sources including:

- Minnesota MDNR.
- USGS (Geoff Plumlee).
- SRK Files.
- Environment Canada (Robert McCandless).
- Other consultants.

Appendix H.1 tabulates 96 mines compiled as part of a broad search for information on low level leaching of nickel.

7.2 Sources of Data

7.2.1 Historical Duluth Complex Testwork Data

All historical Duluth Complex testwork data generated by the MDNR for other sites has been included in the database. These data were obtained from other nearby Duluth Complex intrusions. The data have been included in the database because the bulk and trace mineralogy of the Duluth Complex is relatively homogeneous and believed to be consistent with NorthMet.

7.2.2 NorthMet Project

The MDNR's kinetic test data for the Dunka Road Project (which is now called the NorthMet Project) and PolyMet's kinetic test data have been included in the database.

7.2.3 Dunka Pit Stockpile Monitoring Data

As described in Section 3.2.4, the Duluth Complex waste rock stockpiles at the Dunka Pit are of uncertain composition (Eger 2006). The value of the data in terms of understanding weathering of rock of known composition is therefore limited. The data were included in the database but were not used for interpretations.

7.2.4 USX Bulk Sample Pit, AMAX, Teck Cominco, INCO Bulk Sample Sites

A number of bulk sampling programs have been completed primarily in the nearby Babbitt Deposit. US Steel excavated a test pit in the NorthMet Deposit. As described in Section 3.2.1, AMAX's underground bulk sampling resulted in construction of four waste rock test piles. These data were included in the database because the waste rock is known to be similar to the NorthMet Deposit and the composition of the test piles is known.

No suitable data were identified from other bulk sample sites.

7.2.5 Other Sites in Similar Geological Settings

The USGS has classified the Duluth Complex Deposits as part of "Deposits related to mafic-ultramafic rocks in unstable areas". Its classification is 5a in the USGS System (Cox and Singer 1986) and it is included in the same group as the Noril'sk Deposits in Russia. Eckstrand and Hulbert (2006) call this group "Rift- and continental flood basalt-associated mafic sills and dyke-like bodies" and include other similar deposits (Jinchuan, China; Muskox, Nunavut, Canada and Crystal Lake intrusion, Ontario, Canada). The Noril'sk and Jinchuan Deposits are mined but no information could be found on waste rock drainage geochemistry.

7.2.6 Other Sites in Ultramafic Settings

In addition to the rift association, Eckstrand and Hulbert (2006) identified five other mafic and ultramafic deposit types which host nickel, copper and platinum group elements. These include:

- A meteorite-impact mafic melt sheet that contains basal sulfide ores (Sudbury, Ontario)
- Komatiitic (magnesium-rich) volcanic flows and related sill-like intrusions (Thompson, Manitoba; Raglan and Marbridge, Quebec; Langmuir, Ontario; Kambalda and Agnew, Australia; Pechenga, Russia; Shangani, Trojan, and Hunter's Road, Zimbabwe).
- Other mafic/ultramafic intrusions (Voisey's Bay, Labrador; Lynn Lake, Manitoba; Giant Mascot, British Columbia; Kotalahti, Finland; Råna, Norway; Selebi-Phikwe, Botswana).
- Reef-type or stratiform PGE deposits, which occur in well-layered mafic/ultramafic intrusions (Merensky Reef and UG-2 chromitite layer of the Bushveld Complex, South Africa; J-M Reef of the Stillwater Complex, Montana; Main sulfide zone in the Great Dyke, Zimbabwe).
- Magmatic breccia type, which occur in stock-like or layered mafic/ultramafic intrusions (Platreef deposits of the northern Bushveld Complex, South Africa; Lac des Iles deposit and Marathon deposit, Ontario).

SRK sought information from many of these mines and in some cases made direct contact with the mines. No direct results for waste rock drainage were identified.

Diamond mines have recently been opened in northern Canada and drainage chemistry data are available for waste kimberlite which is a nickel-rich ultramafic rock type with sulfide content close to the concentrations in the NorthMet Deposit. However, kimberlite contains carbonate minerals and was therefore considered inappropriate as an analogue.

7.2.7 Other Sites with Pyrrhotite as Dominant Sulfide

Numerous other deposit types contain pyrrhotite as the dominant sulfide (including volcanogenic and stratiform massive sulfides). These sites were not pursued as sources of data because there are many differences in host rock mineralogy and they are not natural analogues to the deposits in the Duluth Complex.

7.2.8 Other Sites with Reactive Iron Sulfides

For the same reason as described in Section 7.2.7, mines in which pyrite is present as the dominant sulfide mineral were not included.

7.3 QA/QC of Database

The database is provided on the CD attached to this report.

The compiled database contains 28,700 data records of which roughly 5,000 are from the current NorthMet laboratory test program and the balance are data generated by the MDNR since the 1980s. QA/QC of NorthMet data is described in Section 5.2.5 of this report. The quality of the data transmitted to SRK by the MDNR was not evaluated because the MDNR has been involved in compiling and maintaining the database.

7.4 Interpretation of Database

7.4.1 Relationships among Parameters

The database was primarily evaluated for relationships between ion concentrations and pH. Appendix H.2 provides charts showing pH vs element and ion concentrations.

Table 7-1 provides conclusions on the distribution of contaminant concentrations with respect to pH. The main components occurring at elevated concentrations in leachates of all types, in addition to the major ions were sulfate, cobalt, copper, nickel and zinc. These components have been analyzed in the majority of leachates and the distribution with respect to pH is well defined and consistently shown by various data sources (Figure 7-1). The upper bound to the relationships is mainly defined by the AMAX test piles. At higher pHs, the MDNR's long term Dunka reactor experiments and Babbitt Deposit experiments provide some indication of metal solubility. Generally concentrations in leachates obtained from laboratory experiments were lower than indicated under field conditions. This indicates that the release of these elements is a function of pH but that the high liquid to solid ratio in the tests dilutes the reaction products. Concentrations of these parameters increase as pH decreases.

Other components tended to occur at trace concentrations and have only been analyzed in leachates from PolyMet's laboratory tests. As a result, the distribution with respect to pH is not well known. The lack of analyses of acidic leachates from Duluth Complex rocks limits the understanding of low pH solubility of elements such as antimony, arsenic, barium, beryllium, boron, chromium, mercury, molybdenum, selenium, silver and thallium. The data show however that many of these elements are more soluble under alkaline conditions than acidic conditions (Figure 7-1). Because several components occur as oxyanions this finding is consistent with expected chemical behavior.

7.4.2 Evaluation of Geochemical Controls on Metal Solubility

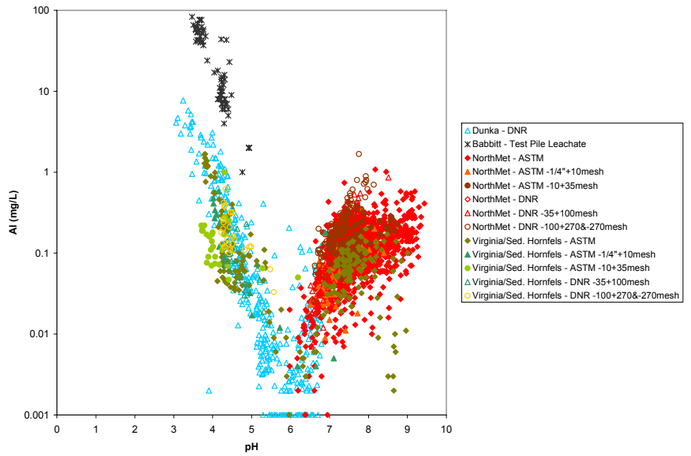
Thermodynamic Controls

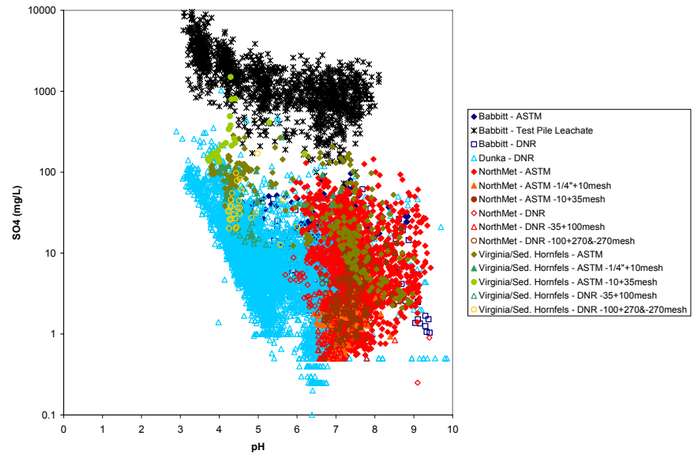
Solubility controls on metal concentrations were evaluated using Geochemist's Workbench. The thermodynamic database was updated to provide data for two nickel silicates (Nepouite and Ni-Kerolite) provided by Schmiermund (2006) and obtained from Golightly (1981). Selected

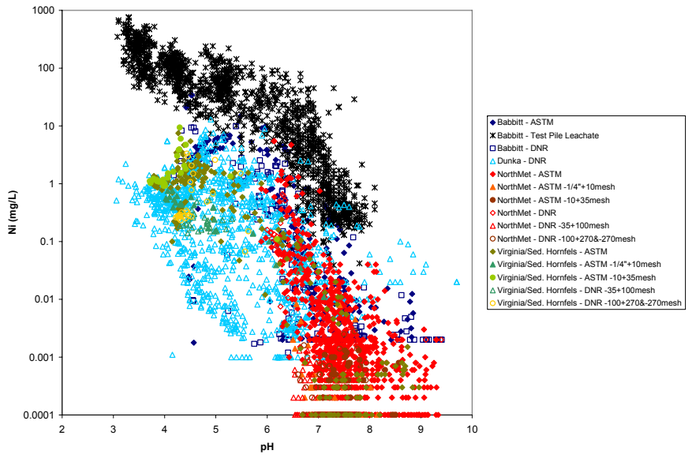
leachate results were input into the model and the resulting saturation indices obtained were used to determine whether any minerals could be controlling the solution chemistry. The samples were selected as pH transects through the data scatter at various pH concentrations. Data for Duluth Complex rocks and Virginia Formation were assessed separately. Saturation indices were plotted on the graph as data point labels. An example for nickel from the Duluth Complex is provided in Figure 7-2.

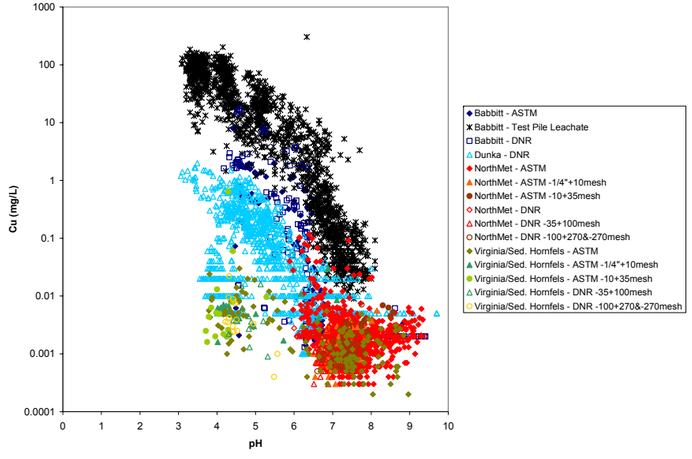
Table 7-1: Distribution of Components with Respect to pH and Maximum Concentrations Indicated at Characteristic pHs

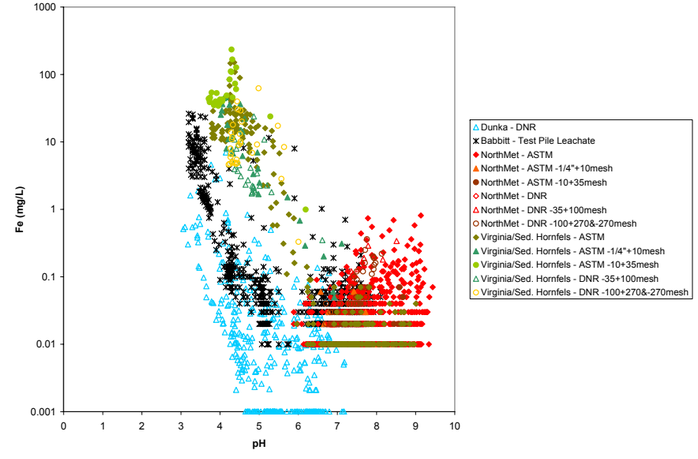
Parameter	Description
Alkalinity	Concentrations increase as pH increases
SO_4	Concentrations increase as pH decreases. Highest concentrations at full scale
Al	Envelope concentration minimum between 5 and 6.
Sb	Upper envelope increases as pH increases.
As	Upper envelope increases as pH increases. No data for Duluth Complex at low pH.
Ba	Upper envelope increases as pH increases. No data for Duluth Complex at low pH.
Be	Mostly undetected
B	Upper envelope increases as pH increases. No data for Duluth Complex at low pH.
Cd	Mostly undetected
Cr	Upper envelope increases as pH increases. No data for Duluth Complex at low pH.
Co	Upper envelope Increases as pH decreases.
Cu	Upper envelope Increases as pH decreases.
Fe	Upper envelope concentration minimum between 5 and 6.
Pb	No pH relationship
Mn	Upper envelope increases as pH decreases but unusual clusters for AMAX test piles.
Hg	Mostly undetected
Mo	No pH relationship
Ni	Upper envelope Increases as pH decreases.
Se	Upper envelope increases as pH increases. No data for Duluth Complex at low pH.
Ag	Mostly undetected
Tl	Mostly undetected
Zn	Upper envelope increases as pH increases.











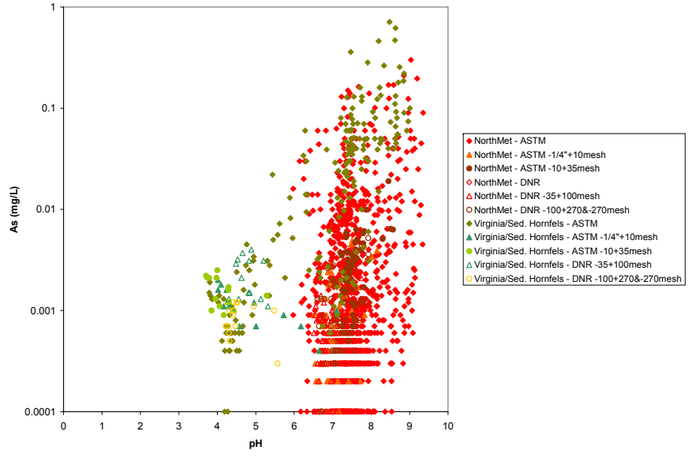


Figure 7-1: Examples of Component Distributions in the Water Chemistry Database

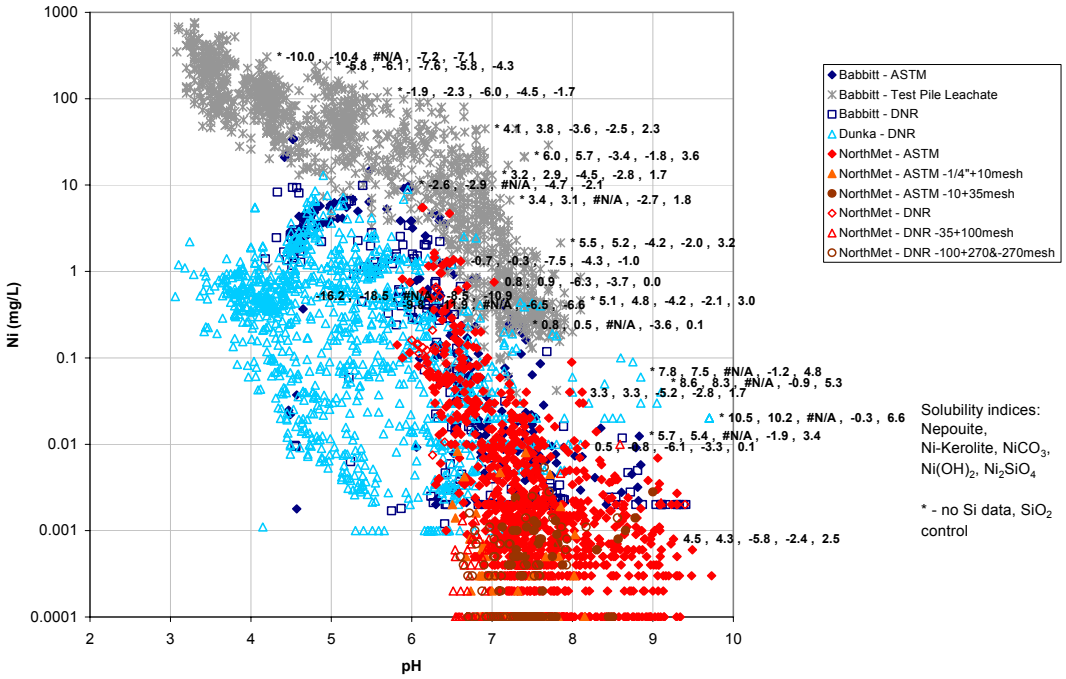


Figure 7-2: Assessment of Solubility for Nickel. Labels show saturation indices for minerals shown in the legend at selected data points.

The data for the AMAX stockpiles did not include silica analyses which were needed to evaluate the solubility of silicates. To simulate silica concentrations, quartz was added as a soluble phase.

Major ion chemistry showed that solubility controls included gypsum, calcite, dolomite, gibbsite and ferrihydrite.

Nickel concentrations lower than the edge of the data scatter (i.e. lower concentrations than the edge for a given nickel concentration) at pHs greater than about 6 tended to be undersaturated with respect to the nickel silicates (nepouite, Ni-kerolite, Ni_2SiO_4). For nickel concentrations near or above the edge, nickel concentrations were over-saturated with respect to the silicates but undersaturated with respect to nickel hydroxide and nickel carbonate.

At pH lower than 6, nickel minerals are highly soluble and no solubility controls were found.

Copper concentrations at pHs greater than 6 for Duluth Complex samples appeared to be controlled by tenorite and malachite. Concentrations were undersaturated with respect to dioptase the only copper silicate in the thermodynamic database. For the Virginia Formation, tenorite appeared to the control.

Zinc solubility for both Duluth Complex and Virginia Formation was predicted to be controlled by silicates ($ZnSiO_3$ and willemite) but not carbonates, hydroxides or oxides. Similarly, cobalt concentrations may be controlled by Co_2SiO_4.

The nickel, zinc and cobalt data therefore indicate a relationship with pH that may be explained by the formation and dissolution of silicates. The strong relationships between metal concentrations and pH provide a reasonable empirical basis for definition of metal solubility controls.

Copper data are consistent with dissolution of well known secondary copper minerals.

Relationships with Source Rock Characteristics

Relationships with source rock characteristics would have been considered if the database included data from a wide range of mine sites. This was not the case, and therefore this was not investigated.

Empirical Relationships

The well-defined upper metal concentration limits indicate that empirical relationships between pH and metal concentrations could be developed for use in predictions.

Selection of Solubility Controls

For the purpose of constraining subsequent water quality predictions, the upper limit of the data scatter for two pHs for each parameter were identified. For non-acidic drainage, a pH value of 8 was selected based on the expectation that drainage chemistry for Category 2 rock will be dominated by alkalinity from the weathering of silicates and relatively low water to rock ratios resulting in formation of secondary carbonate minerals (See Section 6.1.2 and Figure 6-2). For acidic drainage, a pH value of 3.5 was selected to nominally represent strongly acidic conditions. This effectively results in selection of the highest component concentrations in the database.

Maximum observed concentrations at the two pH levels are shown in Table 7-2.

Table 7-2: Maximum Concentrations Indicated at Characteristic pHs and Possible Controlling Minerals

			Maximum Concentrations			
Parameter	Unit	Possible Controlling Minerals	At pH 8	Source	At pH 3.5	Source
Alkalinity	mg/L	Calcium and magnesium carbonates	72.5	NorthMet MDNR Reactor	0	By definition
SO_4	mg/L	Gypsum	2150	AMAX Pile	9600	AMAX Pile
Al	mg/L	Kaolinite, gibbsite	1.68	NorthMet MDNR Reactor	83	AMAX Pile
Sb	mg/L	Fe oxides	0.003	NorthMet MDNR Reactor	0.00001	NorthMet MDNR Reactor
As	mg/L	Fe oxides	0.71	NorthMet HCT	0.71	NorthMet HCT
Ba	mg/L	Barite	0.19	NorthMet MDNR Reactor	0.19	NorthMet MDNR Reactor
Be	mg/L	Fe oxides	0.0002	NorthMet MDNR Reactors and HCTs	0.0023	NorthMet HCT
B	mg/L	Unknown	0.76	NorthMet HCT	0.76	NorthMet HCT
Cd	mg/L	Fe oxides	0.00018	NorthMet HCT	0.0149	NorthMet HCT
Cr	mg/L	Fe oxides	0.0015	NorthMet HCT	0.0015	
Co	mg/L	Fe oxides, cobalt silicates	0.052	AMAX Pile	44	AMAX Pile
Cu	mg/L	Tenorite, malachite	0.092	AMAX Pile	202	AMAX Pile
Fe	mg/L	Fe oxides	0.81	NorthMet HCT	235	NorthMet HCT
Pb	mg/L	Fe oxides	0.0528	NorthMet HCT	0.0528	NorthMet HCT
Mn	mg/L	Mn Oxides	0.75	AMAX Pile	47	AMAX Pile
Hg	ng/L	Fe oxides	6	Low Level Analyses	6	Low Level Analyses
Mo	mg/L	Fe oxides	0.0051	NorthMet HCT	0.0051	NorthMet HCT
Ni	mg/L	Nickel silicates	0.86	AMAX Pile	762	AMAX Pile
Se	mg/L	Fe oxides	0.0029	NorthMet HCT	0.0029	NorthMet HCT
Ag	mg/L	Fe oxides	0.0007	NorthMet HCT	0.0007	NorthMet HCT
Tl	mg/L	Fe oxides	0.00002	NorthMet HCT	0.00006	NorthMet HCT
Zn	mg/L	Zinc silicates	0.09	AMAX Pile	26	AMAX Pile

Mercury

Mercury concentrations were not determined in the MDNR's historical work but characterization work has been completed as part of the overall mass balance for mercury at the site as reported in RS66 (Barr 2007c). This work primarily focused on the ability of geological materials (rock and tailings) to remove mercury from rainfall in the region which typically contains 10 ng/L compared to the Lake Superior basin water quality standard of 1.3 ng/L. Laboratory experiments performed by NTS using Virginia precipitation in contact with Duluth Complex rock showed that mercury concentrations decreased from 12 ng/L to between 1.9 and 3.2 ng/L over 36 days. The control test (no rock) over the same period showed a decrease to 7 ng/L. The results indicate that the rock has the ability to remove mercury from solution. As shown in Table 5-9, mercury concentrations in leachates from waste rock samples were about 6 ng/L. The results imply that the mercury concentrations observed in leachates are indicative of equilibrium concentrations. Table 7-2 shows a source term mercury concentration of 6 ng/L.

8 Stockpile Water Chemistry Modeling

8.1 Objective

The primary intended objective of waste rock characterization was to result in a high level classification of rock into "non-reactive" and "reactive" categories as defined in Minnesota Rules. A "non-reactive" waste is defined as a material that will have leachate that meets the appropriate water quality discharge limits and therefore does not require treatment for discharge to the environment. A "reactive" waste is defined as generating unacceptable water chemistry which could exceed water quality discharge limits.

A secondary objective was the separation of reactive wastes into categories that would define the extent of containment needed to limit leakage from the disposal facilities. For example, waste rock predicted to generate non-acidic water but with unacceptable drainage chemistry would be a low reactivity rock.

This chapter of RS42 presents the evaluation of classification approaches in terms of modeled drainage chemistry.

8.2 Inputs to Water Quality Model

8.2.1 Waste Quantities and Scheduling

Waste Rock Quantities by Potential Classification Categories

In RS78, PolyMet (2007d) provided annual waste rock quantities from the block model and mine plan based on the three sulfur categories indicated in Section 6.4. Sulfur and element content for each category were calculated for components containing sufficient data in the block model. Total tonnages and characteristics for all waste rock produced to Year 20 are shown in Table 8-1.

Table 8-1: Summary of Average Characteristics for Each Category through Life of Mine

Category	Type	Mass US tons	Total S %	Cu %	Ni %	Co mg/kg	As mg/kg	Ba mg/kg	Cr mg/kg	Mn mg/kg	Pb mg/kg	Sb mg/kg	Zn mg/kg
All	All	394,081,962	0.15	0.04	0.02	52	6	67	97	1207	5	4	104
1/2	Waste	327,073,193	0.08	0.03	0.02	52	6	63	97	1224	5	4	100
3	Waste	14,755,777	0.22	0.06	0.03	51	5	59	91	1244	5	4	102
4	Waste	8,892,706	2.38	0.02	0.01	30	34	280	156	601	15	4	285
3	Lean Ore	41,470,125	0.23	0.09	0.04	60	6	60	93	1245	5	4	102
4	Lean Ore	1,890,162	0.88	0.11	0.04	61	6	70	103	1065	5	4	98

Category 1/2 is the largest waste rock component and has an average sulfur content of 0.08%. Annual characteristics indicated that average sulfur concentrations are expected to vary from 0.01% to 0.10%. These concentrations are a result of mixing of rock containing 0.01% to 0.31% sulfur and indicate that the average composition is well below the threshold used to define this category. Category 3 waste rock has average sulfur concentration of 0.22% varying from 0.16% to 0.28% between the criteria of 0.12% and 0.6%. Copper and nickel concentrations are higher than Category 1/2 but concentrations of other elements were comparable. Category 4 waste rock is dominated by Virginia Formation and therefore has an average sulfur concentration of 2.38% with annual averages varying from 0.8% to 2.8%. The effect of Virginia Formation is apparent in the lower copper and nickel content but higher arsenic, lead and zinc content compared to Category 1/2 and 3.

Lean Ore

Two lean ore categories are defined equivalent to the waste rock categories 3 and 4. Category 3 Lean Ore is expected to have similar characteristics to Category 3 waste rock with the exception of higher copper content. Category 4 Lean Ore is quite different from Category 4 waste rock because it consists of Duluth Complex rock containing 0.6% surface rather than Virginia Formation. This is also reflected in the copper and nickel content.

Virginia Formation

Category 4 waste rock is not specifically Virginia Formation but it is dominated but this rock type.

8.2.2 Hydrological Inputs

Annual average infiltration into waste rock was calculated by Barr (2007a) as described in RS21 for Years 1, 5, 10, 15 and 20. Inflow calculations account for progressive reclamation efforts including the placement of evapo-transpiration covers.

Barr (2007) provided two flow scenarios (“low” and “high”). Water quality compliance will be determined by concentrations, therefore all modeling was performed using the low flow scenario because this would result in the highest concentrations.

8.2.3 Physical Inputs

For scale-up calculations, it was necessary to describe the relationship between the particle size distribution used in the humidity cells and the particle size of waste rock, and the proportion of the rock expected to be contacted by infiltration.

All calculations assumed that the rates indicated by humidity cells are equivalent to 20% of the run-of-mine rock mass. In other words, the rate observed in the laboratory from 1 ton of rock is assumed to be seen from 5 tons under field conditions. This reflects the much coarser nature of run-of-mine rock (ROM) compared to the rock crushed to pass a ¼” sieve for humidity cell testing.

The range of possible values is 0 to 100%. Choice of 100% as the factor would require that all ROM rock is finer than ¼".

The proportion of rock assumed to be contacted by infiltration was 50%. The range of possible values is 0 to 100%. Use of 100% as the value would assume that all rock is contacted by precipitation which would ignore channeling effects.

8.2.4 Chemical Inputs

The chemical inputs into the calculations are rates of weathering and rate reductions to reflect temperature. Weathering rates were calculated as follows:

- Average rates (in mg/kg/week) were calculated for each humidity cell test using the period when sulfate release had stabilized.
- The average rates were then grouped according to the numeric geological units (numbered 1 to 6, sedimentary hornfels and Virginia Formation), major rock types (anorthositic, troctolite and ultramafic) and numeric waste rock category (1/2, 3, 4) (as described in Section 6.4.3). No separation was maintained between waste rock and lean ore tests because kinetic test results indicated a continuum regardless of metal content. Lean ore is effectively treated as waste rock.
- The 95th percentile rate for each unit, rock type and category grouping (a total of 28 individual rates) were then calculated. These are groups for which it was predicted more than 1 million tons of rock would be produced. This conservatively assumes that all rock in any particular grouping will function as indicated by the sample containing the highest sulfur concentrations.
- The tonnages of each unit and rock type combination were used to obtain weighed averages for overall rates for each category. The tonnages used are shown in Figure 4-1. A summary of rates is shown in Table 8-2.

Calculations for Category 3 and 4 required the use of acidic rates. For Category 3 rock, none of the test work resulted in acidic pHs. In order to calculate rates under acidic conditions, the MDNR's results for the Dunka Pit (Engstrom 2006a) were reviewed to determine the factor by which release rates increased for this type of rock when leachates become fully acidic. The typical factor was 10 indicating that rates tended to increase by an order of magnitude. This factor was applied to the rates calculated under non-acidic conditions for Category 3 rock. This does not mean that concentrations would only increase by an order of magnitude for acidic conditions because the concentrations calculated are evaluated with respect to the general solubility limits indicated in Table 7-2. For example, nickel concentrations can increase by nearly three orders of magnitude because nickel becomes highly soluble as pH decreases. The same approach was used for Category 3 lean ore.

For Category 4 waste rock, the acidic weathering rates indicated for Virginia Formation and Sedimentary Hornfels were used directly. For Category 4 Lean Ore (which is mainly intrusive rock), the non-acidic rates were scaled using the same approach as Category 3.

The Arrhenius equation indicates that all chemical reactions proceed slower under lower temperatures with the difference in rates determined by the activation energy of the reaction. This finding applies to oxidation of sulfide minerals (SRK and MESH 2007). For pyrrhotite, the range of activation energies is 50 to 60 KJ/mol (Nicholson and Scharer 1998). For NorthMet average site temperature of 2.4°C (Hoyt Lakes), the rate reduction from 20°C (the low end of the laboratory environment) is 0.2 to 0.3 for the range of activation energies. A value of 0.3 would therefore conservatively apply for the NorthMet site. This approach can only be used for low sulfur rock of Category 1/2 because the assumption is that Category 3 and 4 will oxidize rapidly generating excess heat. Category 1/2 rates were therefore reduced by a factor of 0.3, but the rates for Category 3 and 4 were not adjusted.

All rock is assumed to weather at the same rate without limit to the supply of oxygen. Decrease in oxygen availability was not considered, which is conservative. Oxygen limitation could be a factor in Category 3 and 4 waste rock and lean ore piles when oxidation accelerates. It may also be a factor when the waste rock is reclaimed.

Table 8-2: Weighted Averages of 95th Percentile Rates Indicated by Humidity Cells (mg/kg/week)

Category	Acidity (pH=8.3)	Alkalinity	F	Cl	SO_4	Al	As	Ba
2	1.3	7.9	0.027	0.11	2.3	0.087	0.0044	0.0088
3 (non-acidic)	1.4	9.5	0.024	0.14	11	0.052	0.0068	0.0081
4 (non-acidic)	1.4	15	0.03	0.11	11	0.063	0.0044	0.0075
4 (Virginia, acidic)	24	0.88	0.034	0.11	50	0.37	0.00071	0.0052
Ore	1.4	5.2	0.041	0.11	23	0.017	0.0014	0.0059

Category	B	Cd	Ca	Cr	Co	Cu	Fe	Pb
2	0.0039	0.00002	2.2	0.00011	0.000053	0.00085	0.015	0.000063
3 (non-acidic)	0.0054	0.000028	4.7	0.00011	0.0059	0.0084	0.011	0.000069
4 (non-acidic)	0.016	0.000021	3.4	0.00013	0.000086	0.00078	0.03	0.000059
4 (Virginia, acidic)	0.021	0.0032	3.5	0.00012	0.039	0.0048	9.5	0.0011
Ore	0.011	0.000022	7.3	0.0001	0.0028	0.0053	0.0074	0.000076

Category	Mg	Mn	Mo	Ni	Se	Ag	Tl	Zn
2	0.44	0.00096	0.000027	0.00024	0.00011	0.000025	0.00001	0.0013
3 (non-acidic)	0.82	0.023	0.000043	0.07	0.0002	0.000031	0.00001	0.0040
4 (non-acidic)	0.31	0.0033	0.00014	0.0009	0.00042	0.000096	0.00001	0.00069
4 (Virginia, acidic)	3.9	0.12	0.000026	0.56	0.0006	0.000029	0.000012	0.60
Ore	1.5	0.022	0.000034	0.057	0.00012	0.000025	0.00001	0.0021

The delay to onset of acidic conditions for Category 3 and 4 was an input used to semi-quantitatively determine when leachate from the piles would be acidic. For Category 3, the delay was assumed to be 5 years based on the AMAX stock pile data. For Category 4, the rock was assumed to become acidic immediately upon placement.

All rates were assumed to be constant. In reality, depletion of rock components will result in decreasing rates. However, the laboratory results did not provide a basis to predict changes in rates as a function of time and it is conservative from a prediction standpoint to hold rates constant. Total metal content of the rock (Table 8-1) provides a basis to determine when components might run out, in which case the rate would decrease to "zero". For the majority of components of interest, depletion would take centuries and therefore was not a factor in the predictions.

The effect of leaching of explosives residues was based on the explosives recipe provided PolyMet. The overall mixture of emulsion, oxidizer and ANFO (ammonium nitrate and fuel oil) will contain 67% ammonium nitrate. Explosive use will be 0.33 lbs/ton, or 3720 lbs/blast hole. PolyMet have estimated that explosives losses will occur mainly from spillage at the point of loading the blast holes, and that the estimated spillage is 2 lbs/hole, or 0.05% of total explosives use. For safety reasons, PolyMet are committed to re-firing or explosives recovery for blast holes that do not detonate and therefore losses from this source were assumed to be negligible.

8.3 Evaluation of the Effect of Segregation Criteria on Water Quality

These sections of the report discuss the differences in predicted chemistry resulting from the proposed waste rock segregation scheme.

8.3.1 Waste Rock

Effect of Segregation Criteria on Water Quality

Examples of predictions are illustrated in Figure 8-1. Tabulated modeling results are attached in Appendix I.

Sulfate concentrations for all stockpiles were predicted to increase due to increases in mass that are not matched by increasing footprint (i.e. the piles grow vertically rather than laterally). The delayed effect of ARD starting for Category 3 waste rock is shown by the initially slow increase in sulfate, copper and nickel to Year 5. Both Category 3 and Category 4 piles show rapid increases in sulfate reaching a maximum value shown as constrained by maximum observed sulfate concentrations from stockpiles. The Category 3 stockpile shows the 5 year delay until ARD is allowed to develop within the piles.

Although Category 3 rock contains greater nickel concentrations than Category 4 rock (Table 8-1), Category 4 rock is dominated by Virginia Formation which has been shown to leach nickel under laboratory conditions (Table 8-2) and as a result there is no predicted difference between Category 3 and 4 stockpile drainage chemistry.

Recommended Criteria

The predictions indicate that the main benefit of segregation is in producing a large volume of waste rock that has a low potential for ARD (Category 2) from rock that is likely to generate ARD (Categories 3 and 4). The predictions imply that the chemistry of water draining from Category 3 and 4 waste rock will not be different. This reflects the use of common solubility constraints at acidic pHs for both waste categories. In reality, Category 4 waste rock is mainly Virginia Formation rock and will contain higher sulfur concentrations than Category 3. Segregation into these categories is probably beneficial but not readily quantified with the data available.

8.3.2 Lean Ore

Effect of Segregation Criteria on Water Quality

The effect of segregating Lean Ore into Categories 3 and 4 is shown in Figure 8-1. The predicted difference in water chemistry effect is negligible. The segregation results in a small quantity of intrusive Category 4 rock which is continually blended into the ore feed to the plant. This pile will not exist at the end of mine life. The majority of lean ore is classified as Category 3.

Recommended Criteria

Segregation of Lean ore into Categories 3 and 4 is expected to provide an operational benefit in terms of limiting the need to treat ARD though the Category 3 Lean Ore pile drainage will require treatment to address metal leaching.

8.3.3 Deferred Ore

The deferred ore category no longer exists.

8.3.4 Virginia Formation

Effect of Segregation Criteria on Water Quality

Category 4 waste rock is 92% Virginia Formation rock as shown by the sulfur content and distinctive trace element characteristics.

Recommended Criteria

The sulfur criterion of 0.6% between Category 3 and 4 waste rock effectively segregates Virginia Formation rock into Category 4 waste rock because the quantity of waste intrusive rocks containing sulfur concentrations greater than 0.6% is negligible.

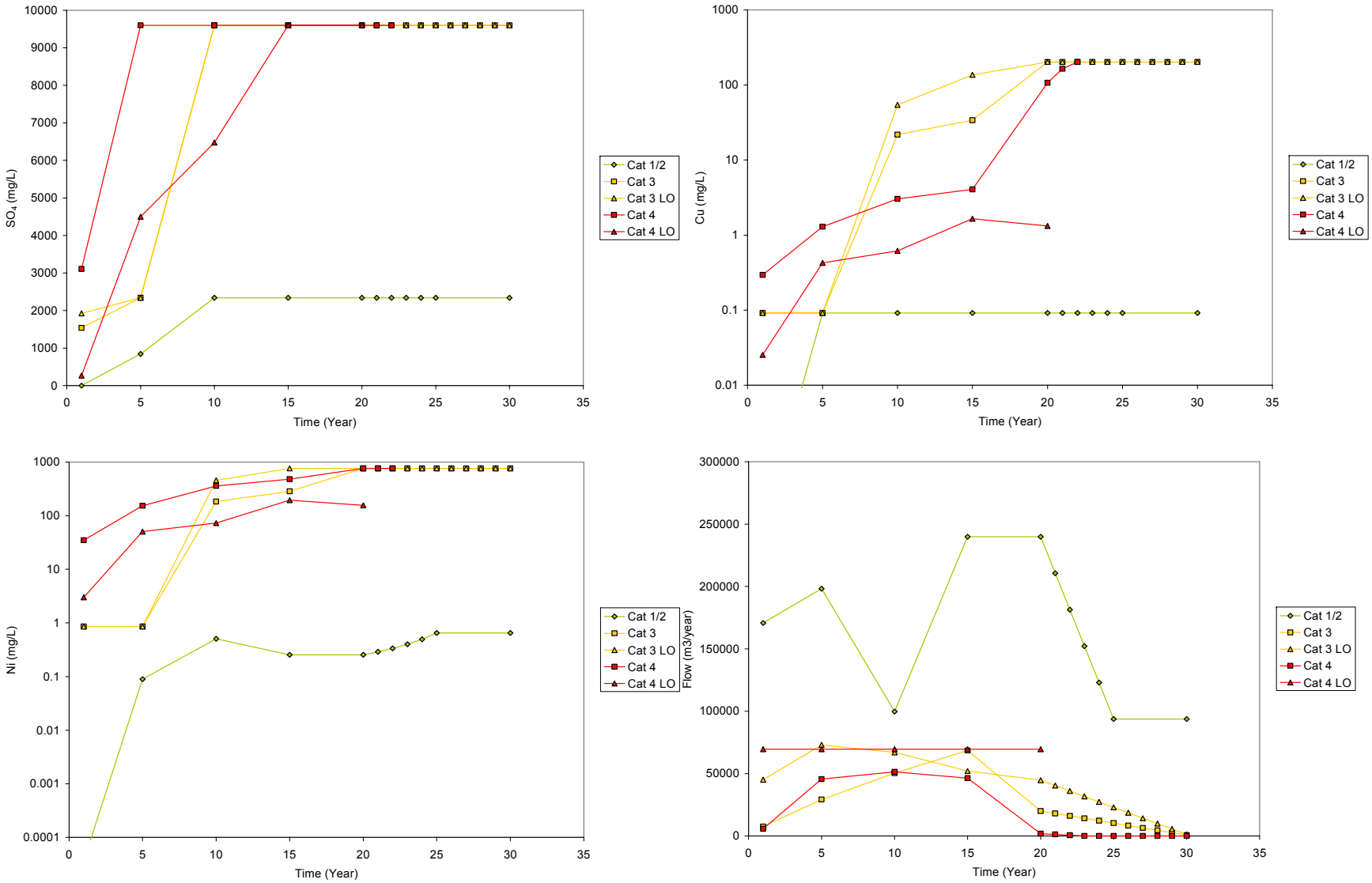


Figure 8-1: Examples of Predicted Chemistry for Waste Rock.

8.4 Water Quality Predictions

This section was originally intended to describe predicted drainage chemistry for the actual waste management. However, project planning has advanced using the proposed categories described above. Section 8.4 provides discussion on Category 1 (non-reactive waste rock) and sensitivity of predictions.

8.4.1 Non-Reactive Waste Rock

Prediction of Water Chemistry with Time

Non-reactive waste rock (Category 1) is defined as rock that will have effluent chemistry below water quality objectives and therefore the drainage is suitable for direct discharge to the environment without treatment. Category 2 was originally intended to satisfy this objective but predictions for Category 2 waste indicated that concentrations can be expected to exceed hardness-based water quality objectives for at least copper and nickel (0.03 and 0.17 mg/L, respectively at highest level of hardness acceptable for use in the calculations), and sulfate (250 mg/L). The Category 2 waste rock is not considered non-reactive by these results.

Consideration was given to a further segregation in Category 2 based on metal and/or sulfur content. As shown in Figure 6-4, average leaching rates are broadly related to metal and sulfur content, but at the low end of the sulfur and metal content of the rock, data are scattered and the ability to develop a significant relationship does not exist. It was concluded that definition of a non-reactive waste rock category based on bulk rock characteristics would not be reliable.

Disposal configuration was considered as a method to produce acceptable drainage quality. Because the calculation of water chemistry is driven by the rate of dissolution and the length of flow path (all other factors considered fixed), the sensitivity of rates used in the metal leaching calculations and the effect of low path length to the chemistry predictions were evaluated.

Decrease in flow path length was considered as an approach that could reduce concentrations primarily to determine if Category 2 type-rock could be used for construction purposes (e.g. road fill). The effect of relatively thin layers on overall water chemistry was evaluated by calculating concentrations for thicknesses of 10, 20 and 30 feet. The calculation also included estimation of hardness using calcium and magnesium concentrations so that the appropriate water chemistry objective could be calculated.

Results of the calculation are shown in Table 8-3. As shown, copper concentrations were predicted to be consistently marginally above the discharge limit. As the thickness decreases, copper concentrations decrease but the hardness is also expected to decrease which further lowers the limit as hardness decreases below 400 mg $CaCO_3$/L.

Table 8-3: Calculated Water Chemistry for Category 2 Rock in Thin Layers

Thickness	Standard or Prediction	Hardness	F	SO_4	Co	Cu	Mn	Ni	Zn
Feet		mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
10	Standard		2	250	0.005	0.021	0.05	0.12	0.27
	Prediction	259	0.9	81	0.002	**0.030**	0.03	0.01	0.047
20	Standard		2	250	0.005	0.030	0.05	0.17	0.39
	Prediction	519	1.9	163	0.004	**0.060**	**0.07**	0.02	0.09
30	Standard		2	250	0.005	0.030	0.05	0.17	0.39
	Prediction	778	**2.8**	244	**0.006**	**0.090**	**0.10**	0.03	0.09

Notes:
1. Bold indicates calculated concentration exceeds the potential discharge limit.

These calculations indicate that the concept of "non-reactive" waste rock cannot be achieved because drainage from waste rock could exceed water quality discharge limits (primarily copper).

Evaluation of Closure Alternatives

Closure alternatives were not evaluated because the concept of non-reactive waste rock has not been advanced.

Sensitivity Analysis

No sensitivity calculations were completed because the concept of non-reactive waste rock has not been advanced.

8.4.2 Reactive Waste Rock

Prediction of Water Chemistry with Time

Waste rock Categories 2, 3 and 4 are considered "reactive" by the definition of predicted drainage chemistry not meeting water quality discharge limits. Chemistry predictions were discussed in Section 8.3.1.

Evaluation of Closure Alternatives

The predictions described above incorporate the closure concepts described in the Detailed Project Description (PolyMet 2007a).

Sensitivity Analysis

The water chemistry predictions are sensitive in a linear fashion to all inputs. If an input factor is increased or reduced by a factor of 50%, the increase or decrease in loading released changes by the same factor. However, many predictions are capped by solubility limits so that changes in inputs do not affect the final prediction. Therefore, the predictions are controlled by the choice of solubility limits which is determined by the water chemistry database. The current predictions use maximum observed concentrations for a given pH and are therefore considered conservative. Use of lower solubility limits would lower the predicted concentrations.

Predictions of drainage chemistry from Category 2 rock are susceptible to the assumption that the overall conditions within the waste rock will remain non-acidic and the composition will reflect rock classified as Category 2 in the block model. Under operational conditions, these assumptions may be affected by the accidental inclusion of small amounts of Category 3 and 4 rock that could become localized sources of acidic water and leaching metals. The effect of these inclusions could be to contribute to metal leaching and lowering of pH resulting in higher concentrations of metals in the drainage. Category 3 and 4 rock could become incorporated into Category 2 rock by a number of routes which could include waste heterogeneity (i.e. small-scale inclusions of Category 3 and 4 rock in Category 2 rock) and operational errors. The latter are factors such as mistakes at the operating face and dumping location. These errors will be minimized by management practices but some level of operational mishaps can be expected.

The ability of Category 2 rock to absorb small quantities of Category 3 and 4 waste rock depends on the generation of excess alkalinity by weathering of silicates. The criteria used to produce Category 2 waste rock are based on an estimate of the critical level at which alkalinity generated by silicate weathering will balance acid produced by sulfide oxidation. All rock below this critical level is assumed to perpetually generate excess alkalinity by silicate weathering. The excess alkalinity then becomes available to neutralize acidity produced by inclusions of Category 3 and 4 rock.

Table 8-4 calculates the ratio of leachate alkalinity from Category 2 rock to acidity from Category 3 and 4 rock in order to achieve ratios of 2 and 1 for mixed alkalinity and acidity. The ratio of 1 indicates an exact balance whereas a ratio of 2 indicates twice much overall alkalinity as acidity. Various levels of alkalinity are shown compared to two levels of acidity. Actual alkalinity is expected to be about 75 mg $CaCO_3$/L from the water chemistry database, whereas acidity is shown for two values (1000 and 5000 mg $CaCO_3$/L) representing the behavior of rock with different levels of sulfur (eg. Category 3 and 4 waste rock). For the expected alkalinity, the tolerable proportion of Category 3 or 4 rock to maintain an alkalinity to acidity rate of 2 is between 1 and 4%.

Table 8-4: Calculation of Category 2 Waste Rock Stockpile Sensitivity to Proportion of Category 3 and 4 Rock Contributing Acidity

Alkalinity from Category 2 Rock	Acidity from Category 3 or 4 Rock	Proportion of Category 3 or 4 Required to Result in Indicated Alkalinity/Acidity Ratio	
mg$CaCO_3$/L	mg$CaCO_3$/L	2	1
50	1000	2%	5%
75	1000	4%	7%
100	1000	5%	9%
150	1000	7%	13%
50	5000	0%	1%
75	5000	1%	1%
100	5000	1%	2%
150	5000	1%	3%

Review of selected drill hole data throughout the deposit indicates that in most cases Category 2 rock occurs separately from Category 3 and 4 rock and is therefore amenable to segregation at a mining scale (Appendix C.2). The ability to maintain non-acidic drainage and therefore limit metal leaching will depend on the implementation of management systems that can ensure errors in placement of rock are limited to a few percent.

8.4.3 Lean Ore

Prediction of Water Chemistry with Time

Chemistry predictions were discussed in Section 8.3.1 and shown in Figure 8.1.

Evaluation of Closure Alternatives

The predictions described above incorporate the closure concepts described in the Detailed Project Description (PolyMet 2007a).

Sensitivity Analysis

Predictions of drainage chemistry for Lean Ore stockpiles are subject to the same sensitivities as all waste rock predictions.

Accidental placement of rock of other categories in the Lean Ore piles is unimportant in terms of drainage chemistry compared to Category 2 waste rock because collection systems are being designed to capture acidic drainage.

8.4.4 Virginia Formation

Prediction of Water Chemistry with Time

Chemistry predictions for Category 4 waste rock were discussed in Section 8.3.1 and shown in Figure 8.1.

Evaluation of Closure Alternatives

The predictions described above incorporate the closure concepts described in the Detailed Project Description (PolyMet 2007a).

Sensitivity Analysis

Predictions of drainage chemistry for Category 4 waste rock are subject to the same sensitivities as all waste rock predictions.

Accidental placement of rock other Categories 2 and 3 in the Category 4 piles are unimportant compared to Category 2 waste rock because collection systems are being designed to capture acidic drainage.

9 Water Chemistry for Subaqueous Disposal Option

9.1 Literature Review

As described in Section 2.2.3, the two principal geochemical issues associated with subaqueous disposal are dissolution of oxidation products formed prior to inundation with water, and continued reaction of the wastes once submerged. SENES (1996) reviewed flooding of pit walls and backfill wastes. Results of that review are summarized below.

SENES (1996) documented over 60 in-pit disposal sites in the U.S. and Canada. This included flooding of pit walls to address wall reactivity, and in-pit disposal of waste rock and tailings. Backfilled pits were identified as being "wet" (i.e. free water following backfilling), "combined" (i.e. backfilling with a cap or cover on the waste) and "dry" (i.e. no water cover) disposal of waste. The latter is a common approach when the open pit becomes part of the waste rock disposal area. Eight sites were listed as disposing of reactive waste rock in open pits resulting in a wet cover. Details on the individual sites were limited but include information for three sites:

- Collins "B", Saskatchewan. This uranium mine disposed of low sulfur (0.05%) rock in a pit. The constituents of concern in the waste rock were arsenic, nickel, uranium, lead-210 and radium-226. Interpretation of the effect of the backfilling on pit water chemistry was complicated by the discharge of waste rock drainage to the open pit.

- Owl Creek, Ontario. At this gold mine, slightly oxidized waste rock exposed for just over a year after ARD was first detected containing several percent sulfur was disposed into a flooding open pit. Limestone was added to the waste during backfilling to offset the effects of leaching of acid salts. Flooding of the waste occurred rapidly. The addition of limestone resulted in pH neutral conditions in the pit water with nickel concentrations between 0.09 and 0.12 mg/L, and copper between 0.01 and 0.06 mg/L.

- Solbec, Quebec. Waste rock was backfilled to a small open pit. No information was available on water chemistry in the pit after flooding.

SENES (1996) conclusions included:

- Pit disposal of reactive waste is becoming common practice;

- Several examples demonstrated that backfilling has resulted in environmental improvements by eliminating or limiting management of contaminant leaching problems.

- Wet covers are the most common approach in Canada. In the US, dry covers are more typical.

- The technology for in-pit disposal is reasonably well developed. The geochemistry, hydrogeology etc. can be determined and models to assess contaminant release and transport are well developed. The design of engineered controls can be reasonably assessed.

MEND (2001) added one additional case history. The Whistle Mine in Sudbury, Ontario was being backfilled with waste rock and final closure was to include a cover.

The Flambeau Mine (Wisconsin) backfilled two types of rock into the open pit at closure (Hill and Benson 2001). Type I rock contained less than 1% sulfur and was not predicted to be acid generating. Limestone was added to Type II rock because it was predicted to be potentially acid generating. Rock types were described as schist. Kuipers et al. (2006) indicate that water in the submerged backfill contained up to 12 mg/L iron, up to 37 mg/L manganese and 1700 mg/L sulfate. No other chemical data were provided.

Literature on waste rock backfilling indicates that it is common approach. Management plans appear to focus on controlling pH in the backfill during disposal to limit leaching effects from oxidation products formed prior to backfilling. Lime and/or limestone may be added to neutralize acidic products. The Flambeau Mine example showed that if a component of the waste rock has natural buffering capacity, addition of extra buffering material is not required.

Oxidation effects from oxygen dissolved in water have not been identified as a significant issue for long term water quality though there is considerable evidence in the literature that subaqueous disposal of reactive wastes significantly inhibits oxidation (MEND 2001).

9.2 Alternatives Considered

9.2.1 Disposal in Existing Pits

Disposal in nearby existing taconite pits is not considered in the Detailed Project Description (PolyMet 2007a). This alternative was not assessed.

9.2.2 Disposal in NorthMet Pits

RS31 (SRK 2007) provides a more detailed discussion of water quality predictions for the open pits. The following sections of RS42 are a summary.

The Detailed Project Description includes backfilling of the East Pit with Category 2 waste rock originating in the West Pit from Years 13 to 20 after East Pit is exhausted. Placement of Category 3 and 4 waste rock in the East Pit has not been evaluated at this time

Various options are presented in RS52 (Barr 2007b) for flooding of the East Pit so it reaches the discharge elevation in 20 years.

Category 2 waste rock would be hauled directly from the West Pit to the East Pit without a stockpiling phase. The backfill would be placed such that waste rock will be flooded very rapidly (within a year of being placed). This will be achieved by placement as lifts from the bottom of the pit rather than end-dumping from a higher elevation in the pit. This approach is being used to minimize exposure of the waste but also to provide access to the Virginia Formation pit walls for application of lime or limestone to limit leaching during filling of the pit. Initial predictions of pit water indicated that pore water chemistry would be elevated as a result of leaching of the oxidized Virginia Formation.

At closure, a wetland will be constructed on the surface of the East Pit backfill.

9.3 Non-Reactive Waste Rock

As described in Section 8.4.1, the concept of non-reactive waste rock has not been advanced for the project.

9.4 Reactive Waste Rock

9.4.1 Soluble Load Release

The literature review indicated that the primary factor considered when backfilling waste rock is the leaching of soluble oxidation products accumulated during weathering prior to subaqueous disposal. The accumulation of weathering products was estimated from the humidity cells by assuming that the weekly rate of release of weathering products accumulates in the rock until it is flushed. The load accumulated in the waste rock before it is inundated is therefore estimated from the time the rock is exposed multiplied by the weekly weathering rate. The calculation assumed that all this load is leached during flooding. RS31 explains in more detail the mechanics of the calculation but the exposure time is calculated by the difference between the time when the rock is flooded and the time when the rock was placed. This is very conservative calculation for metals because it assumes that all load produced is soluble and readily leachable. In fact, solubility constraints will limit leaching and some component of the load will not be leached. Development of reducing conditions in the backfill could be a factor allowing for dissolution of metals associated with ferric hydroxides. Because the closure plan includes construction of a wetland on the backfill, reducing conditions could locally develop in the backfill due to delivery of dissolved organic carbon to the backfill, but the Category 2 waste rock does not contain a reductant that could initiate sufficiently reducing conditions in the backfill to dissolve ferric hydroxides.

The calculation used is conservative but has not been refined. The load originating from the backfill (both the component briefly exposed above the water level and the flooding component) were found to be small compared to load leached from the Virginia Formation pit walls. The pit water predictions were used primarily to evaluate the effect of measures to limit the effect of the Virginia Formation.

9.4.2 Underwater Weathering

Because underwater weathering has not been documented as a significant effect for subaqueously disposed waste rock and the rate of oxidation under these conditions is likely to be very slow compared to leaching of the exposed high walls, this effect was not considered. Backfilled rock was assumed to not oxidize once flooded.

9.4.3 Prediction of Water Chemistry as a Function of Time

RS31 provides results of overall predictions for the East Pit during flooding. As indicated above, the rapid flooding of slowly weathering Category 2 rock with minimal exposure resulted in insignificant effects on the chemistry of backfill pore water quality.

9.5 Further Investigations

Under the current proposal provided in the Detailed Project Description, no additional studies of the effect of subaqueous disposal are recommended because the disposal of Category 2 rock in the East Pit has been predicted in RS31 to have a minor effect on overall loadings from the pit when compared to other sources. In the event that other disposal methods are proposed, or other categories of rock are considered for disposal, specific investigations should be considered.

10 Conclusions and Recommendations

An extensive geochemical characterization program has been completed for waste rock, lean ore and ore at the proposed NorthMet Project. These data and data obtained by the MDNR over nearly two decades of testing indicate the following:

- The concept of "non-reactive" waste rock cannot be defined when drainage from waste rock is required to meet stringent water quality discharge limits. Even thin waste rock placement containing low levels of sulfide mineralization may produce drainage chemistry exceeding the limit for copper (in particular) because the water quality standards are hardness based and result in low water quality discharge limits.

- Three categories of reactive waste rock have been defined using sulfur content as a primary criteria and copper to sulfur ratios as a secondary criterion.

- Category 2 waste rock has been defined as having low potential to generate acid rock drainage (ARD). This rock will have a sulfur content of less than 0.31% or 0.12% for lower and higher copper to sulfur ratios, respectively.

- The sulfur criteria for Category 2 were developed by considering threshold effects for consumption of acid by alkalinity generated from silicate weathering. The thresholds obtained are completely consistent with the MDNR's long term testwork which shows that rock containing less than 0.41% sulfur did not generate ARD for 18 years.

- Category 3 waste rock has been defined as rock with potential to generate ARD but with a delay of several years. The sulfur content of this rock is greater than Category 2 and less than 0.6%. The category was developed primarily with reference to the field scale test work conduced on the Babbitt Deposit.

- Category 4 waste rock has been defined as rock with a potential to generate ARD immediately upon exposure. This waste rock has sulfur content above 0.6% and is dominated by the Virginia Formation metasediments located in the footwall of the Duluth Complex at NorthMet.

- Lean Ore has been divided into Categories 3 and 4. Category 2 does not exist for Lean Ore. Both Lean Ore categories are intrusive rock hence Category 4 lean ore has lower sulfur content than Category 4 waste rock.

- Conservative water quality predictions for three waste rock stockpiles and two lean ore stockpiles were developed by scaling-up laboratory weathering rates and considering solubility limits indicated by review of water chemistry database assembled testwork and field monitoring data obtained by MDNR and PolyMet.

- All chemistry predictions indicate that drainage from the waste rock and lean ore stockpiles will exceed water quality discharge limits for several parameters including sulfate, nickel and copper.

- Drainage chemistry for Category 2 waste rock is sensitive to the inclusion of Category 3 and 4 rock. If the proportion of these potentially ARD producing categories are included in the Category 2 rock, the main concern is that the overall drainage could be acidic and result in increased metal solubility. Segregation errors may occur during mining due to operational practicalities. A target error rate of less than 2% is recommended.
- Rapid subaqueous disposal of Category 2 rock in the East Pit after year 12 is not expected to have a significant effect on pit water chemistry compared to the effects of leaching of walls composed of Virginia Formation.

The findings of RS42 have been used to provide input chemistry to the prediction of drainage treatment (RS29, Barr (2007)) and the Waste Management Plan (RS43).

The ongoing waste rock, lean ore and ore characterization program currently consists of more than 100 individual tests. Substantial reduction in this test program is recommended. A few tests should be continued to support long term predictions.

This report **"1UP005.01 – RS53/RS42 Waste Rock Characteristics/Waste Water Quality Modeling – Waste Rock and Lean Ore - DRAFT"**, has been prepared by SRK Consulting (Canada) Inc.

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Executive Summary

PolyMet Mining Inc (PolyMet) is proposing to develop the NorthMet Project (former Dunka Road Project of US Steel) near Babbitt, Minnesota. As a part of the Minnesota Department of Natural Resources (MDNR) "Permit to Mine" process a complete "mine waste characterization" is required (Minnesota Rules Chapter 6132.1000). This document describes geochemical characterization of tailings from other nearby mineral deposits by the MDNR, geochemical characterization of NorthMet Project tailings produced by PolyMet pilot plant testing and water quality predictions.

PolyMet's characterization plan was developed in close consultation with the MDNR and considered results from the MDNR's historical testwork. The MDNR has tested tailings produced by pilot plants run on ore grade rock from the Babbitt Deposit. This testwork is ongoing and showed that tailings containing 0.2% sulfur did not generate acidic leachate after four years. The protocol used for testing appeared to have a strong influence on the results, a finding that has been incorporated into predictions of the geochemical performance of the tailings under site conditions.

PolyMet's characterization program included bulk chemical tests, mineralogical evaluation and kinetic dissolution tests produced by pilot testing of three ore composites. The pilot testing evaluated the use of copper sulfate to enhance flotation of pyrrhotite. As a result, four bulk tailings containing a range of sulfur concentrations (0.10 to 0.23%) were available for geochemical testing. Size fractions of the tailings were also produced so that the different characteristics of tailings dams, beaches and slimes could be assessed. As of March 23, 2007, kinetic testing on NorthMet Project tailings has been underway for 52 to 80 weeks.

Interpretation of the MDNR's testwork on both rock (reported in RS42, SRK 2007b) and other tailings, and PolyMet's testwork on NorthMet Project tailings indicates that tailings containing less than 0.2% sulfur will not generate acidic leachate despite the absence of carbonate minerals. Long term weathering behavior can be predicted by assuming that weathering of abundant silicate minerals in the tailings results in generation of bicarbonate alkalinity, a conventional weathering process. This alkalinity perpetually offsets the acid produced by sulfide mineral oxidation.

Leaching of nickel appears to be sensitive to pH near 7. The MDNR's testwork on bulk tailings in small reactor tests and PolyMet's humidity cell testwork on coarse (>200 mesh fraction) tailings has shown that nickel and cobalt leaching accelerates when pH falls below 7 due to re-leaching of weathering products formed at higher pH. PolyMet's coarse tailings are likely susceptible to moderate pH depression possibly because the silicate mineral particles have less surface area than the fine (<200 mesh) tailings. Bulk tailings and fine sand tailings have not shown accelerated leaching of nickel.

Testing on LTV Steel Mining Company (LTVSMC) taconite tailings showed that the carbonate content of these tailings makes them more leachable in terms of major (calcium, magnesium and alkalinity) ions than the NorthMet Project tailings. This indicates that the LTVSMC tailings produce hard groundwater with capacity to neutralize acid. Nickel leaching from NorthMet tailings is

expected to exceed that of the LTVSMC tailings but the LTVSMC tailings appear to have capacity to attenuate nickel.

The leaching behavior of tailings has been predicted by coupling models of moisture content, oxygen profile and sulfide mineral oxidation. These models have shown that coarser tailings in the dam embankments and beaches near the embankments will be well oxygenated due to the low moisture content and particle size of the tailings. The low sulfide mineral content is also a factor because oxygen is not consumed near the surface as is commonly observed in tailings at other sites containing higher levels of sulfide minerals. Seepage chemistry was predicted using water balance information coupled with the tailings facility design which includes progressive placement of synthetic membranes on the downstream slopes of the dams and a system of horizontal drains to collect seepage.

Water in the Cell 2E horizontal drain system was predicted to have metal concentrations below most stringent water quality standards. PolyMet is not proposing a point discharge but the water quality standards would be the minimum discharge limits and are used as water quality objectives for the project. Sulfate is predicted to exceed the secondary drinking water standard (for odor and taste) during the operational period and again in about mine Year 60.

Water collecting in the Cell 1E horizontal drains is predicted to exceed the water quality standards in the later part of operations until Year 37. This water will be returned to the tailings ponds during operation and pumped to the Mine Site at closure until it reaches quality such that collection is no longer required.

Water collected at the Cell 1E seepage recovery barrier was predicted to contain sulfate concentrations above the secondary drinking water standard during the same period. Cobalt concentrations were also slightly above the Class 2B surface water quality standards. This water will be returned to the tailings ponds during operation and pumped to the Mine Site at closure until it reaches quality such that collection is no longer required. Attenuation by contact with overburden and soils was not included in the evaluation and would be expected to reduce concentrations to below applicable water quality standards.

Operational process water quality in the tailings pond was predicted using a mass balance approach for all inputs and outputs. Water quality in the pond was predicted to meet most surface water quality standards by controlling the chemistry of the Mine Site Waste Water Treatment Facility (WWTF) effluent to the pond.

All uncollected seepage to groundwater that is not collected in the seepage collection system will flow north and northwest toward the Embarrass River. This uncollected seepage was not predicted to exceed applicable water quality standards.

Manganese, silver and thallium concentrations were universally predicted to exceed their water quality standards. Manganese is naturally elevated in local groundwaters and surface waters and often exceed the manganese secondary drinking water standard. The latter two parameters are affected by the use of analytical data with detection limits at or above the standards for natural groundwaters and surface waters.

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Report CD: Data Listing

1 Introduction

1.1 Background

PolyMet Mining Inc (PolyMet) is proposing to develop the NorthMet Project (former Dunka Road Project of US Steel) near Babbitt, Minnesota. As a part of the Minnesota Department of Natural Resources (MDNR) "Permit to Mine" process a complete "mine waste characterization" is required (Minnesota Rules Chapter 6132.1000).

The potential water quality issues associated with tailings disposal at the NorthMet Project are expected to include acid rock drainage (ARD) and leaching of some heavy metals from the tailings themselves, build-up of solutes in the tailings impoundments due to re-circulation of process water, and effects resulting from interaction between tailings pore waters and the underlying LTV Steel Mining Company (LTVSMC) taconite tailings.

The characterization program for tailings was designed based on extensive consultations with staff at the MDNR resulting in implementation of "Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan NorthMet Project, Minnesota" in May 2006. For reference, Hydrometallurgical Residue Geochemical Characterization is addressed in other documents, RS33 and RS65 (SRK 2007a).

Inputs to the overall water quality predictions for the tailings pond came from a variety of sources detailed elsewhere including pilot plant testwork (RS32, Barr 2006a), tailings pond water balance (RS13, Barr 2007c), tailings pond operation (RS39/40T, Barr 2007a) and mine water treatment testwork (RS29T, Barr 2007b). This report combines the original scopes of RS54 (reporting on tailings characterization) and RS46 (tailings water quality model).

1.2 Objective

The specific objectives of this characterization program included:

- Assessment of the overall reactivity of the tailings solids as required in Minnesota Rules Chapter 6132.1000.
- Development of mass-loading rates for input into water quality predictions for impact assessment and mitigation design.

1.3 Design and Consultation Process

The tailings characterization plan was developed in consultation with staff from the MDNR. The consultation included the following steps:

- June 22, 2005. A draft of the plan was prepared for MDNR Review.

- July 21, 2005. MDNR provided initial comments focused primarily on flotation tailings.
- September 9, 2005. SRK responded to the July 21, 2005 letter.
- September 14, 2005. A conference call was held to discuss the July 21, August 16 and September 9 letters.

The process was begun in June 2005 and concluded with full implementation of "Flotation Tailings and Hydrometallurgical Residue Geochemical Characterization Plan NorthMet Project, Minnesota" in May 2006.

1.4 Structure of Report

This report combines results of two studies. RS54 provides results of characterization of tailings. RS46 is the prediction of leachate chemistry and resulting water chemistry of seepage collected and seepage escaping the facility.

The structure of the RS46 report, which was a combination of RS54 and RS46 was agreed with the MDNR. The final version of the report outline was transmitted to the MDNR on April 26, 2006. The agreed outline has been followed although in places this does not fit with the development of thinking on the project. If any sections are redundant, the section heading is shown with a brief note to explain why the section is no longer relevant.

Information on hydrometallurgical residue characterization and water quality is presented in RS33 and RS65 (SRK 2007a)

2 Water Chemistry Prediction Methods

2.1 Introduction

The final products of this study were chemistry predictions for the tailings pond water, and tailings pore water during operation and closure. The prediction methods to be used were considered as part of the design of the characterization program so that the program would provide the inputs needed to predict water chemistry. The following sections broadly describe the types of modeling approaches considered and the method selected.

2.2 Theoretical Method

Theoretical models for tailings and tailings basins need to couple physical and chemical processes in the tailing profile which include oxygen diffusion, saturated and unsaturated water flow, oxidation of sulfide minerals, dissolution of buffering minerals, oxidation/reduction reactions, ion exchange, and secondary mineral precipitation in addition to basin dynamics.

No single theoretical model has been developed probably because coupling the processes is complex. Models such as RATAP (Scharer et al. 1994) and MINTOX (Wunderly et al. 1996) are best described as "Engineering Models" because they have limited power to reliably predict water quality but provide a good basis to evaluate engineering solutions.

2.3 Analog and Empirical Methods

2.3.1 Analog

The analog method involves direct comparison of the proposed project with existing similar mines. Two analogs were considered. The first would use pond water chemistry from other sites to predict the chemistry of the ponds at the NorthMet Project; however, as ore processing, tailings placement and basin hydrodynamics are unique to every site, direct analogs for tailings management facilities are not easily found.

The second analog would compare the geochemical performance of the tailings solids expected to be produced at NorthMet with the tailings solids from other similar sites. This is a more appropriate comparison because the number of variables or factors affecting geochemical behavior is lower than for the pond water. As will be described below, the MDNR has performed testwork on tailings generated from evaluations of the nearby Babbitt Deposit which is in a similar geological setting.

2.3.2 Empirical

The empirical method uses scale-up of laboratory tests to predict the geochemical performance of the tailings solids. Further explanation of this approach can be found in RS42 (SRK 2007b).

2.4 Method Selected

2.4.1 Tailings Leaching

The method selected to predict the leaching behavior of tailings involved components of the theoretical approach combined with empirical or scale-up methods. The progress of oxidation and pore water chemistry in this case was predicted by combining oxidation rates indicated by laboratory tests with calculation of oxygen diffusion consistent with the moisture content and particle size distribution of the tailings (for example, SRK 2004).

2.4.2 Operational Process Pond

The method used to predict operational process pond water quality is coupled with the tailings leaching prediction but itself is primarily empirical. Experience shows that water quality during closure can be accounted for by carefully tracking all sources and calculating the total masses entering and leaving the tailings impoundment. This is very commonly used approach for prediction of pond chemistry at new and operating mines (e.g. Red Dog Mine – Teck Cominco Alaska; Ekati Diamond Mine - BHP Billiton Diamonds; Kemess North Project - Northgate Resources; Galore Creek - Novagold Resources; Ruby Creek Project – Adanac Molybdenum Corporation).

3 Program Design

The following sections describe geological, geochemical and mine design background considered in the design of the geochemical characterization program for the tailings solids.

3.1 Geological Background

Detailed description of the geological setting of the deposit is provided in ER03 (PolyMet 2007).

The NorthMet Deposit is located in the intrusive Duluth Complex of northern Minnesota. Disseminated copper-nickel-iron sulfides (chalcopyrite, cubanite, pentlandite and pyrrhotite) with associated platinum group element (PGE) mineralization will be extracted from several igneous stratigraphic horizons.

In the vicinity of the NorthMet deposit, the Duluth Complex intruded and assimilated the Virginia Formation, which consists of argillite and greywacke with minor interbeds of siltstone, graphitic argillite, chert, and carbonate. This formation is the stratigraphic footwall of the NorthMet deposit, but also occurs as xenoliths ("inclusions") within the deposit.

Processing of the ore by flotation will result in removal of sulfide minerals to produce tailings composed almost entirely of silicates. Small quantities of residual sulfide minerals will remain.

3.2 Geochemical Background

The tailings characterization program was designed based largely on experience extrapolated from waste rock characterization conducted by the MDNR beginning in the 1970s (Lapakko et al 2001; MDNR 2004a). The MDNR's waste rock testwork showed that sulfur content is the primary variable controlling pH of leachate, delay to onset of acidic leachate, oxidation rates, and metal release rates (Lapakko 1993; Lapakko and Antonson 2006).

The MDNR has completed some testwork on tailings generated in the past from processing of the Babbitt Deposit. These results are summarized in the following sections.

3.2.1 AMAX Tailings Test Plot

In 1978, a field test plot was constructed containing tailings produced by processing of ore extracted from a test shaft (MDNR 2004b). The resulting tailings contained 0.38% sulfur (i.e., over two times the planned concentration in NorthMet tailings). The tailings plot was monitored for 3 years during which time leachate pH was generally above 7 but occasionally between 6 and 7 during high runoff periods. Flow-weighted average sulfate concentrations were near 2000 mg/L, copper concentrations were near 0.05 mg/L and nickel concentrations varied from 0.03 to 0.3 mg/L. Lower nickel concentrations were generally found when pH dropped below 7. Based on review of the data, SRK

found that drainage yields from the pile were very low which strongly suggests that drought conditions and excessive evaporation may have had a strong influence on water chemistry.

3.2.2 Cominco Tailings

In 2002, Cominco Ltd. produced tailings from processing of ore from the Babbitt Deposit (MDNR 2004b). The resulting tailings contained 0.2% sulfur. The MDNR is conducting eight laboratory kinetic tests on the samples in three different configurations (ASTM, MDNR Reactor with no lid (uncovered) and MDNR Reactor with lid (covered). Leachate chemistry data were provided by Folman (2006a,b). All samples produced leachates that had pH greater than 6.5 but the reactors with lids had lowest pHs (6.5 to 7) compared to without lids (pH 7) and ASTM (7.5 declining to 7) which correlated with greatest sulfate production and nickel release for the reactors with lids (Figure 3-1). Nickel release was lowest for the ASTM test. SRK calculated that 29%, 38% and 55% of sulfur, were depleted from the ASTM, no lid and lid tests, respectively after four years of testing. Copper leaching was hardly detected (Figure 3-1).

3.3 Summary of Impoundment Construction and Operation

Details of impoundment construction and operation are provided in RS39/40T (Barr 2007a). Tailings will be deposited in an existing taconite tailings basin previously operated by LTVSMC. Tailings slurry will be discharged from multiple spigot locations. Natural particle size segregation will occur resulting in deposition of coarse particles near the dams and fine particles in the center of the impoundment. Coarse tailings will be used for ongoing construction of the impoundment using the upstream construction method. Initial deposition will occur in Cell 2E. After 8 years, deposition will begin in the combined Cells 2E and 1E. Downstream faces of tailings dams will be progressively covered with a geosynthetic material to limit infiltration and oxidation of coarser tailings used for dam construction. A system of horizontal drains will be installed in part to allow collection of water that may contain concentrations of regulated parameters above minimum water quality discharge limits.

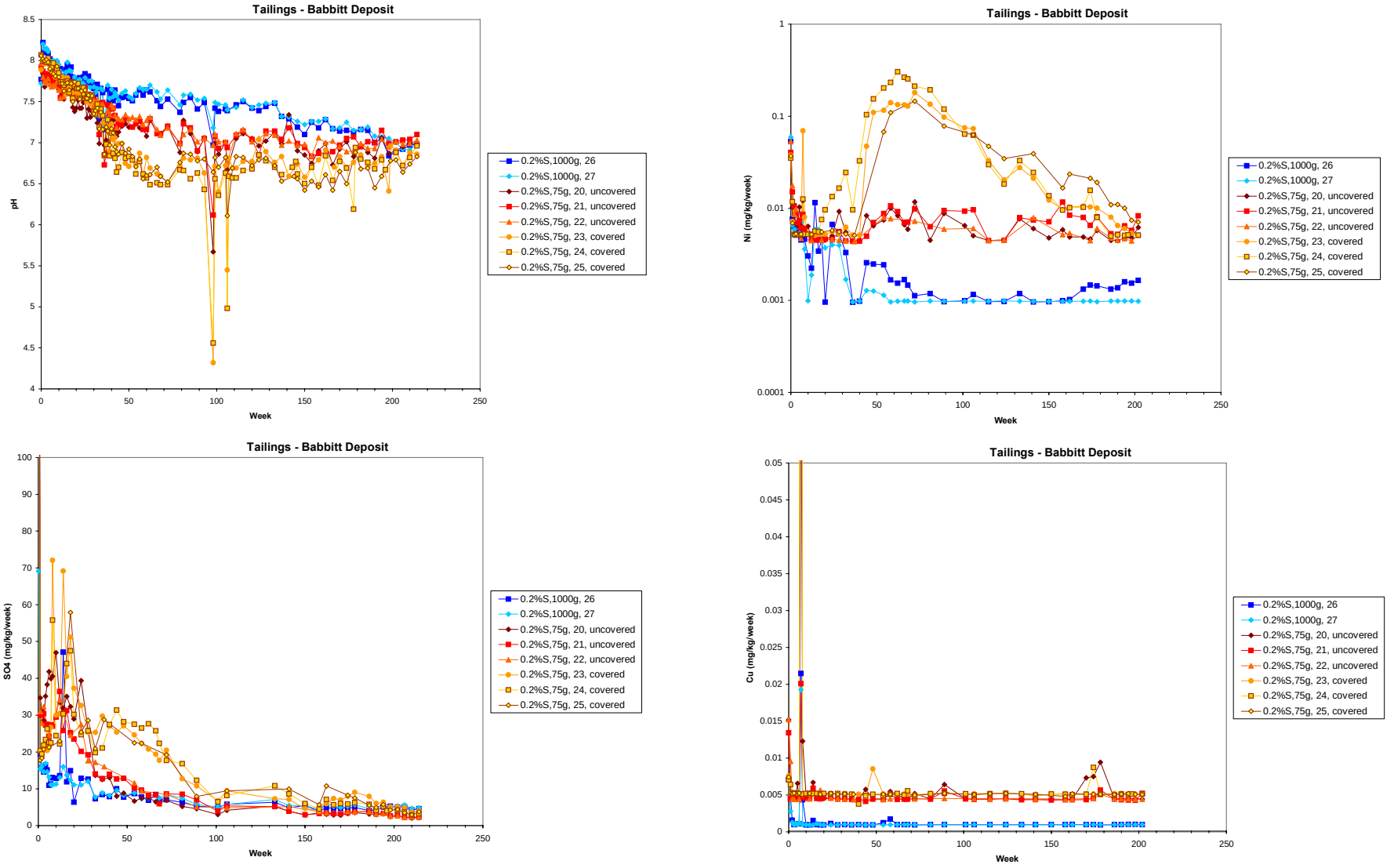


Figure 3-1: Results of MDNR Testing on Cominco Tailings

3.4 Data Requirements

The overall objective of the waste characterization test program was to evaluate the leaching characteristics of the tailings to provide input into prediction of tailings pond water quality and seepage during operation and closure of the impoundments.

3.5 Overall Program Design

The overall characterization program was designed to evaluate the weathering processes that can be expected to occur in exposure of bulk tailings and tailings of different particle sizes as they are deposited. The MDNR's experience has clearly demonstrated that sulfur concentration is an important variable controlling waste rock reactivity and therefore the tailings program was designed to evaluate the effect of sulfur content on reactivity. This was achieved by obtaining tailings samples produced by processing three ore composites with and without the use of copper sulfate to activate the sulfide mineral surfaces to enhance flotation (RS32, Barr 2006a). Tailings produced without copper sulfate contained higher sulfur concentrations than those produced with copper sulfate. The resulting bulk tailings samples were then sieved to obtain three tailings particle size fractions representing coarse to fine sands (+100 mesh, -100+270 mesh and -270 mesh).

An additional factor that could not be evaluated by the MDNR's testwork is that PolyMet intends to deposit the tailings on existing tailings produced by the former LTVSMC taconite operation. Samples of taconite tailings were collected by core sampling to evaluate the interaction between leachates from the NorthMet tailings and the taconite tailing.

4 Sampling and Analytical Methods

4.1 Metallurgical Program

4.1.1 Ore Composite Preparation

Preparation of the three ore composites or parcels (P1-low grade ore, P2-mid grade ore and P3-high grade ore) is described in ER03 (PolyMet 2007).

4.1.2 Production of Tailings Samples

Bulk Tailings

Tailings samples were generated from the three ore parcels using a pilot plant over a three-week period in July and August 2005. Results of environmental sampling of the pilot plant are described in RS32 (Barr 2006b).

Every 2 hours, samples of bulk tailings were collected and analyzed for sulfur content to evaluate the variation of sulfur concentration that could be expected during full-scale production and the degree to which the resulting bulk samples would represent this variability. Results of the sulfur analyses are shown in Figure 4-1.

The trend in sulfur results in tailings are explained by the chronology of the testwork and evaluation of addition of copper sulfate as a reagent:

- Flotation testwork began on July 17 with Parcel 2 without the use of copper sulfate. Parcel 2 was processed entirely without using copper sulfate. As shown, sulfur concentrations varied from 0.05% to 0.25% reflecting adjustment of the process conditions early in the testwork. The average was 0.19%. The composite of Parcel 2 tailings had a sulfur content of 0.2% closely representing the average.

- Testwork continued with Parcel 1 without using copper sulfate. Processing was continuous so one point is shared between Parcel 2 and Parcel 1. The range of sulfur concentrations was 0.19% to 0.28% with an average of 0.24%. The composite sample was 0.23% and is close to the average of the two-hour samples.

- Pilot plant testwork was suspended on July 19 to allow for further bench scale testing on reducing sulfide in tailings.

- The pilot plant resumed on August 8 using Parcel 1. Addition of copper sulfate was evaluated. The effect of copper sulfate on tailings characteristics was immediately apparent for Parcel 1. Total sulfur concentrations decreased to a range of 0.09% to 0.13% (average 0.1%) and the resulting tailings composite was 0.1%.

- Processing continued with Parcel 3 using the copper sulfate additive. Sulfur content of the tailings varied over a wider range (0.09% to 0.25%, average 0.18%) though the range was comparable to the total range indicated by processing of other ore packages. The resulting composite had a total sulfur content of 0.15%.

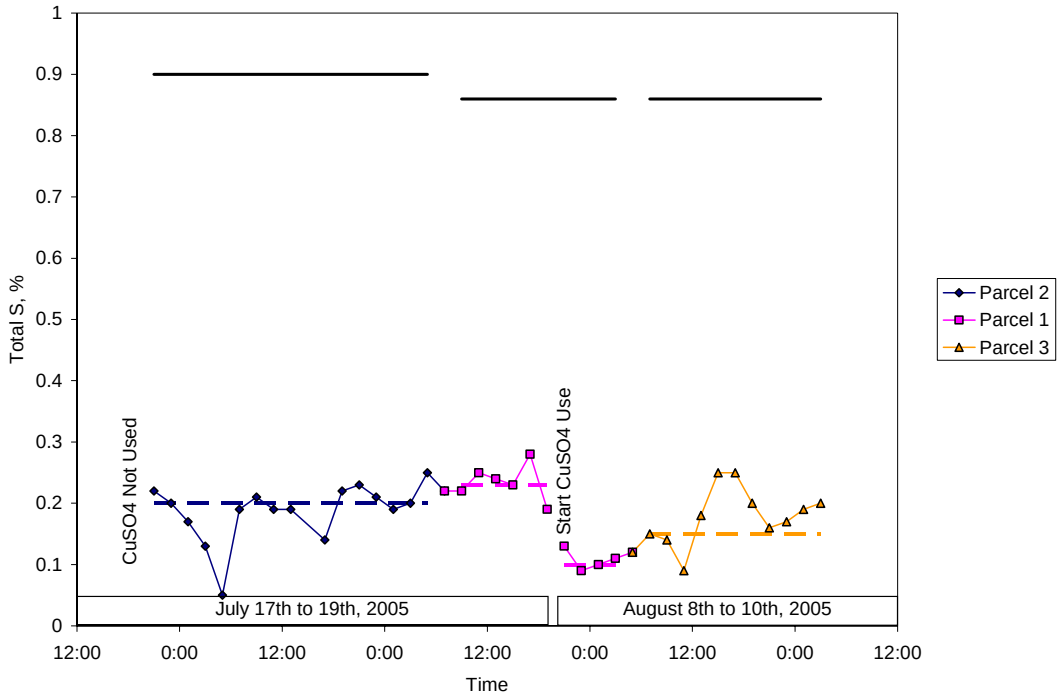


Figure 4-1: Results of 2-Hourly Total Sulfur Analyses. Solid lines and points connect 2-hourly results. Broken solid lines are sulfur concentrations in composite tailings samples representing each stage of testwork. Solid horizontal lines at the top of the graph are the respective ore composite sulfur contents.

In summary, the 2005 Pilot Plant processing resulted in four bulk tailings samples:

- Parcel 2 tailings produced without using copper sulfate.
- Parcel 1 tailings produced without using copper sulfate.
- Parcel 1 tailings produced using copper sulfate.
- Parcel 3 tailings produced using copper sulfate.

The process testwork showed that sulfur concentrations in the tailings can be expected to vary in response to changes in process conditions including the use of copper sulfate. The full scale plant design includes copper sulfate addition. Pilot testing in 2006 using a pilot plant configuration that represents the final plant design demonstrated that full scale plant tailings are expected to be much less than 0.2%S. The average sulfur concentration was 0.12% (range 0.10% to 0.13%).

The lack of copper sulfate for the Parcel 2 and initial Parcel 1 samples may have significantly affected the subsequent reactivity of the residual sulfide minerals in the tailings because copper sulfate would improve flotation of liberated pyrrhotite mineral grains. For the latter part of Parcel 1 and all of Parcel 3, residual sulfide may remain as unliberated or encapsulated grains. These factors indicate that similar sulfur concentrations in samples produced with and without the use of copper sulfate may not represent equivalent reactivity. This particularly affects the evaluation of coarse fraction reactivity.

Tailings Fractions

Initial designs for the tailings impoundments included the use of tailings sands produced using cyclones for construction of the dams. To evaluate the characteristics of the tailings produced from cycloning, bulk tailings samples were screened to produce three size fractions representing the dam material (+100 mesh), beach (-100+200 mesh) and fine sands (-200 mesh). The cycloning concept was not carried through to the Detailed Project Description but instead the dams will be constructed using coarser tailings resulting from natural settling of tailings discharged from spigots. This difference in construction method indicates that the coarse tailings will contain some material finer than 100 mesh entrained during settling and therefore that differences in leaching behavior between the ideal fractions will be less apparent under field conditions.

4.2 LTVSMC Tailings Characterization Program

4.2.1 Drilling

The LTVSMC taconite drilling program plan is described in Appendix A.

Seven holes were drilled in the tailings to a depth exceeding 60 feet using a geoprobe. Mostly continuous samples were obtained from five holes with tailings having the following textural characteristics:

- GP-1 – Mainly coarse sand.
- GP-2 – Interlayered fine sand and slimes.
- GP-3 – Coarse sand grading into fine sand and slimes.
- GP-4 – Interlayered coarse and fine sands.
- GP-5 – Interlayered fine sand and slimes.

Samples were obtained as core and shipped whole to Canadian Environmental and Metallurgical Inc. (CEMI).

4.2.2 Existing Water Quality and Laboratory Testing Data

Existing groundwater and seepage chemistry data for the LTVSMC tailings cells that are proposed for disposal of NorthMet tailings are described in RS64 (Barr 2006d). Data distribution statistics are provided in Table 4-1. Average concentrations were calculated using the absolute value in the case of results reported at the detection limit. The majority of analyses were performed on "unfiltered" samples and therefore concentrations may reflect the presence of suspended matter. Total suspended solids were detected (average 6 mg/L and 11 mg/L for cells 2E and 2W, respectively but with maxima of 47 and 672 mg/L, respectively) which showed that total concentrations can be expected to exceed concentrations in 0.45 μm-filtered waters.

Waters in the impoundment area are bicarbonate-Ca-Mg-Na but sulfate is also an important anion. Fluoride is a minor anion occurring at elevated concentrations compared to typical groundwaters (averaging 2.3 and 3.0 mg/L in Cells 2E and 2W, respectively). Iron concentrations were relatively low (averages of 0.05 and 0.07 mg/L). As shown, manganese concentrations averaged 0.9 and 0.4 mg/L, respectively for the two cells.

Berndt et al. (1999) characterized the LTVSMC tailings as part of a study to evaluate in pit disposal of taconite tailings for several mines. They found that the taconite tailings were composed mainly of quartz (44 to 51%), hematite (3 to 17%) and talc (3 to 12%). Siderite and ankerite were present at concentrations ranging from 4 to 9% each. Elevated fluoride concentrations were believed to be due to the use of wet scrubbers for control of particulate emissions from induration furnaces. The wet scrubbers also remove highly soluble HF gas from the furnace exhaust, resulting in elevated fluoride concentration in the scrubber water.

Table 4-1: Summary Statistics for Groundwater Chemistry in the LTVSMC Tailings Basin

Location Code	Bicarbonate	As	Ba	Be	B	Ca	Cl	Co	Cu	F	Hardness	Fe*	Mg	Total Mn	Mo	Ni	pH	K	Na	SO_4	Zn
	mg/L	µg/L	µg/L	µg/L	µg/L	mg/L	mg/L	µg/L	µg/L	mg/L	mg/L	mg/L	mg/L	mg/L	µg/L	µg/L	su	mg/L	mg/L	mg/L	µg/L
Cell 2E																					
n	67	8	8	1	110	100	9	74	72	110	100	55	100	77	110	61	32	62	72	120	8
Min	270	<2	47	<0.2	85	18	19	<3	<2	<0.1	110	<0.03	15	0.02	<5	<2	6.4	6.2	34	8.1	<10
P5	400	<2	49	<0.2	240	50	20	<1	<2	0.19	370	<0.03	57	0.05	5.5	<2	7	7.6	72	56	<10
Median	490	<2	70	<0.2	400	69	26	<1	<2	2.4	520	0.03	84	0.59	20	<2	7.8	13	98	180	<10
Mean	490	2	73	0.2	380	69	25	1.3	2.4	2.3	510	0.052	81	0.88	21	2.2	7.7	15	97	180	10
P95	550	2.5	100	<0.2	490	85	29	1.4	2.6	3.8	650	0.14	110	2.3	37	2.4	8.1	20	120	300	3.3
Max	630	2.7	110	<0.2	520	94	31	17	24	4.8	760	0.17	130	4	97	5.9	8.8	75	220	470	10
Cell 2W																					
n	130	17	17	3	240	260	13	210	190	250	270	120	260	190	210	180	110	110	120	350	88
Min	120	<2	14	<0.2	<40	1.3	1.2	<3	<2	<0.1	3	<0.03	<0.5	<0.01	<5	<5	6.4	3.3	24	20	<30
P5	240	<2	19	<0.2	240	17	11	<1	<2	0.99	160	<0.03	21	0.01	14	<2	6.8	7.6	44	53	<10
Median	370	<2	30	<0.2	400	54	26	<1	<2	2.8	400	0.03	64	0.34	58	<2	7.7	13	75	170	<10
Mean	370	2	43	0.2	400	55	28	1.4	3.1	3	430	0.071	72	0.43	77	3.5	7.6	14	78	190	11
P95	550	4.7	110	<0.2	560	90	48	3	8.3	5.8	810	0.17	140	1.2	190	11	8.9	22	120	350	13
Max	620	5.3	140	<0.2	630	120	67	8	39	9.6	1100	0.87	190	2.3	290	46	9.4	33	150	530	23

Notes:

1. Element concentrations are for unfiltered samples, except iron.
2. All detectable values are shown to two significant figures.

4.2.3 Sample Collection

At the laboratory, all discrete textural layers were tested for rate of HCl reaction (ie a "fizz" test) and qualitative magnetism as an indicator of magnetite content. Samples were selected from each hole to represent the surface material (i.e. potentially weathered) and two samples of each textural type from each hole. These samples were submitted for relative density, moisture content and particle size determinations, quantitative mineralogy by x-ray diffraction and chemical analysis.

4.3 Dissolution Testwork

4.3.1 Physical Characterization

Physical characterization included density determinations and size fraction analysis as part of the Environmental Sampling and Analysis Plan for the pilot plant test program (RS32, Barr 2006b).

4.3.2 Mineralogy of NorthMet Project Tailings

Optical

Optical analyses were performed by PolyMet on polished thin sections of NorthMet whole tailings samples. Results are provided in Appendix B.1.

Sub-Optical

Results of an extensive evaluation of the elemental composition of minerals present in the NorthMet Deposit are provided in RS42 (SRK 2007b). Results of that study are applicable to the tailings samples.

X-Ray Diffraction

X-ray diffraction was planned but not completed because the optical work provided a better characterization of mineral forms present in ore and tailings than could be expected using XRD.

4.3.3 Analytical Methods

Solids Characterization

A split of each sample was submitted for the following analyses:

- Sulfur forms (total S, S as sulfate).
- Paste pH.
- Neutralization potential and carbonate.
- 50 elements (mostly metals by ICP scan following aqua regia (nitric and hydrochloric acids) digestion).
- Whole rock oxides. This provided total concentrations of major elements in whole tailings.

Results are provided in Appendix B.3.

ASTM Humidity Cell

Humidity cell testing was performed using ASTM Procedure D 5744 – 96 (Reapproved 2001). This procedure was selected for the following reasons:

- Similar procedures have been in use under different names since the late 1980s (e.g. MEND 1991). The results can therefore be evaluated in the context of more than a decade of experience using the procedure.
- It is a standard procedure approved by the ASTM which produces reproducible results (White and Lapakko 2000).

The ASTM procedure provides some options for varying the test procedure. Modifications to the test procedure were described in the overall characterization plan (SRK 2006).

MDNR Reactor

Tailings samples are being tested in an apparatus designed by MDNR to contain 75 g of sample to complement the MDNR's own long term experiments and comparisons using the ASTM humidity cell and MDNR Reactor (Lapakko et al. 2002; Lapakko and White 2000). The procedure was described in the overall characterization plan (SRK 2006).

PolyMet and LTVSMC Tailings Interaction Experiment

Based on the initial understanding of the characteristics of the NorthMet and LTVSMC taconite tailings, the following chemical processes can be expected to occur within the layered tailings basins:

- In NorthMet Tailings
  - Oxidation of residual sulfide minerals resulting in release of acidity, iron, sulfate and trace elements (copper and nickel).
  - Weathering of silicate minerals by carbonic acid resulting in release of alkalinity and base cations.
  - Development and migration of an oxidation front through the tailings.
  - Attenuation of metals as a result of interaction between pore fluids and mineral grains.
- LTVSMC Tailings
  - Dissolution of ankerite, siderite and calcite under saturated conditions resulting in release of calcium, magnesium, reduced (ferrous) iron, reduced manganese and bicarbonate alkalinity.
  - Localized re-precipitation of ferric hydroxides and manganese oxides due to variations in pH and oxidation-reduction conditions.

- Interaction Between NorthMet Tailings Pore Water and LTVSMC Tailings
    - Sorption of metals by ferric hydroxides and manganese oxides, particularly in the immediate contact zone where LTVSMC tailing are probably partially oxidized.
    - Precipitation of metal carbonates, hydroxides and silicates due to alkaline conditions.

To evaluate these processes, a column design was agreed with the MDNR and included the following main features (Figure 4-2):

- Two 5 kg leach columns designed to generate NorthMet tailings pore water as feed into LTVSMC tailings. The leach columns contained bulk tailings P1 and P3 prepared using copper sulfate.
- NorthMet tailings columns open to atmosphere.
- Leachate from the NorthMet tailings directed into a column containing 2 kg of LTVSMC tailings.
- Leachate from first LTVSMC tailings column into a second LTVSMC tailings column.
- Leachate from second LTVSMC tailings column into a third LTVSMC tailings column.
- All connecting pipes between columns and final effluent sampling point operated to allow sampling with exclusion of oxygen to simulate conditions beneath the NorthMet tailings.
- Application of 2 L of deionized water to the NorthMet tailings every week to allow withdrawal of up to 250 mL of water from each of the two intermediate locations and 1.5 L of the final effluent. This application rate represents approximately one pore volume every 4 weeks.

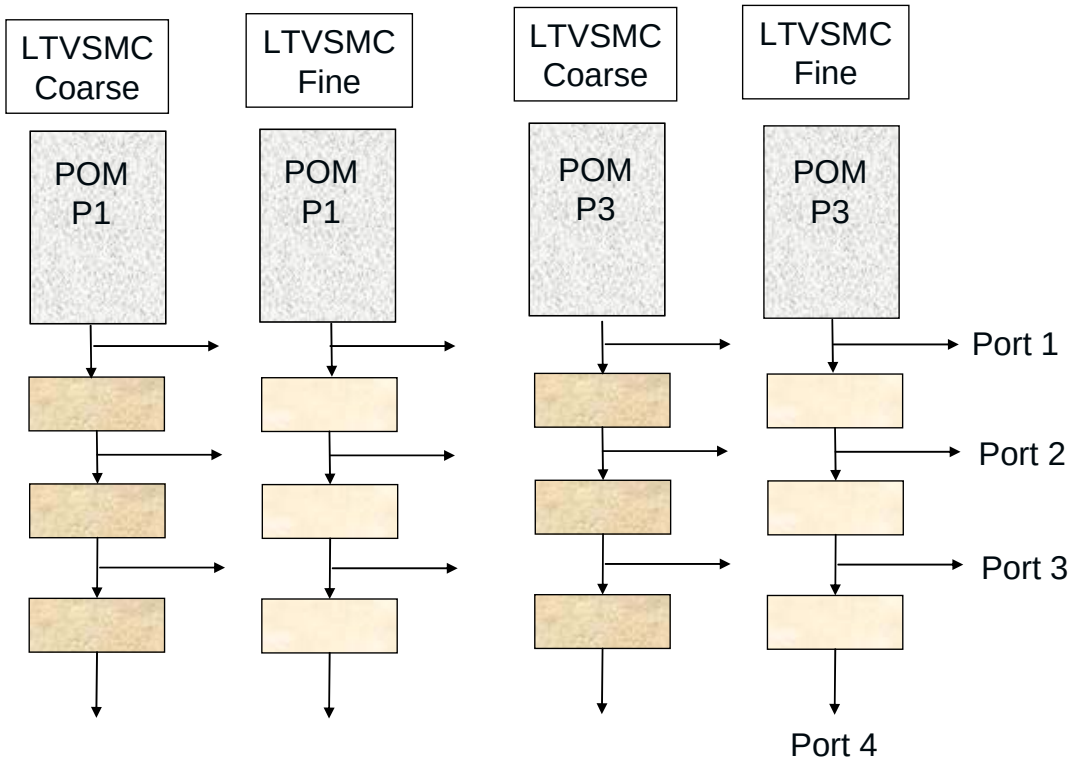


Figure 4-2: Schematic of NorthMet and LTVSMC Taconite Tailings Contact Experiment

Leachate Analysis

Leachates from kinetic tests were analyzed for the parameters indicated in Table 4-2. Conductivity, pH and ORP were analyzed every week. Sulfate, alkalinity, acidity, chloride and fluoride were analyzed on even numbered weeks. Every four weeks beginning on the first rinsing cycle (week 0) the metals indicated in Table 4-2 were analyzed using an ICP-MS scan on filtered samples. On other even numbered weeks (i.e. weeks 2, 6, 10 etc.), the leachates were analysed for a higher level scan (ICP-OES) to evaluate trends in major elements. This scan also provided trace metal concentrations but at higher detection limits.

Leachate analytical results are graphed in Appendix C. Results are available electronically as Microsoft Excel® spreadsheets in the report pocket.

Table 4-2: List of Parameters for Humidity Cell Leachate Analyses. Concentrations in mg/L except where indicated

Parameter	Limit	Parameter	Limit
pH (standard units)	-	Acidity	1
Conductivity (µS/cm)	1	Alkalinity	1
Chloride	0.2	Sulfate	0.5
Fluoride	0.05	Total Inorganic Carbon	1
ORP (mV)	-		
Dissolved Elements (ICPMS Scan)			
Aluminum	0.001	Mercury	0.02 µg/L
Antimony	0.0001	Molybdenum	0.00005
Arsenic	0.0001	Nickel	0.0001 (0.00005)[1]
Barium	0.0001	Potassium	0.02
Beryllium	0.0002	Selenium	0.0002
Bismuth	0.0002	Silicon	0.05
Boron	0.005	Silver	0.00005
Cadmium	0.00004	Sodium	0.01
Calcium	0.01	Strontium	0.0001
Chromium	0.0002	Tellurium	0.0002
Cobalt	0.0001 (0.00005)[1]	Thallium	0.00002
Copper	0.0001	Thorium	0.0001
Iron	0.01	Tin	0.0001
Lead	0.00005	Titanium	0.0002
Lithium	0.0002	Uranium	0.00005
Magnesium	0.005	Vanadium	0.0002
Manganese	0.00005	Zinc	0.001

Notes:
1. Low detection limits are available for cobalt and nickel as shown.

QA/QC

To summarize, QA/QC included the following components:

- Roughly 10% of all solids analyses are performed in duplicate.
- Roughly 10% of all cell and reactor tests are run as duplicates.
- A blank cell and reactor containing no sample is being operated to check for contamination of leachates by construction materials.
- Individual leachate results are reviewed.
- Ion balances on leachate results are reviewed. In general, imbalances of ±10% are considered acceptable. Re-analysis if requested depending on the nature of the imbalance.
- Data of data trends in kinetic test leachates are analyzed to check for anomalies.

Review of antimony data indicated that test apparatus components of the ASTM humidity cell tests constructed from polyvinyl chloride (PVC) were leaching antimony due to the use of antimony oxide in manufacturing. Antimony results from humidity cell results were therefore discarded. The MDNR Reactor tests were unaffected.

4.3.4 Interpretation Methods for Kinetic Tests

Trend Analysis

All results from kinetic plots were plotted as time series which were continually updated as the project progressed to allow trends to be assessed. Results were plotted as raw concentrations and as loadings, or release rates calculated from:

Loading (mg/kg/week) = Concentration (mg/L) x Leachate Recovered (L) / Mass of Sample.

As indicated above, metal concentrations were determined by two different methods on alternate even-numbered weeks. For the purpose of plotting and loading calculations, the following rules were used:

- If the result was determined by ICP-MS and was below the reporting limit, the value on the graph is at the reporting limit.
- If the result was determined by ICP-ES and was determined to be below the reporting limit, no value is plotted.
- If the result was determined by ICP-ES and was determined to be above the reporting limit, the value is plotted.

These rules can result in four cycles between plotted results if the parameter is not detected by ICP-ES (e.g. molybdenum in shake flask leachates).

Occasionally, "sawtooth" trends are apparent in which values alternate between high and low for the ICP-ES and ICP-MS analyses. This results from analytical "noise" around the ICP-ES reporting limit when reported values are slightly above the reporting limit. Aluminum is a particular example that commonly shows reported values above the ICP-ES reporting limit of 0.05 mg/L.

Many graphs are plotted on logarithmic axes to allow data spanning a wide range of concentrations to be compared.

Average Rate Calculations

Average rates (in mg/kg/week) were calculated to evaluate correlations between bulk characteristics (e.g. metal and sulfur content, mineralogical characteristics) and to provide inputs into water quality predictions. The following method was used to calculate average rates:

- The loading trends for sulfate were examined as an indicator of sulfide oxidation rates and the expected main factor driving other parameters such as release of metals and the products of acid neutralization.

- The loading trends typically showed relatively rapid initial release of sulfate followed by decrease, then a longer term trend (stable, increasing, or slow decrease). The initial trend is usually a result of leaching of weathering products produced by oxidation of the sample in storage prior to testing. The trend following the short term effect reflects dissolution of weathering products produced each week. For trends showing relatively stable release, the trend was examined to find the first week when the release rate was below the highest point in the stable trend. If a decreasing or increasing trend was apparent, the trend was visually assessed to estimate when the initial flush ended. The release rates following the development of the stable trend are then used to calculate average release for the entire trend. In the event that the trend showed much more variability than other tests, the average was not calculated.

- Loading trends for other parameters were calculated using the same time period as sulfate so that comparisons between parameters could be made on a consistent basis.

- Some dissolved ions were not determined on a weekly basis, and in some cases have variable analytical frequency depending on detection by ICP-ES or ICP-MS. The average rates for individual weeks were pro-rated between analyses by summing the load leached rather than just averaging weekly rates.

Depletion Calculations

Rates were also used to evaluate depletion of rock components by totaling the load leached over the entire period of the test.

5 Results

5.1 Solids Characteristics

5.1.1 LTVSMC Tailings

Mineralogy

Table 5-1 shows selected data sorted by the main textural groups (coarse sand, fine sand and slimes).

The dominant mineral in all samples was quartz which varied from 58 to 79% (by weight) but was not different in the three textural groups. Residual amounts of the ore minerals (hematite and magnetite) were present as expected. Magnetite was lower in the slimes samples likely resulting from density segregation as the tailings were deposited. Carbonates were a significant mineralogical component varying from 5 to 14%. Total carbonate content was greater in the slimes fraction compared to the coarse sands. Ankerite and siderite dominated and occurred in about equal amounts. The calcite content was lower than either ankerite or siderite.

Pyrite was detected in most samples but at very low levels. The sulfur content of the samples varied from 0.02 to 0.04% equivalent to pyrite content of 0.04 to 0.08%.

Silicates occurring in all samples were hydrobioitite, kaolinite, amphibole (cummingtonite ± grunerite), diopside, ferripyrophyllite (possibly minnesotaite) and albite. Other minerals occurring in a few samples were pyrophyllite, muscovite and hydroxylapatite. There was no evidence that the mineral distribution was related to particle size.

Solids Chemistry

Distribution of metals was also unrelated to particle size with the possible exception of manganese which appeared to be elevated in the slimes (Table 5-1). This is consistent with the higher carbonate content and indicates that manganese is associated with the carbonates.

Table 5-1: Characteristics of LTVSMC Tailings Samples

			Mineralogy Indicated by X-Ray Diffraction											Trace Element Composition										
Core and Tailings Type	Interval Sampled		Quartz	Pyrite	Calcite	Ankerite	Siderite	Hematite	Magnetite	Biotite	Kaolinite	Ferriprophyllite	Albite low	As	Cd	Co	Cr	Cu	Fe	Mn	Ni	P	Pb	S	Zn
	Start ft	Finish ft	%	%	%	%	%	%	%	%	%	%	%	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	ppm
Coarse Sand																									
GP-1	8	12	72	0.2	0.3	3	3	3	4	1	1	2	2	31	<0.5	12	100	14	15.85	5970	5	240	7	0.02	14
GP-1	20	40	71	0.2	0.2	5	8	2	2	2	1	2	2	36	<0.5	14	90	25	15.5	7110	8	250	5	0.04	13
GP-3	8	12	58	0	0.4	6	6	2	3	5	1	6	4	16	<0.5	7	42	7	13.7	3420	3	250	4	0.02	9
GP-4	4	16	59	0.1	0.7	2	4	2	3	10	2	2	3	19	<0.5	7	57	9	15.45	3890	1	240	7	0.02	9
GP-4	20	24	79	0	0.1	3	2	2	3	3	1	2	1	21	<0.5	10	45	8	12.85	4010	3	250	5	0.02	34
Fine Sand																									
GP-1	60	72	72	0.1	0.7	3	2	1	1	2	2	2	4	22	<0.5	11	77	20	12.05	7010	3	330	3	0.04	14
GP-2	0	1	60	0	0.2	6	7	2	2	11	4	3	0	15	<0.5	9	27	7	14.4	4270	-1	490	-2	0.02	10
GP-2	24	28	62	0.4	1	6	3	1	2	5	2	3	3	22	<0.5	11	61	13	14.55	5340	3	550	7	0.02	14
GP-3	44	60	68	0.2	0.5	4	3	3	2	4	1	3	1	15	<0.5	12	45	17	13.4	8510	4	400	7	0.03	12
GP-4	18	20	73	0	0.2	7	3	2	2	4	2	2	1	43	<0.5	10	41	12	13.3	4020	2	290	8	0.02	13
GP-5	8	20	78	0.1	0.1	4	2	3	3	3	1	2	1	14	<0.5	7	42	7	13.45	3630	2	270	2	0.02	7
GP-5	36	48	73	0.4	0.4	4	2	2	3	2	1	3	2	14	<0.5	9	53	14	13.55	5820	2	290	4	0.02	8
Slimes																									
GP-2	28	32	62	0	1	4	4	2	1	5	2	3	5	18	<0.5	14	31	10	14.2	7390	3	530	4	0.02	13
GP-3	60	72	62	0	0.7	8	5	3	1	7	2	2	1	15	<0.5	14	33	20	12	10050	4	590	2	0.04	14
GP-5	20	24	70	0.1	0.3	6	4	3	2	6	2	3	1	25	<0.5	9	38	7	13.75	4830	1	460	4	0.02	8
GP-5	48	52	72	0	0.6	4	6	2	1	4	1	2	1	16	<0.5	16	40	19	13.1	12400	3	550	4	0.03	11

5.1.2 PolyMet Tailings

Mineralogy

The mineralogical report on whole tailings characteristics is provided in Appendix B.1. Photomicrographs are provided in Appendix B.2. The dominant mineral in the tailings was plagioclase (50% to 80% by volume) which microprobe work on plagioclase in the NorthMet Deposit is shown to average $An_{59}Ab_{41}$ (labradorite) (RS42, SRK 2007b). Olivine was the next most abundant mineral (10 to 15%) and clinopyroxene (4 to 5%). A component of the tailings was too fine-grained to be distinguished optically. Calcite was not observed, which is consistent with the lack of calcite in the NorthMet Deposit and Duluth Complex in general (PolyMet 2007).

The main visible sulfide was pyrrhotite described as occurring at 0.25% to 0.5%. Chalcopyrite, sphalerite and galena were described as rare and pentlandite and cubanite were not observed.

Solids Chemistry

Selected chemical characteristics of tailings samples being tested in dissolution experiments are shown in Table 5-2. Results are shown for the three ore composites or packets in the order in which they were generated, as described in Section 4.1.2. The solids results illustrate the decrease in sulfur concentrations that occurred for the bulk (whole) tailings after copper sulfate was introduced to activate pyrrhotite. Sulfur concentrations were 0.2% and 0.23% for P2 and P1, respectively processed without copper sulfate, and 0.1 and 0.15% for P1 and P3, respectively processed with copper sulfate. The activator had no perceptible effect on copper and nickel concentrations in the tailings.

Table 5-2: Selected Chemical Characteristics of Tailings Samples

Fraction	Sulfide Activator	Parcel	pH	Total S	Cu	Ni	CO_2
			Unity	%	%	%	%
Whole	None	P2	8.3	0.2	0.053	0.039	0.2
+100	None	P2	9	0.15	0.046	0.028	<0.2
-100+200	None	P2	9.2	0.17	0.03	0.031	<0.2
-200	None	P2	9	0.24	0.03	0.031	0.2
Whole	None	P1	8.3	0.23	0.025	0.033	0.2
Whole	Copper sulfate	P1	8.3	0.1	0.022	0.032	<0.2
+100	Copper sulfate	P1	9.1	0.11	0.03	0.021	<0.2
-100+200	Copper sulfate	P1	9.3	0.1	0.018	0.027	<0.2
-200	Copper sulfate	P1	8.8	0.09	0.011	0.027	0.2
Whole	Copper sulfate	P3	8.4	0.15	0.042	0.037	<0.2
+100	Copper sulfate	P3	9.2	0.11	0.039	0.023	<0.2
-100+200	Copper sulfate	P3	9.3	0.14	0.025	0.029	<0.2
-200	Copper sulfate	P3	9	0.14	0.014	0.032	0.2

The results show no consistent differences between particle sizes. For Parcel 2 processed without using copper sulfate, the finest fraction had higher sulfur than the two coarser fractions and the bulk, but for P1, the fine fraction had the lowest sulfur concentration but the difference between fractions was small. For P3, the differences between fractions were again small. It appears that use of copper sulfate not only lowered overall sulfur content but may also have resulted in more effective removal of sulfur from the finest size fractions.

As shown in Table 5-2, carbonate was reported as undetectable (<0.2%) or at the detection limit (0.2%). These low values confirmed the optical finding that calcite was effectively absent, or possibly present at trace levels. This is consistent with the magmatic origin of the Duluth Complex.

Neutralization potential results are not shown due to the complications of interpreting NP determined on samples containing reactive silicate and low carbonate content in the context of low sulfide content (Paktunc 1999).

5.2 Description of Leachate Chemistry from Dissolution Testwork

5.2.1 PolyMet Tailings

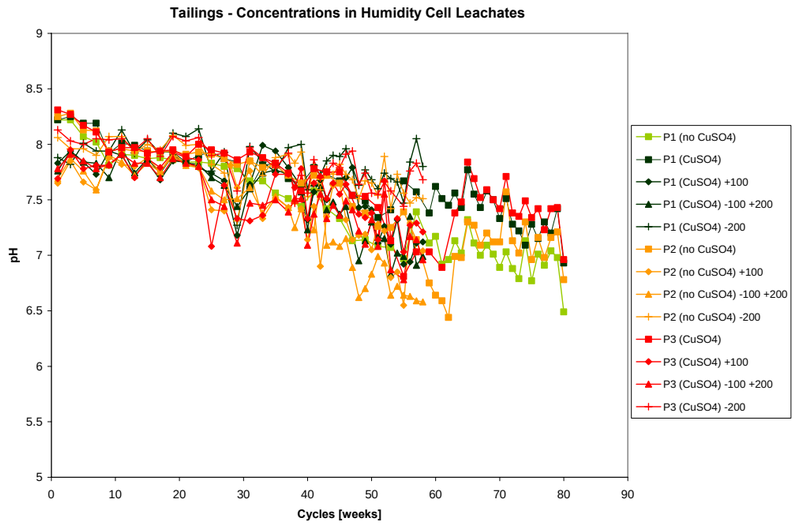
Graphs illustrating results from testwork are provided in Appendix C. Trends for pH, sulfate, alkalinity, nickel, cobalt and copper in humidity cells and MDNR reactors are shown in Figures 5-1 and 5-2, respectively.

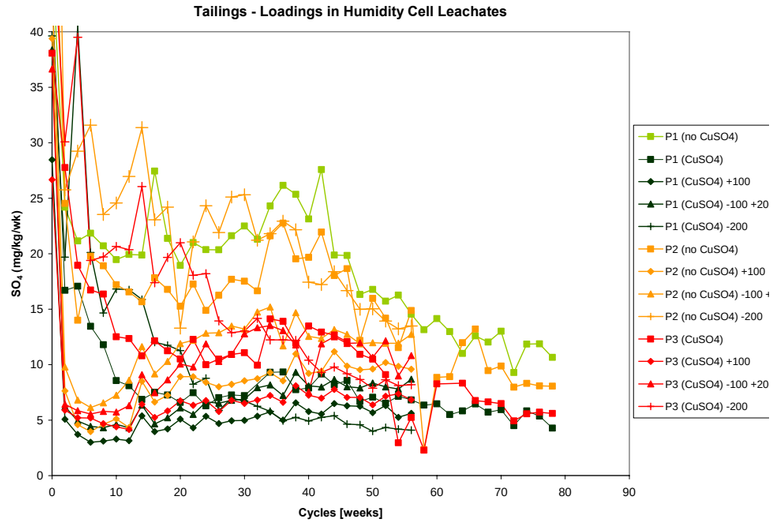
All tests were continuing at the time of preparation of this report. The difference in duration of individual tests reflects different start-up dates. Tests on tailings size fractions were started after bulk tailings samples and the PolyMet/LTVSMC interaction test after that.

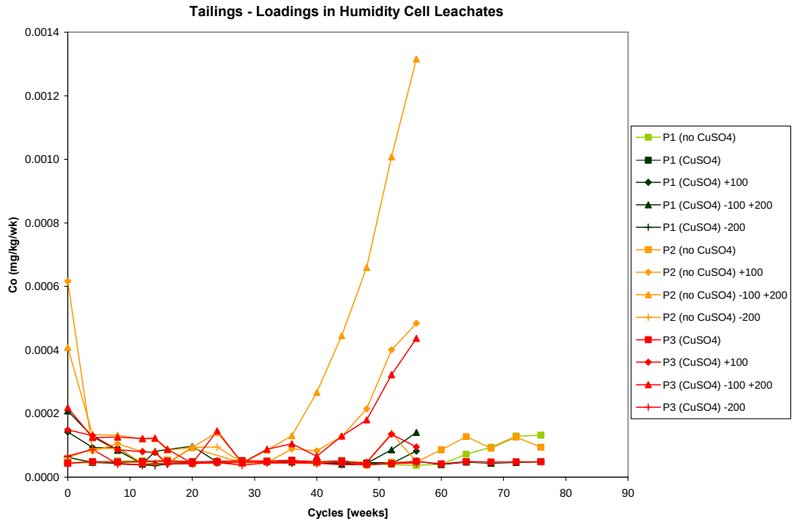
ASTM Humidity Cells

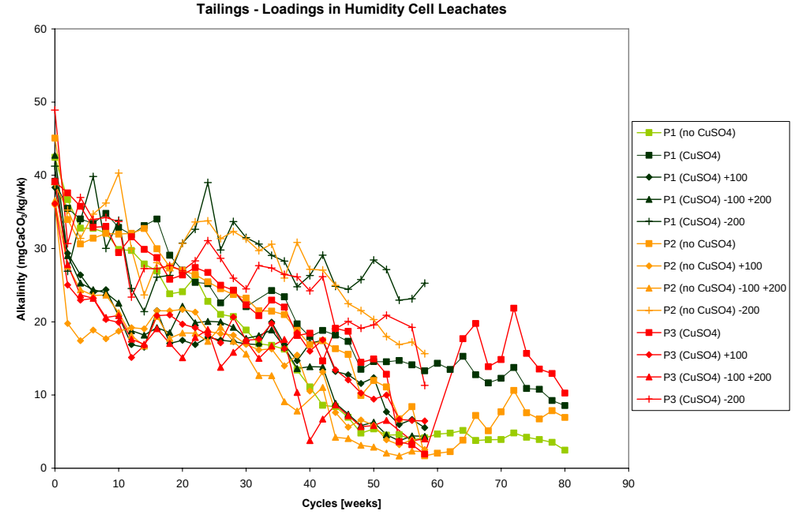
All tests showed a general decline in leachate pH lasting about 1 year followed by stabilizing pH after about a year of testing for the bulk tailings. Leachate pHs appeared to stabilize above pH 7, but with some variations. The P1 tailings produced using copper sulfate showed the highest pH (near 7.2 to 7.5). This sample had the lowest sulfur concentration of any of the bulk samples (0.1%). P3 showed a decrease in pH to 6.8 before increasing to between 6.9 and 7.4. This sample had a higher sulfur concentration (0.15%). The other two samples produced without using copper sulfate showed generally lower pHs. The lowest pH (6.4) was shown by P2 (tailings prepared using copper sulfate) though pH appeared to recover.

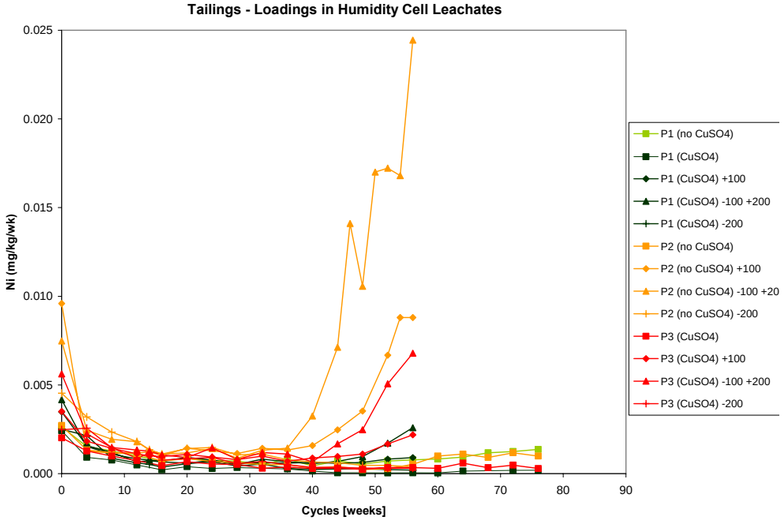
Individual size fractions initially produced leachates with a narrow range in pHs (7.6 to 8.2). After about 30 weeks, the range in pHs increased as the coarser fractions showed declining pHs to below pH 7 while the -270 mesh tailings showed a lesser decline to pHs between 7.2 and 8. More recent leachate pHs for the coarse tailings samples were stabilizing.











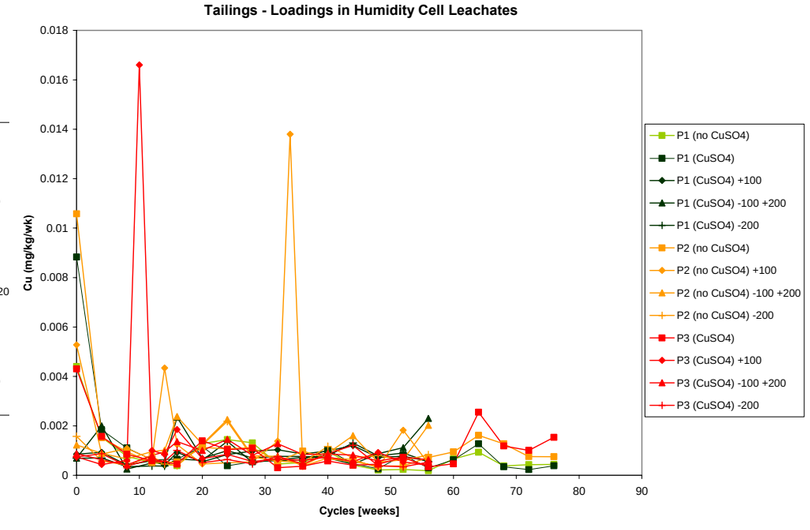
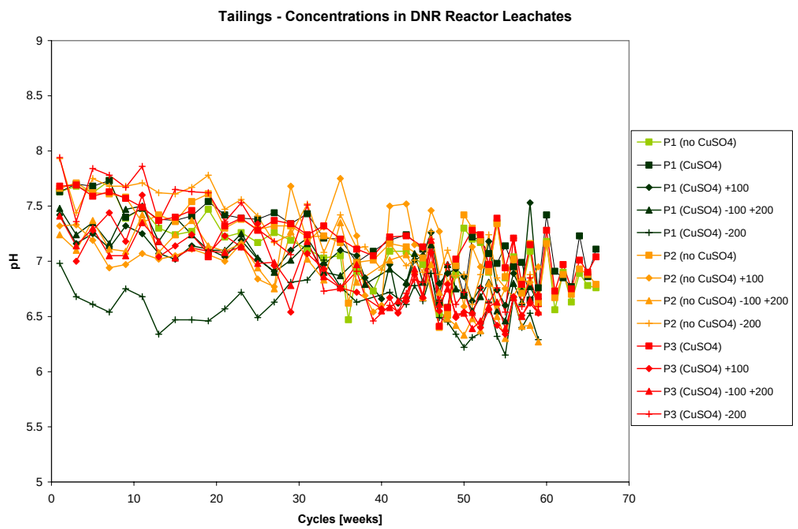
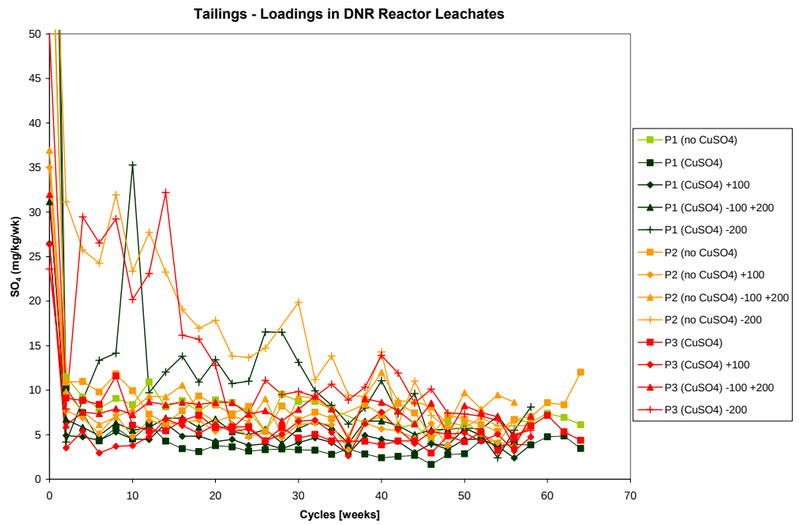
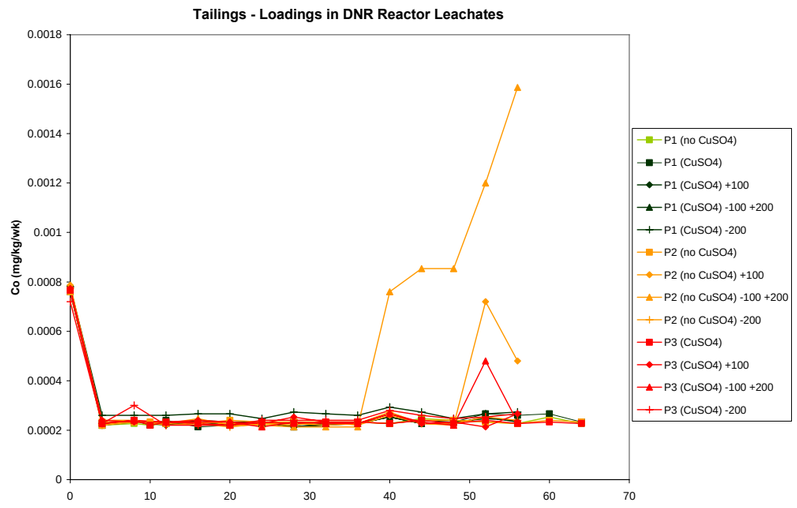
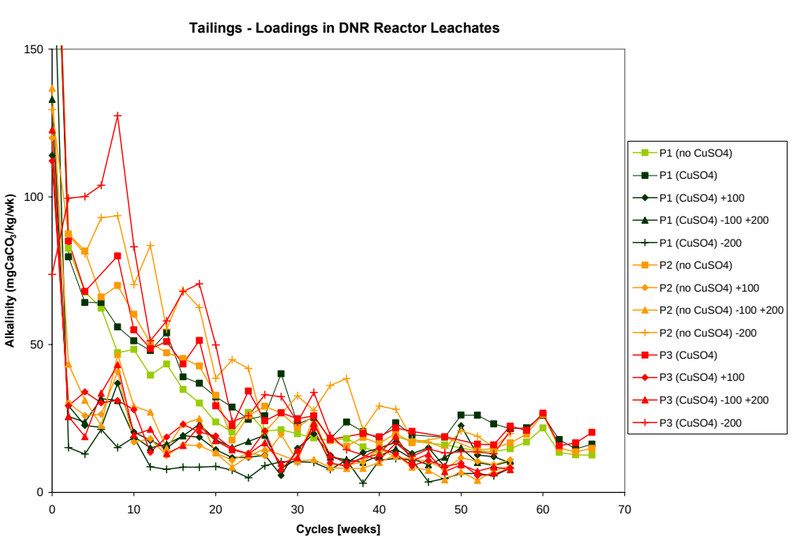


Figure 5-1: Trends in pH, Sulfate, Alkalinity, Nickel, Cobalt and Copper in Tailings Humidity Cells









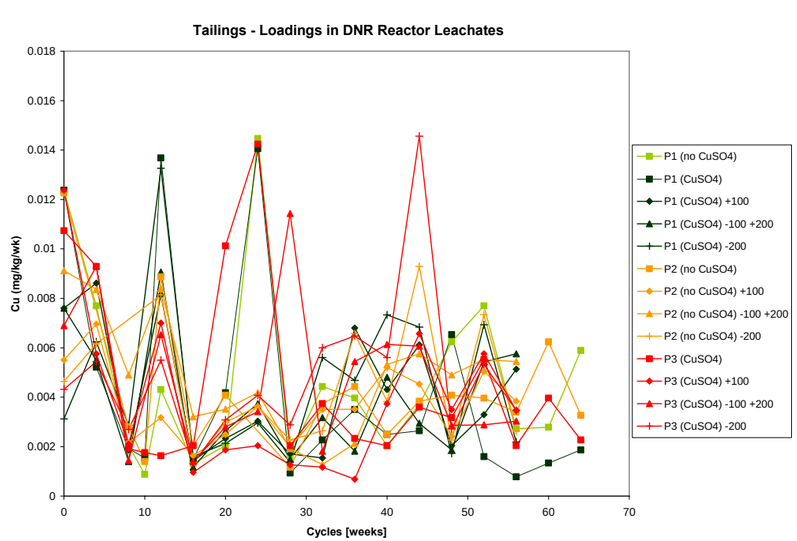


Figure 5-2: Trends in pH, Sulfate, Alkalinity, Nickel, Cobalt and Copper in MDNR Reactor Tests

Major ions in leachates were sulfate, bicarbonate and calcium. Sulfate concentrations decreased very sharply at first, then reached a subdued peak and then in the case of whole tailings showed a decreasing trend. Alkalinity in contrast showed a decreasing trend, with greatest alkalinity leaching from the -200 mesh fractions, followed by the bulk fractions. Alkalinity leaching from all tailings samples appeared to stabilize after about a year. Calcium leaching followed trends close to sulfate (except that the initial decrease was absent). Calcium leaching appears to stabilize after about a year, whereas magnesium, sodium and potassium leaching followed the decreasing alkalinity trend stabilizing after about a year.

Leaching of copper generally occurred at concentrations of less than 5 µg/L. Isolated spikes were observed reflecting occasional detections by ICP-ES. Because the trends were not confirmed by subsequent low level ICP-MS analyses, the results are due to analytical uncertainty. Nickel leaching initially occurred at low levels mostly below 10 µg/L. Around week 40, nickel concentrations began increasing for all the coarser tailings samples concurrent with declining pH. Similar affects were apparent for cobalt but at lower concentrations.

Whole tailings samples prepared without using copper sulfate are showing slightly increasing nickel and cobalt release trends. Lowest nickel and cobalt leaching is being shown by the P1 and P3 whole tailings prepared with the use of copper sulfate and all three -200 mesh samples.

Manganese leaching showed the same trends as nickel and cobalt.

Low level mercury analyses were conducted on 13 leachates collected in June 2006. Results are provided in Appendix C.4. A statistical summary of the data is shown in Table 5-3. The reporting limit for the analyses was 2 ng/L. At 3.1 ng/L, the median was slightly above the reporting limit. The travel blank contained 2.4 ng/L mercury. The results indicated that mercury may have leached from the samples or have been contributed by the laboratory atmosphere. The latter might occur because the samples are exposed to the atmosphere during testing although the effect cannot be confirmed.

Table 5-3: Summary of Low Level Mercury Analyses for ASTM Humidity Cell Leachates (ng/L)

Statistic	Result
n	13
Minimum	Not detected
P_5	2.5
Mean	3.2
Median	3.1
P_{95}	4.0
Maximum	4.2

MDNR Reactors

Leachates from MDNR reactors showed a similar pH trend to the ASTM tests except that pHs were lower and mostly between 6.2 and 7.5 later in the test period (Figure 5-2). Lowest pH (6.3) was indicated by the the P1 -200 mesh sample (prepared using copper sulfate).

Major ions in leachates were bicarbonate, sulfate and calcium. Alkalinity decreased at first though after a year, alkalinity leaching stabilized along with sulfate and other major ions (calcium, magnesium, sodium and potassium). Silica leaching stabilized early in testing. The whole tailings and -200 mesh fractions initially leached higher levels of all ions though the gap between whole tailings and the -200 mesh fraction and the coarser size fraction narrowed as the test proceeded. The exception was the P1 -200 mesh sample which initially consistently lower leaching of major ions compared to the other -200 mesh samples.

Copper leaching was low and erratic with no evidence of trends. Coarse fraction P2 samples (-100+200 and +100 mesh) both showed increasing nickel, cobalt and manganese leaching which matched the same trend shown in the humidity cell. There was also evidence of low level increases in nickel leaching for other coarse size fraction samples though not for the whole tailings.

5.2.2 LTVSMC Tailings

Samples of coarse and fine LTVSMC tailings were leached in a configuration that allowed changes in concentrations along the flow path to be evaluated. Charts of concentrations are shown in Appendix C.3.

Leachates from both samples showed an initial slight increase in pH which then stabilized near 8. There was no indication of pH changes along the flow path. Oxidation-reduction potential readings gradually increased for both samples and showed that reducing conditions had not developed in the columns. Leachate chemistry was initially dominated by sodium and sulfate but then shifted to bicarbonate, calcium and magnesium. Sulfate concentrations were lower than alkalinity but sulfate was present in the tens of milligrams per liter.

Concentrations of major parameters increased along the flow path. Lowest concentrations were seen in the first port and highest in the last of the three ports. Many parameters showed a flushing effect. Highest concentrations were seen early in the test but these slowly dissipated as the test proceeded. This effect was shown for sulfate and fluoride. Despite the textural differences, results for the samples were very similar.

Leaching of copper occurred at low concentrations for both samples and there was no apparent indication of increasing concentrations along the flow path or of a flushing effect. Nickel concentrations were also low but it appeared that concentrations increased along the flow path and there was some initial flushing. The fine tailings sample showed decreasing nickel concentrations and a lack of flow path accumulation toward the latter stages of the test, whereas the coarse sample

showed nickel concentrations increasing between port 1 and 2 but not between port 2 and 3. The effect for the coarse tailings sample diminished as the test progressed.

Manganese leaching showed unusual trends. For the fine tailings sample, concentrations initially increased and showed accumulation along the flow path. At the peak, higher concentrations (0.1 mg/L) were apparent in sampling port 1, and then concentrations decreased very rapidly to below 0.01 mg/L. Ports 1 and 2 then showed stable manganese concentrations near 0.01 mg/L. Port 3 showed further decrease in manganese to below 0.001 mg/L. The more recent samples showed that manganese decreased along the flow path with the greatest decrease at the base of the column. These changes correlated with slowly increasing ORP from 88 mV to over 350 mV.

The coarse sample showed relatively stable manganese concentrations for about 16 weeks then concentrations in Port 3 decreased from 0.04 mg/L to 0.002 mg/L while concentrations in port 1 remained near 0.05 mg/L. Concentrations in port 2 decreased as port 3 concentrations also decreased but to a lesser degree. Like the fine tailings, later results showed decreasing manganese concentrations along the flow path from 0.04 to 0.005 mg/L in the most recent sample set.

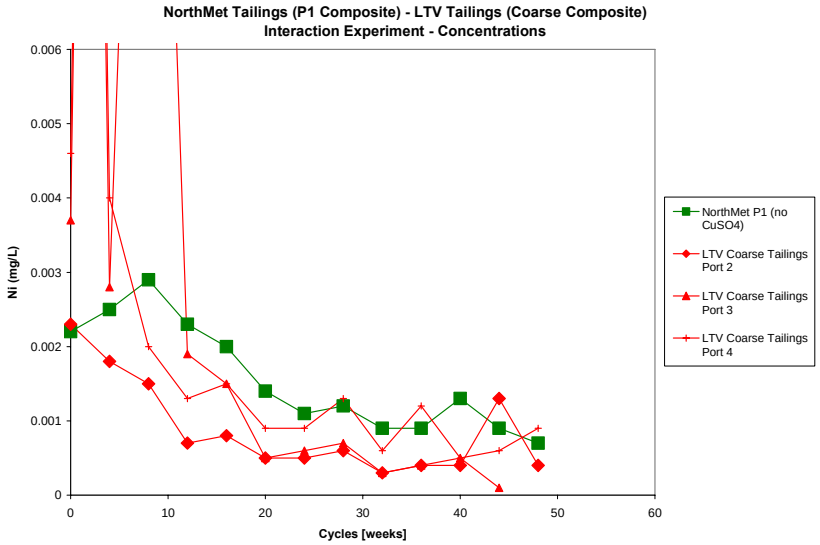
5.2.3 PolyMet and LTVSMC Tailings Interaction

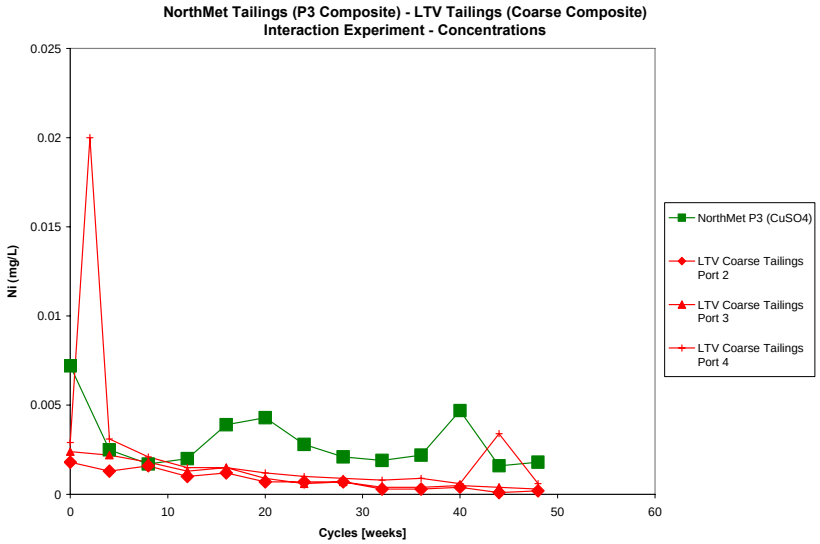
PolyMet P1 Tailings

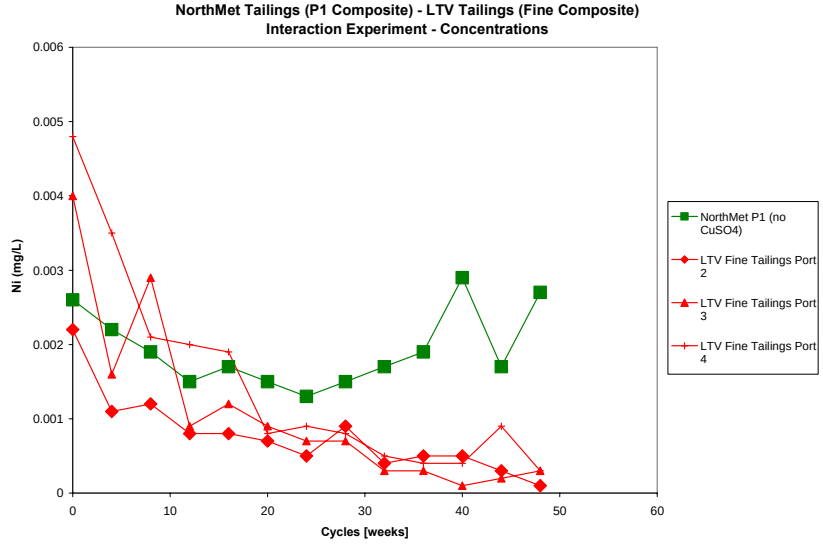
The P1 NorthMet leachates were characterized by pH above 7 and relatively stable leachate chemistry including alkalinity 33 mg $CaCO_3$/L, sulfate between 16 and 20 mg/L and calcium 14 to 15 mg/L. Copper and nickel leaching were below 0.005 mg/L. Neither parameter showed indication of increasing trends.

Concentrations of all major and many trace ions increased as the leachate from the NorthMet tailings entered the LTVSMC tailings and concentrations increased along the flow path through the LTVSMC tailings. Of the trace elements, only nickel leached from the NorthMet tailings at greater concentrations than the LTVSMC tailings (Figure 5-3). However, nickel concentrations variably decreased as the NorthMet tailings leachates entered the LTVSMC tailings. For the fine taconite tailings composite, nickel concentrations decreased from 0.003 mg/L in the NorthMet tailings leachate at week 48 to 0.0003 mg/L in the first port in the taconite tailings indicating a concentration reduction of 90%. In the coarse taconite tailings composite test the effect was initially apparent but became weak toward the later stages of the test

Cobalt did not show the same effect as nickel. It was generally not detected in the PolyMet tailings leachates. In the early stages, cobalt flushed from the LTVSMC tailings but then diminished.







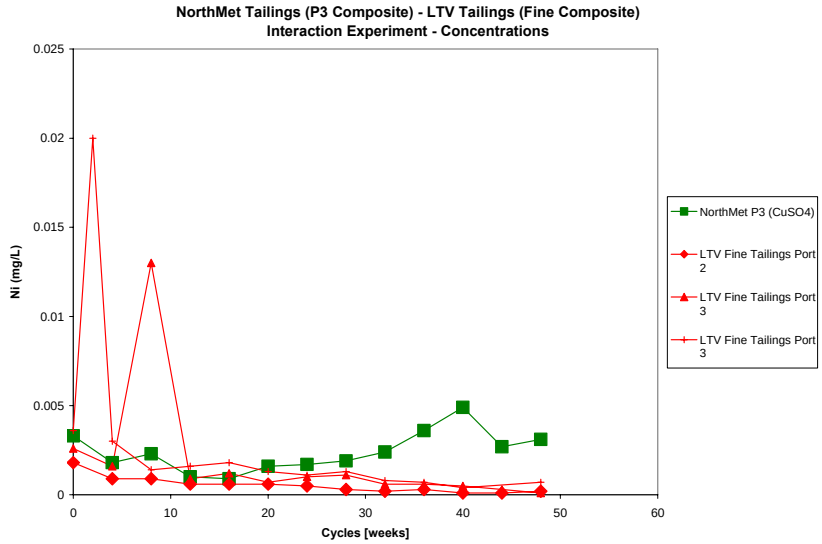


Figure 5-3: Nickel Leaching Trends for PolyMet and LTVSMC Tailings Interaction Experiment

Manganese leaching from Port 4 of the LTVSMC tailings followed similar trends to the control column. Initially, this showed that manganese increased as the PolyMet tailings water contacted the LTVSMC tailings. As the test progressed, manganese leaching from the PolyMet tailings stabilized and in the case of the fine LTVSMC tailings exceeded manganese leaching from the LTVSMC tailings. This implied that manganese leached from the PolyMet tailings was removed by contact with the PolyMet tailings.

PolyMet P3 Tailings

The P3 NorthMet leachates had similar major ion characteristics except that leachates had slightly lower alkalinity, sulfate and calcium concentrations. Copper leaching was comparable to P1 but nickel concentrations were slightly higher reaching a maximum of 0.005 mg/L.

Like the P1 columns, the LTVSMC tailings added load of major ions and many trace elements to the load released from the P3 tailings samples. Nickel concentrations were greater in the P3 leachate and was removed as the leachate contacted the LTVSMC taconite tailings. In the fine taconite tailings composition, nickel concentrations in the first port were 0.0007 mg/L in the latest leachate compared to 0.0031 mg/L leaving the P3 tailings representing a removal factor of 92%. In the coarse tailings, the inflow from the NorthMet tailings was 0.0018 mg/L compared to 0.0006 mg/L in the first taconite port, representing 33% removal.

Low level mercury analyses were conducted on 22 leachates collected in June 2006. Results are provided in Appendix C.4 and a statistical summary of the data is shown in Table 5-4. The reporting limit for the analyses was 2 ng/L. At 3.2 ng/L, the median was slightly above the reporting limit. The travel blank contained 2.4 ng/L mercury indicating minor contribution of mercury from the tailings.

There were no clear increasing or decreasing mercury concentration trends along the flow path through the LTVSMC tailings.

Table 5-4: Summary of Low Level Mercury Analyses for LTVSMC Contact Experiment Leachates (ng/L)

Statistic	Result
n	22
Minimum	2.3
P_5	2.6
Mean	3.2
Median	3.2
P_{95}	3.7
Maximum	5.4

6 Interpretation of Dissolution Testwork

6.1 General Interpretation of Leachate Chemistry

The primary requirement for interpretation of the tailings testwork was a long term prediction of the performance of the tailings solids in terms of potential for ARD and metal leaching. A similar requirement exists for the waste rock for which a much larger database exists. Review of dissolution testwork for waste rock in RS42 (SRK 2007b) indicated that a conceptual dissolution model would need to account for the observed leachate chemistry features shown in Table 6-1. Because kinetic tests on Babbitt Deposit tailings have operated for a maximum of 4 years compared to the 18 years of data for waste rock, and the sulfur content of tailings tested has been less than 0.4%, many of the features observed in the waste rock data have not been observed for tailings. Table 6-1 indicates which observations have been documented for tailings. These include:

- Absence of acidic conditions in any NorthMet or Babbitt Deposit tailings for samples containing less than 0.41% sulfur.
- Generally lower leachate pHs for MDNR reactor tests compared to ASTM humidity cells on the same material when pHs are above about 5.5.
- Increasing nickel and cobalt concentrations as pH decreases below 7 but not necessarily in the presence of increasing sulfate release.
- Decrease in nickel and cobalt release even as pH remains below 7 and continues to decrease.

Based on review of the waste rock data described in RS42 (SRK 2007b), the interpretations shown in Table 6-1 were developed.

Table 6-1: Interpretation of Dissolution Test Observations – Waste Rock and Tailings

OBSERVATION	Observed for Tailings	INTERPRETATION
General		
The variable long term delay in development of acidic conditions in the absence of carbonate mineral buffering capacity.		Silicate minerals provide buffering capacity through carbonic acid weathering and direct reaction of sulfuric acid with silicate minerals.
The absence of acidic conditions for samples containing sulfur concentrations less than 0.41%.	X	Bicarbonate alkalinity produced by carbonic acid weathering of silicates permanently offsets acidity from sulfide oxidation. Sulfide oxidation is strongly correlated with sulfur content.
Leachate pH Trends		
Initially strongly basic alkaline leachates (pH>8) followed by a steep decline in pH to below 8 within a few months of initiation of kinetic tests.		Reaction of fresh silicate mineral surfaces with water ("abrasion pH").
The lack of long term pH decrease for samples that have not generated acidic leachate after 18 years of testing		Steady generation of bicarbonate alkalinity by weathering of silicates.
The generally slow decline in leachate pH in kinetic tests that eventually produce acidic leachate. The initial decline is not necessarily accompanied by increasing sulfate release.		Bicarbonate from silicate weathering is less than acidity from sulfide oxidation and excess acid must react directly with silicate minerals. Blinding of silicate minerals limits this process.
At times stepwise sharp decreases in pH under acidic conditions.		Consumption of weathering mineral buffers leachate acidity. When exhausted, the pH drops sharply until the next buffer is reached.
Steady (at times with steps) recovery of leachate pH following a short term pH minimum.	X	As sulfide oxidation decelerates due to sulfide mineral depletion, silicate minerals become more effective again and pH recovers.
Difference between test work procedures		
Generally lower leachate pHs for MDNR reactor tests compared to ASTM humidity cells on the same material when pHs are above about 5.5.	X	Higher liquid to solid ratios result in greater effect of deionized water when reacting with silicate minerals.
Increase in sulfate in some MDNR reactor tests when pH decreases early in test		Lower pH enhances bacterial activity and accelerates oxidation.
Generally lower leachate pHs for ASTM humidity cells compared to MDNR reactor tests on the same material when pHs are below 5.5.		Higher liquid to solid ratios dilute acidity produced by sulfide oxidation.
Metal Leaching Trends		
Increasing nickel and cobalt concentrations as pH decreases below 7 but not necessarily in the presence of increasing sulfate release.	X	Dissolution of secondary minerals formed above pH 7 as pH decreases.
Decrease in nickel and cobalt release even as pH remains below 7 and continues to decrease.	X	Depletion of secondary minerals formed above pH 7.
Coincident sulfate and nickel, cobalt and copper concentration peaks as pH drops below 5.		Metals released by oxidation of sulfides remain in solution due to lower pH.
Long term declining metal release as pH recovers.		Depletion of sulfide minerals.

6.2 Comparison between Different Test Methods

As described in Section 4.3.3, two different dissolution type tests were performed on tailings samples. The ASTM humidity cell was performed to ensure consistency with a standard method

commonly in use. All samples were also tested using the MDNR's reactor test design because this procedure has been used in the past by the MDNR on both rock and tailing samples. Critical differences between the procedures include:

- The MDNR reactor contains a much smaller sample (75 g) compared to the ASTM humidity cell (1000 g).
- In order to obtain sufficient water for analysis, the MDNR reactor results in a five times higher water application rate of 2.7 mL/g compared to the ASTM reactor (0.5 mL/g).
- The sample in the ASTM cell is stirred during flushing whereas the sample in the MDNR reactor is leached without stirring.
- Water is held in contact with the sample for 1 hour in the ASTM humidity cell before draining. For the MDNR reactor, the water is allowed to drain immediately.

The difference in water application rates has an important effect on overall leachate chemistry. As observed in waste rock tests (Table 6-1), the MDNR reactors generally resulted in lower leachate pHs. Because carbonate minerals are typically absent or rare in Duluth Complex Rocks, leachate pH is controlled by weathering of silicates and alkalinity generation is relatively low and susceptible to the effects of dilution produced by using leaching water with a pH of about 5.5 due to dissolved carbon dioxide. Table 6-2 shows a sample calculation performed using Geochemist's Workbench in which the pH resulting from different water application rates was calculated assuming a constant silicate weathering rate. The difference between the application rate for the MDNR reactor and ASTM humidity cell is 0.7 pH units. This difference is a reasonable explanation for the difference observed in testwork. While this difference is not substantial it can have a significant effect on nickel leaching because nickel and cobalt solubility appears to increase as pH decreases below 7.

Table 6-2: Calculated Effect on pH of Leachate Ratio Using a Constant Silicate Weathering Rate as the Source of Alkalinity

Test Procedure	Water Application Rate mL/g/week	Equivalent Water Application Rate mm/year	Leachate pH
MDNR Reactor	2.7	2700	6.65
ASTM Humidity Cell	0.5	750	7.37

Figure 6-1 compares average sulfate release rates for the ASTM humidity cells and MDNR reactors for each fraction. The comparison for bulk tailings shows that oxidation rates are strongly correlated with sulfur concentration but that the correlation is different. The MDNR reactors showed consistently lower oxidation rates at a given sulfur concentration. Regression relationships for both test types pass close to the origin indicating that all sulfur is theoretically oxidizable. The coarser fractions show qualitatively that the humidity cells oxidized at a higher rate but the difference was less obvious. For the -200 mesh fractions, the difference between methods is not readily apparent with the exception of the highest sulfur sample for which the ASTM test showed a higher average oxidation rate than the MDNR Reactor.

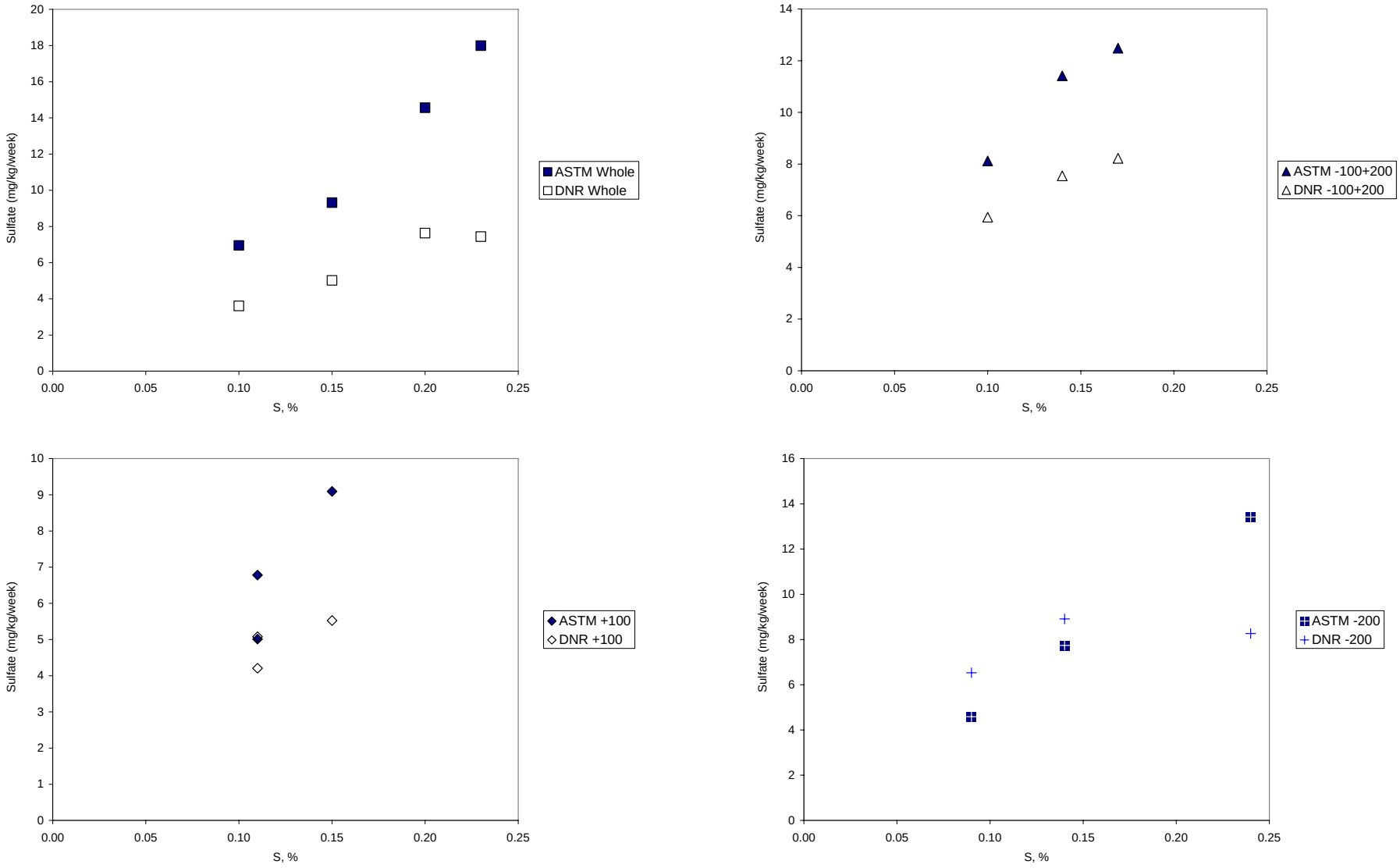
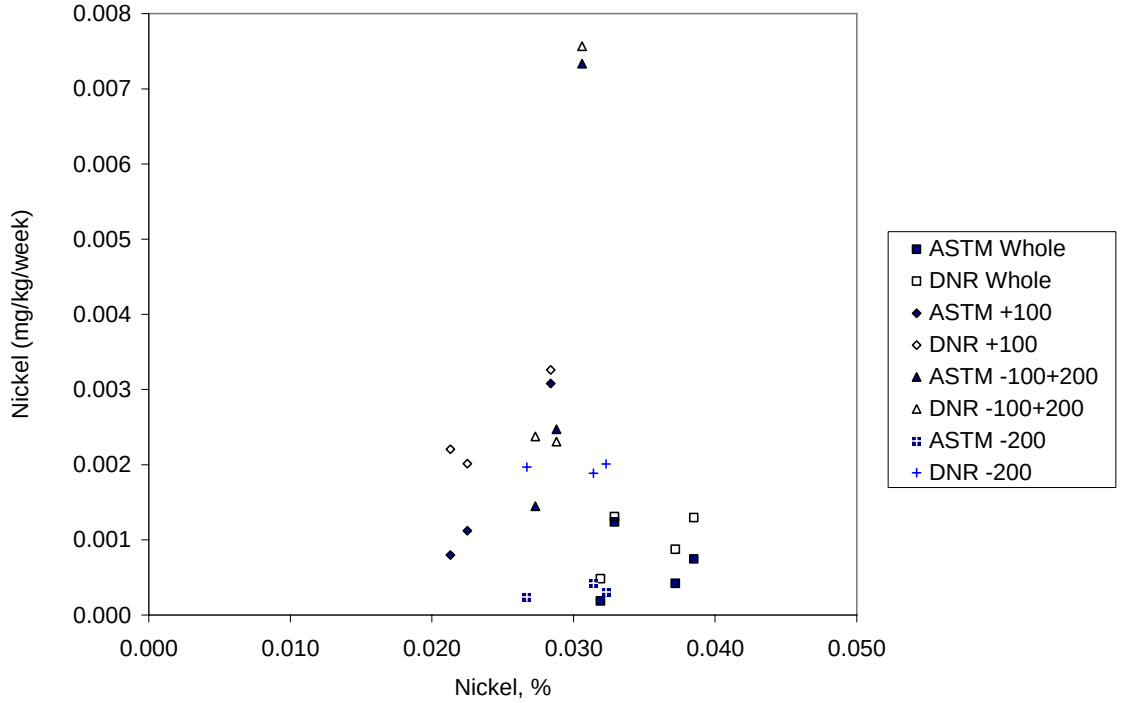


Figure 6-1: Comparison of Average Sulfate Rates as a Function of Sulfur Content for MDNR Reactors and ASTM Humidity cells for Each Size Fraction.



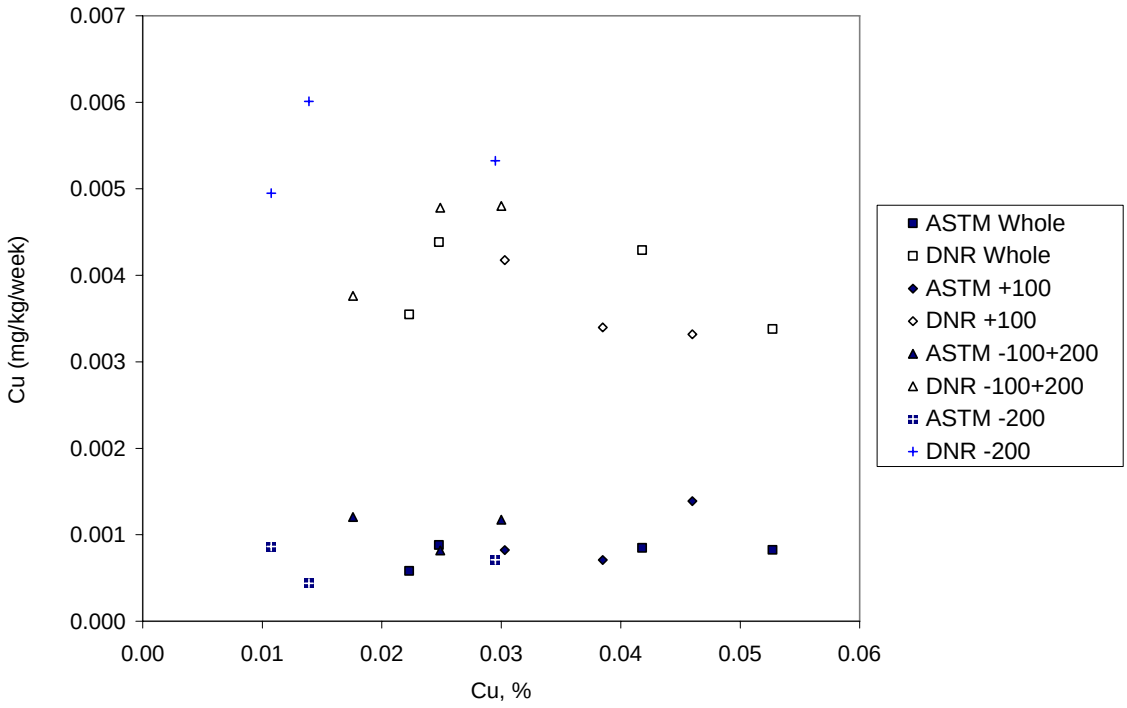


Figure 6-2: Comparison of Average Copper and Nickel Rates as a Function of Copper and Nickel Content for MDNR Reactors and ASTM Humidity cells for Each Size Fraction.

Figure 6-2 compares average copper and nickel release rates with copper and nickel content of the tailings. The average rates for nickel are affected by the increases in leaching that occurred as pH dropped below 7. Copper and to a lesser degree nickel leaching were greater for the MDNR reactors when compared to the ASTM humidity cells.

The difference in sulfate release indicates that some factor is causing oxidation to be enhanced in the humidity cells or that the leaching of sulfate is consistently more efficient for the humidity cells. The main differences that could affect oxidation and leaching is the act of stirring the humidity cell when the deionized water is first applied and the retention of leachate in the cell. Stirring and fluid retention are both expected to lead to more efficient recovery of oxidation products in the humidity cell.

The difference in metal release rates is probably due to the effect of the lower pHs in the MDNR reactors. Actual concentrations are low but the subtle difference in pH appears to have been sufficient to enhance the solubility of copper and nickel.

6.3 Trend Evaluation

6.3.1 ARD Potential

The geochemical processes operating in the tailings are expected to be similar to waste rock because the major mineralogy of the samples is the same as the waste rock. As presented in RS42 (SRK 2007b), the explanation for the lack of acidic conditions in long term (18 year) testwork on rock samples containing less than 0.41% is hypothesized to be due to long term generation of alkalinity by reaction of carbonic acid with silicate minerals (i.e. the conventional process of weathering of silicates as occurs during soil formation, e.g. Drever 1982). In this model, the entire mass of silicate minerals contributes to generation of alkalinity by weathering reactions on plagioclase such as:

$$CaAl_2Si_2O_8 + H_2O + 2H_2CO_3^o \rightarrow Ca^{2+} + Al_2Si_2O_5(OH)_4 + 2HCO_3^-$$

This reaction occurs regardless of whether sulfide minerals are present and effectively surrounds all tailings particles with bicarbonate alkalinity.

For waste rock, this process could be quantified by measuring alkalinity generation from rock containing very low concentrations of sulfide minerals. This rate was then compared to the balancing rate of acid generation.

The critical acid generation rate could be correlated to critical sulfur concentrations (SRK 2007) because acid generation rates have been shown to be well-correlated with sulfur content of the rock (RS42 SRK 2007b, Lapakko and Antonson 2006).

This particular approach could not be applied to tailings because weathering rates for silicate minerals in tailings could not be measured directly. All tailings samples contain some sulfide minerals albeit at low levels and the observed leachate chemistry is a result of oxidation of sulfides

and reaction of the resulting acidity with dissolved alkalinity from silicate mineral weathering and direct reaction with silicate minerals. The alkalinity in leachates from the tests is therefore the net alkalinity remaining after reaction with acidity from sulfide minerals. The presence of alkalinity and pHs above 5.5 in both PolyMet's tests and the longer term MNDR tests on Babbitt Deposit tailings demonstrates that weathering of reactive silicates does occur according to the types of reactions shown above.

A different method was developed to understand the relationship between acid generation from sulfide oxidation, reaction of acidity with silicate minerals and alkalinity provided by weathering of silicate minerals. The following calculation steps were followed:

1. Latest weekly leachate analysis results for each sample were selected. The ASTM humidity cell tests were used because the procedure is believed to provide better recovery of weathering and oxidation products than the MDNR Reactors.

2. The observed leachate chemistry was used to estimate the amount of anorthite (from observed calcium), albite (from observed sodium), forsterite (from observed magnesium), fayalite (from the average composition of olivine, $Fo_{57}Fa_{43}$, indicated by microprobe work presented in RS42 (SRK 2007b) and pyrrhotite (from sulfate) dissolved in the leach cycle. This assumes that these parameters are not retained in any weathering products.

3. The calculated mineral quantities were used as inputs into Geochemist's Workbench to evaluate simulation of actual leachate chemistry. The minerals were assumed to be exposed to the atmosphere allowing excess oxygen and carbon dioxide to react with the minerals. Ferrihydrite and kaolinite were allowed to precipitate.

4. Geochemist's Workbench was found to over-predict the observed alkalinity concentration and the pH of the leachate compared to observed values in the leachate. Therefore, the quantities of reacting silicates were reduced to provide a closer fit to observed alkalinity and pH. The reduction varied from 5 to 45%.

5. The quantity of oxidizing pyrrhotite was then varied from below to above the observed sulfate release and the final pH was calculated using Geochemist's Workbench. The calculation is comparable to a titration in which acid (from pyrrhotite oxidation) is titrated into a source of alkalinity (from silicate weathering).

6. The oxidation rates used in the calculation were then converted to sulfur content using the relationship observed between oxidation rates and sulfide contents (Figure 6-1). Because the calculations were performed on oxidation rates observed after more than a year of testing, the sulfate rate was used to back-calculate the starting sulfur concentration in the sample.

7. The calculated residual pHs were then used to estimate the sulfur content at which the acidity generated would be sufficient to depress pH below that of carbonic acid (approximately 5.5).

Figure 6-3 shows "titration" curves generated for each sample. For each curve, the point used as the basis for development of the curve is shown. The form of the curve shows the transition from excess bicarbonate alkalinity resulting in pHs above 7, to excess acidity in which the pH is near 4 due to buffering by precipitation of alumino-silicates. Each curve represents a unique calculation for the particular sample.

For all particle sizes, the curves shift to the right as the sulfur content of the tailings sample under test increases. If neutralization were occurring only by reaction with dissolved alkalinity, it would be expected that the curves would be very similar or show no relationship to sulfur content. The position of the curves shows that more alkalinity is available at higher sulfur concentrations. This reflects direct reaction of acidity from pyrrhotite oxidation with silicate minerals by reactions of the type:

$$CaAl_2Si_2O_8 + H_2O + H_2SO_4^o \rightarrow Ca^{2+} + Al_2Si_2O_5(OH)_4 + SO_4^{2-}$$

These reactions occur most effectively at lower pHs and will increase in significance as sulfur content increases. This finding indicates that it is not appropriate to estimate critical sulfur contents at which acidity might be produced for each type of tailings based on individual samples, but must also consider the effect of acid consumption by direct reaction with the minerals in addition to dissolved alkalinity from carbonic acid.

Table 6-3 compares actual sulfur content compared to calculated sulfur content equivalent to pHs of 7 and 5.5. These two pHs were selected to provide an indication of the range of sulfur contents associated with the inflexion point on the curves. The table also shows the ratio of the predicted sulfur content to the initial sulfur content to indicate proximity of the actual value to the predicted value. Figure 6-4 shows actual sulfur content of the samples compared to the ratios.

From these calculations, the tendency for low pH conditions increases in a well-defined fashion as the sulfur content increases. For the bulk and coarser tailings samples, the ratio and initial sulfur content are correlated regardless of sample type, and the ratio is below 1 for samples with greater than or equal to 0.2%. This implies that coarse fraction tailings with more than 0.2% sulfur could theoretically generate acid. The -200 mesh tailings show a different correlation at a higher level indicating that sulfur content would need to be higher than for the bulk and coarse tailings before acidic conditions would be apparent.

The finding that 0.2% is a critical level that defines the potential for ARD in tailings is consistent with the MDNR's findings on waste rock that showed Duluth Complex rock from the Dunka Pit with 0.2% sulfur did not generate acid after 18 years of testing. The conclusion is also consistent with similar findings for waste rock presented in RS42.

Further consideration of the potential for acidic conditions to occur in the tailings in the context of the development of the weathering profile is provided in Section 7.2.4.

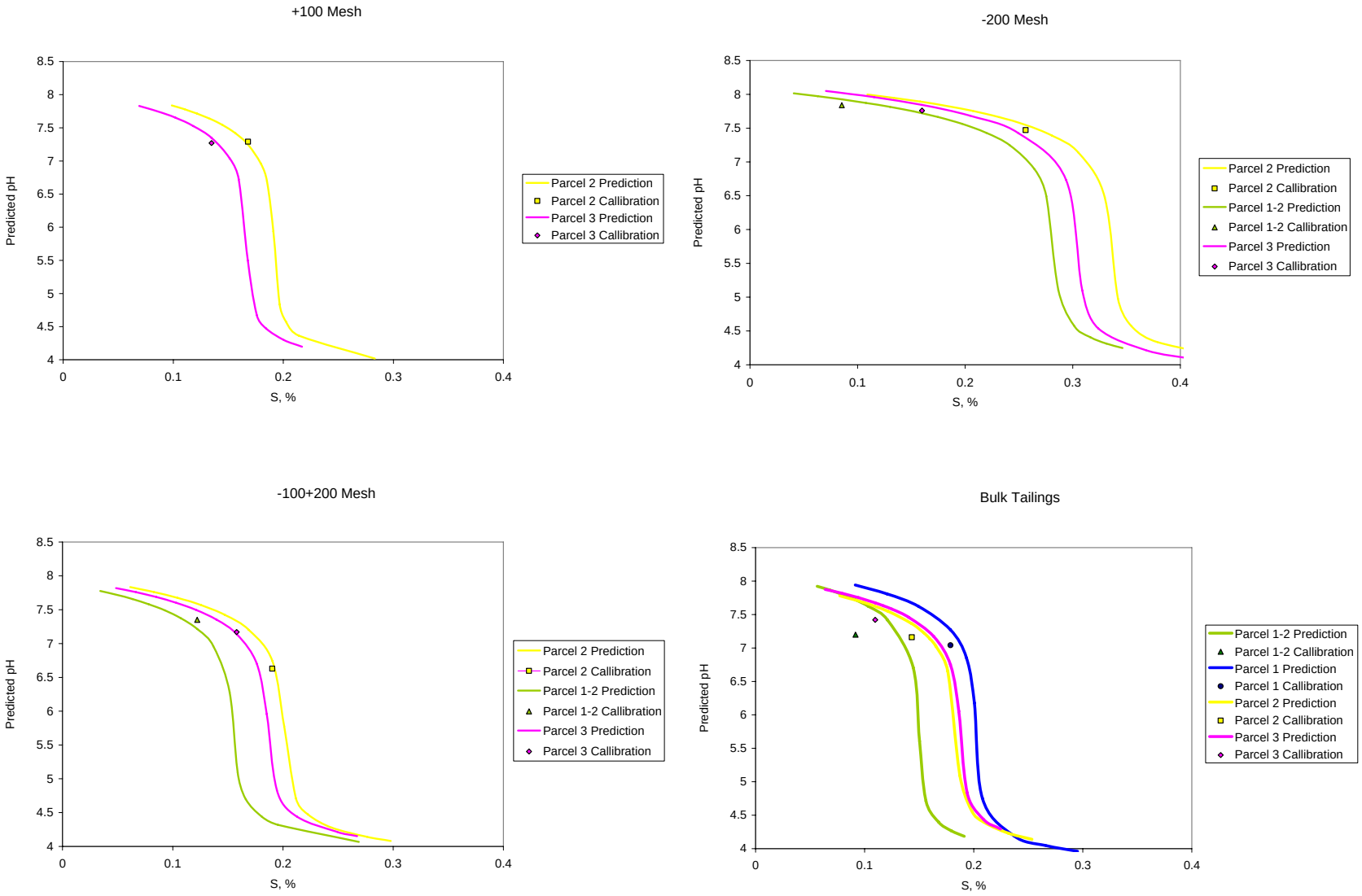


Figure 6-3: Titration Curves for Tailings Samples.

Table 6-3: Comparison of Initial Sulfur Content to Predicted Sulfur Content for pH 7 and 5.5

			S Content Equivalent to Indicated pHs from Titration Curves		Ratio of S Content for Indicated pH to Initial S Content	
Tailings Type	**Ore Package**	**Initial S Content**	**5.5**	**7**	**5.5**	**7**
		%	%	%		
Bulk	P1-2	0.1	0.15	0.14	1.5	1.4
Bulk	P1	0.23	0.21	0.19	0.9	0.8
Bulk	P2	0.2	0.19	0.17	1.0	0.8
Bulk	P3	0.15	0.19	0.18	1.3	1.2
+100	P2	0.15	0.20	0.18	1.3	1.2
+100	P3	0.11	0.17	0.15	1.5	1.4
-100+200	P1-2	0.1	0.16	0.14	1.6	1.4
-100+200	P2	0.17	0.22	0.19	1.3	1.1
-100+200	P3	0.14	0.19	0.18	1.4	1.3
-200	P1-2	0.09	0.29	0.26	3.2	2.9
-200	P2	0.24	0.34	0.33	1.4	1.4
-200	P3	0.14	0.31	0.29	2.2	2.1

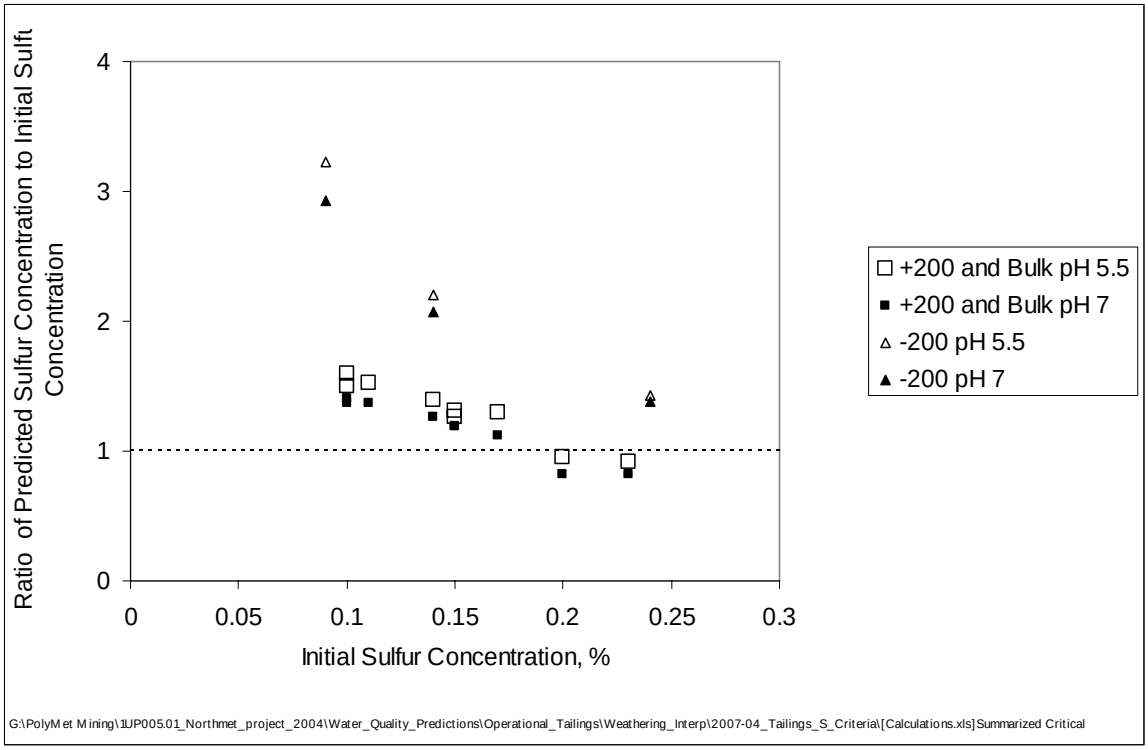


Figure 6-4: Sulfur Content Compared to Ratio of Predicted Sulfur Content to Predicted Content to Initial Sulfur Content

6.3.2 Metal Leaching Potential

Metal leaching potential is clearly related to the pH of leachates. This is demonstrated by the effect of large pH changes in waste rock tests as discussed in RS42 (SRK 2007b), which results in orders-of-magnitude changes in metal concentrations in leachates as predicted from metal solubility. However, it also appears to be important for smaller shifts in pH as shown by the difference in copper and nickel leaching for the ASTM humidity cells and MDNR reactors, the effect of a decrease in pH below 7 for the MDNR's covered Babbitt tailings reactors (Figure 3-1) and the increase in nickel and cobalt leaching as pH decreases below 7 for NorthMet Project coarse tailings samples (Figure 5-1).

The following sections provide a discussion of metal leaching potential for nickel, copper and other parameters.

Release and Solubility of Nickel

Using microprobe data for the nickel composition of olivine and pyrrhotite for the NorthMet Deposit (described in RS42, SRK 2007b), the degree of nickel attenuation was approximated for the NorthMet tailing samples and the MDNR's Babbitt Deposit tailings using magnesium in leachates to indicate olivine leaching and sulfate to indicate pyrrhotite oxidation. Figure 6-5 shows the attenuation of nickel calculated by:

$$1 - \frac{Ni_{leachate}}{Ni_{Olivine} + Ni_{pyrrhoite}}$$

If nickel concentrations in leachate can be completely accounted for by leaching of nickel from olivine and pyrrhotite the attenuation factor would be zero. Positive values indicate attenuation of nickel, while high negative values indicate that more nickel is in the cell leachates than can be accounted for by dissolution of olivine and oxidation of pyrrhotite. A similar approach was used in RS42 (SRK 2007b) to assess nickel leaching from NorthMet Project waste rock.

Figure 6-5 broadly indicates the same trends shown for waste rock. At the start of the tests, nickel was being stored rather than released. The decrease in pH for coarse tailings and covered MDNR reactors resulted in release of nickel beyond the levels that could be accounted for by oxidation of pyrrhotite and dissolution of olivine. This showed that nickel stored in the early stages of the test (and probably also produced in storage prior to the test) was leached, but this was a short term effect. Once this stored load was removed, nickel leaching decreased and the net accumulation of nickel resumed though to a lesser degree than at higher pHs. This effect was not apparent for the bulk and -200 mesh NorthMet tailings and the Babbitt bulk tailings humidity cells due to the higher leachate pHs.

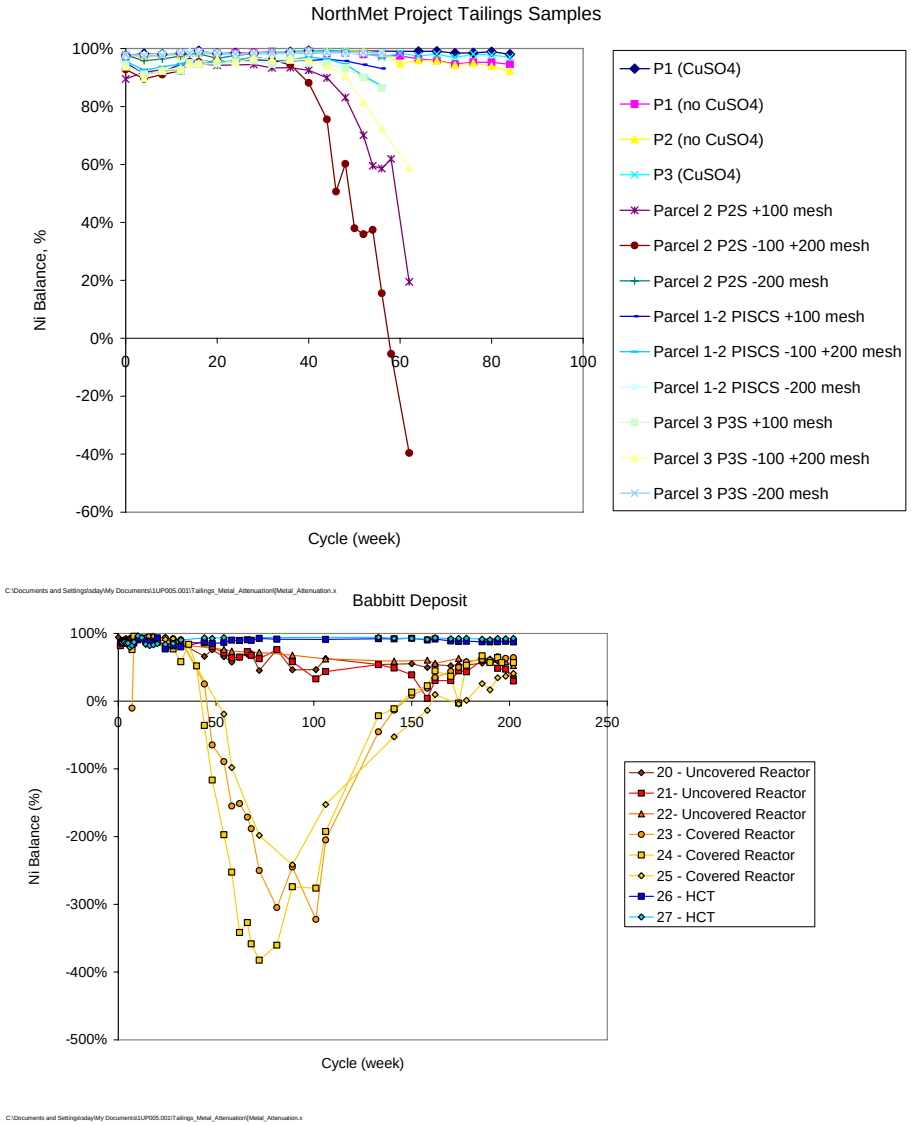


Figure 6-5: Estimated Nickel Attenuation in NorthMet and MDNR Babbitt Tailings Kinetic Tests and Copper Attenuation in NorthMet Project Humidity Cells.

The form of stored nickel was evaluated by comparing nickel concentrations for all types of tests with respect to solubility limits calculated using thermodynamic data for two nickel silicates (nepouite and Ni-kerolite) known to form by weathering ultramafic rocks. The data were determined by Golightly (1981) and provided by Schmiermund (2006). The calculations were performed in Geochemist's Workbench and used to calculate nickel concentration as a function of pH for equations:

Nepouite: $3Ni^{2+} + 5H_2O + 2SiO_2 \rightarrow Ni_3Si_2O_5(OH)_4 + 6H^+$

Ni-Kerolite: $3Ni^{2+} + 4H_2O + 4SiO_2 \rightarrow Ni_3Si_4O_{10}(OH)_2 + 6H^+$

Figure 6-6 shows that highest nickel concentrations in ASTM humidity cells leachates exceeded the solubility of Ni-kerolite by about a factor of three when silica is provided by quartz. This corresponds to a silica concentration of about 5 mg/L which is the same level observed in much of the testwork. The outer edge of the highest nickel concentrations parallels the solubilities of the known minerals indicating that the actual nickel phase involved may be similar to these known nickel silicates (because the slope of the Ni(pH) relationship is the same) but that the thermodynamic data may need to be slightly adjusted to reflect actual solubility.

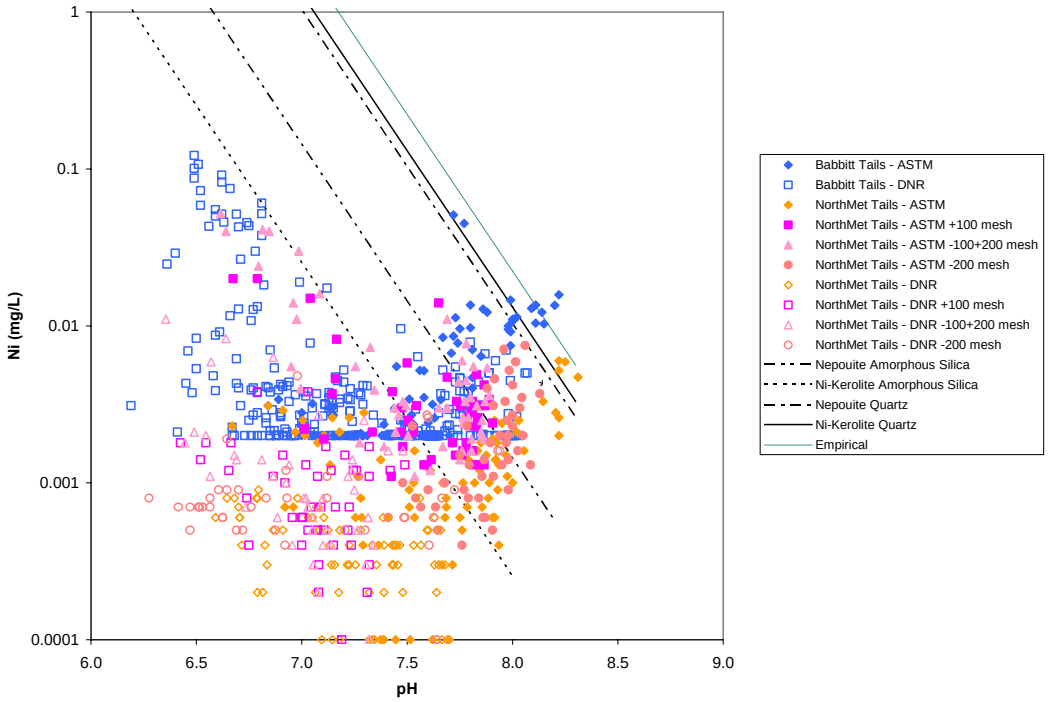


Figure 6-6: Nickel concentrations in kinetic test leachates compared to solubilities of two nickel silicates

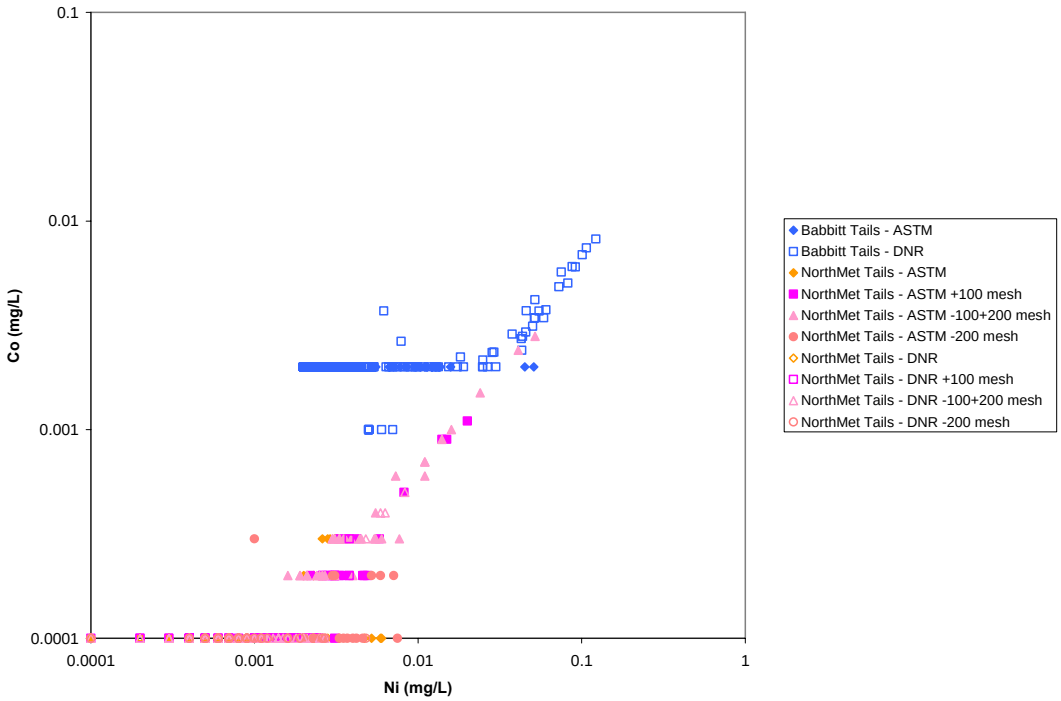


Figure 6-7: Comparison of Nickel and Cobalt Concentrations in Tailings Kinetic Test Leachates

Based on these findings, the increase in nickel release for the NorthMet coarse tailings and the Babbitt covered reactors reflects dissolution of a nickel silicate due to decrease in pH. The peak concentration of 0.12 mg/L is below the solubility of any known nickel minerals including silicates. The rapid but short term release of nickel therefore reflects removal of a finite stored amount of secondary nickel under test conditions.

These results indicate that:

- Exposures of coarser NorthMet tailings can be expected to leach nickel after a delay of several months due to depression of pH below 7.
- Assuming that nickel solubility is controlled by nickel silicates, solubility of nickel below pH 7 will be higher under field conditions than currently demonstrated by testwork. Using Geochemist's Workbench, nickel concentrations could be of the order 2.2 to 2.4 mg/L at pH 6.5, for example, if sufficient leachable secondary minerals are present.

The solubility of cobalt cannot be evaluated using the same methods because it occurs at low concentrations in the minerals and leachates. However, the correlation between nickel and cobalt is very well defined as pH decreases implying that cobalt would show the same behavior as nickel. and allows cobalt concentrations to be predicted from nickel (Figure 6-7). Based on this relationship, cobalt concentrations at pH 6.5 could be about 0.15 mg/L.

Copper Solubility

Neither the MDNR's data for the Babbitt tailings nor the NorthMet Project humidity cells or reactors showed an increase in copper release as pH decreased indicating that the form in which copper is stored in the samples during weathering was not as sensitive to pH as nickel. For the Babbitt tailings the effect may have been obscured by the detection limit of 0.002 mg/L used in the testwork but for the NorthMet Project tailings copper was detected in the leachates. A similar calculation to nickel was performed for NorthMet tailings to assess attenuation. Copper was assumed to originate from olivine and pyrrhotite leaching. Other forms may contribute to copper release (for example, residual chalcopyrite) therefore the calculation tends to underestimate the degree to which copper is attenuated. The calculation showed that copper was attenuated under the weathering conditions in the testwork. It appeared that less than 30% of copper produced by weathering was expressed in the leachates and therefore that at least 70% was stored. As pH decreased, this did not result in the same pattern as observed for nickel. The potential form of stored copper was therefore investigated further.

The solubility of the copper minerals malachite and tenorite which could reasonable be expected to form by weathering of the tailings were evaluated using Geochemist's Workbench. No thermodynamic data were available for copper silicates although by analogy to nickel, any of the numerous known copper silicates might be expected to form.

As shown in Figure 6.8, maximum copper concentrations in the kinetic test database were above the solubility of tenorite but below the solubility of malachite. The two highest copper concentrations in +100 mesh tailings fraction leachates were isolated results that were not part of the trend and are therefore unrelated to dissolution effects. The four highest copper concentrations for bulk tailings in ASTM-style tests were the first flush from the humidity cell. Concentrations dropped rapidly following the first flush indicating that that these results could reflect leaching of pore water from the process rather than oxidation products. Considering these factors, copper concentrations in all other leachates were near or below the solubility of tenorite indicating that this could be the secondary phase controlling copper release.

Based on the solubility of tenorite as a function of pH, the decrease in pH should have resulted in an increase in copper concentrations in kinetic test leachates much like that observed for nickel. However, no increase in copper release was apparent. For the MDNR's experiments, the detection limit may again be a factor but for the NorthMet tailings, copper concentrations were detectable at higher pHs. These findings imply that the secondary form of copper is not tenorite. If copper were co-precipitated with ferric oxides, copper release would not increase until pH dropped low enough to significantly dissolve these minerals. Typical pHs are below 5.

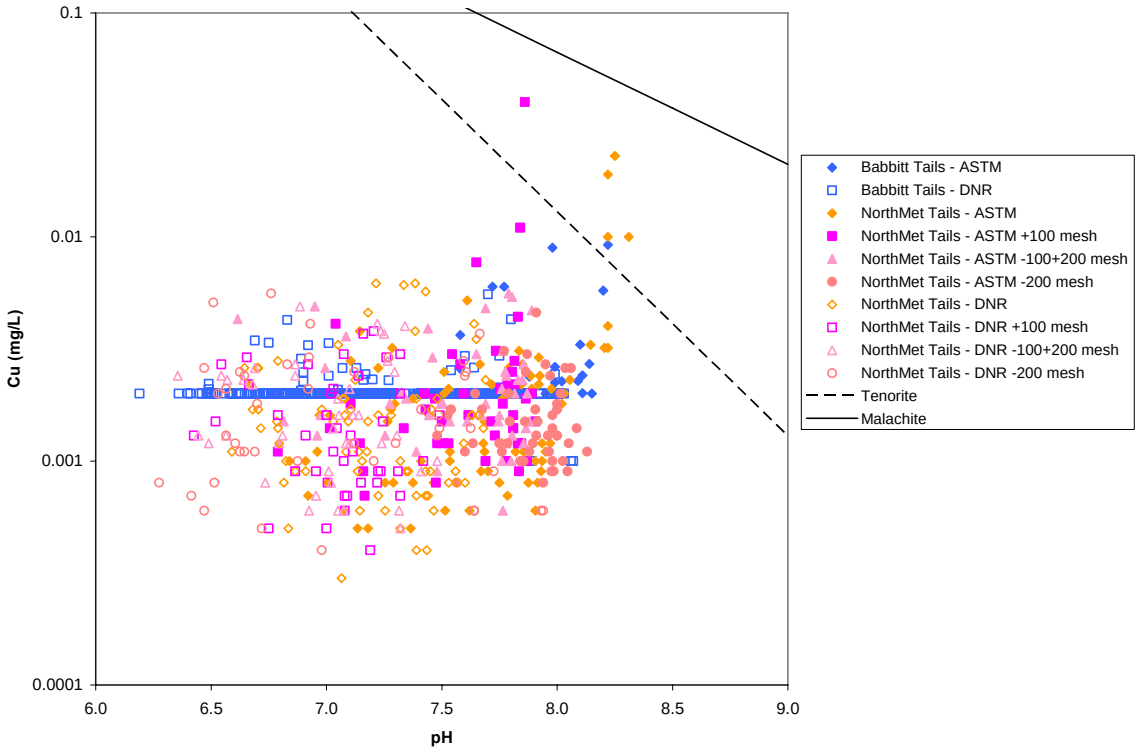


Figure 6-8: Copper concentrations in kinetic test leachates compared to solubilities of tenorite and malachite

Solubility of Major pH-Sensitive Elements

The majority of iron concentrations were below or near the detection limit of 0.01 mg/L with scattered concentrations up to 0.15 mg/L. These concentrations were all above the expected solubility of ferric hydroxide and the concentration expected to result from oxidation of pyrrhotite in the tailings. Iron released from pyrrhotite was therefore precipitated and the resulting sporadic detection of iron was probably due to the formation of iron flocs that passed through the 0.45 µm filter.

Maximum aluminum concentrations for any given pH were negatively correlated with pH but the concentrations were well above the solubility of alumino-silicates or hydroxides that would be expected to form from by weathering of silicates. Aluminum concentrations therefore probably reflect colloids formed by the breakdown of alumino-silicates rather than dissolved aluminum ions.

Solubility of Other Elements

Concentrations of other elements were evaluated with respect to pH but they were too low to be compared with the solubilities of known secondary minerals using the same methods described above for nickel and copper. Instead, the data distributions was reviewed to determine if any relationships existed and how these might relate to the solubilities of major secondary minerals.

The majority of elements showed no relationship to pH. Highest manganese concentrations for NorthMet tailings also showed a negative correlation with pH though at low concentrations. The negative correlation with pH was consistent with formation and dissolution of secondary minerals but concentrations were below their solubility.

In contrast, the highest arsenic, antimony and selenium concentrations at any given pH were positively correlated with pH. Because these elements occur as oxyanions, increasing solubility with higher pHs in natural range is expected. The trend is therefore consistent with this effect. Arsenic and antimony concentrations were well below the solubility of known arsenic and antimony minerals (for example, scorodite and antimony oxide) as is expected because arsenic and antimony content of the tailings is very low (average 2 mg/kg and 0.2 mg/kg, respectively) relative to iron as sulfide (3000 mg/kg). Therefore, the solubility of both elements from solution is expected to be controlled by the low solubility of ferric hydroxides at slightly basic pHs rather than discrete antimony and arsenic minerals.

6.4 Comparison of Results with MDNR Testwork Programs

Results from MDNR's testwork on tailings from the AMAX test shaft and tailings from Cominco's pilot plant processing of the Babbitt Deposit, and PolyMet's pilot plant should be compared cautiously due to potential differences in the ores and generation and management of the pilot tailings products. However, qualitatively, it is worth comparing order-of-magnitude ranges of rates for the various tests to determine if the results are similar though use of low detection limits by PolyMet provided better definition of metal leaching rates. Table 6-4 shows that all test types except

the MDNR covered reactors for nickel leaching had comparable ranges. The rates indicated by the field test are probably affected to some degree by formation of secondary salts and are therefore lower than actual generation rates, but the calculated rates are of the same order as the laboratory tests indicating comparable behavior under field and laboratory conditions.

The MDNR covered reactors showed a markedly different nickel leaching trend as well as a higher sulfate rate (Figure 3-1). Both features may be related to the lower pH of leachate in this test which both allowed stored nickel to be released (Figure 6-5) and caused pyrrhotite oxidation to accelerate resulting in depletion of 55% of sulfur compared to 29% and 38% of sulfur in the humidity cells and uncovered reactors on the same tailings samples. The explanation of the different behavior of the same sample tested under the different conditions is believed to be the retention of moisture resulting from lack of evaporation between cycles. This effect is shown in Figure 6-9 by the consistently higher recovery of leachate from covered reactors. The higher recovery indicates retention of leachate which in turn allows residual carbonic acid to accumulate in the reactors between cycles compared to the uncovered reactors that evaporate the residual moisture allowing less retention of carbonic acid.

The overall effect is that lower pH conditions developed in the covered reactors due to the high liquid to solid application rate compared to the uncovered reactors. Because liquid application rate on an area basis (2700 mm/year, Table 6-2) far exceeds precipitation at the tailings pond (711 mm/year) the covered reactors are an artificial condition both in terms of water application and covering to reduce evaporation. Evaporation will conceivably be lower on the tailings during the winter months but under these conditions reaction rates in the tailings will be greatly curtailed by low temperatures and significant portions of the tailings will be frozen which will result in reduced or near zero infiltration.

The MDNR's testwork performed on whole tailings samples was not directly comparable to the testing being performed on NorthMet Project tailings size fractions. The recent trends in nickel and cobalt release from the -100+200 mesh and +100 mesh tailings samples appear to be similar to the trends observed for the MDNR's covered reactor test. The effect in the coarser size fractions is probably due to the lower surface area available for silicate weathering. The current lack of a similar effect for the -200 mesh samples is consistent with this conclusion because this fraction will have a much higher effective silicate mineral surface area to contribute alkalinity. The effect for the coarse fractions is therefore due to the physical characteristics of the samples rather than excessive water application in the case of the MDNR's covered reactors.

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Table 6-4: Comparison of Rate Ranges for Babbitt Deposit and PolyMet Deposit Tailings Samples

	Humidity Cell (mg/kg/week)				MDNR Reactor (mg/kg/week)				MDNR Reactor (Covered) (mg/kg/week)				Average Rates for Field Test (mg/kg/week)			
Test Program	**pH**	**Cu**	**Ni**	**SO_4**	**pH**	**Cu**	**Ni**	**SO_4**	**pH**	**Cu**	**Ni**	**SO_4**	**pH**	**Cu**	**Ni**	**SO_4**
AMAX Babbitt Shaft	-	-	-	-	-	-	-	-	-	-	-	-	6.3 – 8.3	0.00015	0.0005	5.2
Cominco Babbitt	6.8 – 8.2	<0.001 – 0.003	<0.001 – 0.01	4 – 17	6.7 – 7.9	<0.004 – 0.01	<0.004 – 0.02	2 – 42	6.2 - 8.0	<0.004 – 0.01	<0.004 – 0.3	3 – 72	-	-	-	-
NorthMet (Whole Tailings)	6.4 – 8.3	0.0003 to 0.003	0.00004 – 0.001	2 – 28	6.5 – 7.8	0.0008 – 0.01	0.0002 – 0.002	2 – 12	-	-	-	-	-	-	-	-

Notes:

1. Isolated high values not part of a trend were excluded for the MDNR data.

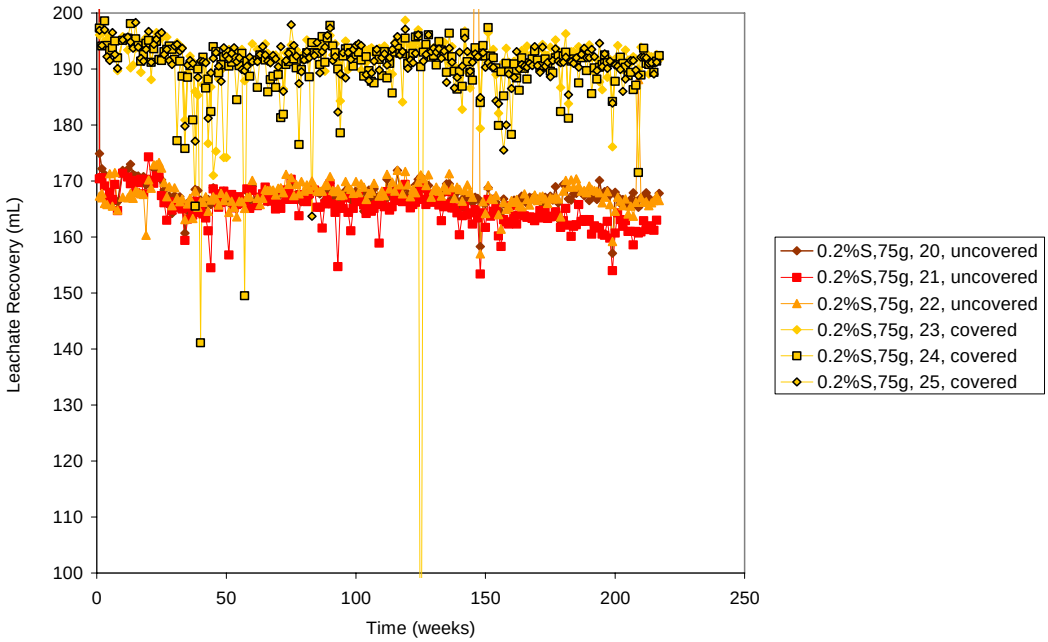


Figure 6-9: Leachate Recovery from MDNR Reactors Containing Babbitt Tailings

6.5 Interaction of PolyMet Tailings Leachate with LTVSMC Tailings

Table 6-5 shows saturation indices for leachates from the control column containing Coarse LTVSMC taconite tailings calculated using Geochemist's Workbench (Bethke 2005). Leachate chemistry for the fine tailings control was similar and therefore shows similar saturation indices. Saturation indices exceeding 0 for dolomite, strontianite, calcite and magnesite indicate that leachate chemistry is controlled by the dissolution of abundant carbonate. Ankerite and siderite would also be expected to be dissolving but iron concentrations were low and the leachates were not chemically reduced indicating that iron was present in ferric form. It is probable that the iron-bearing carbonates are dissolving but the iron is oxidized to ferric and immediately precipitated. The week 1 cycle shows that ferrihydrite ($Fe(OH)_3$) was over-saturated which is consistent with precipitation of this mineral.

Another major component of the taconite tailings is quartz. Silica concentrations are consistent with dissolution of some form of silica. Other minerals that were close to saturation in the early stages were barite and fluorite. Fluoride concentrations in particular probably indicate that a fluoride-bearing mineral is present and dissolving. Berndt et al (1999) suggested that fluoride may be present as fluorite or a sorbed phase. Gypsum is not a known component of the LTVSMC tailings, and sulfate concentrations were well below the saturation level for gypsum. Other gypsum-bearing minerals could be contributing to sulfate concentrations, or the results could reflect flushing of sorbed sulfate as overall sulfate concentrations in the column leachate decreased.

Table 6-5: Saturation Indices for Coarse LTVSMC Tailings (Control Column)

Mineral	Week 1	Week 6	Week 36
Hematite[1]	11.4	-	-
Dolomite	2.5	3.0	3.1
$Fe(OH)_3$[1]	0.8	-	-
Strontianite	0.6	1.0	1.0
Calcite	0.5	0.8	1.1
Magnesite	0.4	0.6	0.4
Quartz	0.3	0.4	0.2
Chalcedony	0.0	0.1	0.0
Barite	-0.1	-	-1.3
Fluorite	-0.3	-1.1	-1.9
Rhodochrosite	-0.4	-0.3	-1.1
Amorphous Silica	-1.0	-0.9	-1.1
Gypsum	-1.2	-2.3	-2.1

Notes:
1. Iron was not detected in these leachates

The main process that occurred as NorthMet tailings water entered the LTVSMC tailings is that concentrations increase as the soluble components of the taconite tailings add to the relatively dilute NorthMet tailings leachate. The only exception to this finding is nickel, which showed decreasing concentrations as the NorthMet tailings leachate enters the LTVSMC tailings. This effect was evaluated by calculating saturation indices for nickel minerals to see if there was evidence that nickel was forming a discrete phase. Results for this calculation of leachates produced at week 20 for the P3 NorthMet tailings entering the coarse taconite tailings are shown in Table 6-6 with particular emphasis on nickel minerals. The thermodynamic database was augmented using data for two additional nickel silicate minerals (nepouite and Ni-kerolite) provided by R. Schmiermund (personal communication).

Leachate leaving the NorthMet P3 tailings was dilute and undersaturated with respect to most of the minerals, which subsequently become saturated in contact with the taconite tailings. The exception is quartz, which was calculated to be saturated in both column leachates. Amorphous silica however was undersaturated for both column leachates. Silica concentrations increased from 10 to 15 mg/L by contact with the taconite tailings.

Nickel concentrations were well under-saturated with respect to all the nickel minerals in the database, which indicates it is very unlikely that these minerals are forming and resulting in the observed decrease in nickel concentration. A more likely explanation is that the dissolution of iron-bearing carbonates and the subsequent formation of ferrihydrite is removing nickel from solution by co-precipitation and adsorption. The elevated pHs of the column leachates provides good conditions for removal of metals by this process. If this is occurring, the LTVSMC tailings have a large capacity to attenuate nickel. Because the first sampling port in the taconite tailings did not show break through in 36 weeks of testing, it is not possible to calculate the attenuation capacity of the taconite tailings. A minimum attenuation capacity of 0.02 mg Ni/kg of taconite tailings was calculated for the duration of the P3 NorthMet tailings to coarse taconite tailings experiment.

Table 6-6: Saturation Indices for NorthMet P3 Tailings Leachate into Coarse LTVSMC Tailings (Final Leachate)

Mineral	P3 Tailings Week 20	Coarse Taconite Tailings Week 20
Dolomite	-1.14	3.30
Calcite	-0.84	1.24
Strontianite	-1.15	1.14
Magnesite	-1.93	0.43
Quartz	0.22	0.39
Rhodochrosite	-1.95	-0.45
Amorphous silica	-1.07	-0.90
Ni-Kerolite	-3.15	-1.29
Ni_2SiO_4	-2.70	-1.74
Nepouite	-3.28	-1.77
NiO	-4.74	-4.35
$Ni(OH)_2(s)$	-5.07	-4.68
$NiCO_3$	-6.36	-5.59

In summary, the contact experiment showed that leachate produced by the LTVSMC taconite tailings is dominated by alkalinity produced by the dissolution of carbonates. The NorthMet tailings had no perceivable effect on the major ion chemistry because the taconite minerals are considerably more soluble than the silicates in the NorthMet tailings. Only nickel leached more readily from the NorthMet tailings but was attenuated by contact with the taconite tailings probably due to sorption effects.

6.6 Conclusions

The following were concluded from testing performed by MDNR on Babbitt tailings and by PolyMet on NorthMet Project tailings:

- NorthMet tailings will consist of crushed Duluth Complex minerals, which are mainly plagioclase and olivine. Pyrrhotite is a minor component, which accounts for the sulfur content of the tailings. Carbonate minerals are virtually undetectable in the tailings, which is consistent with the magmatic origin of the deposit.

- Long term leachate chemistry can be explained by the oxidation of pyrrhotite and weathering of silicate minerals. The former produces sulfuric acid whereas the latter produces secondary silicate minerals and dissolved alkalinity.

- Because the weathering processes produced weak leachates, test protocol has a significant effect on leachate chemistry. The ratio of deionized water to sample solid affects pH because a high ratio introduces a greater amount of carbonic acid during the leach cycle. Lower pHs occur when the liquid to solid ratio is high.

- The effect of lower pHs is to increase the solubility of metals and in particular nickel. The testwork indicates that as pH decreases below 7, nickel stored in weathering products at higher pHs is leached resulting in a spike in nickel release lasting possibly 2 years.

- Based on this finding, metal leaching is best represented by the ASTM humidity cells and has been used as the input to subsequent predictions of operational tailings beach runoff and pore water aqueous chemistry at closure.
- Nickel leaching appears to be explainable by dissolution of nickel silicates.
- The overall finding is that NorthMet tailings generally containing less than 0.2% sulfur and produced using the copper sulfate process to enhance recovery of pyrrhotite will be very unlikely to generate acid. This finding is consistent with the MDNR's waste rock and tailings testing which has not shown generation of acidic leachate for rock or tailings containing less than 0.41% sulfur.
- Coarse tailings (+100, -100+200 mesh) appear to be susceptible to moderate pH decrease below 7 resulting in enhanced leaching of nickel and cobalt.
- The experiment to evaluate the interaction of NorthMet tailings leachate with the LTVSMC tailings showed that the LTVSMC tailings are more soluble and can be expected to produce alkaline leachate due to the presence of carbonates. The NorthMet tailings produced weaker leachates with lower levels of metals with the exception of nickel. Nickel leached from the NorthMet tailings was attenuated by contact with the taconite tailings.

6.7 Recommendations for Future Testing

Prediction of tailings water chemistry provided in Section 7 relies on scale-up of laboratory testwork on pilot plant tailings samples to tailings produced at full-scale weathering under field conditions.

Once the plant is in full production, a program of characterization of the performance of full-scale plant tailings using field test plots and laboratory tests should be designed to extend the application of ongoing test work to the operating tailings basin.

7 Water Chemistry Modeling

7.1 Introduction

Iterative modeling of the geochemical behavior of the tailings solids resulted in selection of an approach that overlaps the operational and closure periods. The original format of the outline agreed with the MDNR has been replaced with the following structure:

- Description of the detailed modeling used to predict the weathering and leaching behavior of tailings solids; and
- Description of the coupling of the result from this modeling with prediction of the chemistry of the process water in the tailings pond.

7.2 Leaching Behavior of Tailings Solids

7.2.1 Approach

Over time, the fine and coarse tailings will respond differently to the physical and geochemical processes that will control water quality. The coarser sandy tailings have a higher porosity and are more permeable. As a result, water will infiltrate and drain away more readily, and oxygen will diffuse more rapidly and deeper into these tailings. The finer silty tailings will be less permeable than the coarser tailings and, as a result, will more effectively retain moisture and reduce the rate of oxygen diffusion.

Sulfide minerals, when exposed to ambient conditions (air and water), will oxidize to form free acid (sulfuric acid) which will react with neutralizing minerals. Under neutral but oxidizing conditions, iron released by the oxidation of pyrrhotite would be expected to form iron oxy-hydroxides which will limit the solubility of iron. These oxy-hydroxides also have a potential to co-precipitate or sorb dissolved metals that may be released from sulfide oxidation. However, where conditions are not sufficiently oxidizing, it may be possible that iron could remain in solution as ferrous iron. The oxidation of galena (which is present in trace amounts in the PolyMet tailings, see Section 5.1.2) could result in the dissolution of lead; however, under conditions of elevated sulfate concentrations it is expected that anglesite would be formed and lead concentrations would remain low. Similarly, the oxidation of chalcopyrite could lead to the release of soluble copper; however its concentration would be limited at neutral pH conditions by the formation of secondary minerals.

Conservatively, however, for the parameters addressed in this report, solubility controls were generally disregarded and the mass release rates, as determined from the kinetic leach tests, were utilized directly in the oxidation release calculations. Conclusions on metal mobility provided in Section 6.3.2 provide a basis for evaluation predictions of tailings pore water chemistry.

Therefore, to determine the water quality of seepage from the tailings, the following steps were undertaken:

- First, the geochemical properties and kinetic test results were used to establish the rates at which the sulfide minerals oxidize and to estimate the associated metal and other solute release rates.
- Second, because the moisture content has a major impact on the rate at which oxygen may diffuse into the tailings, the average moisture content of the coarse and fine tailings were estimated using the physical properties of the tailings. The moisture content and physical properties were then used based on well-established correlations to establish the effective oxygen diffusion coefficients for the coarse and fine tailings respectively.
- Third, oxygen diffusion calculations were then undertaken to determine the rate of oxygen consumption (i.e. sulfide mineral oxidation rates) with depth. These were then used together with the estimated solute release rates to determine mass loadings to the porewater in the coarse and fine tailings respectively. This yielded porewater concentrations in the unsaturated fine and coarse tailings respectively.
- Fourth, infiltration rates to the coarse and fine tailings during deposition were estimated based on the depositional strategy and assumed beach conditions. These estimates were used in the updated RS13 (Draft 03) (Barr 2007c) to estimate transport rates of surface infiltration to the base of the LTVSMC tailings. The transport rates were then used to estimate the volume of seepage from each of the coarse and fine tailings beaches, and combined with the estimated porewater concentrations determined in the previous step to estimate solute concentrations in the seepage.

In the following sections, first the intrinsic oxidation rate of the tailings is established. The physical properties of the tailings are then used to determine potential controls on oxygen diffusion into the tailings. The outcomes from these assessments are then used to determine the inputs for the diffusion modeling which is used to estimate sulfide depletion rates.

7.2.2 Geochemical and Physical Test Results

Geochemical Properties and Oxidation Rates

The humidity cell test results provide some insight into the geochemical mechanisms that could prevail in the tailings and can be used to calculate the intrinsic oxidation rates.

The oxidation rates were estimated from the final 20 cycles of testing to represent a 'steady state' oxidation rate. The oxidation rates were calculated assuming the pyrrhotite is oxidized first according to the following reaction:

$$FeS(s) + 2\,O_2 \rightarrow FeSO_4\,(aq) \qquad \ldots(1)$$

The ferrous iron is then oxidized to ferric and precipitated as ferri-hydrate as follows:

$$2Fe^{2+} + ½\ O_2 + 5H_2O \rightarrow 2Fe(OH)_3\ (s) + 4H^+ \quad ...(2)$$

With the overall reaction as follows:

$$2\ FeS(s) + 9/2\ O_2 + 5H_2O \rightarrow 2Fe(OH)_3\ (s) + 2H_2SO_4 \quad ...(3)$$

Overall oxidation rates, expressed as oxygen uptake rates, were calculated and are summarized in Table 7-1.

Table 7-1: Summary of Tailings Oxidation Rates

Test Type	Fraction	Humidity Cell Test Full ID	Acid Potential kg $CaCO_3$/t	S %	SO_4 (mg/kg/wk)	Oxidation Rates (mol O_2/kg/s)
MDNR Reactor	+100	Parcel 1-2 PISCS +100 mesh	3.4	0.11	4.44	$1.72x10^{-10}$
MDNR Reactor	+100	Parcel 3 P3S +100 mesh	3.4	0.11	5.24	$2.03x10^{-10}$
Humidity Cell	+100	Parcel 3 P3S +100 mesh	3.4	0.11	6.11	$2.37x10^{-10}$
Humidity Cell	+100	Parcel 1-2 PISCS +100 mesh	3.4	0.11	4.61	$1.79x10^{-10}$
Average			3.4	0.11	5.10	$1.98x10^{-10}$
MDNR Reactor	-100+200	Parcel 1-2 PISCS -100 +200 mesh	3.1	0.1	5.78	$2.24x10^{-10}$
MDNR Reactor	-100+200	Parcel 3 P3S -100 +200 mesh	4.4	0.14	7.90	$3.06x10^{-10}$
Humidity Cell	-100+200	Parcel 1-2 PISCS -100 +200 mesh	3.1	0.1	6.22	$2.41x10^{-10}$
Average			3.5	0.11	6.63	$2.57x10^{-10}$
MDNR Reactor	-200	Parcel 1-2 PISCS -200 mesh	2.8	0.09	8.39	$3.25x10^{-10}$
MDNR Reactor	-200	Parcel 3 P3S -200 mesh	4.4	0.14	10.1	$3.91x10^{-10}$
Humidity Cell	-200	Parcel 1-2 PISCS -200 mesh	2.8	0.09	5.84	$2.26x10^{-10}$
Humidity Cell	-200	Parcel 3 P3S -200 mesh	4.4	0.14	16.6	$6.43x10^{-10}$
Average			3.6	0.12	10.2	$3.96x10^{-10}$
MDNR Reactor	Whole	P1S	3.1	0.10	3.45	$1.34x10^{-10}$
MDNR Reactor	Whole	P3S Lapakko	4.7	0.15	5.08	$1.97x10^{-10}$
Humidity Cell	Whole	P1 (CuSO4)	3.1	0.10	7.45	$2.89x10^{-10}$
Humidity Cell	Whole	P3 (CuSO4)	4.7	0.15	10.5	$4.06x10^{-10}$
Average			3.9	0.13	6.62	$2.56x10^{-10}$

As shown in the table, although the oxidation rates fall within a relative narrow range, the results suggest that the fine tailings on average may oxidize more rapidly than the coarse tailings.

These oxidation rates were established for laboratory conditions where a relatively thin layer of tailings was assessed. Under these conditions, the oxidation rates would not be expected to be limited by oxygen availability. Under field conditions once the tailings have been deposited, the near surface tailings would react at the rates indicated. However, at greater depth the rate of oxidation would be limited by the rate at which oxygen diffuses into the tailings. The rate of diffusion would depend on a number of factors, including the porosity, permeability and precipitation which would dictate the degree of saturation of the tailings.

Diffusion Coefficient

The effective diffusion coefficient for oxygen into tailings was calculated from the equation presented by Elberling et al. (1993). The Elberling equation is:

$$D_e = \tau D_a [1 - S_w]^c + S_w D_w / K_H \quad ...(4)$$

Where:

D_a is the free diffusion coefficient of oxygen in air (2.0_10_ 5 m^2/s, Cussler, 1997),

D_w is the free diffusion coefficient of oxygen in water (1.8_10_ 9 m^2/s, Cussler, 1997),

τ is a tortuosity factor (~0.3),

c is an empirical coefficient (~3.3),

S_w is the water saturation, and,

K_H is Henry's constant for oxygen.

The equation provides a bulk effective diffusion coefficient corresponding to parallel diffusion in the gas and liquid phases, with the right-hand term ($S_w D_w / K_H$) corresponding to the liquid phase. Differences between the tortuosity factors for the liquid and gas phases are accounted for by the parameter a.

Therefore, to estimate the effective diffusion coefficient it is necessary to determine the level of saturation of the tailings. During tailings deposition, the spigot points will be cycled around the perimeter embankment of the tailings deposition area. Each 'active area' (i.e. down-slope from the spigot point) will therefore for a short time receive excess water during the deposition period over the area that the deposition fan will develop. Thereafter, the rate of infiltration will be dictated by the site precipitation.

The average site precipitation is about 28.2 inches, with a net precipitation of about 8.2 inches. Depending on the final surface of the tailings once deposition ceases, the net infiltration to the tailings would be expected to vary, and the degree of saturation would change accordingly. There is likely to be seasonal variation in the tailings moisture content.

HYDRUS-2D modeling was undertaken to estimate the saturation profiles that may develop respectively in the coarse and fine tailings. The modeling conditions and summary results are provided in Appendix D.1. The results indicated that the coarse tailings are relatively free draining and the level of saturation would be expected to decrease rapidly to about 38% of saturation. The fine tailings however will remain relatively saturated at about 90% of saturation.

A summary of the tailings porosity, permeability and density is provided in Table 7.2. The table also shows the estimated effective diffusion coefficients, calculated from Equation 4.

Table 7-2: Summary of Average Coarse and Fine Tailings Properties

Zone	Units	Coarse Tailings	Fine Tailings
Porosity	unitless	0.480	0.500
Bulk Dens	kg/m^3	1.560	1.500
Permeability	m/s	$1.20x10^{-5}$	$2.24x10^{-7}$
Saturation	%	38%	89%
D_e	m^2/s	$1.02x10^{-6}$	$3.54x10^{-9}$

It should be noted that oxygen ingress to the tailings will also be affected by snow accumulation and by freezing conditions should sufficient moisture remain in the tailings prior to winter freeze-up.

In the next section, the rate of oxygen diffusion is assessed.

7.2.3 Oxygen Diffusion Modelling

The rate of oxygen transport into the tailings by diffusion is governed by Fick's law. Integrating the one-dimensional form of Fick's Law and incorporating a first order oxygen consuming reaction leads to the conservation equation:

$$\frac{dC}{dt} = \frac{dC}{dx}\left(D \frac{dC}{dx}\right) - rC \qquad ...(5)$$

Where:

C is the oxygen concentration

t is time

D is the effective diffusion coefficient

x is depth and r is the reaction rate constant.

A numerical solution for this equation was coded in Visual Basic in an Excel spreadsheet.

The average reaction rate constant for the tailings was calculated from the humidity cell tests as $3.38x10^{-8}$ s^{-1} for the coarse tailings, and $6.50x10^{-8}$ s^{-1} for the fine tailings.

Two correction factors were applied to the reaction rate constant. First, the average temperature in the tailings is expected to be well below room temperature at which the humidity cell tests were conducted. The temperature in the tailings are expected about 15 ^oC lower than the test conditions and, using the Arrhenius equation, it can be shown that the reaction constant will be lower by a factor of about 0.3 at the lower temperature. Second, for about 3 to 4 months of the year the tailings are expected to be frozen and/or covered by snow, which will severely restrict the diffusion of oxygen into the tailings. Effectively, this will reduce the annual average oxidation rate by a factor of about 0.75. Therefore, the effective reaction rate constant was obtained by multiplying the laboratory determined rate by 0.3 and 0.75 respectively, as shown in Table 7-3. It should however

be noted that during spigoting the tailings beneath the deposition delta will be saturated which will further reduce oxygen ingress and thus oxidation of the tailings. Conservatively, this effect has been disregarded in the current evaluation.

Table 7-3: Summary of Assumed Effective Reaction Rate Constants

Zone	Calibration Reaction Rate Constant (s^{-1})	Temperature Correction	Frozen Conditions	Effective Reaction Rate Constant (s^{-1})
Coarse	3.38×10^{-8}	0.3	0.75	7.61×10^{-9}
Fines	6.50×10^{-8}	0.3	0.75	1.46×10^{-8}

An initial sulfide content of 0.11 % was assumed for the coarse tailings, and 0.12 % for the fine tailings.

Using these starting conditions, the oxygen concentration profiles with depth and the corresponding sulfide oxidation in the tailings with time were calculated. The results are summarized in Appendix D.1. Example plots of the oxygen concentration in the pore gases are shown in Figure 7.1 at years one, five and ten. The progress of the sulfide depletion front in the coarse tailings is in Figure 7.2. The corresponding plots for the fine tailings are provided in Figure 7.3 and Figure 7.4.

It is important to note that the depth of oxygen penetration into the coarse tailings essentially extends to 20 m (~65 ft) after one year; and after five years to the base of tailings column with a height of 30 m (i.e. ~100 ft) column indicating that the tailings would oxygenate very rapidly and would remain oxygenated. In the context of the construction sequence of 5 m (15 ft) lifts, it is apparent that the tailings would tend to oxidize through the entire column. This would also mean that iron would be oxidized to ferric and would therefore not be mobile provided neutral pH conditions prevail. In contrast to the coarse tailings, the depth of oxygenation of the fine tailings is limited to the near surface (note y-axis full-scale is 10 m) and the rate of oxidation will be limited by the flux of oxygen and no oxidation would be expected at depth.

The rate of depletion is significantly faster in the coarse tailings than in the fine tailings as shown in Figure 7.2 and Figure 7.4. Sulfide depletion would commence from the surface layer of the coarse tailings in about 55 years, and the sulfide in the coarse tailings could be depleted to a depth of 15 m (~ 45 ft) in about 160 years. Sulfide depletion in the near surface fine tailings is expected to commence after about 30 years because the fine tailings are more reactive than the coarse tailings. However, due the limitations on oxygen diffusion, the sulfide depletion would extend only to a depth of about 1 m (~3 ft) after 160 years (note the 'steps' in the plot are a consequence of the discritization of the tailings column in the numerical model). It is also apparent that the rate of depletion slows down over time. Therefore, because the coarse and fine tailings have similar sulfide mineral contents, it is apparent that the coarse tailings represent a significantly greater source of oxidation related solute release.

The oxidation rates obtained by this analysis were then used in conjunction with laboratory determined metal release ratios to estimate porewater quality within the oxidation zone as discussed subsequently.

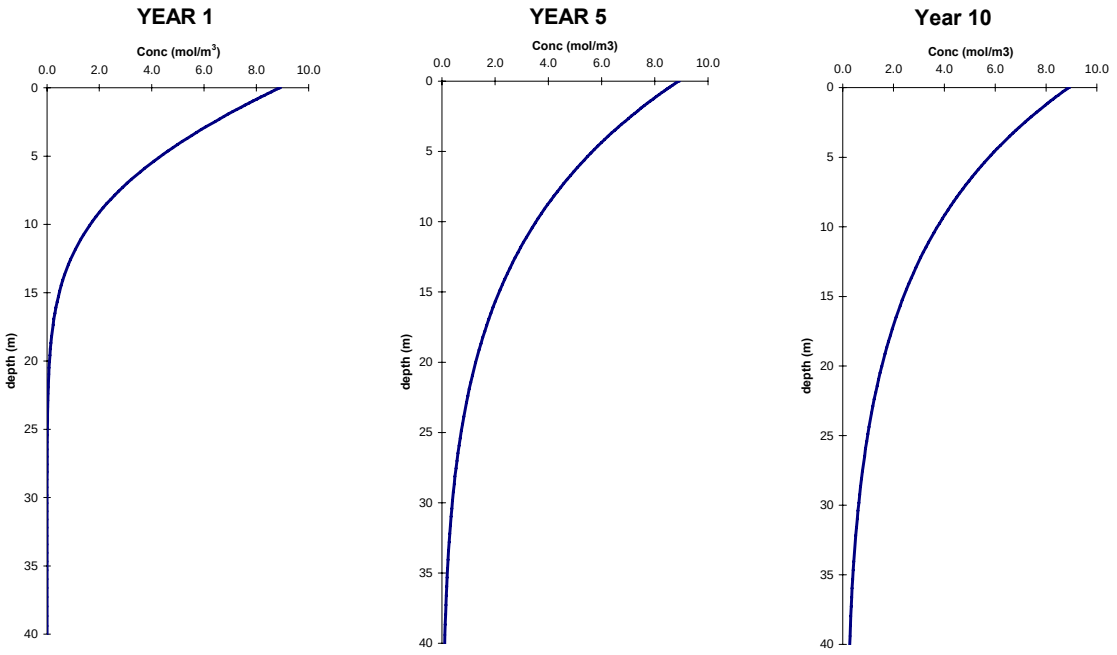


Figure 7-1: Oxygen Concentration Profiles in Coarse Tailings

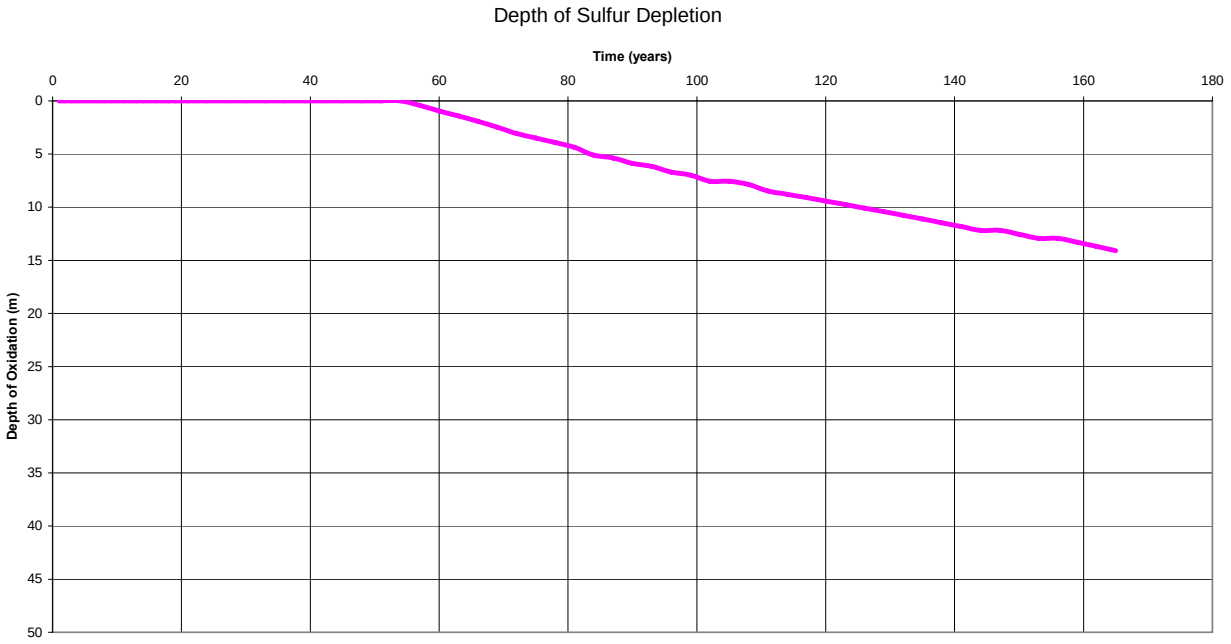


Figure 7-2: Sulfide Depletion in Coarse Tailings

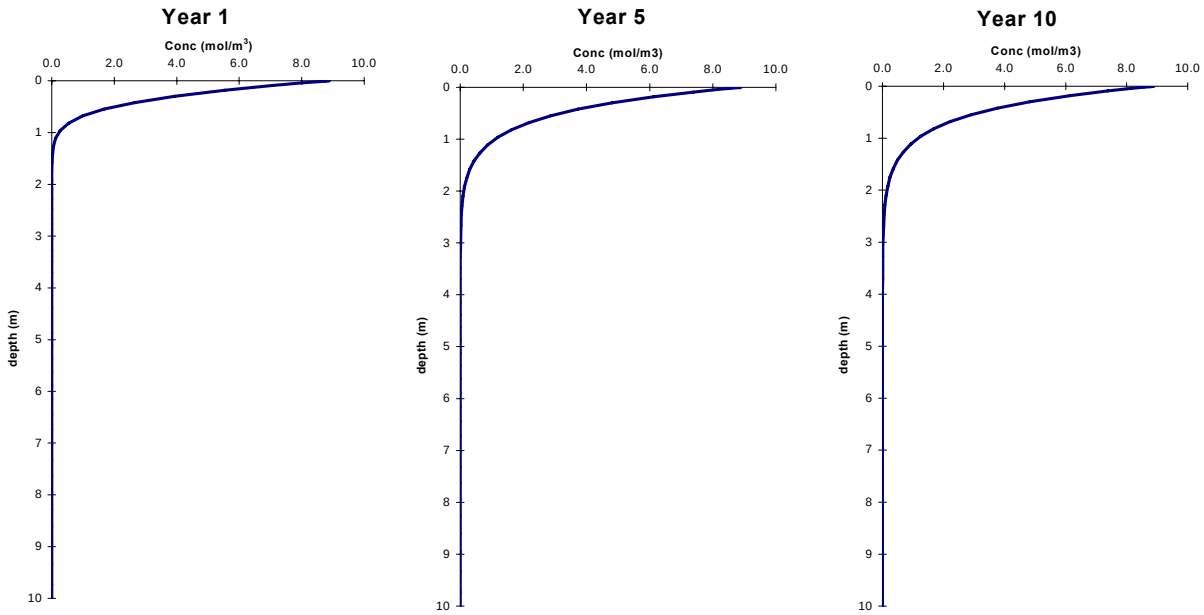


Figure 7-3: Oxygen Concentration Profiles in Fine Tailings

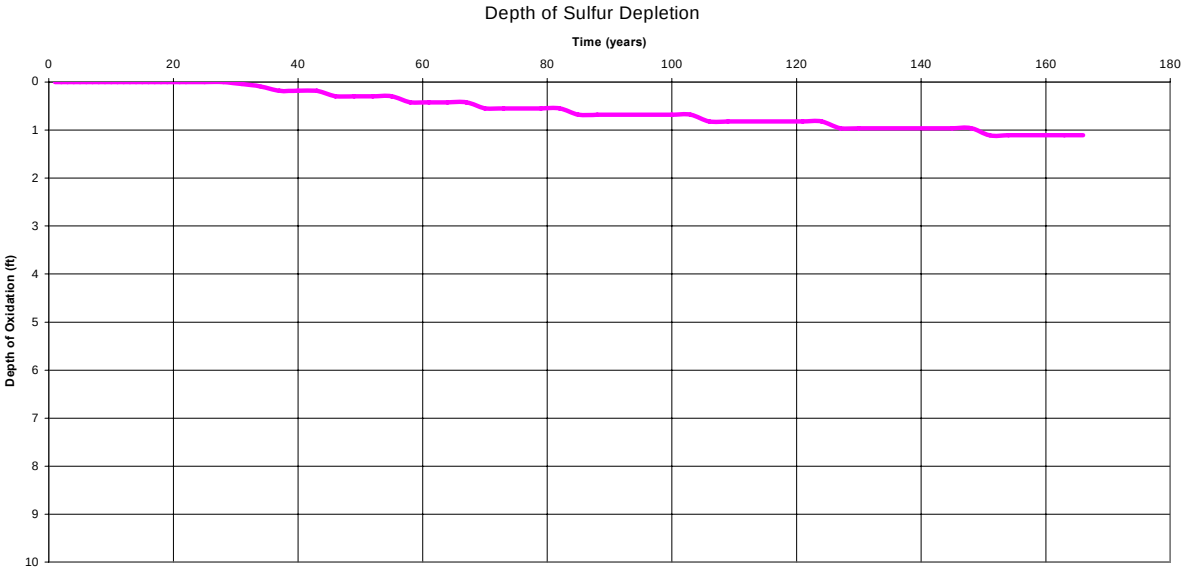


Figure 7-4: Sulfur Depletion in Fine Tailings

7.2.4 Acid Generation – Neutralization Consumption Balance Calculations

Alkalinity Release

This section provides further discussion of the availability of alkalinity to neutralize acid produced by sulfide oxidation as it relates to development of the tailings weathering profile.

The alkalinity release rates in excess of that consumed by the acid generated by oxidation were observed in the humidity cells. The results are summarized in Table 7.4. The table shows the average alkalinity concentration in the leachate from the final 20 cycles of testing. The leach rate represents the rate of loss of alkalinity in excess of that which is required for acid neutralization. For reference, the corresponding acidity equivalent alkalinity consumption rates, and the combined or total (i.e. consumption plus leached excess alkalinity) consumption rates are also shown. The alkalinity release rate from the coarse tailings appears to be lower than that from the fine tailings which is consistent with observed susceptibility of coarse tailings to pH depression.

Table 7-4: Summary of Estimated Steady State Alkalinity Release and Consumption Rates

Sample ID	Average Alkalinity Concentration (mg $CaCO_3$ eq/L)	Leach Rate Indicated by Test Work (mg $CaCO_3$ eq/kg/s)	Acidity Eq. Alkalinity Consumed Indicated by Sulfate Release (mg $CaCO_3$ eq/kg/s)	Total Alkalinity Consumed and Leached (mg $CaCO_3$ eq/kg/s)
P1S	30.8	$2.34x10^{-5}$	$1.14 x10^{-5}$	$3.48 x10^{-5}$
P3S	24.4	$1.90 x10^{-5}$	$1.28 x10^{-5}$	$3.18 x10^{-5}$
Average		$2.12 x10^{-5}$		
Parcel 1-2 PISCS +100 mesh	35.3	$2.72 x10^{-5}$	$9.47 x10^{-6}$	$3.67 x10^{-5}$
Parcel 3 P3S +100 mesh	36.2	$2.77 x10^{-5}$	$1.18 x10^{-5}$	$3.95 x10^{-5}$
Average		$2.74 x10^{-5}$		
Parcel 1-2 PISCS -100 +200 mesh	34.3	$2.67 x10^{-5}$	$1.35 x10^{-5}$	$4.02 x10^{-5}$
Parcel 3 P3S -100 +200 mesh	27.9	$1.98 x10^{-5}$	$2.15 x10^{-5}$	$4.13 x10^{-5}$
Average		$2.32 x10^{-5}$		
Parcel 1-2 PISCS -200 mesh	61.2	$4.74 x10^{-5}$	$9.85 x10^{-6}$	$5.72 x10^{-5}$
Parcel 3 P3S -200 mesh	59.2	$4.33 x10^{-5}$	$2.12 x10^{-5}$	$6.45 x10^{-5}$
Average		$4.54 x10^{-5}$		

Because the mineralogical assessment identified only traces of carbonate minerals, the alkalinity appears to be generated from weathering of silicate minerals (i.e. plagioclase) due to interaction with atmospheric carbon dioxide. Because the mineralogical assessment also indicated that quartz is not present or very rarely, it is concluded that the silica release is indicative of the weathering of the plagioclase. The presence of minor amounts of clay further supports this conclusion.

Because the humidity cell tests were conducted on well-aerated thin layers of tailings, the leach tests therefore are likely to indicate the maximum rate of alkalinity release that can be expected from the tailings. The calculation of the maximum alkalinity release that may occur under field conditions

will however require an understanding of the depth of carbon dioxide penetration into the tailings by gaseous diffusion. It is, however, recognized that rain water will also be saturated with carbon dioxide which will further contribute to alkalinity release but that mechanism is not considered herein.

Calculation of the depth of diffusion requires an understanding of the rate at which carbon dioxide is consumed by the weathering reactions. The theoretical ratio will depend on the end-product (i.e. clay mineral) that is formed and it is probable that a range of clay minerals are present such that the actual release ratio is the end-product of a combination of ratios. Therefore, bicarbonate to silicate ratios were calculated from the later cycles of the humidity cell leachate concentrations as summarised in Table 7-5. The table shows first the molar ratio of the total release of alkalinity (including that consumed by oxidation reactions) to the silica present in the leachate. As shown in the table, the ratio in some cases (e.g. the coarse samples) is substantially higher than would ordinarily be expected from the weathering of silicate minerals. The very high ratios suggest that calcite or dolomite may in fact participate in the reactions. However, it is anticipated that these carbonates had been formed as a result of the initial high reactivity of the silicates, and therefore remain indicative of the overall potential for generating alkalinity. Nonetheless, the ratios were recalculated with the calcium equivalent bicarbonate concentration subtracted from the total prior to calculating the ratio. These results are shown in the second column.

As shown in the table, the 'whole samples' indicate a net release ratio of about 1.9 on average. The coarse and mid size tailings samples indicate higher ratios of about 7.0 and 5.5 respectively. Two of the fine tailings samples indicate a ratio similar to that of the whole samples (i.e. about 2.0). The third sample however returned a much higher ratio. The reason for this is not certain. Overall it appears that a ratio of about 2 is not unreasonable for weathering reactions unaffected by carbonates.

Table 7-5: Summary of Bicarbonate to Silicate Molar Concentration Ratios in Tailings Leachate

Test	Description	(Total HCO_3) : Si	(Total – Calcium eq HCO_3) : Si
14	P1S	7.26	2.60
15	P1 Solids	4.76	1.62
16	P2S	5.26	1.51
17	P3S	5.77	1.69
18	Parcel 2 P2S +100 mesh	19.28	7.53
21	Parcel 1-2 PISCS +100 mesh	18.04	7.31
24	Parcel 3 P3S +100 mesh	17.49	6.29
19	Parcel 2 P2S +200 mesh	13.20	5.01
22	Parcel 1-2 PISCS -100 +200 mesh	14.98	4.80
25	Parcel 3 P3S -100 +200 mesh	18.83	6.76
20	Parcel 2 P2S -200 mesh	9.01	1.89
23	Parcel 1-2 PISCS -200 mesh	30.49	16.00
26	Parcel 3 P3S -200 mesh	10.60	2.18

Using a bicarbonate to silicate release ratio of 2.0, the carbon dioxide consumption rates were estimated for the coarse and fine tailings, in much the same way as the oxygen consumption rates were calculated. The results are shown in Table 7-6.

These rates were then used in diffusion calculations as described for the oxygen transport modeling, to estimate the depth of carbon dioxide diffusion into the tailings, as shown in Figure 7-5. The results indicate that the depth to which carbon dioxide is expected to diffuse into the tailings is about 0.25 m (or 0.8 ft) in the fine tailings and about 0.8 m (2.6 ft) in the coarse tailings. Using these depths, and assuming a tailings bulk density of 1.2 and the rates of alkalinity release shown in Table 7-6, it can be shown that the approximate alkalinity release rate in the tailings deposit will be about 0.576 kg $CaCO_3$ eq/m^2/year from the fine tailings, and about 1.225 kg $CaCO_3$ eq/m^2/year from the coarse tailings.

Table 7-6: Summary of Carbon Dioxide Consumption Rates Estimated from Silicate Molar Release Ratios in Tailings Leachate

Test	Procedure	Size	Sample	Si Release in Tests (mol/kg/s)	CO_2 Consumption mol/kg/s
L21	DNR	+100	Parcel 1-2 PISCS +100 mesh	$3.56x10^{-11}$	$7.11x10^{-11}$
L24	DNR	+100	Parcel 3 P3S +100 mesh	$3.87x10^{-11}$	$7.73x10^{-11}$
T8	ASTM	+100	Parcel 1-2 PISCS +100 mesh	$3.28x10^{-11}$	$7.40x10^{-11}$
T11	ASTM	+100	Parcel 3 P3S +100 mesh	$3.70x10^{-11}$	$6.57x10^{-11}$
			Average	$3.60x10^{-11}$	$7.20x10^{-11}$
L23	DNR	-200	Parcel 1-2 PISCS -200 mesh	$2.36x10^{-11}$	$4.72x10^{-11}$
L26	DNR	-200	Parcel 3 P3S -200 mesh	$9.99x10^{-11}$	$2.00x10^{-10}$
T10	ASTM	-200	Parcel 1-2 PISCS -200 mesh	$6.14x10^{-11}$	$1.23x10^{-10}$
T13	ASTM	-200	Parcel 3 P3S -200 mesh	$5.92x10^{-11}$	$1.18x10^{-10}$
			Average	$6.10x10^{-11}$	$1.22x10^{-10}$

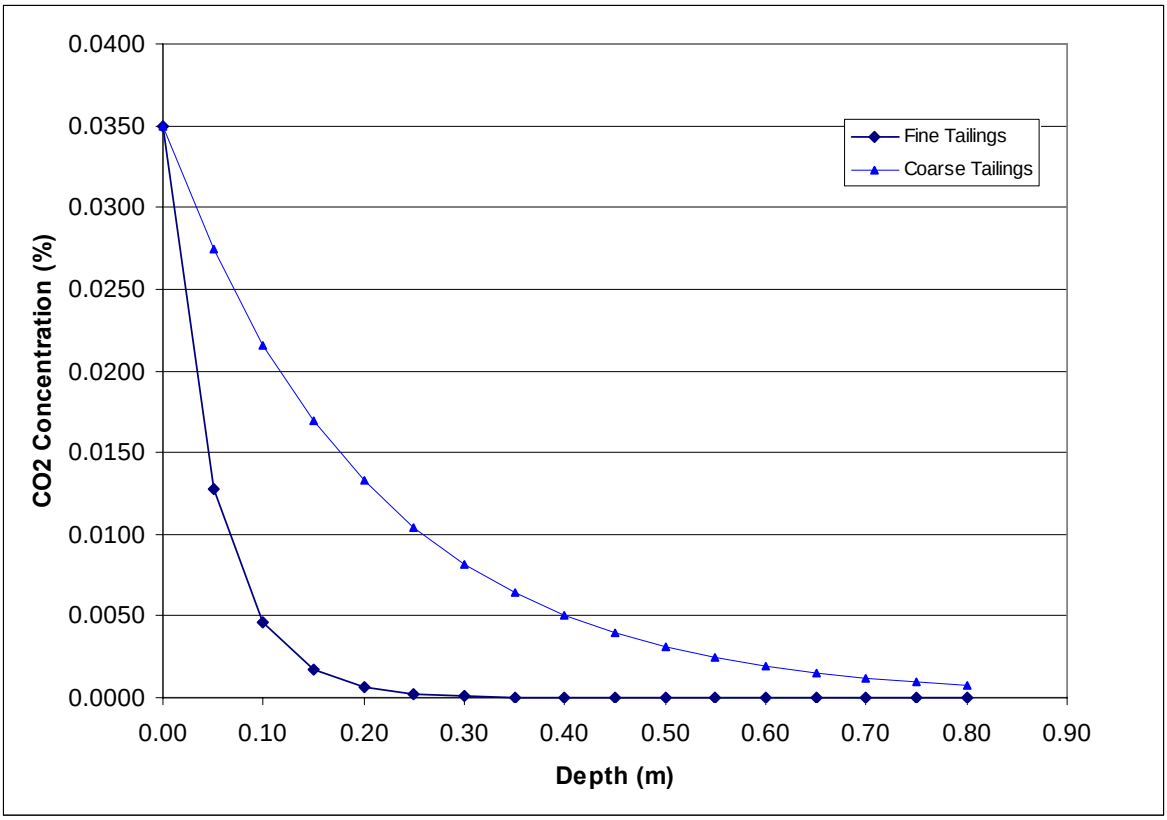


Figure 7-5: Calculated Carbon Dioxide Concentration Profiles in Fine and Coarse Tailings

The petrographic assessment indicated that the tailings contain about 65 % plagioclase on average. Assuming kaolinite is formed when plagioclase reacts with carbon dioxide the reaction proceeds as follows:

0.38 Plagioclase + 0.52 CO_2 + 0.78 H_2O => 0.26 Kaolinite + 0.235 Na^+ + 0.144 Ca^{2+} + 0.47 SiO_2 + 0.52 HCO_3^- …(5)

For this reaction the HCO_3^- : SiO_2 is about = 1.11, which is a much lower bicarbonate yield than measured in the laboratory and would therefore result in a conservative estimate of plagioclase depletion. At the above rates of alkalinity release and using the stoichiometry in equation 5, plagioclase will be depleted at a rate of about 17.9 mol/m^2/year from the zone to which carbon dioxide would diffuse (i.e. about 0.8 m or 2.6 ft) in the coarse tailings, and at a rate of about 8.4 mol/m^2/year from the corresponding zone (0.25 m or 0.8 ft) from the fine tailings. At a bulk tailings density of 1.2, and an average plagioclase content of 65%, it can be shown that the 0.8 m zone in the coarse tailings can sustain an alkalinity release at the estimated maximum rate for about 129 years. The carbon dioxide influenced zone in the fine tailings can sustain the maximum alkalinity release rate for about 89 years. It should however be noted that as the consumption rate of the carbon dioxide in the near surface decreases (as plagioclase is fully converted) the depth to which carbon dioxide will diffuse will increase and additional plagioclase will be available for reaction and alkalinity generation, albeit at a lower rate.

These generation rates are compared to the acid generation rates in the next section.

7.2.5 Overall Acidity - Alkalinity Release Balance

The estimated rates of acidity generated, expressed in $CaCO_3$ equivalents, calculated for the oxidation rate of the coarse tailings over time are summarized in Table 7-7, and the corresponding rates for the fine tailings are shown in Table 7-8.

Table 7-7: Estimated Acidity and Alkalinity Generation in Coarse Tailings

Time Years	Oxygen Flux into Tailings mol/m^{2}year	Corresponding Rate of Acidity Generation kgCaCO$_3$eq/m^2/year	Rate of Alkalinity Generation kgCaCO$_3$eq/m^2/year
1	12.35	0.659	1.225
10	6.84	0.365	1.225
15	6.07	0.324	1.225
30	3.97	0.211	1.225
60	2.67	0.143	1.225
90	2.11	0.113	1.225
125	1.64	0.087	1.225

Table 7-8: Estimated Acidity and Alkalinity Generation in Fine Tailings

Time Years	Oxygen Flux into Tailings mol/m^{2}year	Corresponding Rate of Acidity Generation kgCaCO3eq/m^2/year	Rate of Alkalinity Generation kgCaCO3eq/m^2/year
1	3.37	0.180	0.576
10	1.83	0.097	0.576
15	1.43	0.076	0.576
30	0.97	0.052	0.576
60	0.67	0.036	0.576
90	0.54	0.029	0.576
120	0.47	0.025	< 0.576*

Note: * While the plagioclase will in theory have been depleted from the near surface in the fine tailings, carbonate will progress deeper into the tailings and may reduce the rate of alkalinity release.

As discussed in the previous section, the maximum rate of alkalinity from the coarse tailings is expected to be sustained for about 129 years which is about the time it is estimated that all of the sulfur will be depleted from the coarse tailings to a depth of about 80 feet. It should however be noted that carbon dioxide will be released within the oxidation zone when bicarbonate alkalinity reacts with acidity as follows:

$$HCO_3^- + H^+ \Rightarrow H_2CO_3 \Rightarrow H_2O + CO_2 \qquad ...(6)$$

The carbon dioxide generated then becomes available to react again with plagioclase within the oxidation zone to release additional alkalinity. Therefore, excess alkalinity will be generated to neutralize the acidity and maintain neutral pH conditions even well after the estimated period of release from the near surface layer.

In the case of the fine tailings, the initial alkalinity release rates can be maintained from the near surface tailings (i.e. 0.8 ft) for about 89 years. At that time the rate of oxidation will have decreased to about 16 % of the initial rate, i.e. the alkalinity requirement to maintain neutral pH conditions will be considerably lower. The net rate of alkalinity is nonetheless anticipated to exceed the acid generation rate for two reasons. First, as the plagioclase is depleted from near surface, carbon dioxide diffusion will extend deeper into the tailings thus additional plagioclase will become available for reaction. Alkalinity release will continue albeit at a reduced rate. Second, as noted above, bicarbonate neutralization will release carbon dioxide within the oxidation zone and will result in the 'regeneration' of alkalinity from plagioclase present within the oxidation zone.

It is therefore concluded that the tailings will remain pH-neutral indefinitely.

7.2.6 Infiltration and Seepage Rates

Background

Tailings deposition will commence in Cell 2E of the LTVSMC tailings storage facility. The tailings will be spigoted from the perimeter of the cell so that a beach of coarser tailings will form immediately inside the perimeter and further away from the spigot location a beach of finer tailings will accumulate before the excess water flows into the supernatant pond. The embankment itself will be constructed from these coarse tailings. The coarse tailings beach is expected to be about 400 feet wide and the zone of finer tailings is expected to be about 300 feet wide.

Tailings deposition will continue in Cell 2E until the tailings reach the elevation of the tailings in Cell 1E. This is anticipated to occur at about Year 8. Thereafter, i.e. from Year 9 onwards, Cell 2E and Cell 1E will be operated as a single disposal facility. This will mean that tailings will be deposited along the outer embankments of both cells to raise the embankments simultaneously. The embankments will be raised in lifts of about 15 ft. However, only the exterior embankments along the north edge of Cell 2E and south and southeastern edge of Cell 1E will be constructed of coarse tailings (note – the reference to Cell 2E and 1E is only to indicate locations within the single large pond). As each embankment is completed, and construction of the next lift commences. The exterior embankments constructed of coarse tailings will be capped with a synthetic membrane to reduce oxidation and limit infiltration to the embankment.

To manage the phreatic surface within the coarse embankment a series of horizontal drains (finger drains) will be installed along the entire outer (north) embankment of Cell 2E. The drains will be spaced about 100 ft apart and will be located at the base of the LTVSMC tailings. For Cell 1E, horizontal drains will be required only along the perimeter where the tailings overlie bedrock. Figure 7-6 illustrates the configuration of Cell 2E and Cell 1E after they had been merged as a single

pond. The red lines indicate the locations of the horizontal drains. As described in the following sections, water quality predictions were completed for seepage collected at each of the horizontal drain systems.

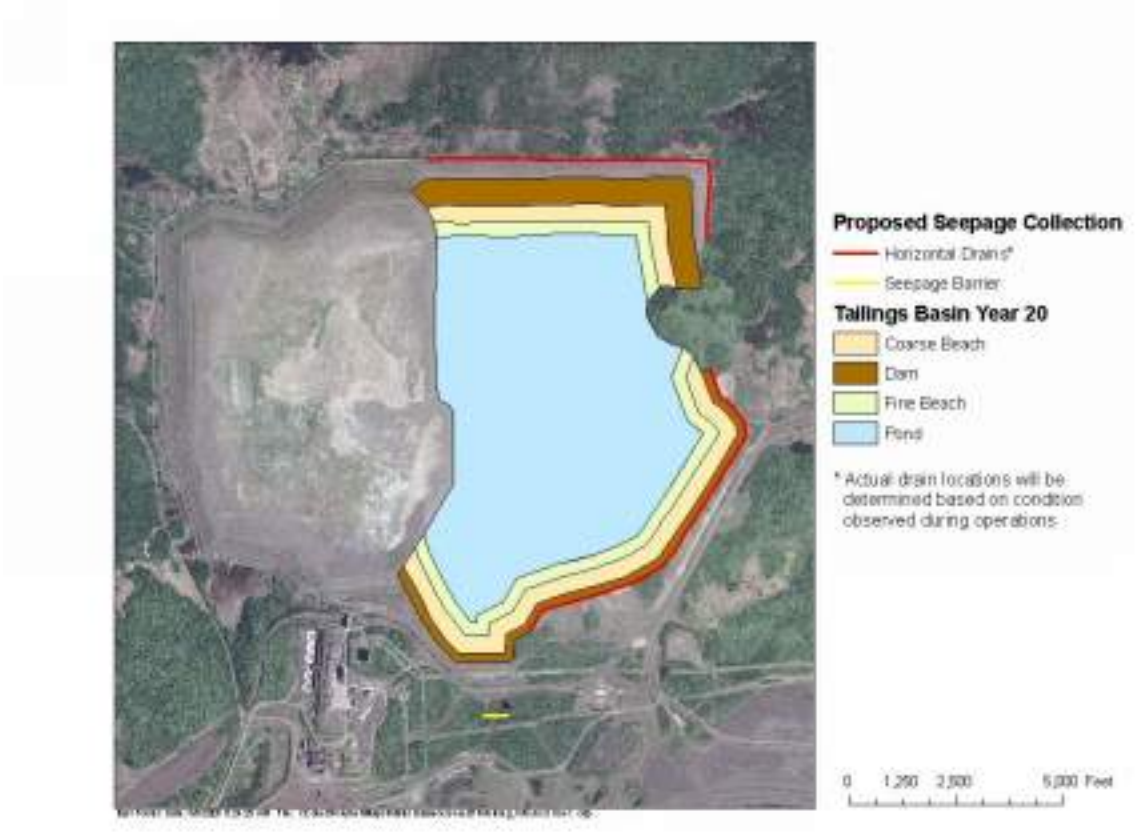


Figure 7-6: Plan View of Cell 2E and Cell 1E after Merging to a Single Pond. Source: Barr Engineering

As indicated previously, the coarse tailings are likely to be most susceptible to oxidation and the above operating strategy has been developed specifically to limit coarse tailings beach size and to restrict active oxidation during the operational period. After operations cease, the coarse tailings beach adjacent to the exterior embankments will be capped with a synthetic membrane to limit both infiltration and restrict oxidation of the coarser tailings.

The total rate of oxidation will therefore depend on the total beach area that is exposed. The estimated beach areas for each of coarse and fine beaches are shown in Tables 7-9 and 7-10, for Cells 2E and Cell 1E, respectively. These were used to estimate ingress of oxygen to the tailings.

Table 7-9: Estimated Beach Area Exposure During Tailings Deposition to Cell 2E

		Embankment	Beach			
Elevation	Height	Coarse to Fine	Coarse to Fine	Fine to Slime	Pond	Total Area
feet	feet	square feet	square feet	square feet	square feet	square feet
1570	0	0	0	0		
1585.0	15.0	430782	2999560	2249670	12649574	18329586
1600.0	30.0	886149	3072000	2304000	13485480	19747629
1615.0	45.0	1349373	3104320	2328240	14857688	21639621
1630.0	60.0	1815117	3105600	2329200	15757140	23007057
1645.0	75.0	2304341	3081300	2310975	16610208	24306824
1660.0	90.0	2867682	3064280	2298210	16886345	25116517
1675.0	105.0	3608470	3068400	2301300	17629980	26608150
1690.0	120.0	3938030	3068400	2301300	18708730	28016460
1705.0	135.0	4438078	3068400	2301300	19504186	29311963
1720.0	150.0	4951640	3068400	2301300	20321140	30642480
1726.1	156.1	5157741	3068400	2301300	20648997	31176437

Table 7-10: Estimated Beach Areas for Cell 1E from Year 9 Onwards

		Embankment	Beach			Total Area
Elevation	Height	Coarse to Fine	Coarse to Fine	Fine to Slime	Pond	
feet	feet	square feet	square feet	square feet	square feet	square feet
1683.5	9	366785	5287600	3965700	24805941	34426025
1690	15	932186	5270320	3952740	24235652	34390897
1704.8	29.8	1660056	5173360	3880020	23327334	34040770
1720	45	2442068	5486800	4115100	21754282	33798249
1726.1	51.1	2522212	5198640	3898980	21863048	33482880

Active Disposal Infiltration

During the operational period, tailings will be discharged from a single movable point spigot. It is anticipated that a delta will be formed at an angle of about 70^{o} that will extend about 700 ft from the discharge point. As noted before, the coarser tailings will tend to accumulate within the first 400 ft, and the finer tailings will be deposited beyond that range.

HYDRUS-2D modeling was undertaken to assess the rate of infiltration that may occur at the anticipated water discharge rate associated with the tailings slurry. The results indicated that for the duration of discharge, sufficient water will be available to result in an infiltration rate equal to the saturated hydraulic conductivity. Therefore, based on the assumed dimensions of the delta and the saturated hydraulic conductivity (K_{sat}) of $2x10^{-4}$ cm/s for the coarse tailings and 2.2×10^{-5} cm/s for the fine tailings, the net infiltration to the tailings beaches were calculated as shown in Table 7-11 for the coarse tailings beach and in Table 7-12 for the fine tailings beach. Note that the estimates are averaged for the entire beach area, and encompass both cells.

Table 7-11: Estimated Infiltration Rates to Coarse Tailings Beaches During Active Tailings Deposition

Elevation ft	Area ft^2	Flow m^3/year	Spigot Inches/year	Precipitation Inches/year	Total Inches/year
1570					
1585	2999560	215,818	30.5	8	38.5
1600	3072000	215,818	29.8	8	37.8
1615	3104320	215,818	29.5	8	37.5
1630	3105600	215,818	29.4	8	37.4
1645	3081300	215,818	29.7	8	37.7
1660	3064280	215,818	29.8	8	37.8
1675	3068400	215,818	29.8	8	37.8
1690	8338720	215,818	11.0	8	19.0
1705	8241760	215,818	11.1	8	19.1
1720	8555200	215,818	10.7	8	18.7
1726.1	8267040	215,818	11.1	8	19.1

Table 7-12: Estimated Infiltration Rates to Fine Tailings Beaches During Active Tailings Deposition

Elevation ft	Area ft^2	Flow m^3/year	Spigot Inches/year	Precipitation Inches/year	Total Inches/year
1570					
1585	2249670	49,854	9.4	8	17.4
1600	2304000	49,854	9.2	8	17.2
1615	2328240	49,854	9.1	8	17.1
1630	2329200	49,854	9.1	8	17.1
1645	2310975	49,854	9.1	8	17.1
1660	2298210	49,854	9.2	8	17.2
1675	2301300	49,854	9.2	8	17.2
1690	6267000	49,854	3.4	8	11.4
1705	6254040	49,854	3.4	8	11.4
1720	6181320	49,854	3.4	8	11.4
1726.1	6416400	49,854	3.3	8	11.3

These infiltration rates were used in the unsaturated flow modeling together with particle tracking to determine the net transport rate of seepage from the tailings storage facility (RS13, Barr 2007c). The output from that modeling provided an estimate of the coarse tailings pore water, fine tailings pore water and pond water that would be expected to report to the horizontal drains and seepage from the tailings storage facility. The application of the results from the unsaturated flow modeling is discussed in the next section.

7.2.7 Seepage Transport Rates

The transport rate modeling was undertaken for infiltration occurring in Year 1, Year 8, Year 9, Year 14 and Year 20. Typically, output from the transport modeling indicated a percentage of flow from each of the sources (coarse tailings, fine tailings, pond) that reported to the horizontal drain and the seepage recovery barrier in the case of Cell 1E. The flow modeling also provided overall water balances for the tailings basin.

The results from the flow modeling are reported in RS13 (Barr, 2007c). However, the following briefly describes the use of the flow modeling results to estimate the seepage water quality.

Figure 7-7 illustrates the results from the flow modeling for Year 1 infiltration to the coarse tailings that report to the horizontal drain system of Cell 2E over time. The solid line represents a curve that was fitted to the data (diamond shapes). Similar plots for the fine tailings and the embankment are shown in Figure 7-8 and Figure 7-9.

The formulation for the curve that was fitted to the data was then used to estimate the annual flow intercepted by the horizontal drains. The results are shown in Figure 7-10. The figure shows the results for each source for Year 1 and those for Year 9.

The Year 1 flows were assumed to repeat for Year 2 to Year 8 because the infiltration rates remain constant over this period. For example, infiltration in Year 1 from the coarse tailings first appears in the horizontal drains in Year 4. Therefore, the infiltration from Year 2 would first appear in the horizontal drains in Year 5, that in Year 3 in Year 6 and so on. The combined flow from the coarse tailings would be the sum of these flows up and including Year 8. In years subsequent to Year 8, the infiltration rates change and the flow profiles for Year 9 were adopted for infiltration occurring from Year 9 to Year 20, when operations cease and infiltration rates decrease further. The sums of these combined flows over time are illustrated in Figure 7-11. These flow rates together with the source concentrations derived in Section 7.2.8 were used to estimate the seepage water quality from each of the cells.

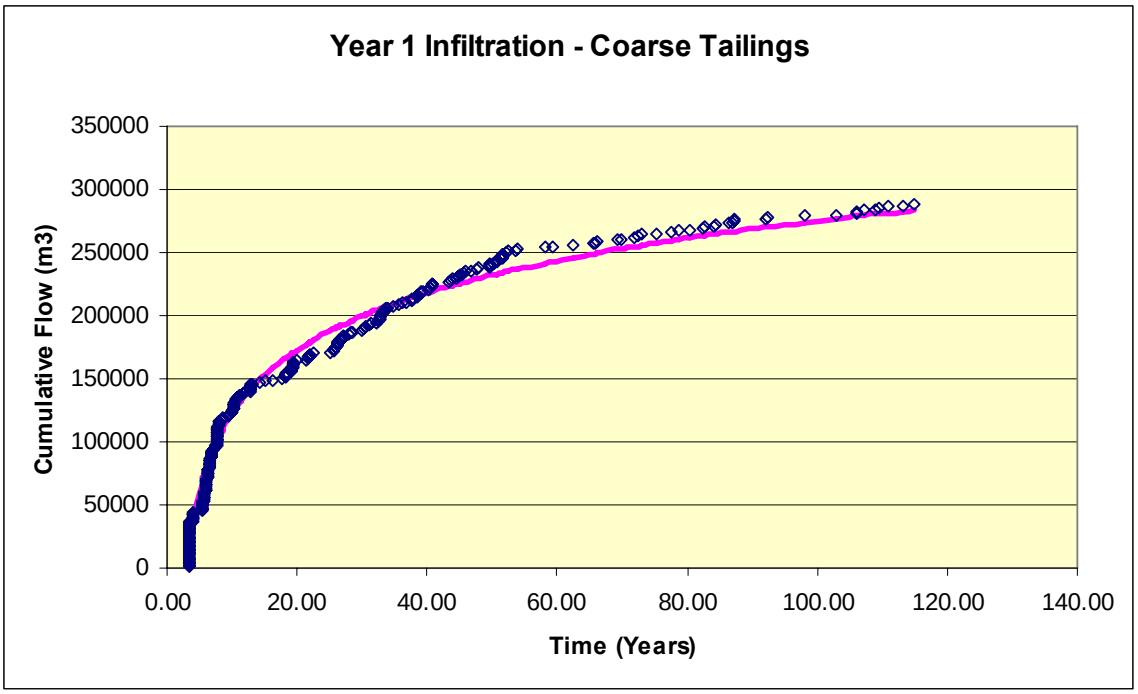


Figure 7-7: Cumulative Seepage from Year 1 Infiltration to Cell 2E Coarse Tailings Beach

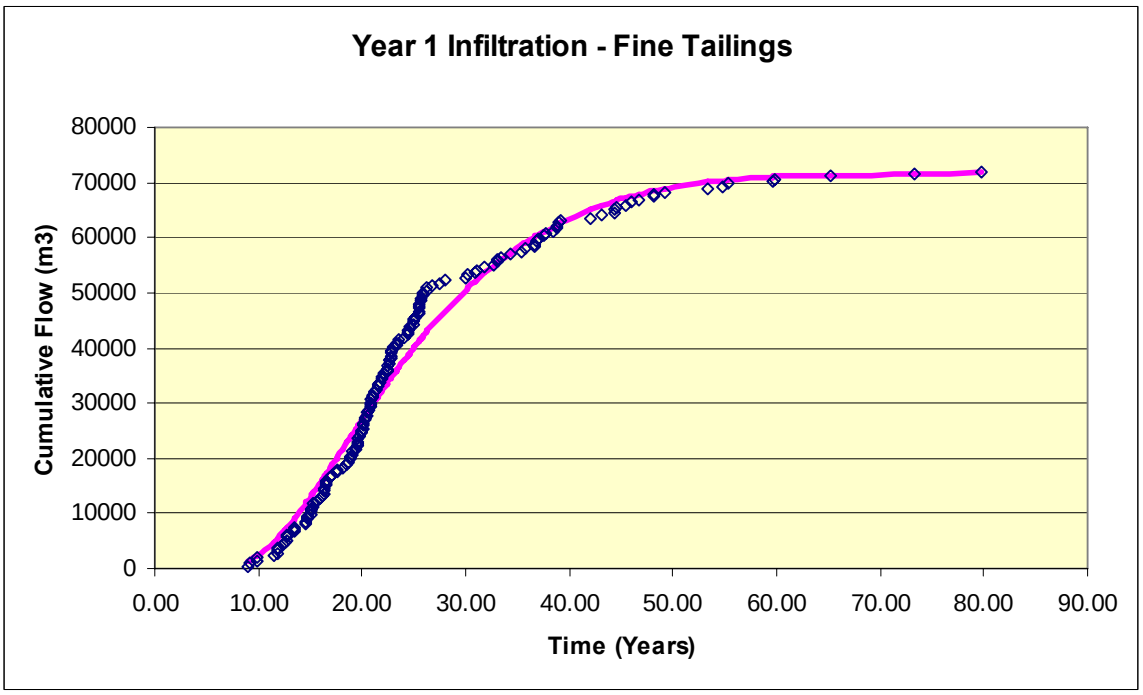


Figure 7-8: Cumulative Seepage from Year 1 Infiltration to Cell 2E Fine Tailings Beach

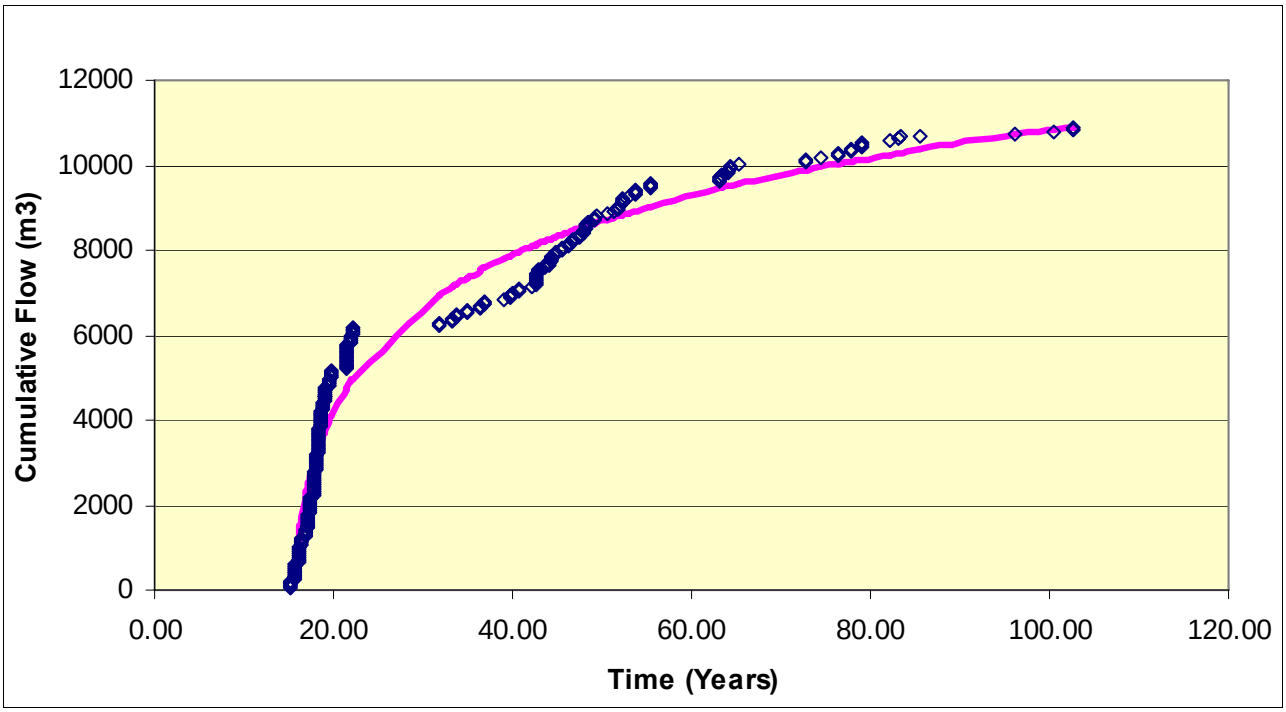


Figure 7-9: Cumulative Seepage from Year 1 Infiltration to Cell 2E Embankment

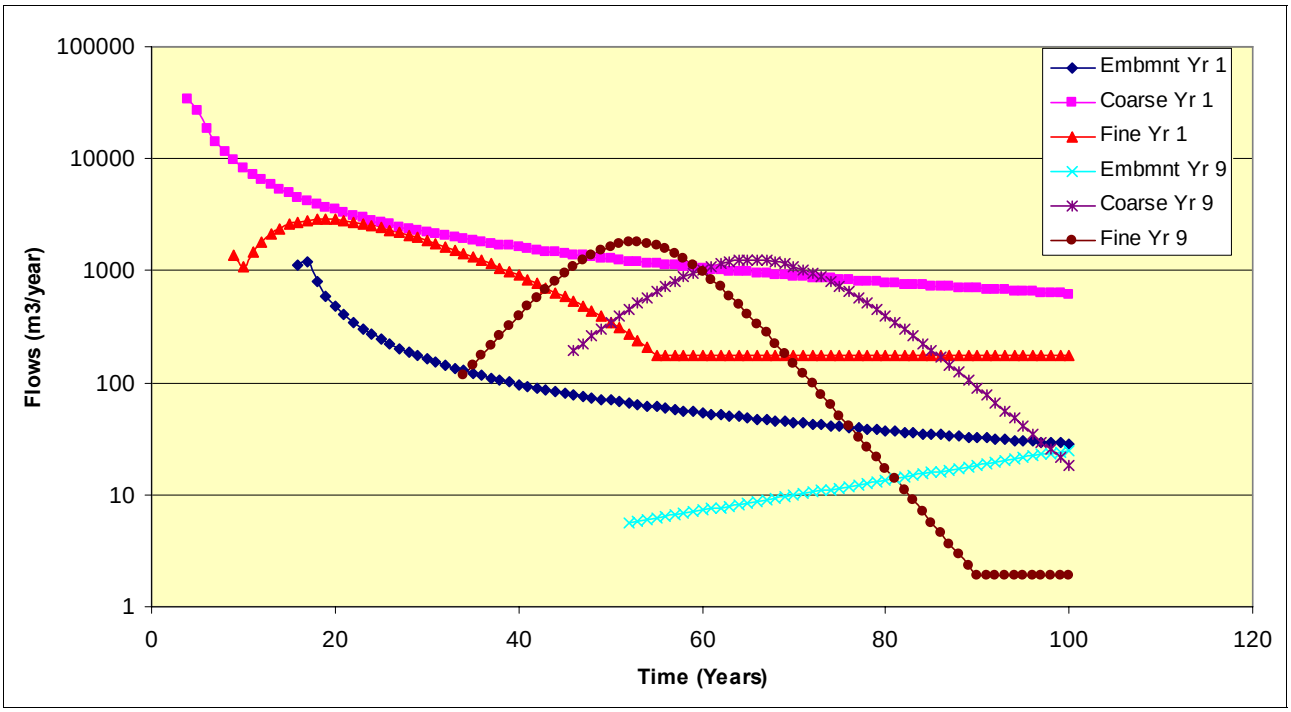


Figure 7-10: Flows from Infiltration in Year 1 and in Year 9

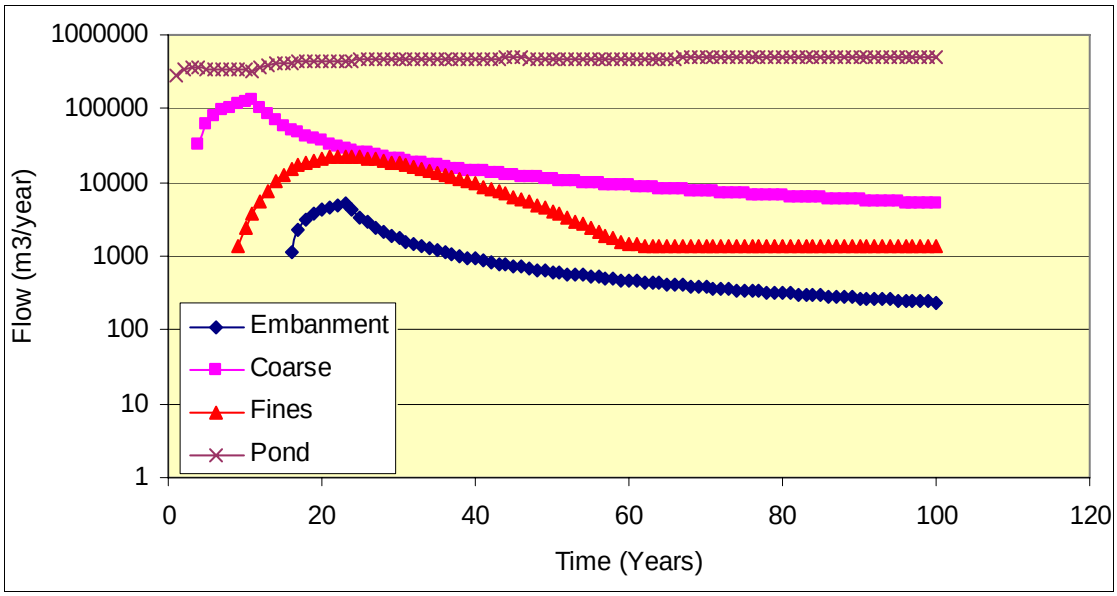


Figure 7-11: Combined Drainage Flows from Infiltration Commencing Year 9 to Year 20

7.2.8 Solute Concentrations in Seepage Water

Oxidation Release Rates and Source Concentrations

To estimate potential metal concentrations in seepage from the tailings, the molar ratios of key elements relative to sulfate were calculated. The results are shown in Table 7.13. Except for antimony, the average of the large scale ASTM humidity cell tests was used to estimate metal release rates. In the case of antimony, the results from the smaller scale MDNR cells were used. As described in Section 4.3.3, the materials used to construct the humidity cells were found to leach antimony but the same was not the case for the MDNR reactor tests.

Table 7-13: Molar Metal to Sulfate Release Ratios Calculated from Humidity Cell Test Results

Ratio	Coarse Tailings	Fine Tailings
Sb/SO_4	$4.0x10^{-5}$	$4.0 x10^{-5}$
As/SO_4	$2.0 x10^{-4}$	$4.2 x10^{-4}$
Cu/SO_4	$1.5 x10^{-4}$	$1.4 x10^{-4}$
Ni/SO_4	$6.6 x10^{-4}$	$6.5 x10^{-5}$
Zn/SO_4	$1.2 x10^{-3}$	$6.4 x10^{-4}$
Co/SO_4	$3.9 x10^{-5}$	$1.3 x10^{-5}$
Ca/SO_4	$1.8 x10^{-1}$	$4.1 x10^{-2}$
Mg/SO_4	$3.8 x10^{-1}$	$4.5 x10^{-1}$
Na/SO_4	$1.1 x10^{-1}$	$2.3 x10^{-1}$
K/SO_4	$3.3 x10^{-1}$	$2.8 x10^{-1}$
Ag/SO_4	$2.99 x10^{-6}$	$2.7 x10^{-6}$
B/SO_4	$5.97 x10^{-4}$	$8.5 x10^{-4}$
Be/SO_4	$1.43 x10^{-4}$	$1.3 x10^{-4}$
Cd/SO_4	$2.30 x10^{-6}$	$2.1 x10^{-6}$
Pb/SO_4	$1.56 x10^{-6}$	$1.4 x10^{-6}$
Se/SO_4	$1.63 x10^{-5}$	$1.5 x10^{-5}$
Tl/SO_4	$6.31 x10^{-7}$	$5.8 x10^{-7}$

As discussed earlier, oxidation rates were estimated from oxygen flux rates into the tailings assuming that all of the oxygen is consumed by sulfide mineral oxidation reactions. By converting the oxidation rates into sulfate generation rates, and then multiplying the ratios presented in Table 7-13, the overall oxidation related metal release within the oxidation zone can be calculated at various times into the future. These release rates were then divided by the infiltration volumes to determine the pore water concentrations for each of the embankment, coarse, and fine tailings areas.

The infiltrating water quality will vary over time. For example, during tailings deposition, the water infiltrating the beaches and embankments will predominantly consist of process water. Post operations, the infiltrating water will reflect unaffected meteoric precipitation. Therefore, to calculate the pore water concentrations, the infiltrating water was assumed to be constant at the estimated process water concentrations shown in Table 7-14. Concentrations were assumed to remain constant for the modeling period. Once deposition ceases, the infiltrating water was assumed to have no dissolved solutes.

Table 7-14: Summary of Assumed Process Water Concentrations

Parameter	Units	Concentration
SO4	mg/L	200
Sb	mg/L	0.015
As	mg/L	0.037
Cu	mg/L	0.012
Ni	mg/L	0.027
Zn	mg/L	0.091
Co	mg/L	0.003
Ca	mg/L	106
Mg	mg/L	26
Na	mg/L	52
K	mg/L	11
Ag	mg/L	0.0012
B	mg/L	0.23
Be	mg/L	0.00054
Cd	mg/L	0.00082
Pb	mg/L	0.0040
Se	mg/L	0.0018
Tl	mg/L	0.0022

Note: Concentrations are based on average for process pond modeling because the process adds low metal loads to the process water as it is re-cycled. Sulfate was calculated from the addition of copper sulfate reagent.

Detailed results of the estimated pore water concentrations are provided in Appendix D.3. The results for Cell 2E beaches are summarized in Table 7-15 and the corresponding summary for Cell 1E beaches are presented in Table 7-16. Calculated pore water concentrations for sulfate, cobalt and nickel are elevated but consistent with the conclusions on metal solubility provided in Section 6.3.2. The sulfate concentrations are consistent with the solubility of gypsum in the presence of magnesium and sodium.

Table 7-15: Summary of Cell 2E Source Concentrations

Time	Embankment Porewater				Coarse Beach Porewater				Fine Tailings Porewater			
	SO_4	Cu	Ni	Co	SO_4	Cu	Ni	Co	SO_4	Cu	Ni	Co
Year	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
1	823	0.075	0.280	0.018	332	0.025	0.081	0.006	230	0.014	0.028	0.003
2	667	0.059	0.217	0.014	332	0.025	0.081	0.006	238	0.015	0.029	0.003
3	741	0.067	0.247	0.016	332	0.025	0.081	0.006	241	0.015	0.029	0.003
4	894	0.082	0.309	0.019	332	0.025	0.081	0.006	229	0.014	0.028	0.003
5	1090	0.102	0.388	0.024	334	0.025	0.082	0.006	230	0.014	0.028	0.003
6	1214	0.115	0.438	0.027	340	0.026	0.084	0.006	238	0.015	0.029	0.003
7	1273	0.121	0.462	0.029	353	0.027	0.089	0.006	241	0.015	0.029	0.003
8	1360	0.130	0.498	0.031	385	0.030	0.102	0.007	229	0.014	0.028	0.003
9	1549	0.149	0.574	0.035	725	0.065	0.240	0.015	245	0.016	0.029	0.003
10	1755	0.170	0.658	0.040	1,040	0.097	0.368	0.023	256	0.017	0.029	0.003
11	1780	0.172	0.668	0.041	1,755	0.170	0.658	0.040	262	0.017	0.030	0.003
12	1791	0.173	0.672	0.041	2,401	0.236	0.919	0.056	243	0.016	0.029	0.003
13	1841	0.179	0.693	0.042	2,554	0.251	0.982	0.060	245	0.016	0.029	0.003
14	1786	0.173	0.670	0.041	2,610	0.257	1.004	0.061	256	0.017	0.029	0.003
15	1809	0.175	0.679	0.041	2,609	0.257	1.004	0.061	262	0.017	0.030	0.003
16	1854	0.180	0.698	0.043	2,557	0.251	0.983	0.060	243	0.016	0.029	0.003
17	1862	0.181	0.701	0.043	2,440	0.240	0.936	0.057	245	0.016	0.029	0.003
18	1903	0.185	0.718	0.044	2,261	0.221	0.863	0.052	256	0.017	0.029	0.003
19	1922	0.187	0.726	0.044	1,982	0.193	0.750	0.046	262	0.017	0.030	0.003
20	1943	0.189	0.734	0.045	1,556	0.150	0.577	0.035	243	0.016	0.029	0.003
21	1965	0.191	0.743	0.045	1,559	0.150	0.578	0.035	245	0.016	0.029	0.003
22	1967	0.191	0.743	0.045	3,434	0.341	1.339	0.081	280	0.019	0.030	0.003
24	1967	0.191	0.744	0.045	3,414	0.339	1.331	0.080	288	0.020	0.031	0.003
27	1968	0.192	0.744	0.045	3,446	0.342	1.343	0.081	262	0.017	0.030	0.003
30	1968	0.192	0.744	0.045	3,483	0.346	1.359	0.082	200	0.012	0.027	0.003
33	1968	0.192	0.744	0.045	3,532	0.351	1.378	0.083	200	0.012	0.027	0.003
36	1968	0.192	0.744	0.045	3,596	0.357	1.404	0.085	200	0.012	0.027	0.003
39	1969	0.192	0.744	0.045	3,571	0.355	1.394	0.084	200	0.012	0.027	0.003
42	1969	0.192	0.744	0.045	3,843	0.382	1.504	0.091	200	0.012	0.027	0.003
45	1969	0.192	0.744	0.045	4,031	0.401	1.580	0.095	200	0.012	0.027	0.003
48	1969	0.192	0.744	0.045	4,371	0.436	1.718	0.103	313	0.022	0.032	0.003
51	1969	0.192	0.744	0.045	4,864	0.486	1.918	0.115	202	0.012	0.027	0.003
54	1969	0.192	0.744	0.045	5,582	0.559	2.210	0.133	201	0.012	0.027	0.003
57	1969	0.192	0.744	0.045	5,582	0.559	2.210	0.133	202	0.012	0.027	0.003
60	1969	0.192	0.744	0.045	5,583	0.559	2.210	0.133	202	0.012	0.027	0.003
63	1969	0.192	0.744	0.045	5,583	0.559	2.210	0.133	202	0.012	0.027	0.003
66	1769	0.180	0.717	0.043	5,383	0.548	2.183	0.130	2	0.000	0.000	0.000
69	1756	0.179	0.712	0.042	5,383	0.548	2.183	0.130	113	0.011	0.004	0.001
72	1719	0.175	0.697	0.042	5,383	0.548	2.183	0.130	2	0.000	0.000	0.000
75	1654	0.168	0.671	0.040	5,383	0.548	2.183	0.130	1	0.000	0.000	0.000
78	1552	0.158	0.629	0.038	5,384	0.548	2.183	0.130	2	0.000	0.000	0.000
81	1429	0.145	0.580	0.035	5,385	0.548	2.183	0.130	2	0.000	0.000	0.000
84	1282	0.130	0.520	0.031	5,386	0.548	2.184	0.130	2	0.000	0.000	0.000
87	1108	0.113	0.449	0.027	5,390	0.548	2.186	0.130	2	0.000	0.000	0.000
90	930	0.095	0.377	0.023	5,396	0.549	2.188	0.131	113	0.011	0.004	0.001
93	767	0.078	0.311	0.019	5,406	0.550	2.192	0.131	2	0.000	0.000	0.000
96	628	0.064	0.255	0.015	5,423	0.552	2.199	0.131	1	0.000	0.000	0.000
99	515	0.052	0.209	0.012	5,287	0.538	2.144	0.128	2	0.000	0.000	0.000

Table 7-16: Summary of Cell 1E Source Concentrations

Time	Embankment Porewater				Coarse Beach Porewater				Fine Tailings Porewater			
	SO_4	Cu	Ni	Co	SO_4	Cu	Ni	Co	SO_4	Cu	Ni	Co
(year)	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
9	823	0.075	0.280	0.018	462	0.0383	0.1336	0.0089	279	0.0190	0.0303	0.0032
10	502	0.042	0.150	0.010	515	0.0437	0.1548	0.0102	202	0.0118	0.0272	0.0026
11	906	0.083	0.313	0.020	749	0.0674	0.2496	0.0158	201	0.0117	0.0272	0.0026
12	1136	0.107	0.407	0.025	1407	0.1345	0.5167	0.0318	202	0.0118	0.0272	0.0026
13	1298	0.123	0.472	0.029	2020	0.1969	0.7654	0.0466	202	0.0118	0.0272	0.0026
14	1488	0.143	0.549	0.034	2164	0.2114	0.8234	0.0501	202	0.0118	0.0272	0.0026
15	1421	0.136	0.522	0.032	2232	0.2184	0.8513	0.0517	202	0.0118	0.0272	0.0026
16	1481	0.142	0.546	0.034	2271	0.2224	0.8669	0.0527	279	0.0190	0.0303	0.0032
17	1618	0.156	0.602	0.037	2295	0.2248	0.8765	0.0532	202	0.0118	0.0272	0.0026
18	1744	0.169	0.653	0.040	2172	0.2123	0.8269	0.0503	201	0.0117	0.0272	0.0026
19	1903	0.185	0.718	0.044	1989	0.1937	0.7526	0.0458	202	0.0118	0.0272	0.0026
20	1924	0.187	0.726	0.044	1707	0.1649	0.6381	0.0390	202	0.0118	0.0272	0.0026
21	1943	0.189	0.734	0.045	1279	0.1214	0.4647	0.0287	202	0.0118	0.0272	0.0026
22	1963	0.191	0.742	0.045	2767	0.2728	1.0679	0.0647	202	0.0118	0.0273	0.0026
24	1965	0.191	0.743	0.045	2769	0.2731	1.0690	0.0647	313	0.0221	0.0317	0.0034
27	1966	0.191	0.743	0.045	2771	0.2733	1.0697	0.0648	202	0.0118	0.0273	0.0026
30	1967	0.191	0.744	0.045	2772	0.2734	1.0703	0.0648	201	0.0117	0.0272	0.0026
33	1968	0.191	0.744	0.045	2773	0.2735	1.0707	0.0648	202	0.0118	0.0273	0.0026
36	1768	0.180	0.717	0.043	2574	0.2619	1.0438	0.0623	202	0.0002	0.0001	0.0000
39	1768	0.180	0.717	0.043	2575	0.2620	1.0441	0.0623	202	0.0002	0.0001	0.0000
42	1768	0.180	0.717	0.043	2575	0.2620	1.0442	0.0623	202	0.0002	0.0001	0.0000
45	1768	0.180	0.717	0.043	2576	0.2621	1.0444	0.0623	113	0.0105	0.0045	0.0009
48	1769	0.180	0.717	0.043	2762	0.2810	1.1200	0.0668	113	0.0002	0.0001	0.0000
51	1769	0.180	0.717	0.043	3101	0.3156	1.2576	0.0750	204	0.0001	0.0000	0.0000
54	1769	0.180	0.717	0.043	3594	0.3657	1.4574	0.0870	253	0.0002	0.0001	0.0000
57	1769	0.180	0.717	0.043	4312	0.4388	1.7485	0.1043	313	0.0082	0.0035	0.0007
60	1769	0.180	0.717	0.043	4312	0.4388	1.7486	0.1043	399	0.0372	0.0158	0.0031
63	1769	0.180	0.717	0.043	4313	0.4388	1.7487	0.1043	406	0.0378	0.0161	0.0032
66	1769	0.180	0.717	0.043	4313	0.4388	1.7488	0.1043	407	0.0379	0.0161	0.0032
69	1769	0.180	0.717	0.043	4313	0.4389	1.7488	0.1043	407	0.0379	0.0161	0.0032
72	1769	0.180	0.717	0.043	4313	0.4389	1.7488	0.1043	407	0.0379	0.0161	0.0032
75	1769	0.180	0.717	0.043	4313	0.4389	1.7488	0.1043	407	0.0379	0.0161	0.0032
78	1769	0.180	0.717	0.043	4313	0.4389	1.7488	0.1043	407	0.0379	0.0161	0.0032
81	1769	0.180	0.717	0.043	4313	0.4389	1.7488	0.1043	407	0.0379	0.0161	0.0032
84	1769	0.180	0.717	0.043	4313	0.4389	1.7488	0.1043	407	0.0379	0.0161	0.0032
87	1769	0.180	0.717	0.043	4313	0.4389	1.7489	0.1044	384	0.0357	0.0152	0.0030
90	1770	0.180	0.718	0.043	4313	0.4389	1.7490	0.1044	360	0.0335	0.0143	0.0028
93	1712	0.174	0.694	0.041	4315	0.4391	1.7497	0.1044	315	0.0293	0.0125	0.0025
96	1606	0.163	0.651	0.039	4319	0.4395	1.7515	0.1045	316	0.0294	0.0125	0.0025
99	1491	0.152	0.605	0.036	4328	0.4404	1.7549	0.1047	319	0.0297	0.0127	0.0025

Predicted Concentrations in Horizontal Drains and at Cell 1E Seepage Recovery Barrier

Using the pore water concentrations presented in the preceding section and the seepage flow rates, the seepage water quality was estimated for the water collected in the horizontal drains and at the Cell 1E Seepage Recovery Barrier (Figure 7-6). As discussed, the seepage water comprises seepage from the dam embankment, the coarse tailings beach, the fine tailings beach and the pond. Because the horizontal drains for Cell 2E are located within the LTVSMC tailings, the initial seepage will also contain LTVSMC pore water. For the purpose of this assessment it was assumed that the 'balance of seepage', i.e. total seepage less the seepage from the different sources will be LTVSMC pore water for the first seven years. Thereafter it was assumed that most of the porewater will have been displaced and that the pore water would reflect pond water quality. The estimated solute concentrations in the LTVSMC pore water and the pond water were as shown in Table 7-17. The estimated concentrations of key parameters in water collected in the horizontal drains and at the Cell 1E Seepage Recovery Barrier are summarized the following sections. Detailed results are provided in Appendix D.4.

Table 7-17: Summary of LTVSMC Tailings Pore Water and Pond Water Concentrations

Parameter	Units	LTVSMC Porewater	Pond Water
SO_4	mg/L	183	234
Sb	mg/L	0.0030	0.015
As	mg/L	0.0021	0.037
Cu	mg/L	0.0024	0.012
Ni	mg/L	0.0022	0.027
Zn	mg/L	0.010	0.091
Co	mg/L	0.0013	0.0026
Ca	mg/L	69	106
Mg	mg/L	81	26
Na	mg/L	97	52
K	mg/L	15	11
Ag	mg/L	0.0010	0.0012
B	mg/L	0.23	0.23
Be	mg/L	0.0002	0.0005
Cd	mg/L	0.0002	0.0008
Pb	mg/L	0.0071	0.0040
Se	mg/L	0.0020	0.0018
Tl	mg/L	0.0020	0.0022

Cell 2E – Horizontal Drains

Time profiles for the estimated sulfate, copper and nickel, and cobalt concentrations in seepage expected to be collected in the Cell 2E horizontal drains are shown in Figure 7-12. As shown, the concentration profiles indicate and initial peak concentration at about Year 12. A secondary peak is predicted to occur between about Year 60 and Year 80. To understand the development of these concentration profiles it is necessary to review the total volume of water from each source that will report to the horizontal drains. Figure 7-13 shows the combined flows from each of the sources for Year 1-8 and for the period Year 9 and above. The total represents the combined flows from all sources. Figure 7-14 illustrates the LTVSMC tailings pore water (Year 1 to 7) and the pond water flows, and the total volume of seepage collected in the horizontal drains. As shown in these plots, the initial peak in the concentrations occurs when the flow from the coarse tailings from the Year 1 to 8 period reaches a maximum. The secondary peak coincides with the maximum flows from the fine tailings beach from the period above Year 9. At that time, the embankment flows are also approaching its maximum. The source concentrations of the embankments are elevated and even at low flows have a significant impact on the water quality.

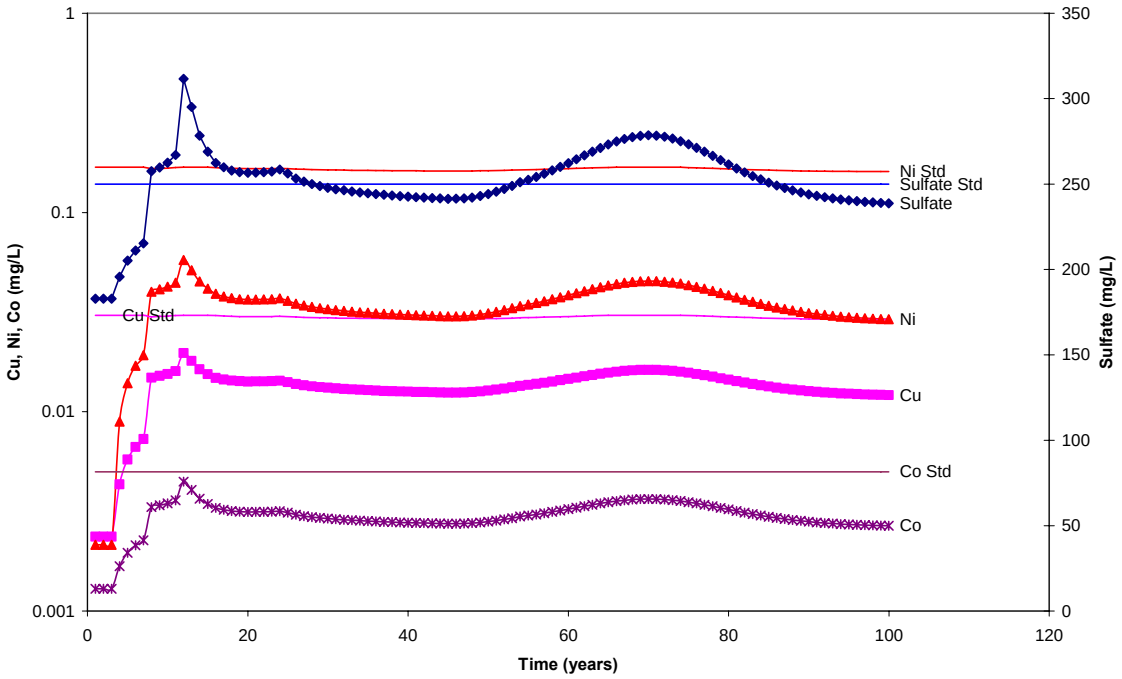


Figure 7-12: Estimated Sulfate, Cobalt, Copper and Nickel Concentrations in Seepage Collected in the Cell 2E Horizontal Drains

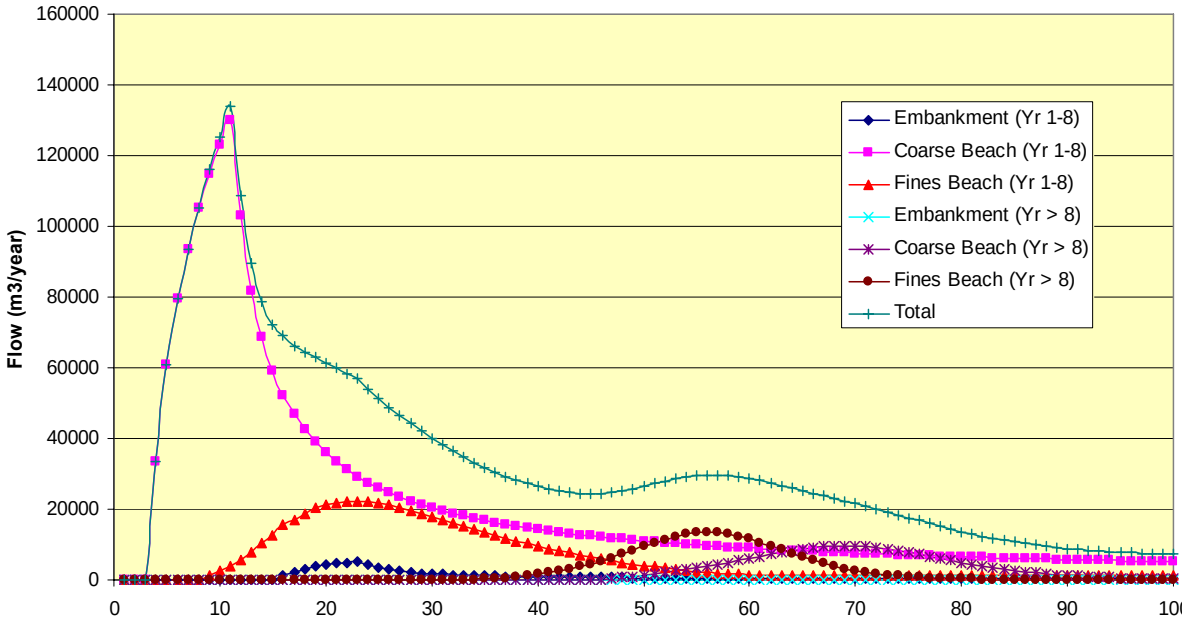


Figure 7-13: Estimated Flows Contributing to Seepage in the Cell 2E Horizontal Drains

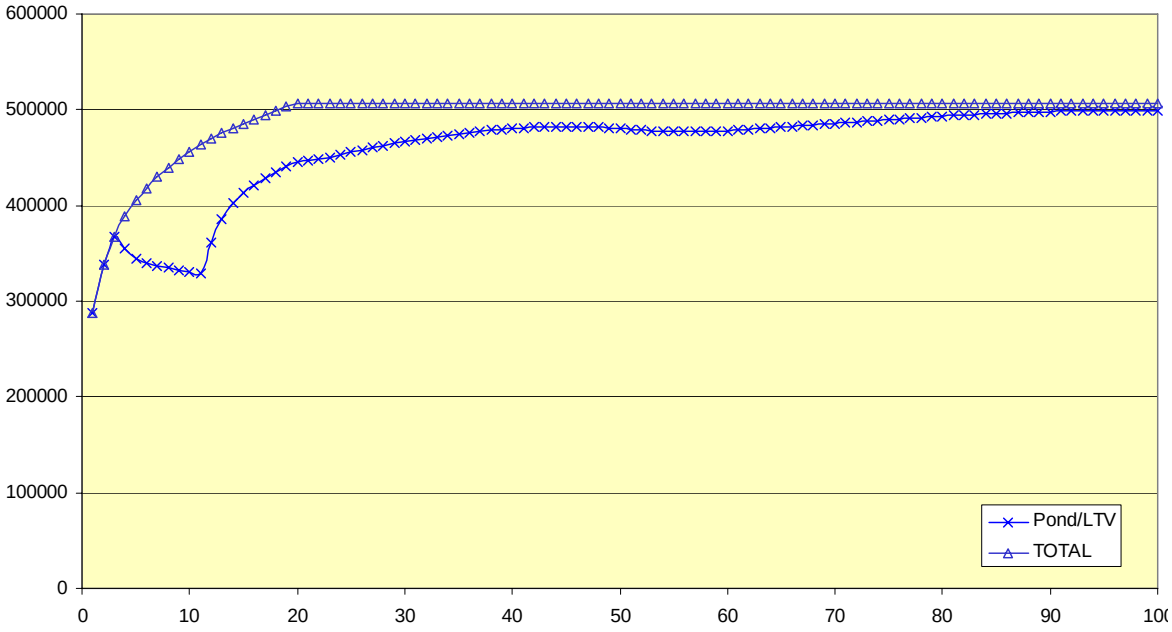


Figure 7-14: Estimated LTVSMC Porewater and Pond Water Flows Contributing to Seepage in the Cell 2E Horizontal Drains. Axis units are same as Figure 7-14.

Cell 1E Seepage to Horizontal Drains

Time profiles for the estimated sulfate, copper and nickel, and cobalt concentrations in seepage expected to be collected in the Cell 1E horizontal drains are shown in Figure 7-16. As shown, the concentration profiles indicate concentrations are predicted to increase rapidly from about Year 16 onwards. Peak concentrations are expected to occur at about Year 26 after which concentrations are expected to decrease again, to reach a plateau after about Year 35. These peak concentrations are caused by seepage originating from the coarse tailings beach.

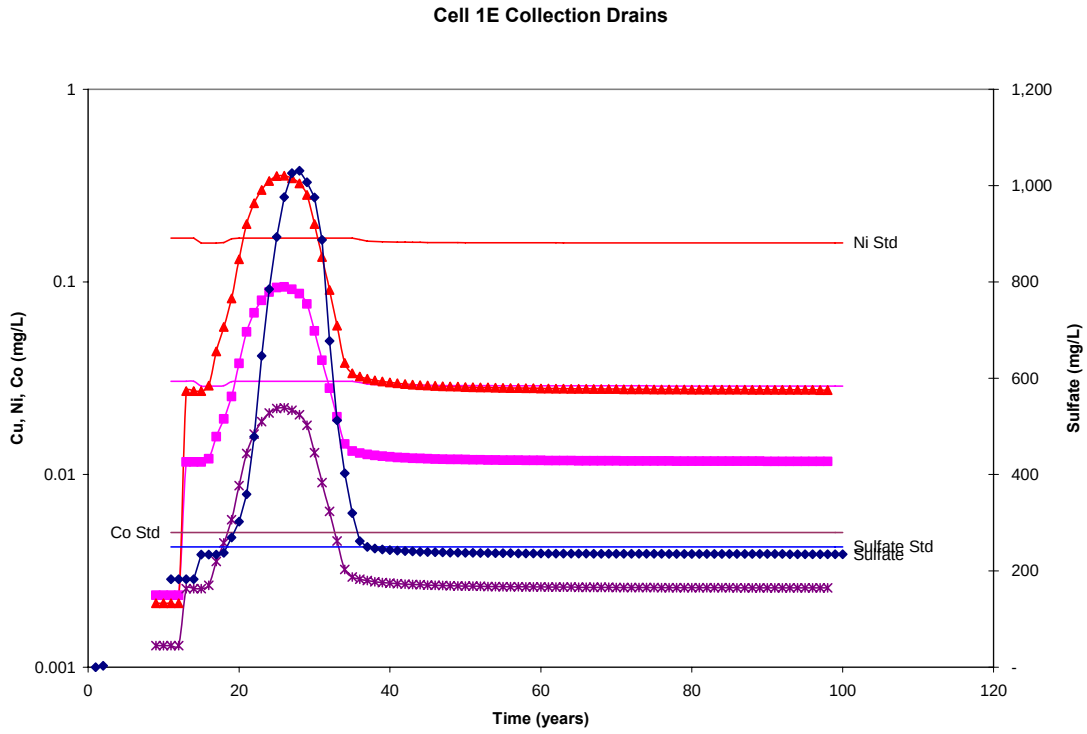


Figure 7-15: Estimated Sulfate, Cobalt, Copper and Nickel Concentrations in Seepage to Cell 1E Horizontal Drains

Cell 1E Seepage to Recovery Barrier

Time profiles for the estimated sulfate, copper and nickel, and cobalt concentrations in seepage expected to be collected in the Cell 1E seepage recovery barrier are shown in Figure 7-17. As with the horizontal drains collection system, the concentration profiles indicate concentrations are predicted to increase rapidly from about Year 16 onwards. However, maximum concentrations are substantially lower than those estimated for the Cell 1E horizontal drains due to significantly higher pond seepage contributions. Peak concentrations are expected to occur at about Year 26 after which concentrations are expected to decrease again, to reach a plateau after about Year 35. As with the horizontal drains, peak concentrations are caused by seepage originating from the coarse tailings beach.

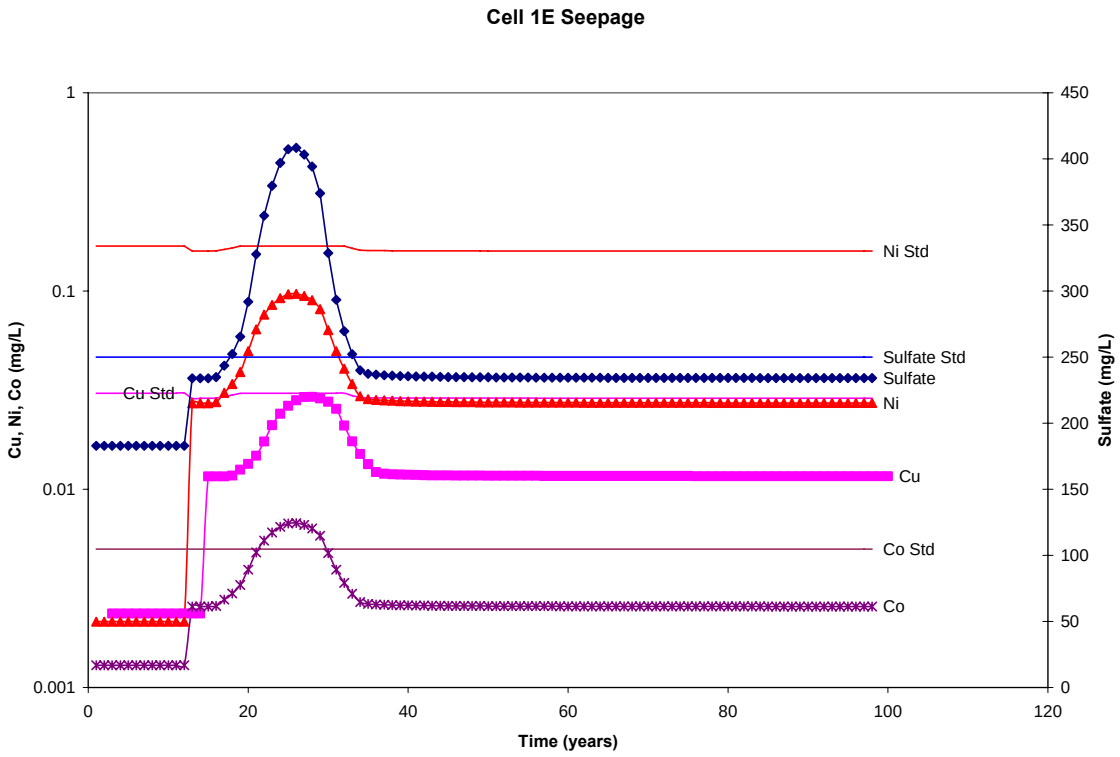


Figure 7-16: Estimated Sulfate Concentrations in Recovery Barrier Seepage from Cell 1E

7.3 Process Pond Model

7.3.1 Explanation of Modeling Approach

The water chemistry model couples water balance and chemical inputs to the operating tailings basin as described in RS13 (Barr 2007c). The model is calculated on a monthly time step, with the main output being the calculated chemical concentrations in the tailings basin at the end of each month. All calculations are done in Excel spreadsheets.

The tailings basin is subdivided into three separate cells (or basins): 2W, 1E and 2E. Only Cells 1E and 2E receive tailings and process water during operations. The model's flow components for Cells 1E and 2E are shown in Figure 7-18. The model simulates the effects of depositing tailings in only one cell (2E) for the first eight years (single pond operation) with Cell 1E used as a clear water basin to clarify flows prior to being returned to the process plant. After eight years, Cells 1E and 2E will merge and be operated as single pond.

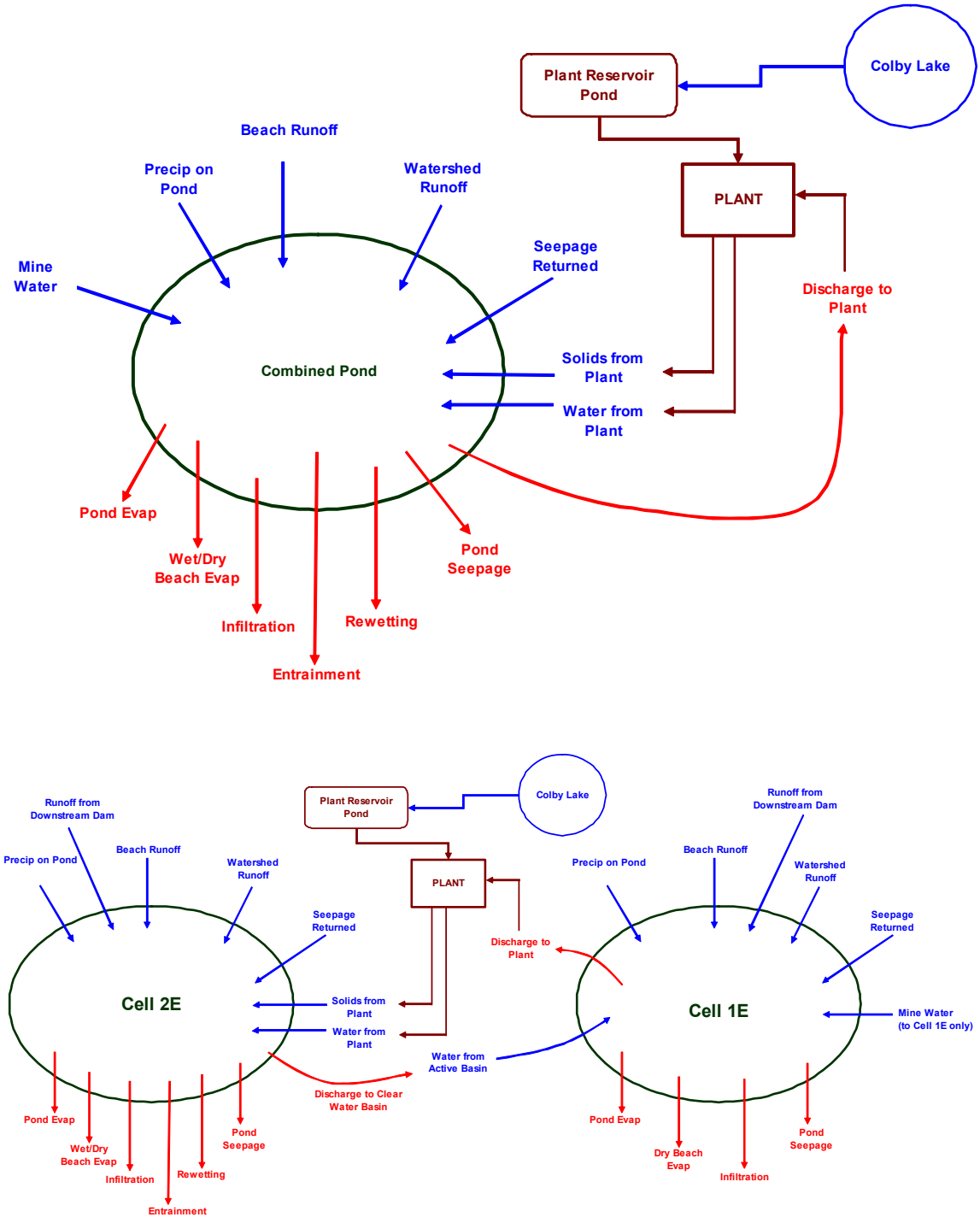


Figure 7-17: Water and Load Balance Components for Combined and Two Pond Operations

The overall flow balance is constrained by the required tailings beach and the associated water volume to maintain the beach. Each month, the net inflow (or outflow) to the basin is calculated such that the desired water volume (and beach) is maintained:

$$Net\ Monthly\ Inflow = \sum Inflows - \sum Outflows = Desired\ Volume\ Change$$

The variable that is changed to maintain the desired volume in each basin is either the reclaim to the flotation plant or the overflow to the inactive basin (Years 1 to 8).

The coupled load balance is represented as follows:

1. Load at the start of the month:

$$Load\ at\ Start\ of\ Month\ (L_1) = Concentration\ at\ Start\ of\ Month \times Volume\ in\ Pond\ at\ Start\ of\ Month$$

2. Load added to pond for each inflow:

$$Total\ Load\ added\ to\ Pond\ (L_2) = \sum [Inflows \times Concentration]$$

3. New concentration in pond assuming no outflows other than evaporation and rewetting:

$$Concentration_{NEW} = \frac{Load\ at\ Start\ of\ Month\ (L_1) + Load\ Added\ to\ Pond\ (L_2)}{Volume\ in\ Pond\ at\ Start\ of\ Month\ (V_1) + \sum Inflows\ (V_2)}$$

4. Load removed from pond at newly calculated concentration:

$$Load\ \mathrm{Re}moved\ from\ Pond\ (L_3) = \sum [Outflows \times Concentration_{NEW}]$$

5. Concentration at end of month is then the same as above-calculated concentration:

$$Concentration_{END} = \frac{Load\ at\ Start\ of\ Month\ (L_1) + Load\ Added\ (L_2) - Load\ \mathrm{Re}moved\ (L_3)}{Volume\ in\ Pond\ at\ Start\ of\ Month\ (V_1) + \sum Inflows\ (V_2) - \sum Outflows\ (V_3)}$$

The calculation of concentration assumes the ponds are perfectly mixed and that there is no load in the ponds at the beginning of the model. The model does not currently consider chemical attenuation effects, which could include precipitation of solids due to build-up of load in the pond and sorption effects that could serve to remove trace metal components. The model is therefore conservative from this standpoint.

The water and load balance in the flotation plant is calculated separately. The water inputs include reclaim from the inactive basin and make-up water from Colby Lake. Water from Colby Lake is adjusted in the model to maintain the required inflow to the flotation plant (see below). The balance does not consider the small change in flow and load losses when operating in Concentrate Mode.

The total inflows to and from the plant are set to the same values for the entire operations period, as provided in RS13 (Barr 2006):

Inflow = 1971 m^3/hr
Outflow = 1986 m^3/hr

The plant water balance is represented by:

$$\sum Inflows = \sum Outflows$$

The slightly higher water outflow from the plant is balanced by adding a small volume of water to the inputs so that the inflow is equal to the outflow. No load is associated with this inflow.

The load to the plant is calculated by combining the water inflows and associated chemical concentrations, along with the loads due to ore and reagents:

$$Total\ Load\ Added\ to\ Plant = \sum \left[Inflows \times Concentration \right] + Ore\ Load + \mathrm{Re}agent\ Load$$

The concentration of the plant water at the end of the month is calculated as follows (where the outflow is the value listed above):

$$Concentration_{END} = \frac{Load\ Added\ to\ Plant}{Outflow}$$

This concentration is then used to calculate the load added to the tailings pond from the plant for the following month.

All calculated metal concentrations are indicated as equivalent to "filtered" values. However, some of the parameters for watershed runoff and seepage return are totals as filtered values were not available. Unfiltered or "total" metal concentrations will be higher in the pond waters due to the presence of suspended matter.

7.3.2 Inputs to Water Quality Model

Solids and Water Balances

The solids and water balance has been calculated separately and are provided in RS13 (Barr 2007c). This calculation considers the operation of the pond, climatic effects and hydrogeological conditions in the vicinity of the pond. The load balance spreadsheet uses the flow and water balance information in RS13 (Barr 2007c) as an input.

Inputs to Loading Balance

The following sections describe the inputs to the load balance. Inputs were selected to be conservative so that calculated concentrations in the pond are reasonable worst case.

- Tailings Discharge

For the first month of the model, the tailings discharge load exiting from the flotation plant is calculated by multiplying the outflow from the plant by the concentrations derived from the pilot plant. In subsequent months, the tailings discharge chemistry is calculated from the load added as water passes through the flotation plant, as described in the previous section. The total load to the flotation plant is:

Load Withdrawn from Tailings Pond

+ Load Leached from Ore

+ Load Added by Reagents

+ Load in Colby Lake water

The concentration of the tailings discharge water at the end of the month is calculated by taking this total load and dividing it by the outflow from the plant during the month. This concentration is then equal to the beginning of month concentration for the following month. The load from the tailings discharge to the tailings pond is calculated by multiplying the beginning of month concentration of the plant water and the flow from the plant during the month.

- Ore Leaching

The contribution from ore as it is processed is expected to include dissolution of oxidation products and direct leaching of ore in the process by reaction of the sulfide minerals with the process waters. The latter factor cannot be readily quantified. The pilot plant testing (RS32, Barr 2006a) was performed on a core composite that had been exposed in some cases for several years, therefore, the pilot plant water chemistry data reflected the combination of both dissolution of oxidation products and reaction during processing. The approach taken to include the effect of ore leaching was therefore to calculate leached load based on weathering prior to arrival at the flotation plant (as described below) and compare this to the load contributed by ore leaching in the pilot plant. The calculation was performed using sulfate because this is expected to be chemically conserved in the process.

Because the calculated effect of leaching of oxidation products was more than the measured contribution from ore leaching in the pilot plant, the calculation proceeded using the calculated leaching effect.

The load leached from the ore was calculated using humidity cell results for the three ore samples (RS42, SRK 2007b). This load was calculated using the following:

Load = Maximum Average Rate of Leaching (mg/tonne/month) x

Temperature Factor (unitless) x

Ore Production (tonnes/month) x

Particle Size Factor (unitless) x

Exposure Period (months) x

(1 - Leaching Factor) (unitless)

The maximum average leaching rate was calculated based on the period of testing equivalent to the exposure factor. Tabulated rates used in the calculation are provided in Table 7-18. Conservatively, the input used to the calculations was the maximum of the three averages.

The temperature factor accounts for the lower overall average temperatures at the site compared to laboratory temperatures, and the approximate reduction in oxidation rates indicated by the Arrhenius Equation. The factor used was 0.3.

Table 7-18: Ore Weathering Rates

Ore Composite	Units	P1	P2	P3	Maximum
Acidity	mg/kg/week	1.2	1.2	1.3	1.3
Alkalinity	mg $CaCO_3$/kg/week	5.5	5	8.5	8.5
Hardness	mg $CaCO_3$/kg/week	26	19	23	26
F	mg/kg/week	0.062	0.024	0.026	0.062
Cl	mg/kg/week	0.11	0.1	0.11	0.11
SO_4	mg/kg/week	24	16	17	24
Al	mg/kg/week	0.019	0.013	0.025	0.025
Sb	mg/kg/week	0.0027	0.0027	0.0024	0.0027
As	mg/kg/week	0.0017	0.00091	0.00074	0.0017
Ba	mg/kg/week	0.0051	0.0037	0.0053	0.0053
Be	mg/kg/week	0.000095	0.000095	0.0001	0.0001
B	mg/kg/week	0.0065	0.0091	0.014	0.014
Cd	mg/kg/week	0.000019	0.000019	0.000021	0.000021
Ca	mg/kg/week	8.2	5.1	6.4	8.2
Cr	mg/kg/week	0.000095	0.000095	0.0001	0.0001
Co	mg/kg/week	0.00029	0.00022	0.00045	0.00045
Cu	mg/kg/week	0.002	0.0017	0.0017	0.002
Fe	mg/kg/week	0.0053	0.0067	0.0055	0.0067
Pb	mg/kg/week	0.000024	0.000024	0.000026	0.000026
Mg	mg/kg/week	1.6	1.5	1.5	1.6
Mn	mg/kg/week	0.0069	0.0042	0.0064	0.0069
Mo	mg/kg/week	0.000024	0.000024	0.000026	0.000026
Ni	mg/kg/week	0.0044	0.0042	0.011	0.011
Se	mg/kg/week	0.000095	0.000095	0.0001	0.0001
Ag	mg/kg/week	0.000024	0.000024	0.000026	0.000026
Zn	mg/kg/week	0.0006	0.00048	0.00051	0.0006

Ore production is indicated by the mine plan described in RS18 (PolyMet 2006e).

The particle size factor accounts for the difference in particle size of ore prior to crushing in the plant compared to the standard particle size of crushed ore in the humidity cell test. The factor used was 0.2. This indicates the average weathering and oxidation rate of ore at the site can be represented by a rate that is 20% of the laboratory measured rate.

The exposure period is the time the ore is exposed to weathering prior to being processed. This period starts when the ore is first exposed by removal of adjacent rock. This period was set as 6 months and was estimated by PolyMet based on the mine plan. This reflects the expected maximum time that ore will be exposed in the walls prior to being blasted and trucked to the rail transfer hopper. Time spent in the rail transfer hopper is relatively short.

The leaching factor accounts for rinsing of soluble load that will occur as rock is exposed to infiltration in the pit walls and rail transfer hopper over a 6 month time period. The factor was set as 0.5. This indicates that 50% of load will be leached and report to the pit or rail transfer hopper water collection system whereas the balance will remain to contribute to the tailings pond water. These loads are included in the water quality predictions provided elsewhere for these sources RS31 – Pit Water Quality (SRK 2007c), RS42 – Waste Rock Water Quality (SRK 2007b).

The effect of dissolution of explosives residues was based on the explosives recipe provided in the Mine Plan (RS18, PolyMet 2006). The overall mixture of emulsion, oxidizer and ANFO (ammonium nitrate and fuel oil) will contain 67% ammonium nitrate. Explosive use will be 0.33 lbs/ton, or 3720 lbs/blast hole. PolyMet have estimated that explosives losses will occur mainly from spillage at the point of loading the blast holes, and that the estimated spillage is 2 lbs/hole, or 0.05% of total explosives use. For safety reasons, PolyMet is committed to re-firing or explosives recovery for blast holes that do not detonate and therefore losses from this source were assumed to be negligible.

Contribution from explosives residues was calculated by assuming that all residues will be leached from the rock as it passes through the processing plant. In reality, some portion of the residues will be leached in the ore stockpile.

- Reagent Contribution

The only reagent that is expected to contribute to metal and sulfate concentrations is copper sulfate. The process will involve addition of 55 g/t copper sulfate pentahydrate ($CuSO_4.5H_2O$). Because the purpose of addition of copper sulfate is to activate pyrrhotite and improve flotation by reaction of copper with the pyrrhotite surfaces, only the sulfate portion was added to the load contributed by the flotation plant. The Pilot Plant test confirmed that this was an appropriate assumption. The pilot plant showed that copper load was removed from the process waters as they were re-cycled through the plant (RS29T, Barr 2007b).

- Colby Lake Water

The load added by Colby Lake is included based on the flow and concentrations. Concentrations used in the calculation are shown in Table 7-19. Data were not available for some parameters. If data were found for watershed runoff, these were applied to Colby Lake. In the absence of watershed runoff data, the seepage data were used. Few analytical results were available for nitrogen forms and low level mercury data were not available. Concentrations of nitrogen forms were entered as 0.1 mg/L.

Table 7-19: Averages for Colby Lake, Watershed Runoff and Seepage Pumpback Water Chemistry

Parameter	Unit	Watershed Runoff	Colby Lake
Alkalinity	mg/L as CaCO3	290	36
Cl	mg/L	3.3	6.1
F	mg/L	0.84	0.26
Hardness	mg/L as CaCO3	340	91
SO4	mg/L	33	53
Al	mg/L	-	0.31
Sb	mg/L	-	-
As	mg/L	-	0.0014
Ba	mg/L	-	-
Be	mg/L	-	-
B	mg/L	0.14	-
Cd	mg/L	-	0.000054
Ca	mg/L	40	23
Cr	mg/L	-	-
Co	mg/L	0.0013	0.00051
Cu	mg/L	0.0022	0.0048
Fe	mg/L	0.1	0.84
Pb	mg/L	-	0.00049
Mg	mg/L	52	8.3
Mn	mg/L	0.9	-
Hg	mg/L	0.0002	0.0002
Ni	mg/L	0.0026	0.0028
K	mg/L	3.3	1.7
Se	mg/L	-	0.0009
Ag	mg/L	-	-
Na	mg/L	11	6.3
Tl	mg/L	-	-
Zn	mg/L	-	0.0069
TDS	mg/L	440	130

Notes:

"-" indicates insufficient data or not analyzed. In calculations, concentrations were assumed to be zero.

- Precipitation

Precipitation incident on the tailings basin will fall on open water or bare tailings. Water falling on open water is assumed to be pure H_2O.

Water falling on exposed tailing beaches can be expected to dissolve soluble products of weathering and oxidation. Tailings particle sizes are expected to vary from coarse near the dams to fine near the pond due to particle size segregation as the tailings flow from the discharge point. Conservatively, the weathering rate applied to the beaches was calculated based on the maximum rates observed in testwork on the three tailings size fractions. The contribution of tailings weathering is a function of the exposed area of tailings. To estimate this affect for the short term when tailings will only briefly be exposed and wet due to placement, laboratory-measured humidity cell rates were expressed as $mg/m^2/month$, where m^2 reflects the aerial exposure of tailings and the load contribution was calculated using the area of exposed tailings as calculated in the water balance.

The area-based rates used in the calculation are shown in Table 7-20. These rates are based on the first six months of the testing and do not include the effects of accelerated nickel and cobalt leaching because there is an operational commitment to ensure that tailings are not exposed for more than six months. Because the rates are based on laboratory experiments, the six month period is conservative because weathering rates will be lower under site conditions.

- Watershed Runoff from Adjacent Areas

  The chemistry of water entering the pond as runoff from adjacent areas was based on data for sampling location WS009 reported in RS64 (Barr 2006d, Table 4) (Table 7-19). The input values used were averages because these are measured rather than predicted values. As shown, data were not available for a number of parameters. If data were available for Colby Lake, those values were used. If no data were available for Colby Lake, the values shown for groundwater were used in the prediction. Few analytical results were available for nitrogen forms and low level mercury data were not available. Concentrations of nitrogen forms were entered as 0.1 mg/L.

- Seepage Return

  The calculation of the chemistry of seepage returned to the impoundment from the horizontal drains is provided in Section 7.2.8.

- Make-up Water

Make-up water will originate from the Waste Water Treatment Facility (WWTF). The derivation of this water chemistry is provided in RS29T (Barr 2007b). Concentrations are provided in Table 7-21.

Table 7-20: Tailings Weathering Rates

		Maximum Average Rates		
Parameter	**Unit**	**Coarse Tailings**	**Fine Tailings**	**Maximum Rate Used in Calculations**
Alkalinity	mg as $CaCO_3/m^2$/month	2400	2500	2500
Cl	mg/m^2/month	26	25	26
F	mg/m^2/month	2.9	3.0	3.0
Hardness	mg as $CaCO_3/m^2$/month	3100	3500	3500
SO_4	mg/m^2/month	1000	1600	1600
Al	mg/m^2/month	7.1	7.5	7.5
Sb	mg/m^2/month	0.28	0.25	0.28
As	mg/m^2/month	2	0.096	2
Ba	mg/m^2/month	0.12	0.14	0.14
Be	mg/m^2/month	0.012	0.012	0.012
B	mg/m^2/month	2.1	1.8	2.1
Cd	mg/m^2/month	0.0024	0.0024	0.0024
Ca	mg/m^2/month	940	1100	1100
Cr	mg/m^2/month	0.016	0.018	0.018
Co	mg/m^2/month	0.009	0.011	0.011
Cu	mg/m^2/month	0.23	0.17	0.23
Fe	mg/m^2/month	1.2	2.0	2.0
Pb	mg/m^2/month	0.012	0.0094	0.012
Mg	mg/m^2/month	210	190	210
Mn	mg/m^2/month	0.71	0.8	0.8
Hg	mg/m^2/month	0.0013	0.0012	0.0013
Mo	mg/m^2/month	0.066	0.053	0.066
Ni	mg/m^2/month	0.16	0.15	0.16
K	mg/m^2/month	230	240	240
Se	mg/m^2/month	0.014	0.013	0.014
Ag	mg/m^2/month	0.003	0.003	0.003
Na	mg/m^2/month	75	67	75
Tl	mg/m^2/month	0.0016	0.0012	0.0016
Zn	mg/m^2/month	0.11	0.11	0.11

Table 7-21: Make-Up Water Chemistry from Waste Water Treatment Facility

Parameter	units	Year 1	Year 5	Year 10	Year 15	Year 20
Flow	gpm	674	1313	1378	875	1159
Hardness	mg/L	97	255	468	895	443
F	mg/L	0.29	0.39	0.86	1.83	1.60
Cl	mg/L	1.60	4.61	4.77	3.09	2.27
SO_4	mg/L	121	243	258	338	240
Al	mg/L	0.010	0.036	0.08	0.14	0.067
As	mg/L	0.004	0.008	0.008	0.020	0.012
Ba	mg/L	0.023	0.037	0.039	0.068	0.045
Be	mg/L	0.00029	0.00033	0.00047	0.00060	0.00041
B	mg/L	0.07	0.14	0.15	0.23	0.17
Cd	mg/L	0.0011	0.0010	0.0017	0.0023	0.0011
Ca	mg/L	150	150	150	150	150
Cr	mg/L	0.00062	0.0009	0.0010	0.0011	0.0011
Co	mg/L	0.00045	0.00121	0.0048	0.0092	0.0045
Cu	mg/L	0.00052	0.00113	0.0163	0.043	0.040
Fe	mg/L	0.046	0.086	0.080	0.034	0.027
Pb	mg/L	0.0015	0.0062	0.0069	0.013	0.008
Mg	mg/L	0.52	1.58	4.15	7.41	3.46
Mn	mg/L	0.0022	0.0051	0.0079	0.014	0.0086
Hg	mg/L	$5.0x10^{-6}$	$5.5 x10^{-6}$	$6.4 x10^{-6}$	$1.1 x10^{-5}$	$7.7 x10^{-6}$
Mo	mg/L	0.0020	0.0036	0.0038	0.0036	0.0043
Ni	mg/L	0.0065	0.0126	0.039	0.076	0.042
P	mg/L	0.0072	0.012	0.012	0.025	0.014
K	mg/L	3.77	6.54	6.24	12.6	7.7
Se	mg/L	0.0012	0.0017	0.0018	0.0020	0.0020
Si	mg/L	0.59	0.84	0.59	1.45	0.92
Ag	mg/L	0.00043	0.00072	0.00076	0.00071	0.00086
Na	mg/L	14.8	58.3	44.0	117	72
Tl	mg/L	0.00070	0.0012	0.0013	0.00094	0.0014
Zn	mg/L	0.036	0.13	0.21	0.32	0.16
NO_3	mg/L	0.051	0.13	0.13	0.023	0.012
NH_4	mg/L	0.051	0.13	0.13	0.023	0.012

Loading Balance Outputs

The majority of water outputs (return to flotation plant, entrainment in tailings interstices, infiltration and loss through seepage) are assigned the calculated chemistry of the pond. The exceptions are evaporation (no load assigned) and beach wetting (assigned flotation plant water chemistry).

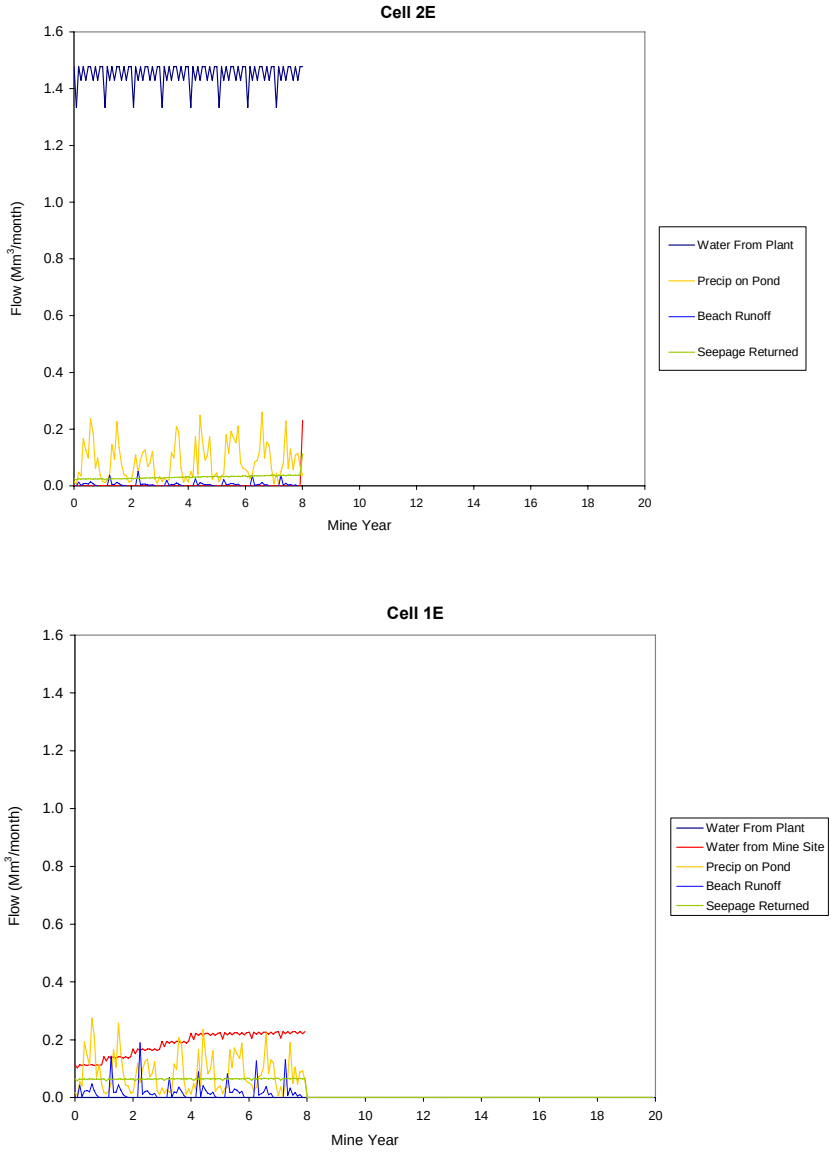
7.3.3 Results

Water Chemistry Prediction as a Function of Time

The water balance is described in RS13 (Barr 2007c). Flows to both cells are shown in Figure 7-19. The process plant provides the majority of water to both ponds and provides a constant loading resulting mainly from leaching of ore as it is processed but also Colby Lake water. The next largest sources of water are the treated water from the mine area and direct precipitation. The volume of water from the mine site reflects both increases in the footprint of disturbed area at the mine site (initial flow increases), and decreases in flow due to completion of mining in the East Pit, use of West Pit water to fill the East Pit and decrease in requirement for treatment due to reclamation. Seepage pumpback and precipitation on the tailings beaches are lesser sources of flow.

Predicted concentrations for sulfate, hardness, copper and nickel are shown in Figure 7-19. Listings for all parameters for which water quality standards are provided in Appendix D.5. The water quality standards are applicable in the receiving waters. During permitting, water quality discharge limits are developed to maintain the standards. Because the point of compliance is typically at a facility boundary, allowance can be made for mixing zones and site specific standards can be developed, a conservative assumption is that the water discharge limits will be no less than the water quality standard. PolyMet is not proposing a discharge but the water quality standards are used as water quality objectives for the project.

The predicted trend is for concentrations to sharply increase. The initial increase results from assuming that initial pond waters are very dilute. As concentrations increase, loss of water from the system (primarily by entrainment in the tailings pores) also removes load which balances the load added to the pond. The long term trends are different for each parameter and reflect different inputs as described in the following sections.



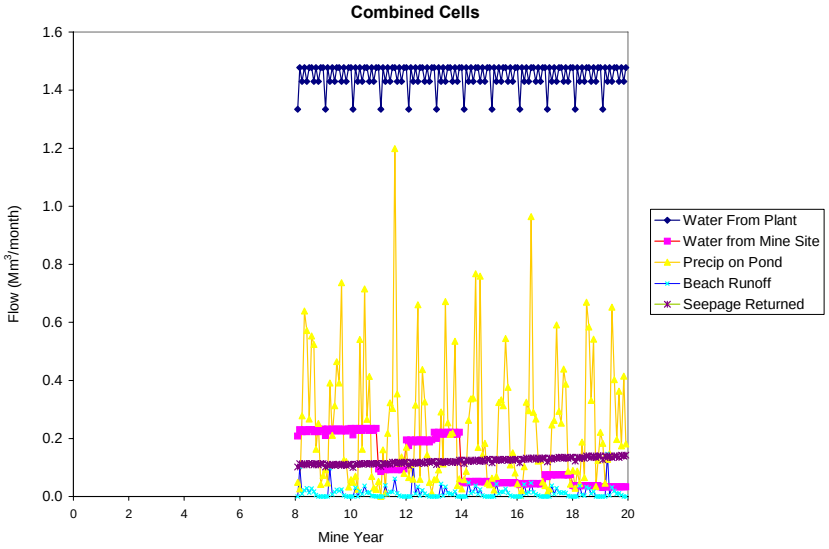


Figure 7-18: Flow Sources to Tailings Cells.

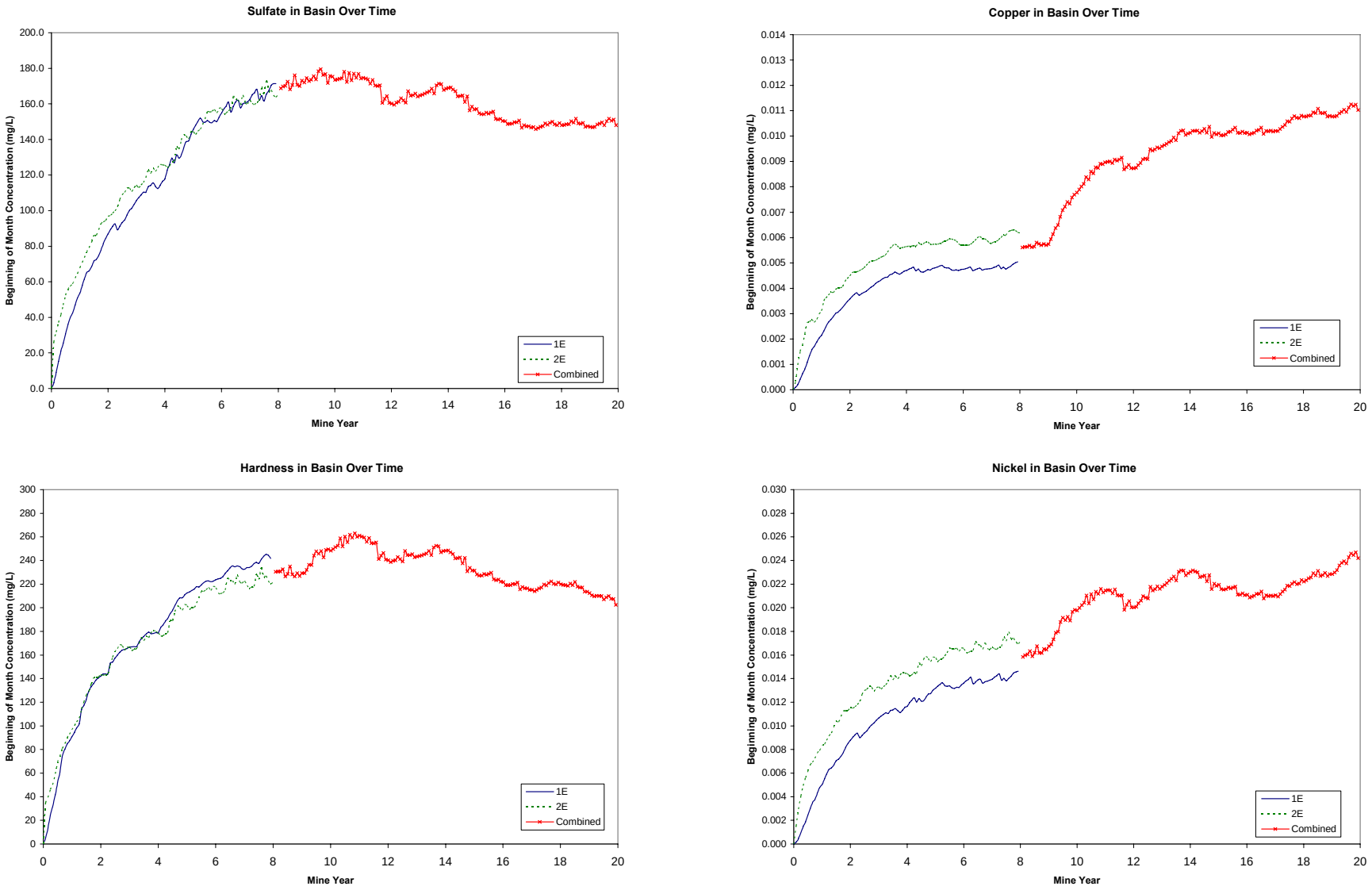


Figure 7-19: Predicted Sulfate, Hardness, Copper and Nickel in Tailings Ponds

The sulfate trend shows concentrations reaching a maximum of 180 mg/L in the combined pond in year 10 followed by a general decrease and then stabilization at about 150 mg/L. The main sources of added sulfate discharging to the pond are the WWTF flow which varies from 121 mg/L (Years 1 to 4) to 338 mg/L (Years 15 to 19), and seepage pumpback from the 2E horizontal drains (peaking over 300 mg/L during the operational period. The 1E horizontal drains do not show peak sulfate until after Year 20. Recirculation of water through the plant and addition of load by ore leaching results in accumulation of load. The hardness trend (i.e. calcium and magnesium) is similar to sulfate for the same reasons.

The copper trend was different from sulfate. Copper trended upward reaching maximum concentrations of 0.011 mg/L. Initial loadings to the pond are low and the main factor causing the increase is accumulation of load due to re-circulation through the mill. The increase in concentrations that occur in about Year 9 is due to a predicted increase in copper concentrations in WWTF effluent (from 0.001 mg/L to 0.016 mg/L). Copper loading from the Cell 2E horizontal drains also increases at about the same time. Copper concentrations are predicted to be below the hardness based water quality standard.

Likewise most of the initial accumulation of nickel load can be accounted for by re-circulation of water through the plant. Contributions from ore leaching and discharge from the WWTF are similar. The increase in nickel concentrations at Year 10 is due to increase in nickel in the WWTF discharge (from 0.01 mg/L to 0.04 mg/L in Year 15). Also, later in operations, nickel loading from Cell 1E and 2E horizontal drains becomes important. Nickel concentrations are predicted to be below the hardness based water quality standard. Cobalt (not shown) has a comparable trend to nickel and maximum predicted concentrations of 0.002 mg/L are expected to be below the water quality standard of 0.005 mg/L.

Overall predictions indicate that the tailings pond water can meet water quality objectives for most parameters by controlling the chemistry of WWTF effluent. Parameters for which water quality objectives are predicted to be exceeded are manganese, silver and thallium. The predictions for manganese are driven by naturally elevated concentrations in natural concentrations in surface water (Table 7-19) and groundwater (Table 4-1). Thallium and silver predictions are affected by the use of analytical detection limits above (thallium) or at (silver) their respective standards for some chemical balance inputs.

Mercury concentrations were not predicted for the pond due to the lack of low level analyses for several sources. Contact with tailings in the plant is expected to result in control of mercury concentrations in the pond. Testwork performed for RS54 showed that mercury concentrations in contact with tailings are between 2 and 5 ng/L. Likewise, NTS (2006) showed that contact with regional rain water containing 10 ng/L resulted in a similar range of concentrations. Pond concentrations are expected to be in the same range or less.

Effect of Lined vs. Un-Lined Disposal Basins

The Detailed Project Description assumes the basins will be constructed without liners. The effect of liners was not considered.

Sensitivity Analysis

To assess the sensitivity of the predictions for process water in the tailings ponds, the proportion of loadings from each source Year 8 (last year of single pond deposition) and Year 20 (last year of operation) was evaluated (Table 7-22).

For sulfate, the largest single source in Year 8 is the mine site water (40%), following by comparable loads from ore leaching, the use of copper sulfate in the process, make-up water from Colby Lake and total seepage collection. Colby Lake is a relatively large flow at low concentration. Toward the end of the operating period, reduced flow from the ine site (due to reclamation efforts and use of treated water to flood the West Pit) reduces the contribution from this source and total seepage becomes the main source (43%). Make-up water from Colby Lake continues to contribute significantly in Year 20 (21%).

The pattern of increasing contribution from seepage return is apparent for the three metals shown. In Year 20, the total loading from seepage is 60%, 51% and 59% for cobalt, copper and nickel, respectively. At the same time, the contribution from the mine site decreases to become insignificant at Year 20. The contribution from other sources varies by parameter. For cobalt, 40% of load in Year 8 comes from Colby Lake make-up and watershed runoff. For copper, Colby Lake and watershed runoff account for 55% of the load. For nickel, ore leaching is the dominant load in Year 8 (60%), and is second to seepage return in Year 20 (32% compared to 59%).

Table 7-22: Proportion of Loads by Source Contributing to the Process Water in the Tailings Pond

Year	Ore	Reagent	1E Seepage	2E Seepage	Colby Lake	Watershed Runoff	Tailings Beaches	Mine Site
SO_4								
8	0.12	0.14	0.07	0.05	0.17	0.05	0.01	0.40
20	0.13	0.14	0.35	0.08	0.21	0.00	0.02	0.06
Co								
8	0.26	0.00	0.07	0.04	0.18	0.22	0.00	0.23
20	0.19	0.00	0.52	0.08	0.16	0.01	0.01	0.03
Cu								
8	0.30	0.00	0.06	0.02	0.45	0.10	0.02	0.05
20	0.16	0.00	0.43	0.07	0.30	0.00	0.04	0.00
Ni								
8	0.60	0.00	0.02	0.01	0.10	0.04	0.01	0.23
20	0.32	0.00	0.53	0.06	0.06	0.00	0.01	0.02

This review indicates that reagent contributions and runoff from tailings beaches are relatively small loading sources for metals. Potentially the more significant sources in terms of loading are ore, tailings seepage collected in horizontal drains, Colby Lake and mine site water. Colby Lake is more a source of dilution but it affects the base level available to assimilate the load contributed by other sources.

The ore term is "worst" case because it is based on the highest rate of ore leaching observed in testwork. Because the samples under test had been oxidized in core boxes longer than will be exposed in the mine prior to mining, the leaching rates indicated by the tests are also likely to be worse than would naturally be observed.

The seepage terms contain several conservative factors, some of which may over-state leaching from the tailings seepage source. These include:

- The calculation of oxidation profile development assumed that coarse tailings are uniformly of the same particle size. No allowance for fine and coarse layers was included which would increase moisture content and reduce oxygen penetration.
- Long term (decades) rates of metal leaching were based on metal leaching effects observed in testwork which are known to be short-lived (years).
- No allowance for attenuation of metals within the NorthMet Project tailings deposit or in the materials below the tailings was included.
- Colby Lake is a significant source but its loading contribution is well-defined. The chemistry of the lake has been measured and the volume of make-up water is defined by the metallurgical process.

The contribution from the mine site is influenced by prediction of the water quality from the waste rock stockpiles and mine pits, and the ability of the water treatment process to achieve the concentrations used to predict process pond water quality. As discussed in RS42 (SRK 2007b) and RS31 (SRK 2007c), the waste rock and open pit predictions are based mainly on the conservative use of 95th percentile weathering rates and consequently are considered "reasonable worst case". Furthermore, water treatment studies are based on known and well demonstrated technology.

In conclusion, the predictions are expected to be sensitive mainly to the seepage return term and water from the mine site but both sources were estimated using methods that were reasonable worst case for loadings. Actual loadings are expected to be lower than calculated.

7.4 Seepage to Groundwater

The final step in the water quality prediction is for all uncollected seepage flowing north and northwest into groundwater toward the Embarrass River. This uncollected seepage water is predicted to not be intercepted by the Cell 2E horizontal drains or the other seepage collection systems. For Years 1 to 8, this reflects the operation of Cell 2E for tailings disposal and Cell 1E as the clear water

pond. Following Year 8, the seepage originates from the combined pond and the seepage chemistry reflects the influence of tailings deposition occurring in the original Cell 2E and Cell 1E. The uncollected seepage and groundwater that flows beneath the tailings facility are only to the north and northwest.

Time profiles for the estimated sulfate, copper and nickel, and cobalt concentrations in seepage to groundwater are shown in Figure 7-20. As shown, the seepage water quality is expected to vary little over time. The seepage water quality is expected to mainly be dominated by the pond water quality. All concentrations are predicted to be below water quality standards with the exception of manganese, silver and thallium, for the reasons discussed previously.

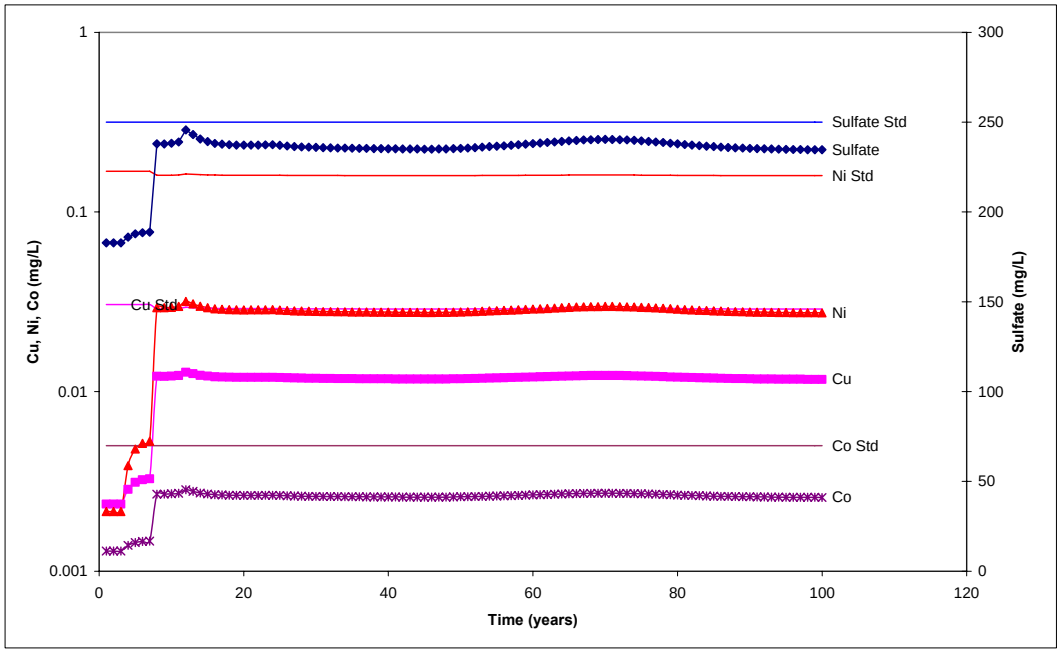


Figure 7-20: Estimated Sulfate, Cobalt, Copper and Nickel Concentrations in Seepage to Groundwater

8 Conclusions

The following conclusions have been reached based on historical testwork completed by the MDNR and PolyMet's recent project-specific characterization of tailings:

- Pilot plant testwork showed that the proposed flotation process can produce tailings containing much less than 0.2% sulfur. This level of sulfur can be achieved using copper sulfate to improve flotation of pyrrhotite.
- Tailings from the NorthMet Project containing negligible carbonate which is consistent with the origin of the mineral deposit.
- Four years of testwork on tailings containing 0.2% sulfur from the Babbitt Deposit produced by Cominco Ltd. did not result in acidic leachate despite depletion of over 50% of sulfur.
- The explanation for the lack of acidic leachate is that alkalinity produced by weathering of silicates exceeds the acidity produced by oxidation of sulfides.
- Geochemical predictions supported by the MDNR's 18 years of testwork on waste rock samples discussed in RS42 shows that at the sulfur concentrations expected in the NorthMet tailings, alkalinity produced by weathering of silicates will always exceed acid produced by oxidation of sulfides.
- Small decreases in pH near 7 appear to cause short term (several months) peaks in nickel and cobalt leaching possibly due to dissolution of silicate weathering products. This effect has been demonstrated by MDNR testwork conditions that artificially depressed pH in the tailings, and PolyMet testwork on coarse (>200 mesh) tailings samples.
- Provided that sulfur concentration in tailings produced using copper sulfate to enhance flotation of pyrrhotite remain below 0.2%, tailings pHs are expected to remain above pH 5.5.
- Column testwork showed that the LTVSMC taconite tailings are more leachable in terms of major ion chemistry than the PolyMet tailings. PolyMet tailings are expected to leach more nickel than the LTVSMC tailings but the LTVSMC removed nickel from the NorthMet tailing water.
- Predictions of tailings seepage chemistry show that water chemistry collected in the horizontal drains will exceed water quality standards for some parameters.
- Water collected at Cell 1E seepage recovery barrier will only exceed the sulfate standard and marginally the cobalt standard. The predictions did not consider metal attenuation and are based on worst case assumptions, therefore actual exceedances for cobalt are not expected.
- A mass balance model for the water in the operational tailings impoundment showed that concentrations should be below water quality standards for most parameters. Silver and

thallium could not be reliably predicted because analytical detection limits for some inputs were above the standards. Manganese exceeds it standard due to naturally elevated concentrations in surface water and groundwater.

- All uncollected seepage from the tailings basin will flow with groundwater to north and northwest toward the Embarrass River. Overall, it is predicted that this water will not exceed water quality standards except for manganese, silver and thallium for the reasons indicated previously.

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This report, **1UP005.001 – RS54/RS46 – Waste Water Modelling – Tailings, NorthMet Project - DRAFT,** was prepared by SRK Consulting (Canada) Inc.

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Appendices

Appendix A

LTVMSC Tailings Drilling Program

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Memo

To:	Jennifer Engstrom, MDNR	**Date:**	September 23, 2005
cc:	Jim Scott, PolyMet	**From:**	Stephen Day
	John Borovsky, Barr		
Subject:	Drilling Program	**Project #:**	1UP005.001
	LTVSMC Tailings Area		

Jennifer

As a follow-up to our meeting on September 14, 2005, PolyMet, SRK and MDNR have designed a program to obtain initial samples for characterization of LTVSMC tailings. These data are needed to design the laboratory column testing on the interaction of NorthMet Project tailings water with the LTVSMC tailings. This memorandum describes the program.

1 Background and Program Design

Review of the history of deposition at the tailings area indicates that the tailings are a result of processing ore feed from several pits and working faces in those pits. The ore was blended by truck delivery to loading pockets and then train delivery to crusher. Once in the plant ore was further blended in the coarse ore bins by a coarse ore tripper which continuously spread coarse ore across 7 fine crushing lines and in the fine ore bins by a fine ore tripper which continuously spread fine ore across 34 mill lines. The fine ore was then processed through 34 mill lines in parallel. The tailings are a recombination from the 34 mill lines pumped to the basin and discharged at many spigots at the periphery of the basin. The tailings were deposited over many years as many layers in the basin.

Since deposition occurred at the periphery, tailings near the margins are likely to be coarse whereas slimes tailings in the center of the impoundment will be finer. Based on this distribution, the preferential pathway for groundwater flow from the impoundment will be through the dam tailings along the beaches. This zone will likely have the higher groundwater velocities, shorter flow path and smaller contact area for attenuation reactions. Actual flow paths will depend on internal layering in the tailings, and the design and operating plan for management of NorthMet Project Tailings. Overall, material in the beaches represents the least favourable conditions for attenuation reactions.

Based on this history and conceptual flow model, a few deep holes in the dam and extending to the slimes area should provide an initial indication of the physical, chemical and mineralogical variations due to historical deposition and particle size segregation.

PolyMet is not aware of any existing characterization data that will allow the location of the proposed holes to be specifically selected to capture known variability. Therefore, a line of holes will be drilled perpendicular to the dam where access is favourable and safe. Once a tailings basin construction and operating plan has been finalized, submitted and approved, additional drilling can be defined to more fully characterize the variability in the basin.

2 Field Program

PolyMet will mobilize a geoprobe to the site. This rig advances a 2" core barrel producing continuous solid core for logging and sampling. The approximate proposed drilling locations are shown in Figure 1[1].

One hole will be drilled in the dam area of Cell 2E (Figure 1) where flow from the disposal area will be focussed and where tailings are expected to be coarse. Five additional holes will be drilled perpendicular to this location extending into the slimes area. If practical all holes will be advanced to the foundation. The penetration depth of the geoprobe is approximately 80'. As indicated above, actual locations will be selected based on access and safety considerations in the field.

During drilling, core will be logged for visual characteristics including colour, particle size, moisture content, odor, reaction with dilute hydrochloric acid and magnetism. Sampling intervals will be defined to characterize relatively homogenous layers expected to be indicated primarily by particle size and possibly colour, or zones of finely interbedded coarser and fine layers. Individual sampling intervals will likely be no thinner than 3' unless field data indicate a need for narrower intervals.

3 Analysis

Core intervals will be selected based on review of core logs and analyzed as follows:

- Physical parameters (particle size, moisture, relative density)
- Chemical parameters (total oxides, trace elements).
- Mineralogy. XRD will be used to provide screening level results to select suitable samples for column testing. Detailed optical and sub-optical mineralogical analyses of samples used in column testing, including variation in composition as a function of particle size will be completed to assist with interpretation of results.

4 Application of Characterization Data to Column Test Design

Field observations and chemical analyses will be used to design column tests to evaluate the interaction of NorthMet Project tailings water with the LTVSMC tailings. The column tests could use a few (two to three) composites of the LTVSMC beach tailings. The composites will be prepared to represent a range of conditions that could influence interactions such as mineralogical differences (inferred from chemical analyses and colour) and particle size. A detailed program will be developed by PolyMet, SRK and MDNR prior to initiation.

[1] Note that the tailings disposal concept involves use of Cell 2W for disposal of hydrometallurgical wastes in lined cells and use of basins 1E and 2E for disposal of flotation tailings.

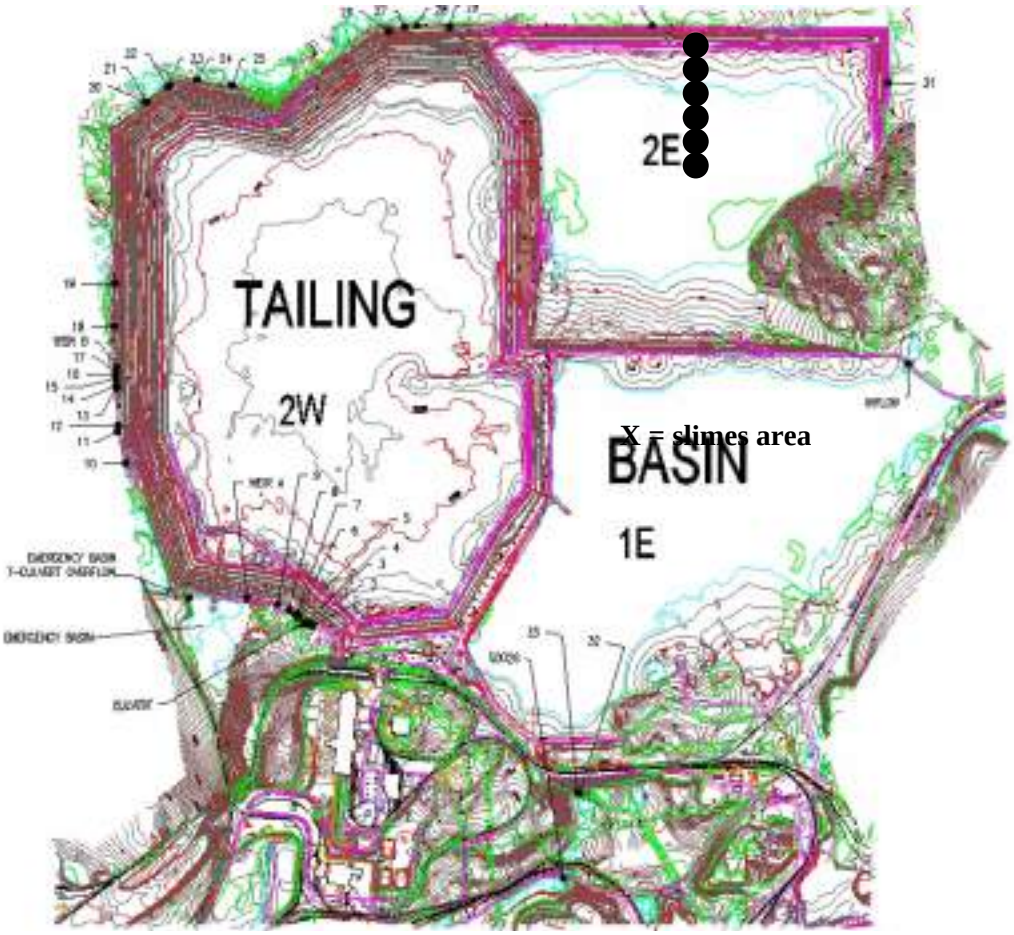


Figure 1. LTVSMC Tailings Area. Conceptual drilling locations in Cell 2E are shown. Actual locations will be finalized in the field based on access and safety considerations.

Appendix B

Static Characterization of PolyMet Tailings Samples

Appendix B.1

Tailings Petrographic Description

Appendix B.1
Tailings Petrographic Description

Description of Samples for Petrographic Report

The subsequent mineralogical report received from PolyMet Mining Inc. describes whole PolyMet tailings characteristics. PolyMet tailings samples were generated from three ore parcels (P1, P2, and P3). Parcel 2 (sample P2S) was processed entirely without using copper sulfate. Parcel 1 was evaluated both with (samples P1S and P1SA) and without (sample P1SOLIDS) copper sulfate. Parcel 3 (sample P3S) was prepared using copper sulfate additive.

Appendix B.1
Tailings Petrographic Description

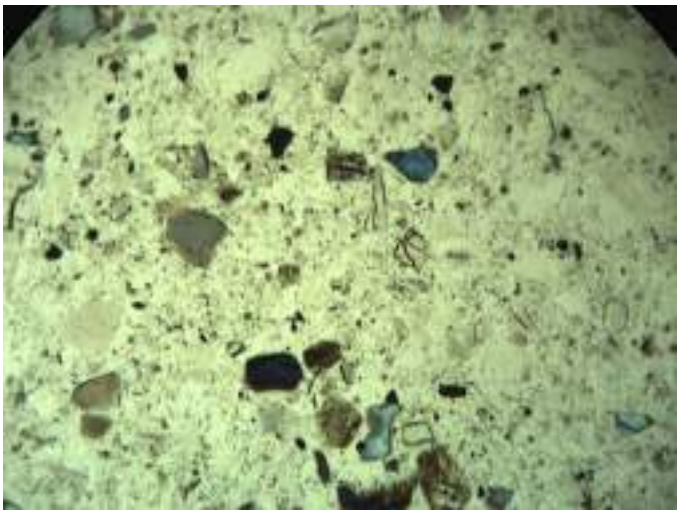
Sample Identification	Vancouver	Description	Mineralogy								
	Petrographic #		% Gypsum	Grain Size	% Plagioclase	Grain Size	% Olivine	Grain Size	% CPX	Grain Size	% OPX
P1S	Z-1	Tailings prepared with CuSO4			80.00	0.01-0.30	12.00	0.01-0.20	4.00	0.01-0.20	1.00
P1SA	Z-1A	Tailings prepared with CuSO4			75.00	0.01-0.20	15.00	0.01-0.15	5.00	0.01-0.20	2.00
P1SOLID	Z-2	Tailings prepared without CuSO4			60.00	0.01-0.10	15.00	0.01-0.10	5.00	0.01-0.10	1.00
P2S	Z-3	Tailings			50.00	0.01-0.15	10.00	0.01-0.10	4.00	0.01-0.15	
P3S	Z-4	Tailings			60.00	0.01-0.20	10.00	0.01-0.15	5.00	0.01-0.30	1.00
			Sulfides								
			% Pyrite	Grain Size	% Pyrrhotite	Grain Size	% Chalcopyrite	Grain Size	% Bornite	Grain Size	% Cubanite
P1S	Z-1	Tailings prepared with CuSO4	rare	0.05	0.25	0.01-0.10	rare	0.05			
P1SA	Z-1A	Tailings prepared with CuSO4	rare	0.05-0.10	0.25	0.01-0.10	rare	0.01			
P1SOLID	Z-2	Tailings prepared without CuSO4	rare	0.01-0.10	0.25	0.01-0.05	rare	0.04			
P2S	Z-3	Tailings	rare	0.01-0.10	0.50	0.01-0.10	rare	0.01-0.05			
P3S	Z-4	Tailings	rare	0.15	0.25	0.01-0.10	rare	0.01-0.04			
*Note: All grain sizes in mm											

Sample Identification												
	Grain Size	**% Biotite**	**Grain Size**	**% Chlorite**	**Grain Size**	**% Serpentine**	**Grain Size**	**% Sericite/Musc.**	**Grain Size**	**% Quartz**	**Grain Size**	**% Calcite**
P1S	0.01-0.10	1.00	0.05-0.25	0.50	0.01-0.20			0.25	0.01-0.05			
P1SA	0.01-0.10	1.00	0.05-0.30	0.25	0.01-0.10			0.50	0.01-0.02			
P1SOLID	0.01-0.05	1.00	0.05-0.15	1.00	0.01-0.05			1.00	0.01-0.05			
P2S		1.00	0.05-0.10	1.50	0.01-0.05			2.00	0.01-0.02			
P3S	0.01-0.10	1.00	0.05-0.30	1.00	0.01-0.15	0.25	0.01-0.05	1.00	0.01-0.03			
	Grain Size	**% Pentlandite**	**Grain Size**	**% Violarite**	**Grain Size**	**% Silver/PGE**	**Grain Size**	**% Mack./Vall.**	**Grain Size**	**% Sphalerite**	**Grain Size**	**% Galena**
P1S										rare	0.01-0.10	rare
P1SA										rare	0.01-0.10	rare
P1SOLID										rare	0.01-0.10	rare
P2S												rare
P3S										rare	0.01-0.05	rare
*Note: All grain sizes in m												

Sample Identification										
	Grain Size	**% Ilmenite**	**Grain Size**	**% Magnetite**	**Grain Size**	**% Graphite**	**Grain Size**	**% Clay/Unidentified**		**Total**
P1S		1.00	0.01-0.10							100.00
P1SA		1.00	0.01-0.15							100.00
P1SOLID		0.75	0.01-0.10					15.00		100.00
P2S		1.00	0.01-0.10					30.00		100.00
P3S		0.50	0.01-0.15					20.00		100.00
	Grain Size									**Sulfide (only) Total**
P1S	0.04									0.25
P1SA	0.05									0.25
P1SOLID	0.01-0.05									0.25
P2S	0.01-0.02									0.50
P3S	0.03									0.25
*Note: All grain sizes in m										

Appendix B.2

Tailings Photomicrographs



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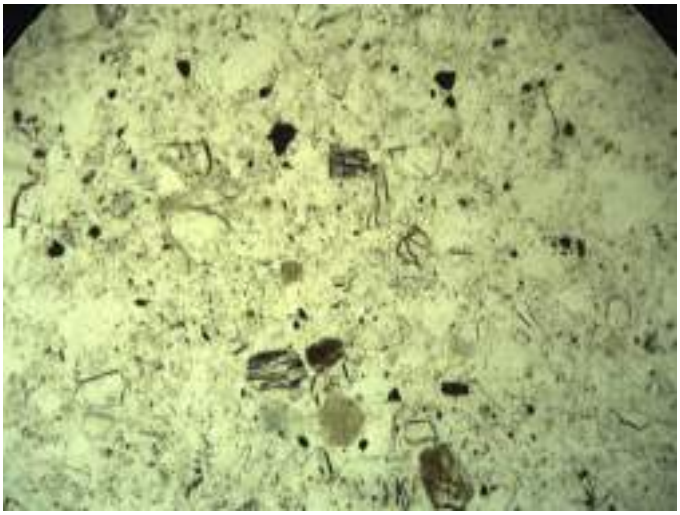
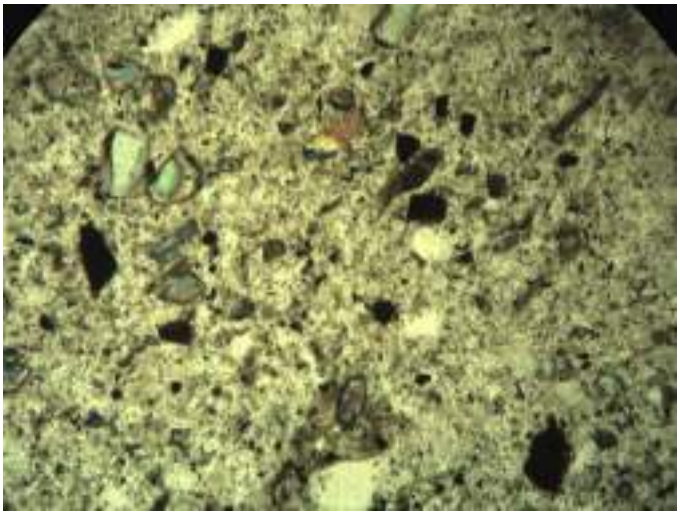


B

C

P1S Bulk Tailings: A) Cross Polarized Light, Field of View = ~0.8 mm, B) Plain Polarized Light, Field of View = ~0.8 mm, C) Reflected Light, Field of View = ~0.8 mm



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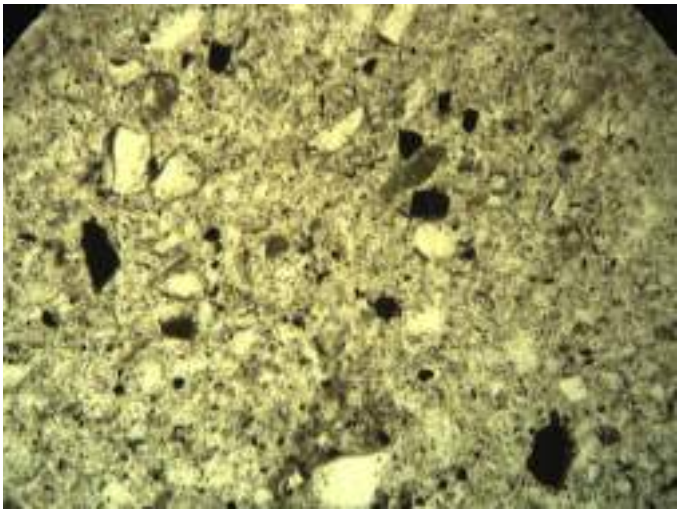

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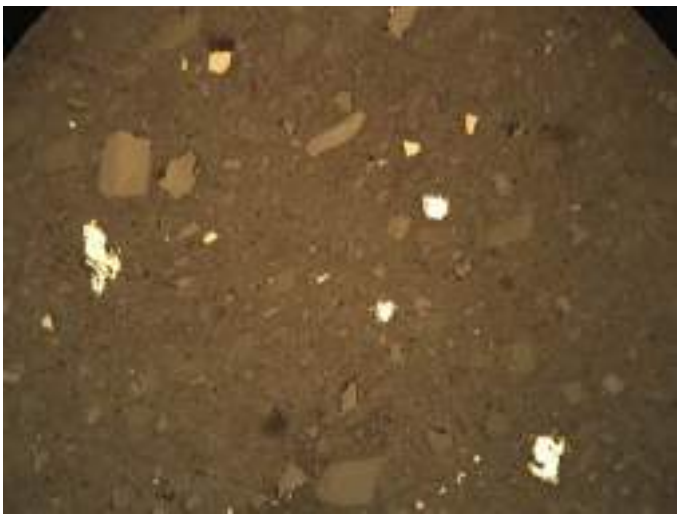

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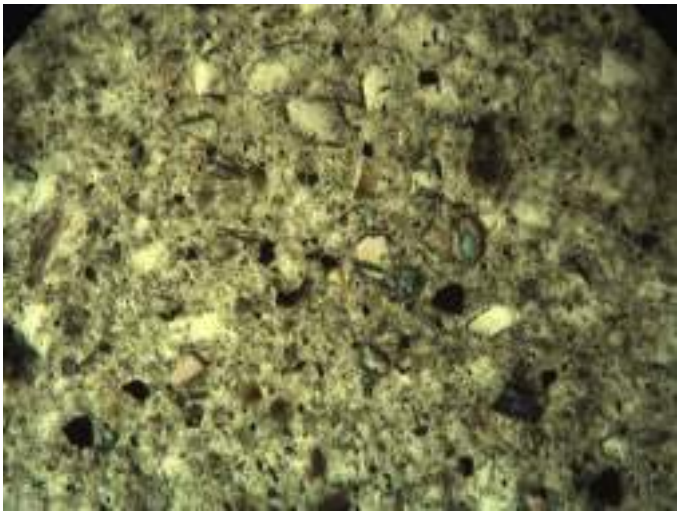
P1Solids Bulk Tailings: A) Cross Polarized Light, Field of View = ~0.8 mm, B) Plain Polarized Light, Field of View = ~0.8 mm, C) Reflected Light, Field of View = ~0.8 mm



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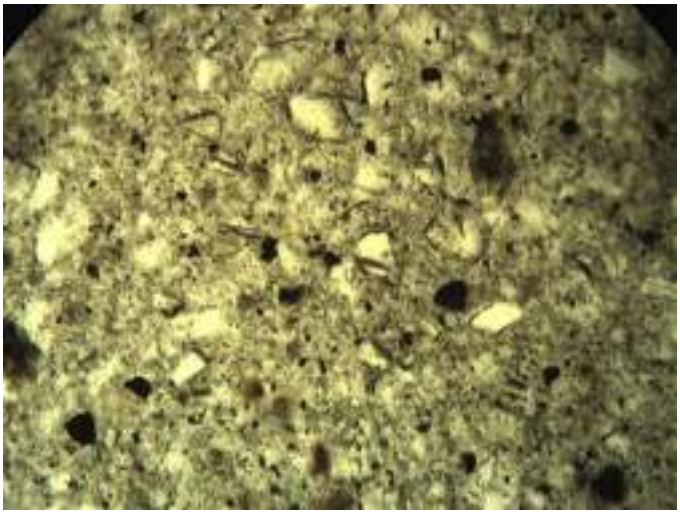
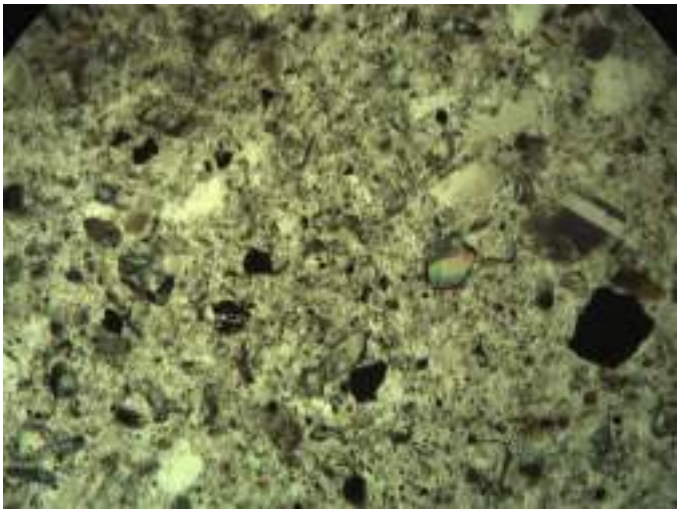


B

C

P2S Bulk Tailings: A) Cross Polarized Light, Field of View = ~0.8 mm, B) Plain Polarized Light, Field of View = ~0.8 mm, C) Reflected Light, Field of View = ~0.8 mm



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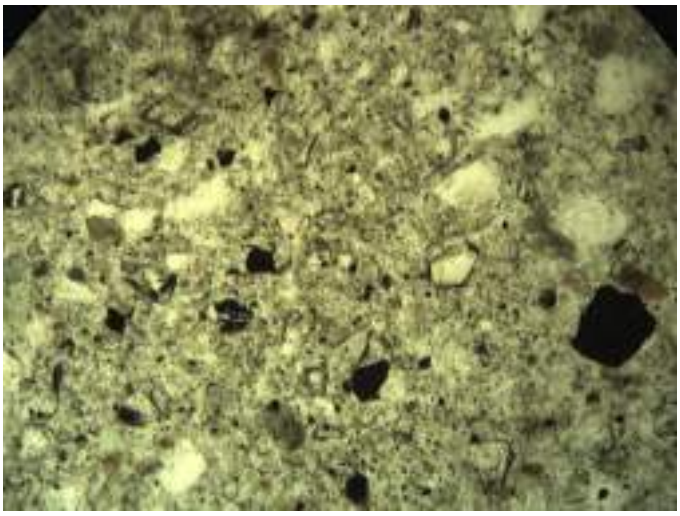


B

C

P3S Bulk Tailings: A) Cross Polarized Light, Field of View = ~0.8 mm, B) Plain Polarized Light, Field of View = ~0.8 mm, C) Reflected Light, Field of View = ~0.8 mm

Appendix B.3

Chemical Analysis

			ABA										
Sample Description	**Test ID**	**Type of Material**	**FIZZ**	**pH**	**NP**	**MPA**	**NNP**	**NP:MPA**	**Total S**	**S as Sulfide**	**S as SO_4**	**C**	**CO_2**
			Unity	**Unity**	**tCaCO3/1000t ore**	**tCaCO3/1000t ore**	**t CaCO3/ 1000 t ore**	**Unity**	**%**	**%**	**%**	**%**	**%**
P1 (CuSO4)	T1	Tailings with CuSO4	1	8.3	21	3.1	18	6.8	0.10	0.01		<0.05	<0.2
P1 (no CuSO4)	T2	Tailings without CuSO4	1	8.3	20	7.2	13	2.8	0.23	<0.01		<0.05	0.2
P2 (no CuSO4)	T3	Tailings without CuSO4	1	8.3	20	6.3	14	3.2	0.20	0.01		<0.05	0.2
P3 (CuSO4)	T4	Tailings with CuSO4	1	8.4	20	4.7	15	4.3	0.15	0.01		<0.05	<0.2
Parcel 2 P2S +100 mesh	T5	Tailings without CuSO4	1	9.0	20	4.7	15	4.3	0.15	<0.01	0.10	<0.05	<0.2
Parcel 2 P2S -100 +200 mesh	T6	Tailings without CuSO4	1	9.2	22	5.3	17	4.1	0.17	0.01	0.12	<0.05	<0.2
Parcel 2 P2S -200 mesh	T7	Tailings without CuSO4	1	9.0	21	7.5	14	2.8	0.24	0.01	0.18	<0.05	0.2
Parcel 1-2 PISCS +100 mesh	T8	Tailings with CuSO4	1	9.1	22	3.4	19	6.4	0.11	0.02	0.07	<0.05	<0.2
Parcel 1-2 PISCS -100 +200 mesh	T9	Tailings with CuSO4	1	9.3	20	3.1	17	6.4	0.10	<0.01	0.07	<0.05	<0.2
Parcel 1-2 PISCS -200 mesh	T10	Tailings with CuSO4	1	8.8	22	2.8	19	7.8	0.09	0.02	0.05	<0.05	0.2
Parcel 3 P3S +100 mesh	T11	Tailings with CuSO4	1	9.2	18	3.4	15	5.2	0.11	0.02	0.08	<0.05	<0.2
Parcel 3 P3S -100 +200 mesh	T12	Tailings with CuSO4	1	9.3	21	4.4	17	4.8	0.14	0.01	0.10	<0.05	<0.2
Parcel 3 P3S -200 mesh	T13	Tailings with CuSO4	1	9.0	21	4.4	17	4.8	0.14	0.01	0.10	0.05	0.2

Notes:
Blank cells indicate no analysis was performed.
NP: Neutralization Potential as determined by the standard Sobek method.
MPA: Maximum Potential Acidity. Calculated from Total Sulfur.
NNP: Net Neutralization Potential.

	Metals by 4-Acid Digestion														
Sample Description	**Cu**	**Ni**	**S**	**S%ICP**	**Pt**	**Pd**	**Au**	**Co**	**Ag**	**Zn**	**Cd**	**Mo**	**Pb**	**As**	**Cr**
	%	**%**	**%**	**%**	**ppb**	**ppb**	**ppb**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**
P1 (CuSO4)	0.022	0.032	0.10	0.10	0.015	0.046	0.008	59	<0.5	95	<0.5	<1	8	<5	186
P1 (no CuSO4)	0.025	0.033	0.23	0.26	0.011	0.045	0.008	62	<0.5	97	<0.5	<1	8	<5	188
P2 (no CuSO4)	0.053	0.039	0.20	0.23	0.016	0.073	0.011	61	<0.5	93	<0.5	<1	8	<5	199
P3 (CuSO4)	0.042	0.037	0.15	0.17	0.010	0.059	0.014	60	0.5	92	<0.5	1	6	<5	175
Parcel 2 P2S +100 mesh	0.046	0.0284	0.15												
Parcel 2 P2S -100 +200 mesh	0.03	0.0306	0.17												
Parcel 2 P2S -200 mesh	0.0295	0.0314	0.24												
Parcel 1-2 PISCS +100 mesh	0.0303	0.0213	0.11												
Parcel 1-2 PISCS -100 +200 mesh	0.0176	0.0273	0.1												
Parcel 1-2 PISCS -200 mesh	0.01075	0.0267	0.09												
Parcel 3 P3S +100 mesh	0.0385	0.0225	0.11												
Parcel 3 P3S -100 +200 mesh	0.0249	0.0288	0.14												
Parcel 3 P3S -200 mesh	0.0139	0.0323	0.14												

Notes:
Blank cells indicate no analysis was performed
NP: Neutralization Potential as determined by t
MPA: Maximum Potential Acidity. Calculated fr
NNP: Net Neutralization Potential.

	Metals by 4-Acid Digestion															
Sample Description	**V**	**Ti**	**Al**	**Ca**	**Fe**	**K**	**Na**	**Mg**	**Mn**	**P**	**Ba**	**Be**	**Bi**	**Sb**	**Sr**	**W**
	ppm	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**
P1 (CuSO4)	137	0.82	10.05	6.13	9.09	0.47	1.97	5.29	1130	790	170	0.6	<2	<5	302	10
P1 (no CuSO4)	136	0.80	10.35	6.14	9.43	0.48	1.96	5.41	1140	800	170	0.6	<2	<5	300	<10
P2 (no CuSO4)	136	0.81	10.15	5.96	9.22	0.47	1.87	5.24	1125	750	160	0.6	<2	<5	282	<10
P3 (CuSO4)	131	0.76	10.45	6.14	9.03	0.5	1.94	5.27	1110	770	170	0.6	<2	<5	296	<10
Parcel 2 P2S +100 mesh																
Parcel 2 P2S -100 +200 mesh																
Parcel 2 P2S -200 mesh																
Parcel 1-2 PISCS +100 mesh																
Parcel 1-2 PISCS -100 +200 mesh																
Parcel 1-2 PISCS -200 mesh																
Parcel 3 P3S +100 mesh																
Parcel 3 P3S -100 +200 mesh																
Parcel 3 P3S -200 mesh																

Notes:
Blank cells indicate no analysis was performed
NP: Neutralization Potential as determined by t
MPA: Maximum Potential Acidity. Calculated fr
NNP: Net Neutralization Potential.

	Metals by Aqua Regia Digestion													
Sample Description	**Cu**	**Ni**	**S**	**Co**	**Ag**	**Zn**	**Cd**	**Mo**	**Pb**	**As**	**Cr**	**V**	**Ti**	**Al**
	%	**%**	**%**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**%**	**%**
P1 (CuSO4)	0.0206	0.0267	0.10	46	0.2	69	<0.5	<1	5	<2	76	44	0.16	3.72
P1 (no CuSO4)	0.0238	0.0282	0.26	46	0.2	68	<0.5	1	2	7	77	44	0.16	3.61
P2 (no CuSO4)	0.0474	0.034	0.23	48	0.3	67	<0.5	1	2	4	82	46	0.16	3.55
P3 (CuSO4)	0.0377	0.032	0.16	47	0.3	65	<0.5	1	<2	4	79	45	0.18	3.89
Parcel 2 P2S +100 mesh			0.15	47.4	0.31	57	0.08	1.6	3.2	1.1	109	34	0.116	4.04
Parcel 2 P2S -100 +200 mesh			0.16	57	0.24	71	0.06	1.85	2.8	1.4	130	39	0.142	3.45
Parcel 2 P2S -200 mesh			0.22	56.3	0.23	70	0.07	1.41	3.3	3.3	120	49	0.18	4.05
Parcel 1-2 PISCS +100 mesh			0.1	38.4	0.24	52	0.11	2.82	6.9	1.8	167	29	0.106	4.71
Parcel 1-2 PISCS -100 +200 mesh			0.1	52.2	0.18	70	0.06	1.79	3.1	2.9	135	39	0.151	3.91
Parcel 1-2 PISCS -200 mesh			0.09	52.8	0.16	69	0.06	1.42	2.8	2.2	115	49	0.184	4.18
Parcel 3 P3S +100 mesh			0.11	35.3	0.29	45	0.08	2.09	4.3	1.8	133	33	0.114	5.15
Parcel 3 P3S -100 +200 mesh			0.12	53.8	0.25	66	0.07	2.03	3.1	1.1	134	36	0.13	3.85
Parcel 3 P3S -200 mesh			0.15	58.3	0.22	79	0.09	1.63	7	2.8	135	53	0.21	4.47

Notes:
Blank cells indicate no analysis was performed
NP: Neutralization Potential as determined by t
MPA: Maximum Potential Acidity. Calculated fr
NNP: Net Neutralization Potential.

	Metals by Aqua Regia Digestion														
Sample Description	**Ca**	**Fe**	**K**	**Na**	**Mg**	**Mn**	**P**	**B**	**Ba**	**Be**	**Bi**	**Ga**	**Hg**	**La**	**Sb**
	%	**%**	**%**	**%**	**%**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**
P1 (CuSO4)	2.12	6.59	0.21	0.52	3.73	724	660	<10	50	<0.5	<2	<10	<1	10	<2
P1 (no CuSO4)	2.06	6.84	0.2	0.51	3.73	728	660	<10	50	<0.5	<2	10	2	10	<2
P2 (no CuSO4)	2.03	6.87	0.2	0.5	3.73	745	680	10	50	<0.5	<2	10	1	10	<2
P3 (CuSO4)	2.22	6.68	0.22	0.55	3.69	731	700	<10	60	<0.5	<2	10	1	10	2
Parcel 2 P2S +100 mesh	2.17	5.34	0.2	0.56	3.15	612	290	<10	50	0.21	0.11	7.81	0.01	3.8	0.12
Parcel 2 P2S -100 +200 mesh	1.88	6.4	0.2	0.49	3.78	732	390	10	50	0.19	0.12	7.24	0.02	4.3	0.14
Parcel 2 P2S -200 mesh	2.25	6.78	0.22	0.56	3.83	753	810	10	60	0.23	0.17	8.71	0.01	7.9	0.18
Parcel 1-2 PISCS +100 mesh	2.55	4.51	0.21	0.67	2.72	518	280	<10	60	0.23	0.12	9.12	0.05	4.1	0.25
Parcel 1-2 PISCS -100 +200 mesh	2.14	6.23	0.21	0.55	3.76	714	400	<10	60	0.2	0.15	7.77	0.01	4.5	0.17
Parcel 1-2 PISCS -200 mesh	2.33	6.61	0.23	0.57	3.92	748	880	<10	60	0.21	0.11	8.64	0.01	8.3	0.17
Parcel 3 P3S +100 mesh	2.77	4.1	0.26	0.74	2.44	468	260	<10	70	0.26	0.16	10.05	<0.01	4.3	0.18
Parcel 3 P3S -100 +200 mesh	2.08	5.88	0.22	0.53	3.53	678	370	<10	60	0.2	0.12	8.06	0.01	4.5	0.11
Parcel 3 P3S -200 mesh	2.47	7.18	0.24	0.61	4.16	808	870	<10	60	0.24	0.1	9.3	0.04	8.6	0.22

Notes:
Blank cells indicate no analysis was performed
NP: Neutralization Potential as determined by t
MPA: Maximum Potential Acidity. Calculated fr
NNP: Net Neutralization Potential.

	Whole Rock Oxides															
Sample Description	**SiO_2**	**Al_2O_3**	**TiO_2**	**Fe_2O_3**	**FeO**	**CaO**	**MgO**	**MnO**	**Na_2O**	**K_2O**	**P_2O_5**	**Cr_2O_3**	**BaO**	**SrO**	**LOI**	**Total**
	%	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**%**
P1 (CuSO4)	45.8	17.7	1.36	13.7	10.85	8.99	8.79	0.09	2.55	0.62	0.13	0.03	0.02	0.03	0.21	100.0
P1 (no CuSO4)	45.4	17.5	1.35	14.0	11.1	8.86	8.73	0.08	2.52	0.60	0.10	0.04	0.02	0.03	0.66	99.8
P2 (no CuSO4)	46.1	18.0	1.32	13.9	11.25	8.66	8.68	0.14	2.43	0.58	0.11	0.04	0.02	0.03	0.67	100.5
P3 (CuSO4)	46.3	17.9	1.19	13.1	10.55	8.57	8.41	0.13	2.42	0.59	0.12	0.03	0.02	0.03	0.79	99.6
Parcel 2 P2S +100 mesh																
Parcel 2 P2S -100 +200 mesh																
Parcel 2 P2S -200 mesh																
Parcel 1-2 PISCS +100 mesh																
Parcel 1-2 PISCS -100 +200 mesh																
Parcel 1-2 PISCS -200 mesh																
Parcel 3 P3S +100 mesh																
Parcel 3 P3S -100 +200 mesh																
Parcel 3 P3S -200 mesh																

Notes:
Blank cells indicate no analysis was performed
NP: Neutralization Potential as determined by t
MPA: Maximum Potential Acidity. Calculated fr
NNP: Net Neutralization Potential.

Appendix C

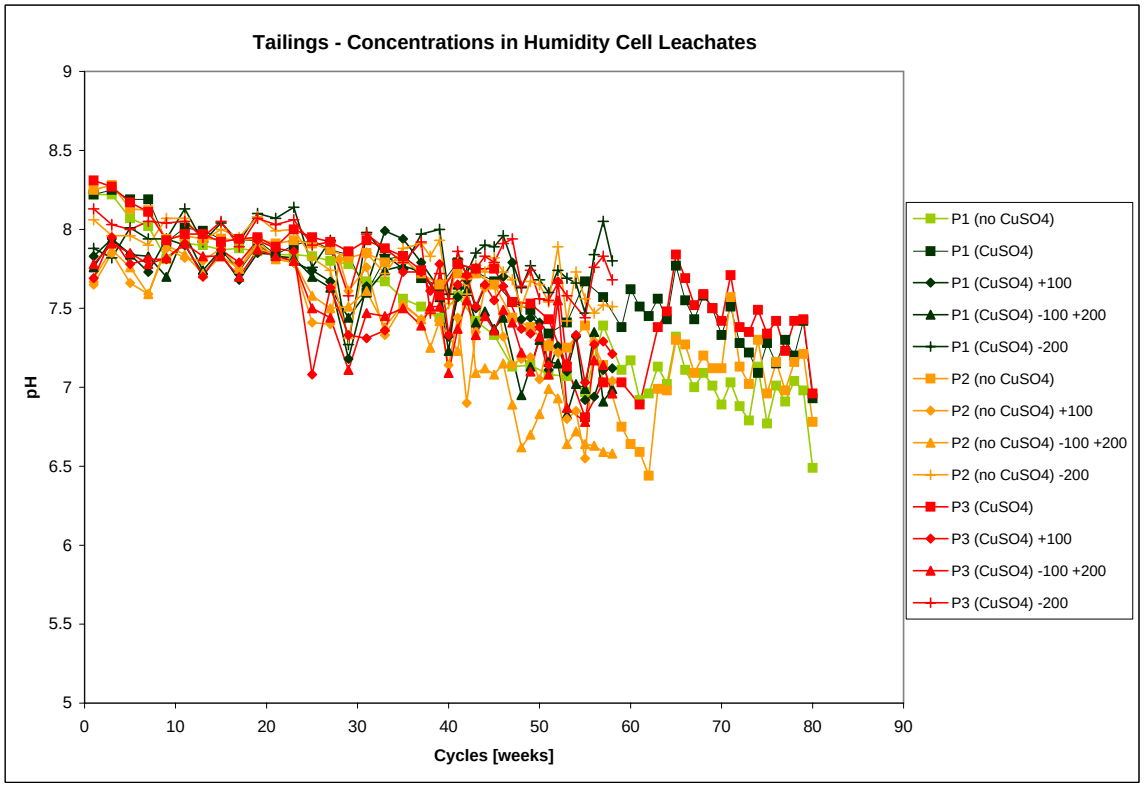
Kinetic Test Results

Appendix C.1

ASTM Cells (Loadings)

Appendix C.1
ASTM Cells (Loadings)

Graph C.1.1



loadings_Tailings_format.xls

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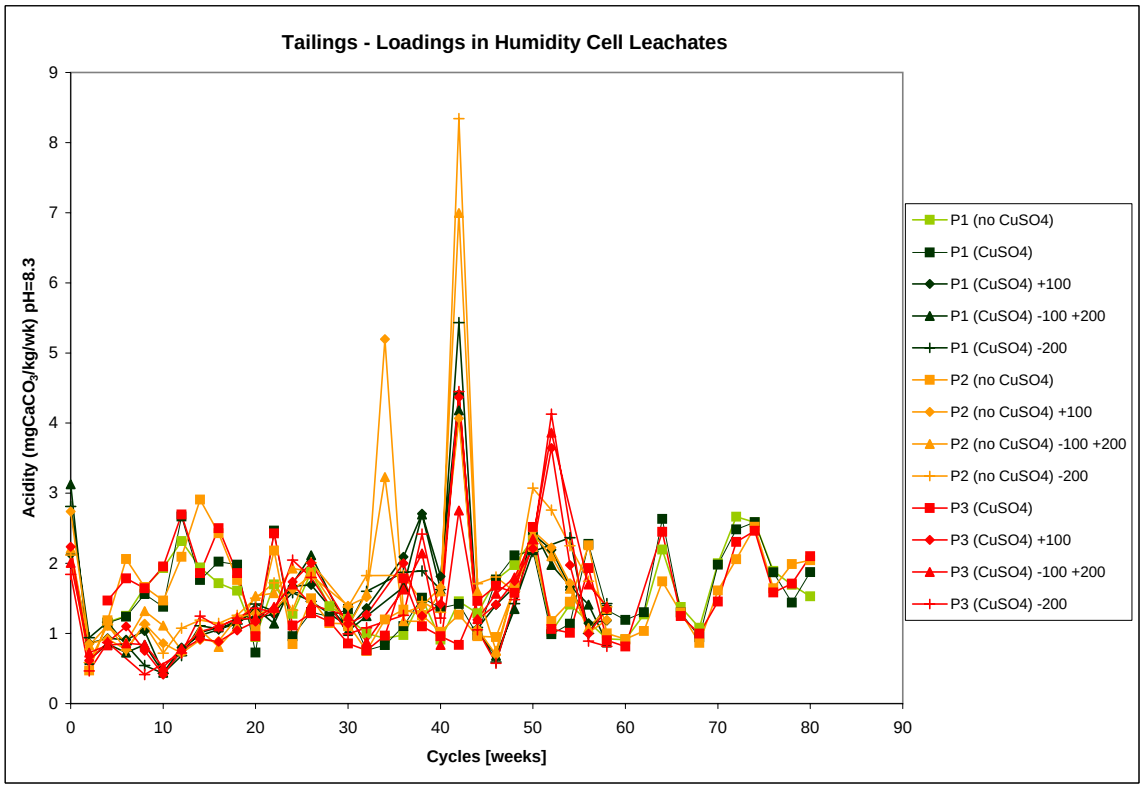
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.2



loadings_Tailings_format.xls

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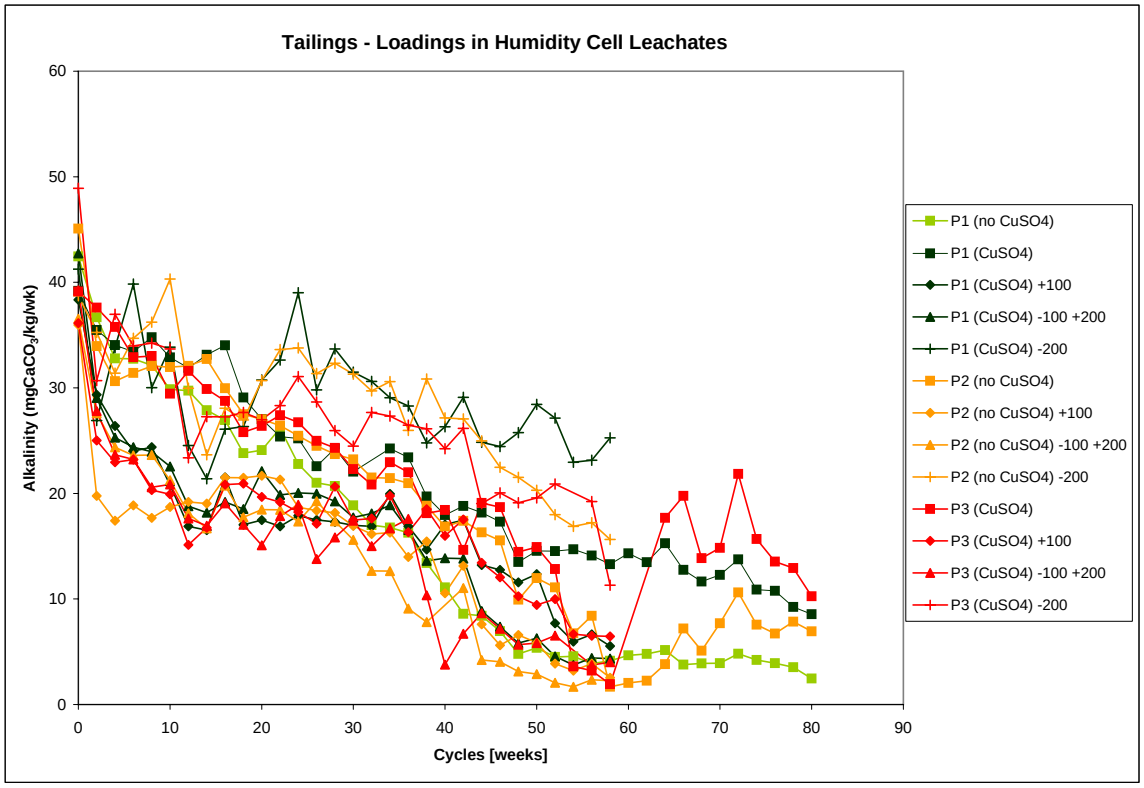
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.3



loadings_Tailings_format.xls

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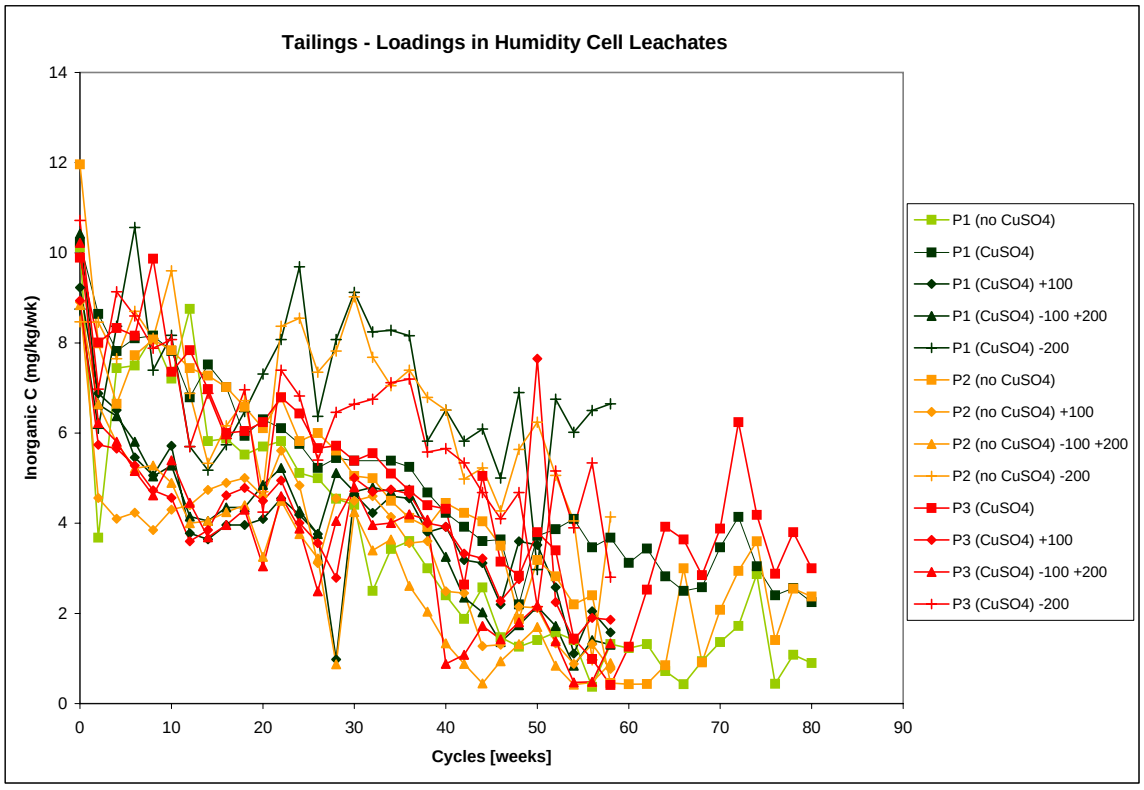
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.4



loadings_Tailings_format.xls

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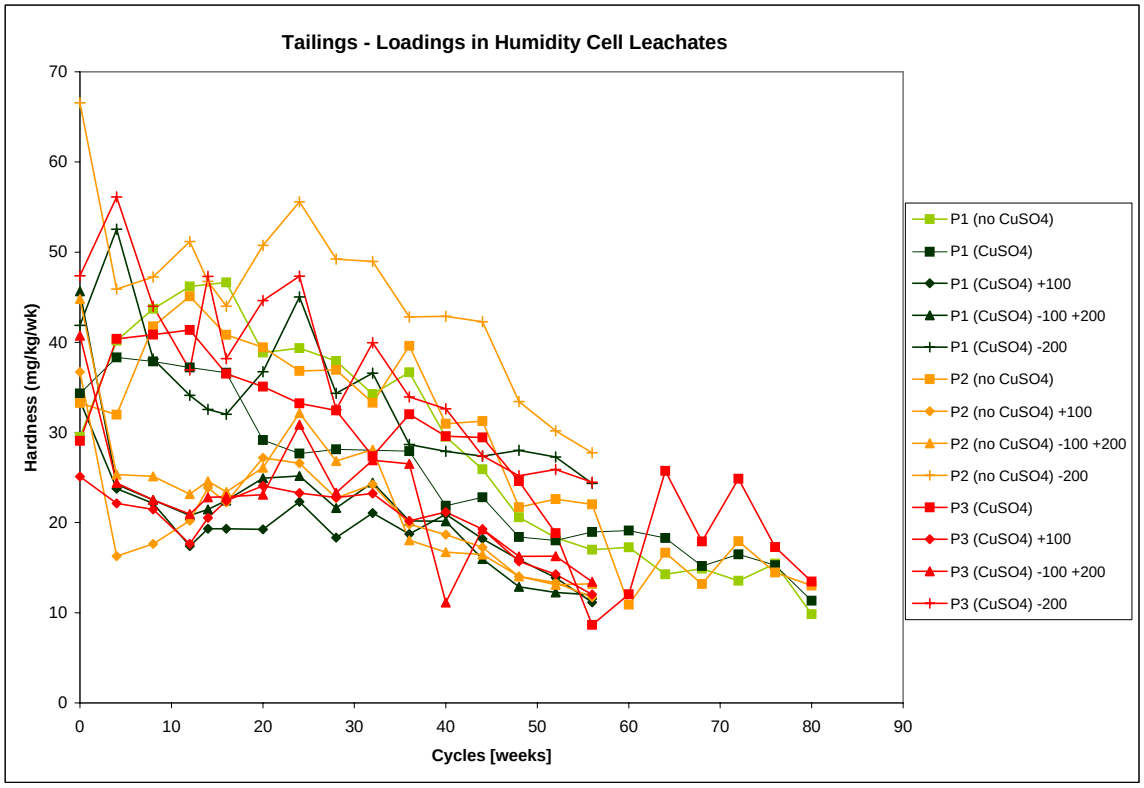
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.5



loadings_Tailings_format.xls

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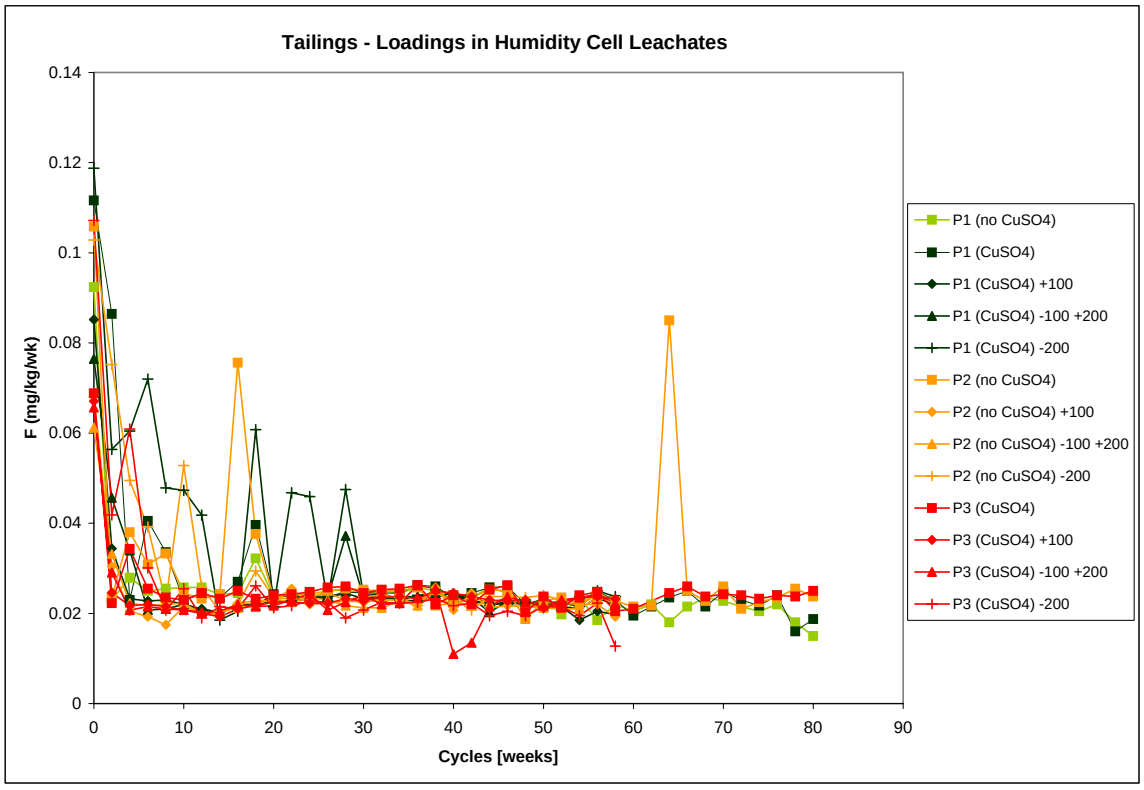
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.6



loadings_Tailings_format.xls

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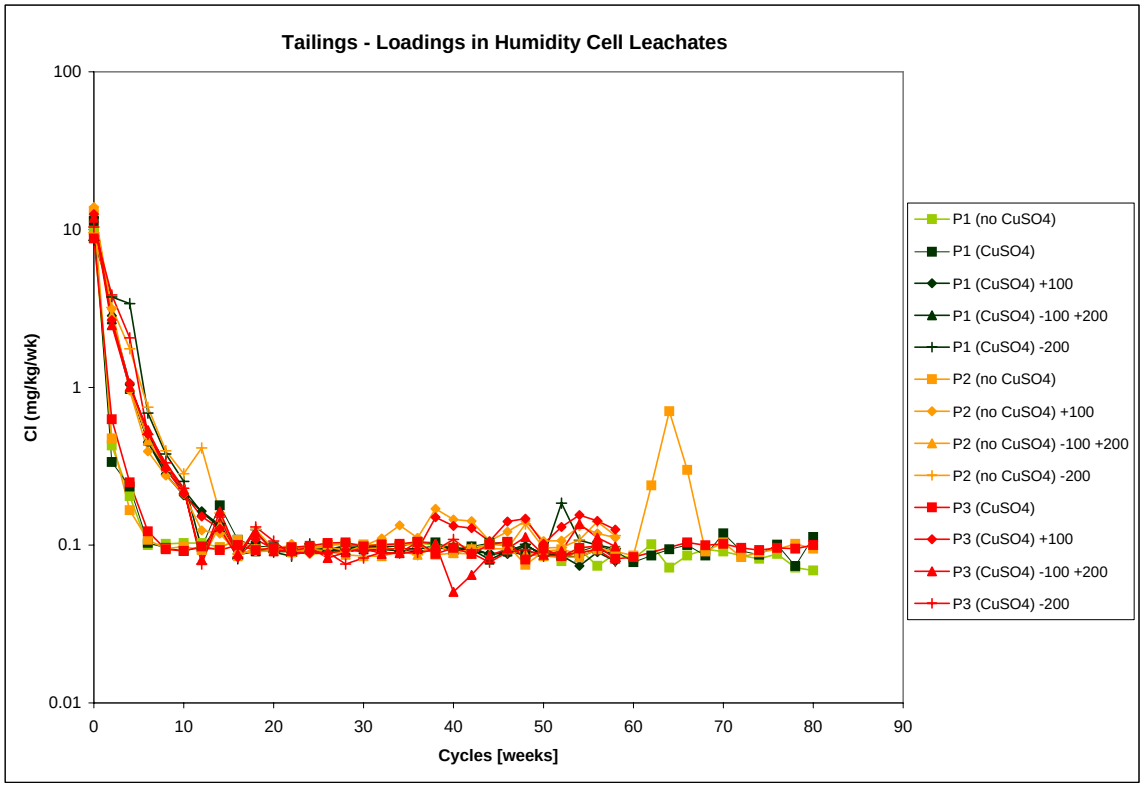
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.7



loadings_Tailings_format.xls

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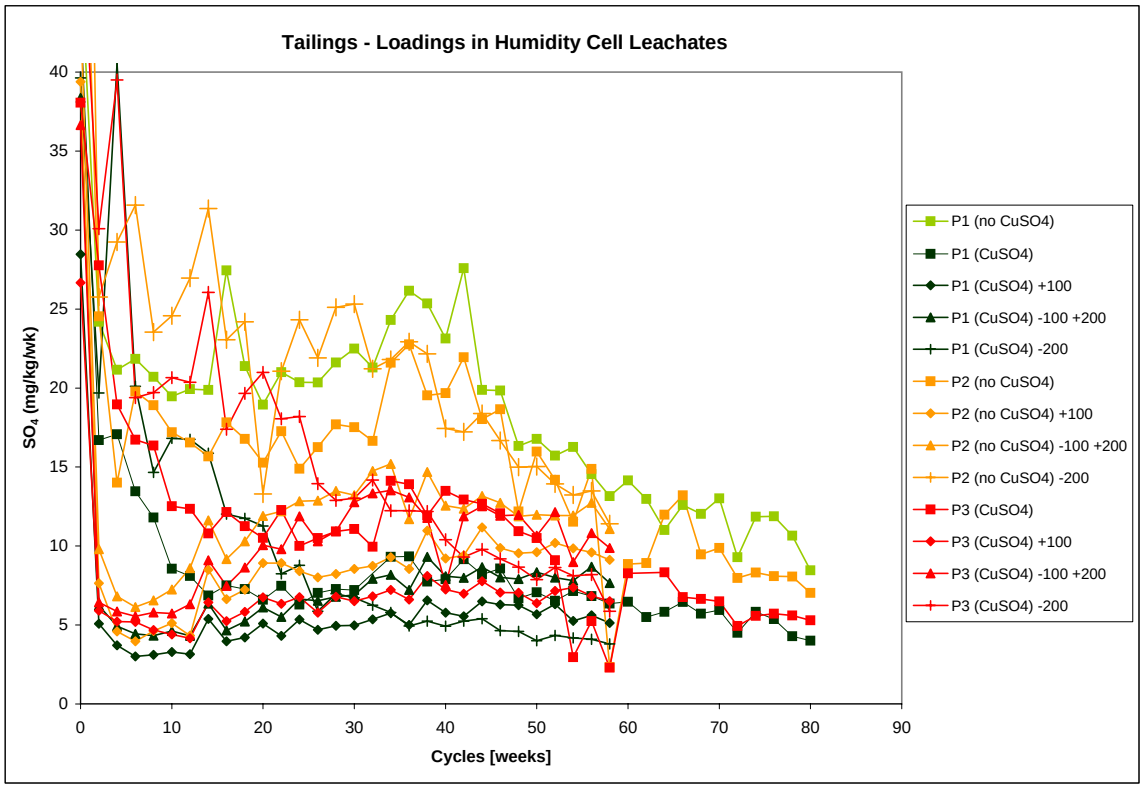
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.8



loadings_Tailings_format.xls

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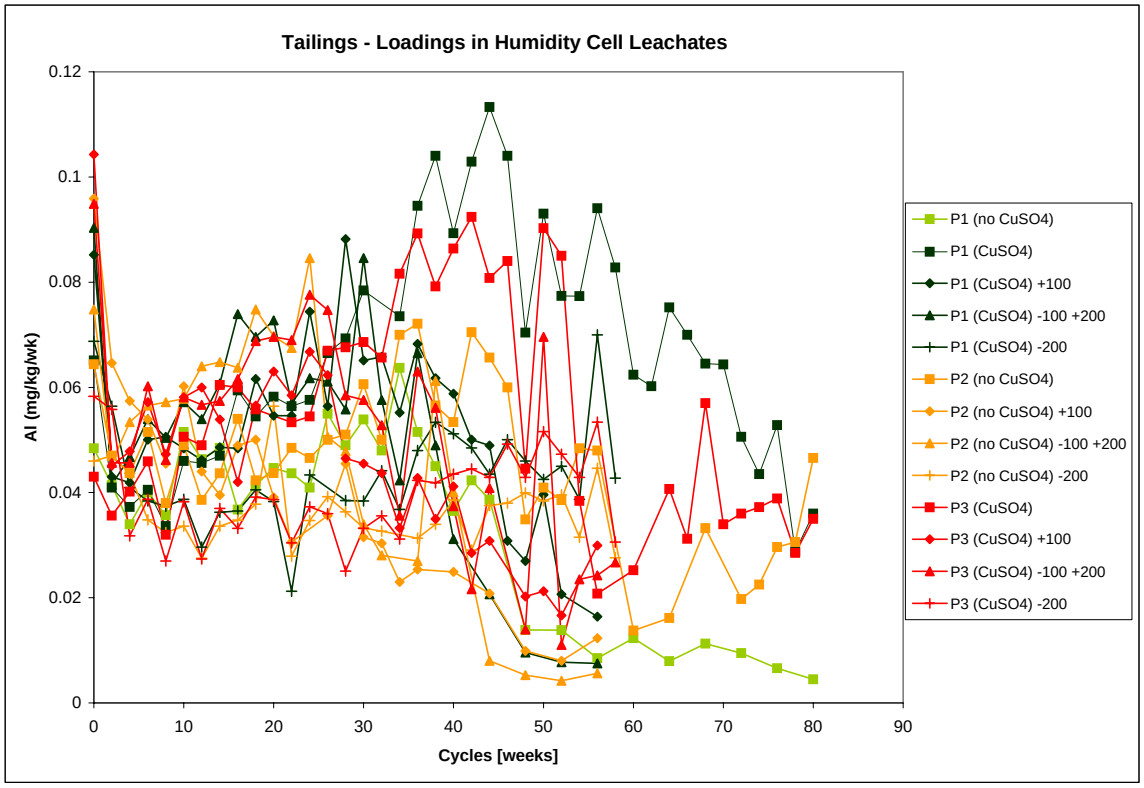
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.9



loadings_Tailings_format.xls

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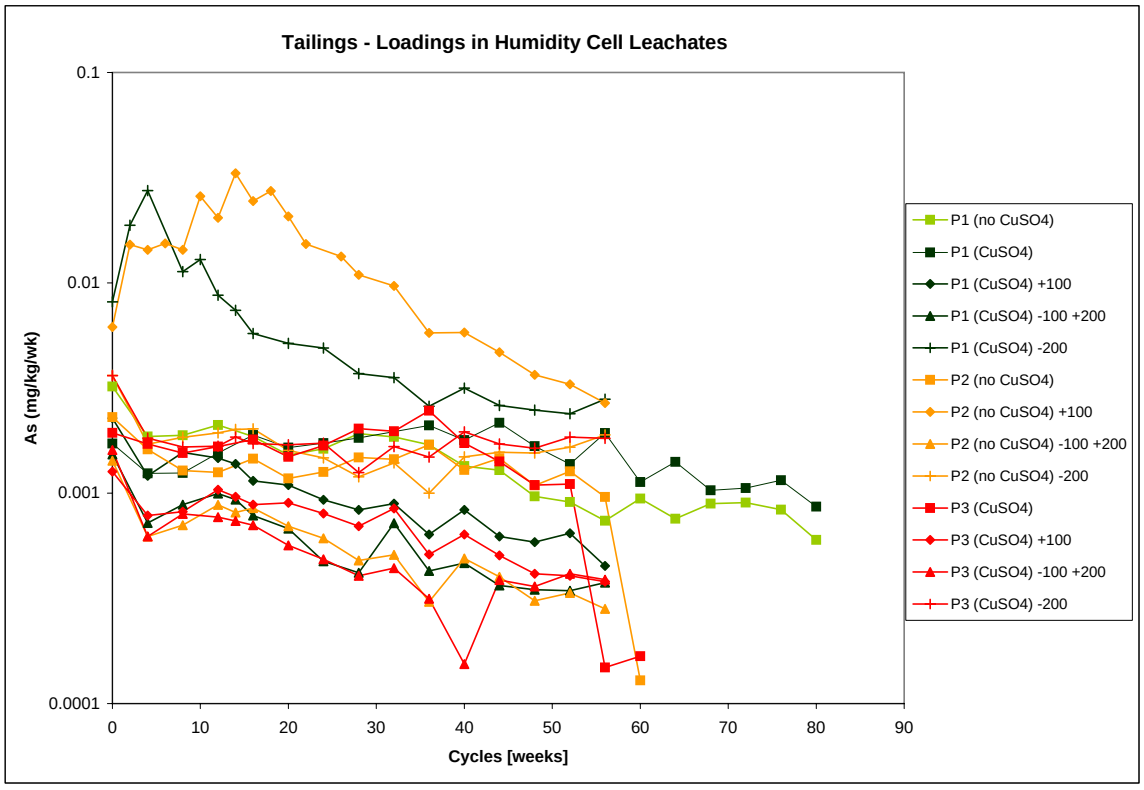
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.10



loadings_Tailings_format.xls

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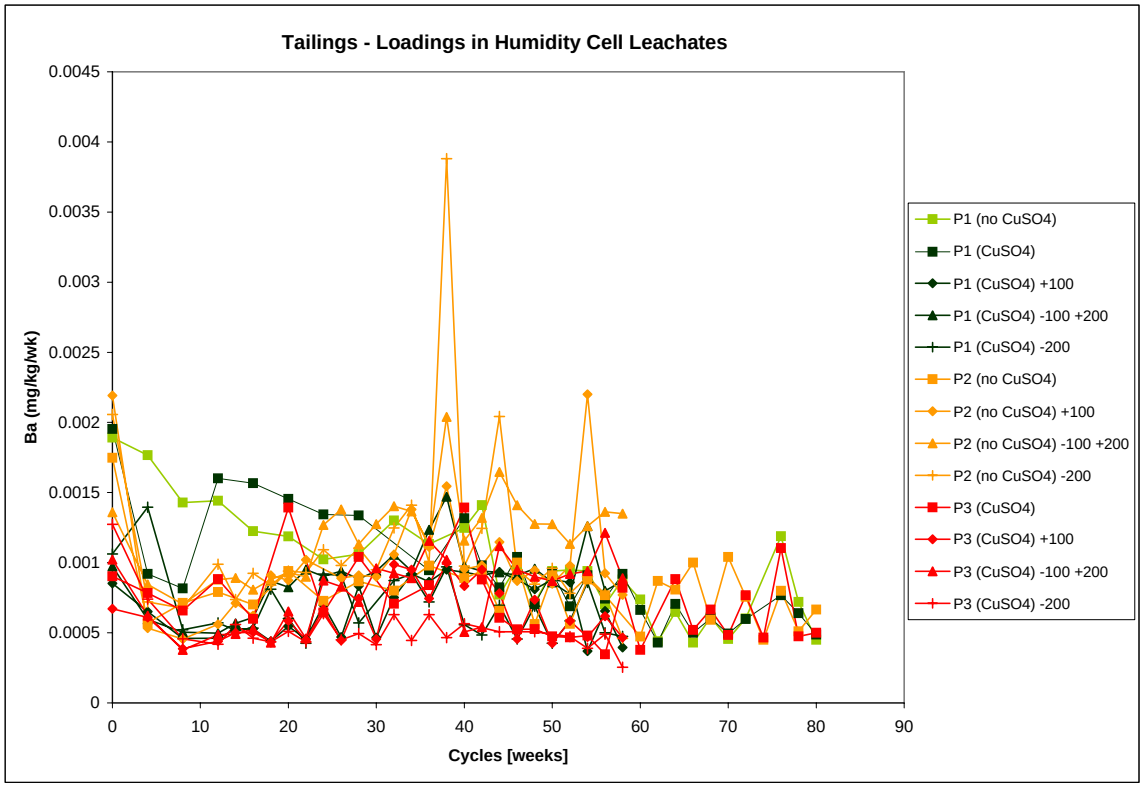
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.11



loadings_Tailings_format.xls

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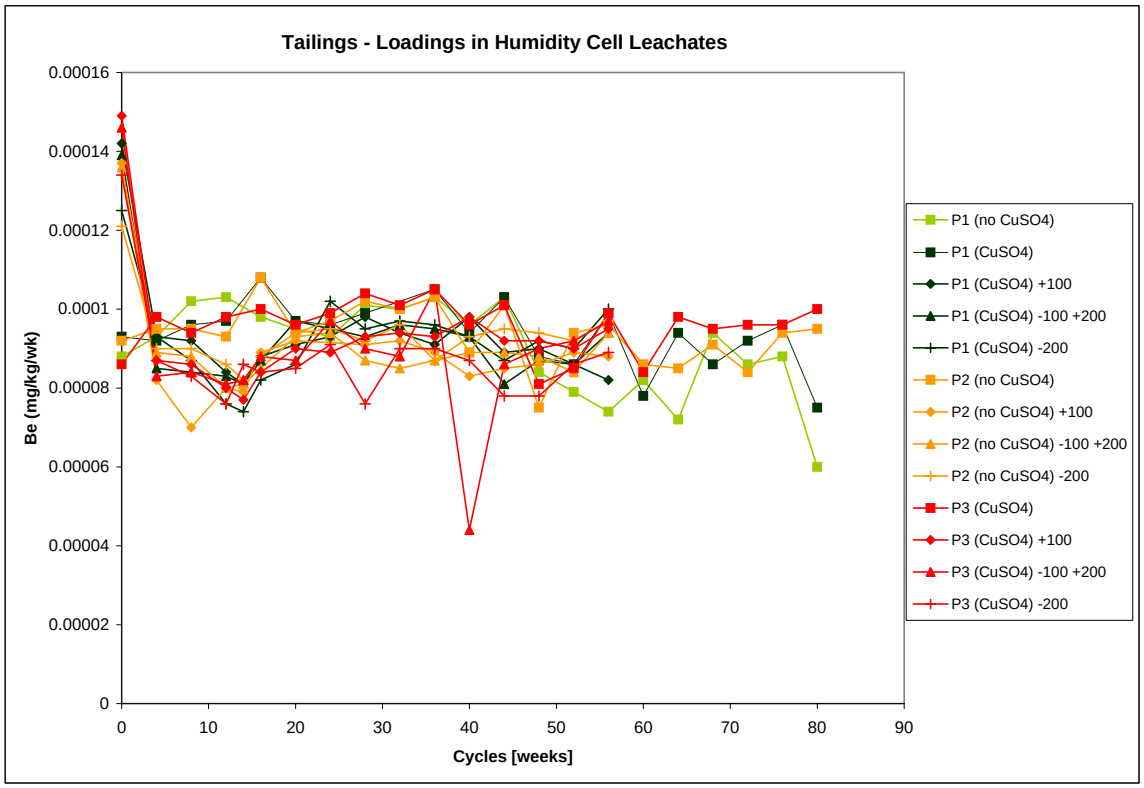
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.12



loadings_Tailings_format.xls

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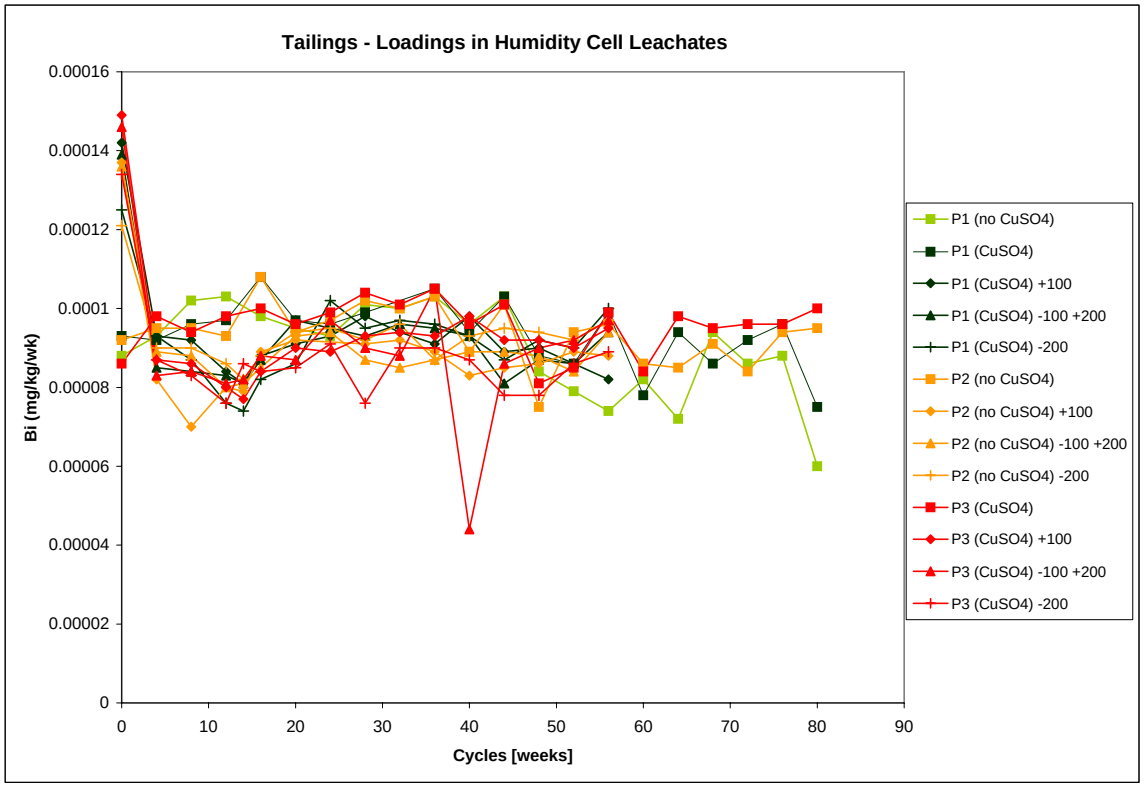
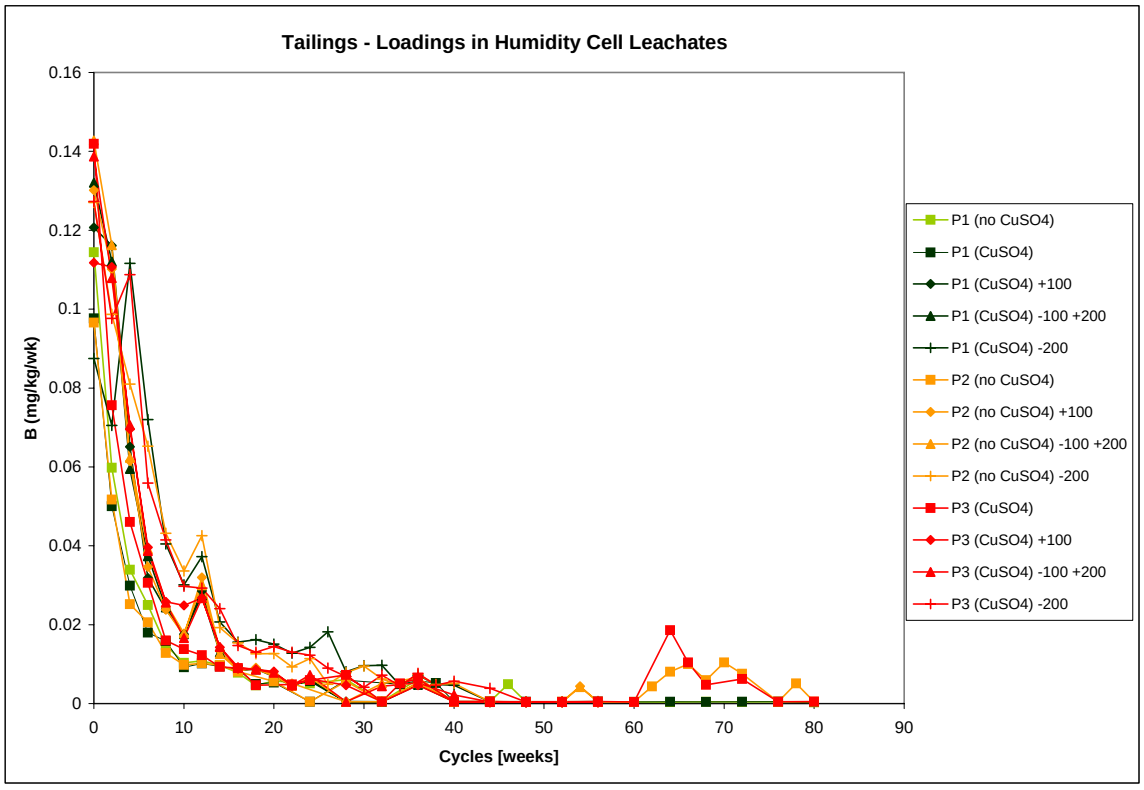
Graph C.1.13

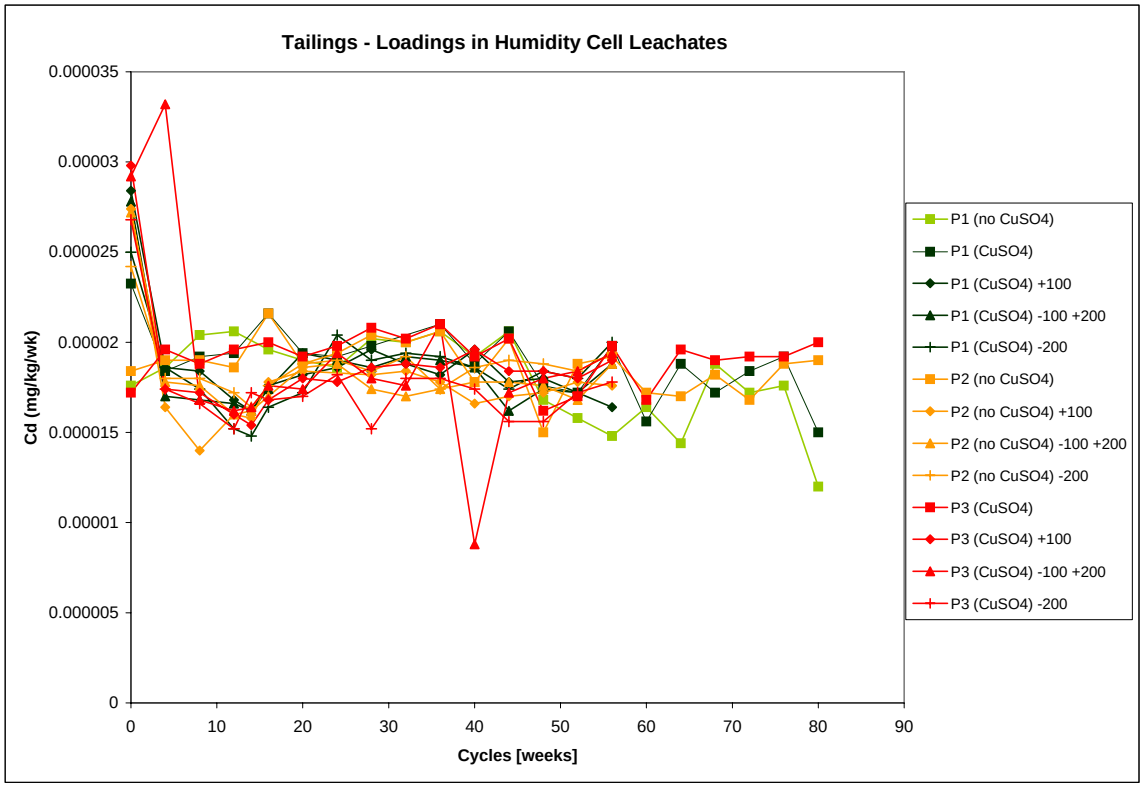


Graph C.1.14



Appendix C.1
ASTM Cells (Loadings)

Graph C.1.15



loadings_Tailings_format.xls

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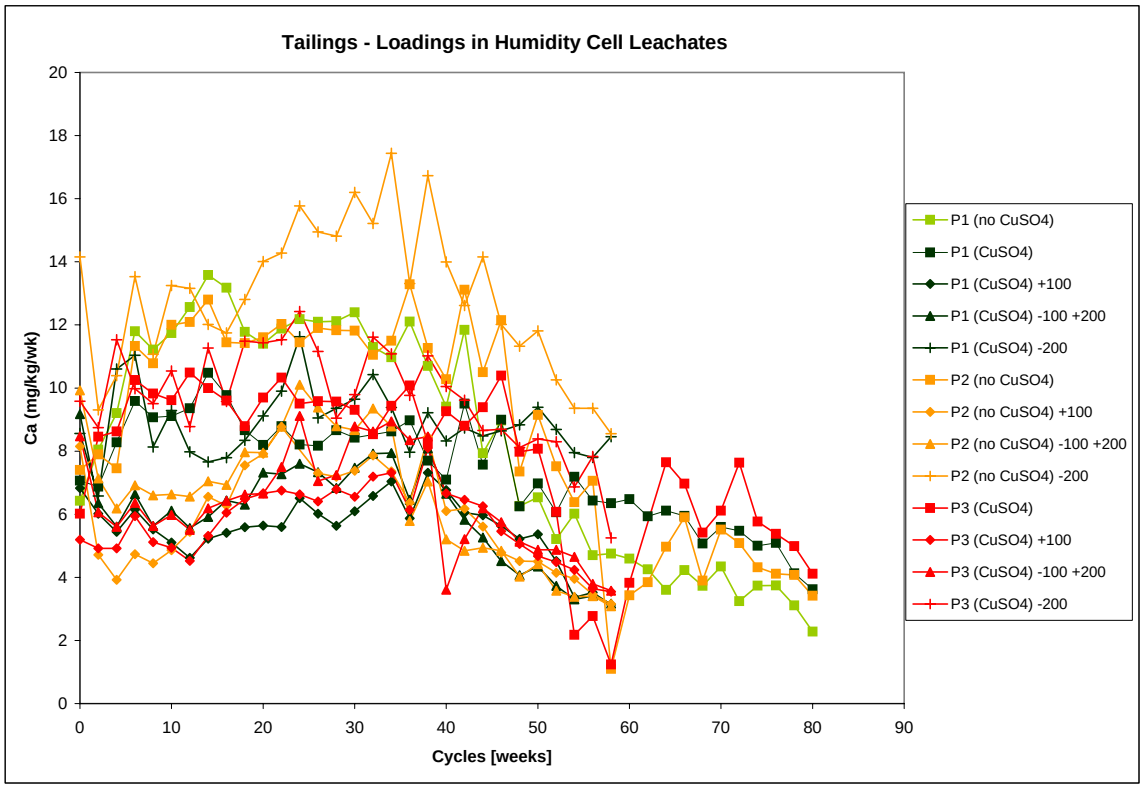
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.16



loadings_Tailings_format.xls

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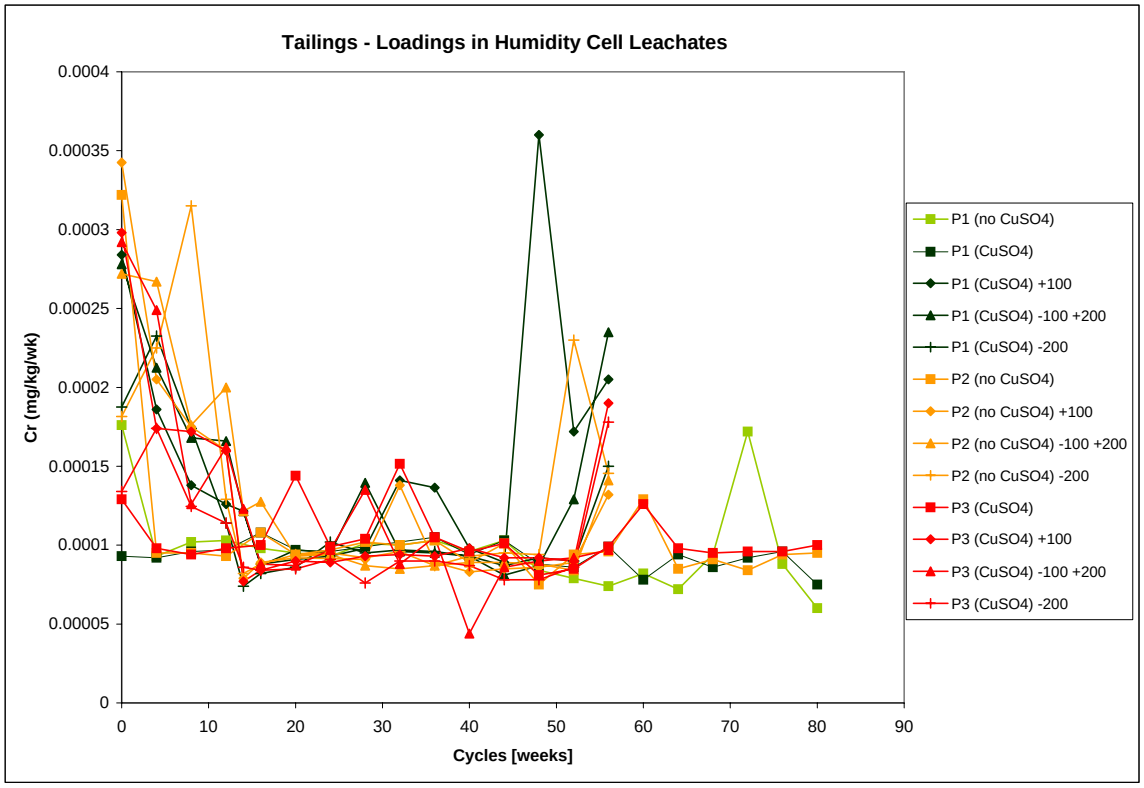
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.17



loadings_Tailings_format.xls

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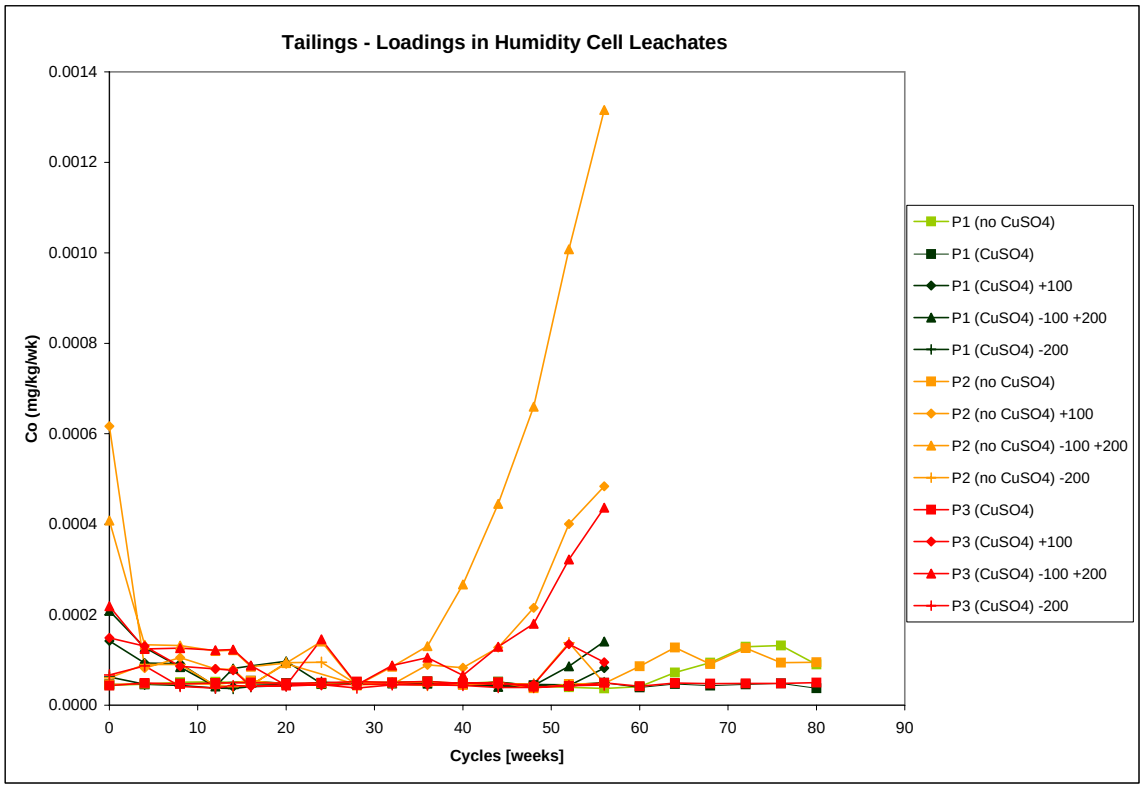
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.18



loadings_Tailings_format.xls

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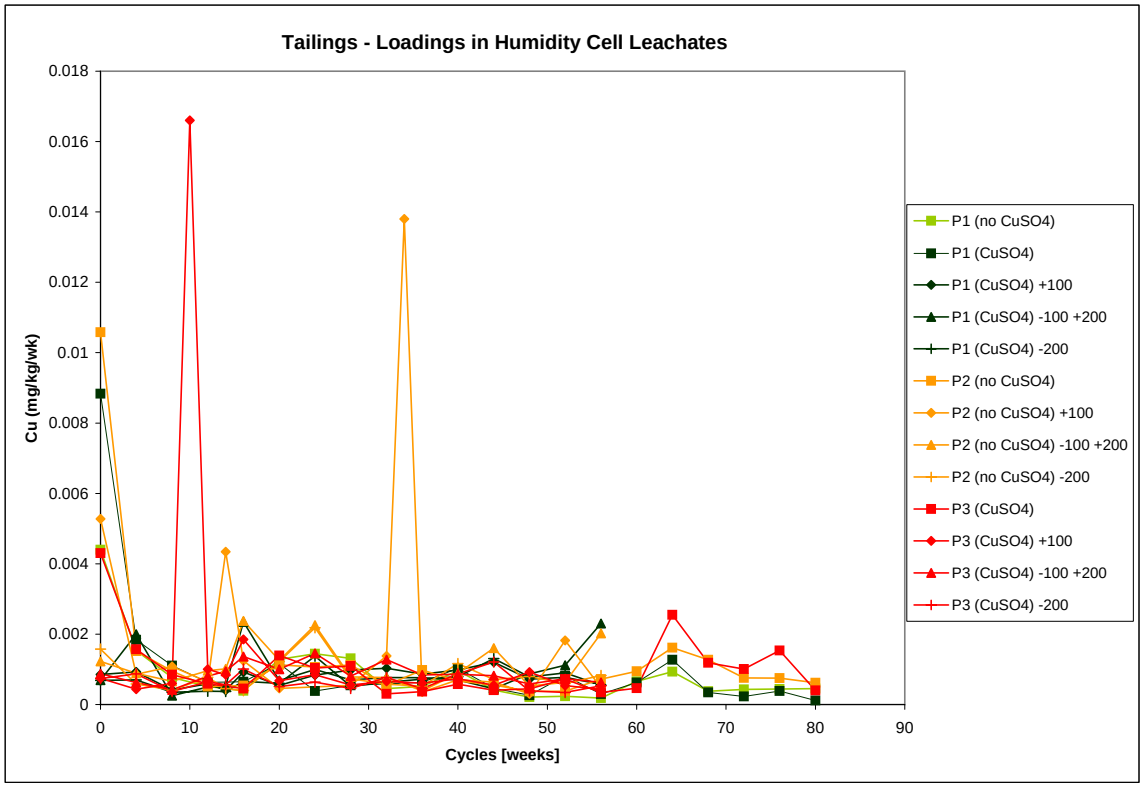
Graph C.1.19



loadings_Tailings_format.xls

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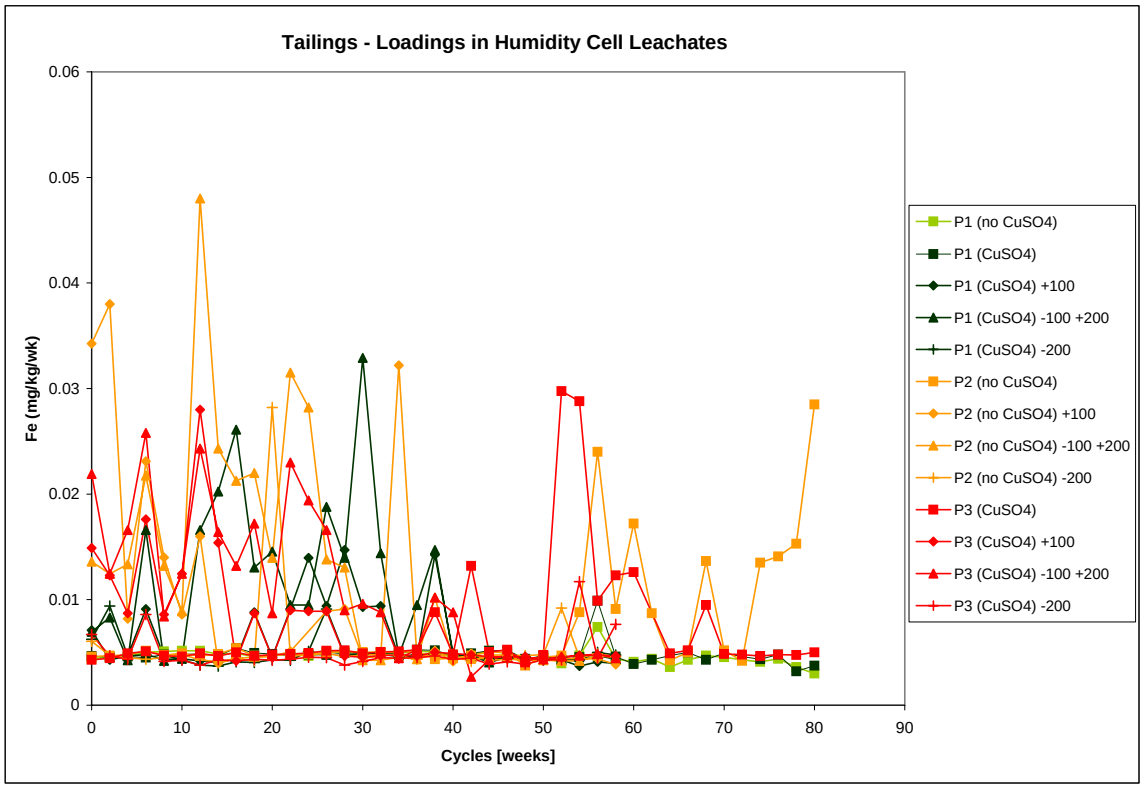
Graph C.1.20



loadings_Tailings_format.xls

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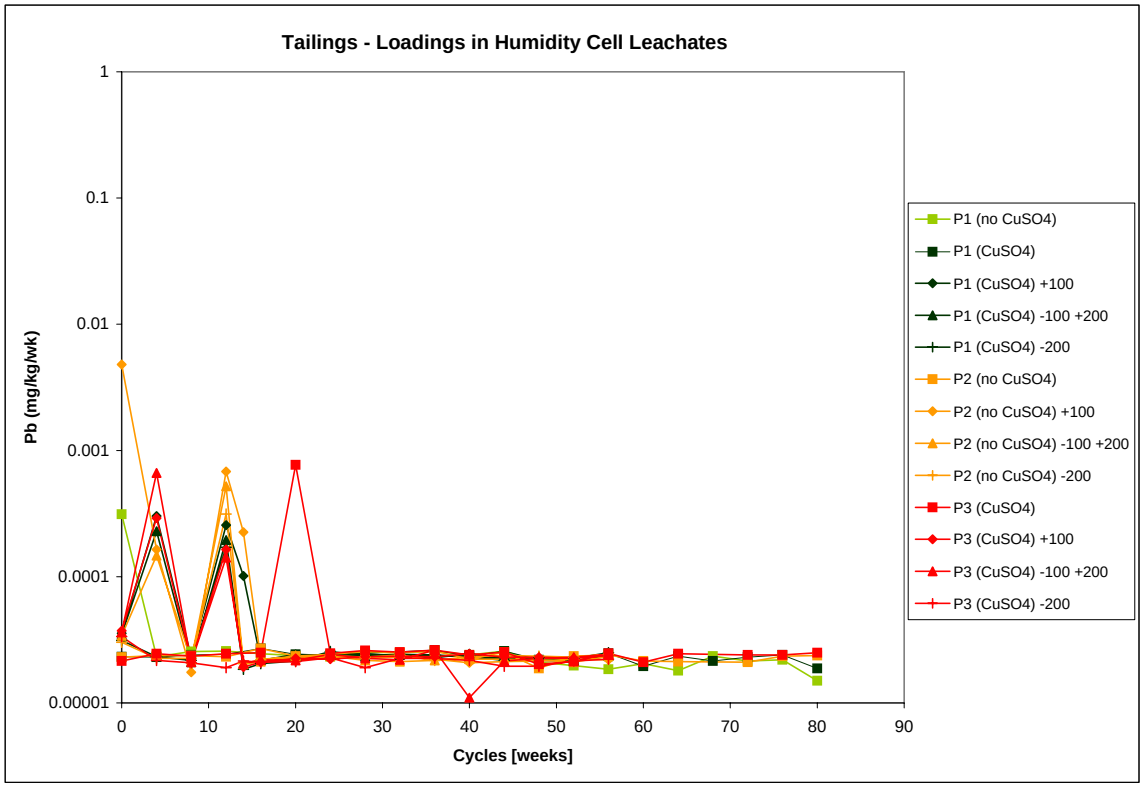
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.21



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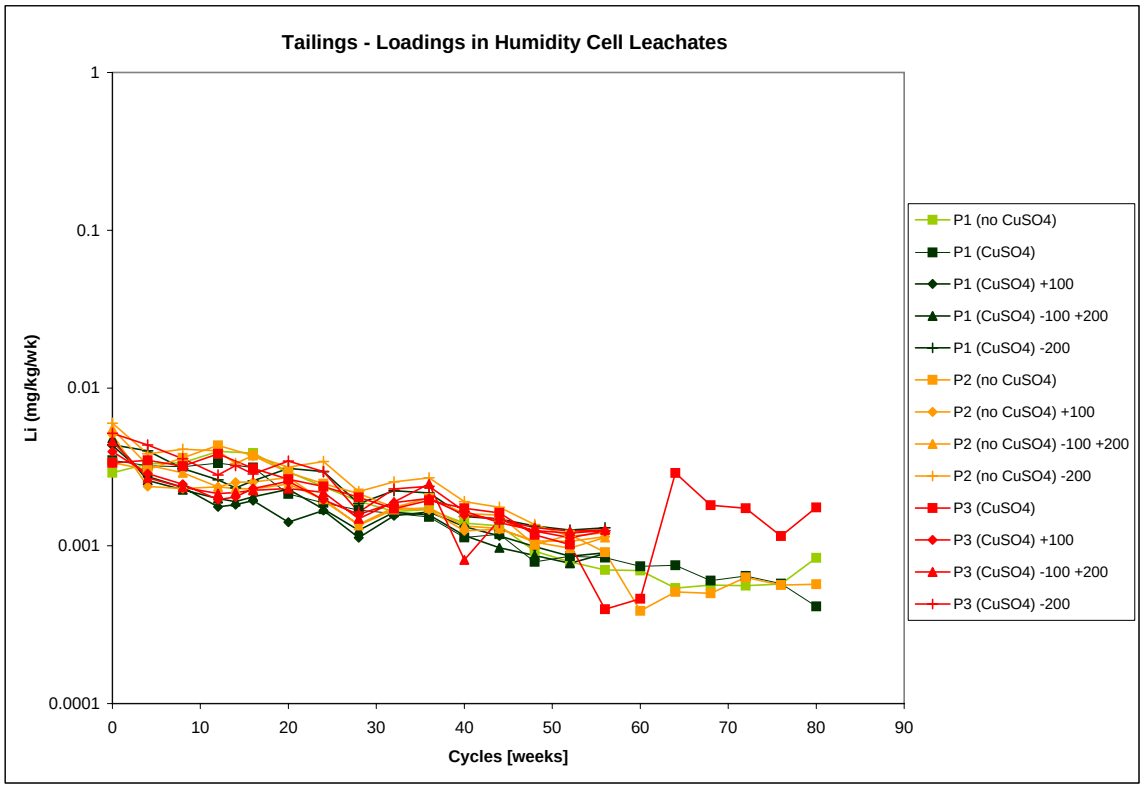
loadings_Tailings_format.xls

Appendix C.1
ASTM Cells (Loadings)

Graph C.1.22



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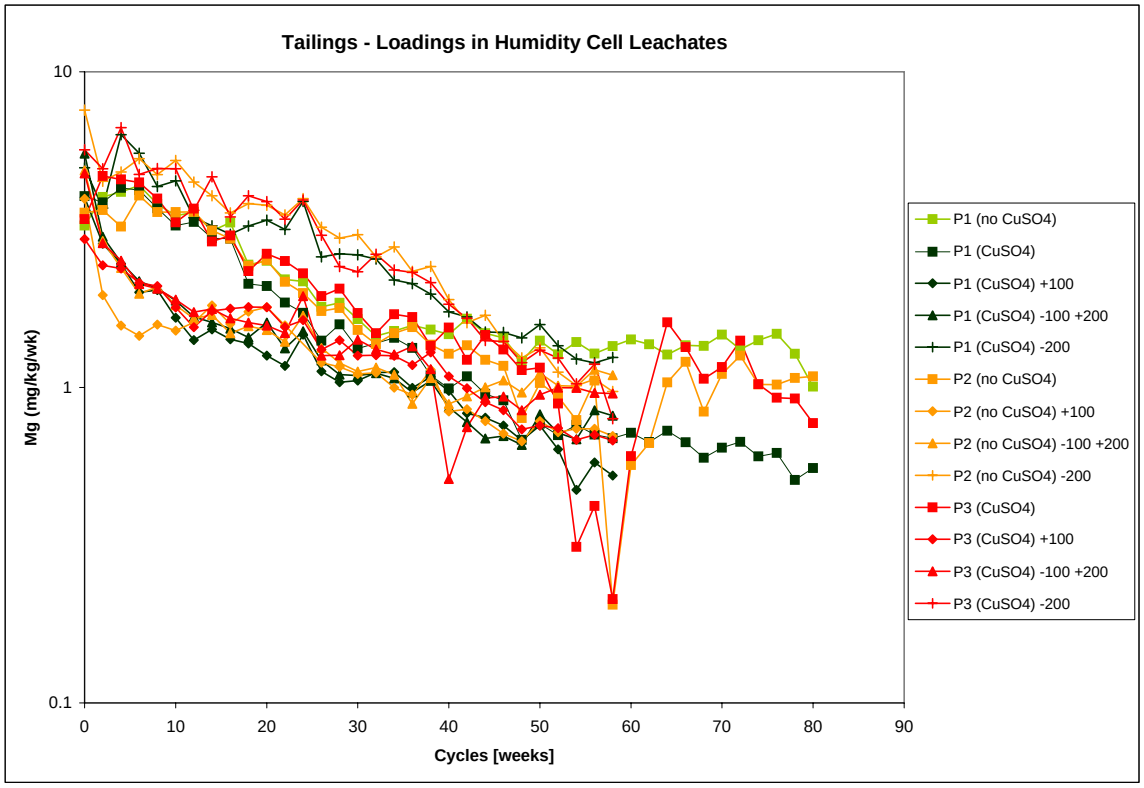
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.23



loadings_Tailings_format.xls

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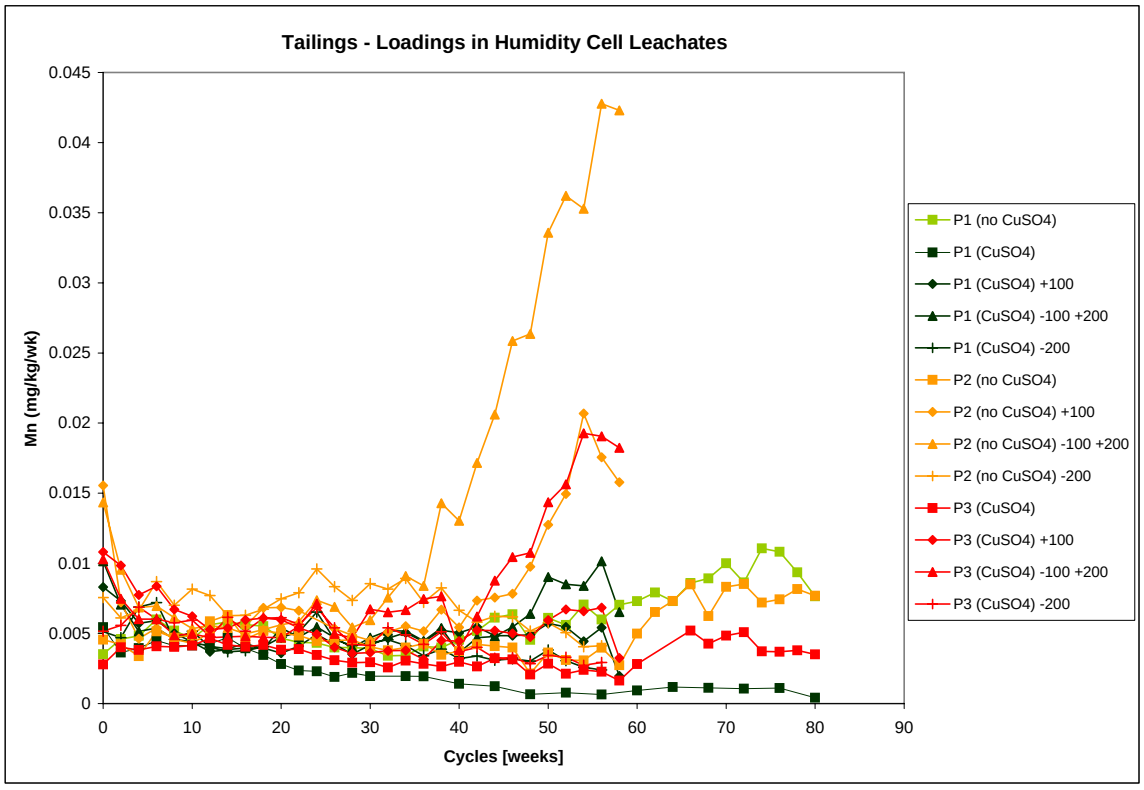
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.24



loadings_Tailings_format.xls

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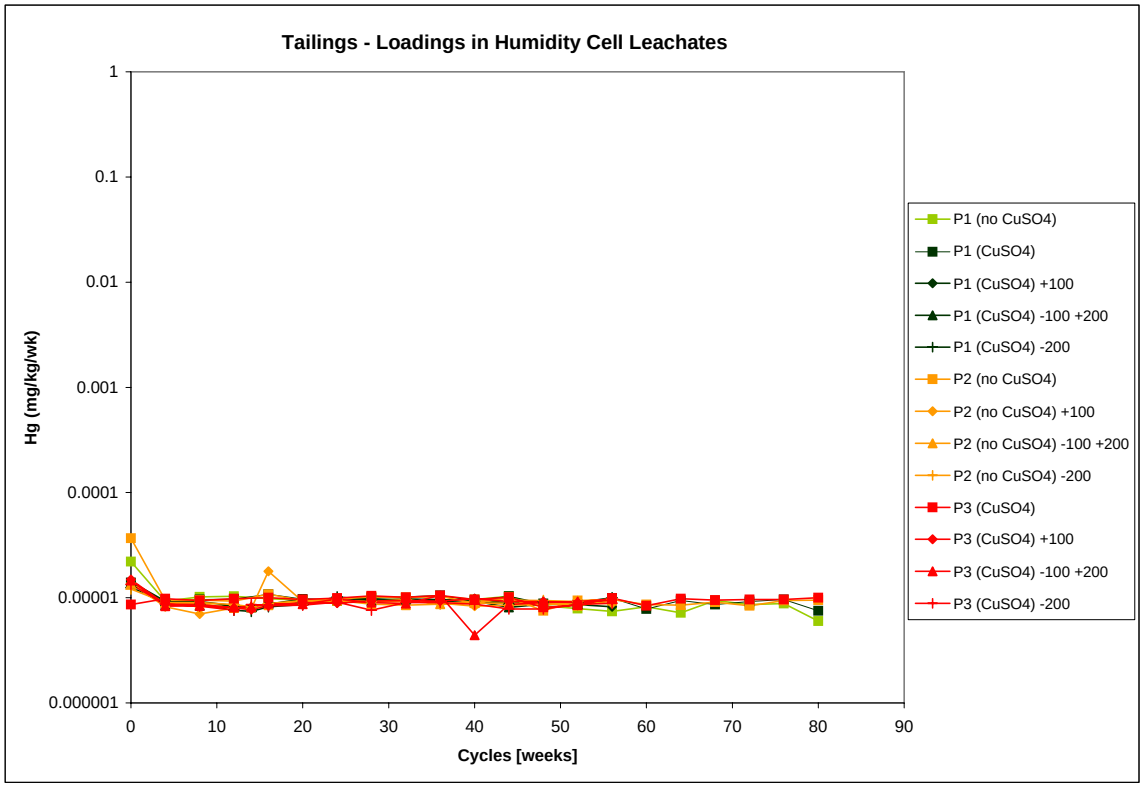
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.25



loadings_Tailings_format.xls

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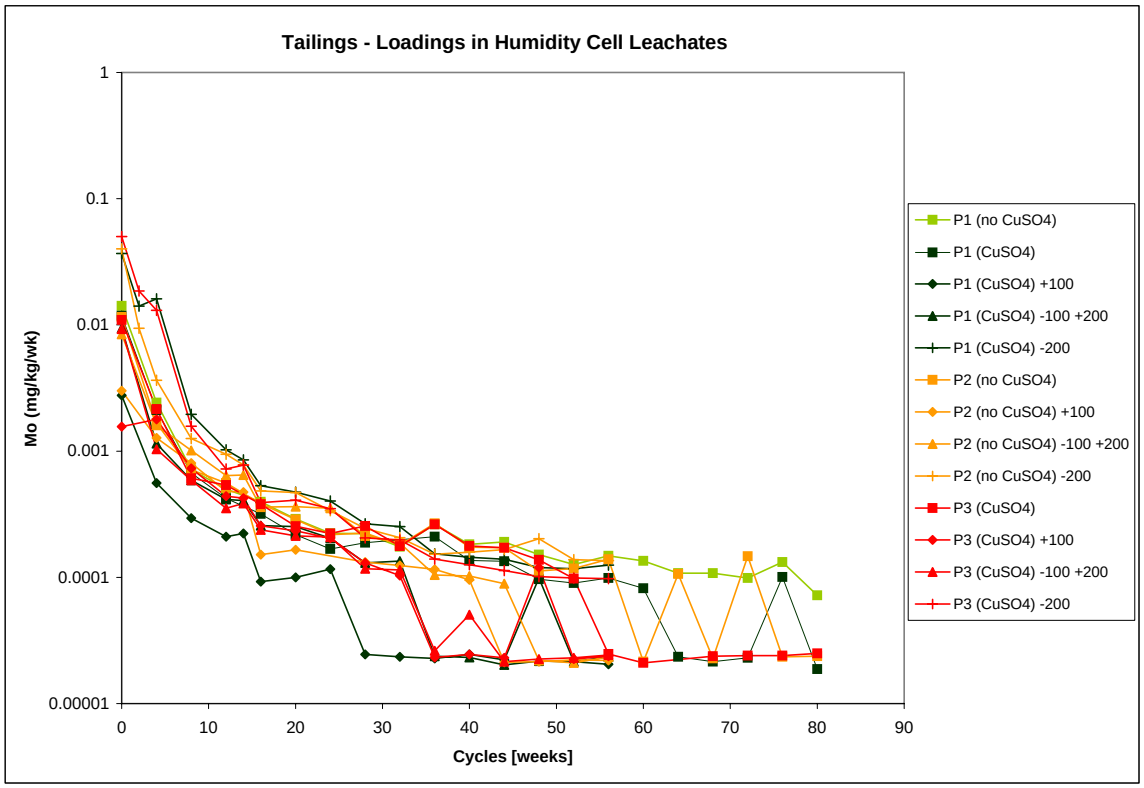
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.26



loadings_Tailings_format.xls

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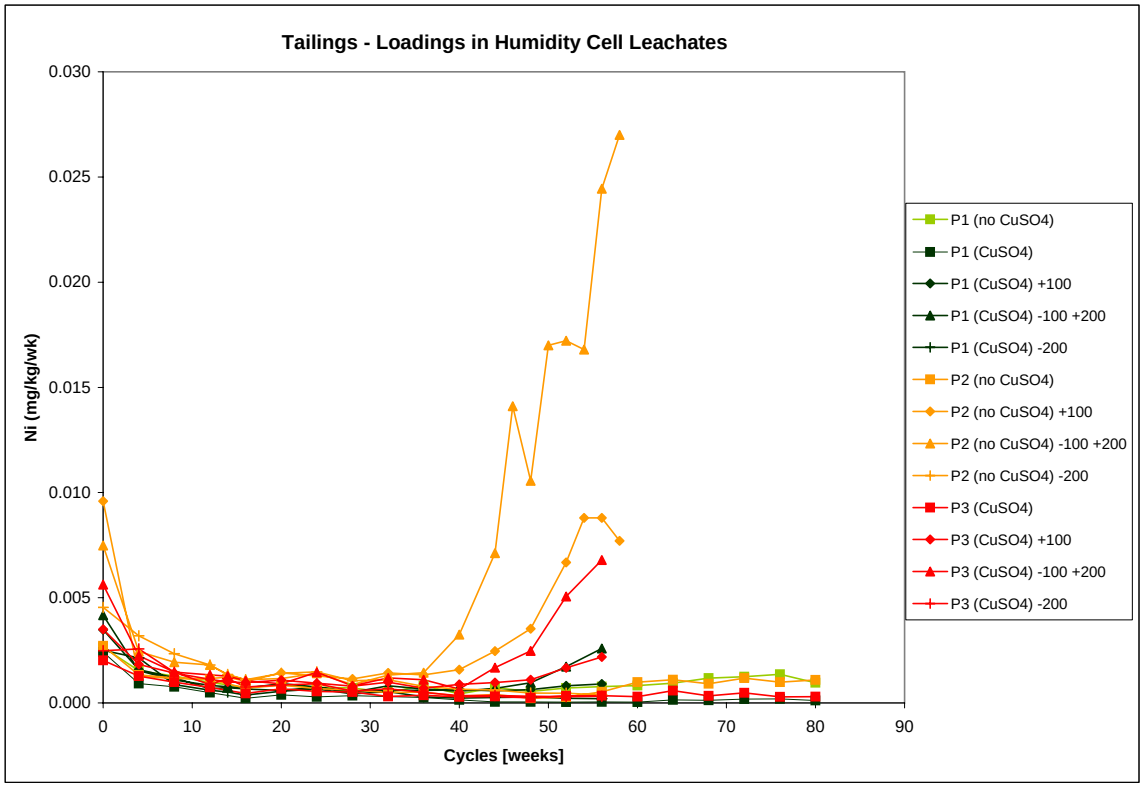
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.27



loadings_Tailings_format.xls

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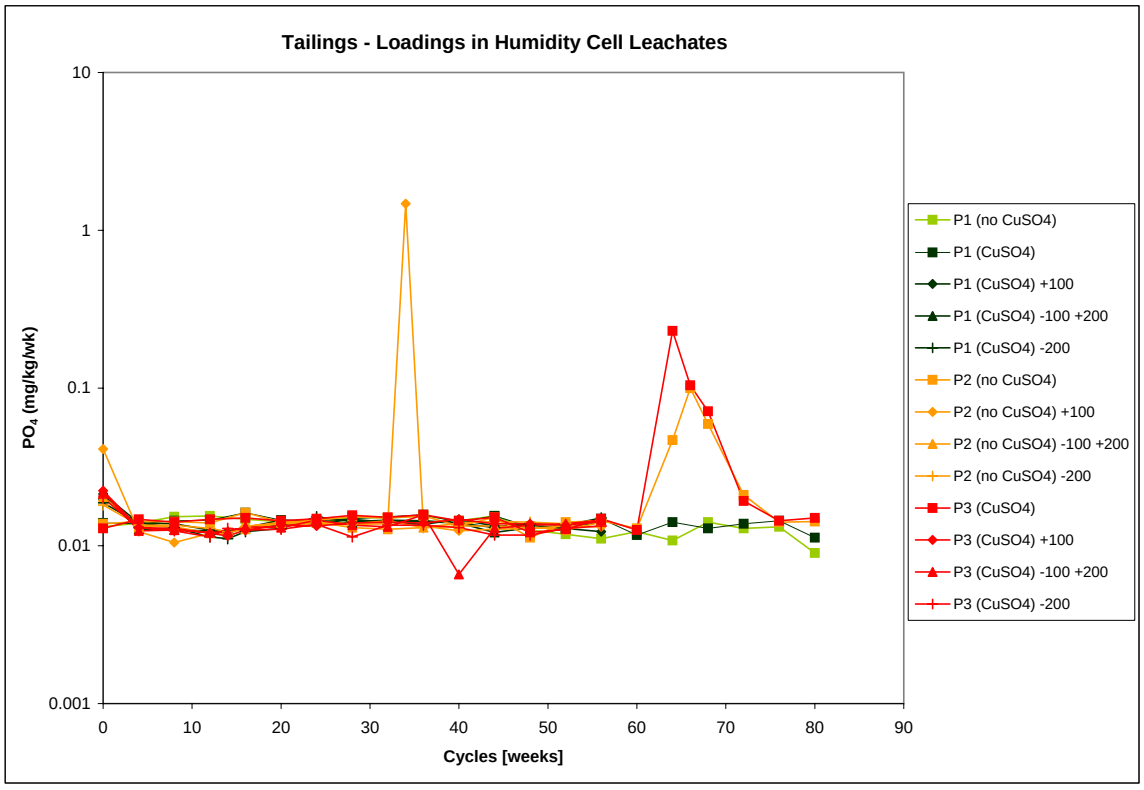
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.28



loadings_Tailings_format.xls

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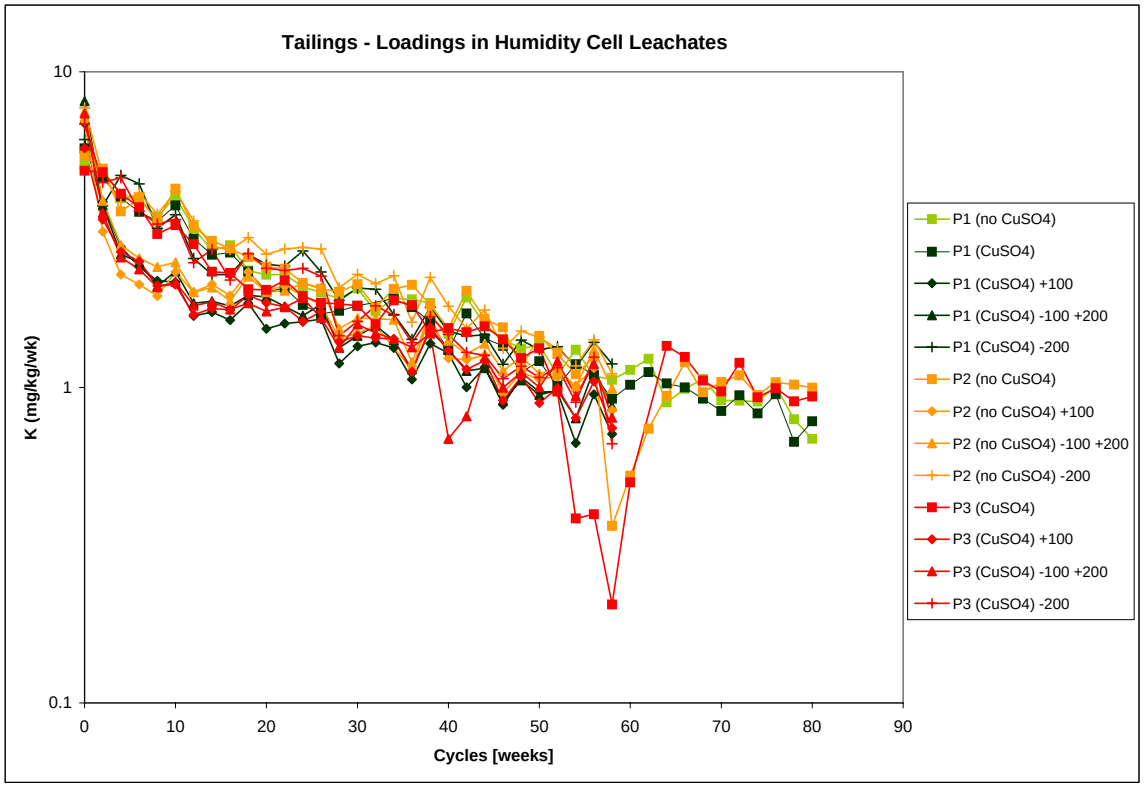
Graph C.1.29



loadings_Tailings_format.xls

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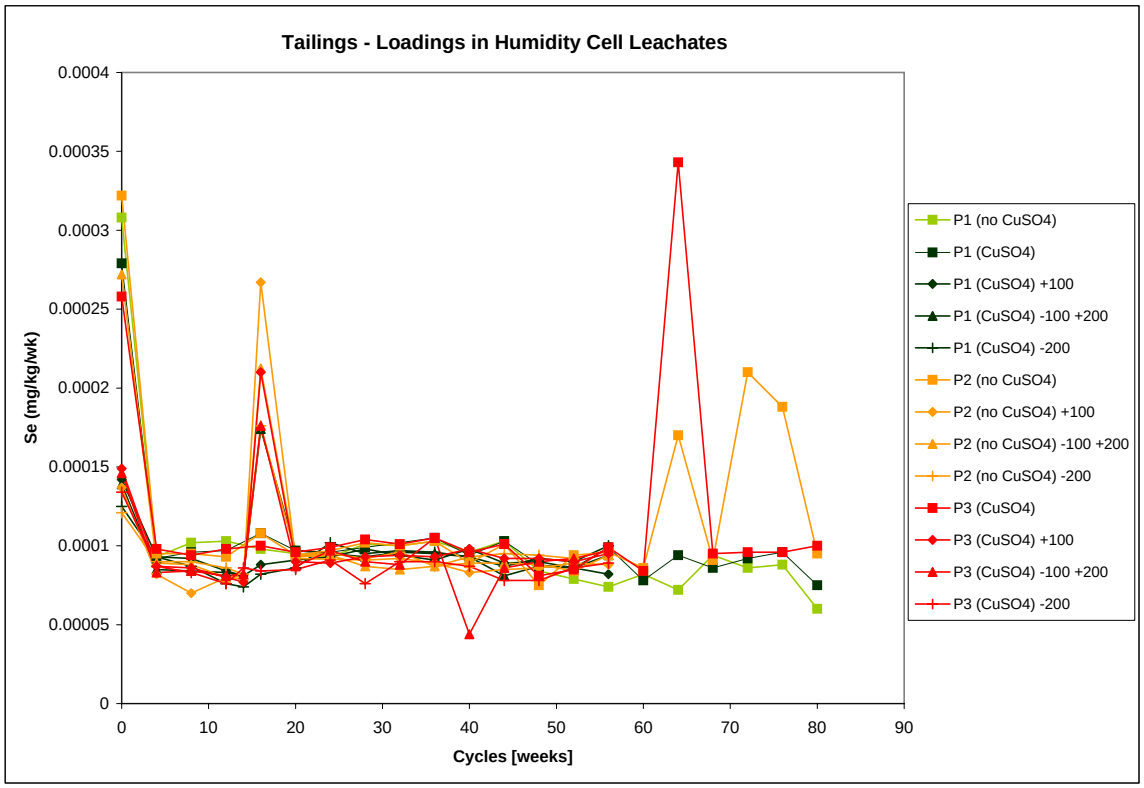
Graph C.1.30



loadings_Tailings_format.xls

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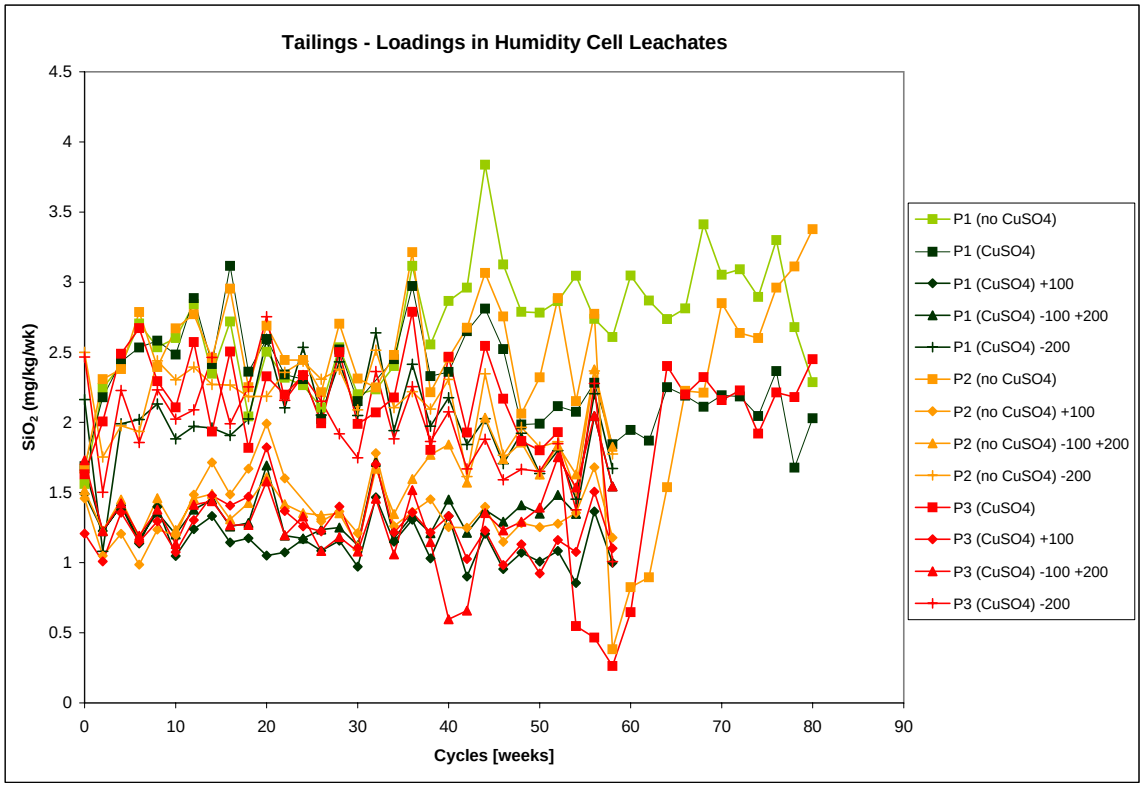
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.31



loadings_Tailings_format.xls

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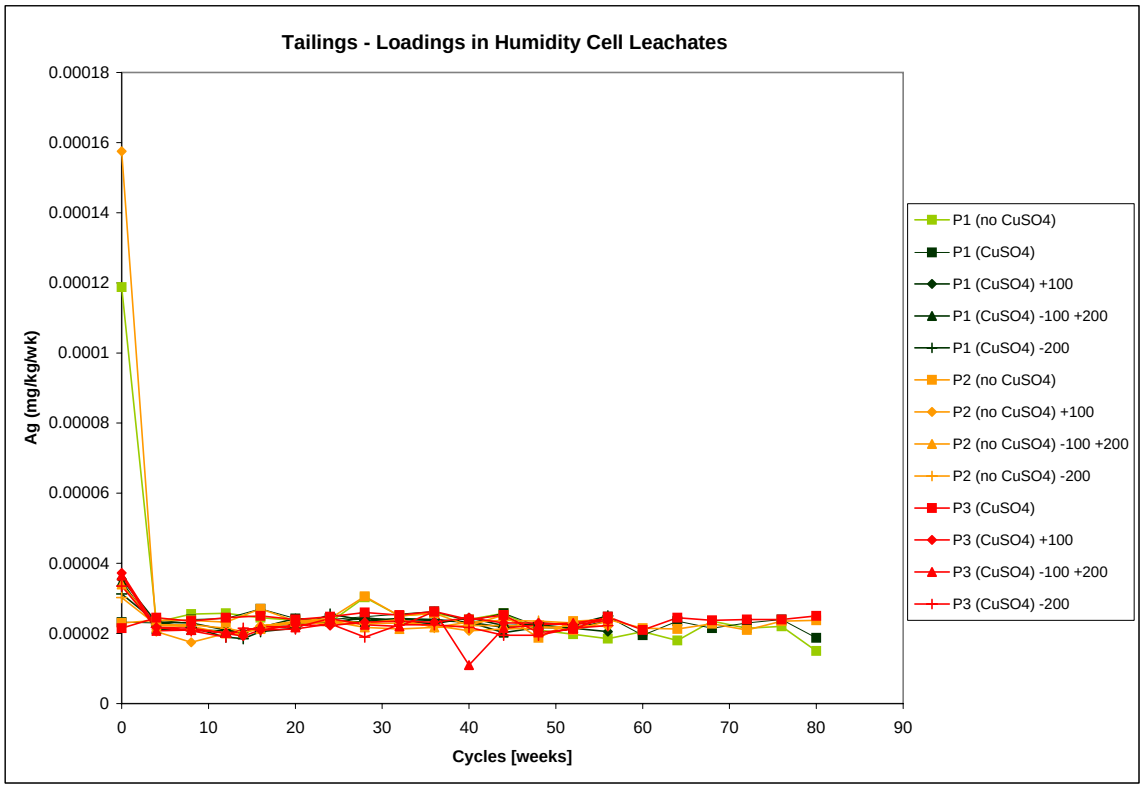
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.32



loadings_Tailings_format.xls

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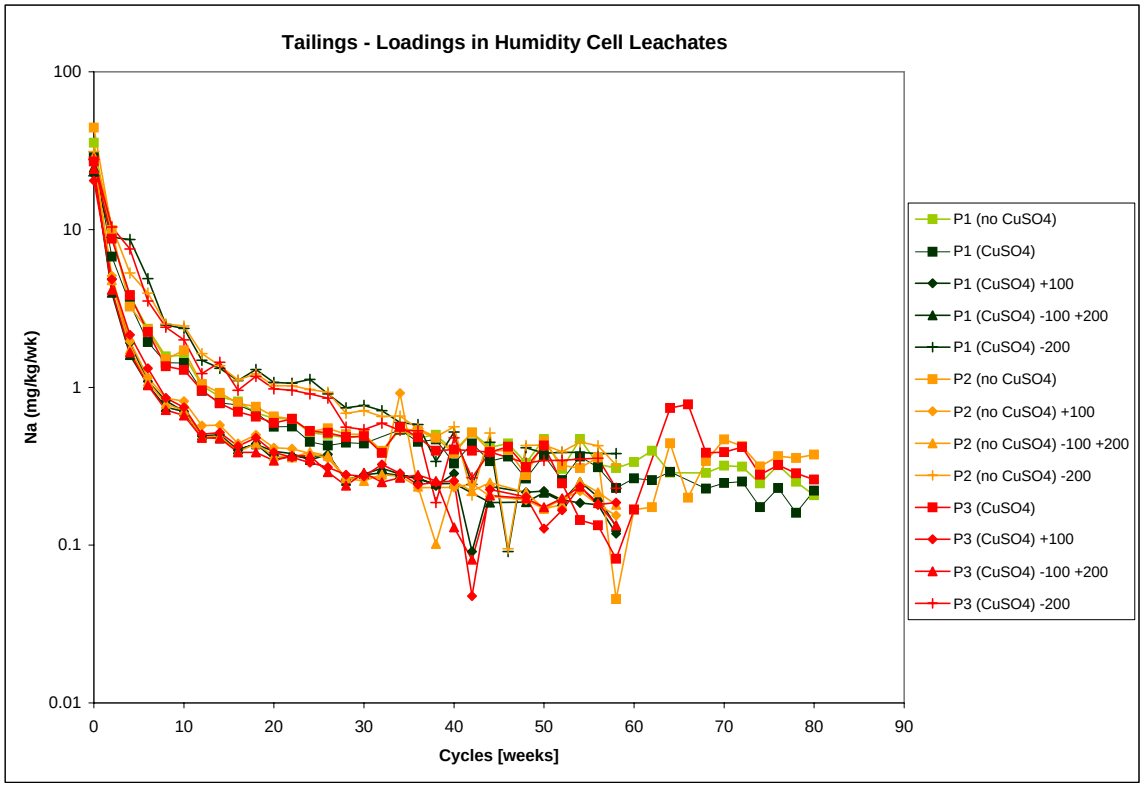
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.33



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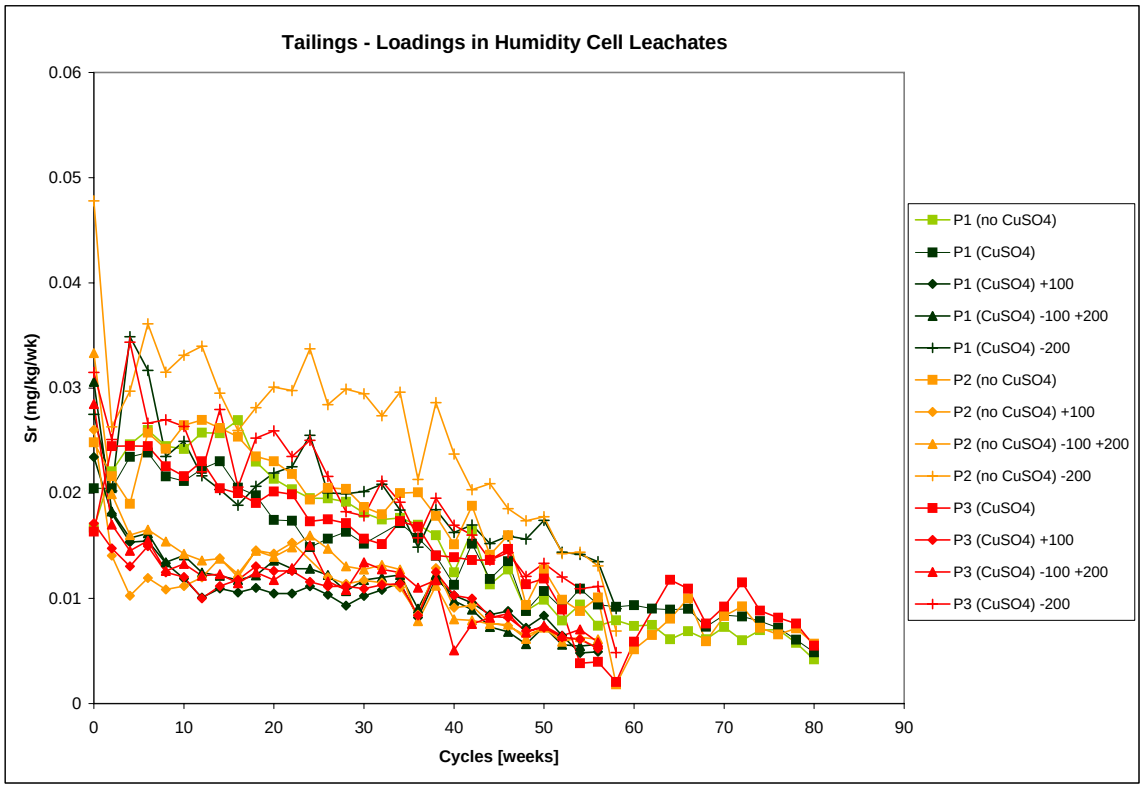
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.34



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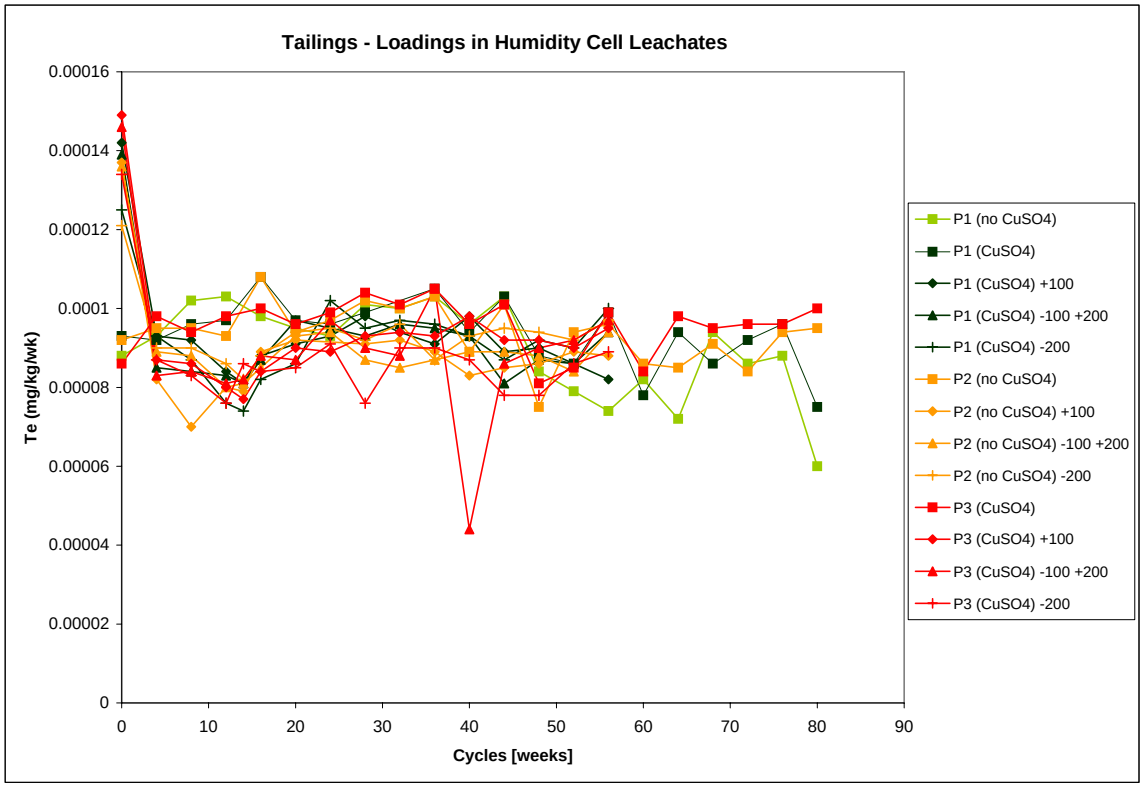
Graph C.1.35



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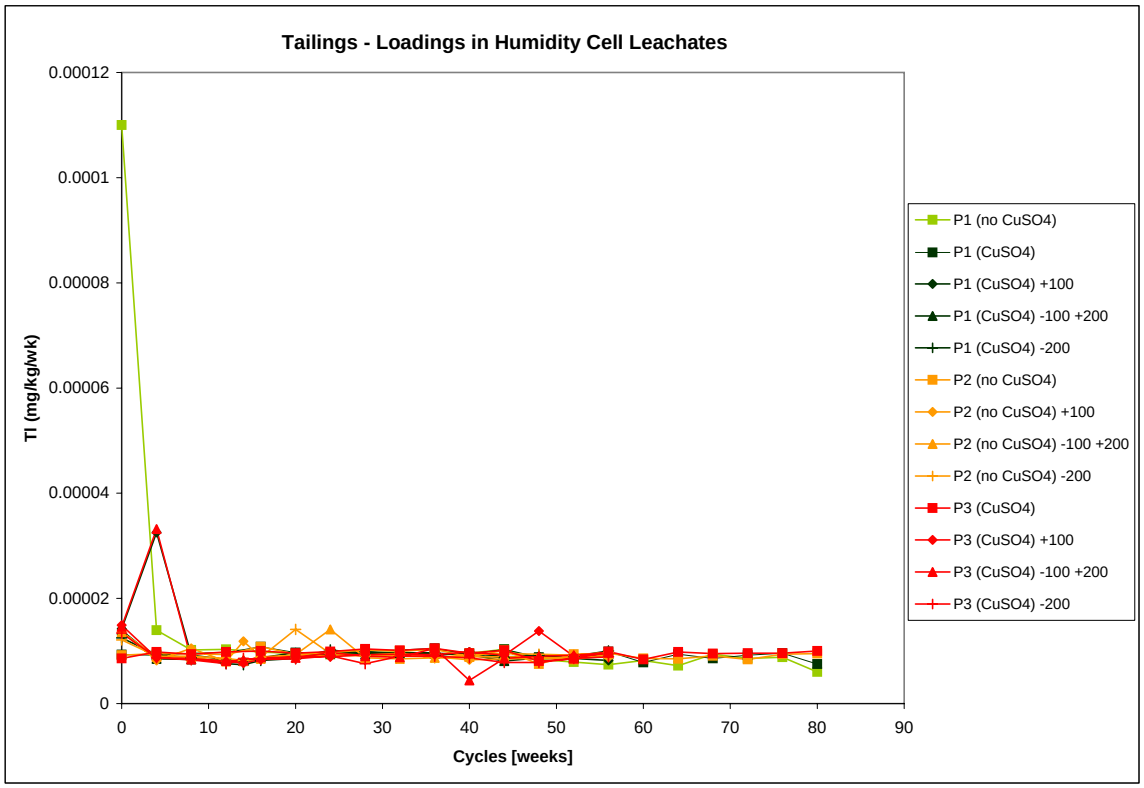
Graph C.1.36



loadings_Tailings_format.xls

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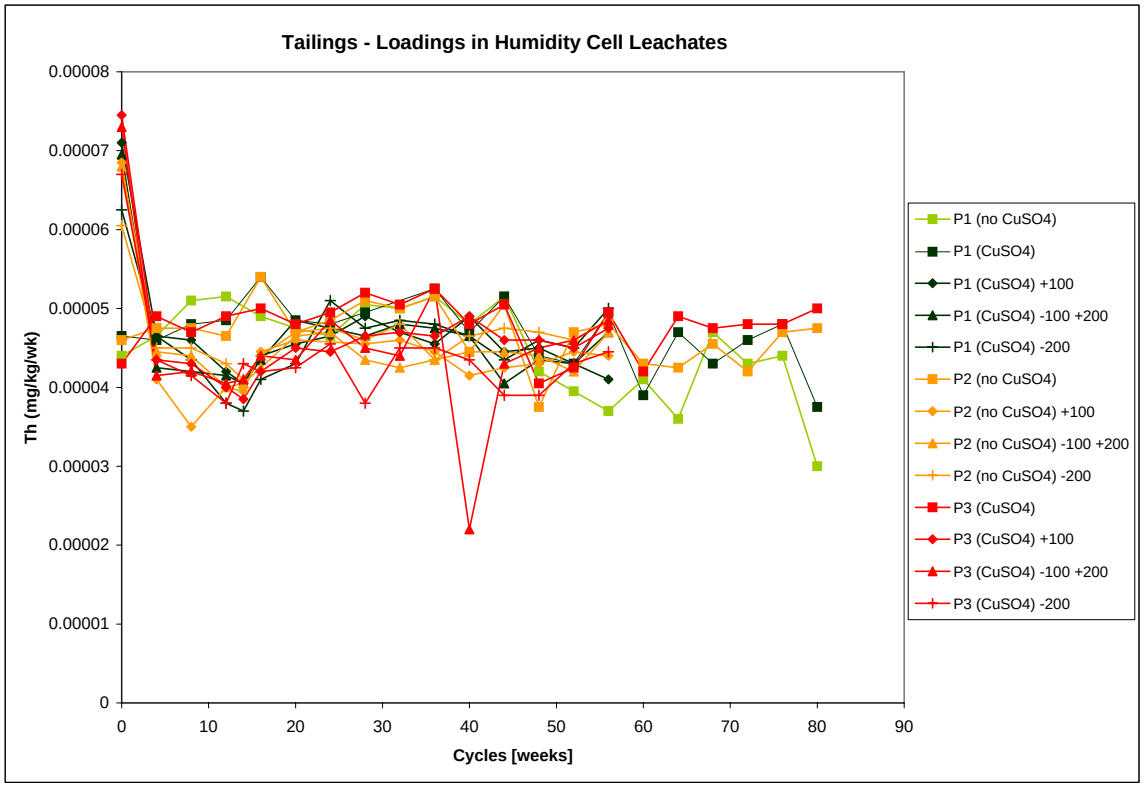
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.37



loadings_Tailings_format.xls

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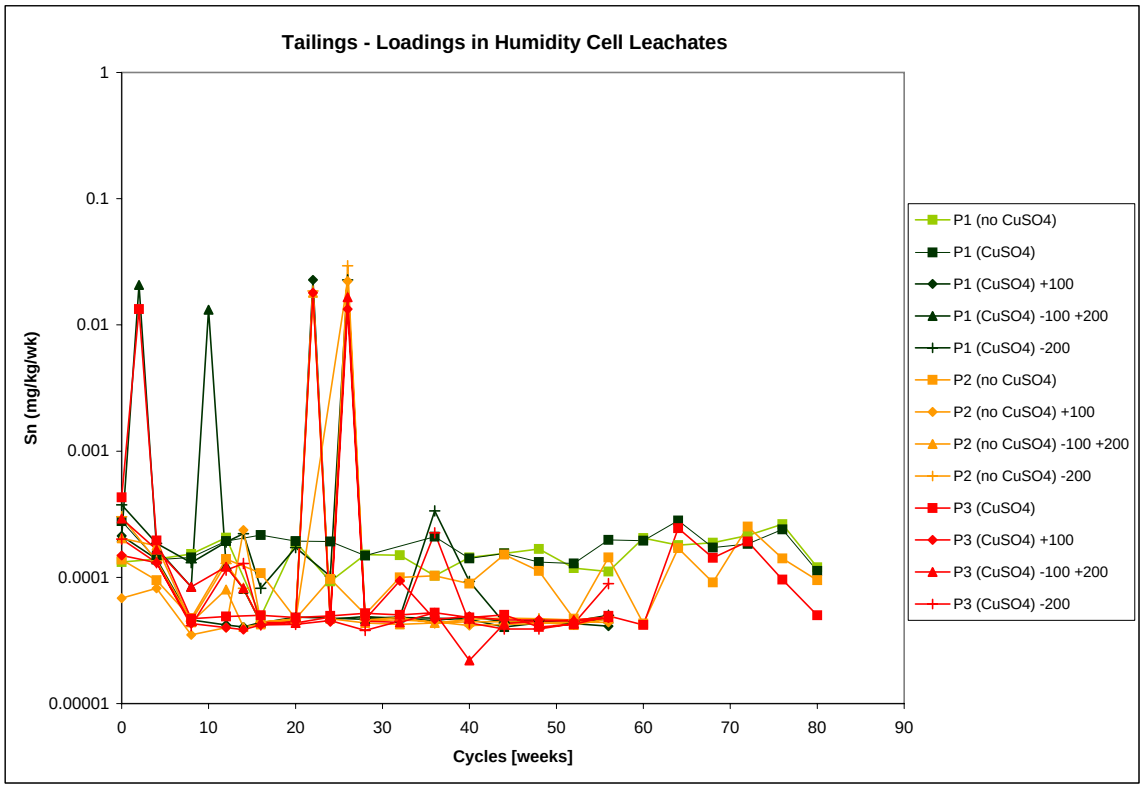
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.38



loadings_Tailings_format.xls

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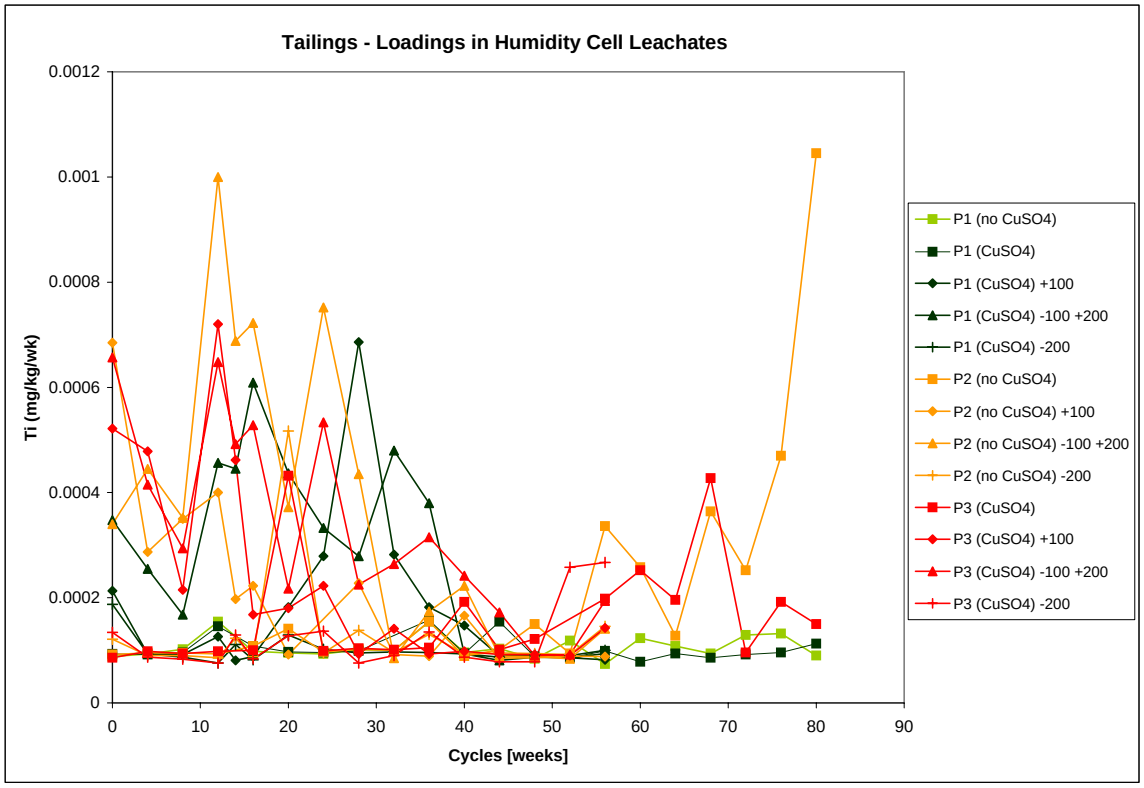
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.39



loadings_Tailings_format.xls

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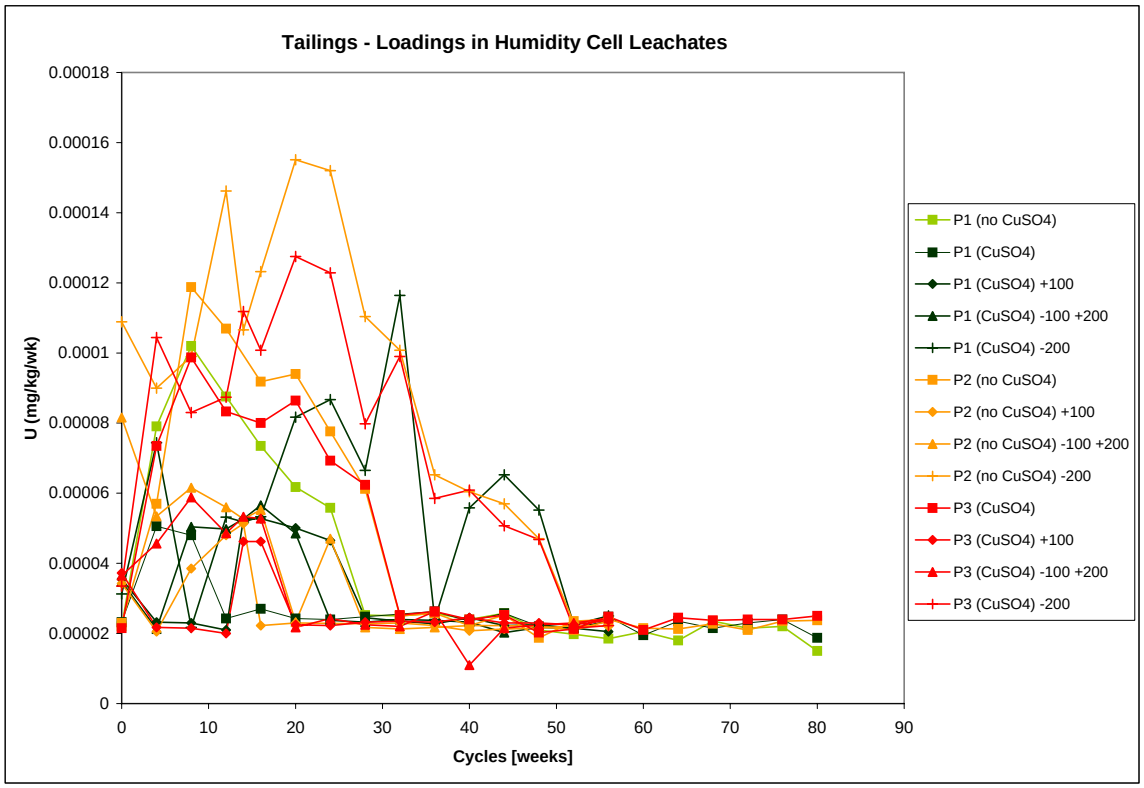
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.40



loadings_Tailings_format.xls

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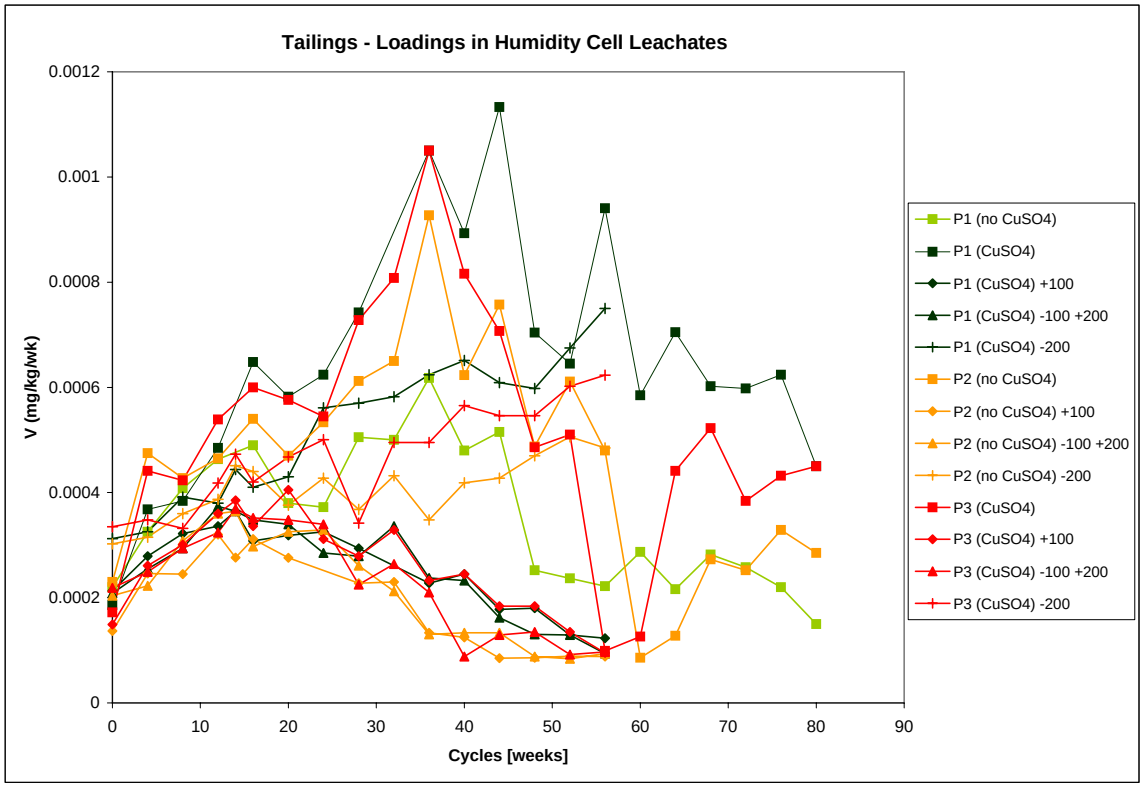
Graph C.1.41



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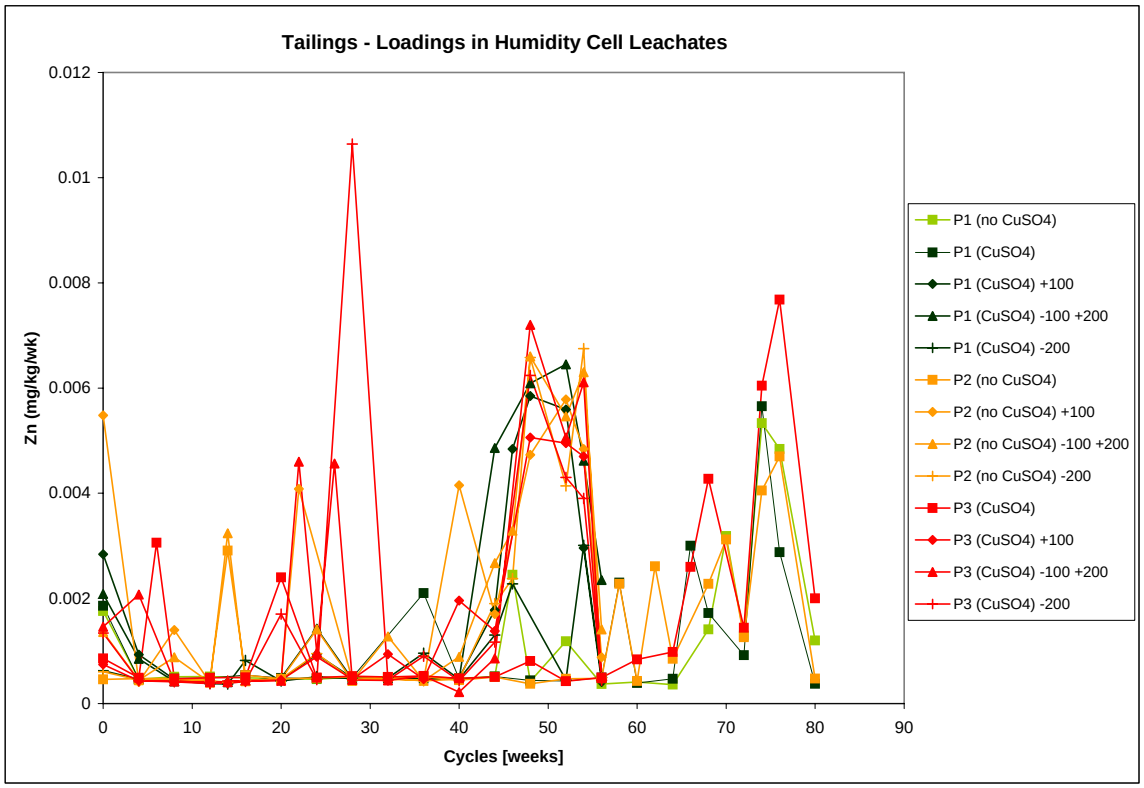
Graph C.1.42



loadings_Tailings_format.xls

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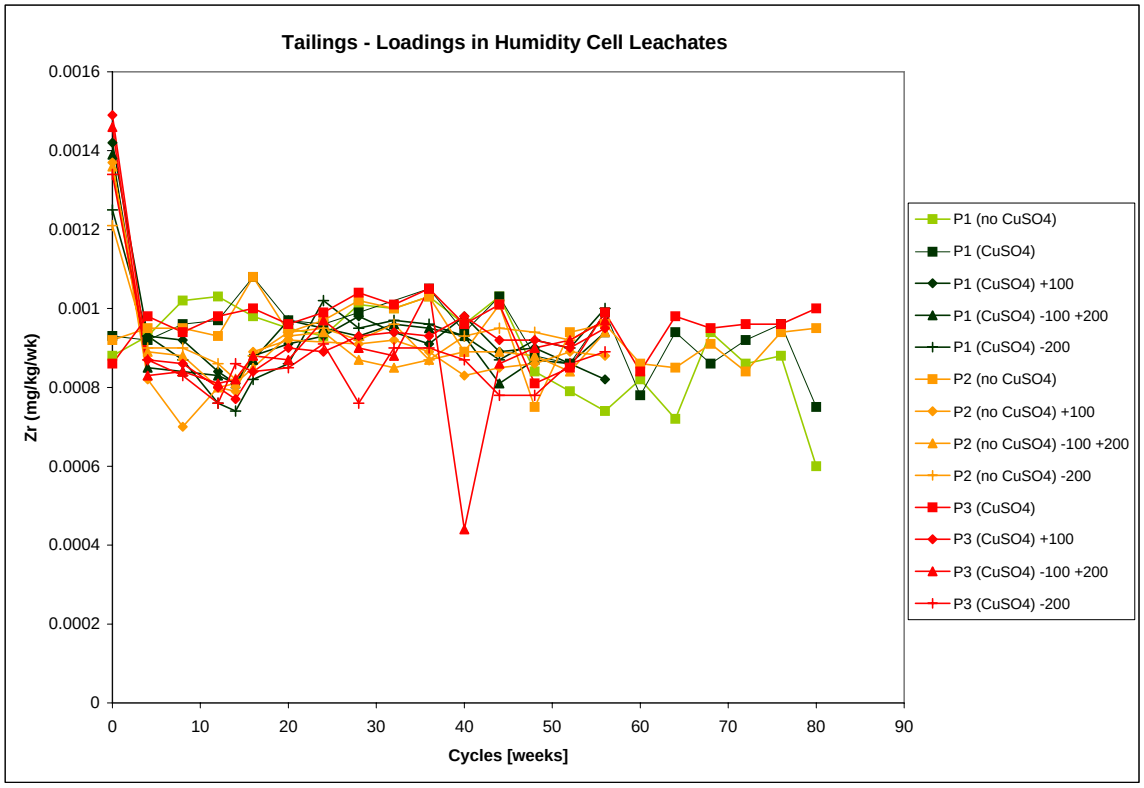
Appendix C.1
ASTM Cells (Loadings)

Graph C.1.43



loadings_Tailings_format.xls

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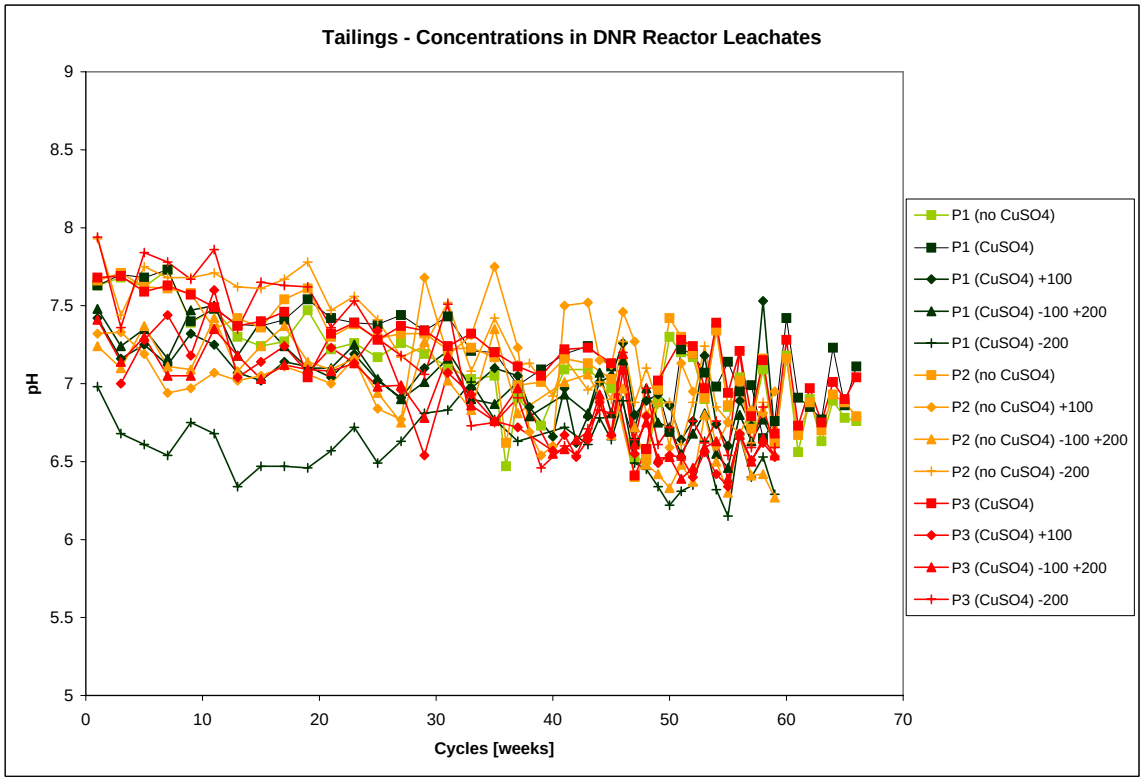
Appendix C.2

MDNR Reactors (Loadings)

Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.1



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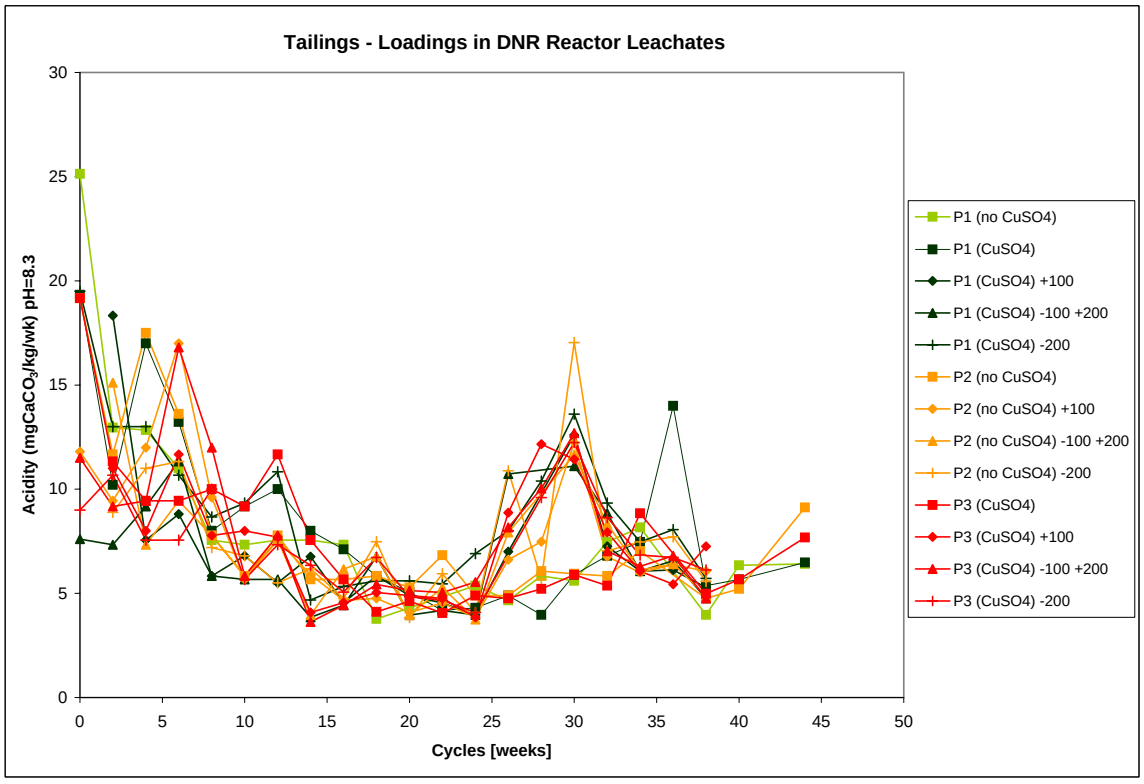
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.2



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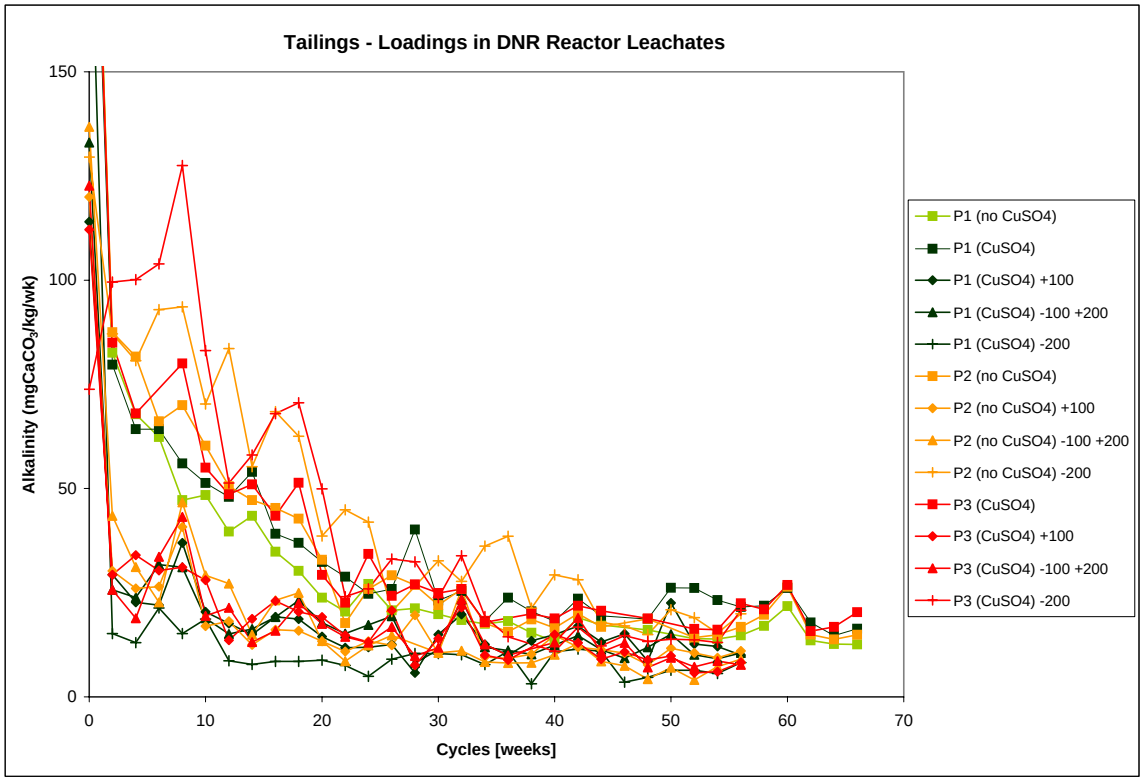
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.3



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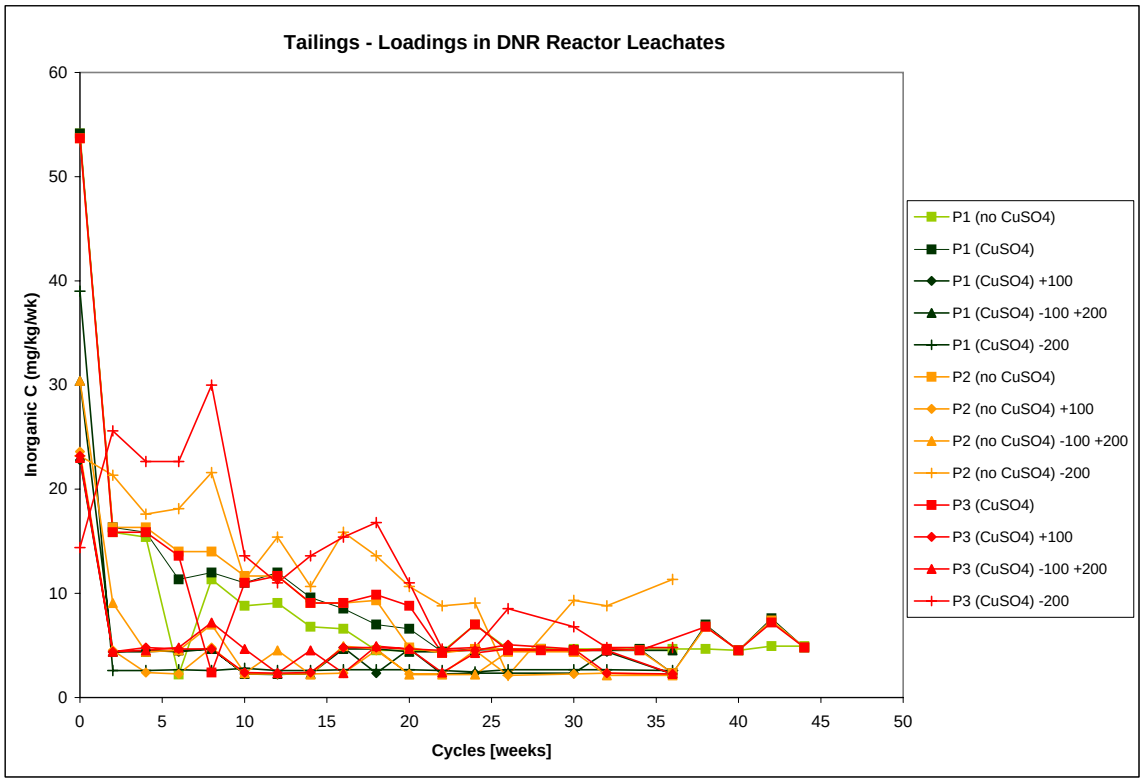
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.4



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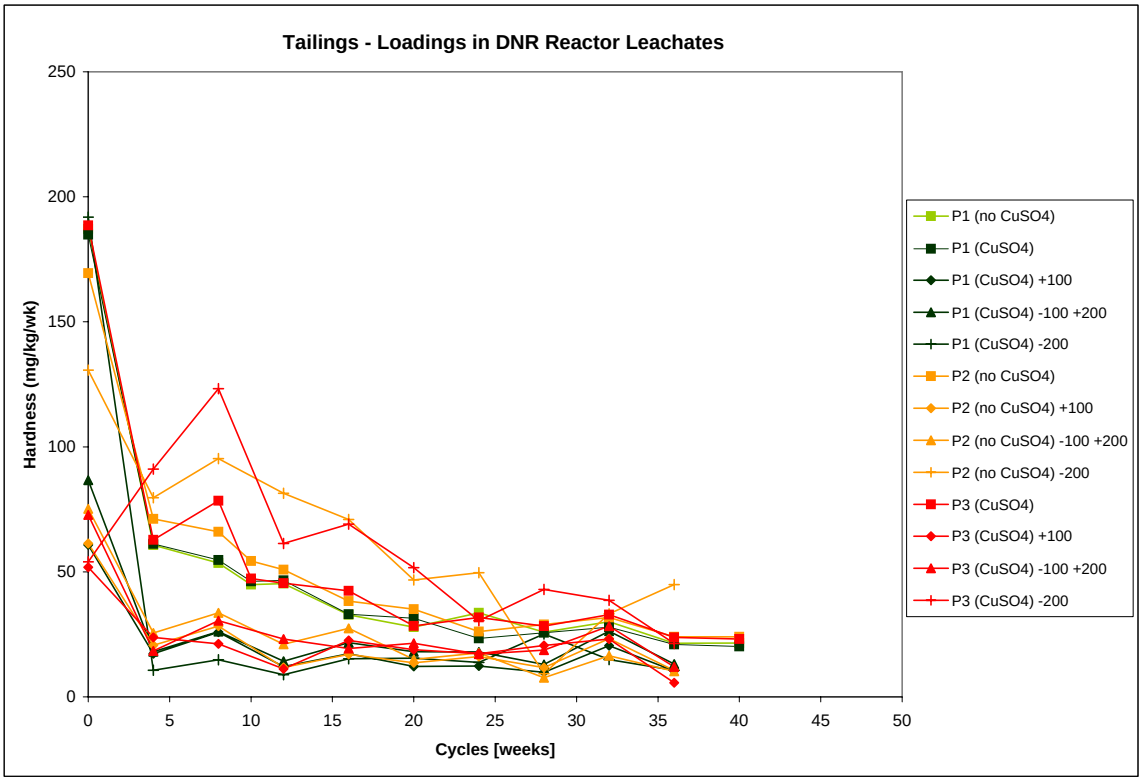
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.5



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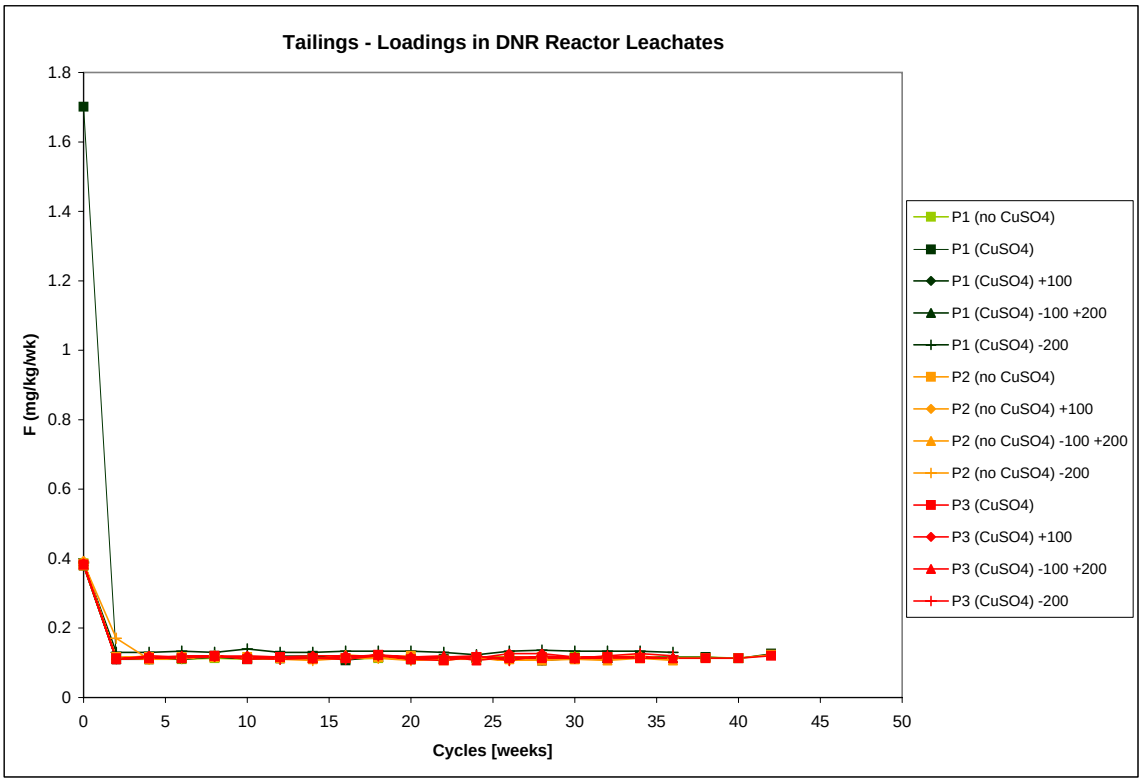
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.6



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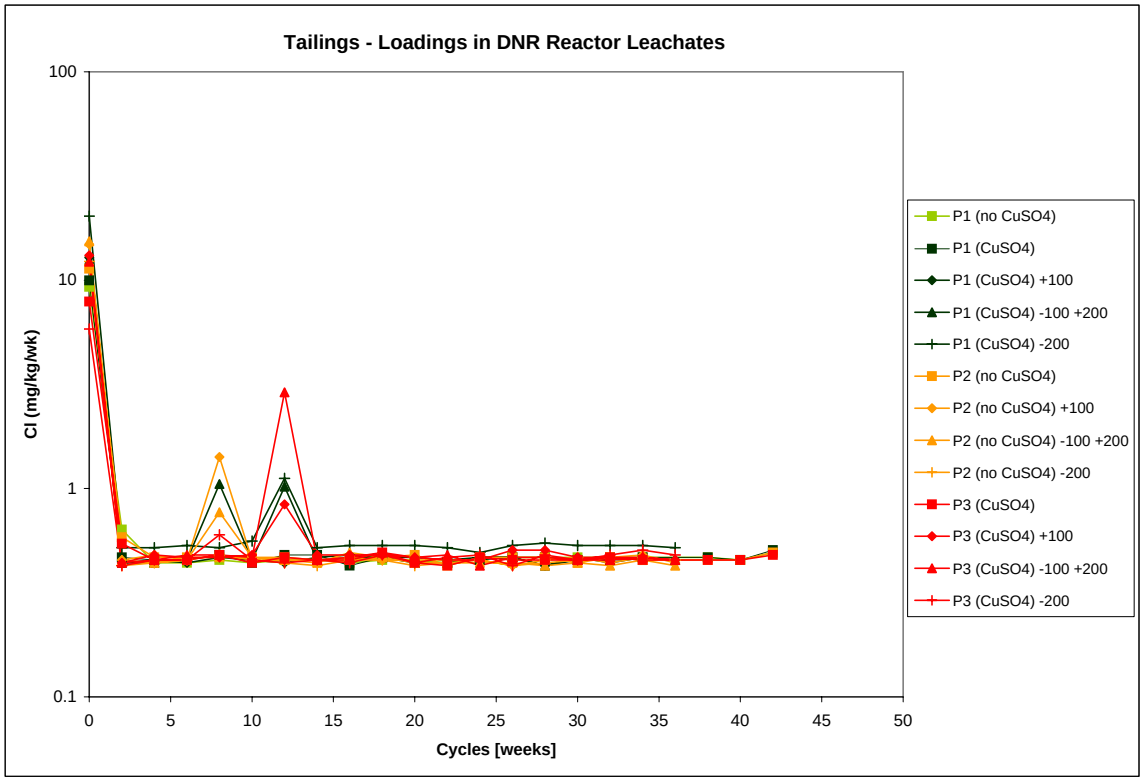
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.7



loadings_Lapakko.xls

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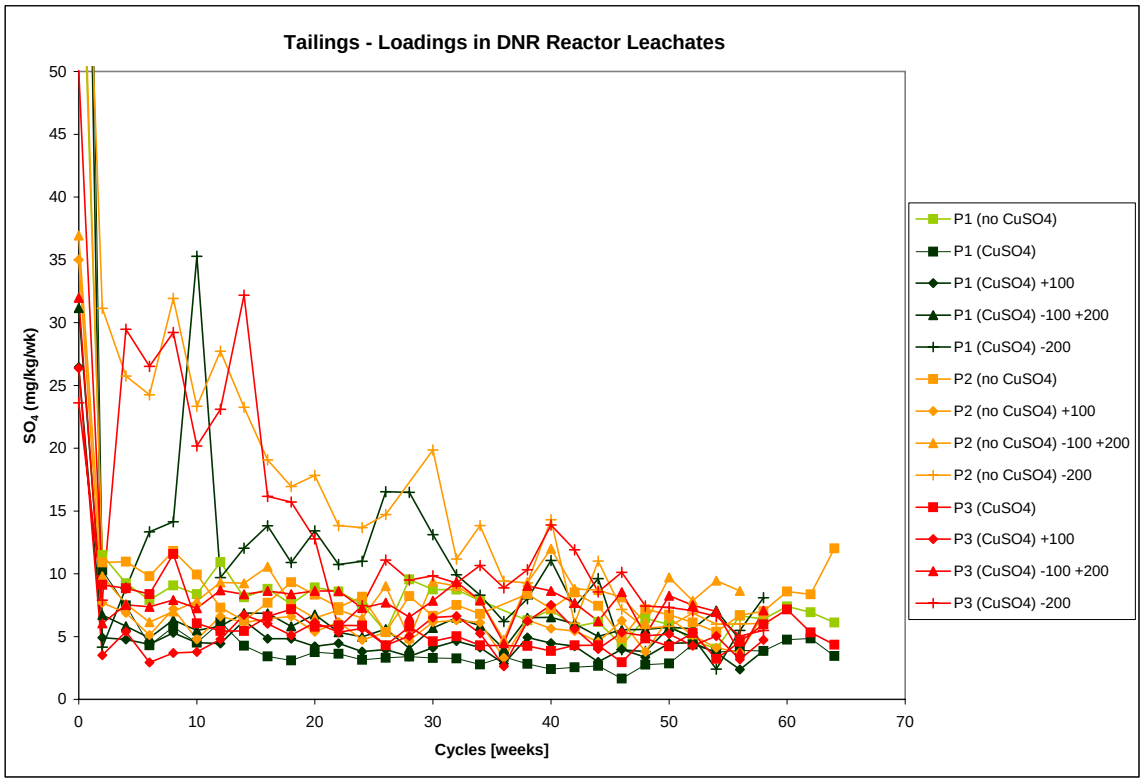
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.8



loadings_Lapakko.xls

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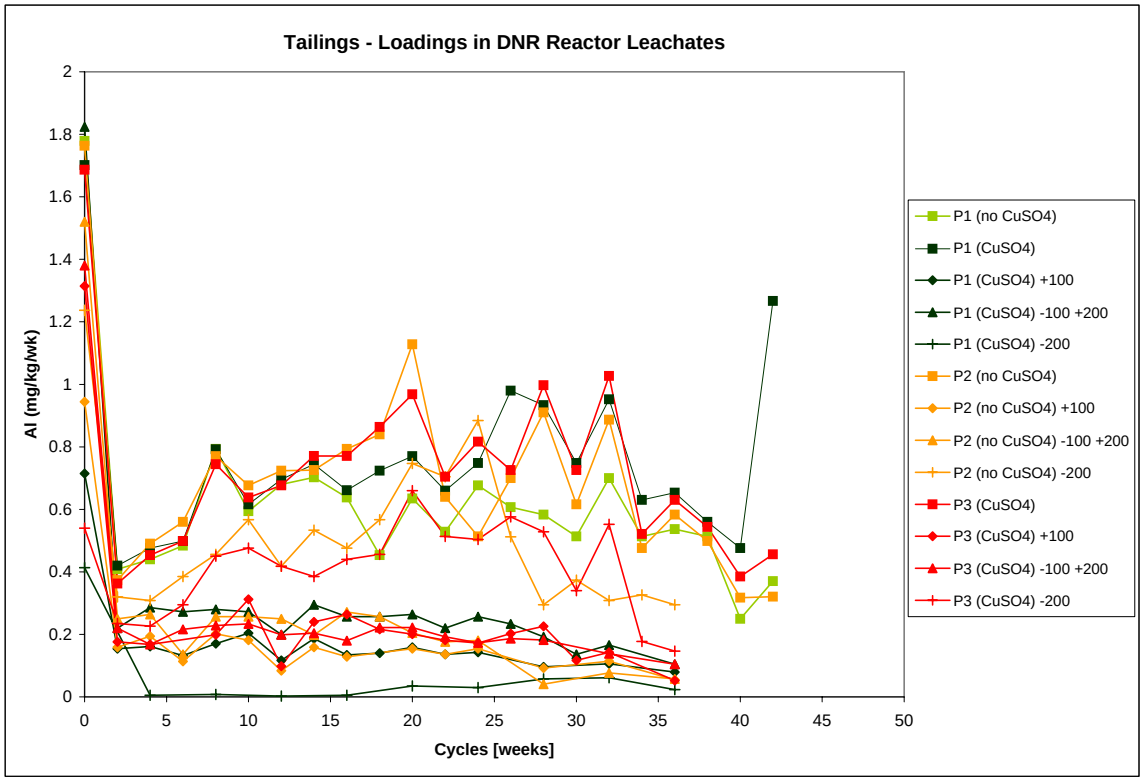
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.9



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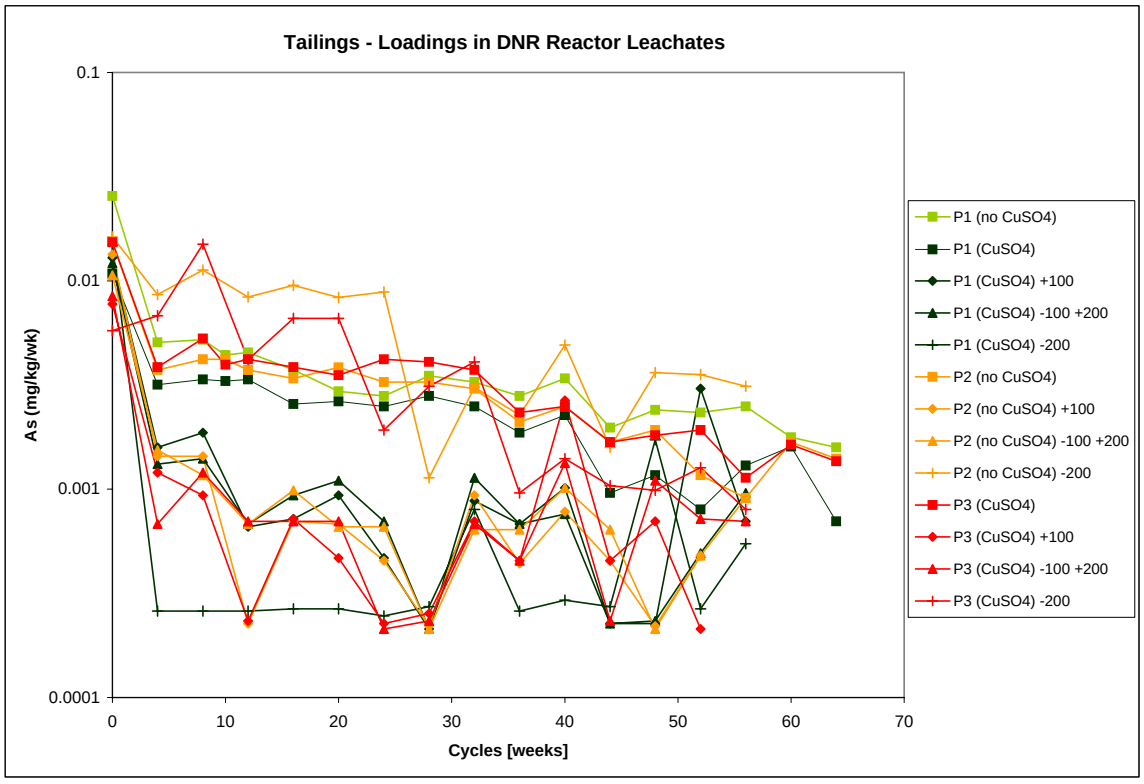
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.10



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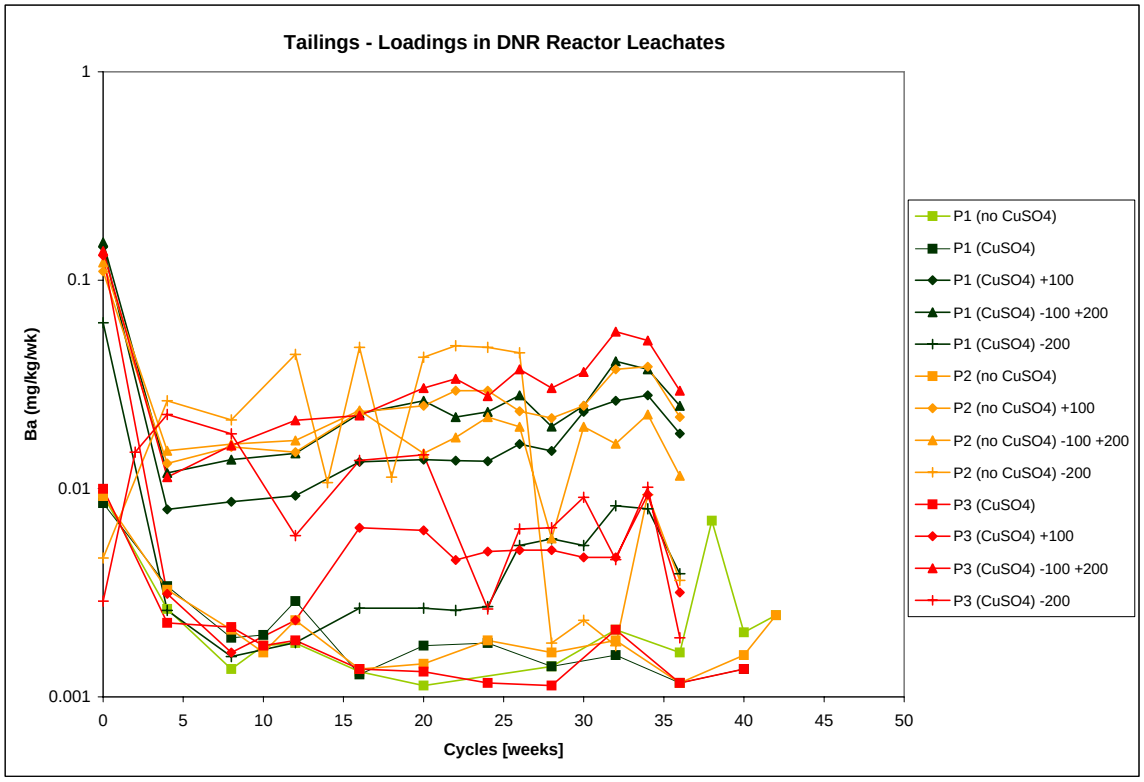
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.11



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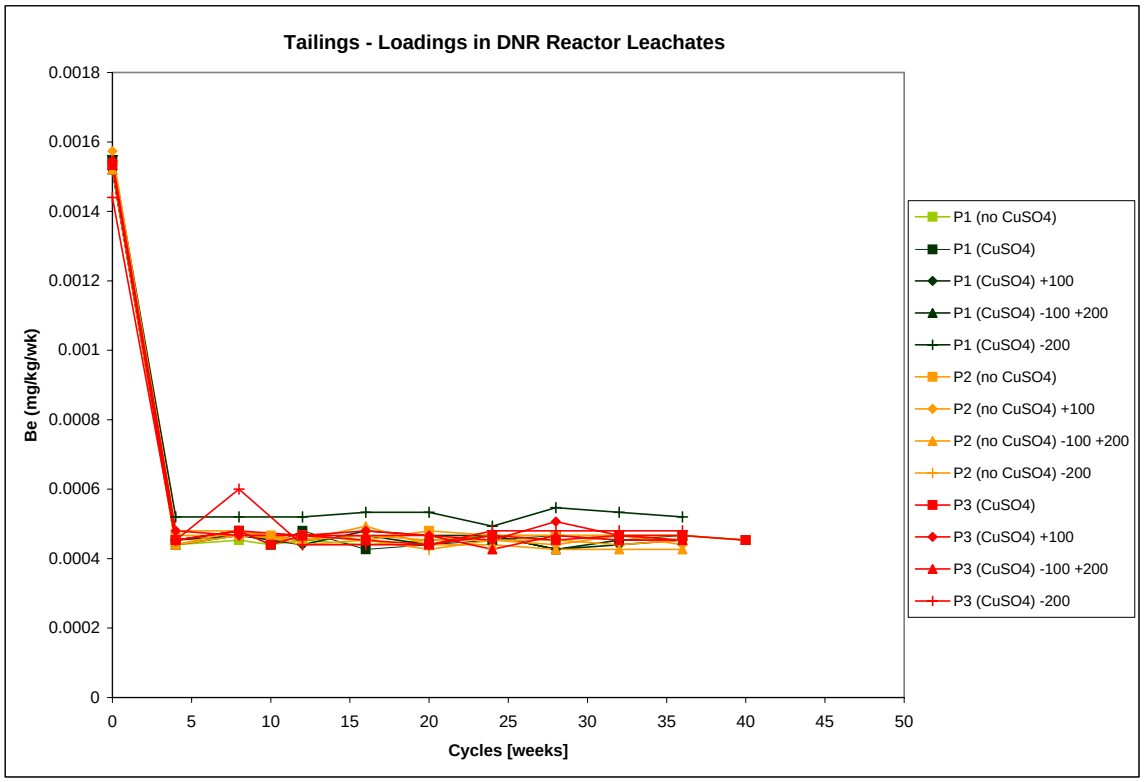
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.12



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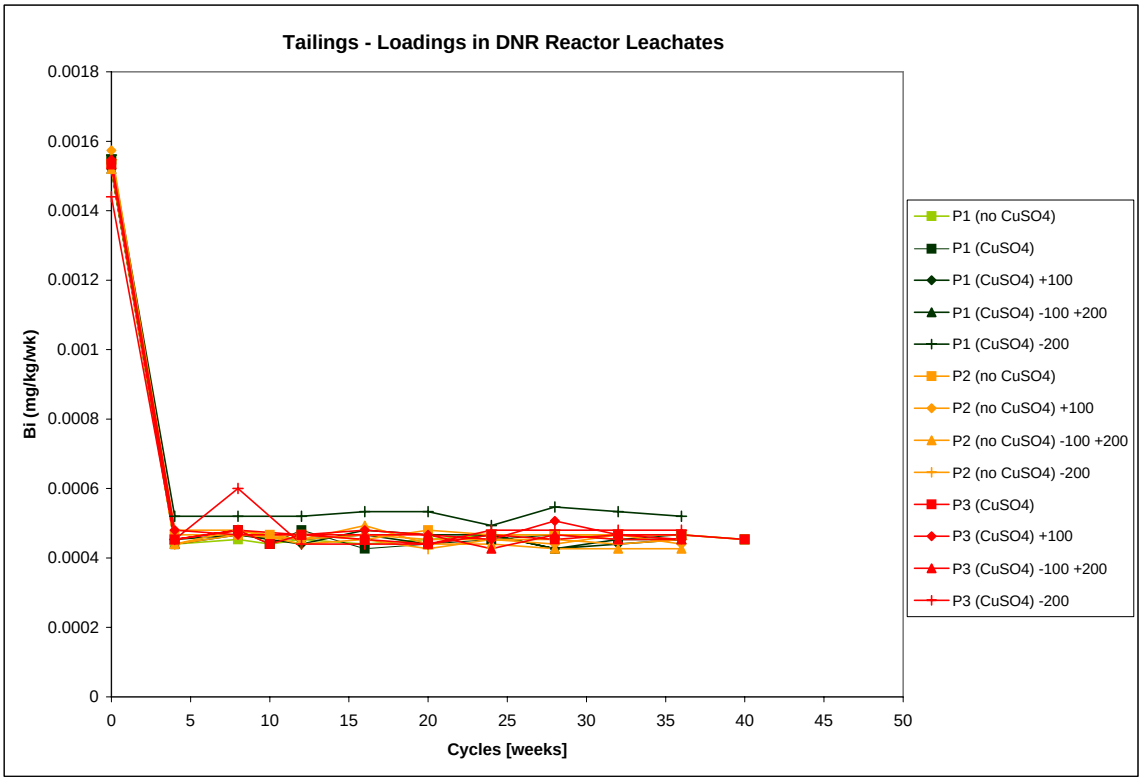
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.13



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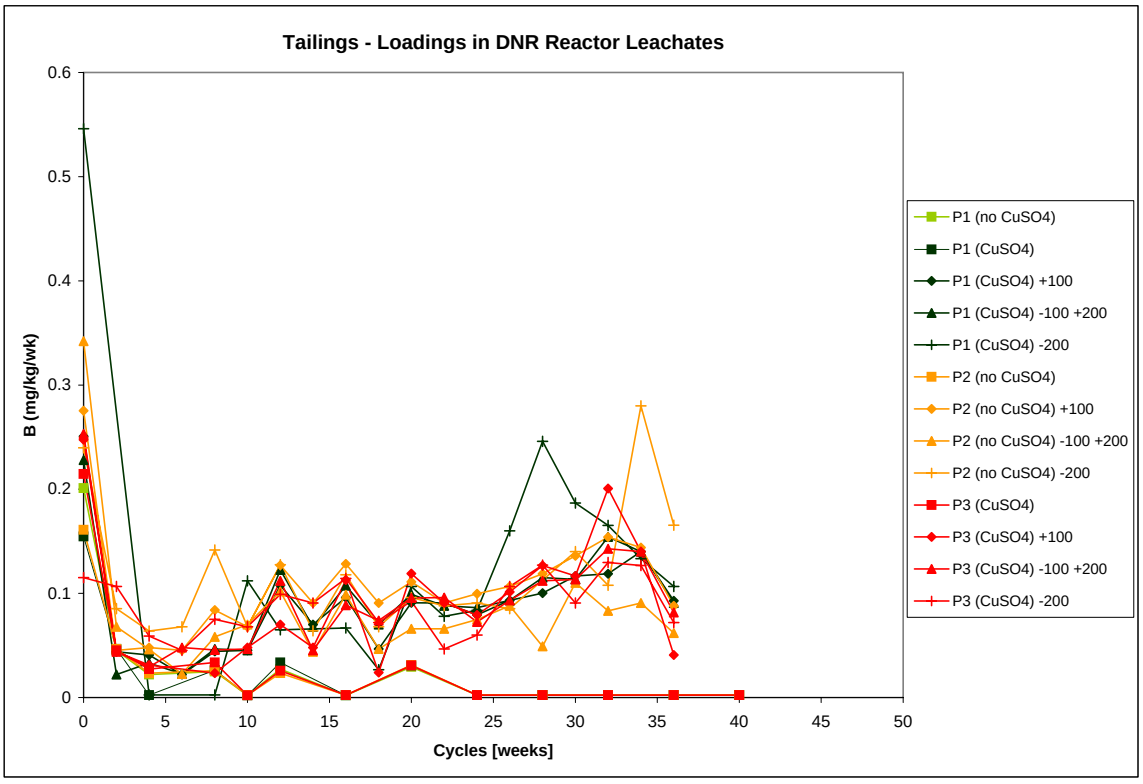
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.14



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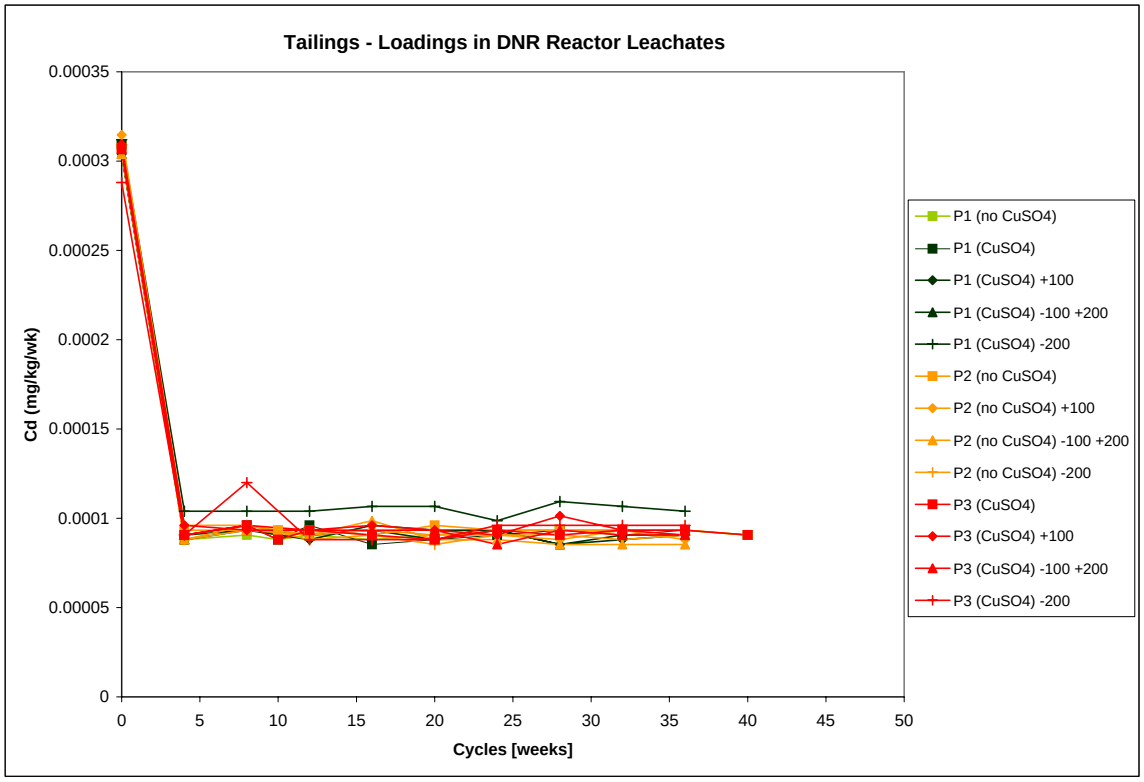
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.15



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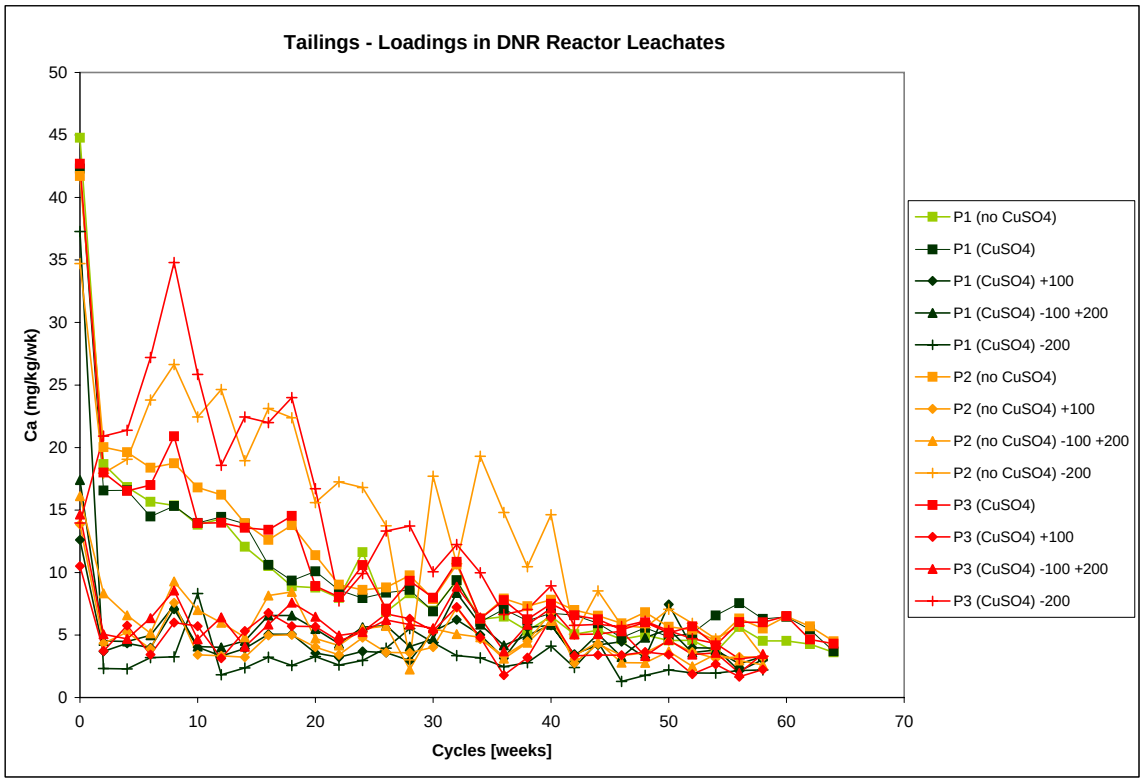
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.16



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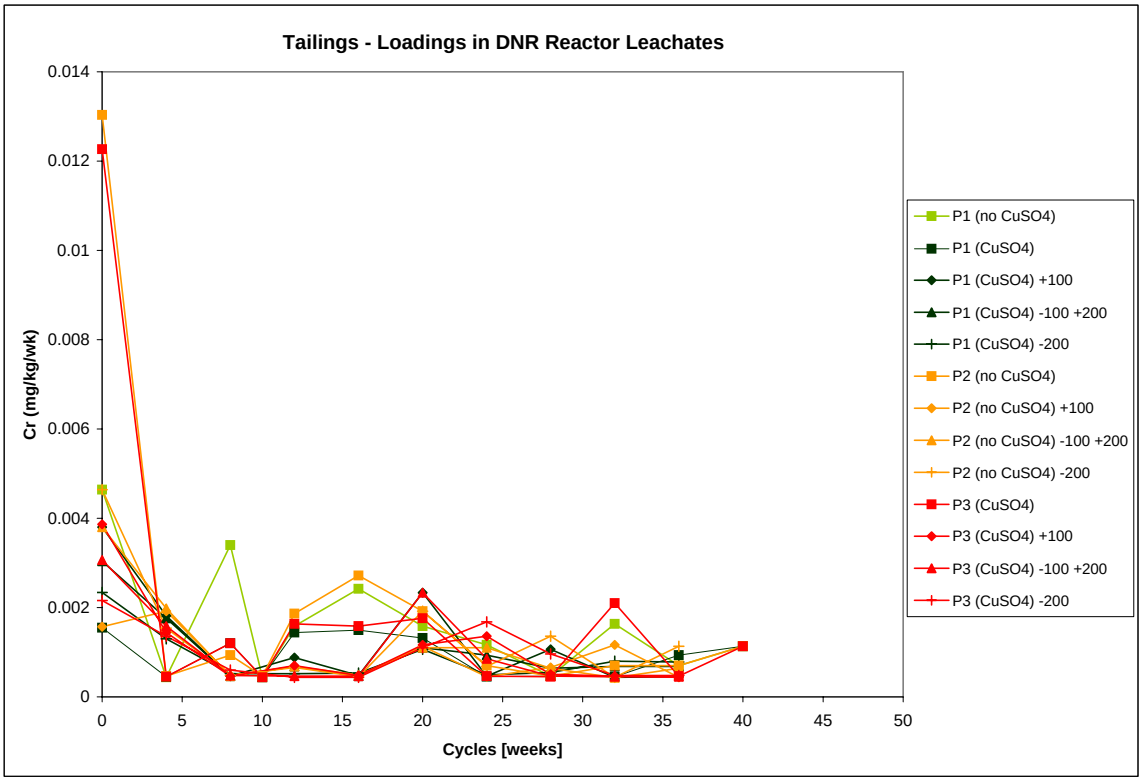
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.17



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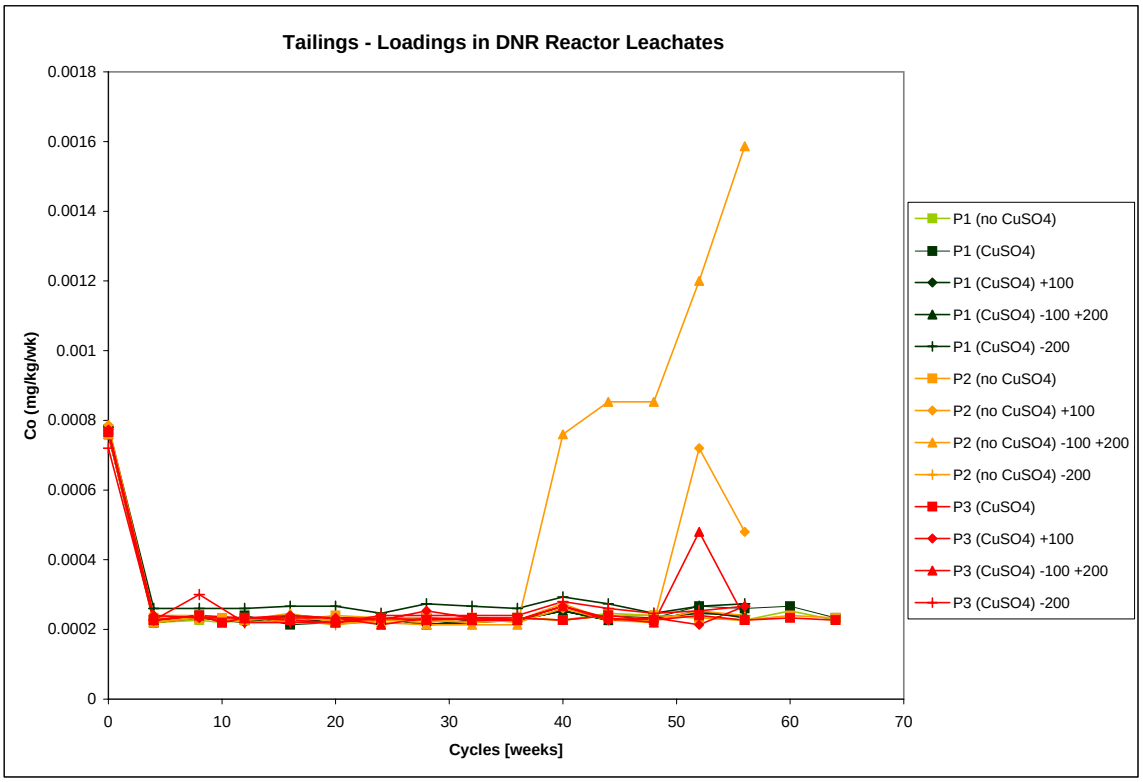
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.18



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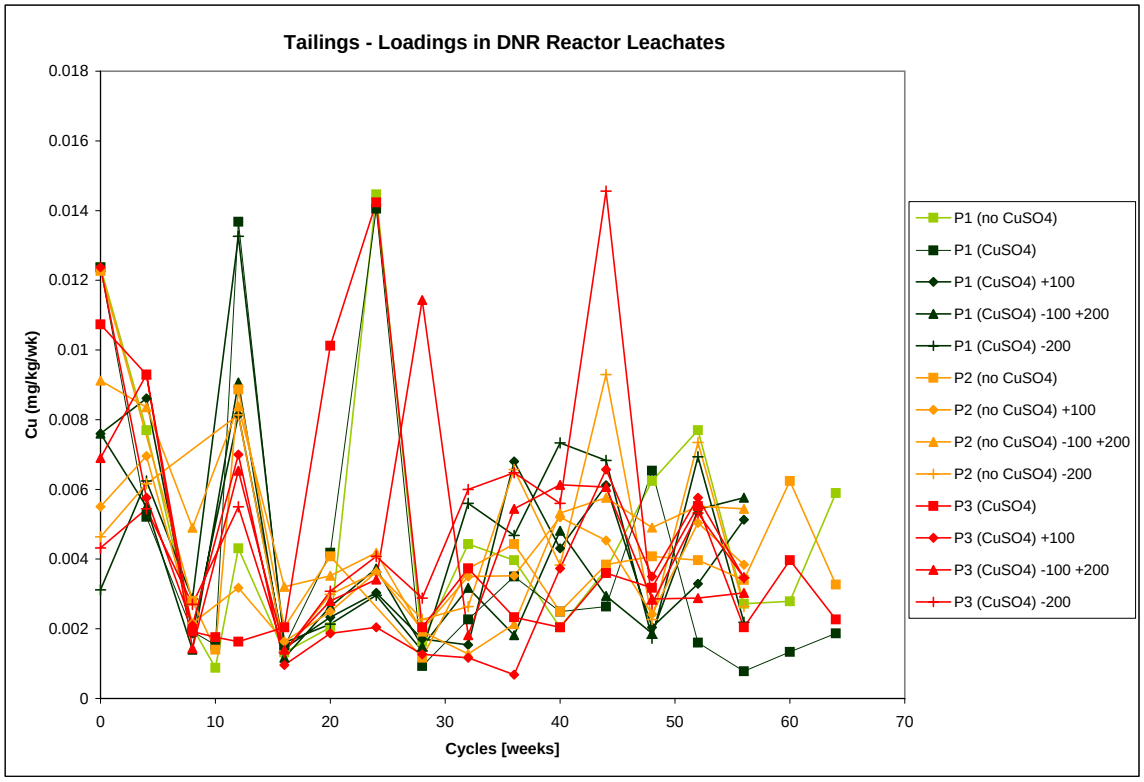
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.19



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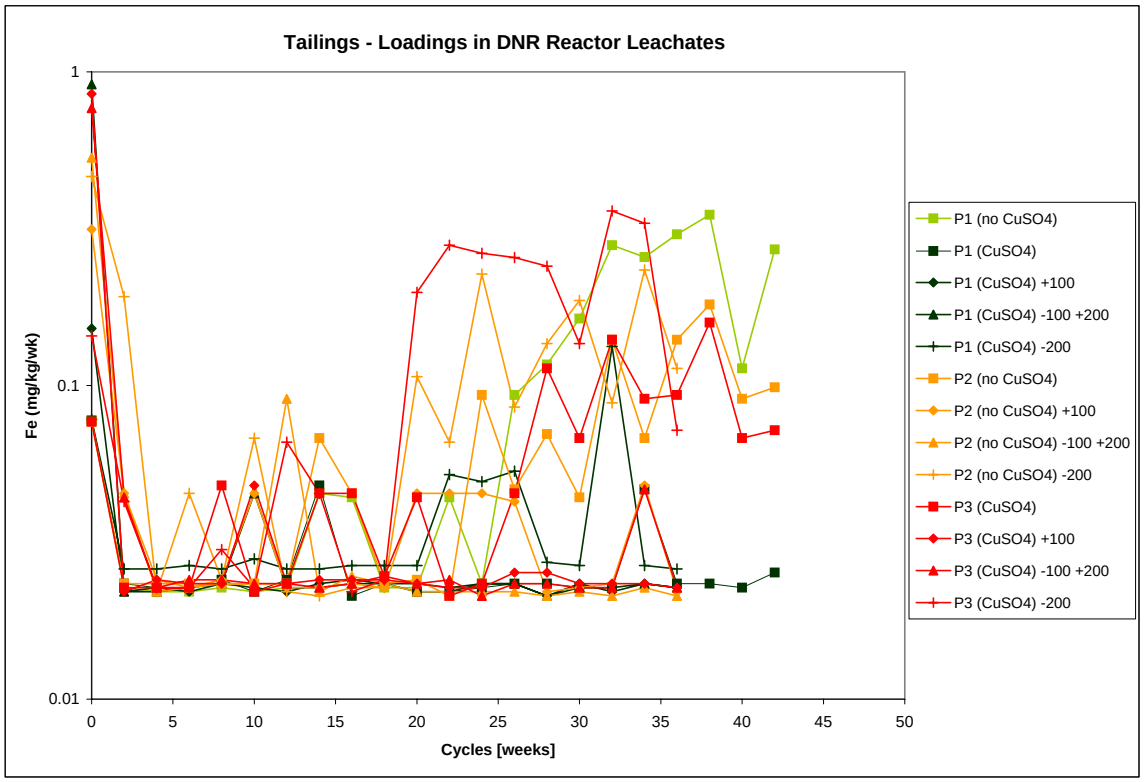
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.20



loadings_Lapakko.xls

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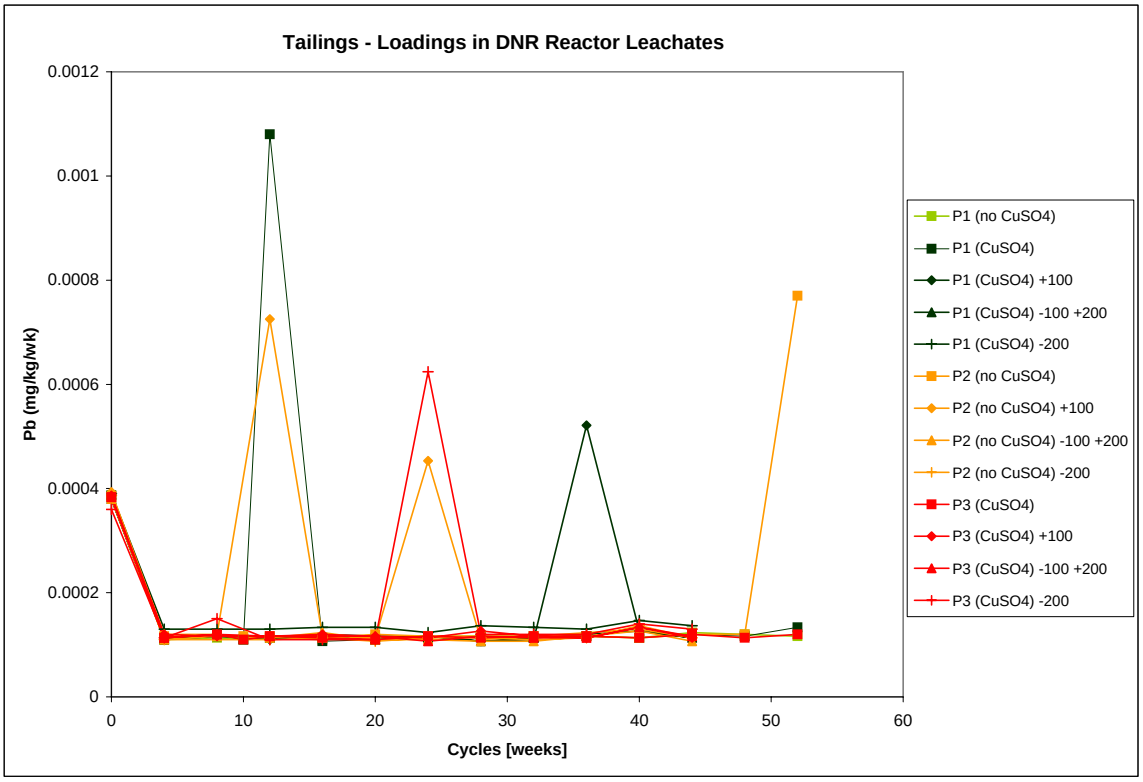
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.21



loadings_Lapakko.xls

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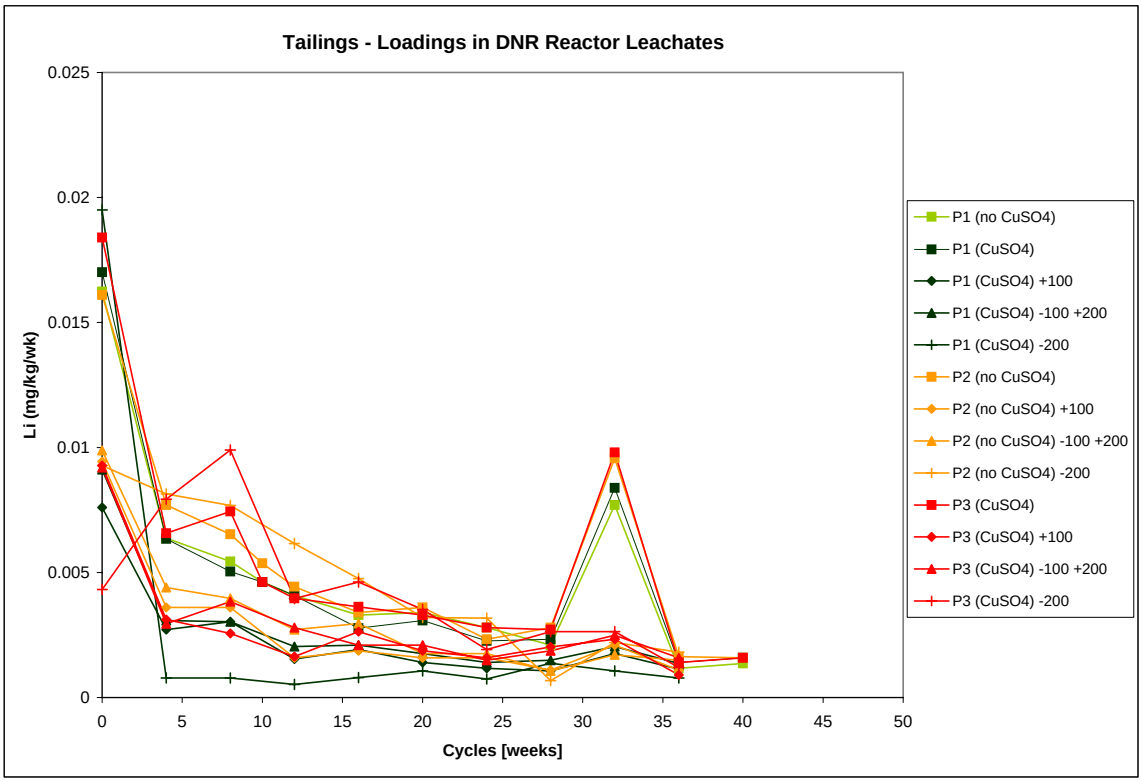
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.22



loadings_Lapakko.xls

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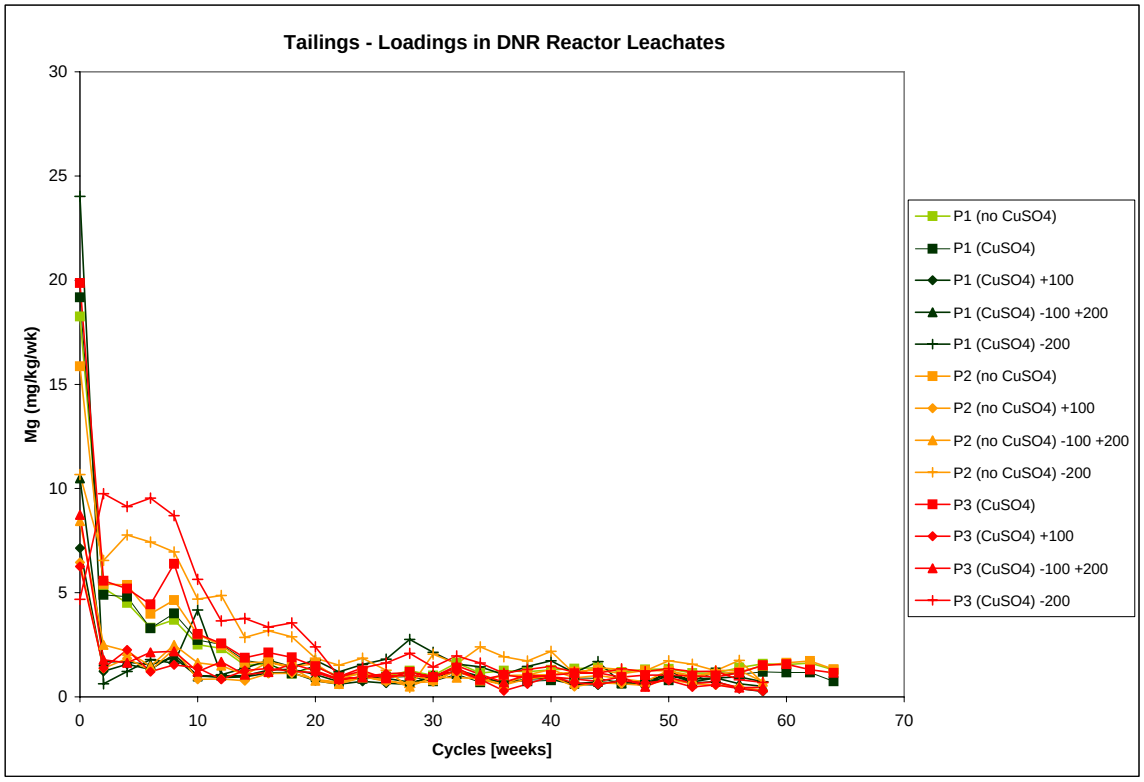
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.23



loadings_Lapakko.xls

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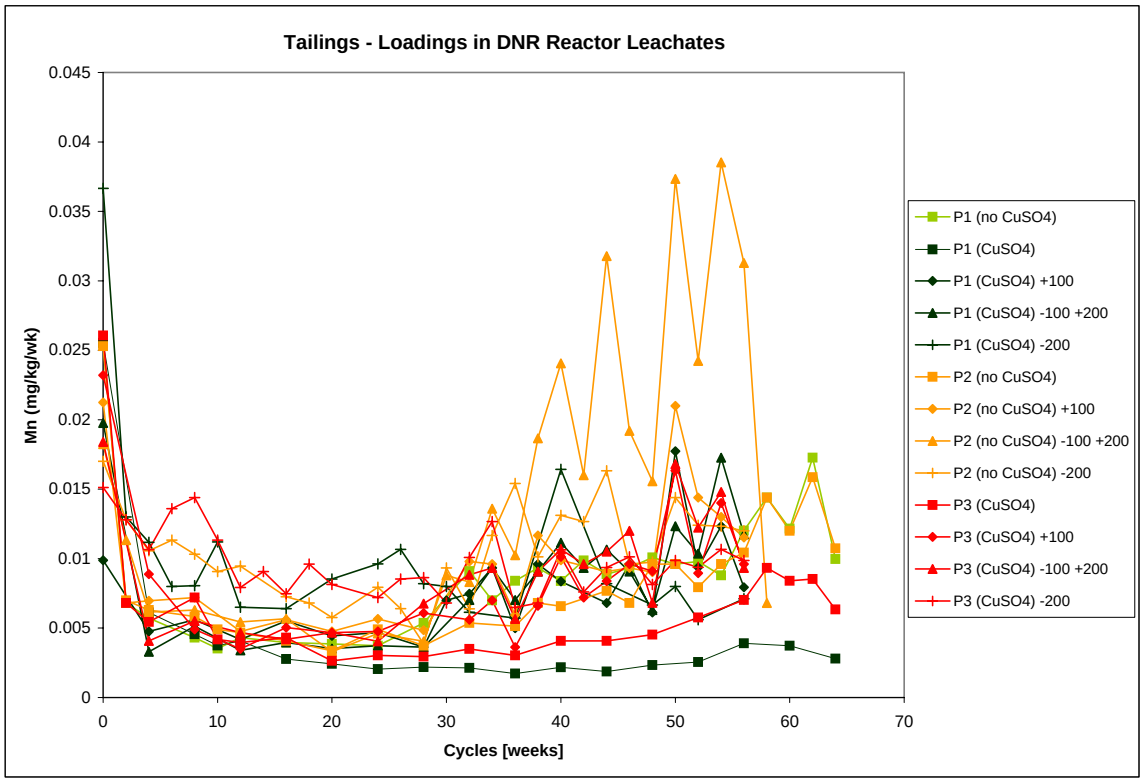
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.24



loadings_Lapakko.xls

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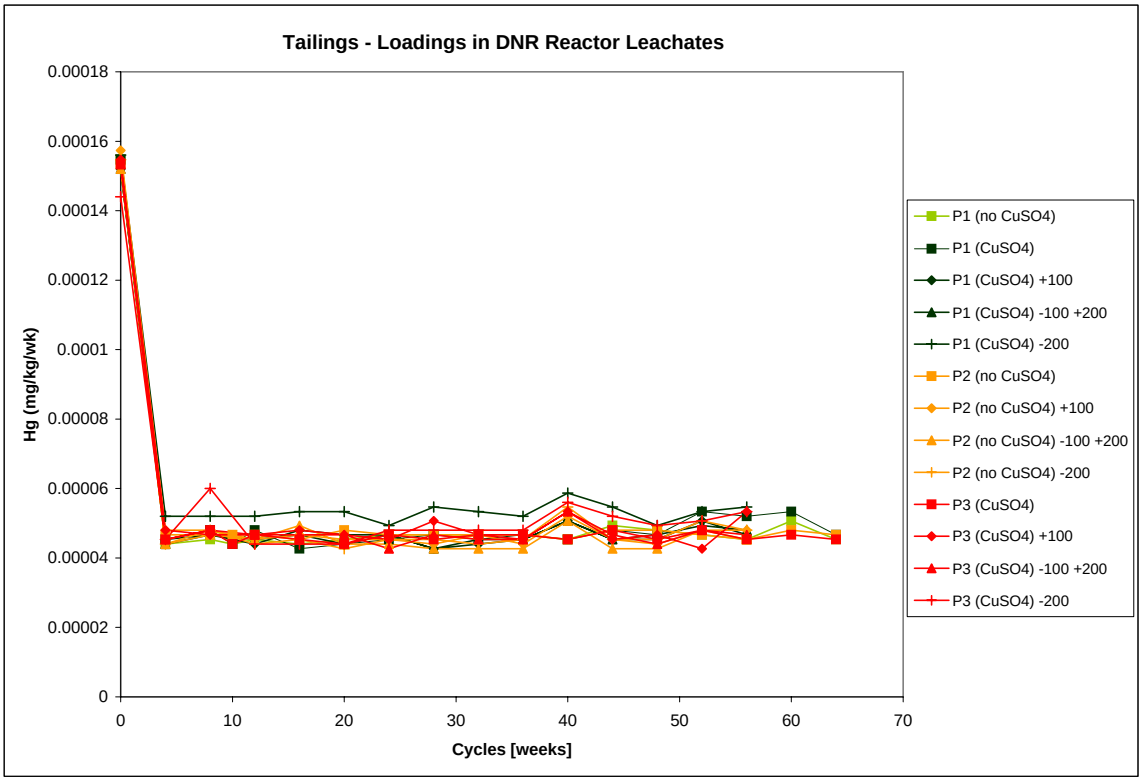
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.25



loadings_Lapakko.xls

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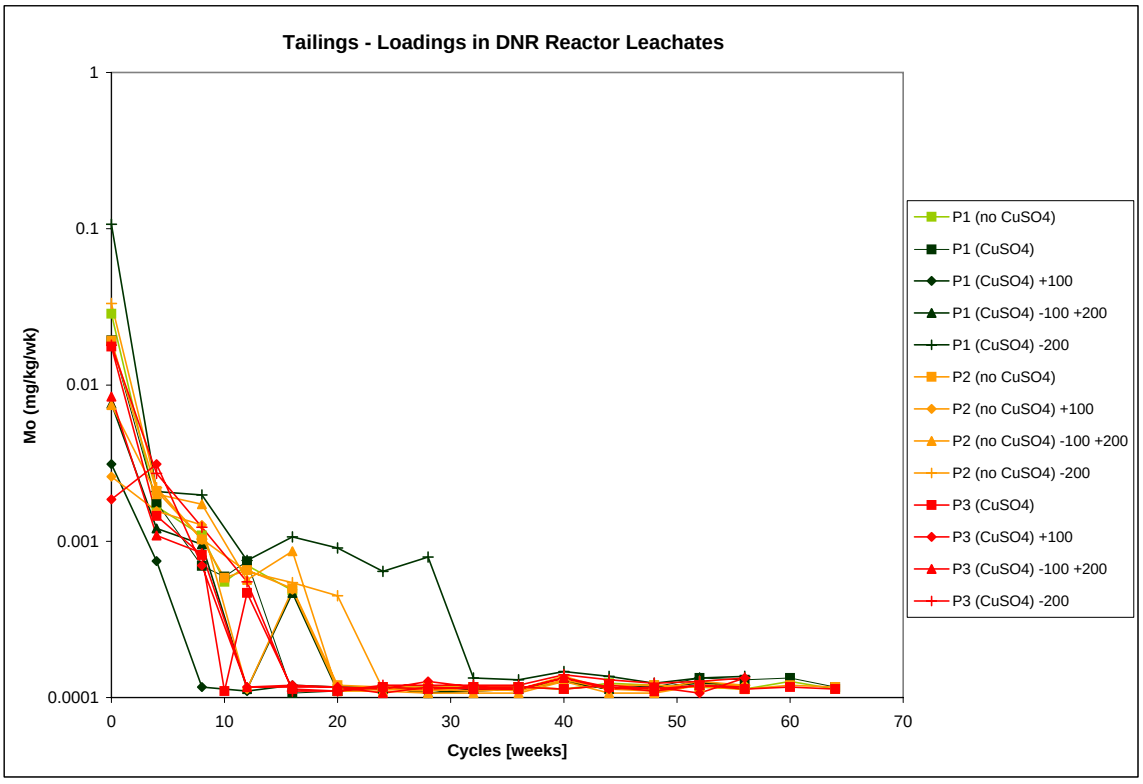
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.26



loadings_Lapakko.xls

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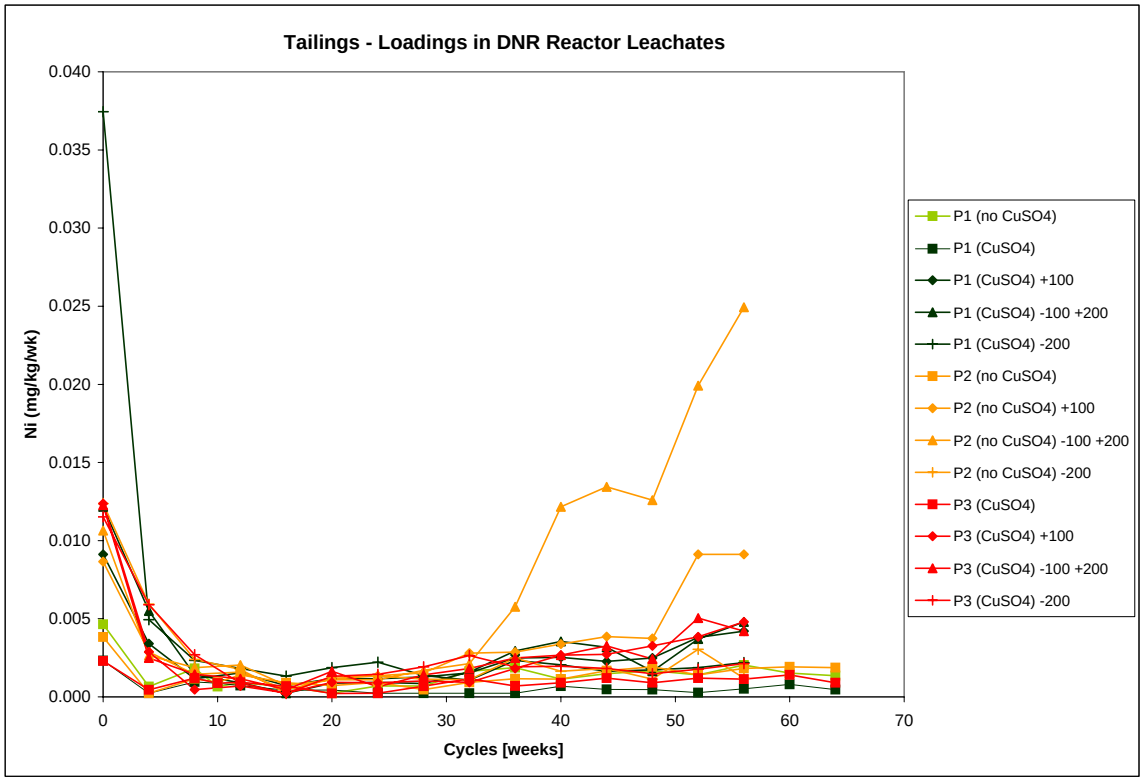
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.27



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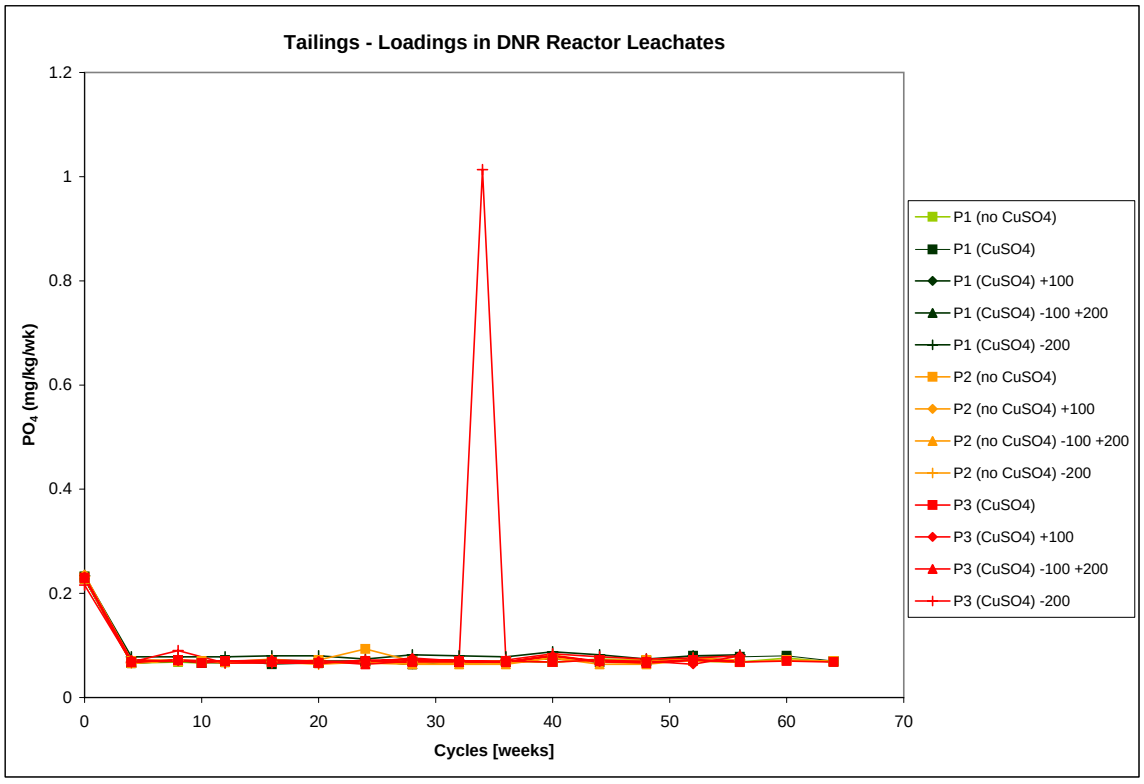
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.28



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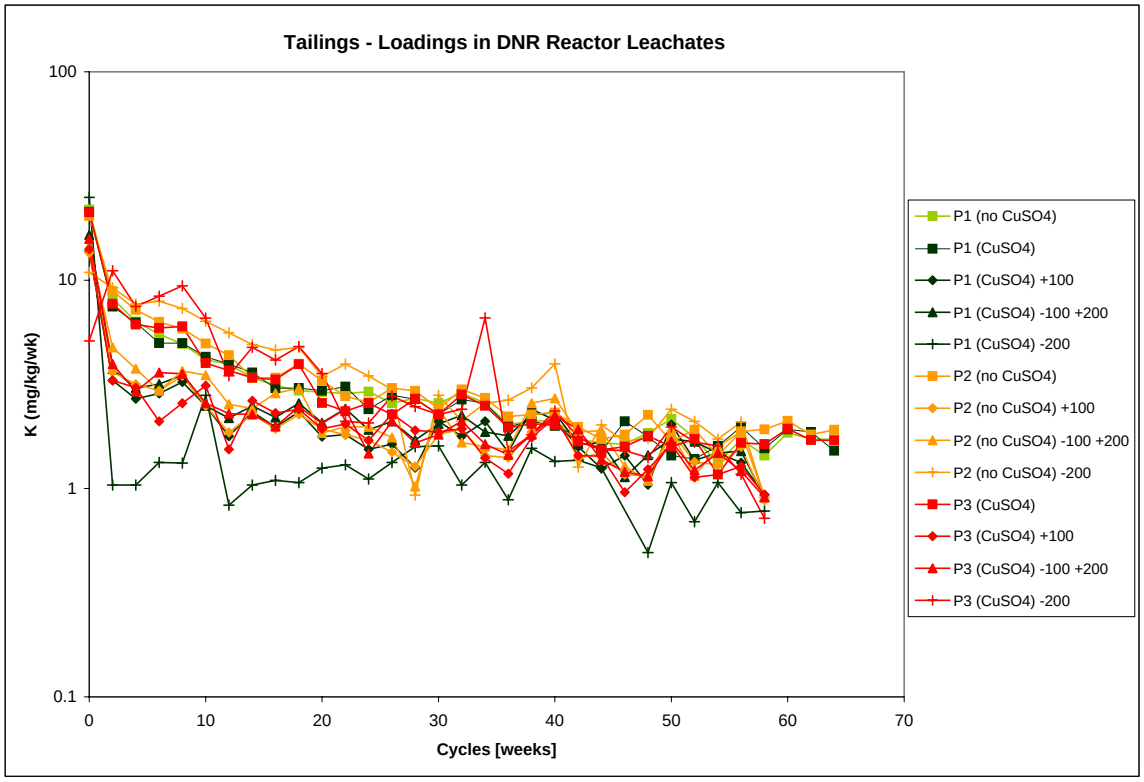
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.29



loadings_Lapakko.xls

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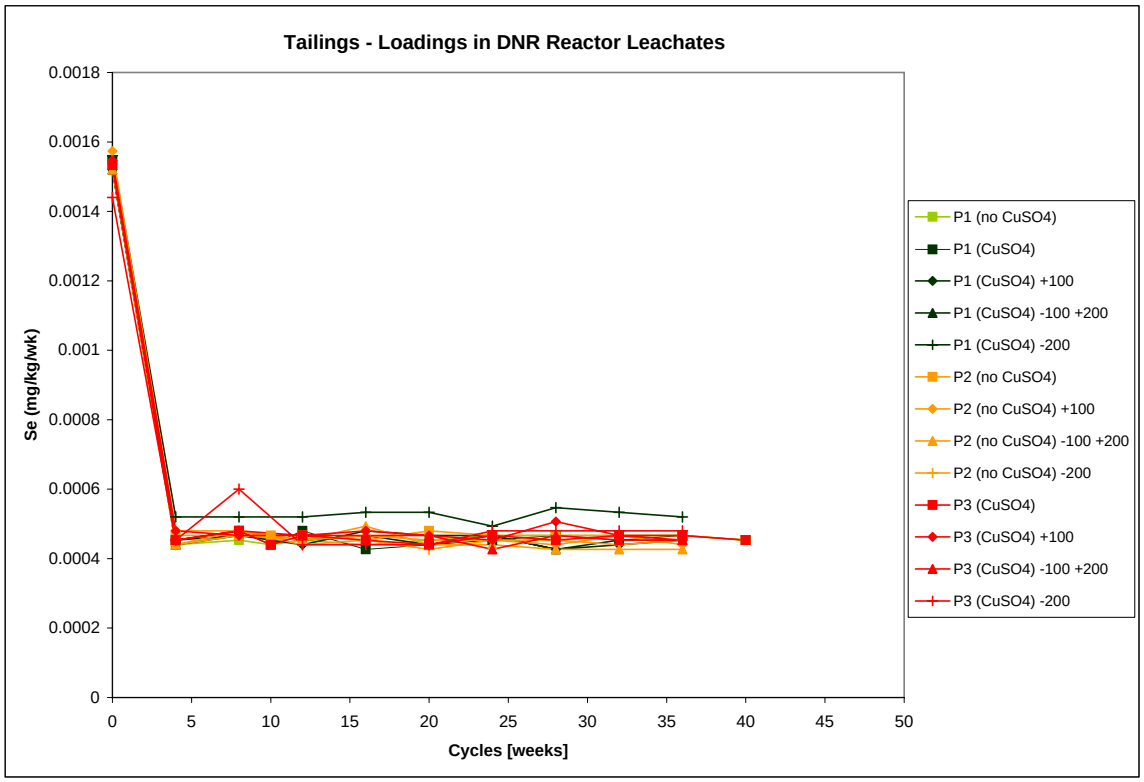
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.30



loadings_Lapakko.xls

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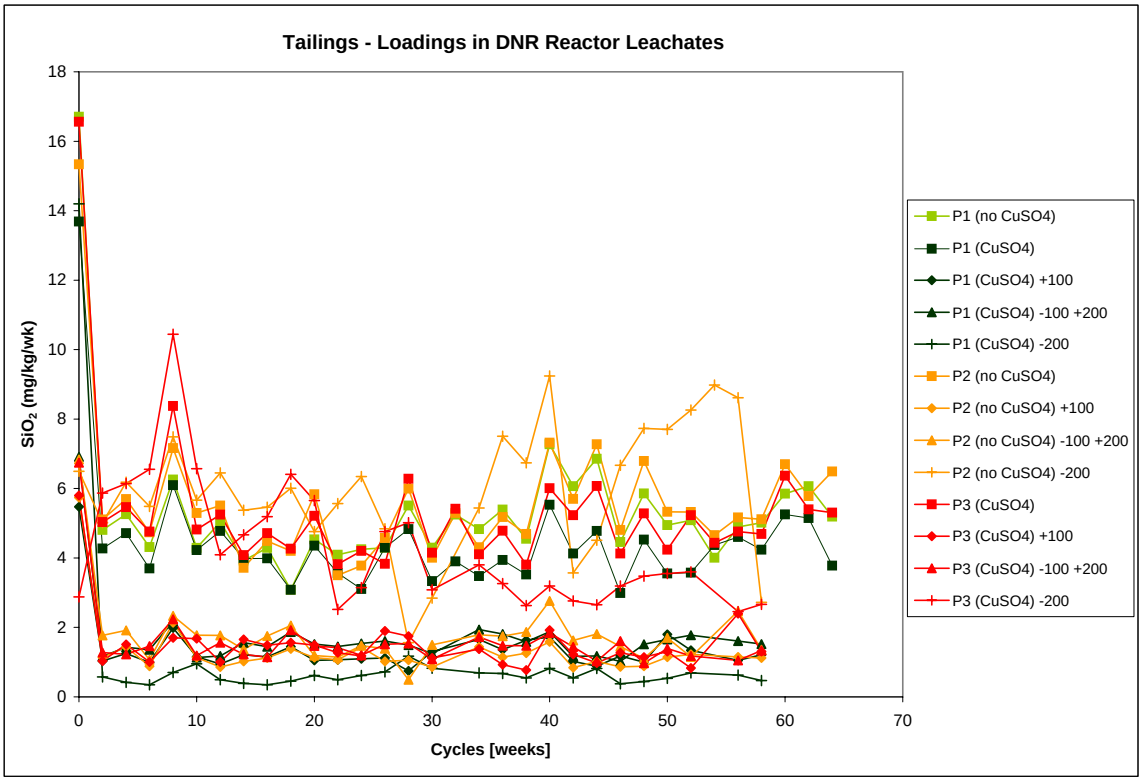
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.31



loadings_Lapakko.xls

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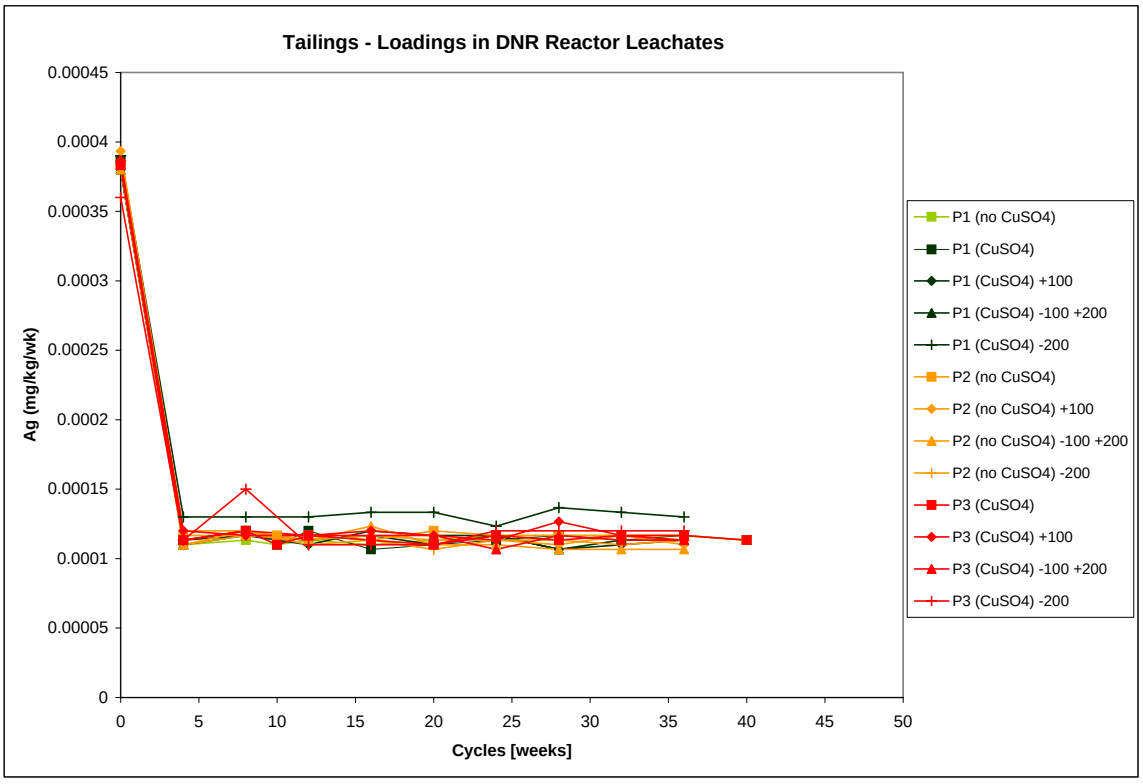
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.32



loadings_Lapakko.xls

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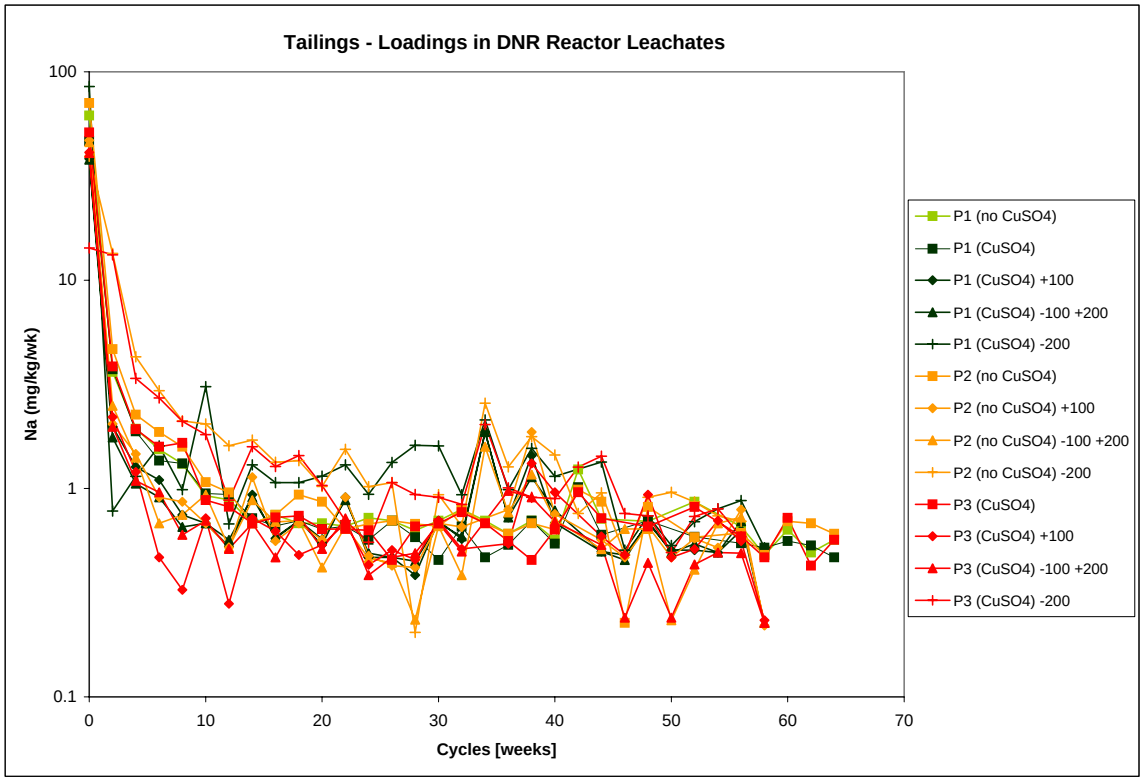
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.33



loadings_Lapakko.xls

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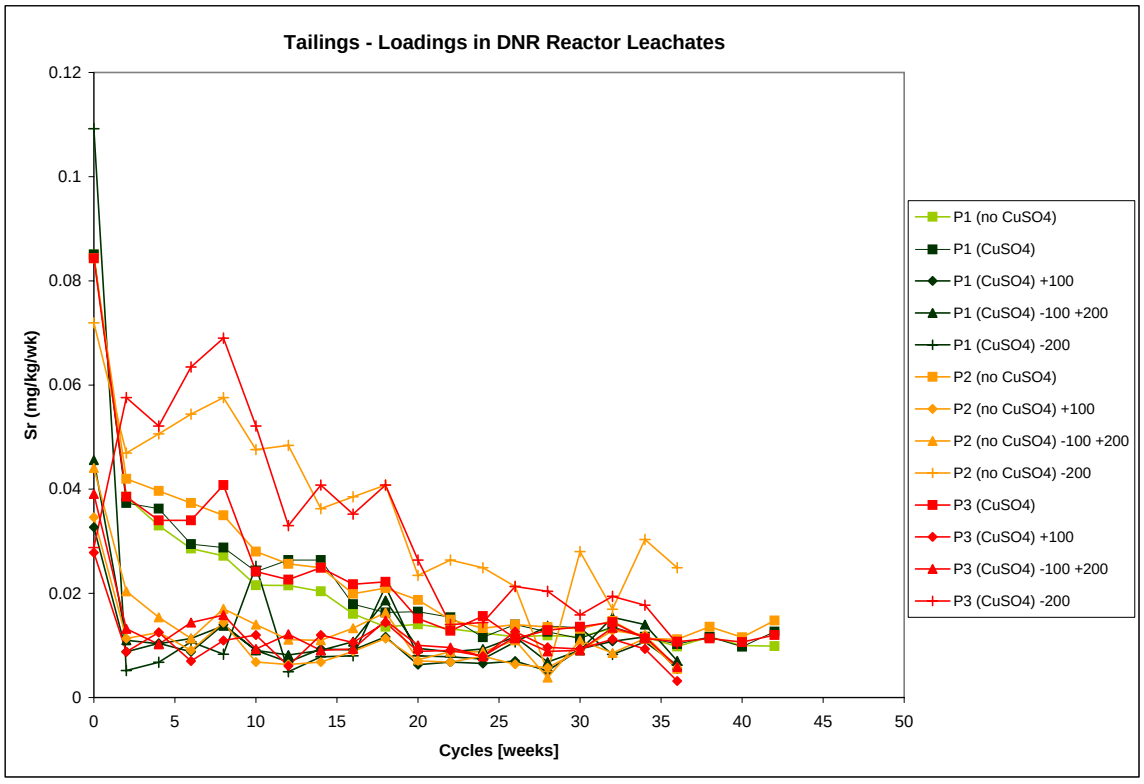
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.34



loadings_Lapakko.xls

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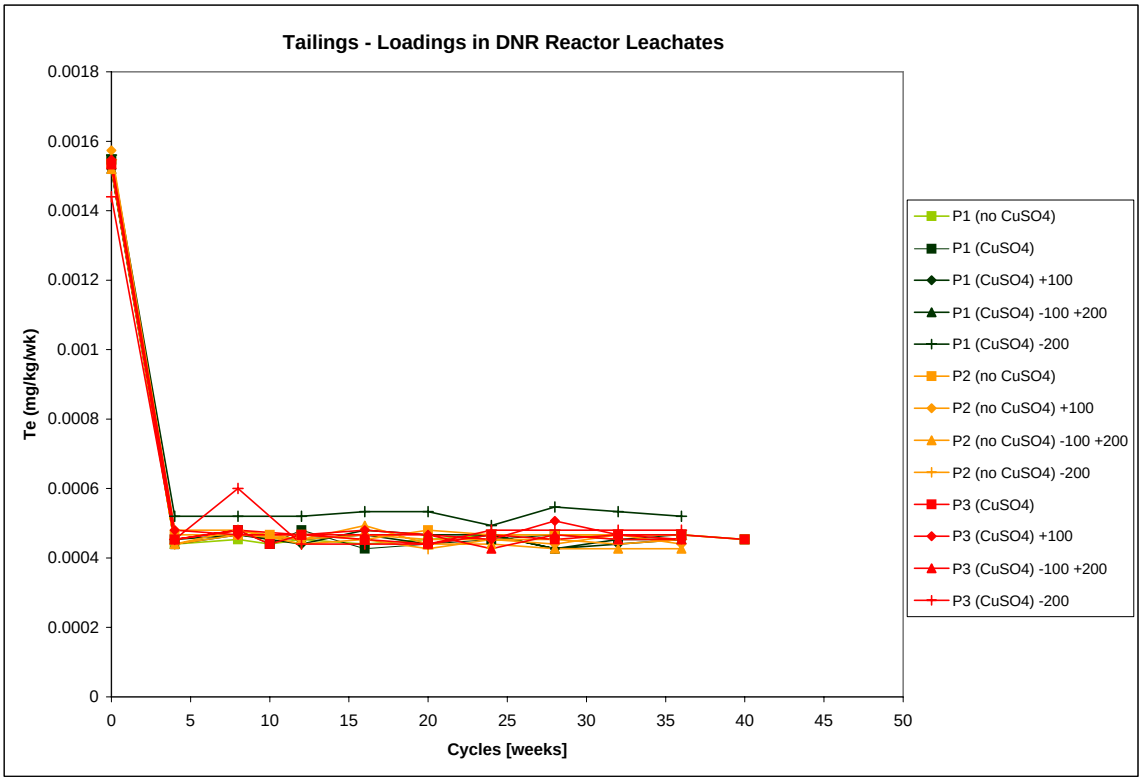
Tailings - Concentrations in DNR Reactors

Graph C.2.35



loadings_Lapakko.xls

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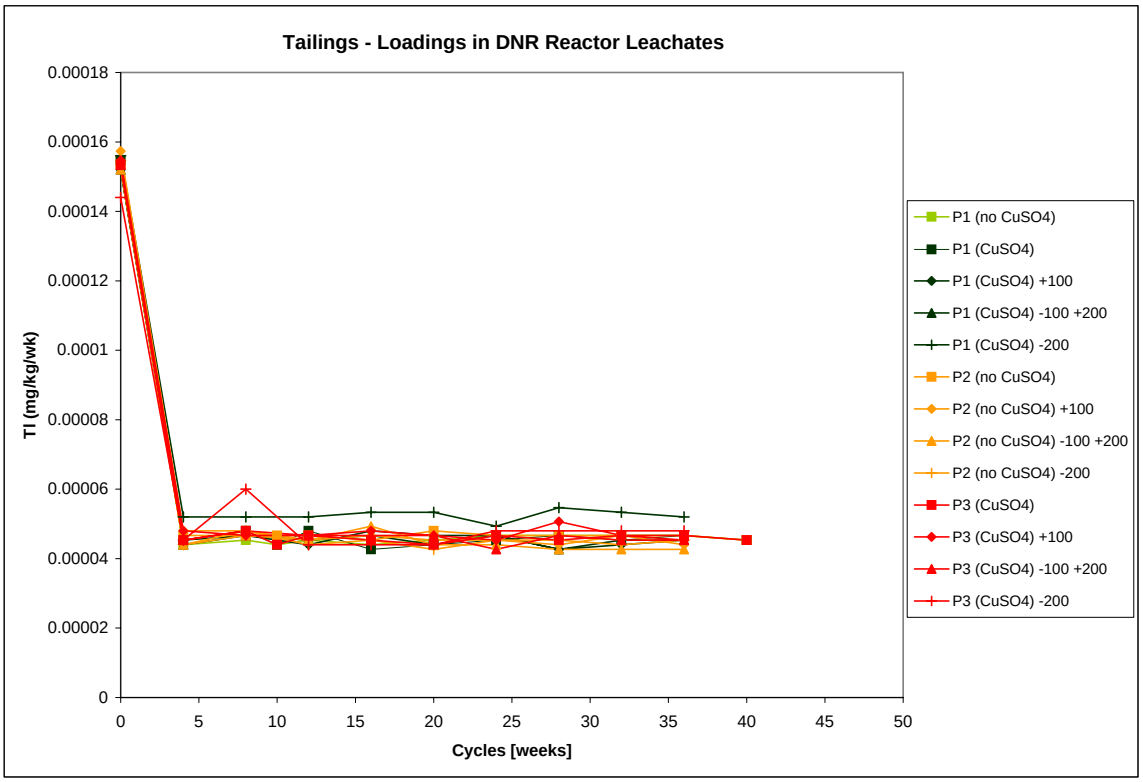
Tailings - Concentrations in DNR Reactors

Graph C.2.36



loadings_Lapakko.xls

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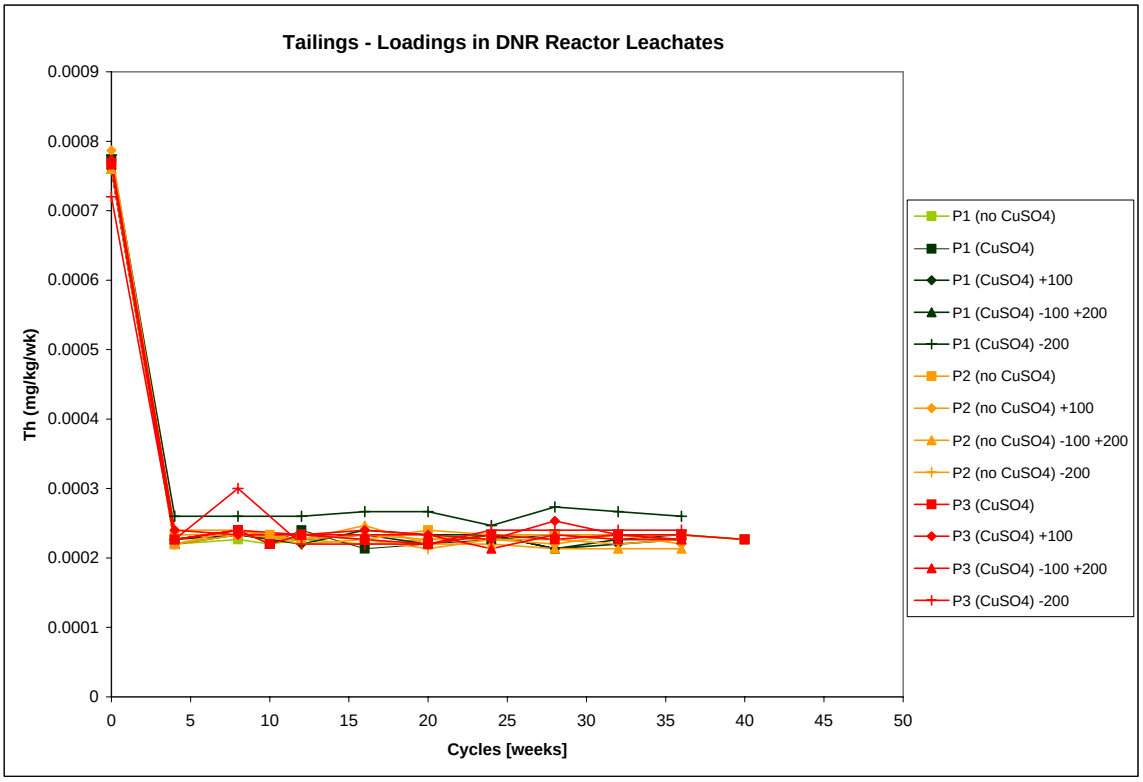
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.37



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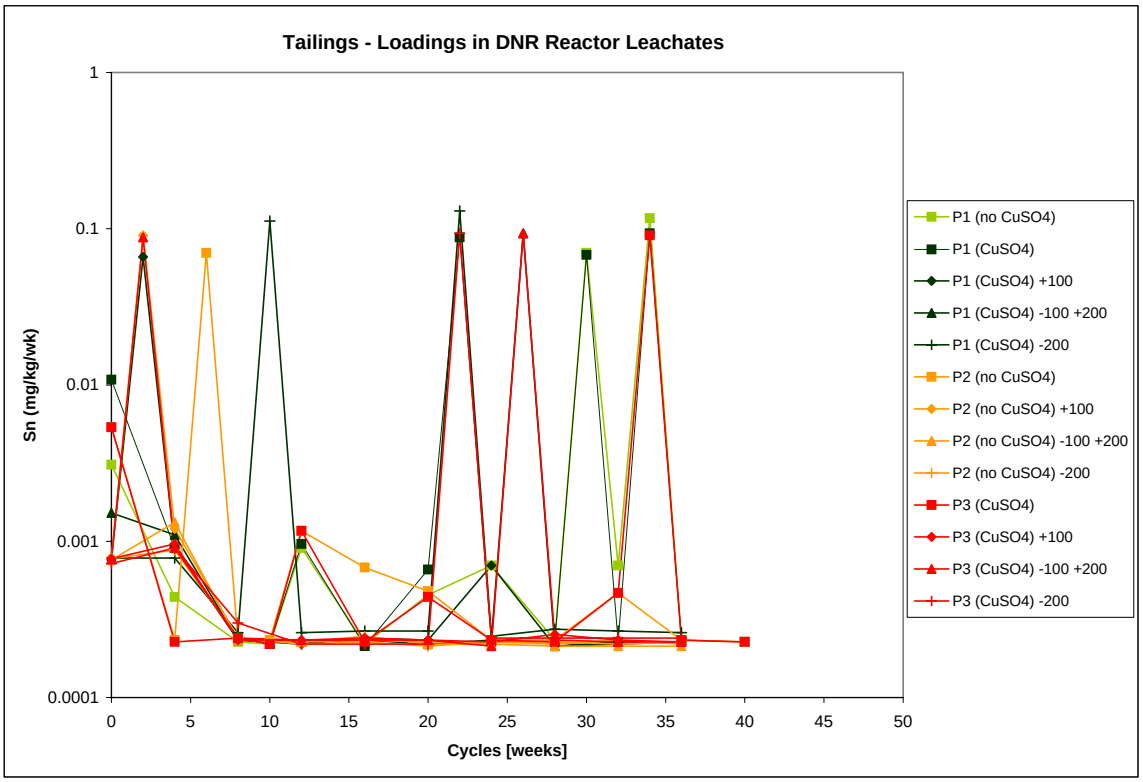
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.38



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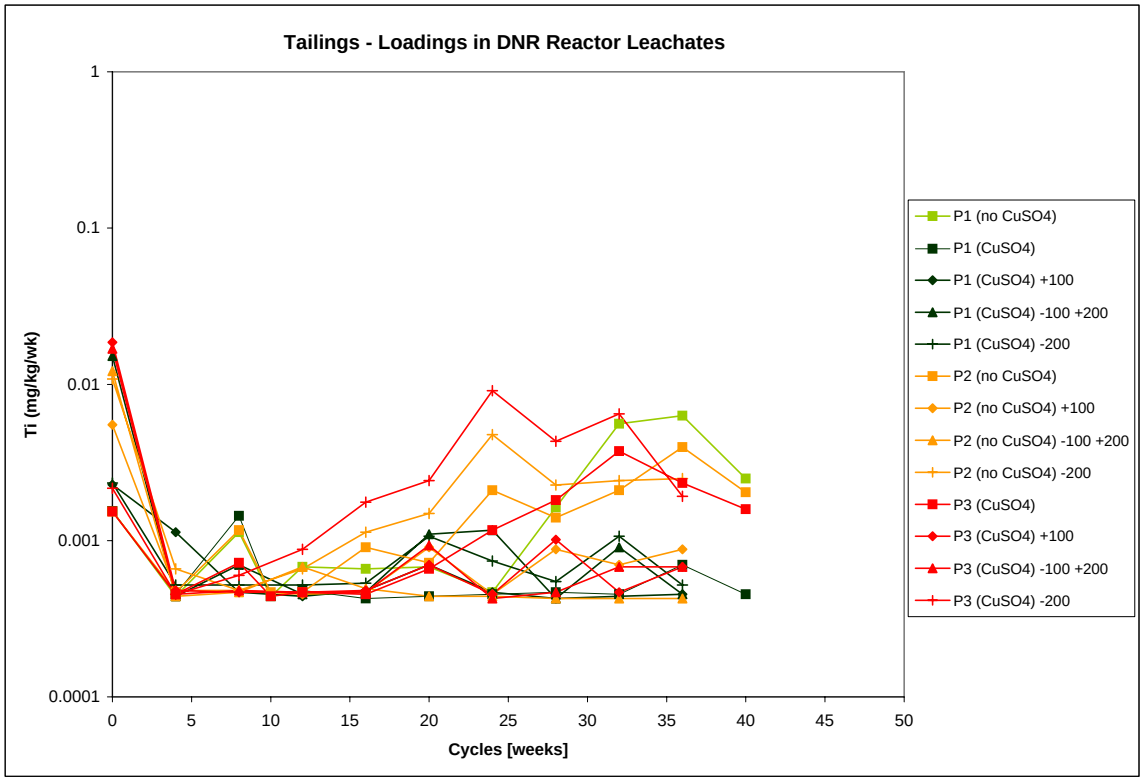
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.39



loadings_Lapakko.xls

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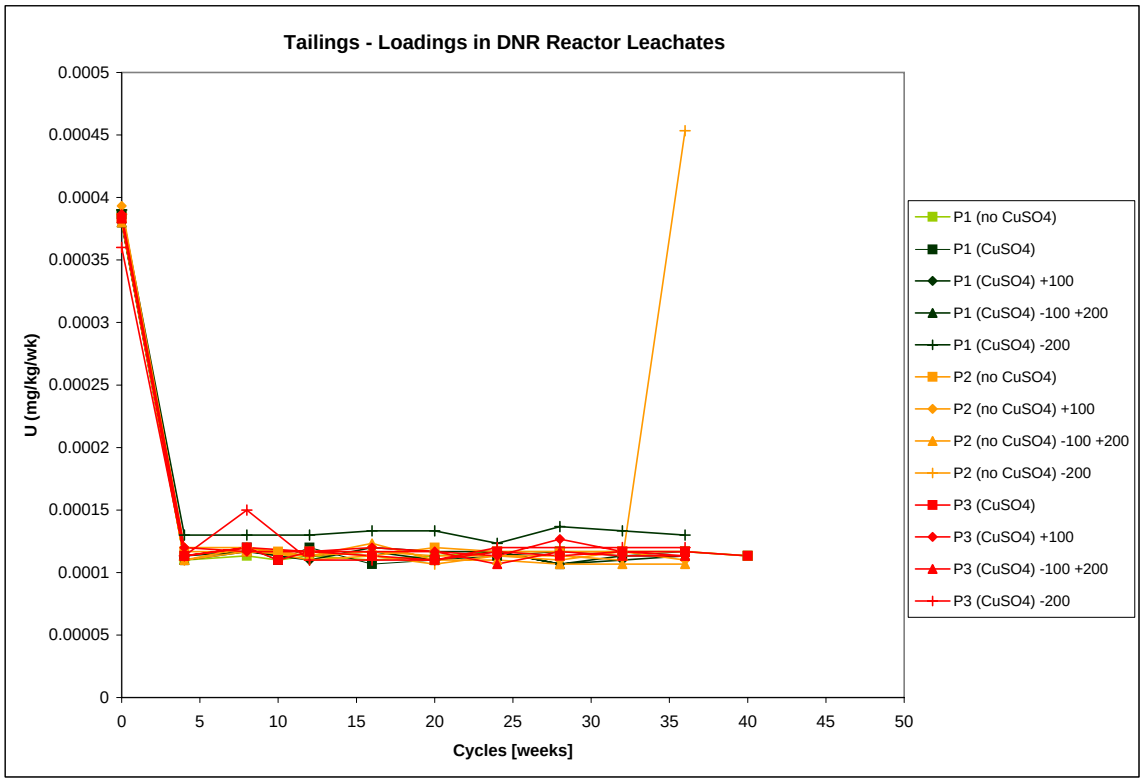
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.40



loadings_Lapakko.xls

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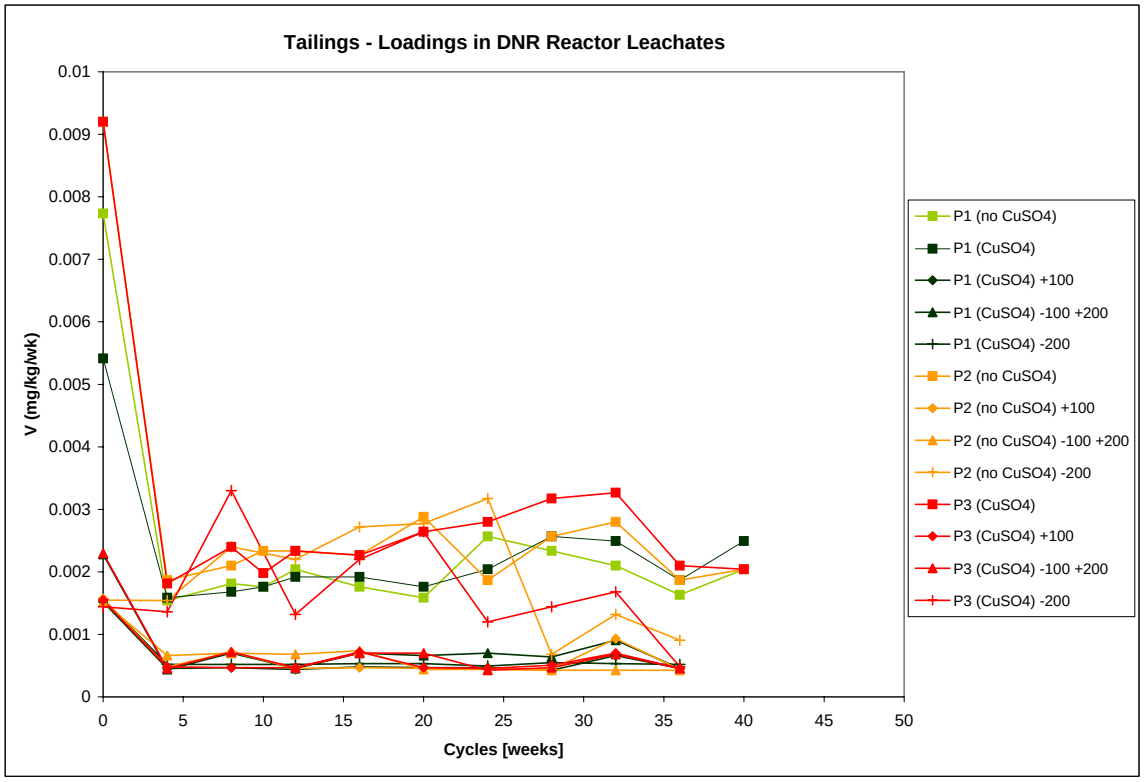
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.41



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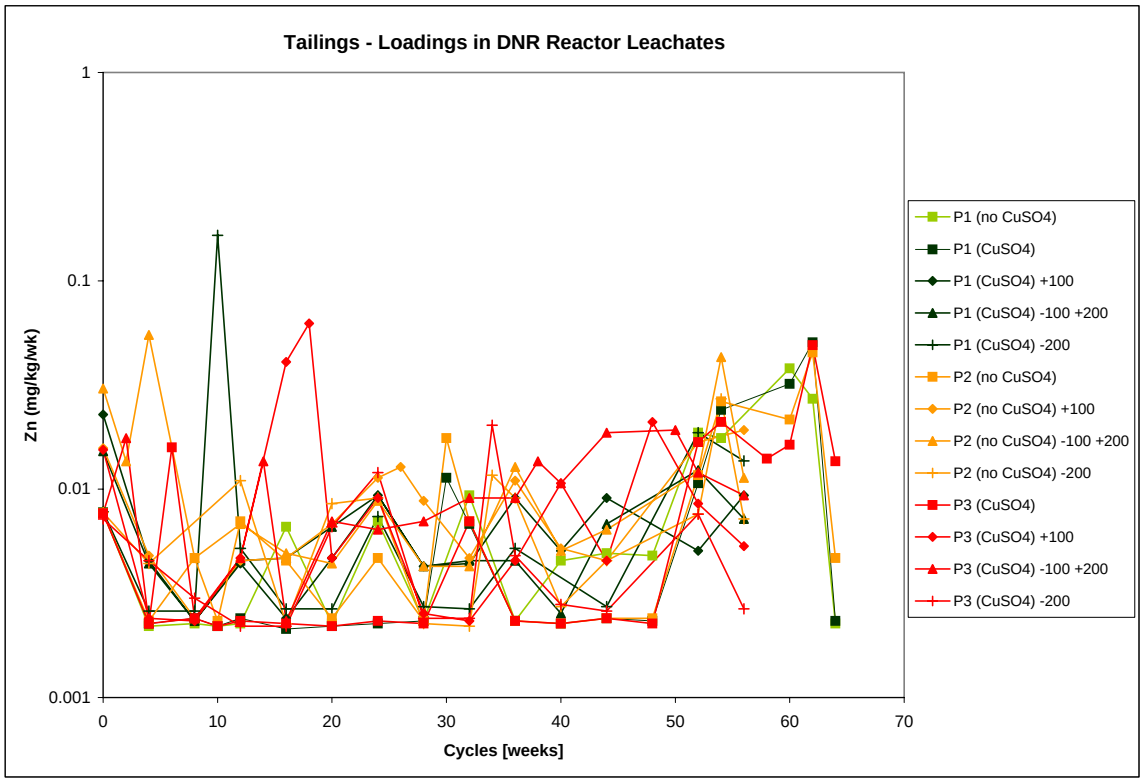
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.42



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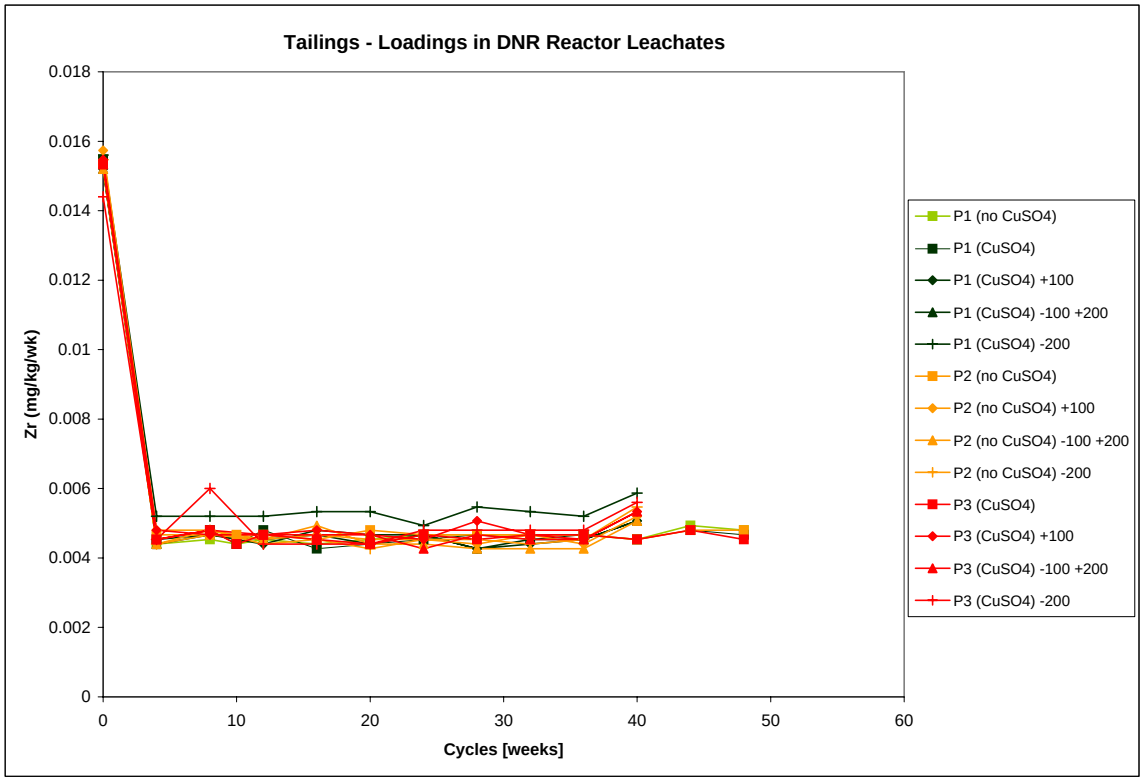
Attachment C.2
Tailings - Concentrations in DNR Reactors

Graph C.2.43



loadings_Lapakko.xls

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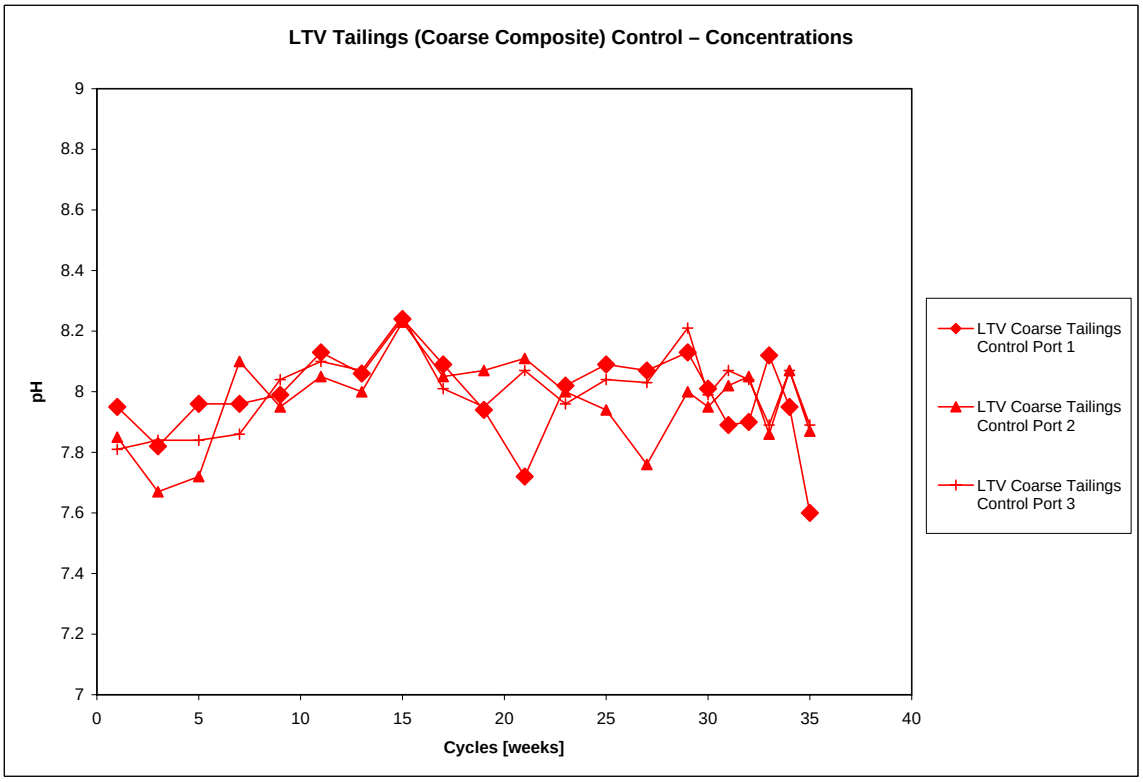
Appendix C.3

LTVSMC Contact Experiment

Appendix C.3
LTVSMC Contact Experiment

Graph C.3.1



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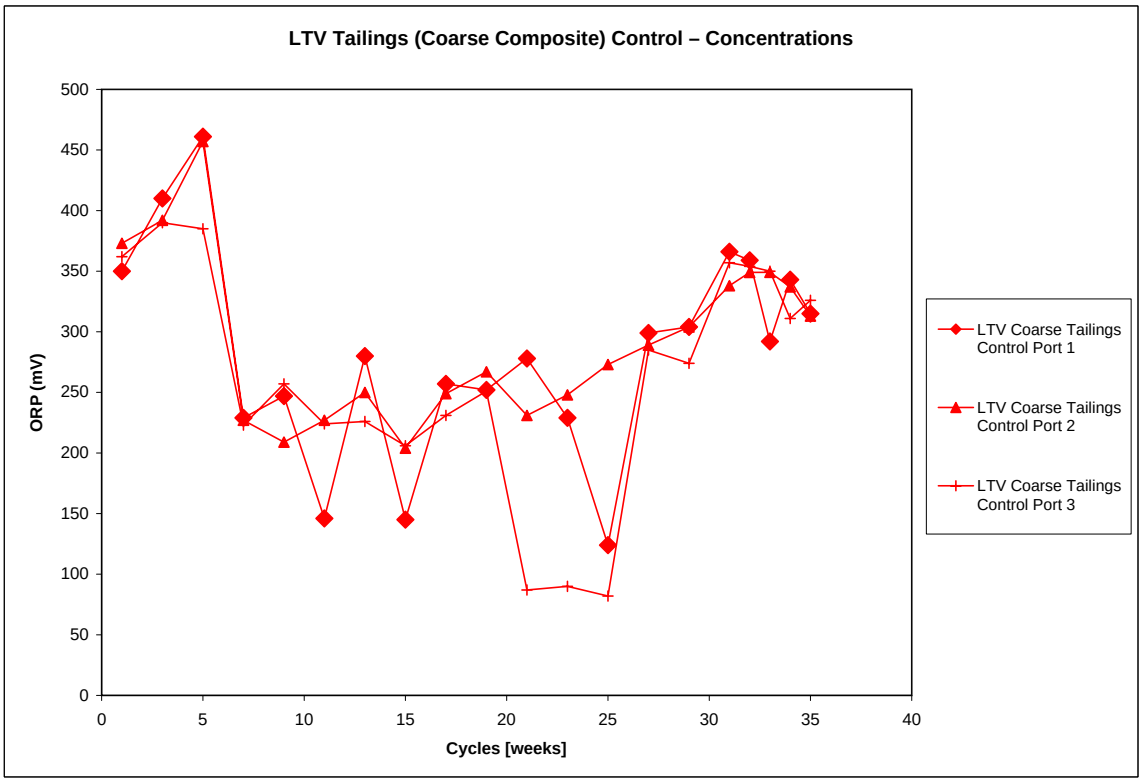
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.2



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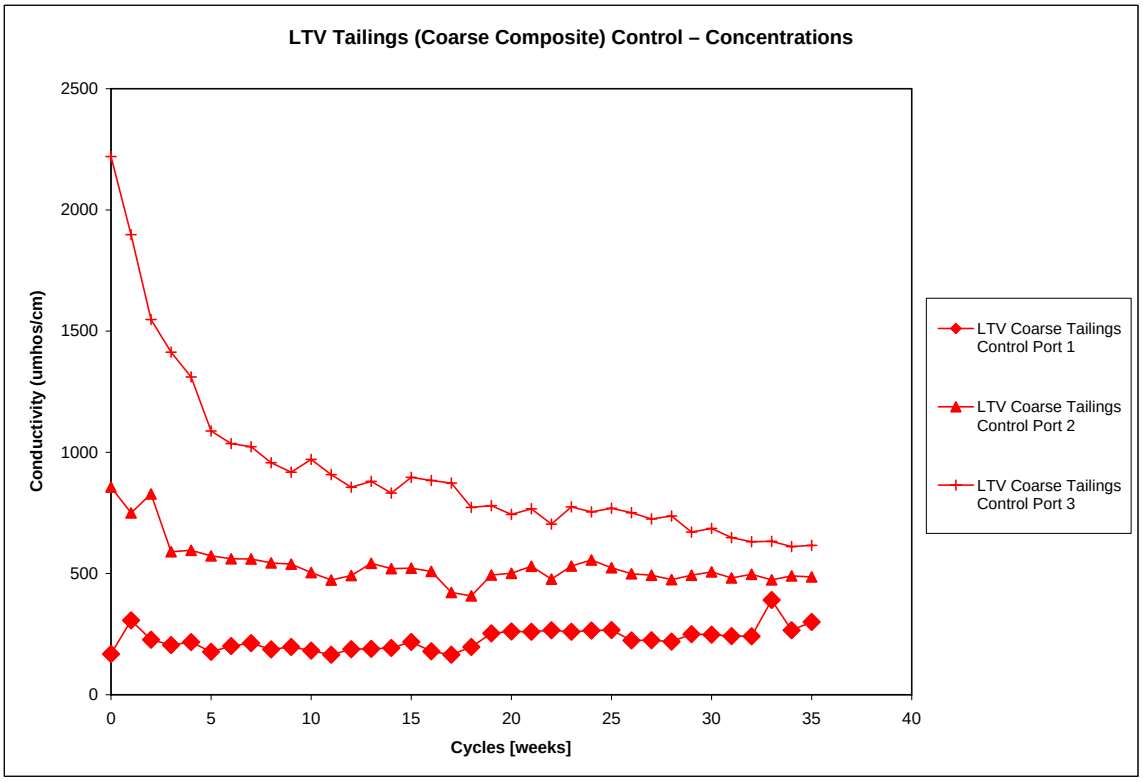
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.3



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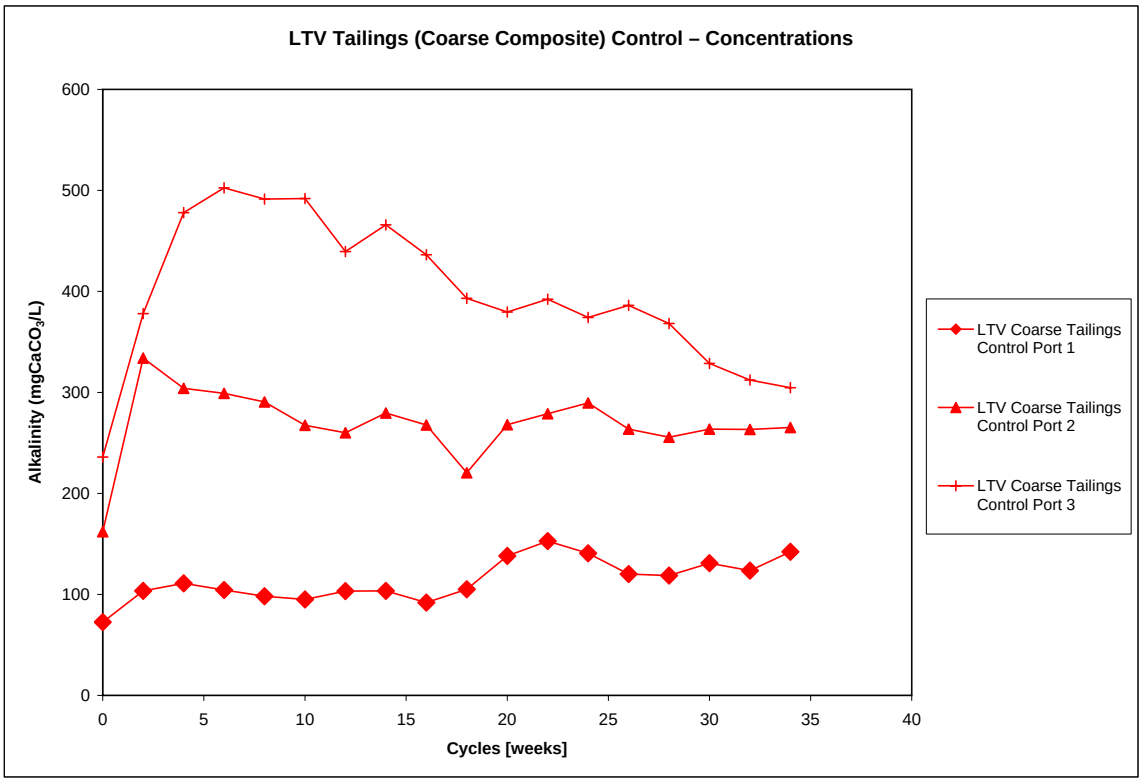
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.4



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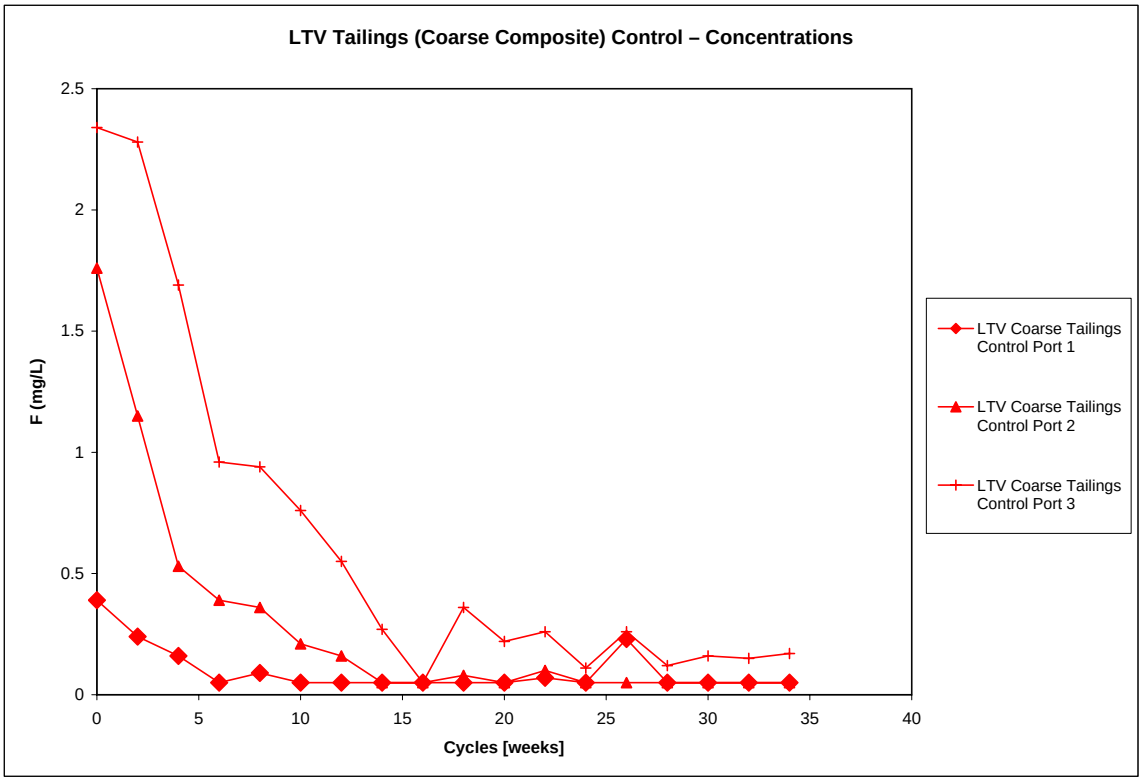
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.5



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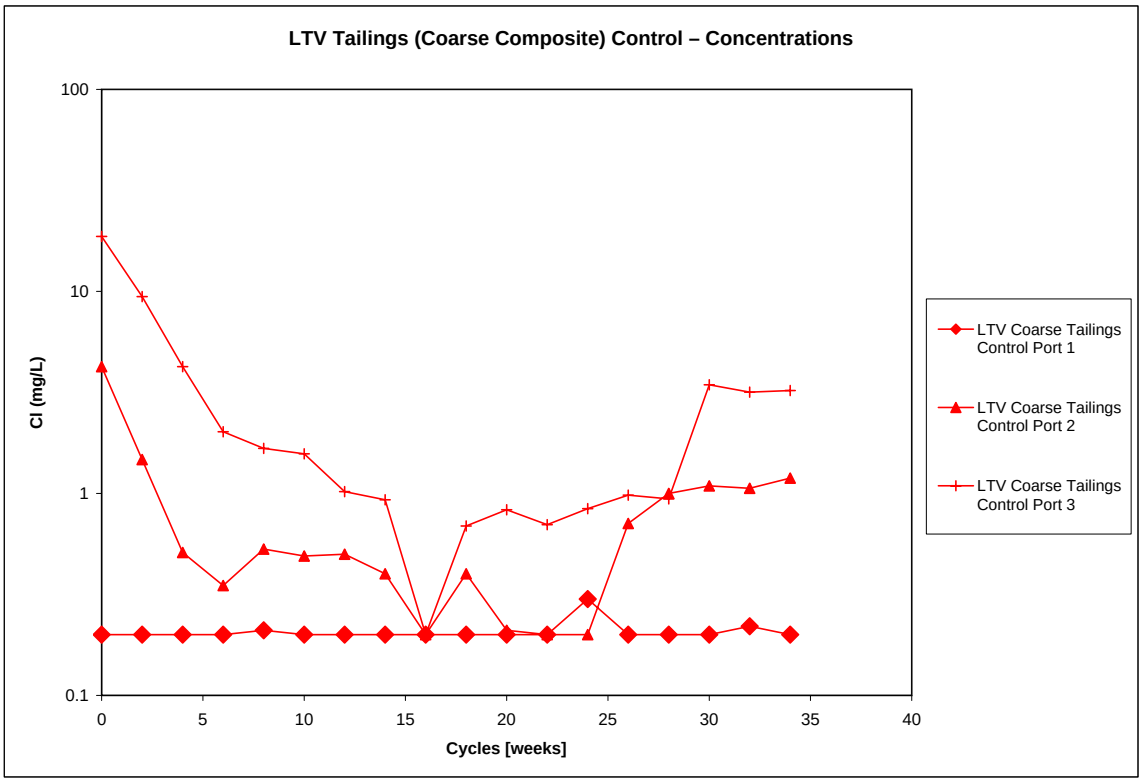
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.6



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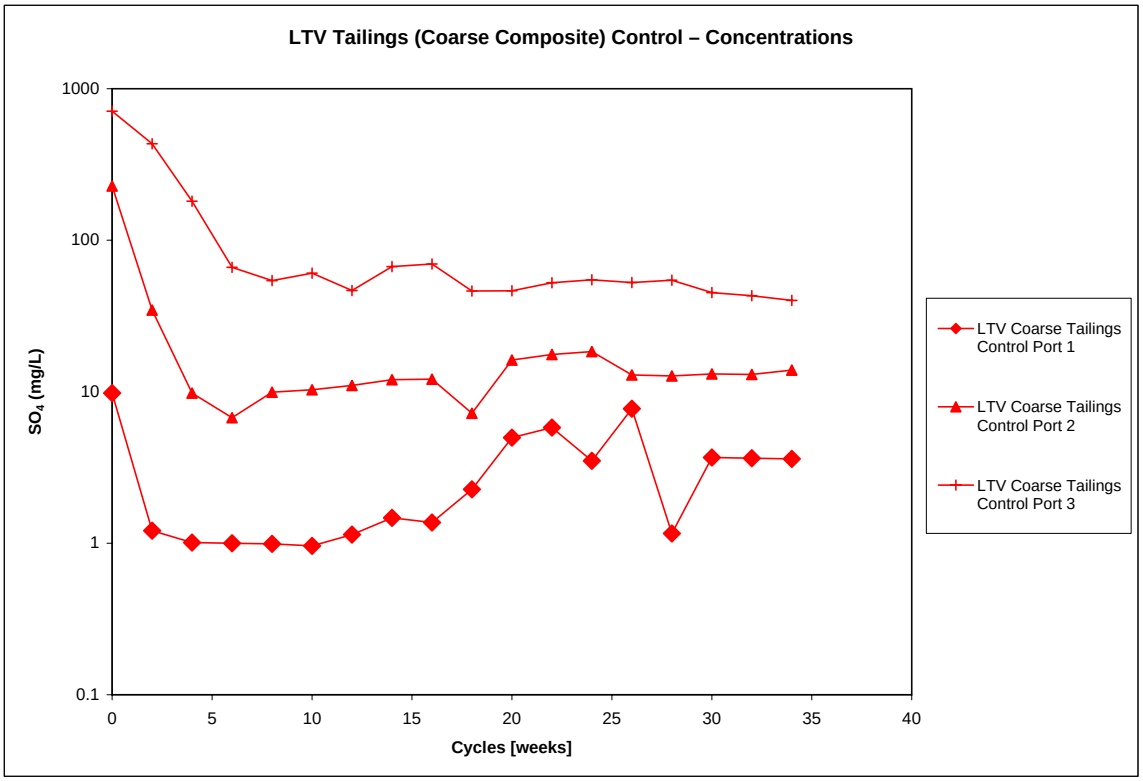
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.7



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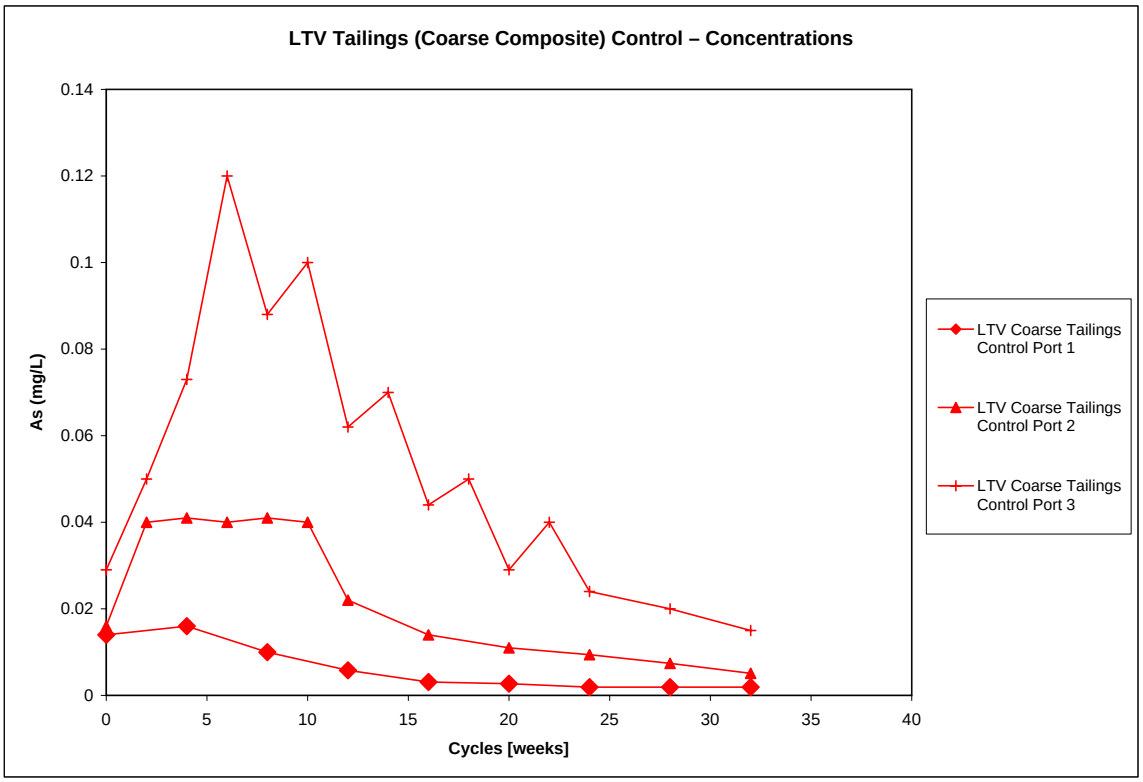
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.8



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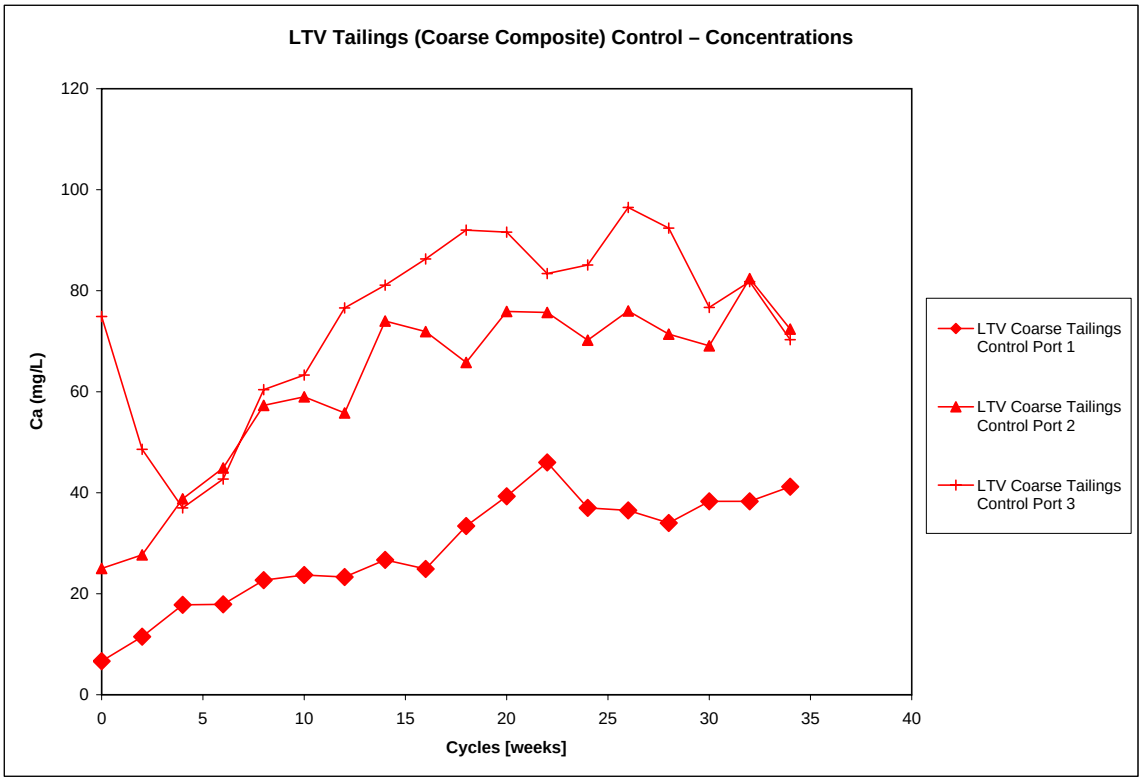
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.9



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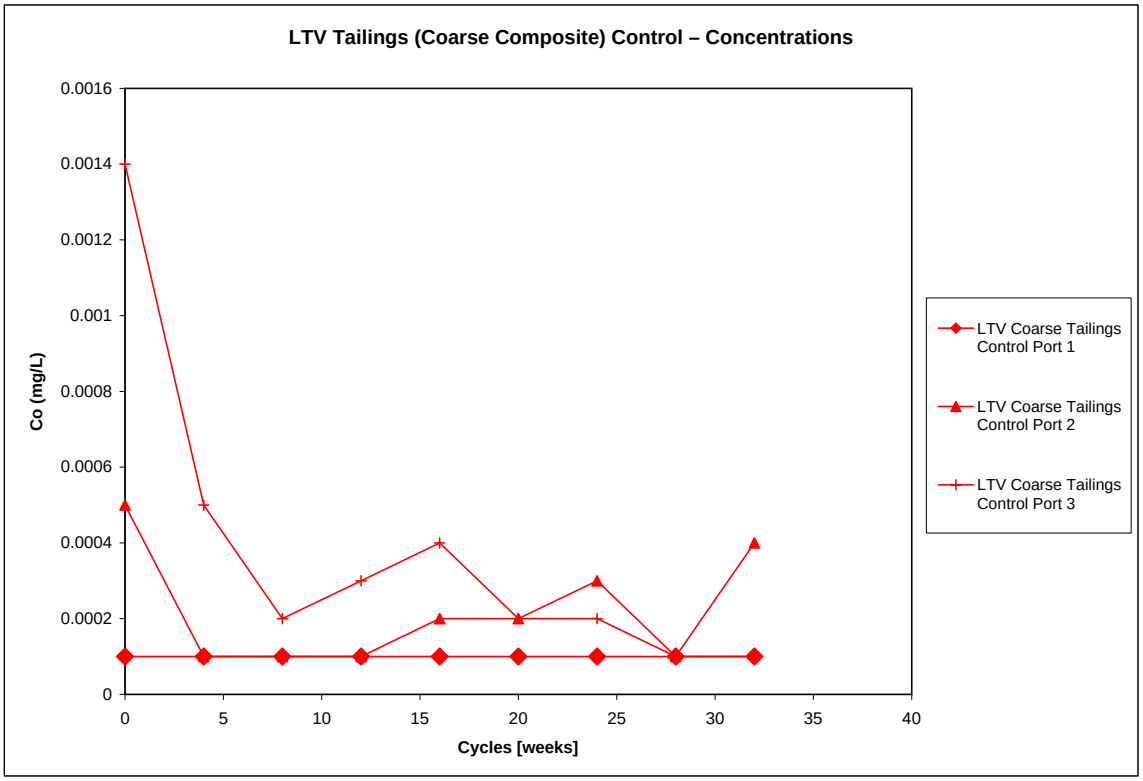
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.10



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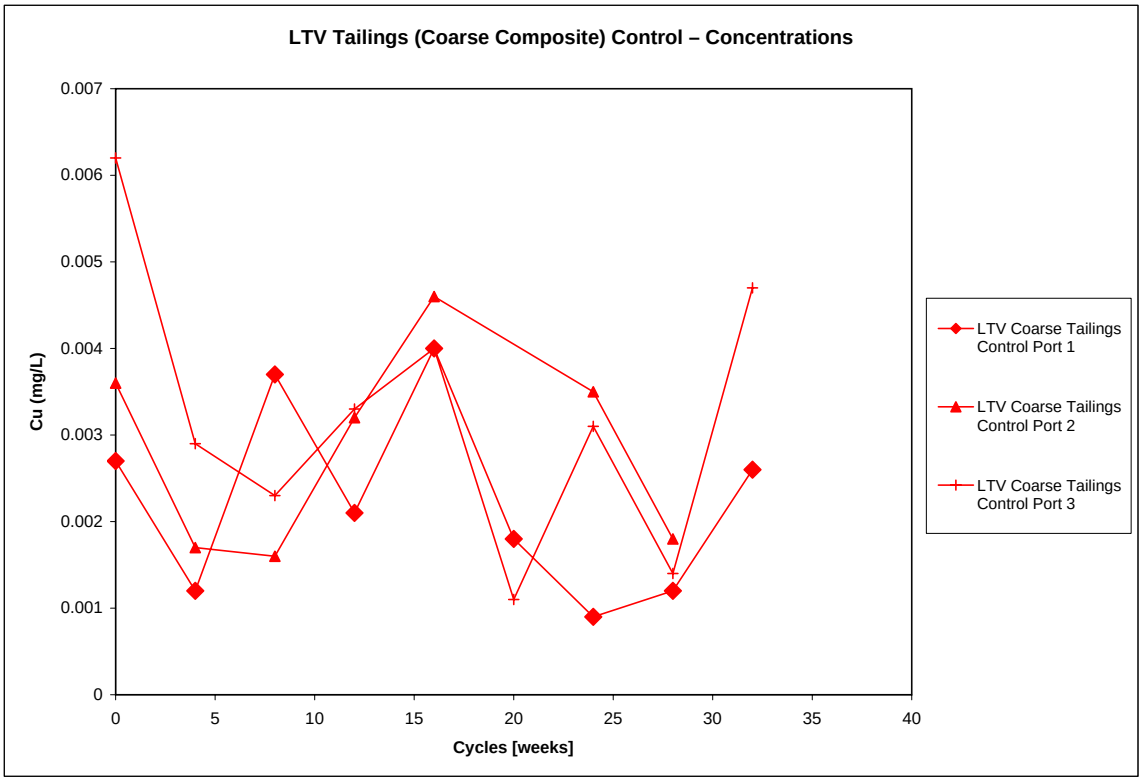
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.11



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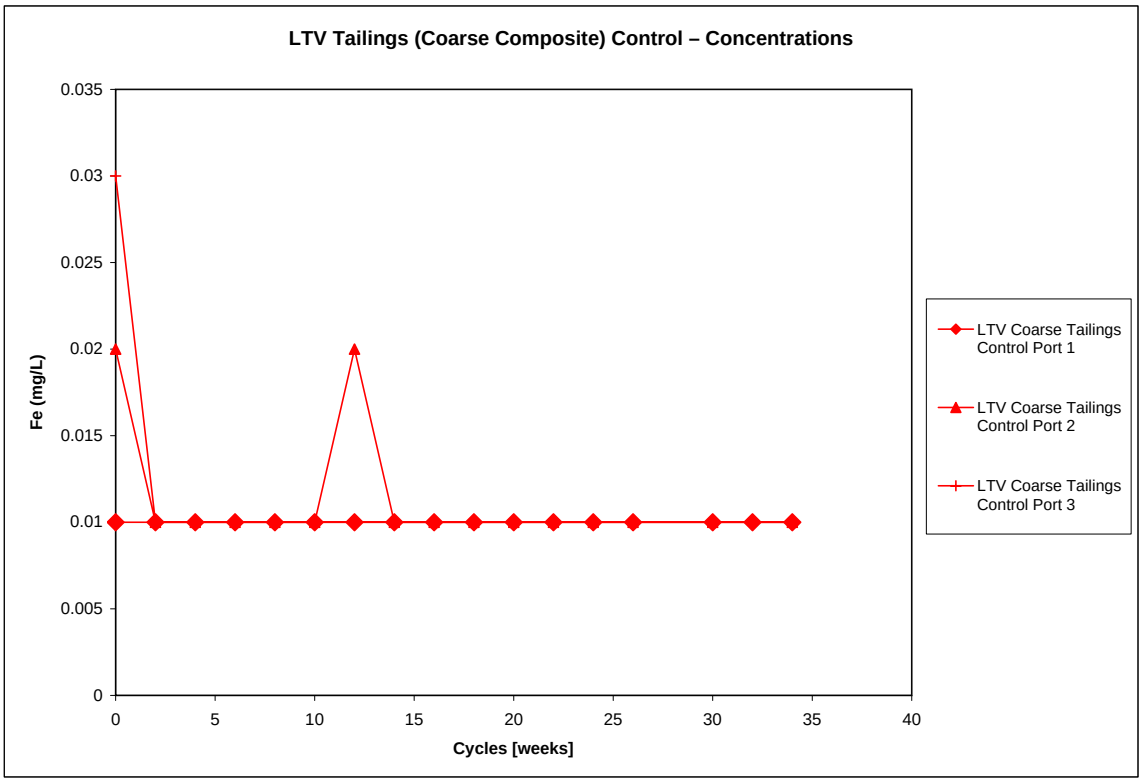
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.12



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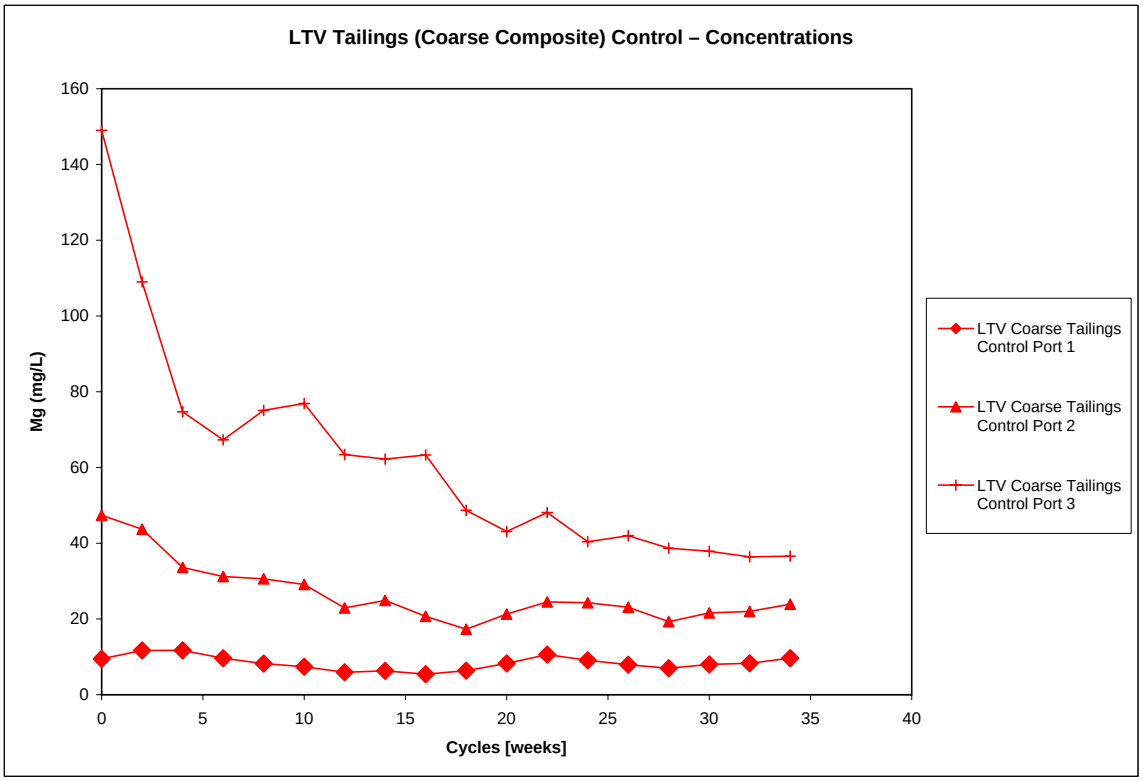
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.13



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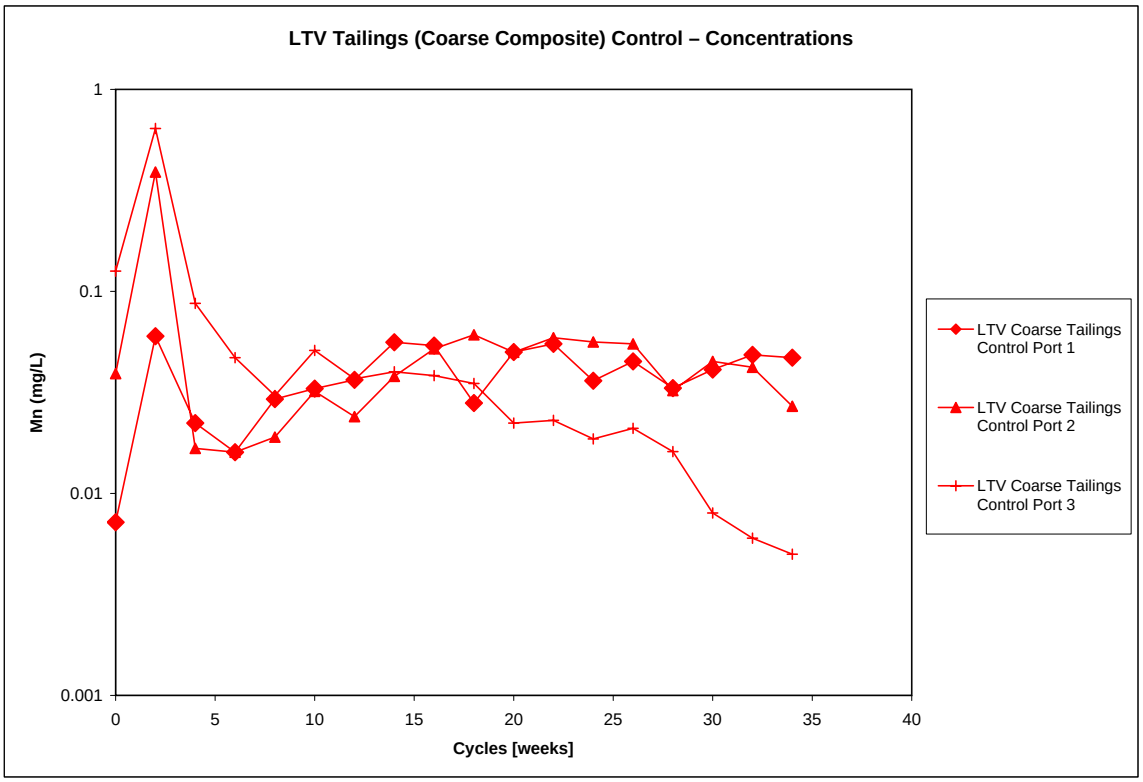
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.14



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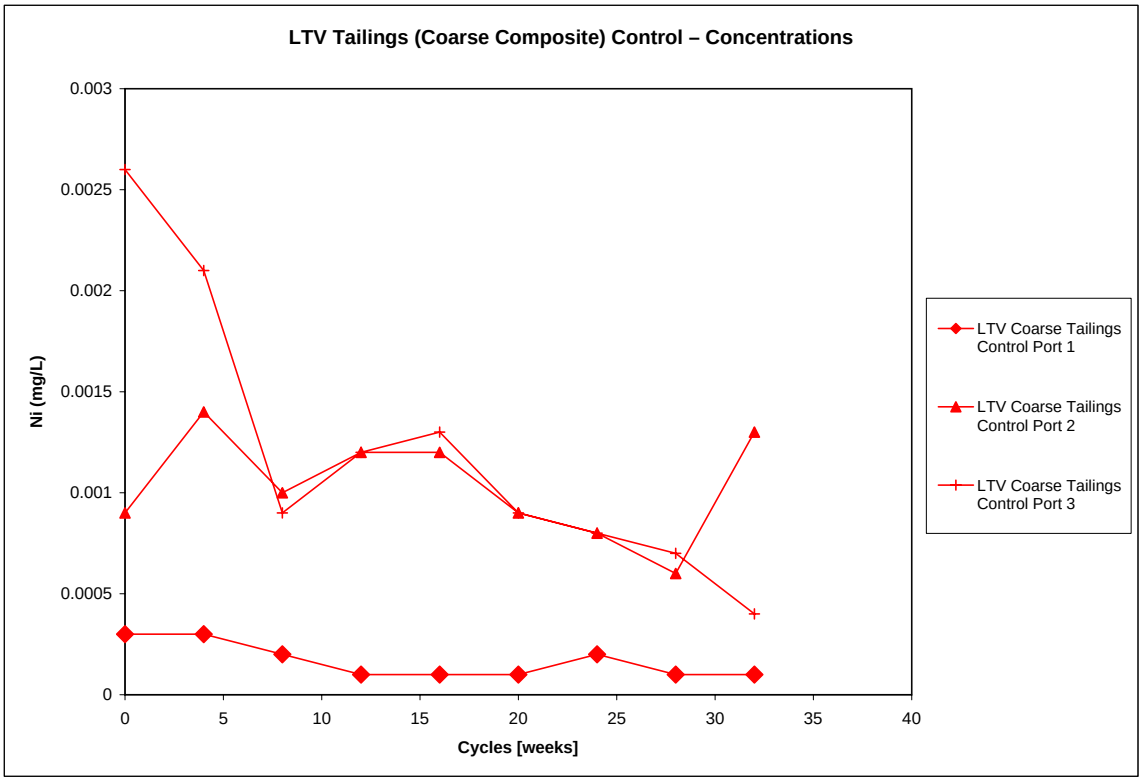
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.15



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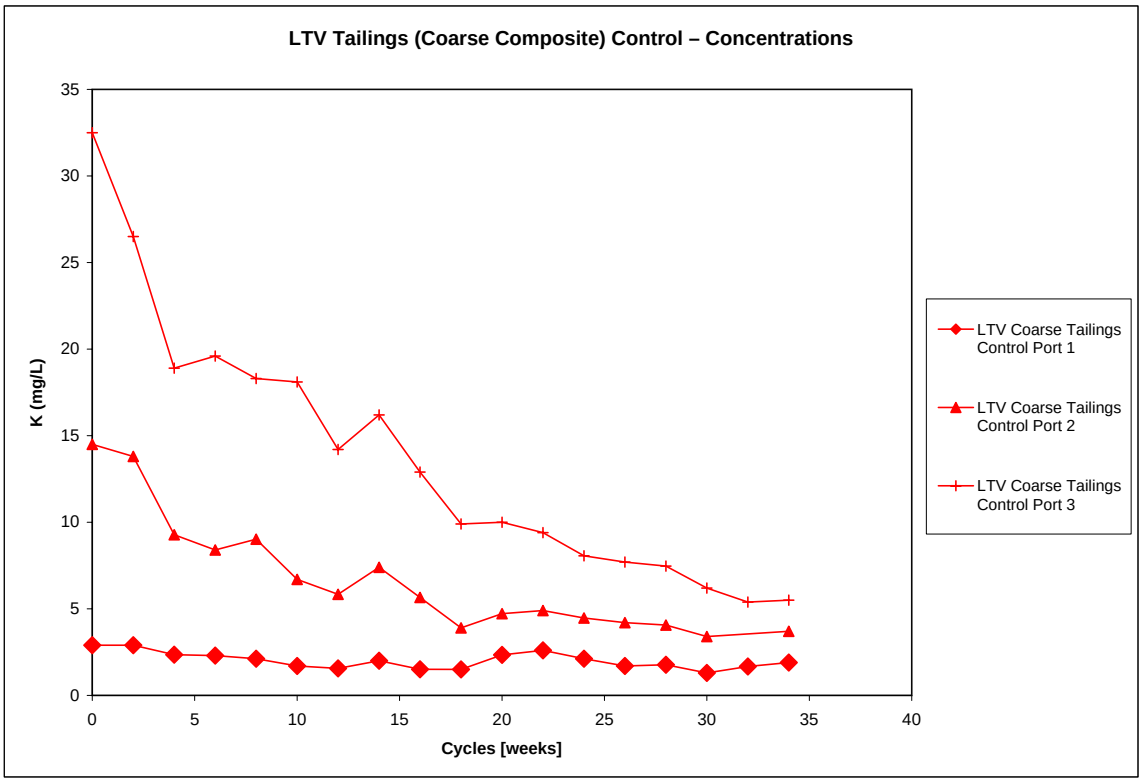
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.16



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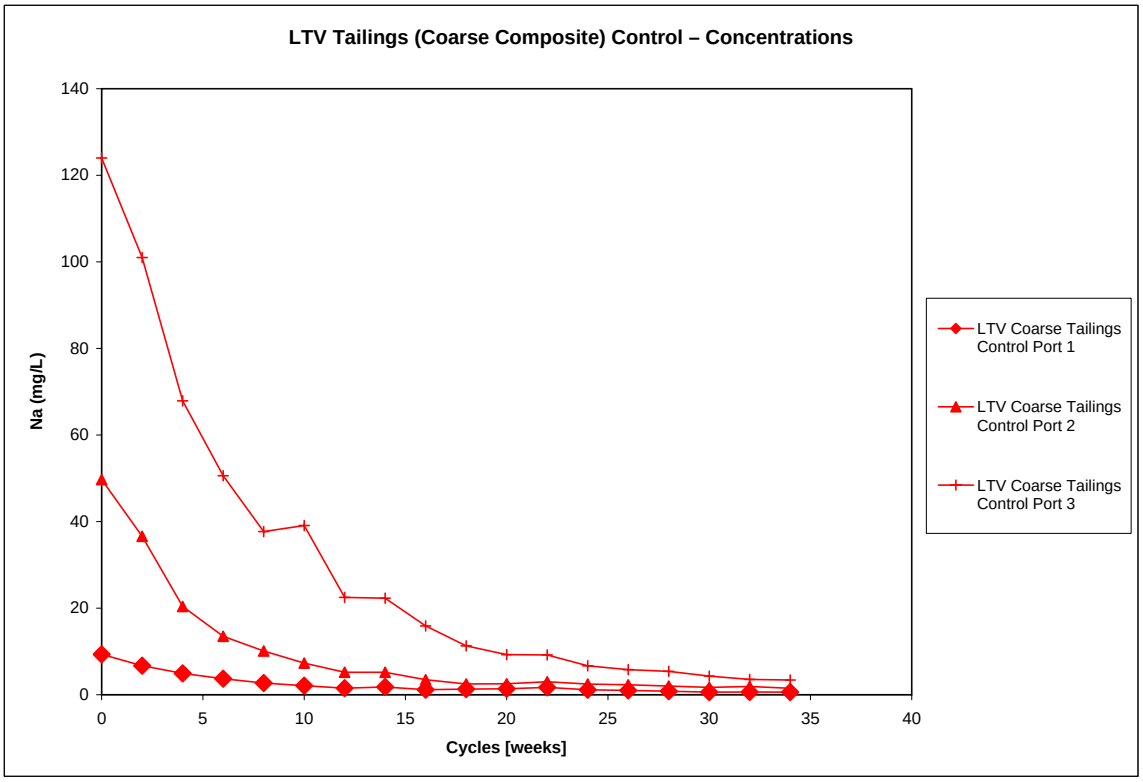
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.17



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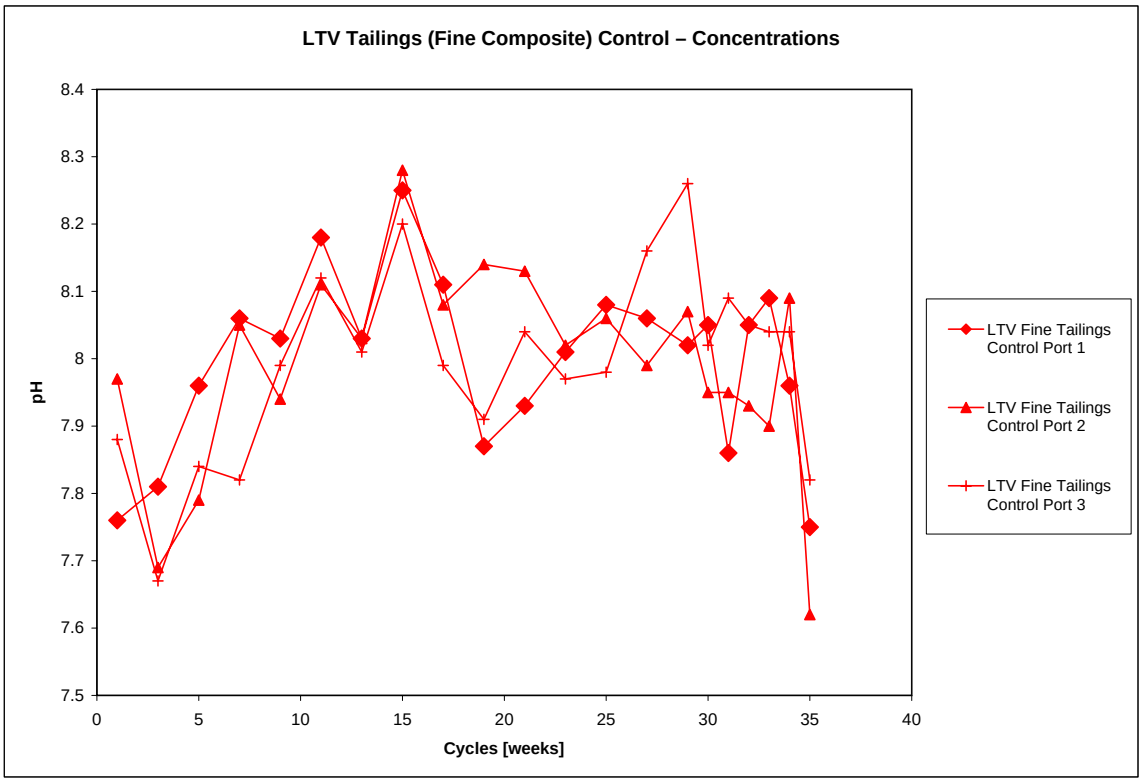
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.18



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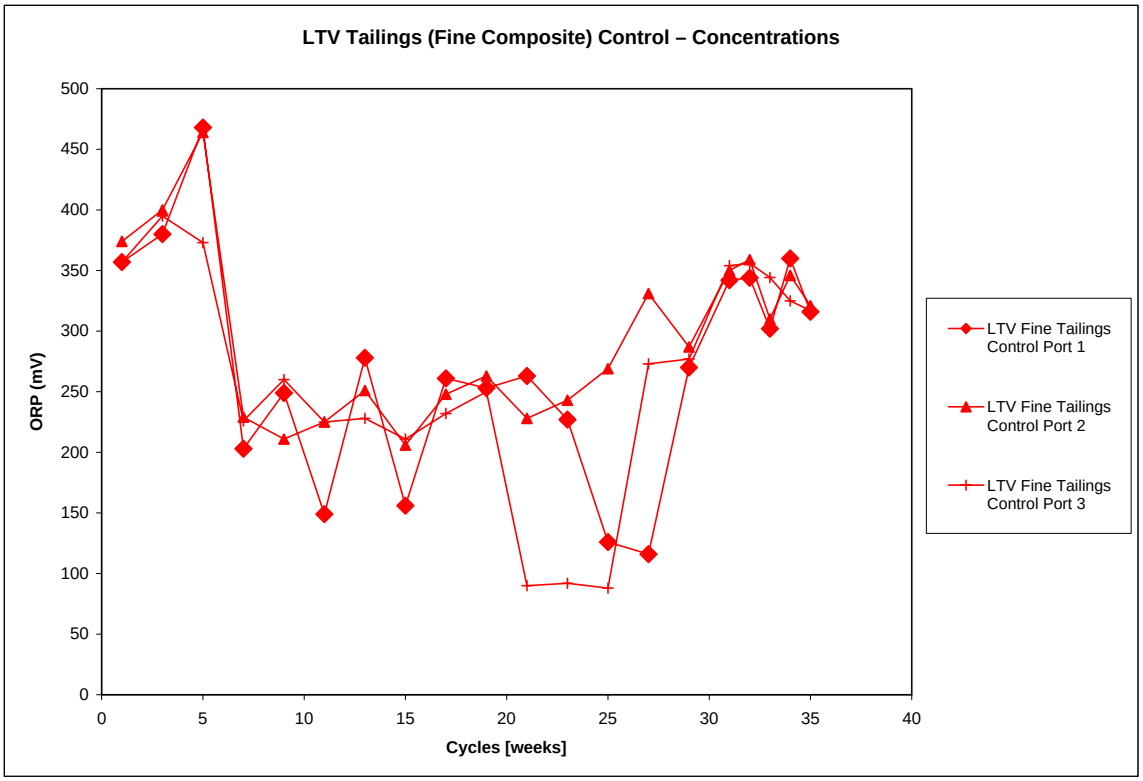
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.19



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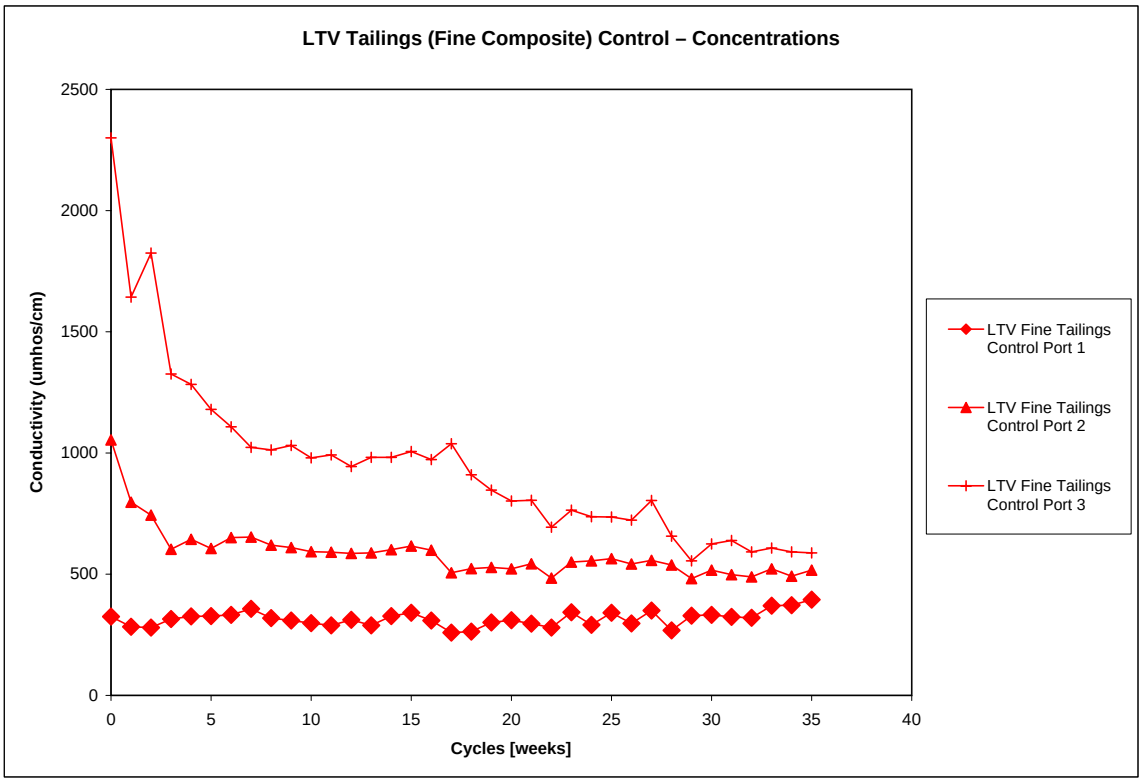
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.20



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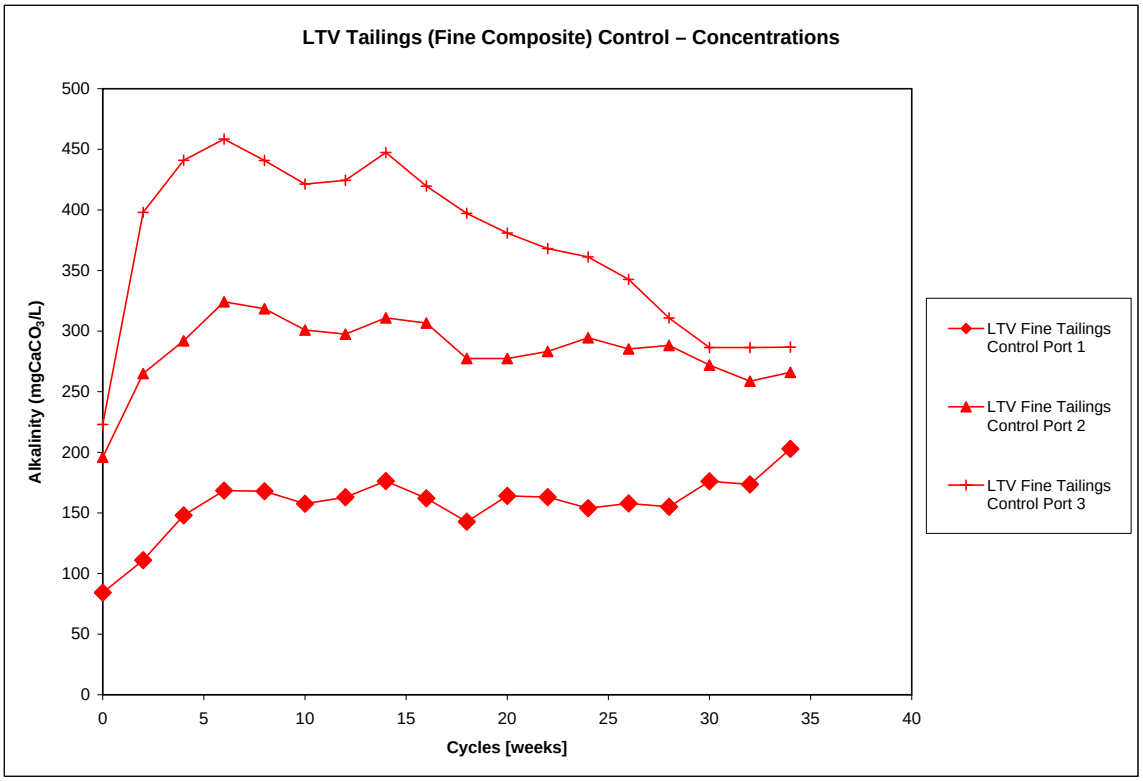
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.21



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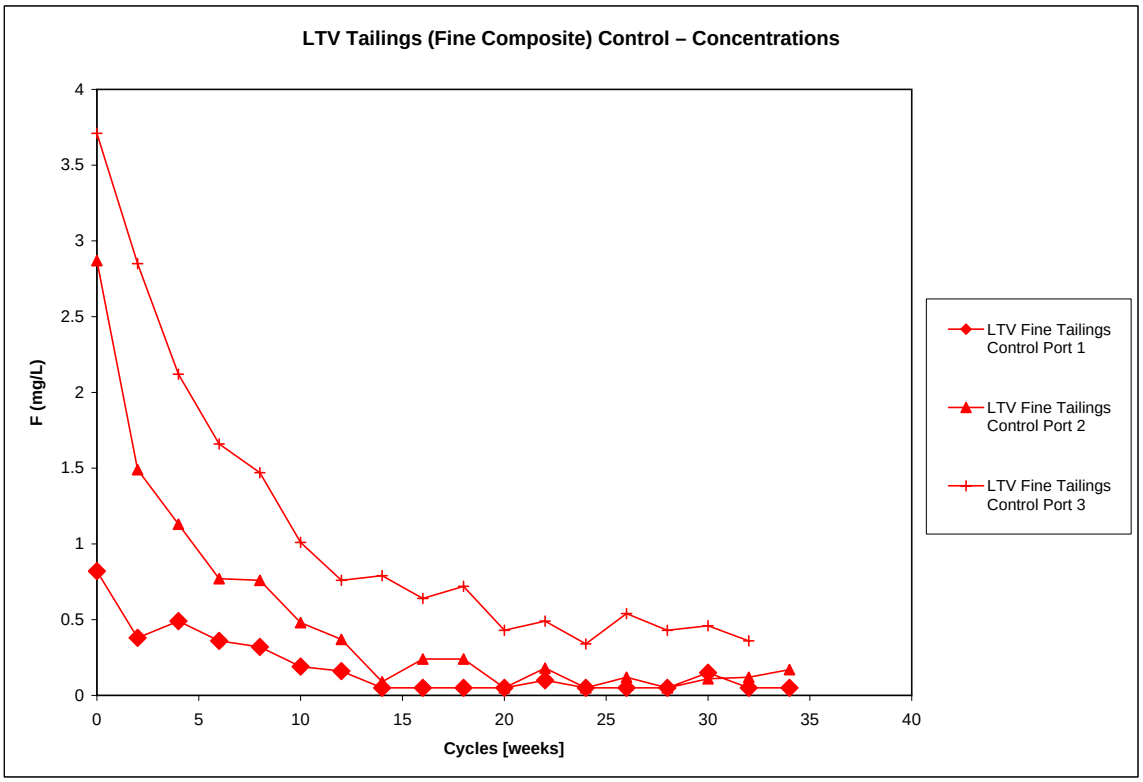
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.22



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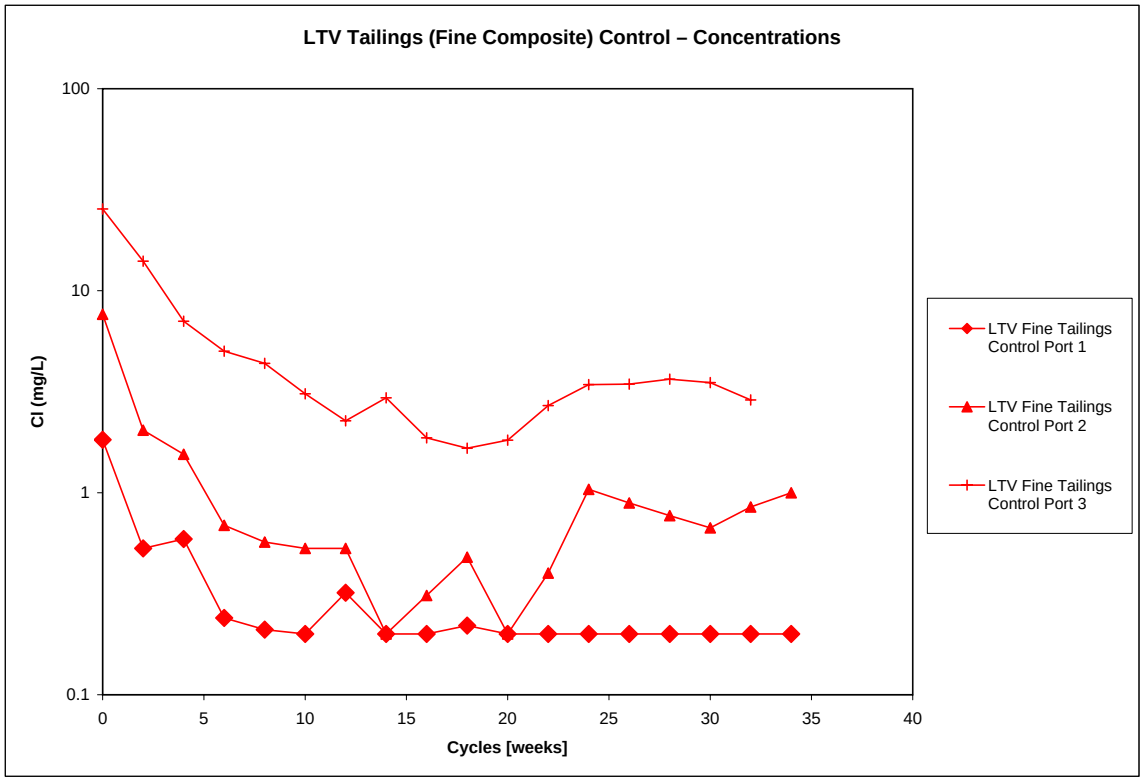
Graph C.3.23



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Appendix C.3
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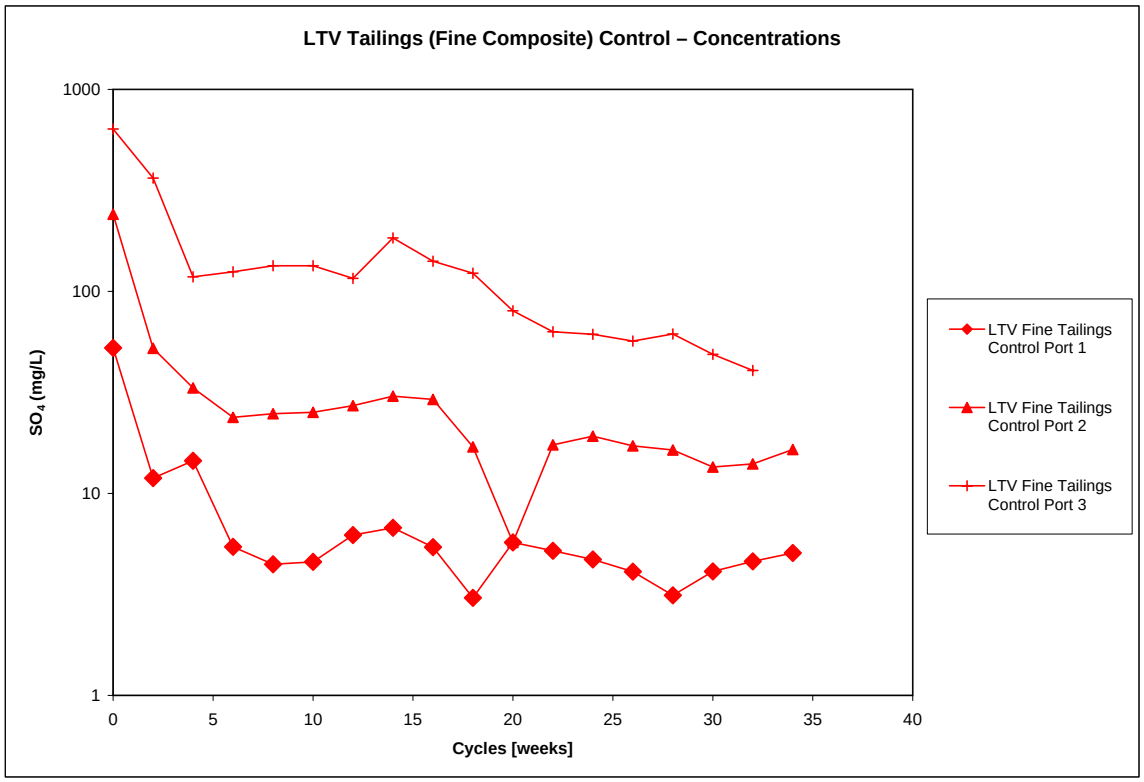
Graph C.3.24



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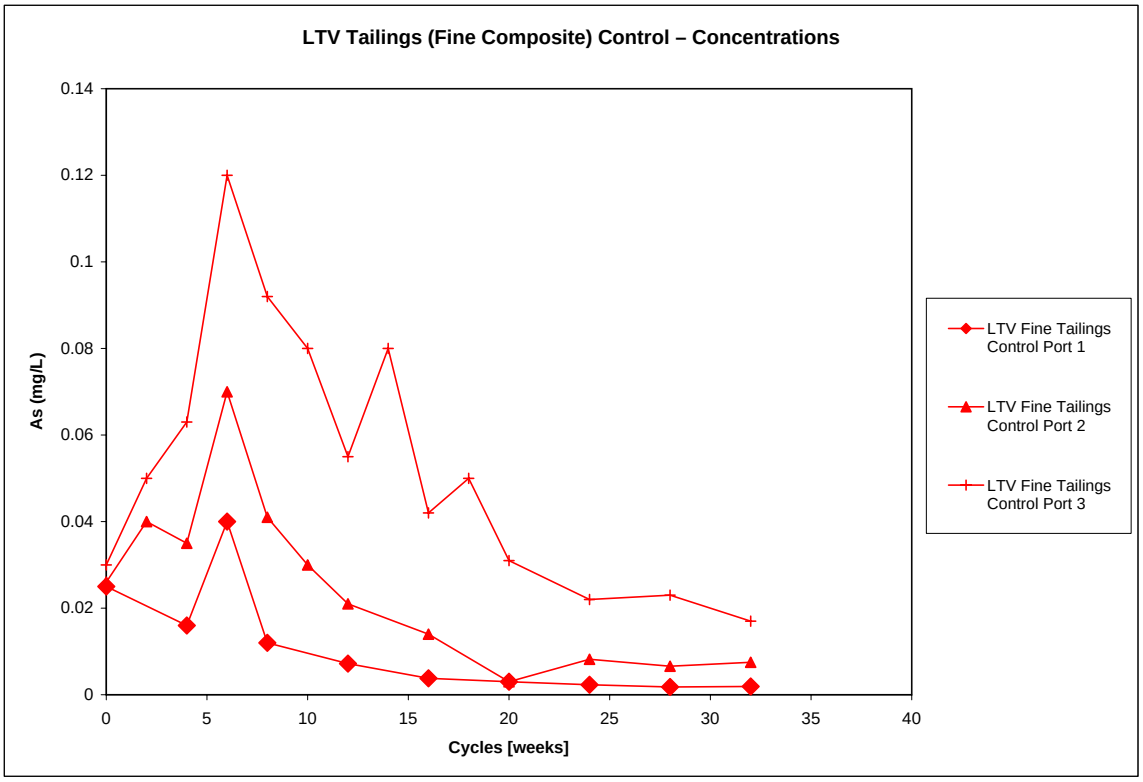
Graph C.3.25



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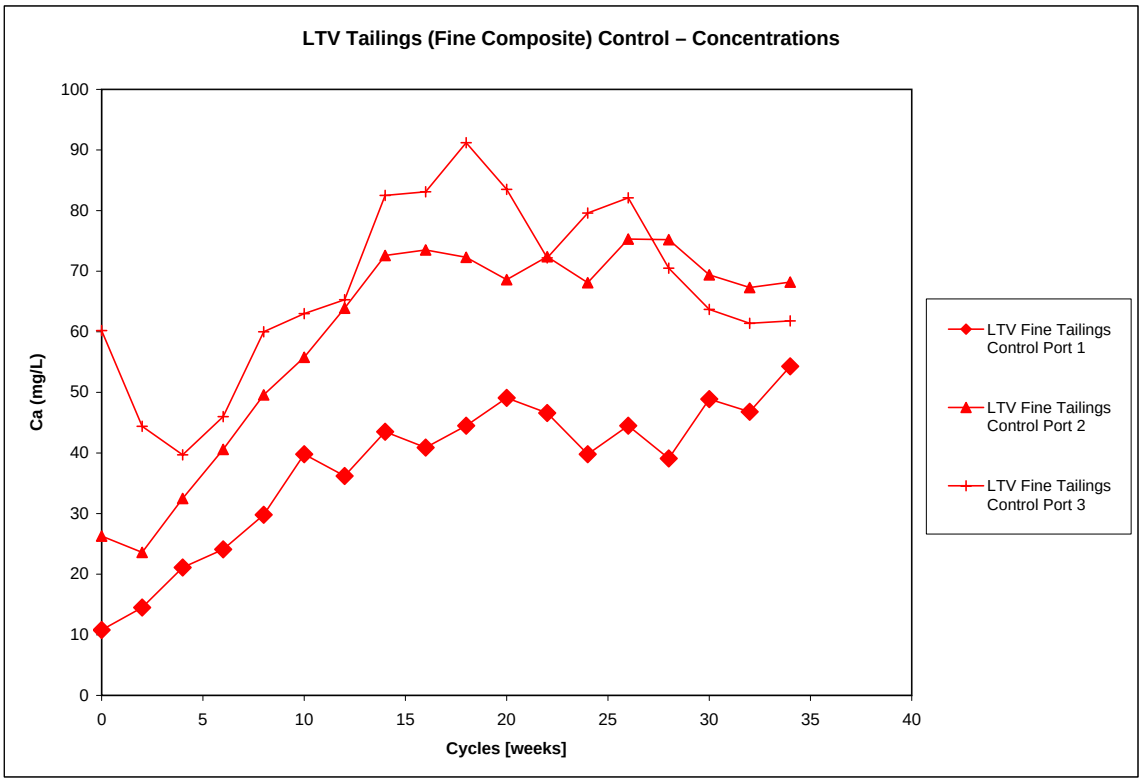
Graph C.3.26



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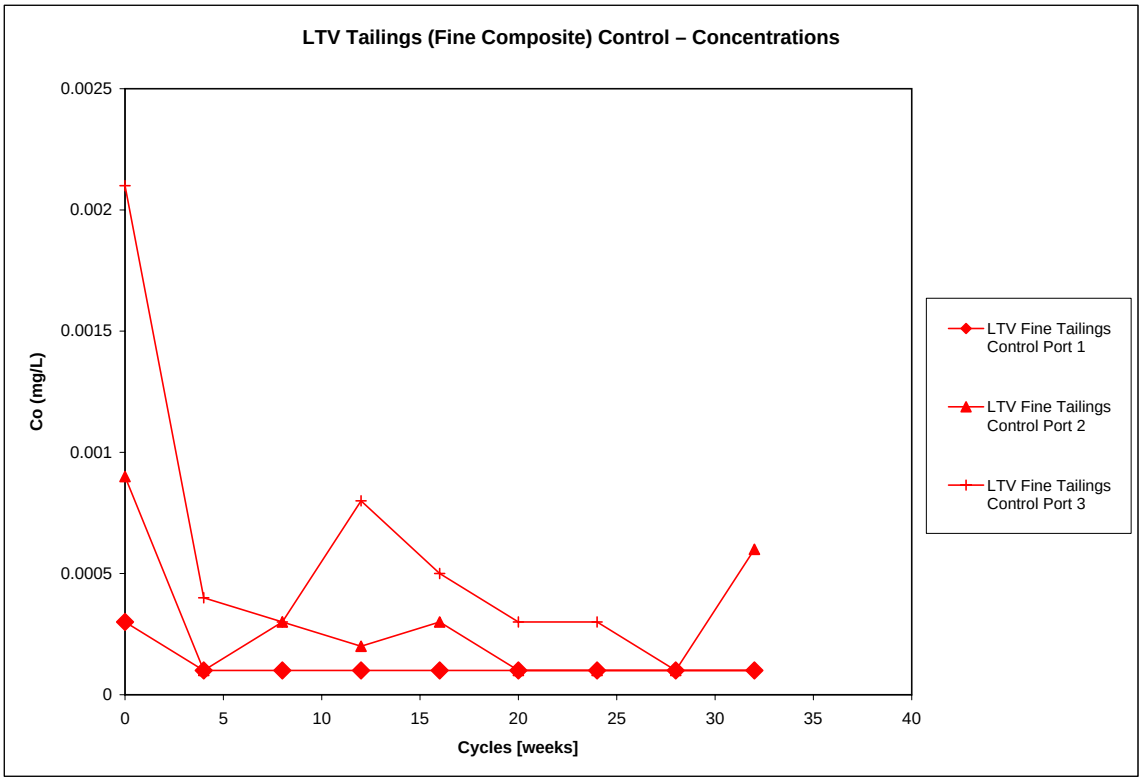
Graph C.3.27



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Appendix C.3
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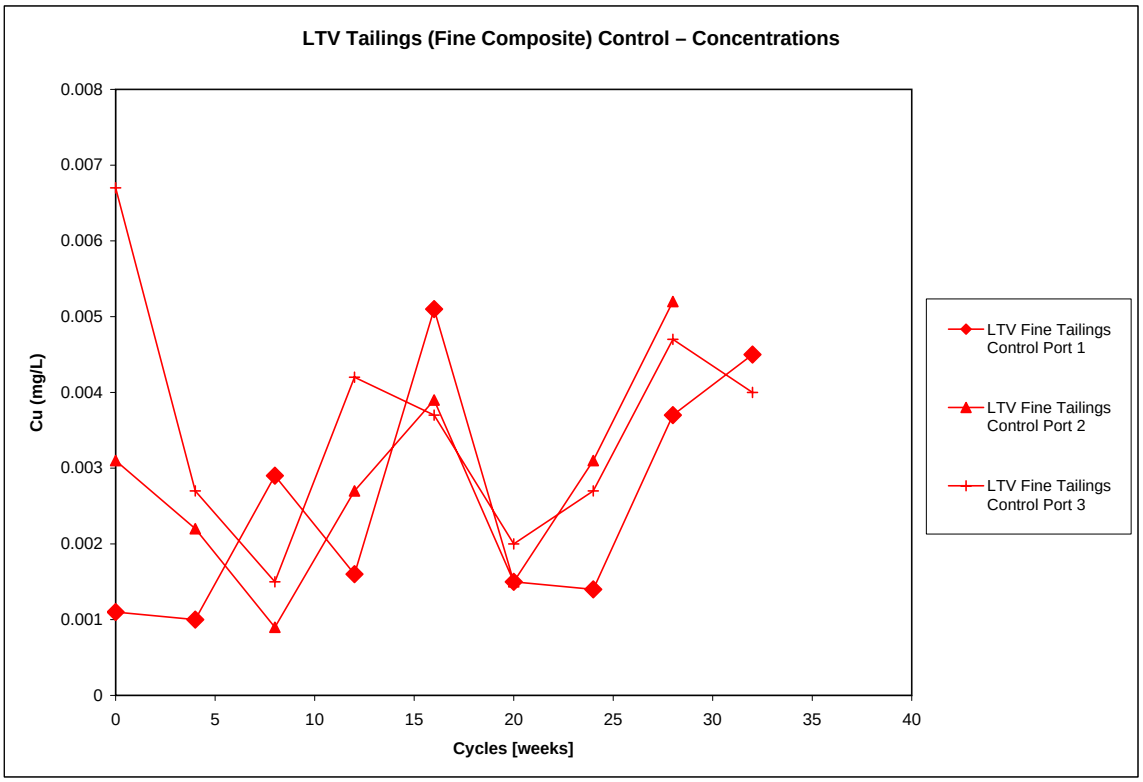
Graph C.3.28



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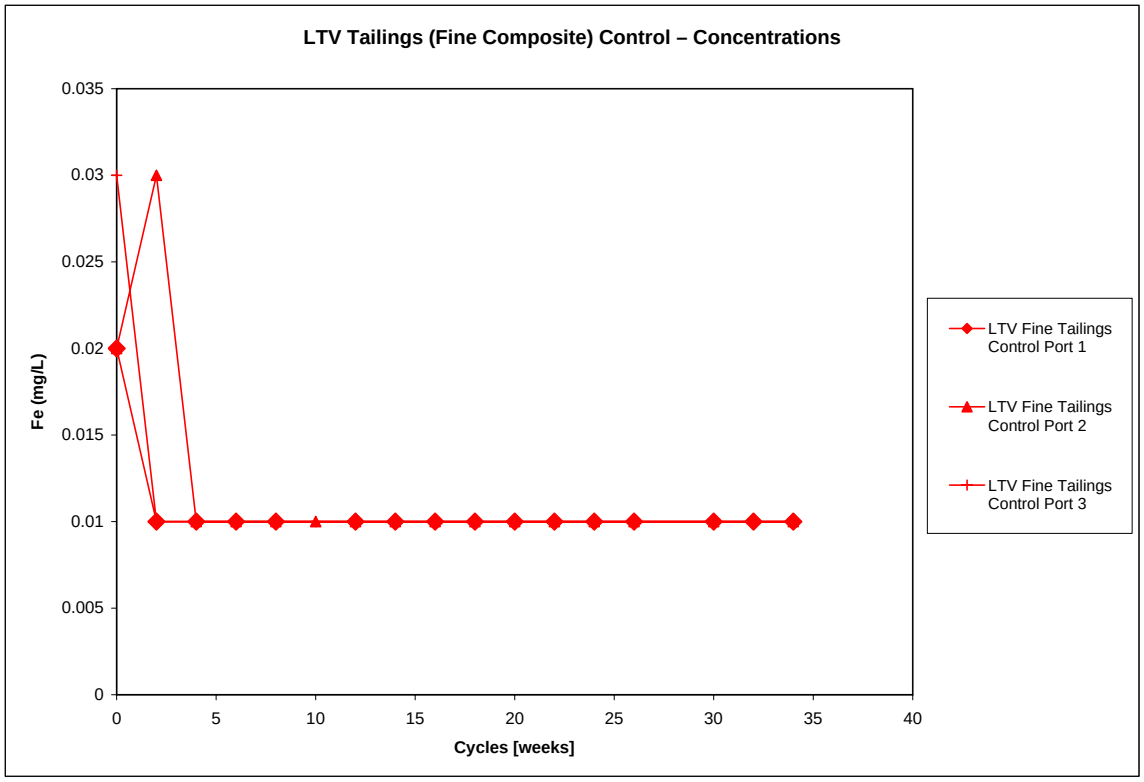
Graph C.3.29



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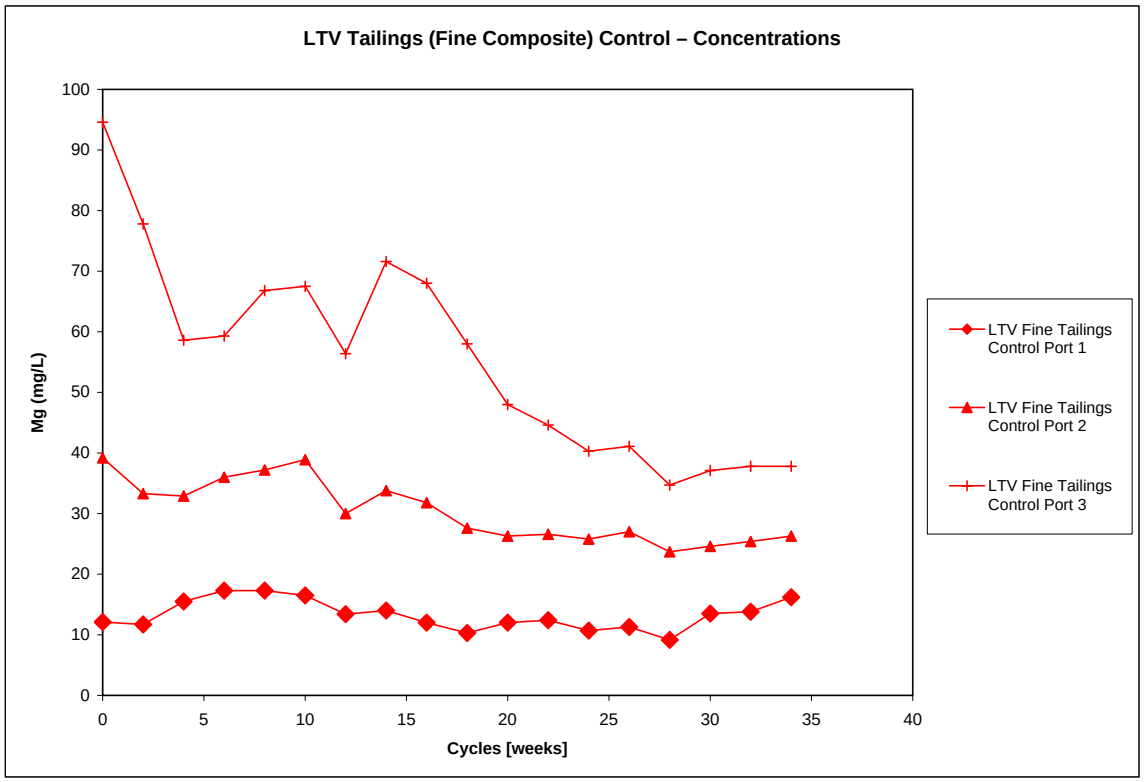
Graph C.3.30



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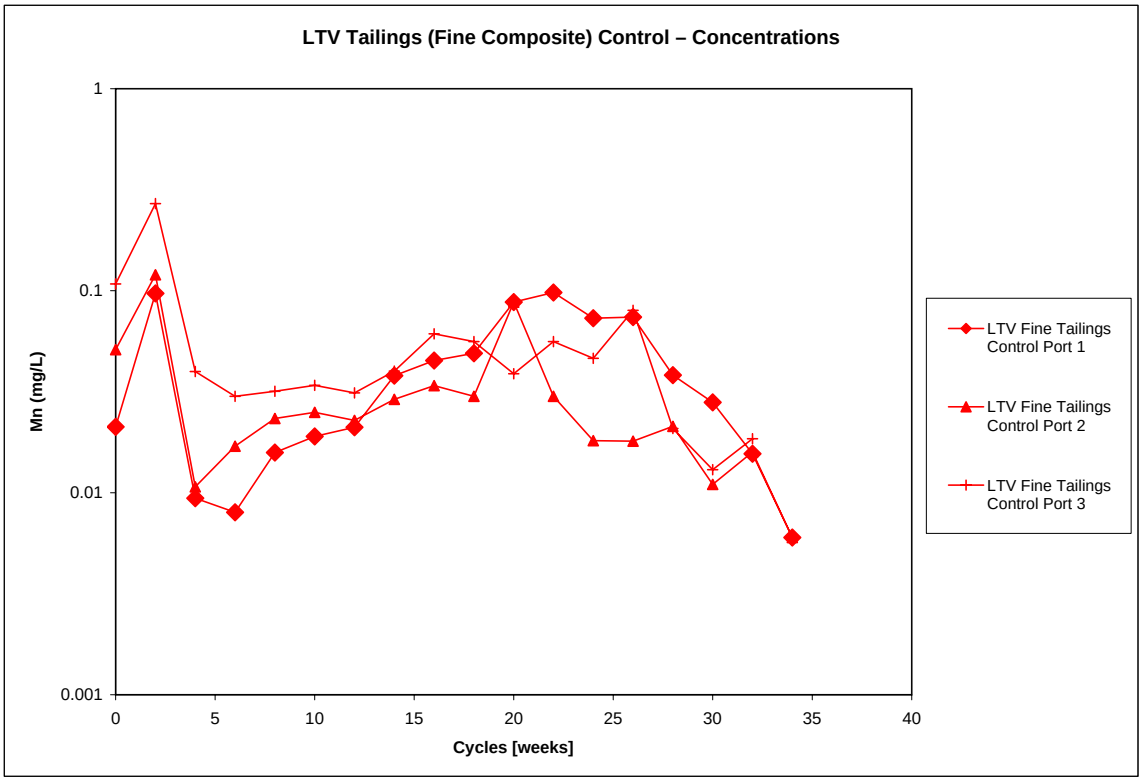
Graph C.3.31



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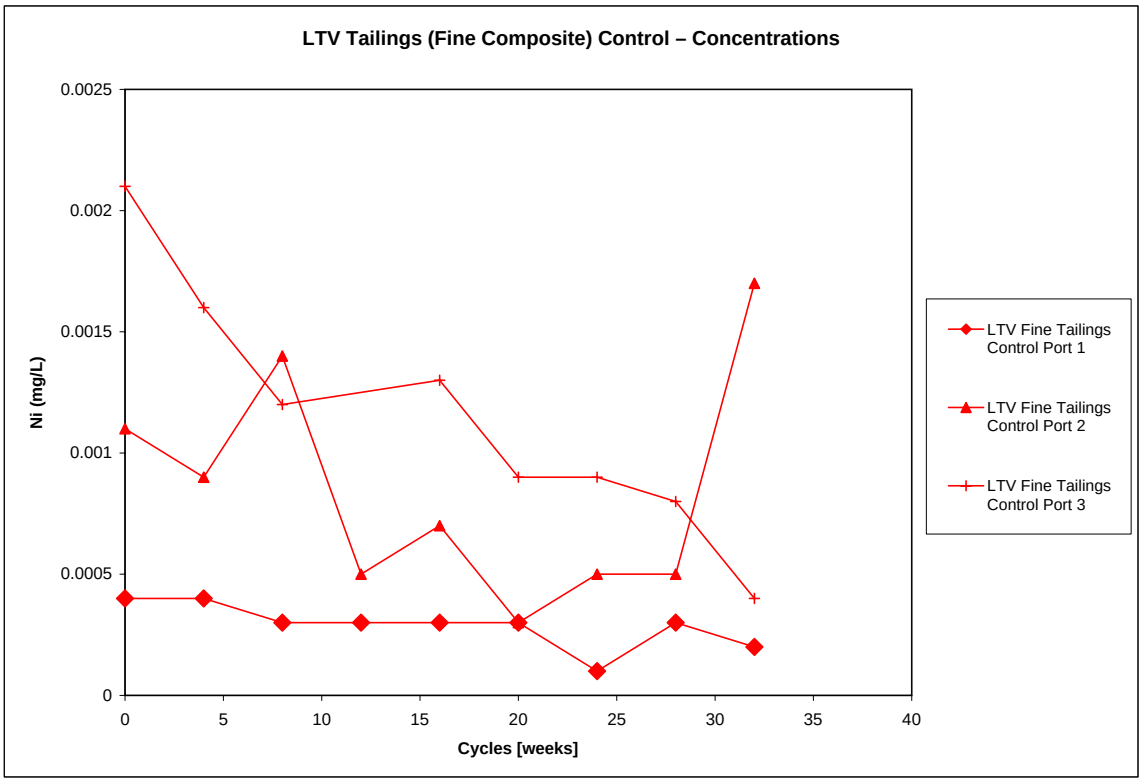
Graph C.3.32



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Appendix C.3
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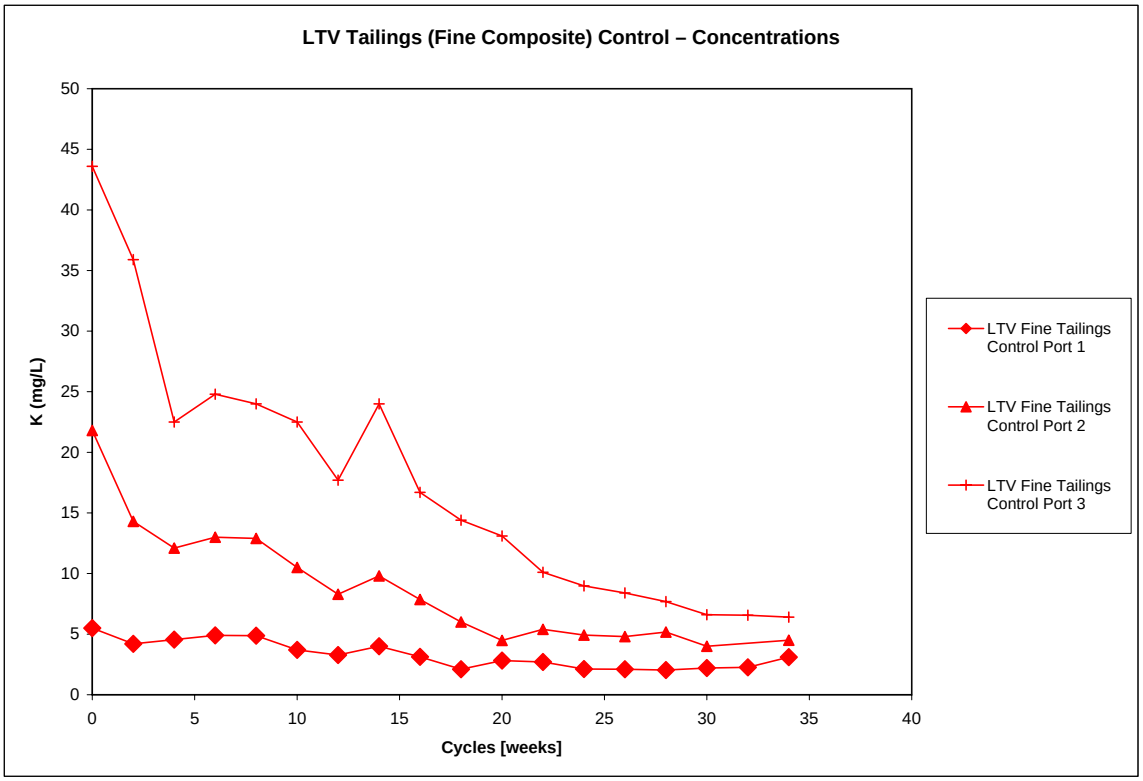
Graph C.3.33



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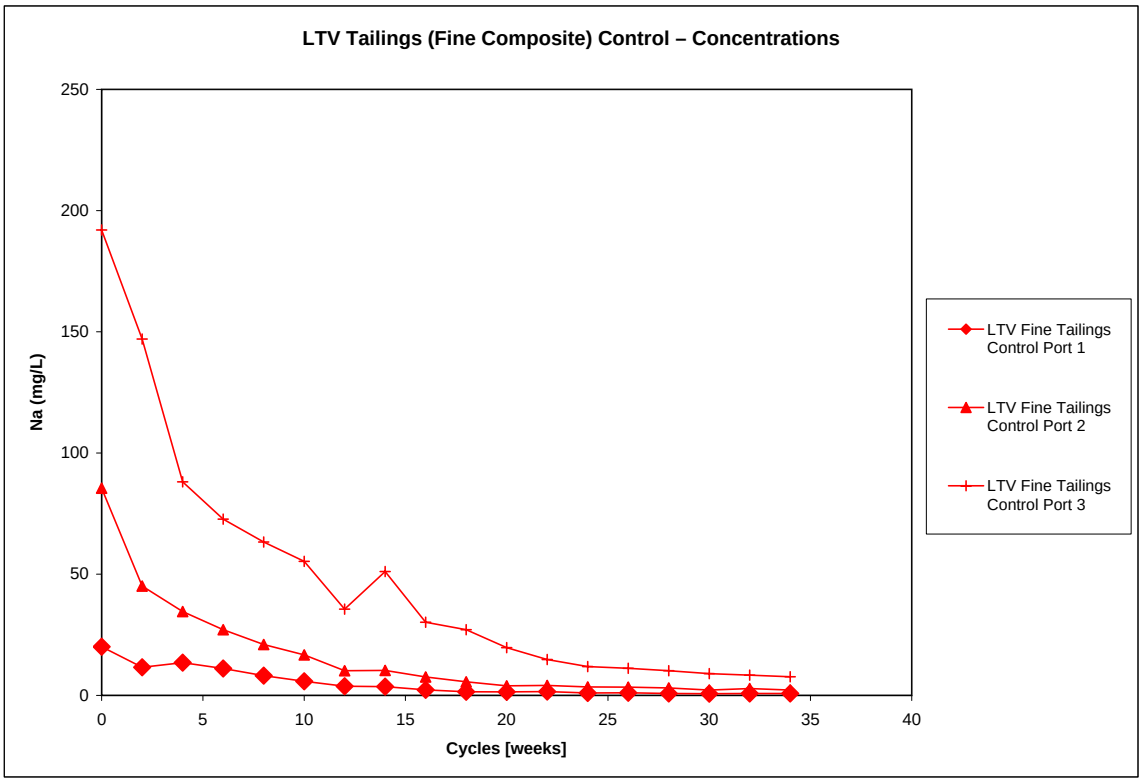
Graph C.3.34



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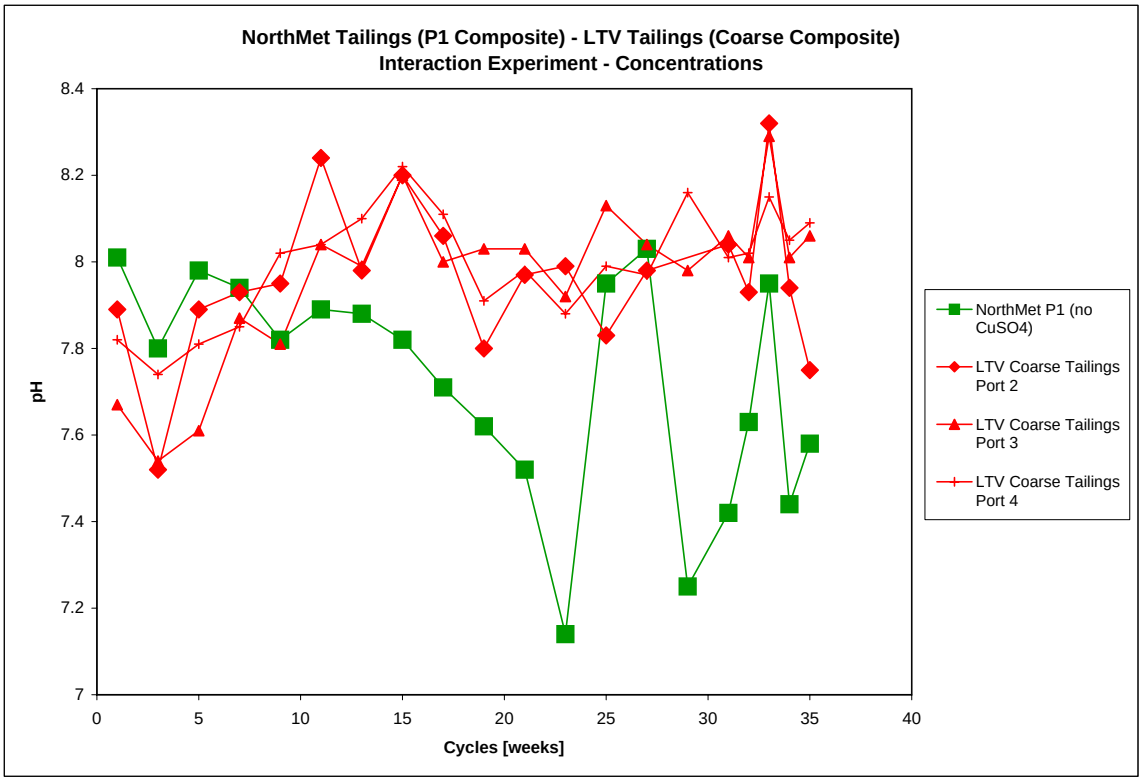
Graph C.3.35



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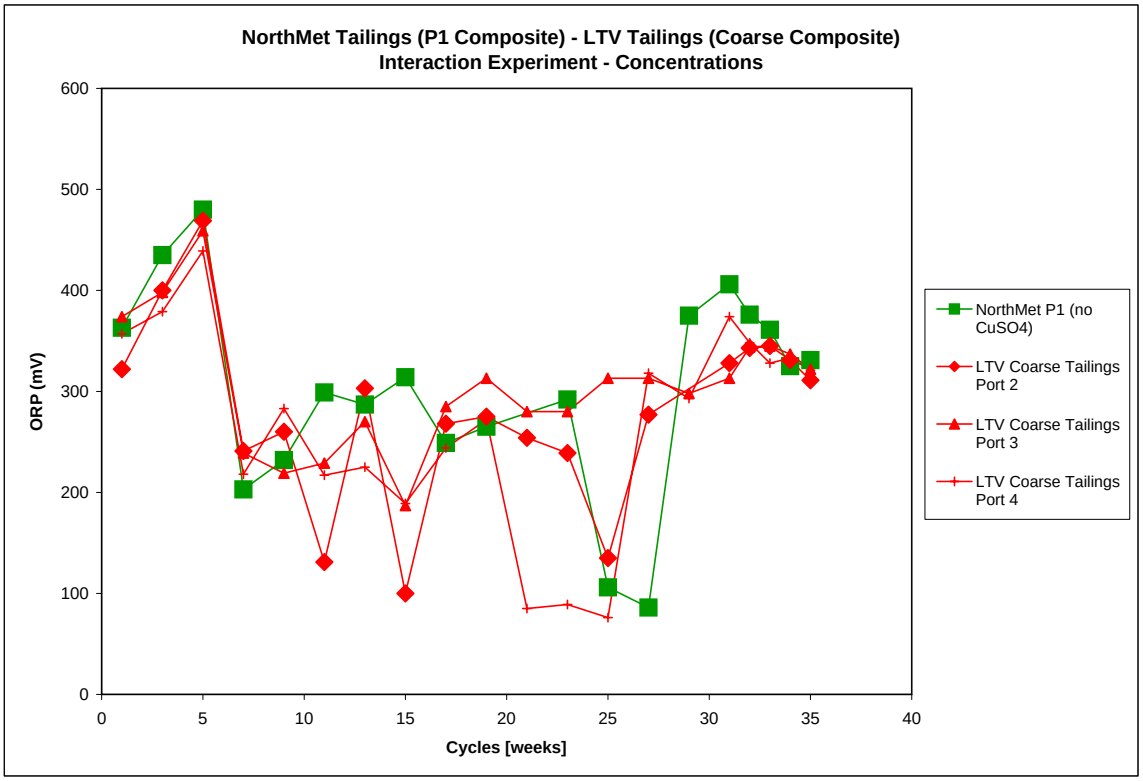
Graph C.3.36



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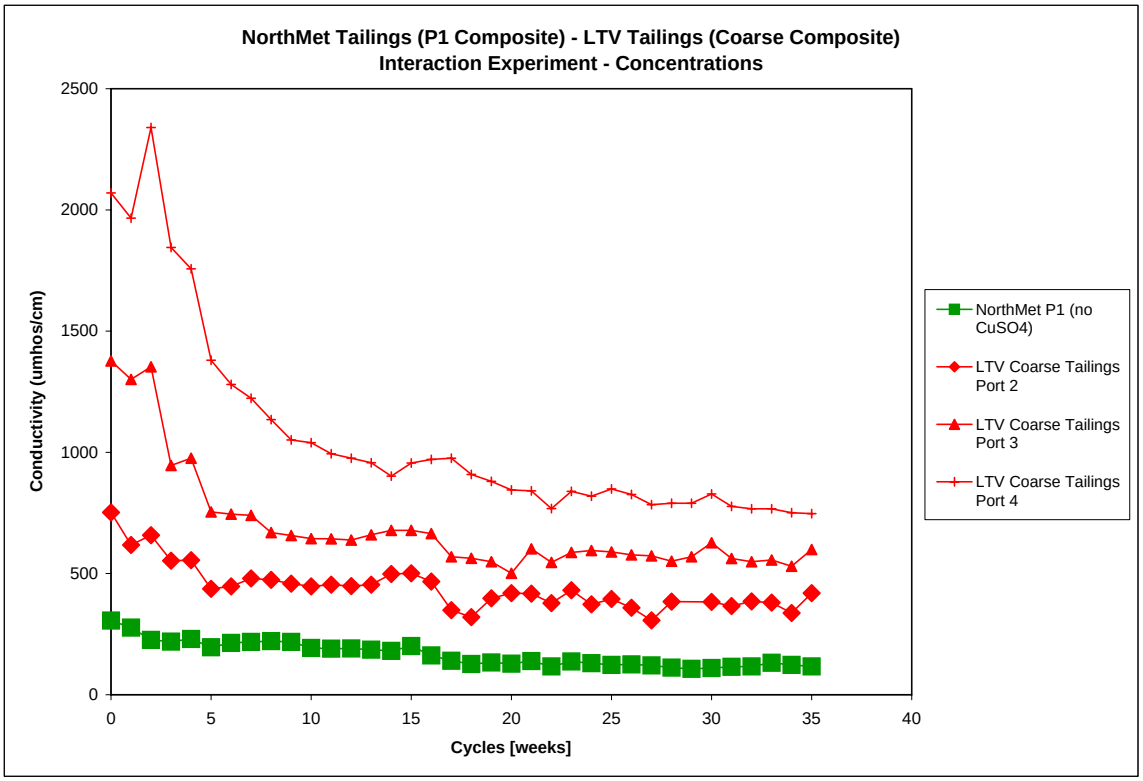
Graph C.3.37



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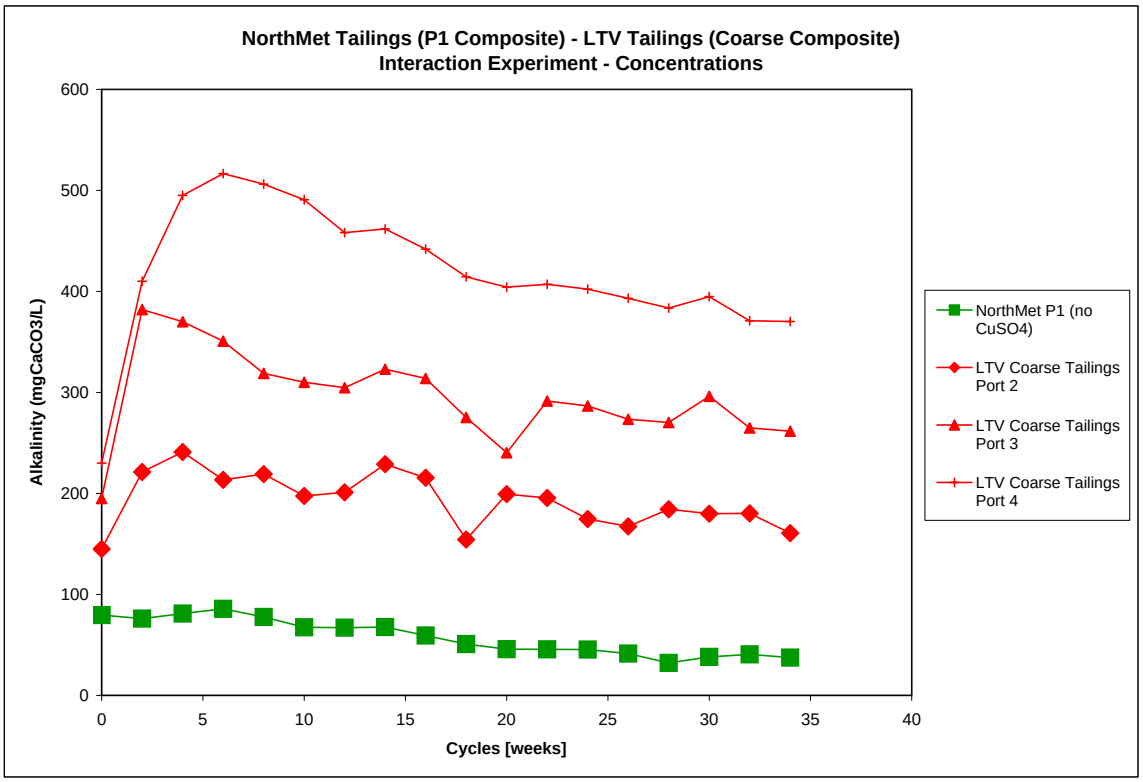
Graph C.3.38



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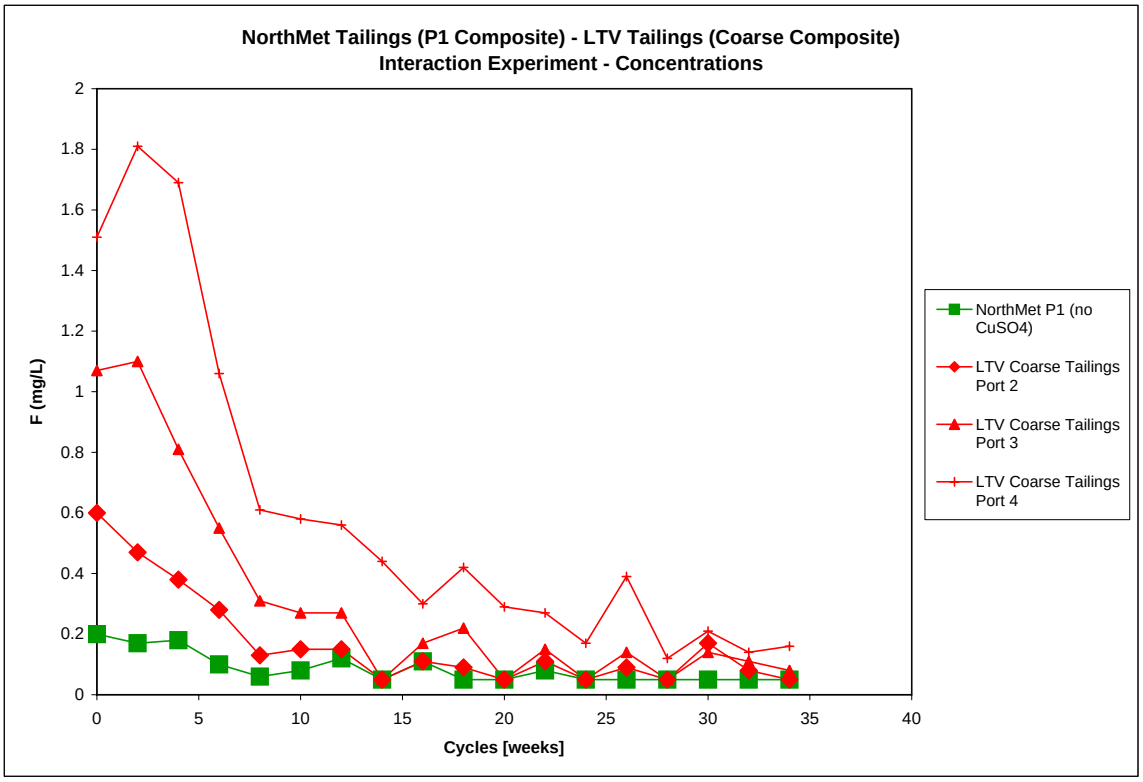
Graph C.3.39



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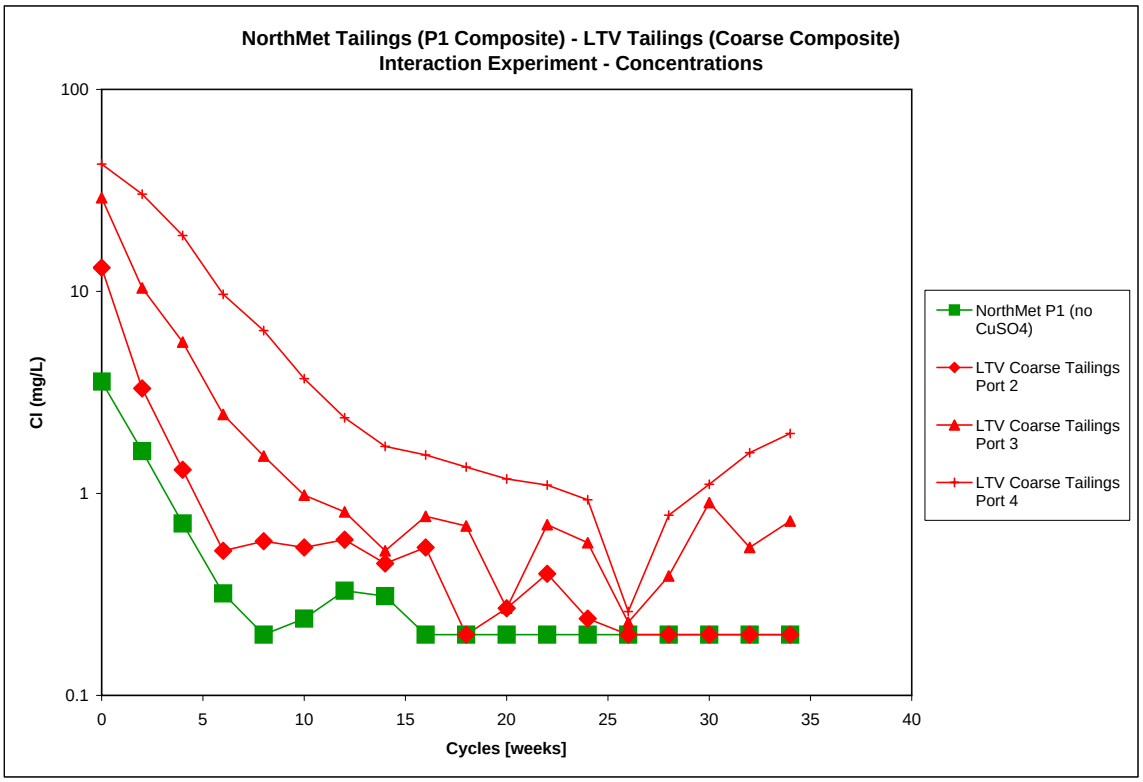
Graph C.3.40



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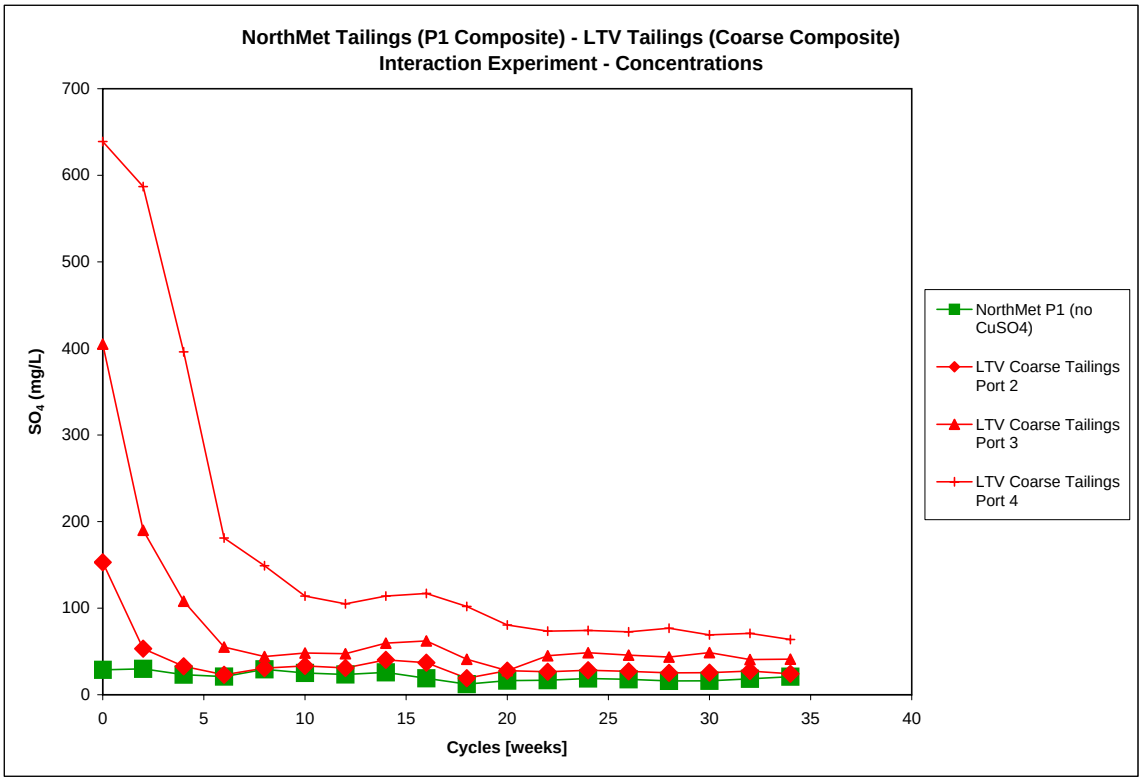
Graph C.3.41



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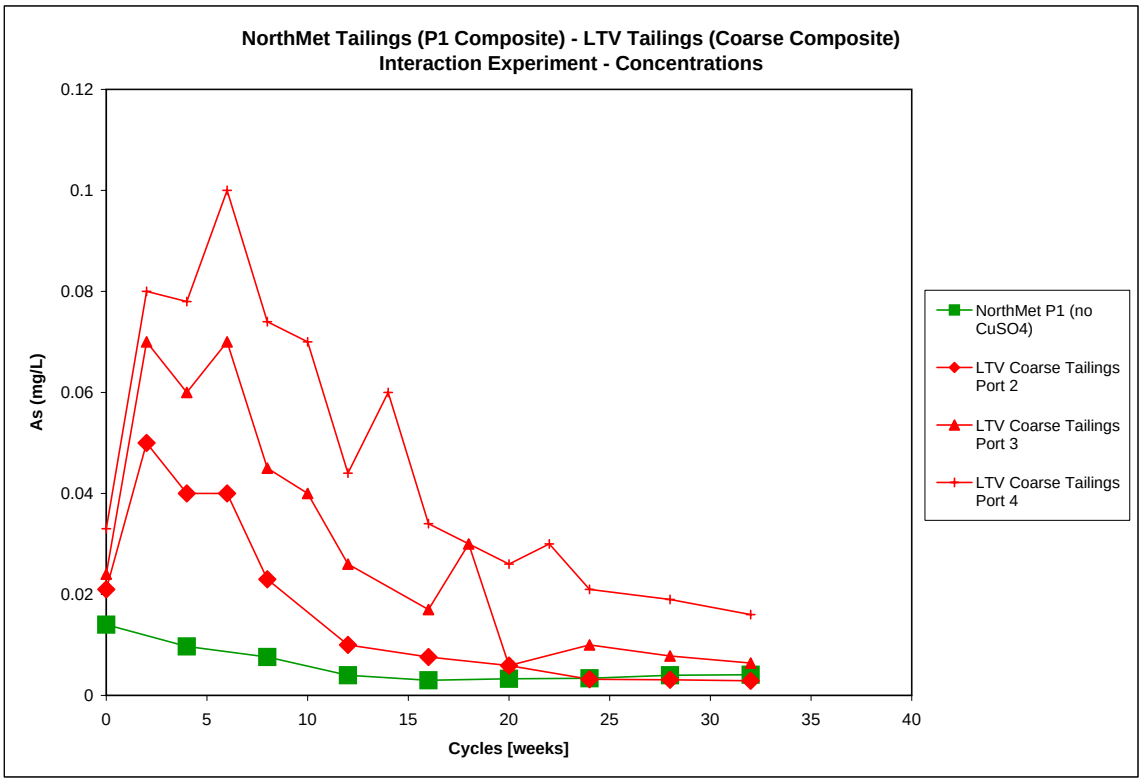
Graph C.3.42



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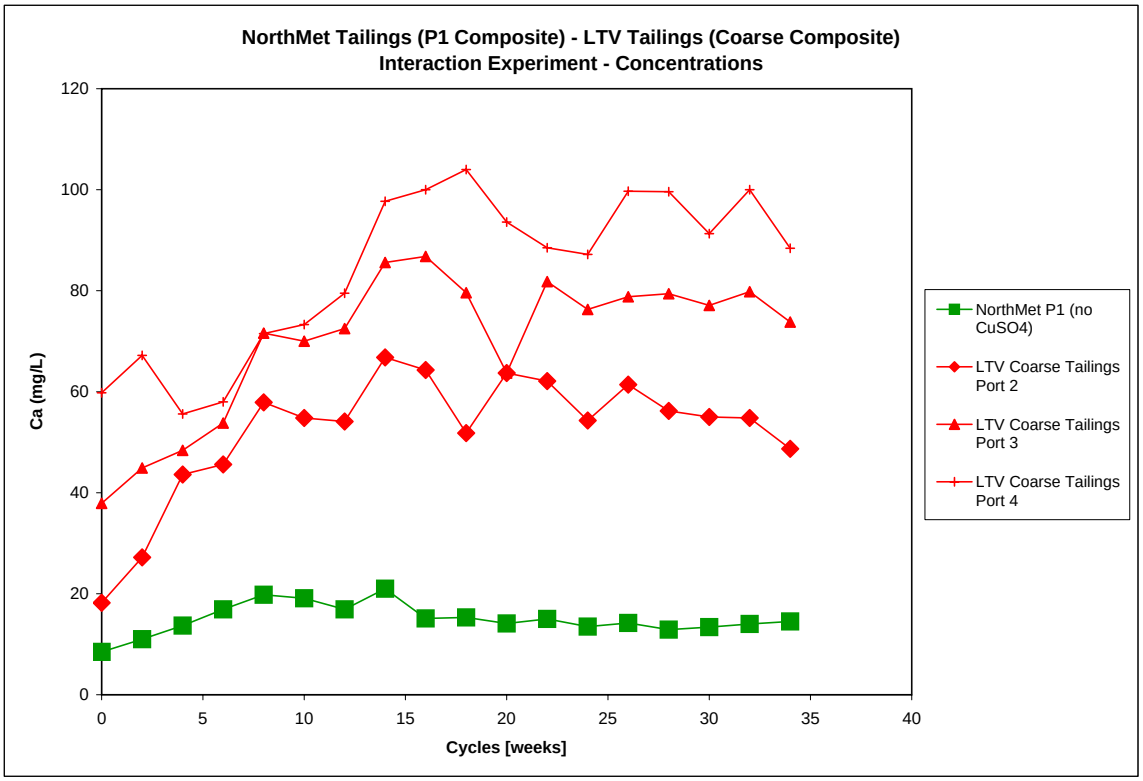
Graph C.3.43



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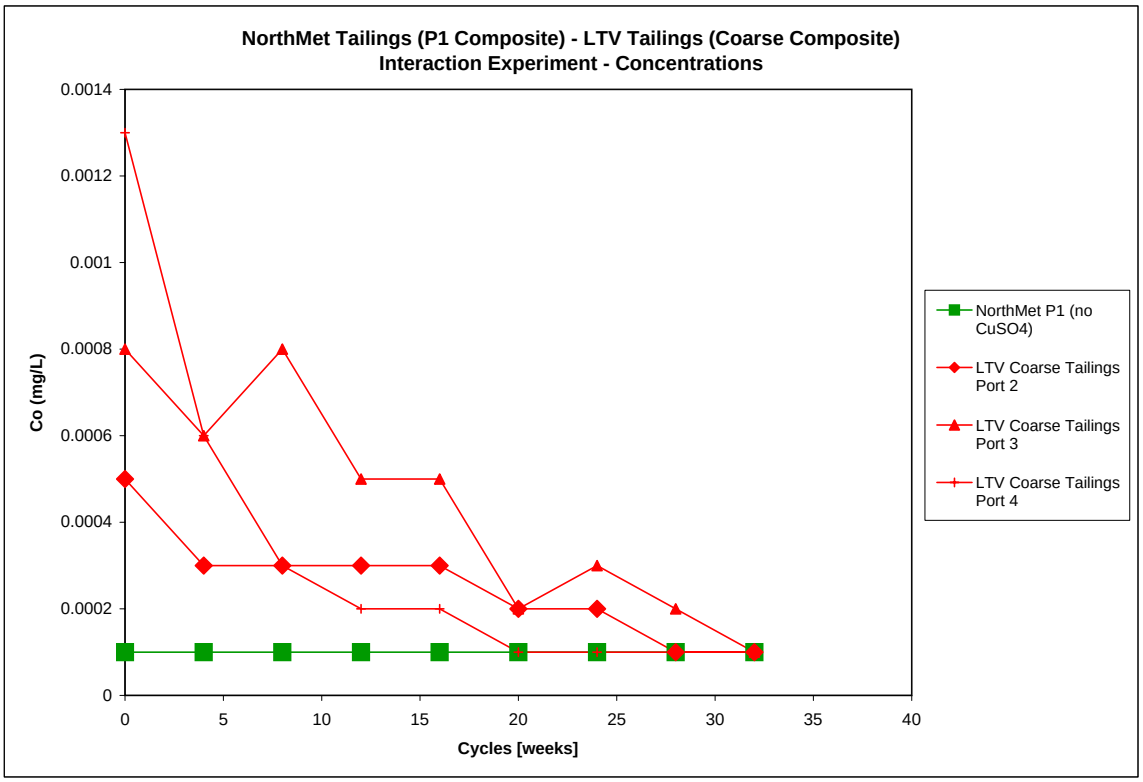
Graph C.3.44



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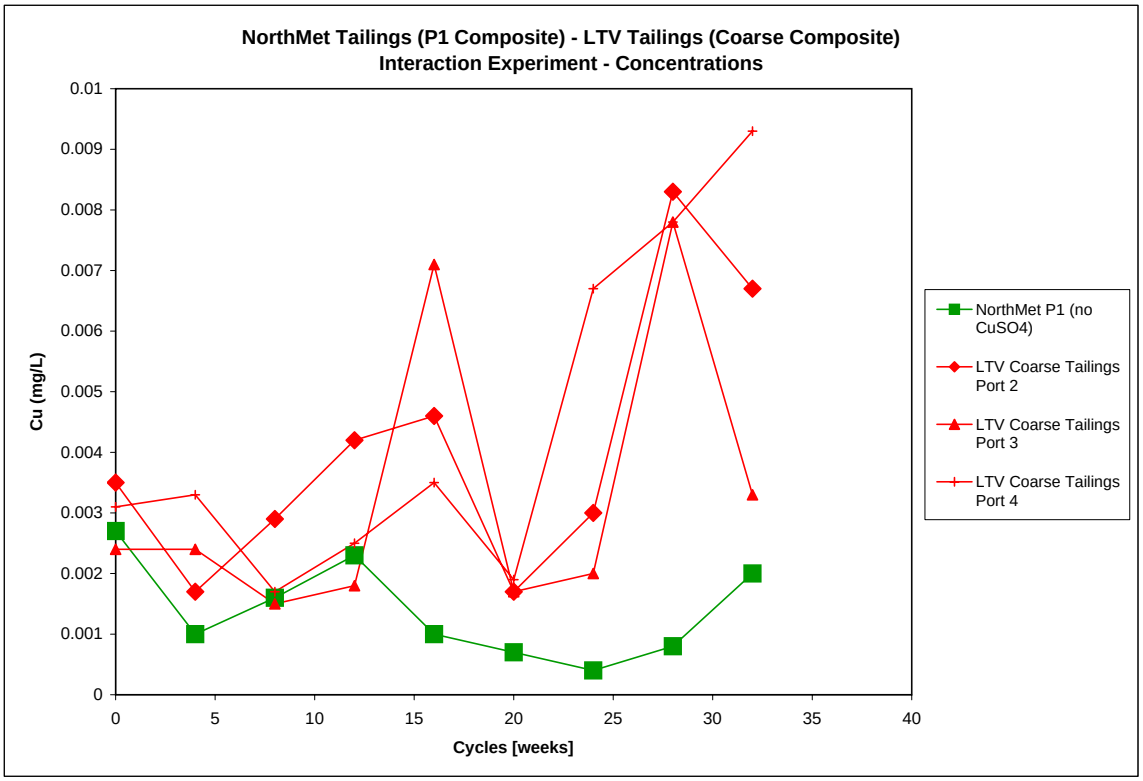
Graph C.3.45



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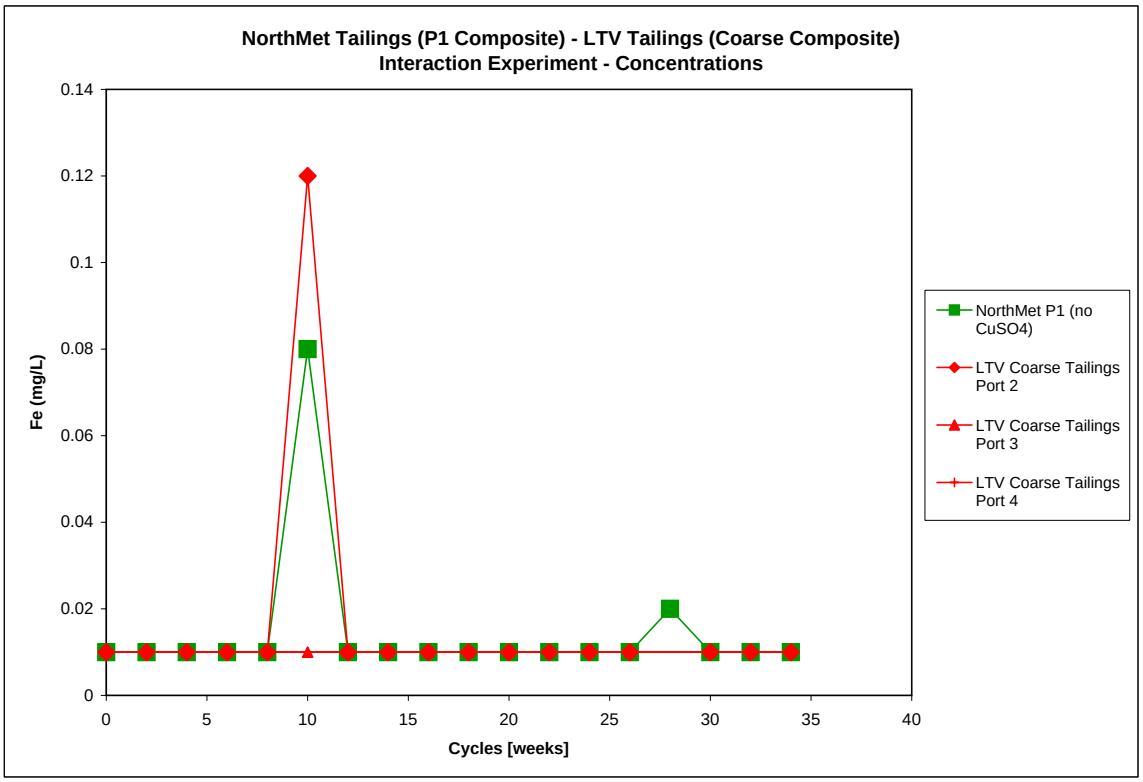
Graph C.3.46



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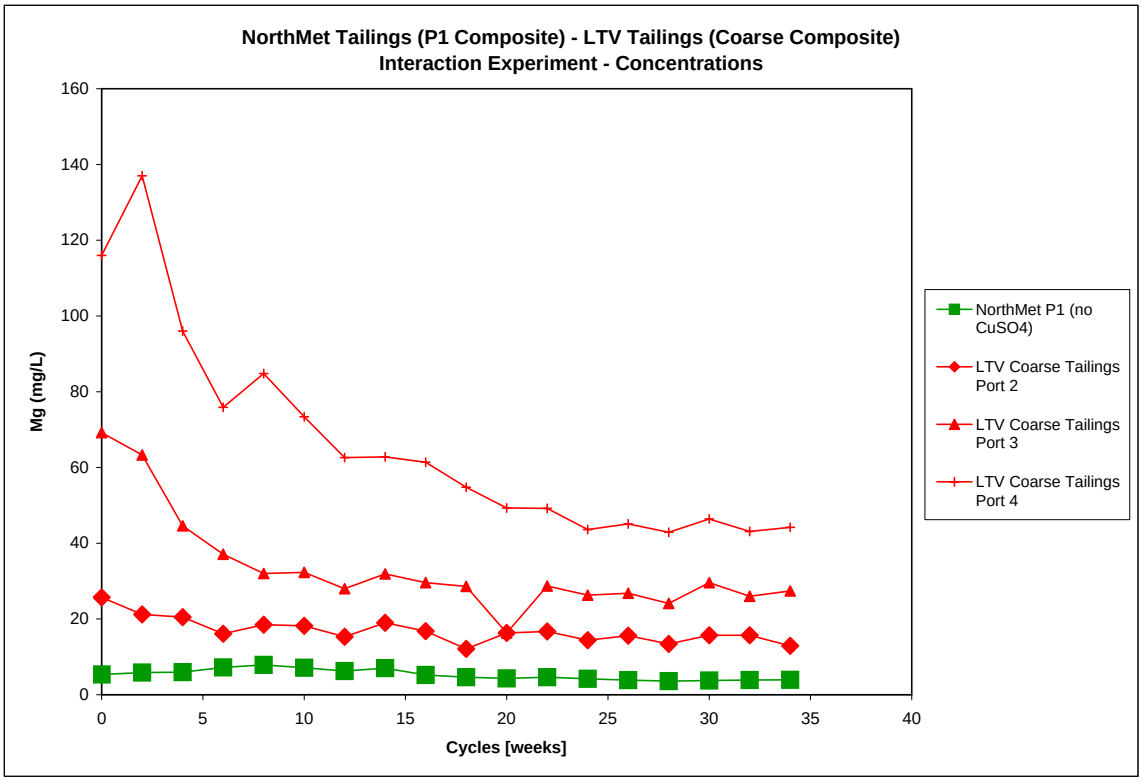
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.47



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Appendix C.3
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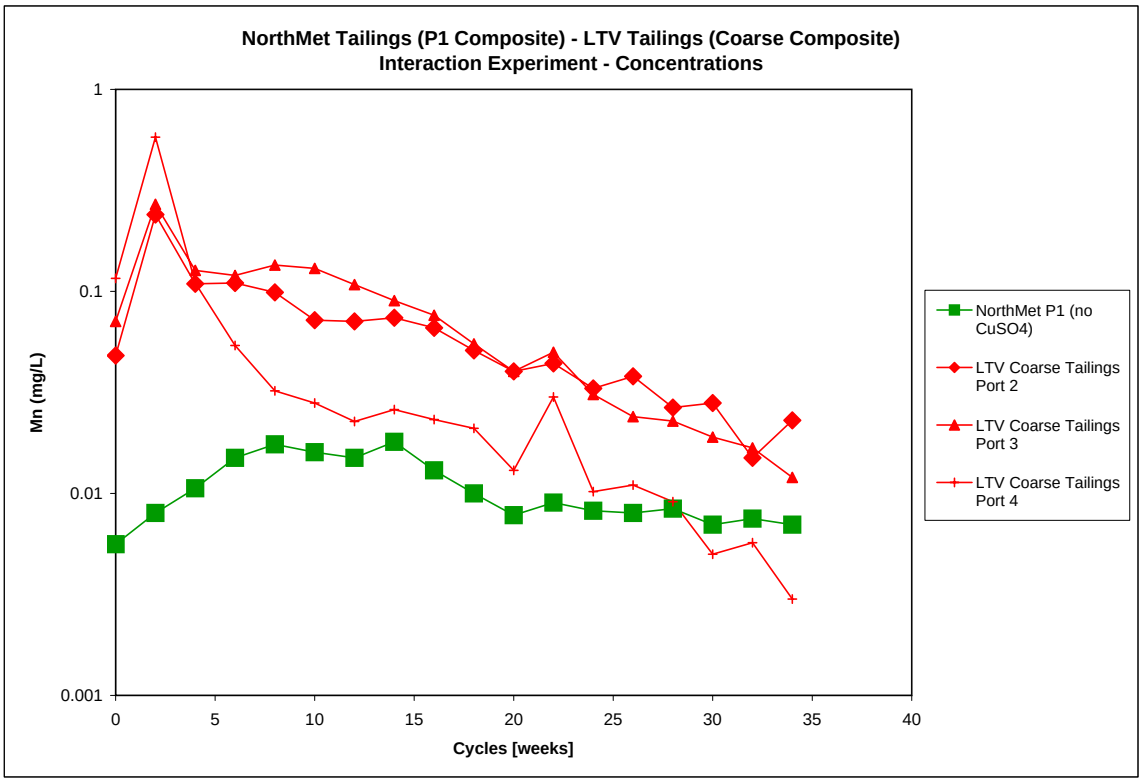
Graph C.3.48



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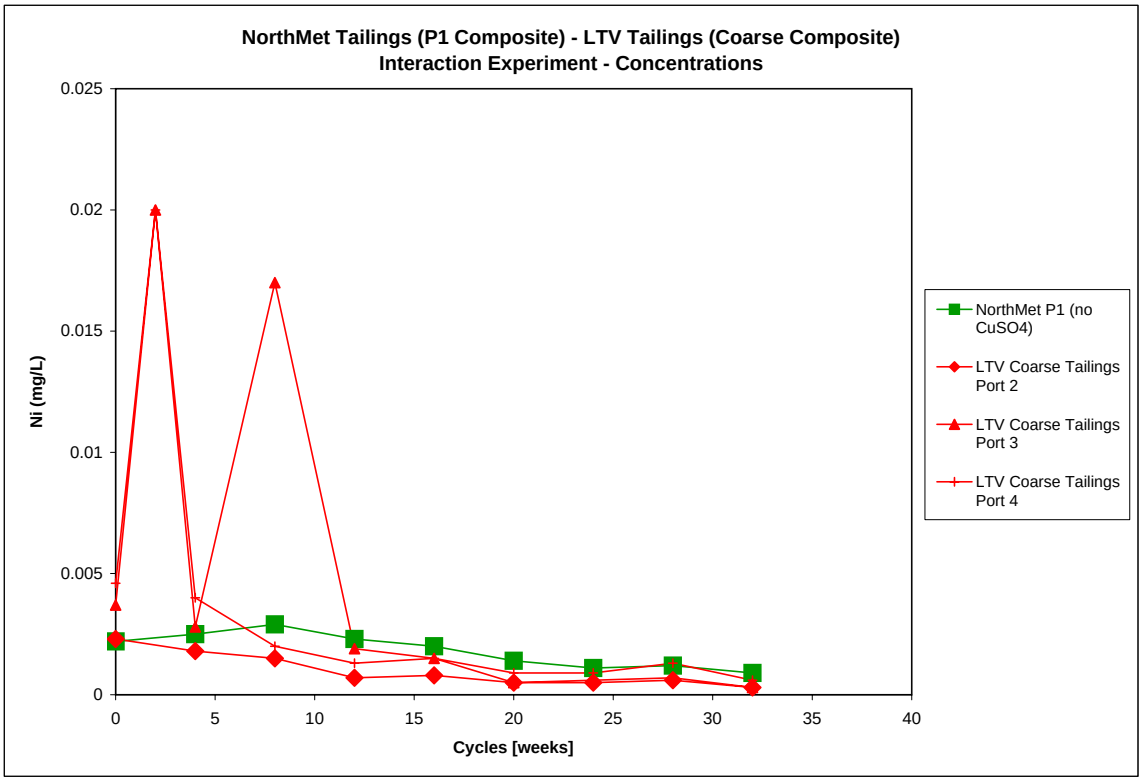
Graph C.3.49



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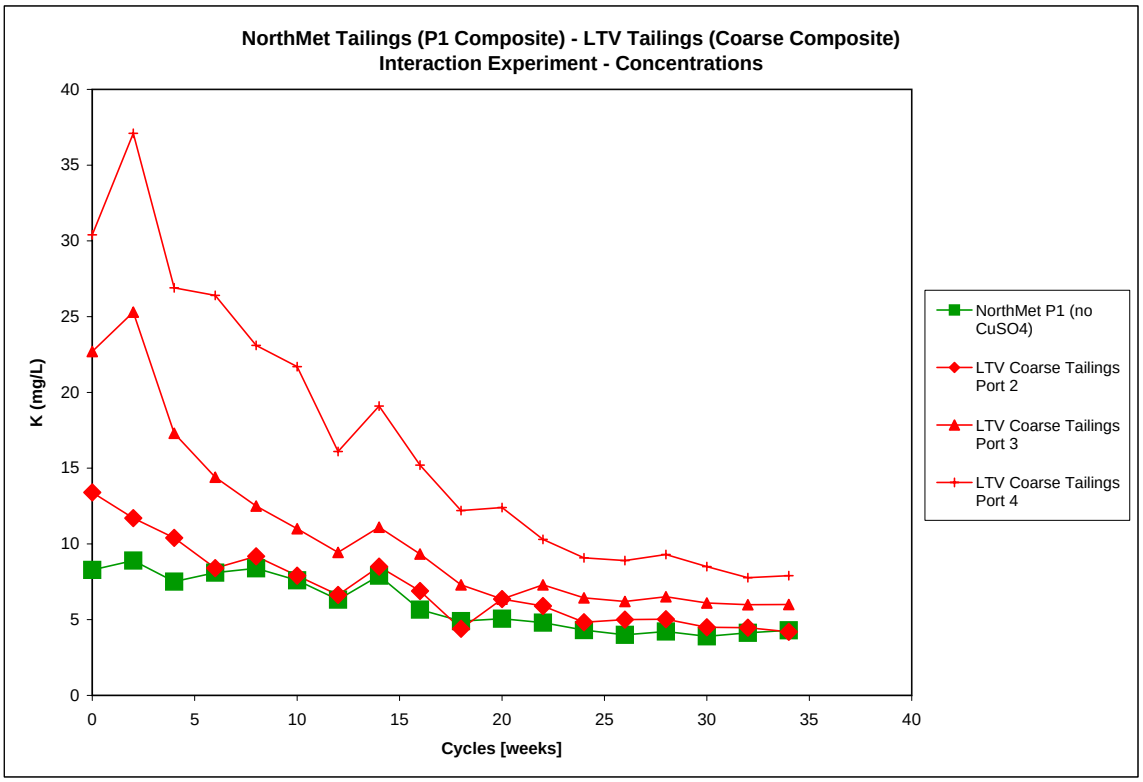
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.50



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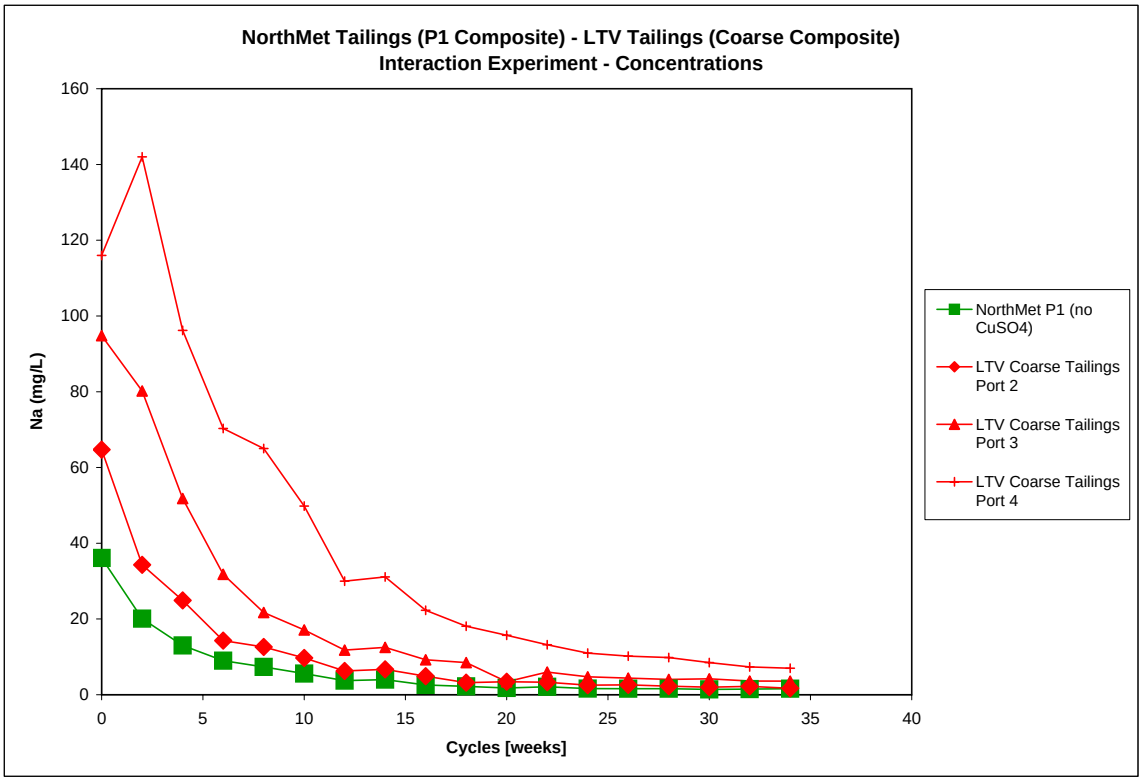
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.51



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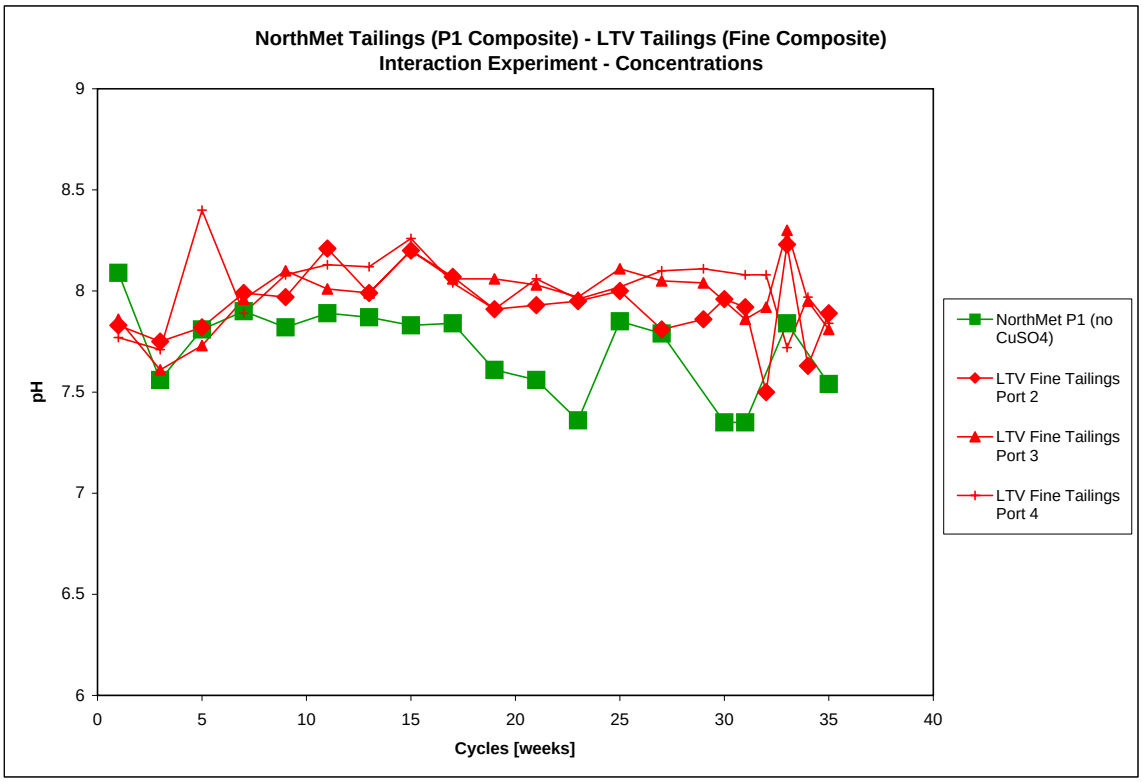
Graph C.3.52



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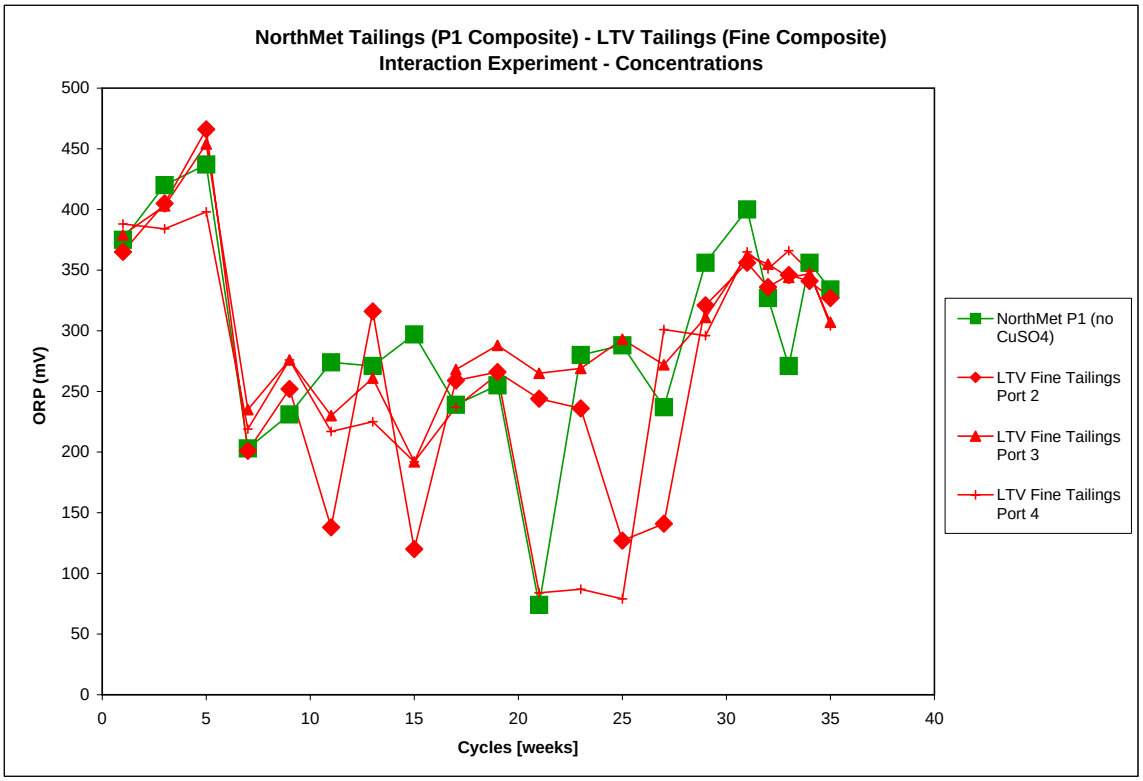
Graph C.3.53



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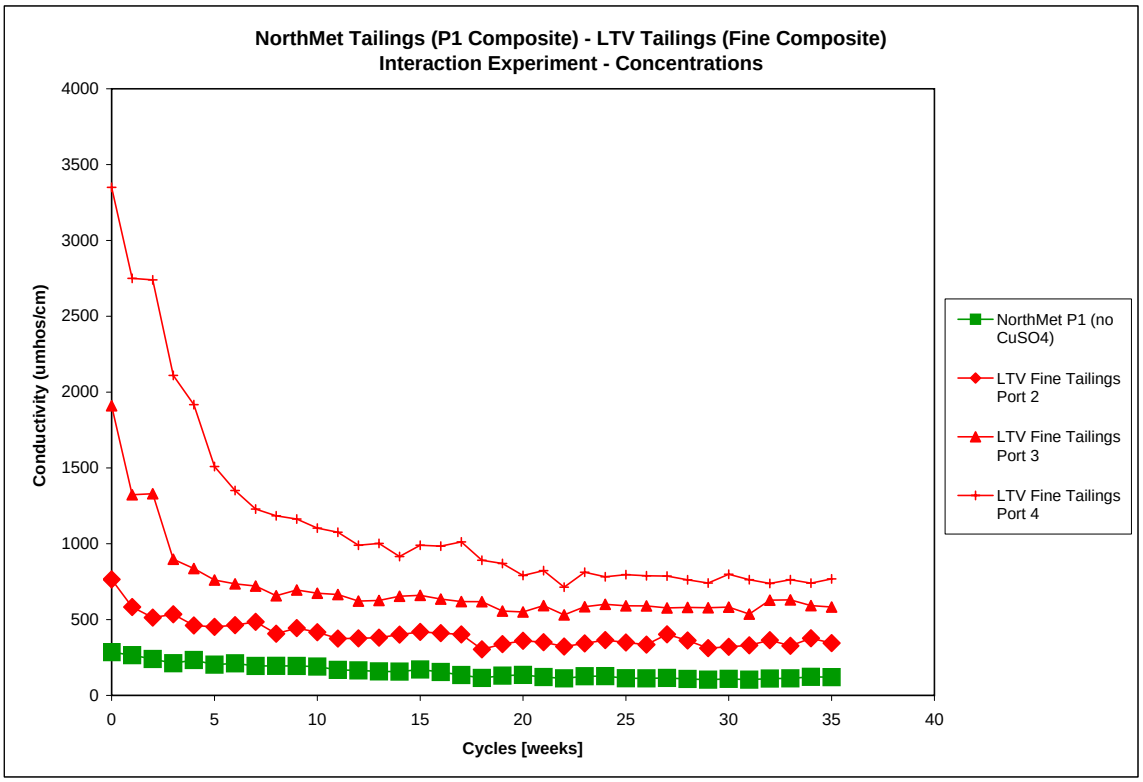
Graph C.3.54



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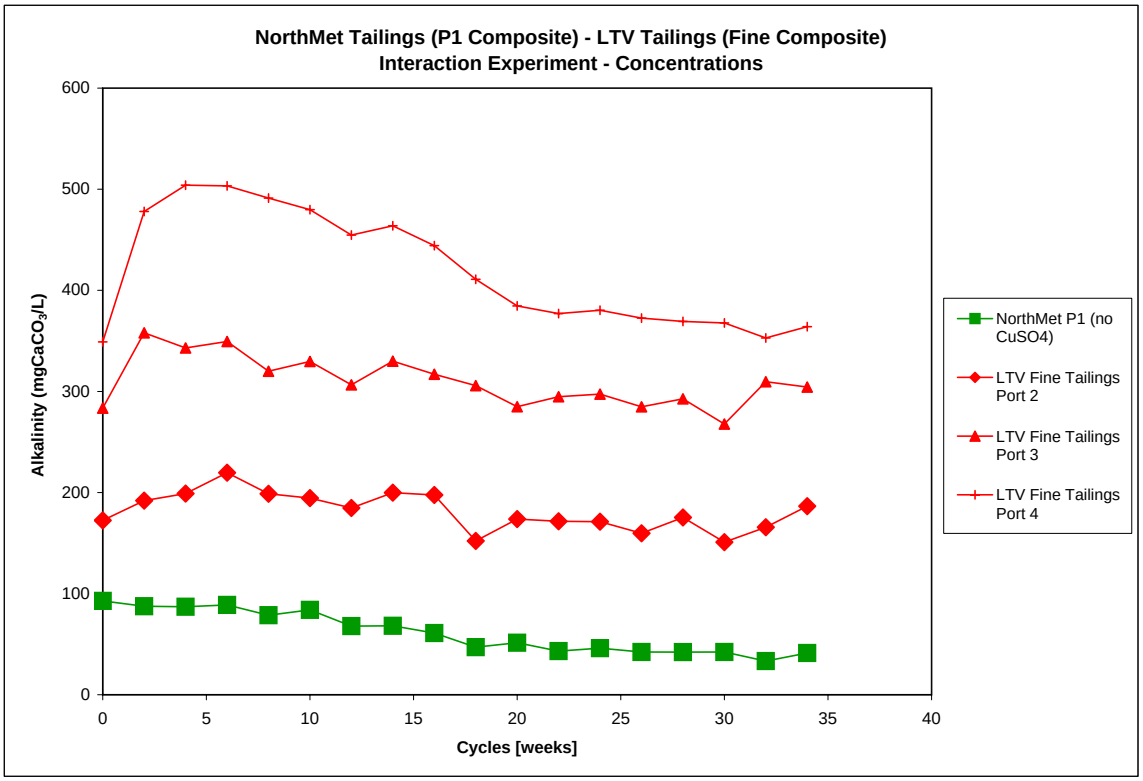
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.55



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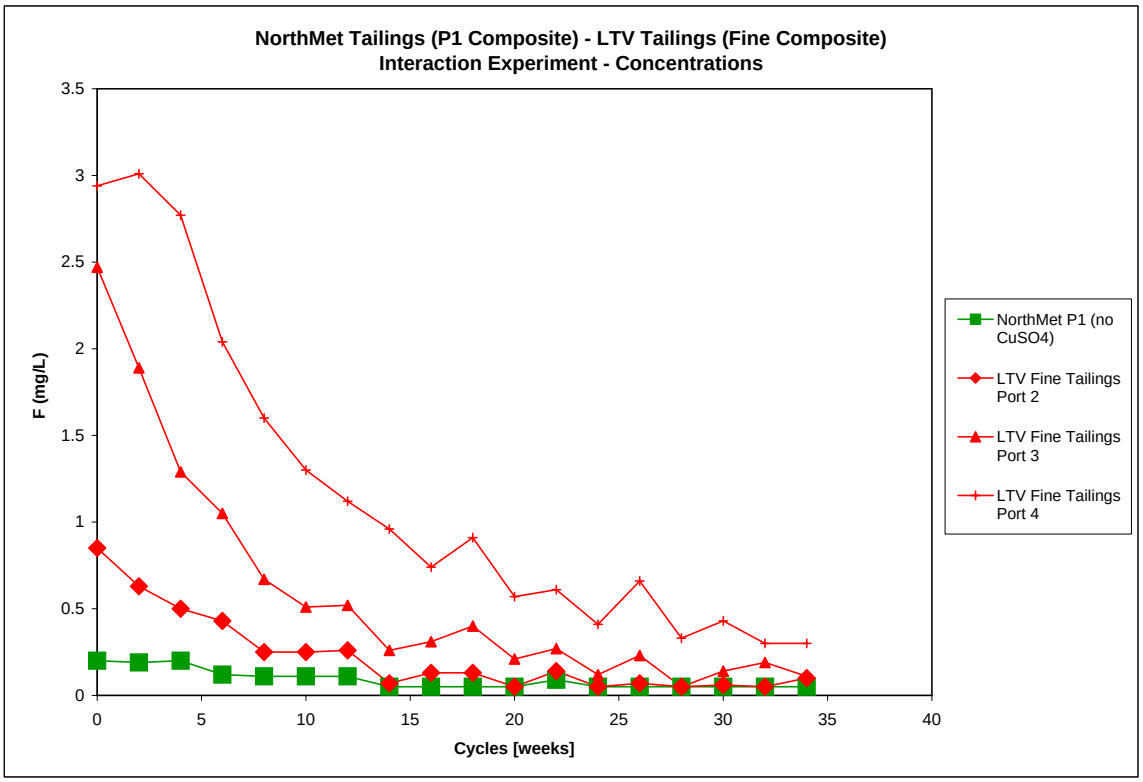
LTVSMC Contact Experiment

Graph C.3.56



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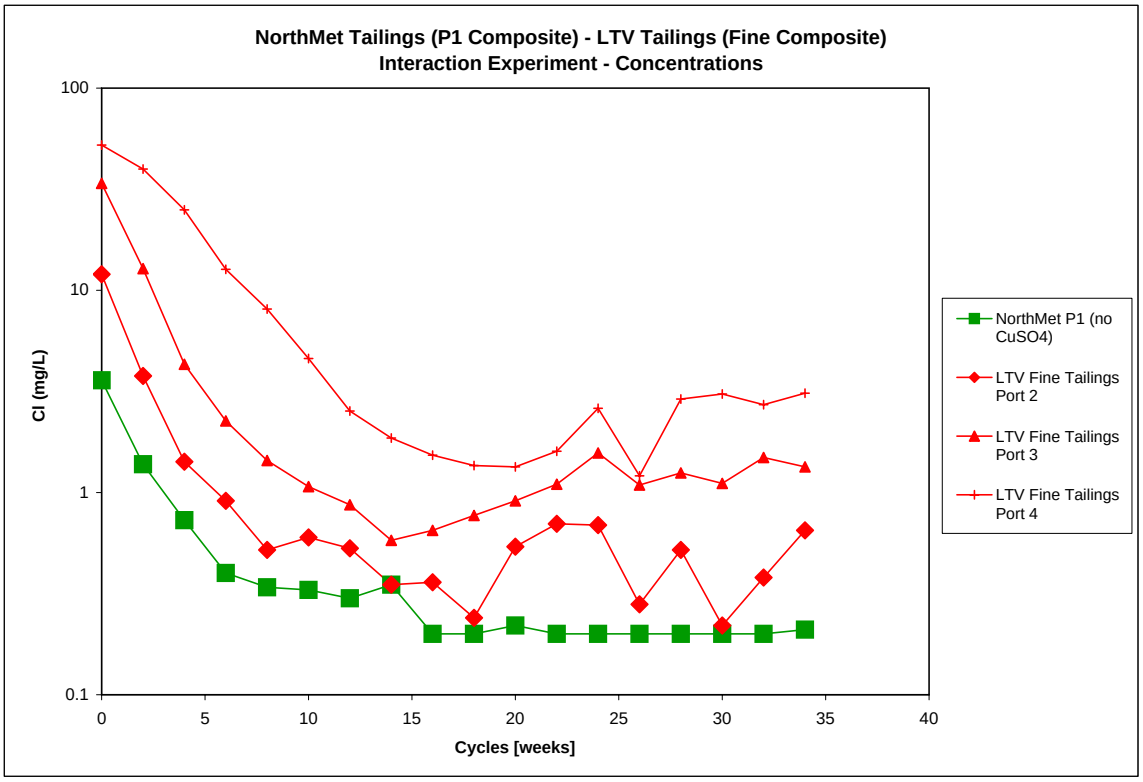
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.57



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Appendix C.3
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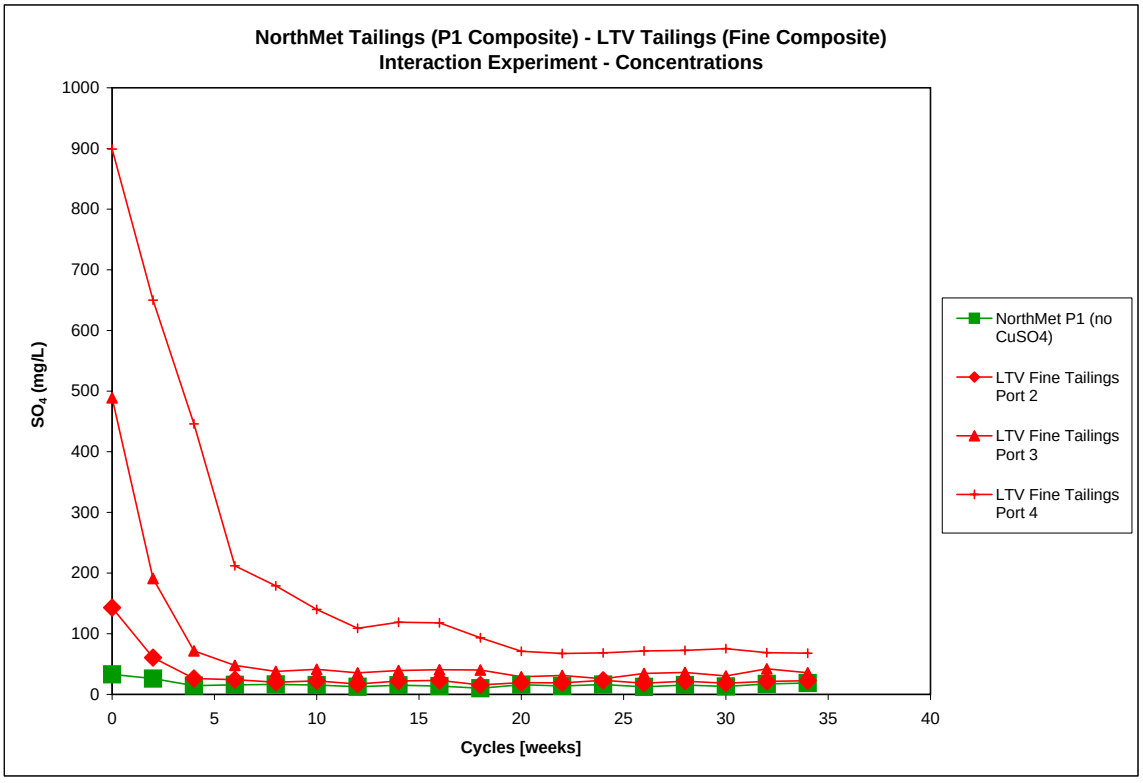
Graph C.3.58



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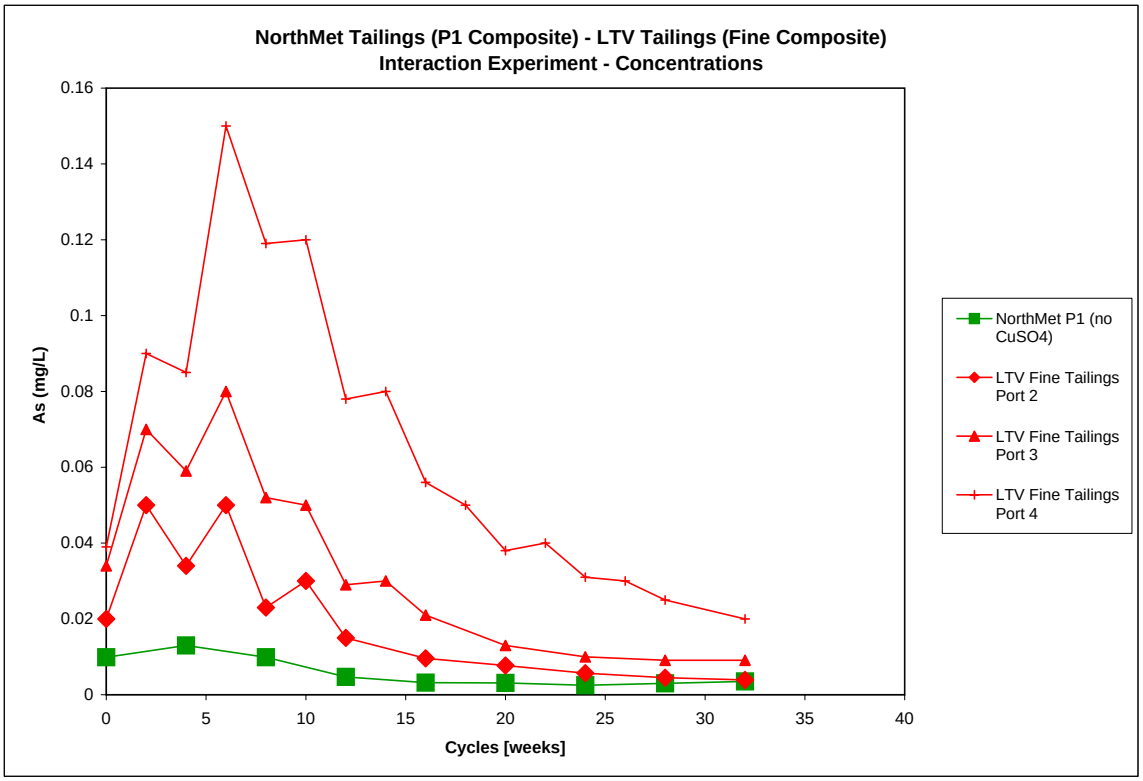
Graph C.3.59



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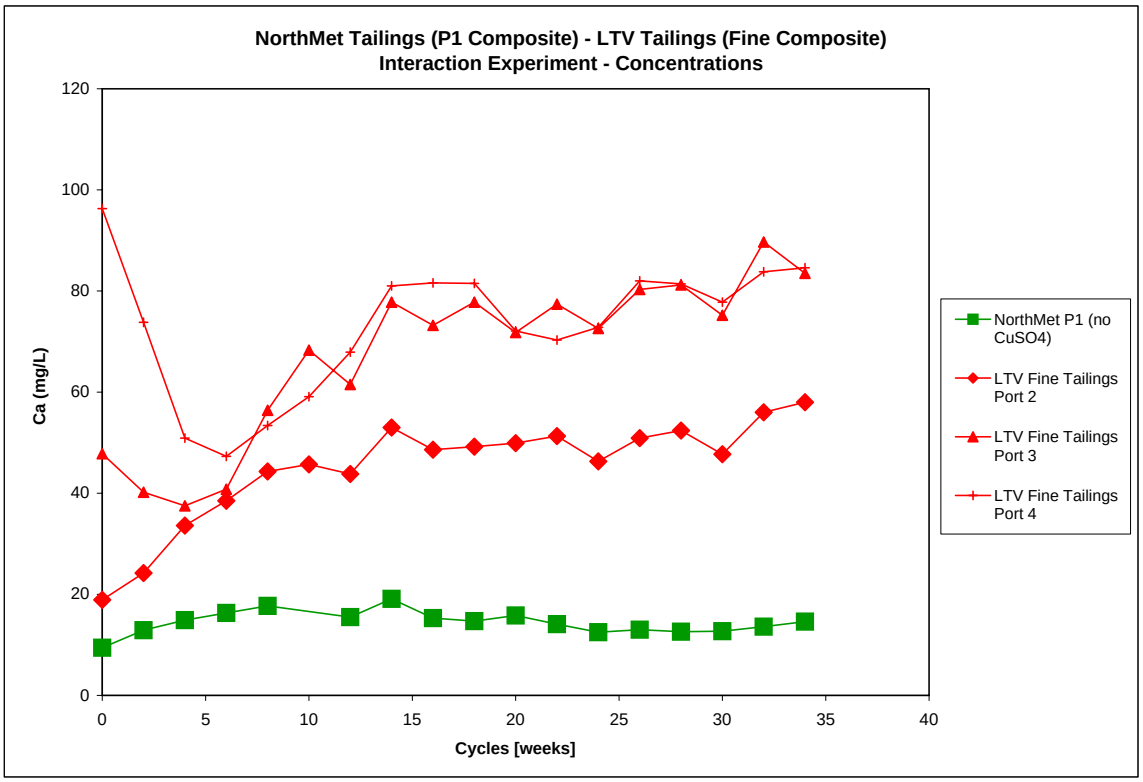
Graph C.3.60



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Appendix C.3
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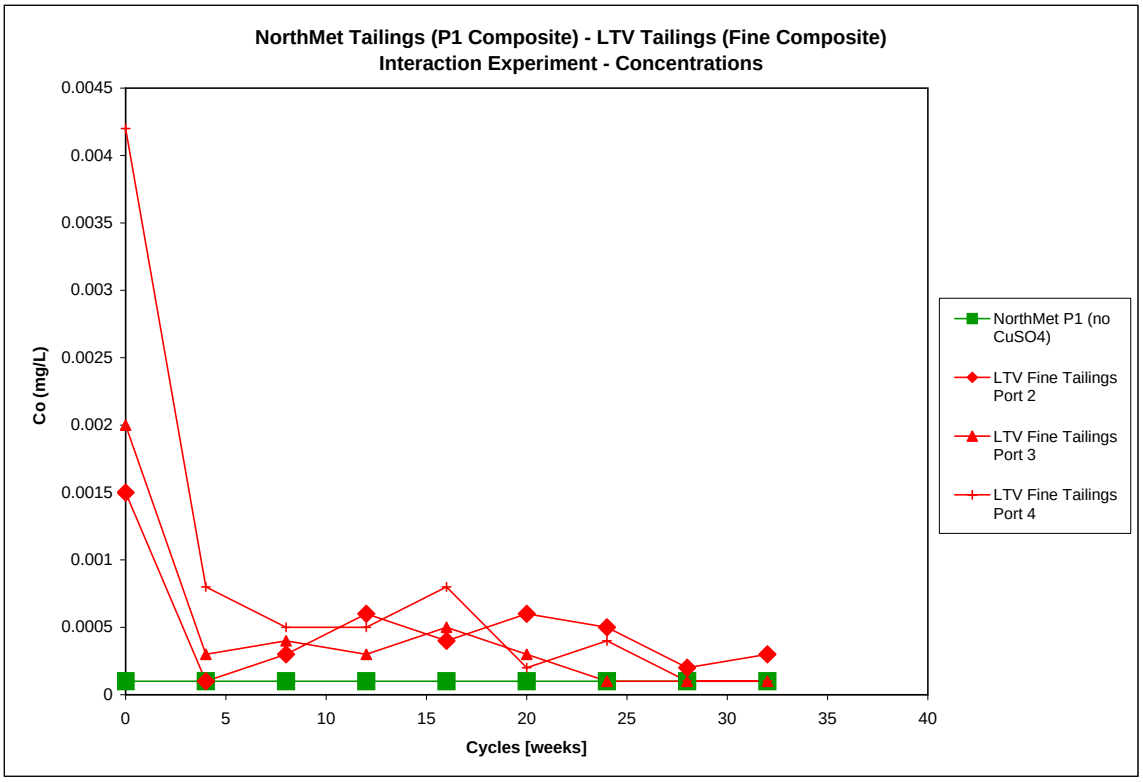
Graph C.3.61



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Appendix C.3
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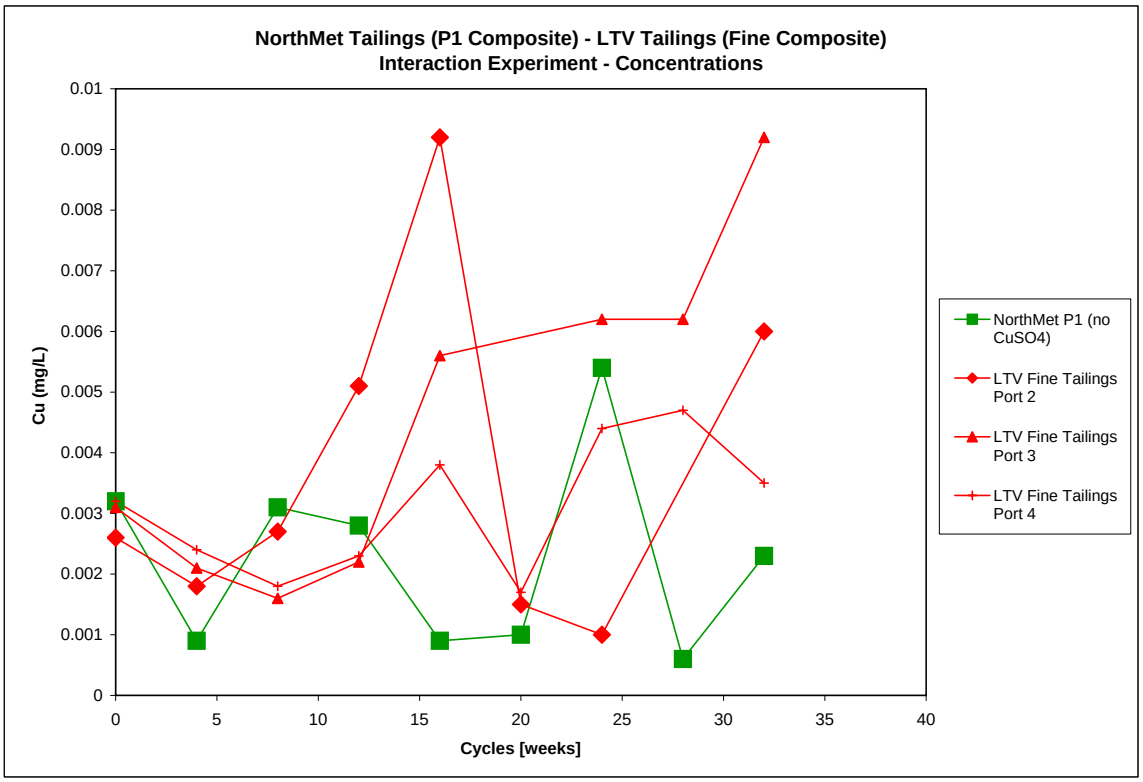
Graph C.3.62



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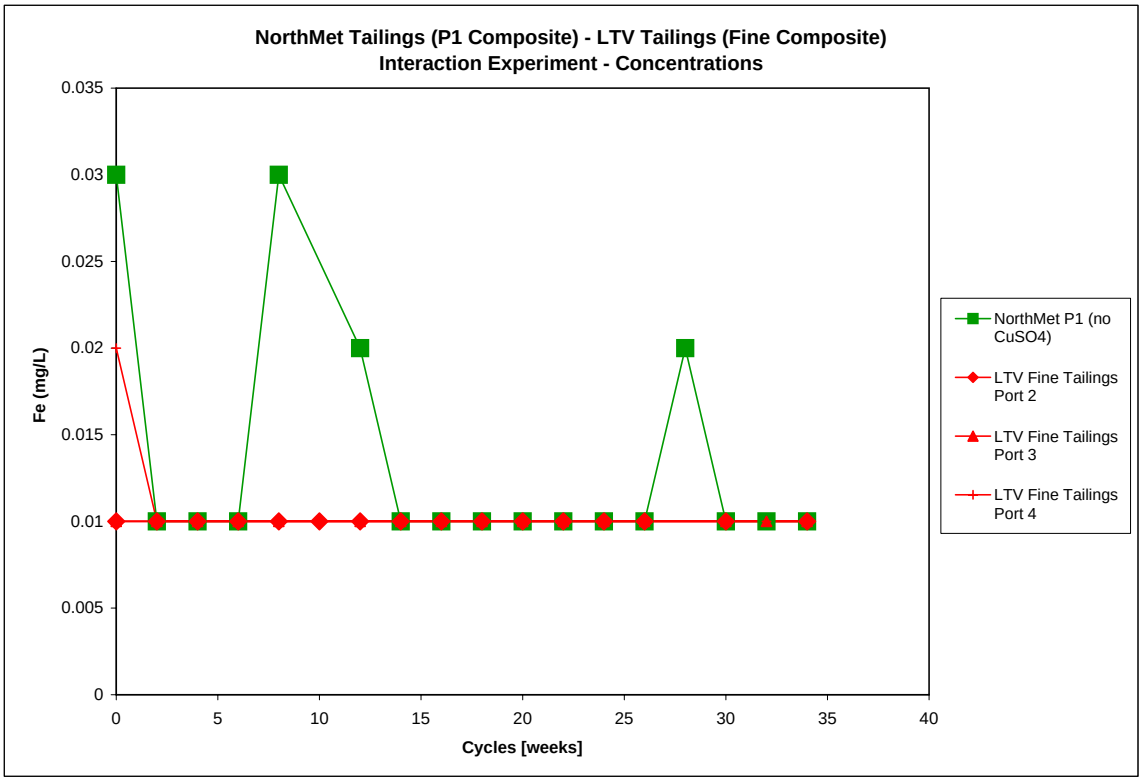
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.63



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Appendix C.3
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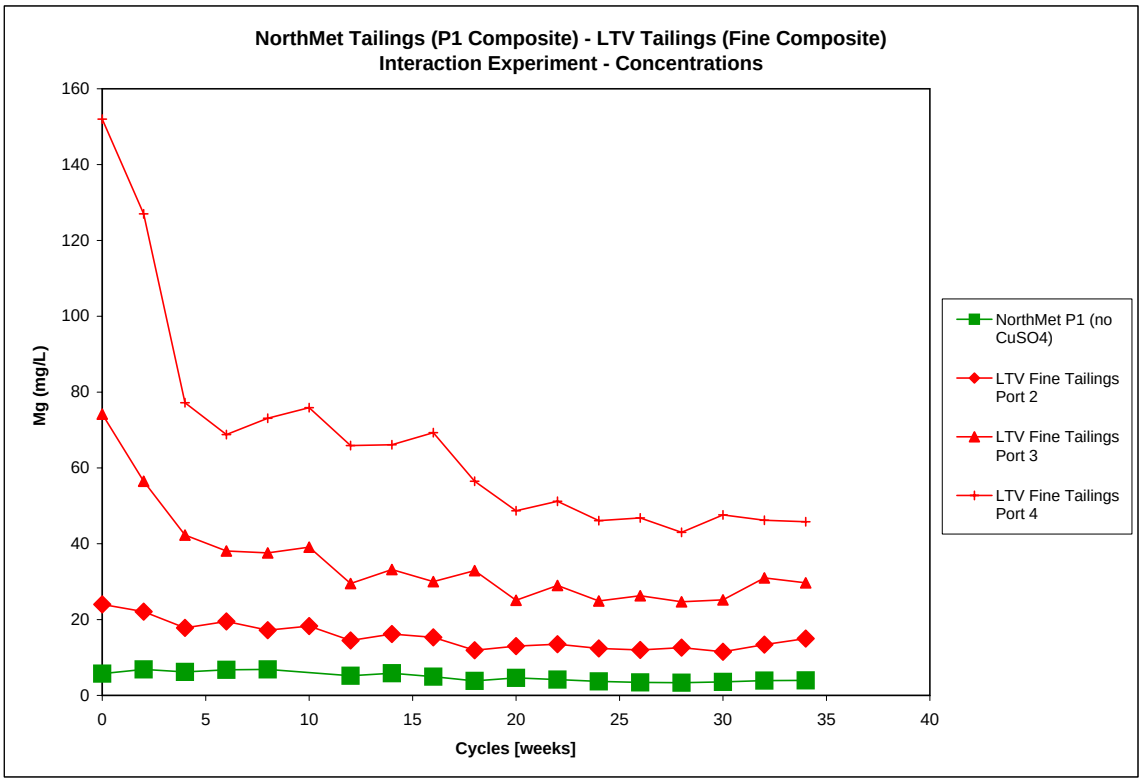
Graph C.3.64



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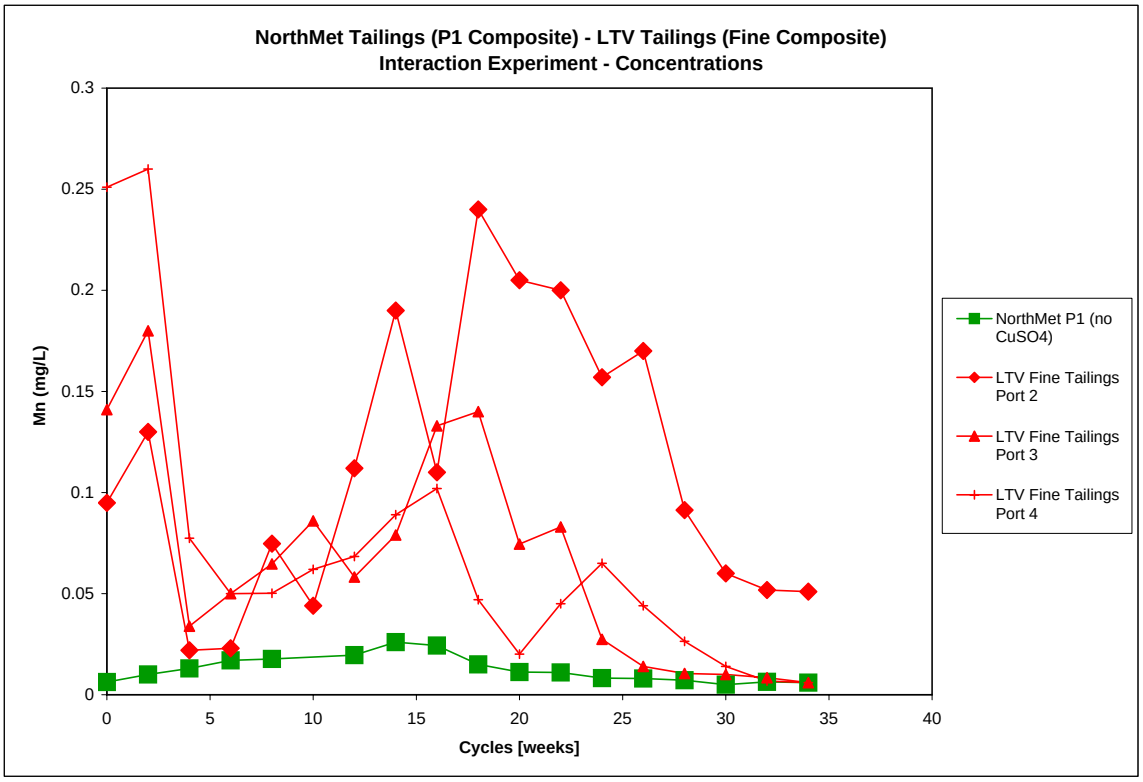
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.65



LTV_combined.xls

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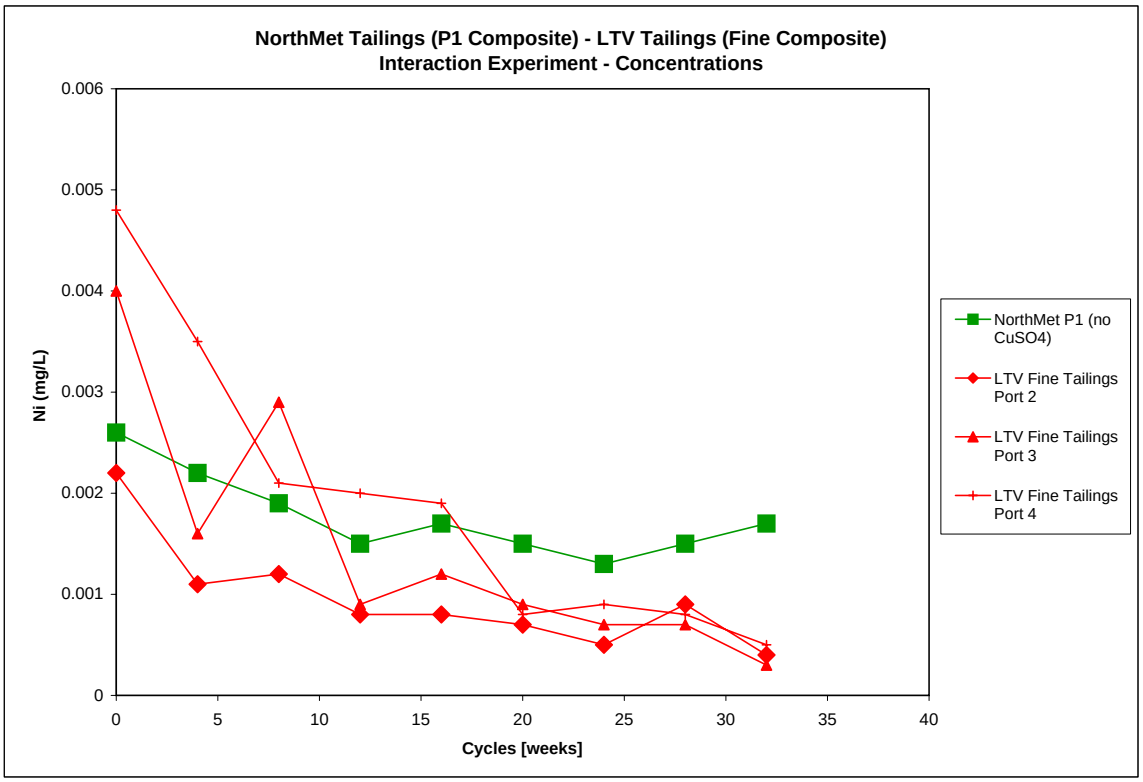
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.66



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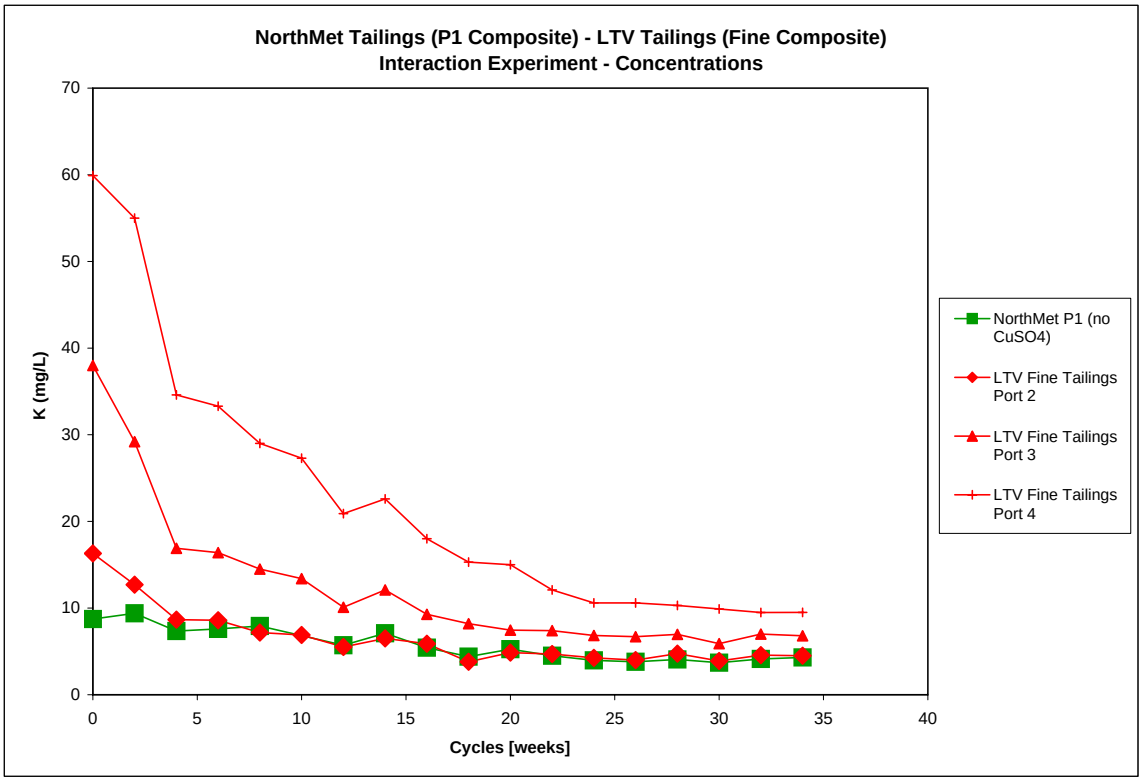
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.67



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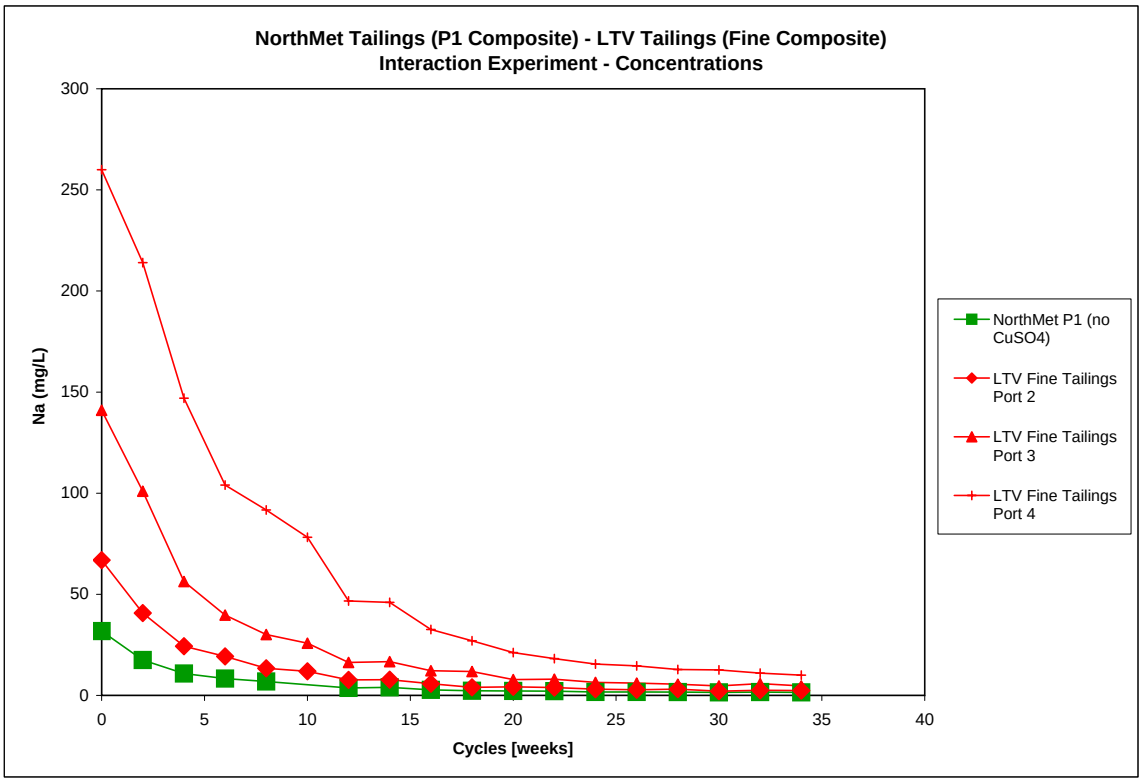
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.68



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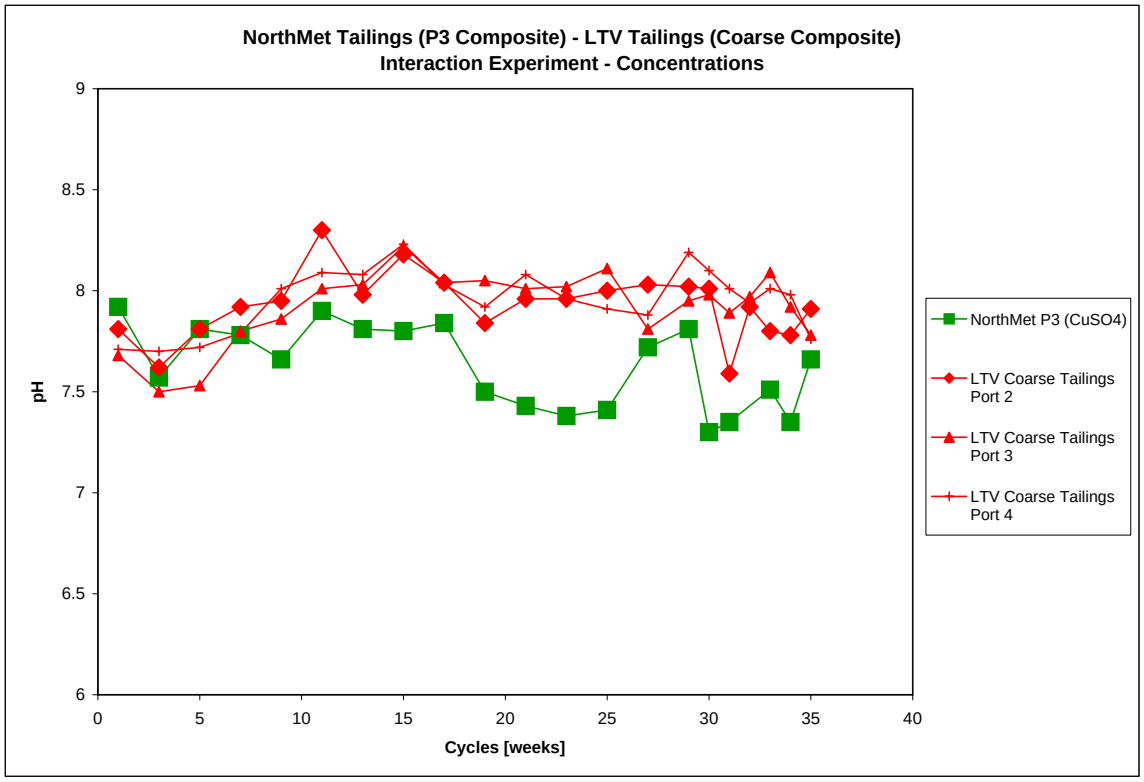
Graph C.3.69



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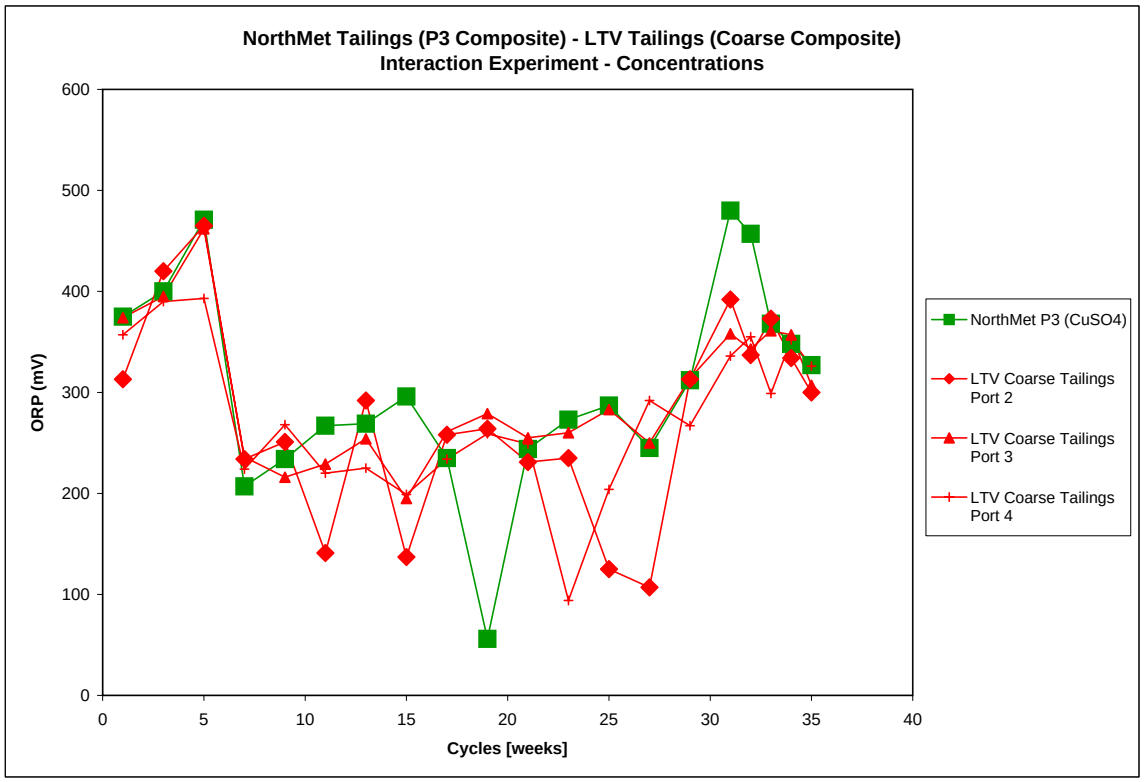
Graph C.3.70



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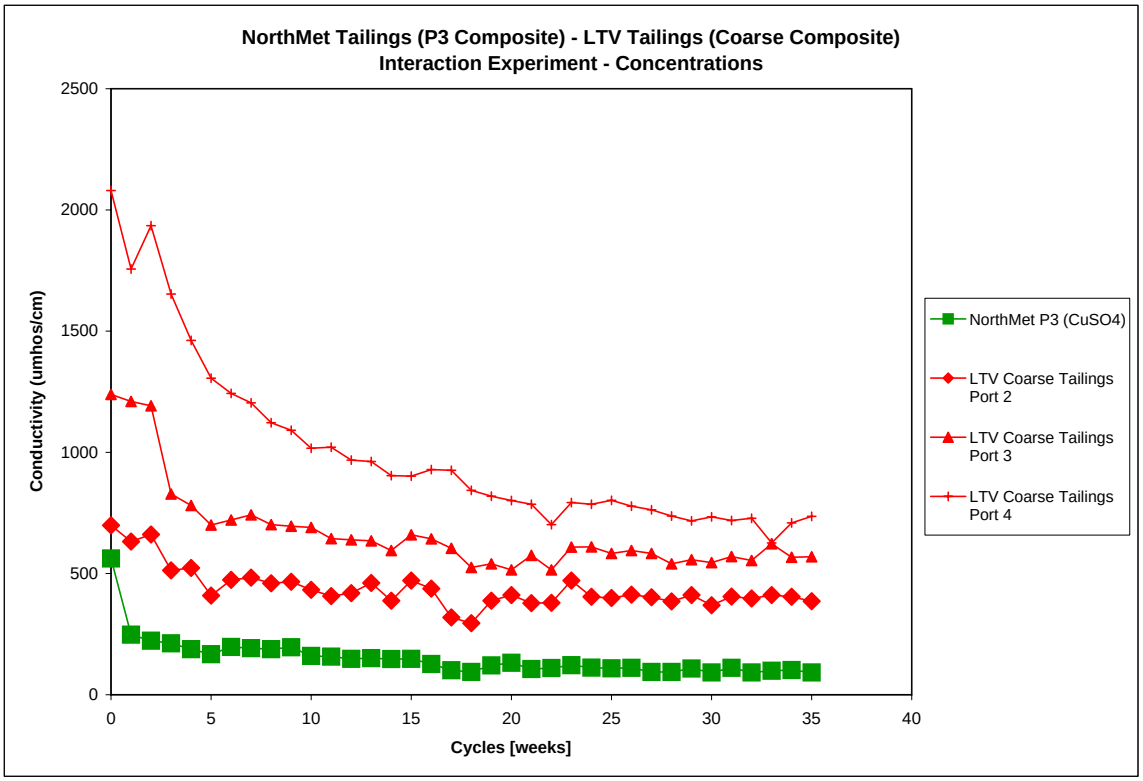
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.71



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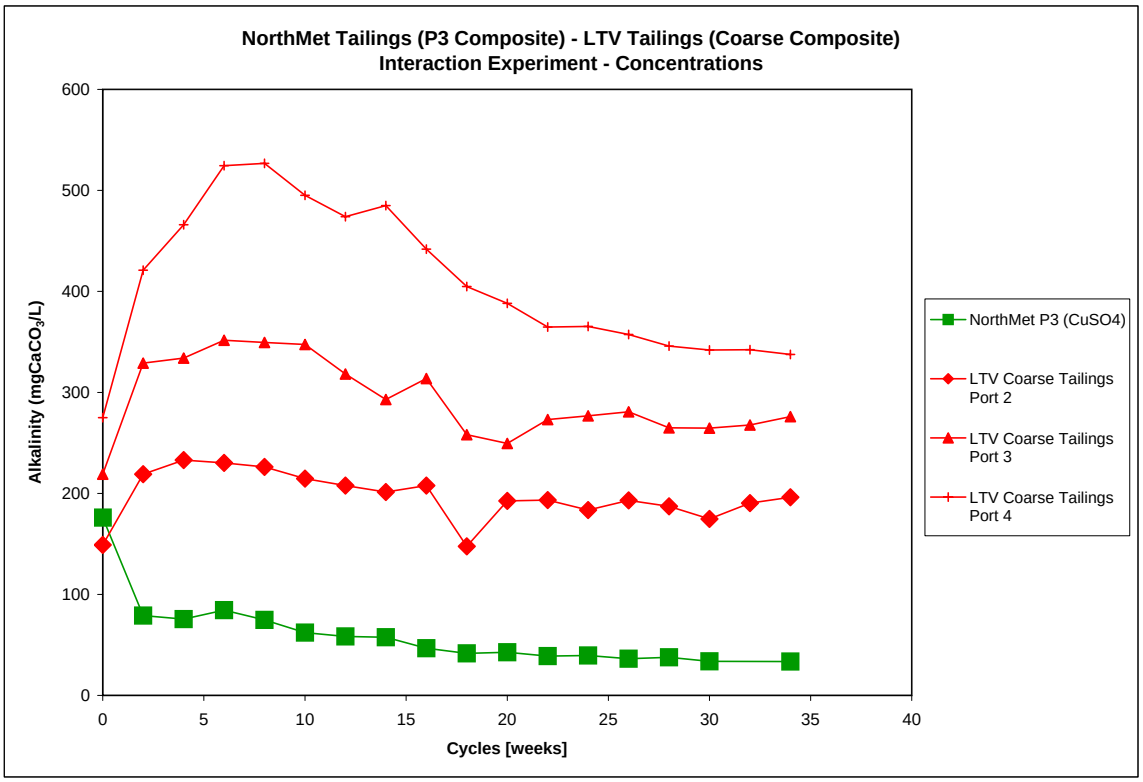
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.72



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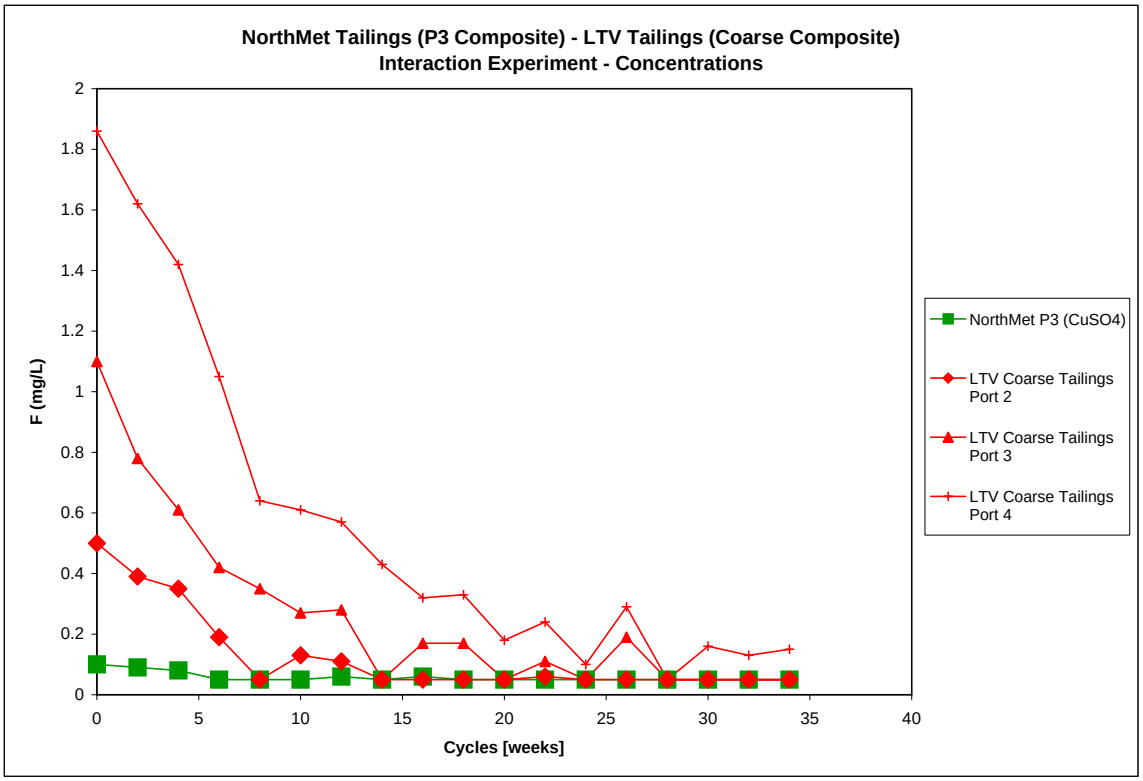
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.73



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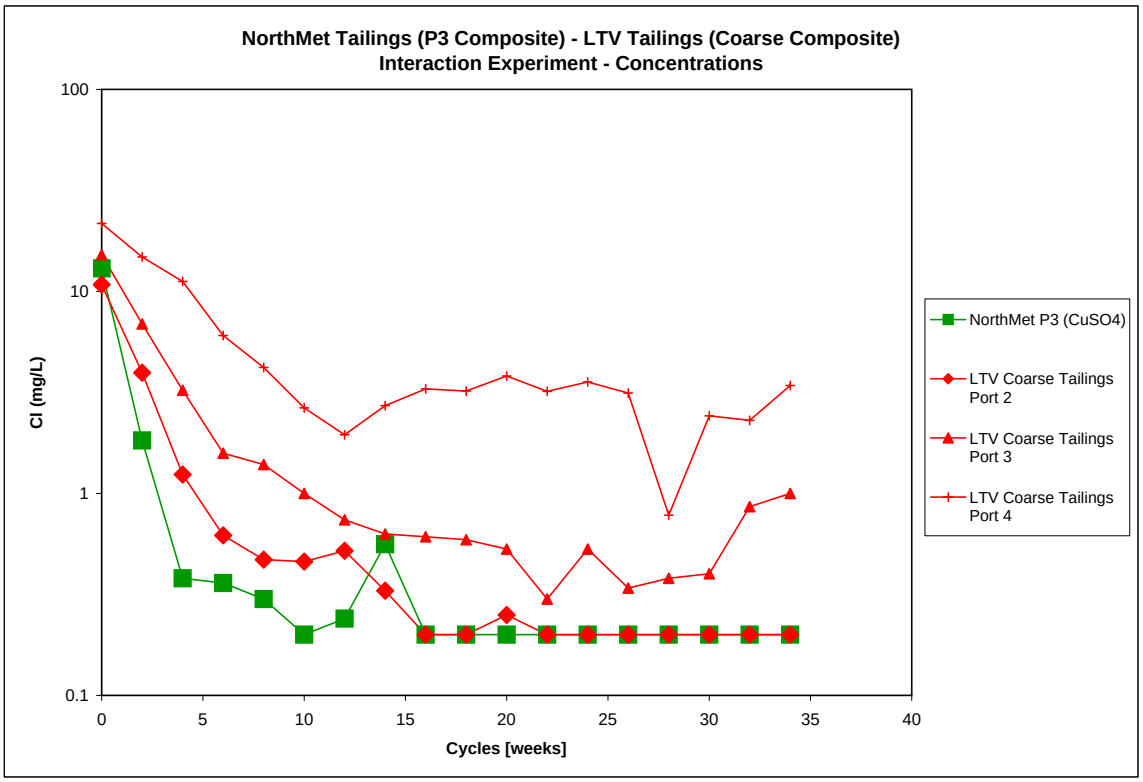
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.74



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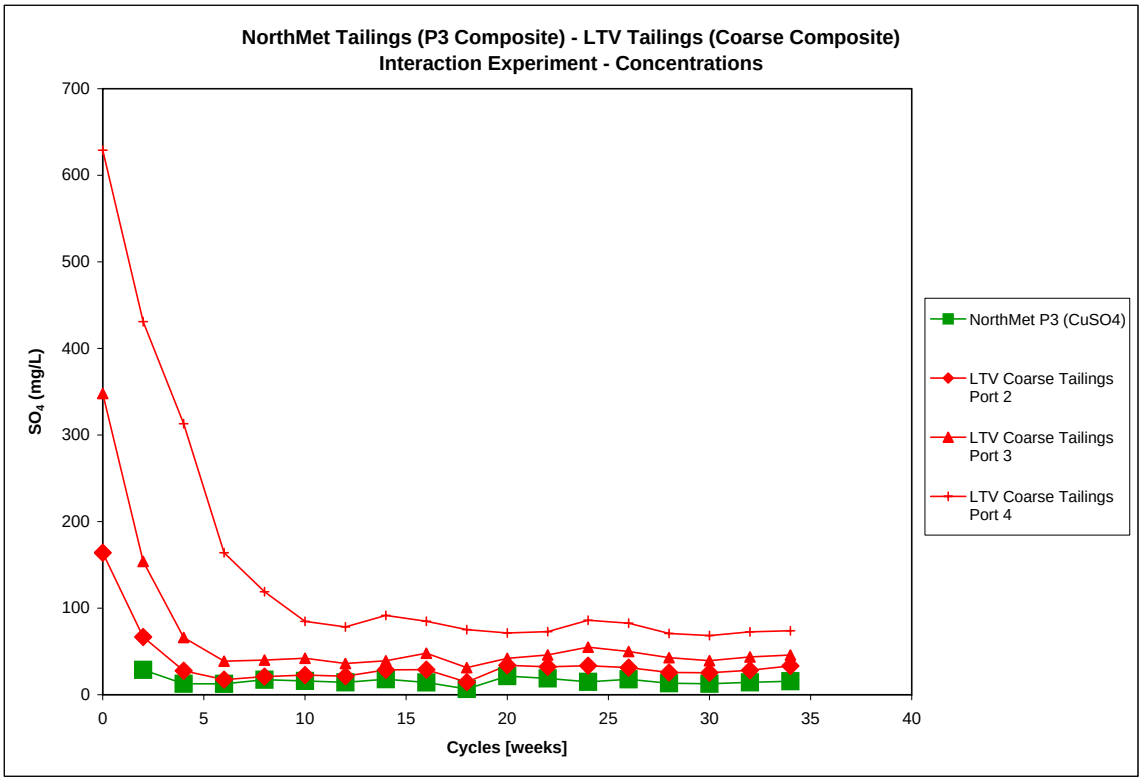
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.75



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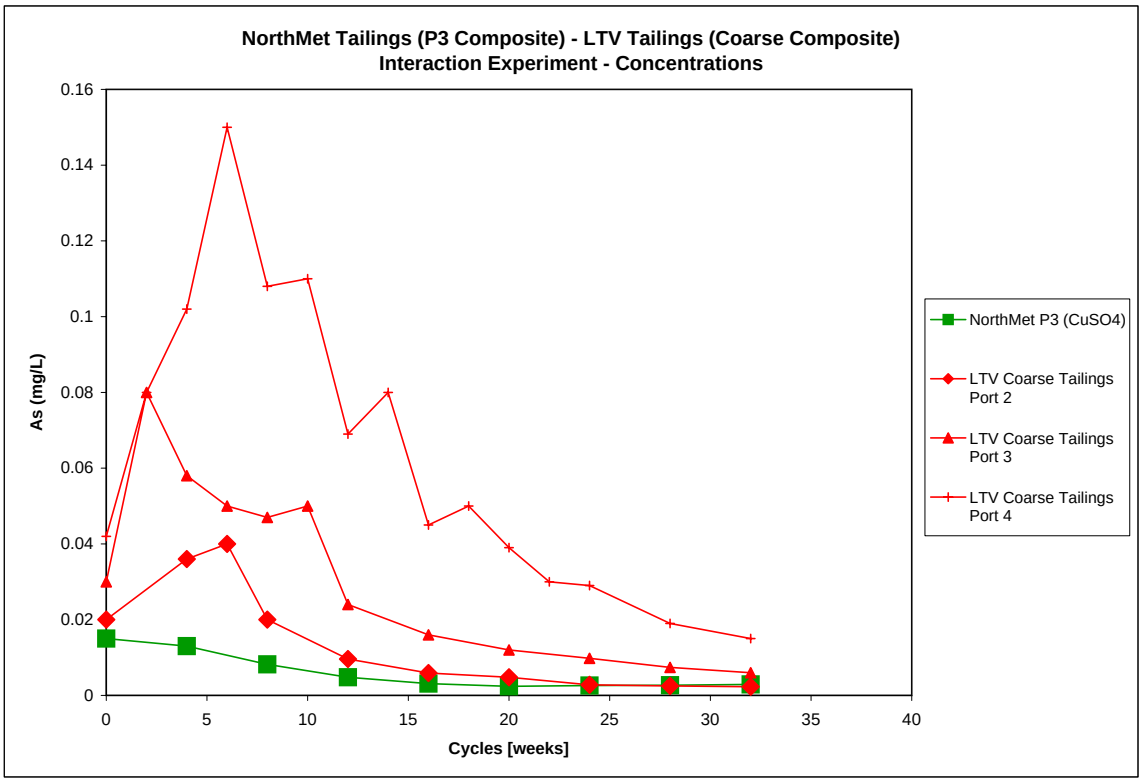
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.76



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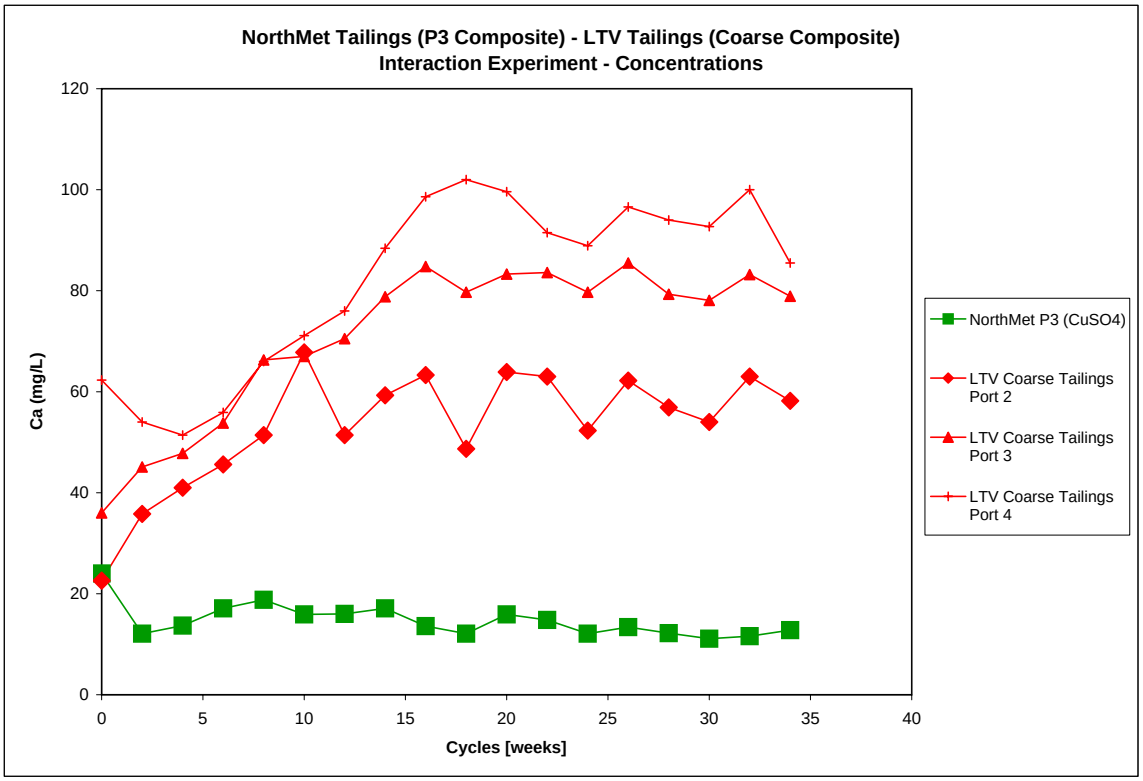
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.77



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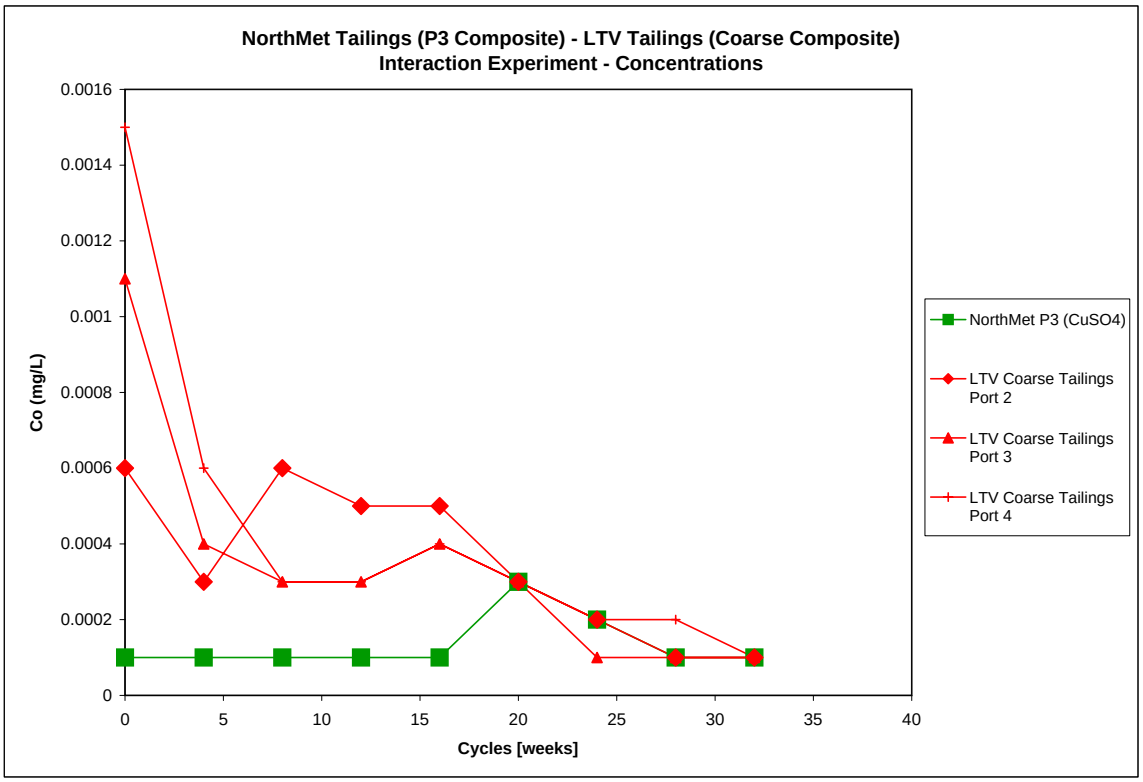
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.78



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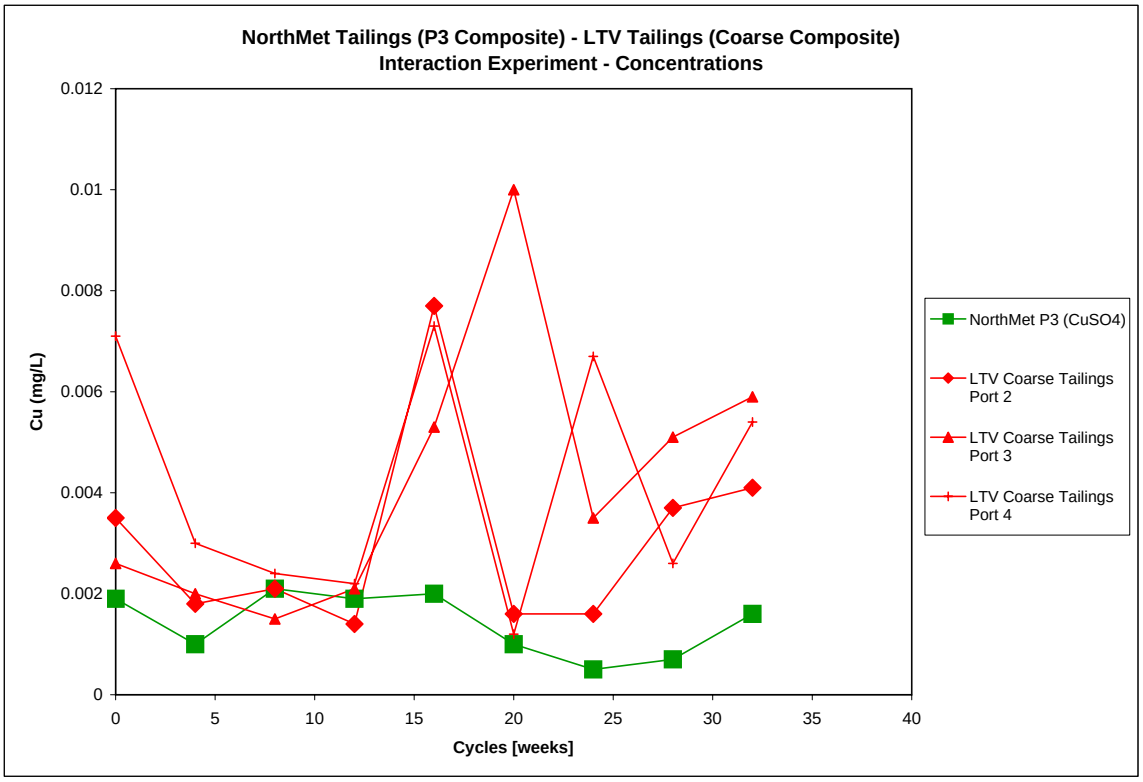
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.79



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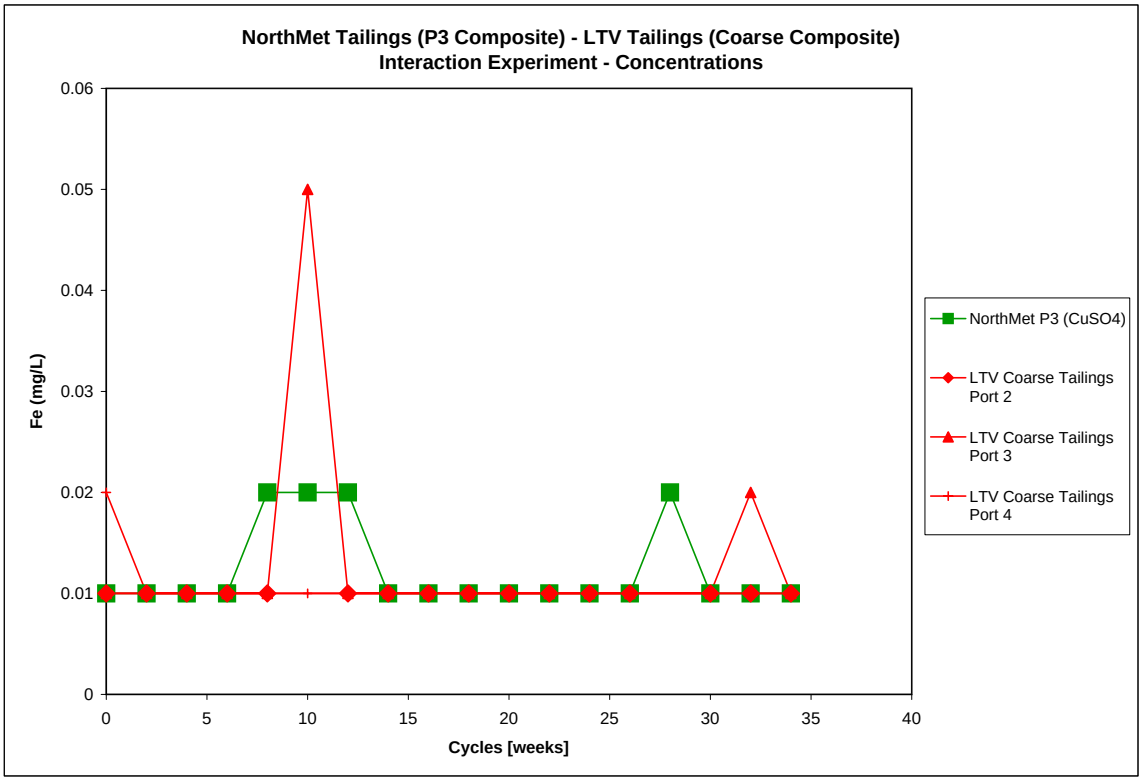
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.80



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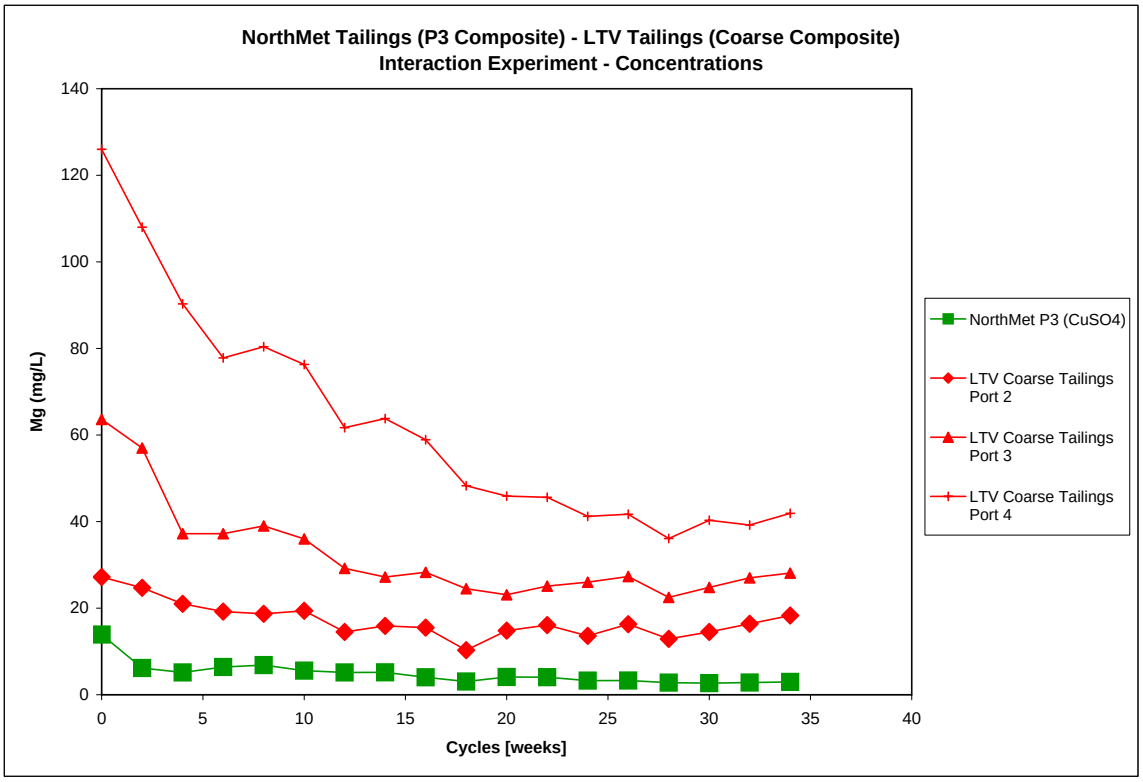
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.81



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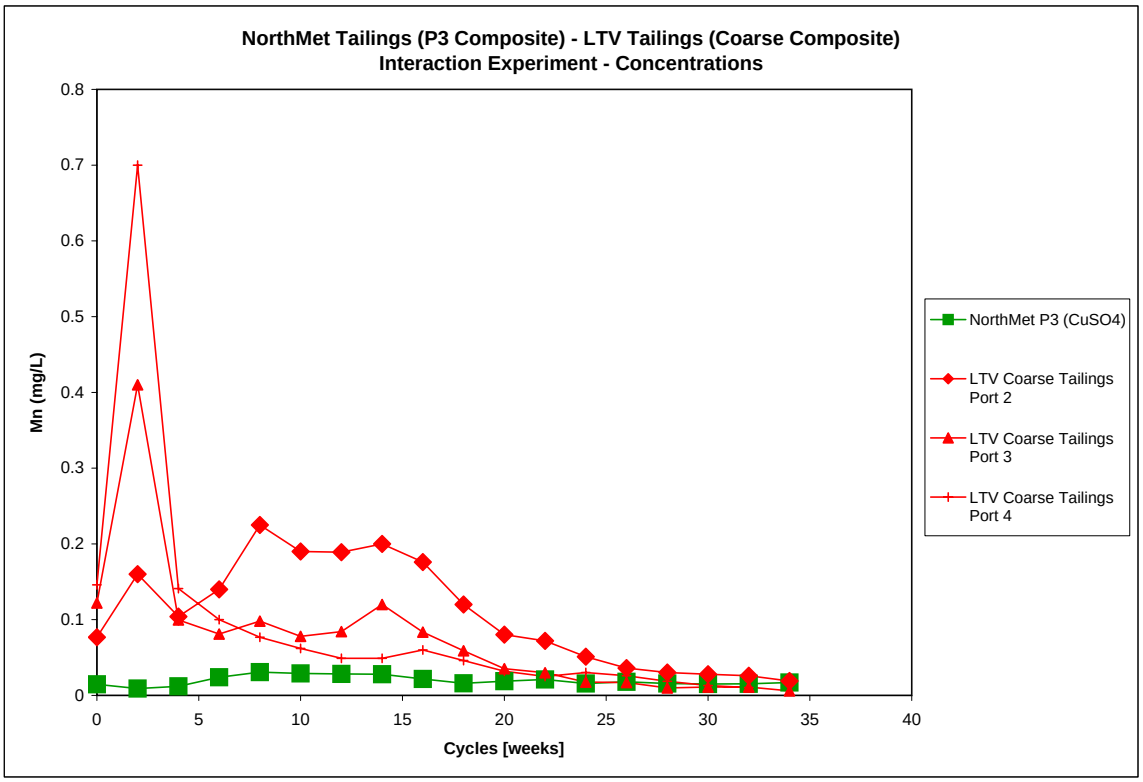
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.82



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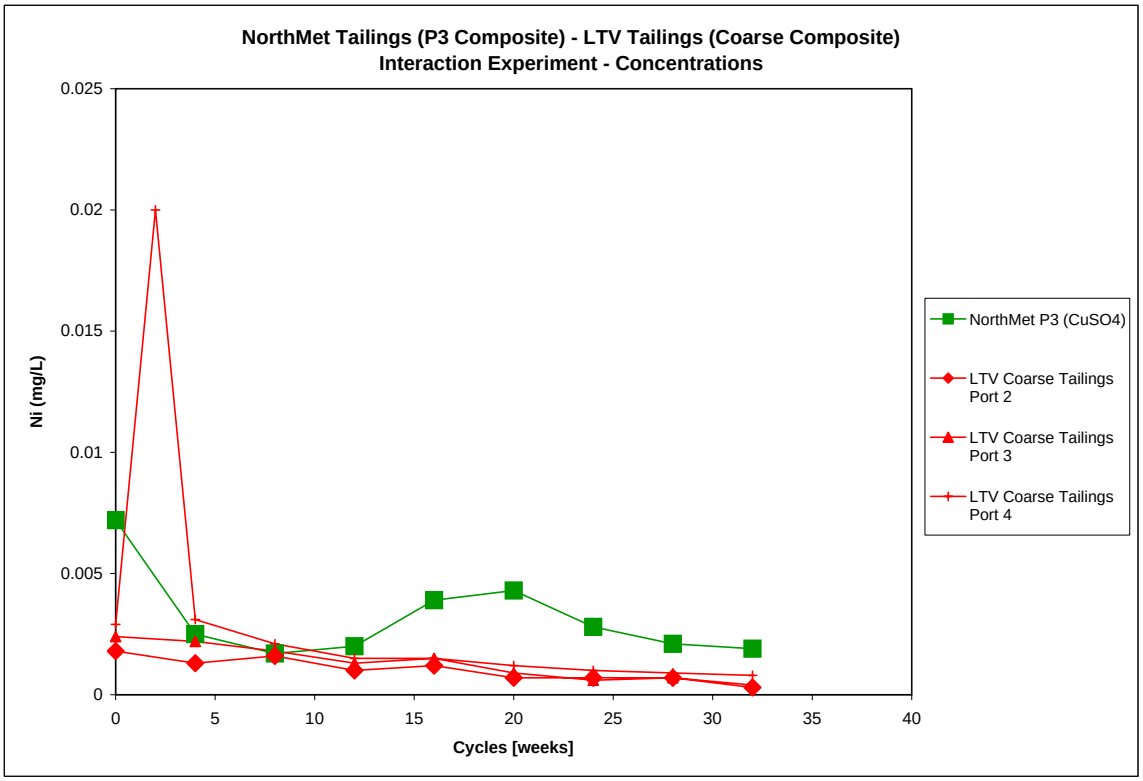
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.83



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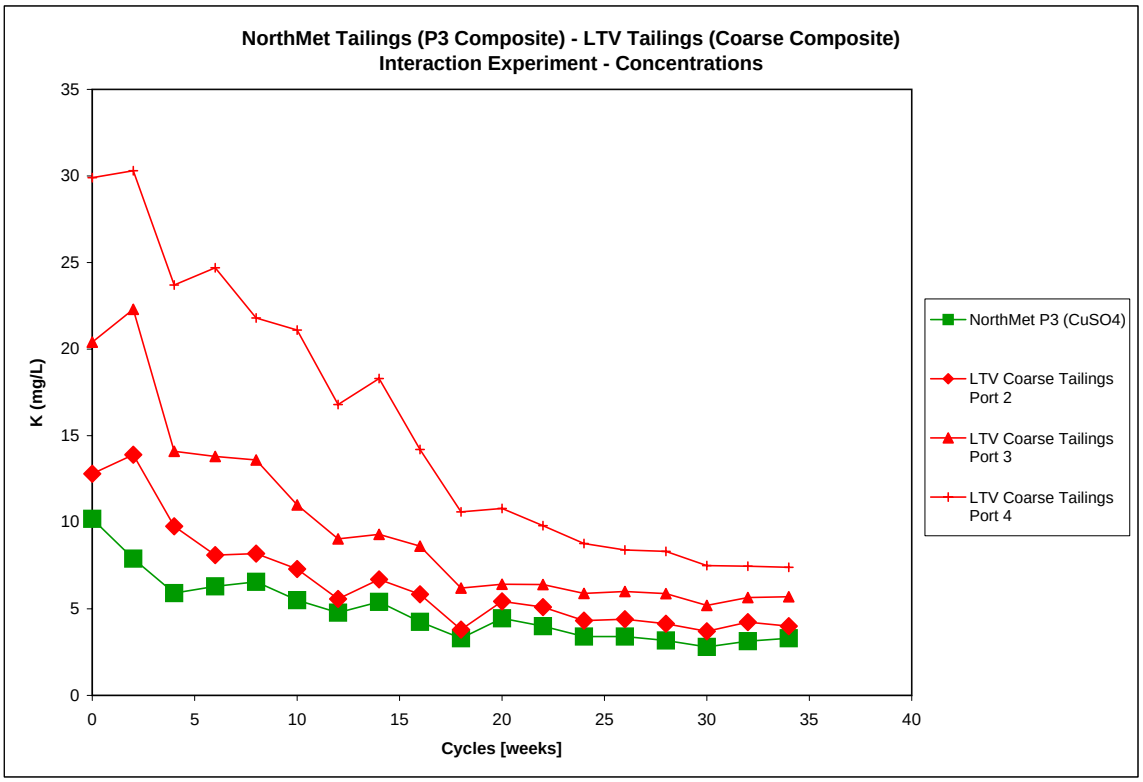
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.84



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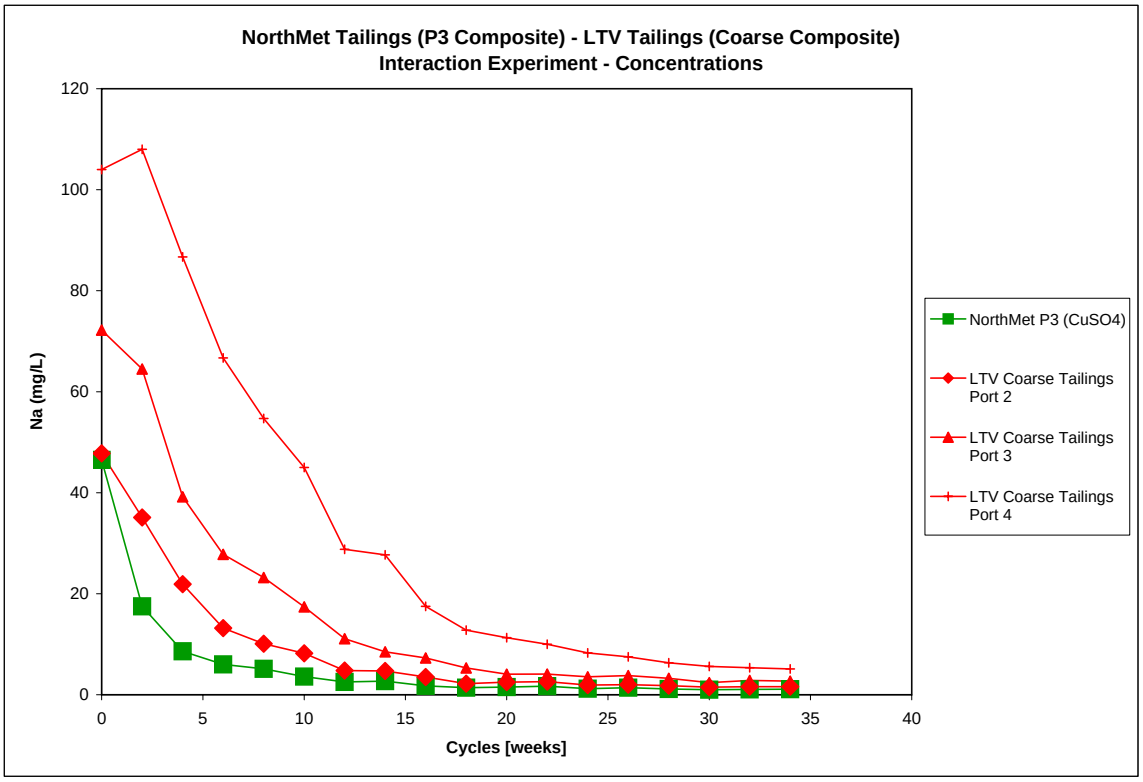
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.85



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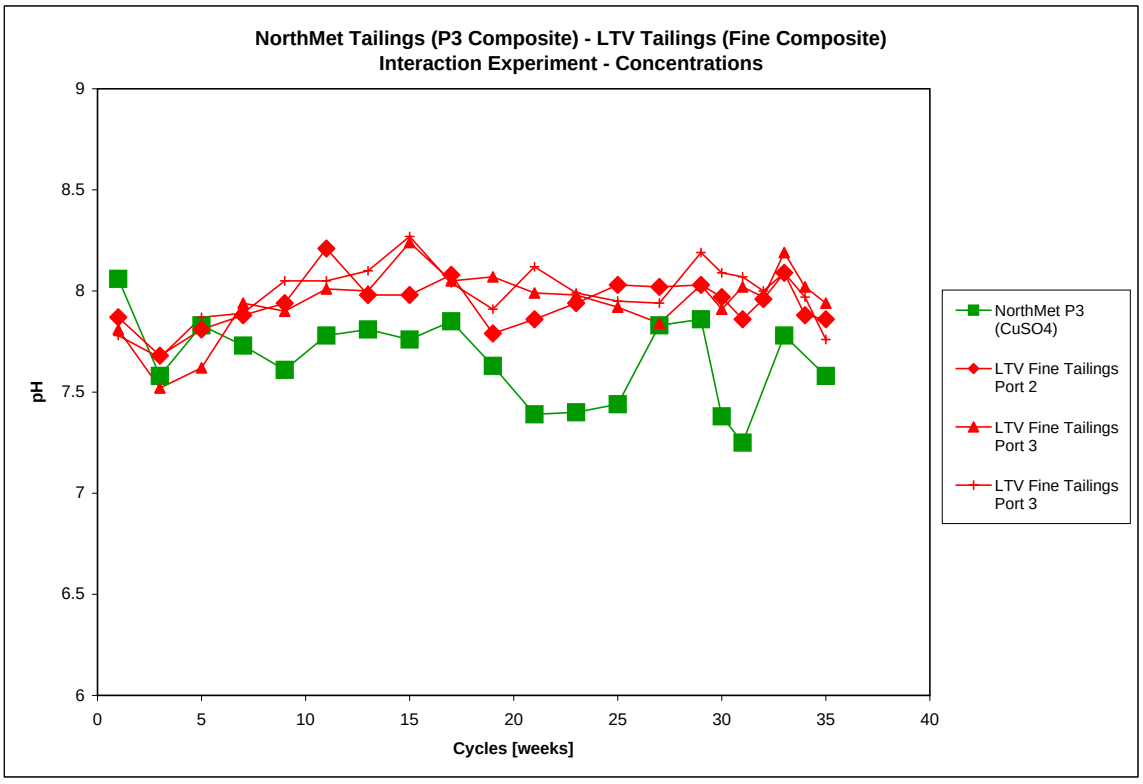
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.86



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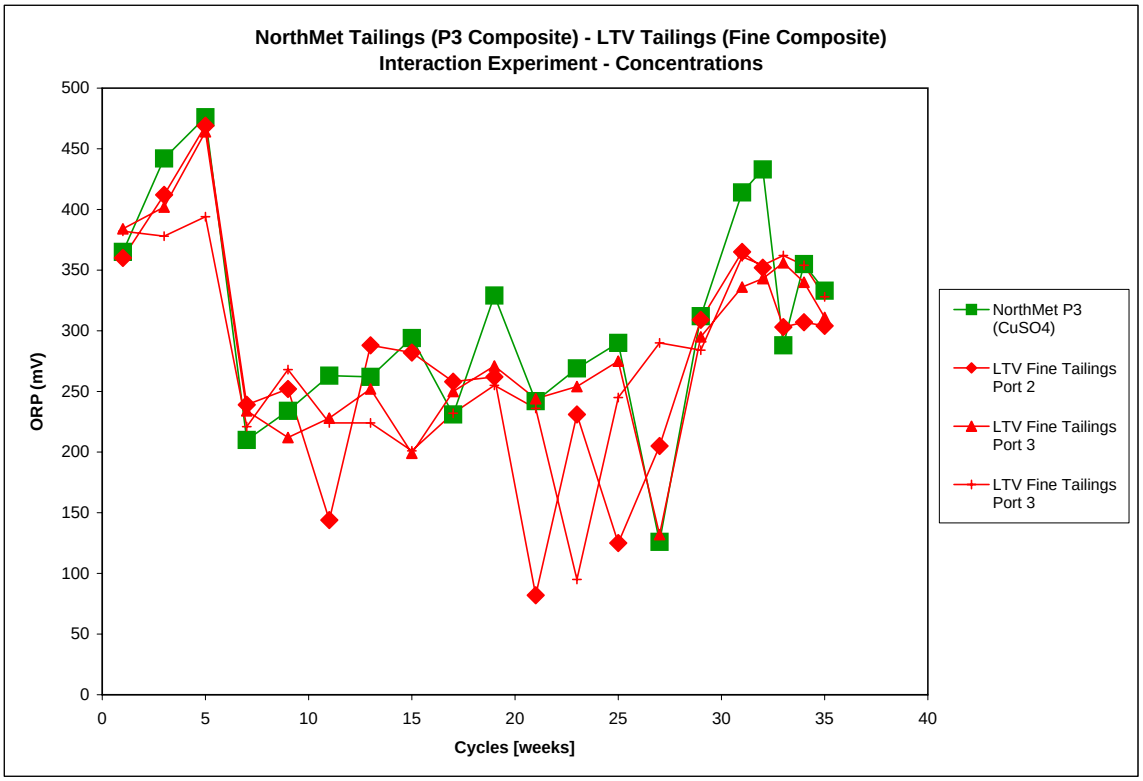
Graph C.3.87



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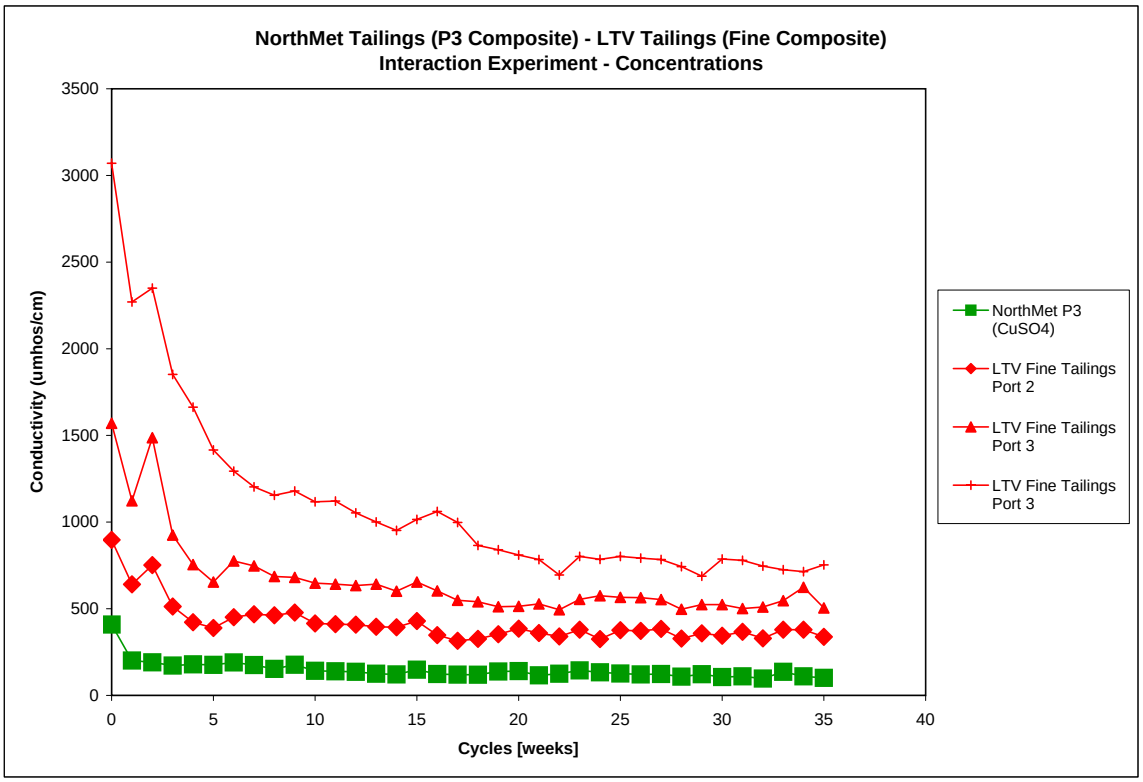
Graph C.3.88



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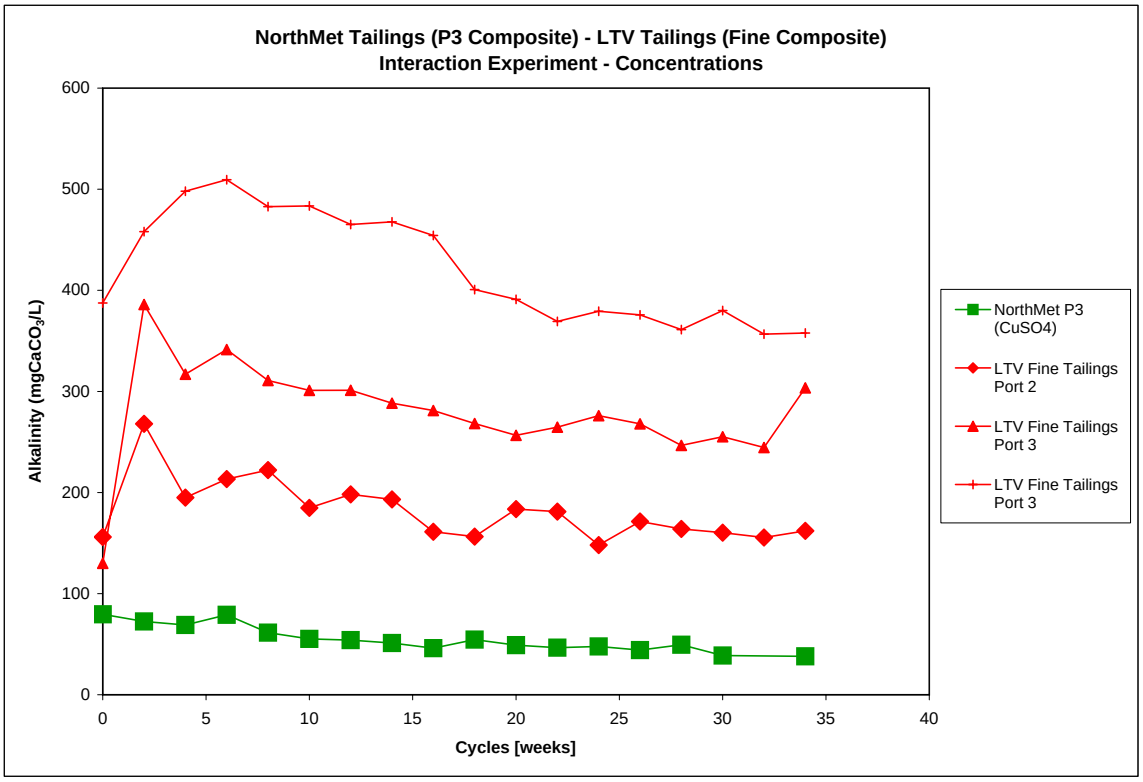
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.89



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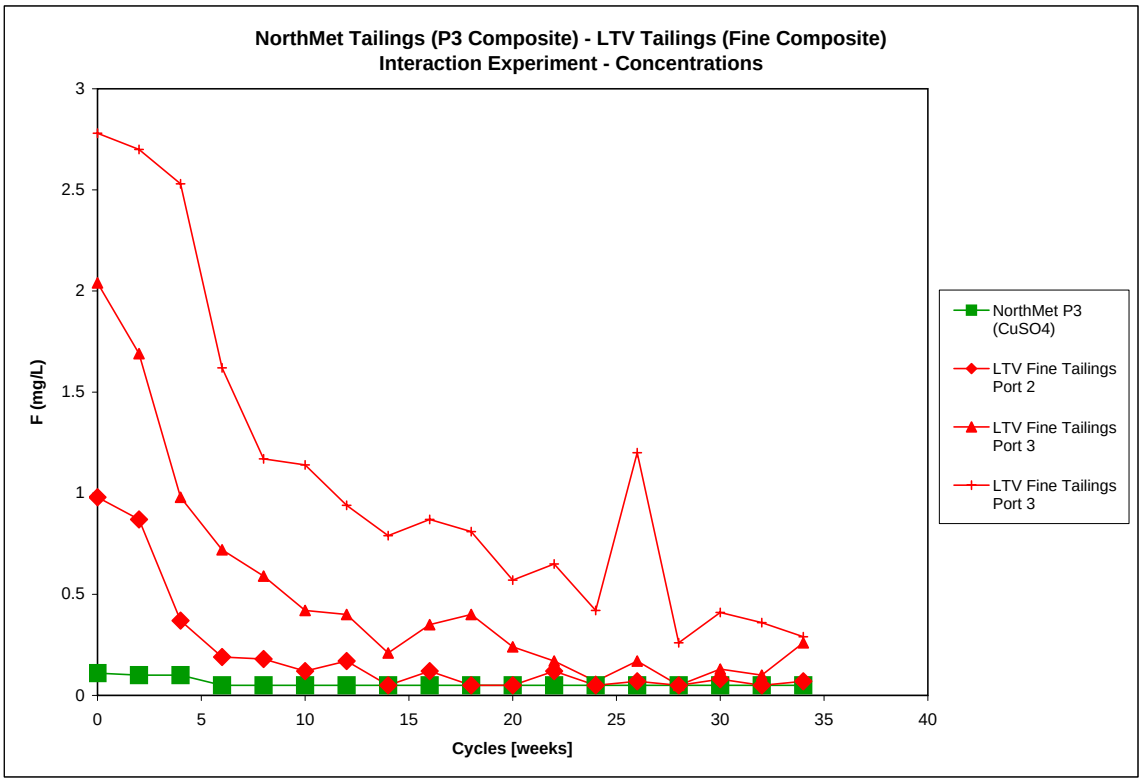
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.90



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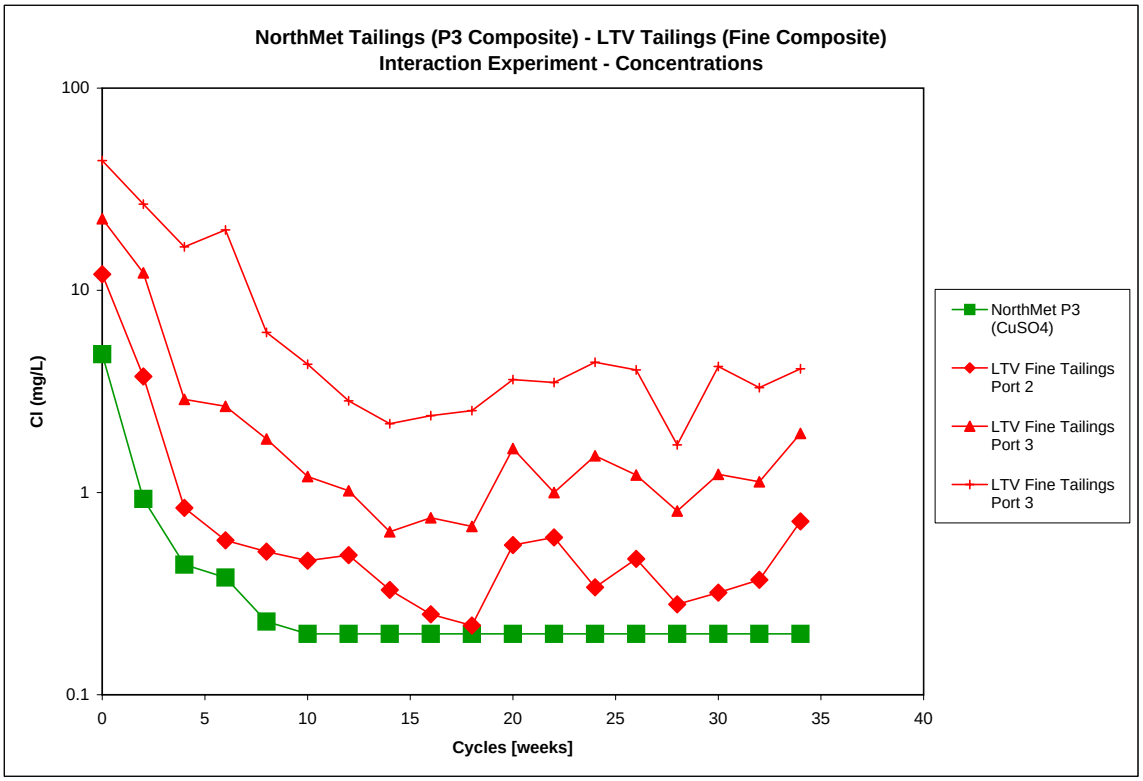
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.91



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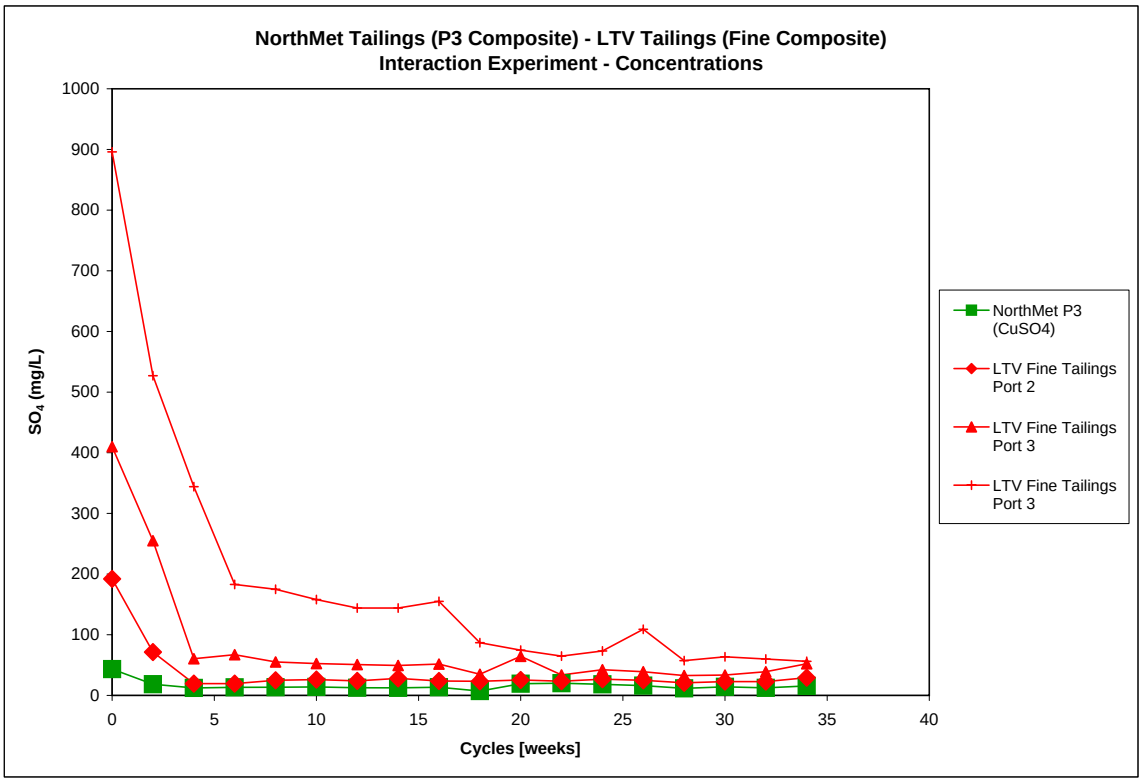
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.92



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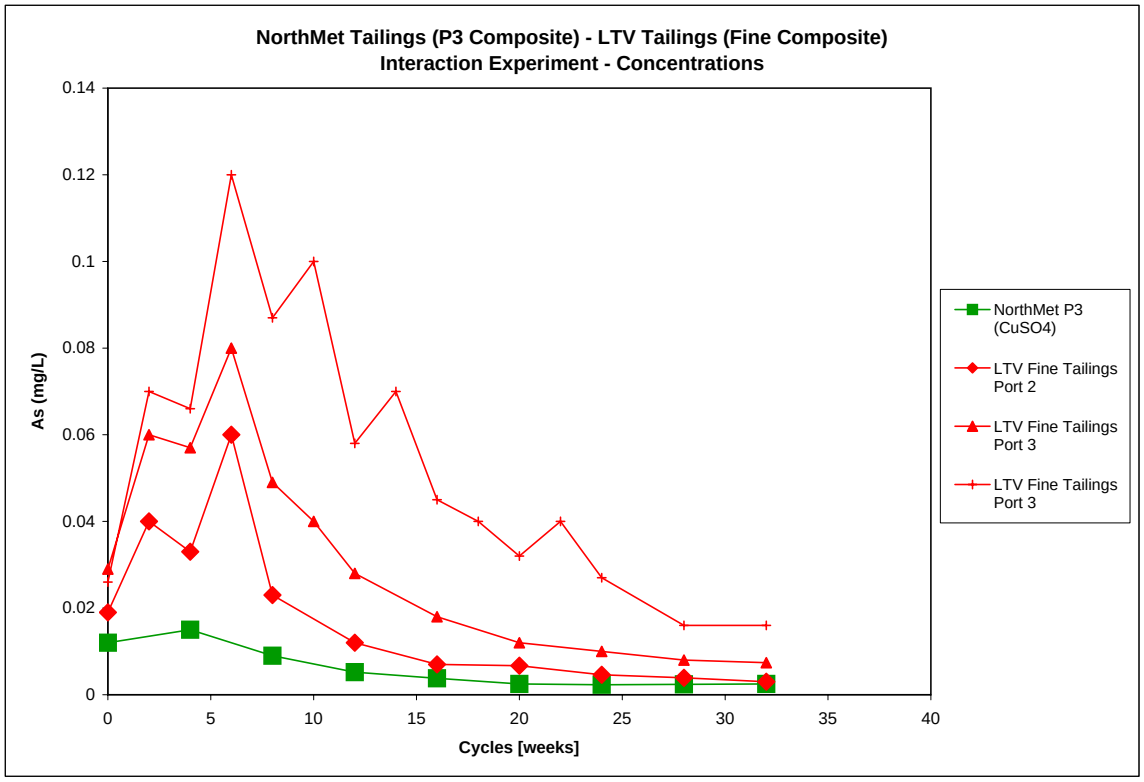
Graph C.3.93



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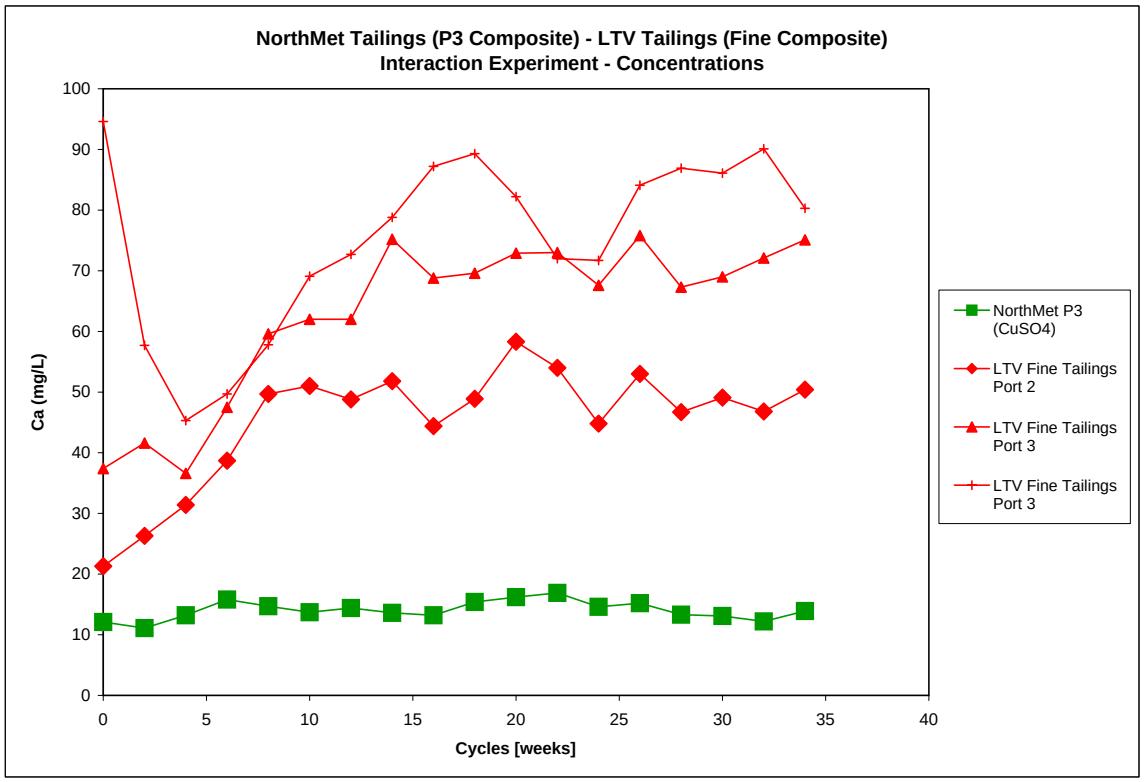
Graph C.3.94



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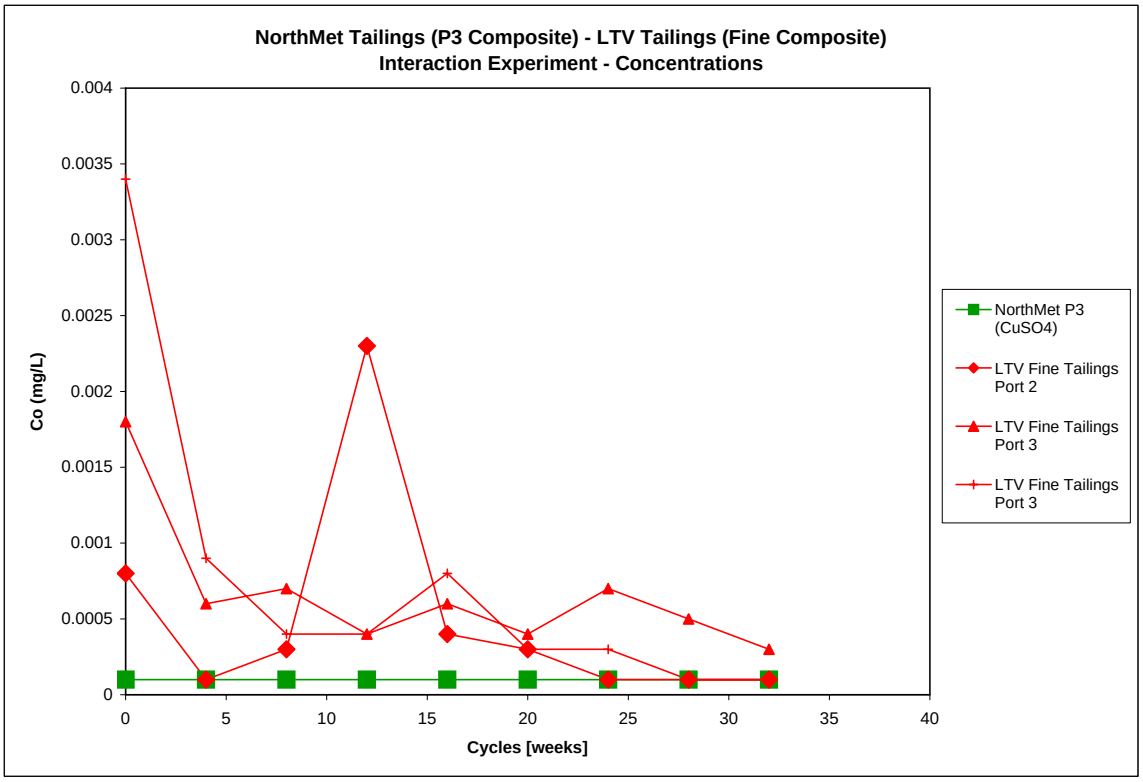
Graph C.3.95



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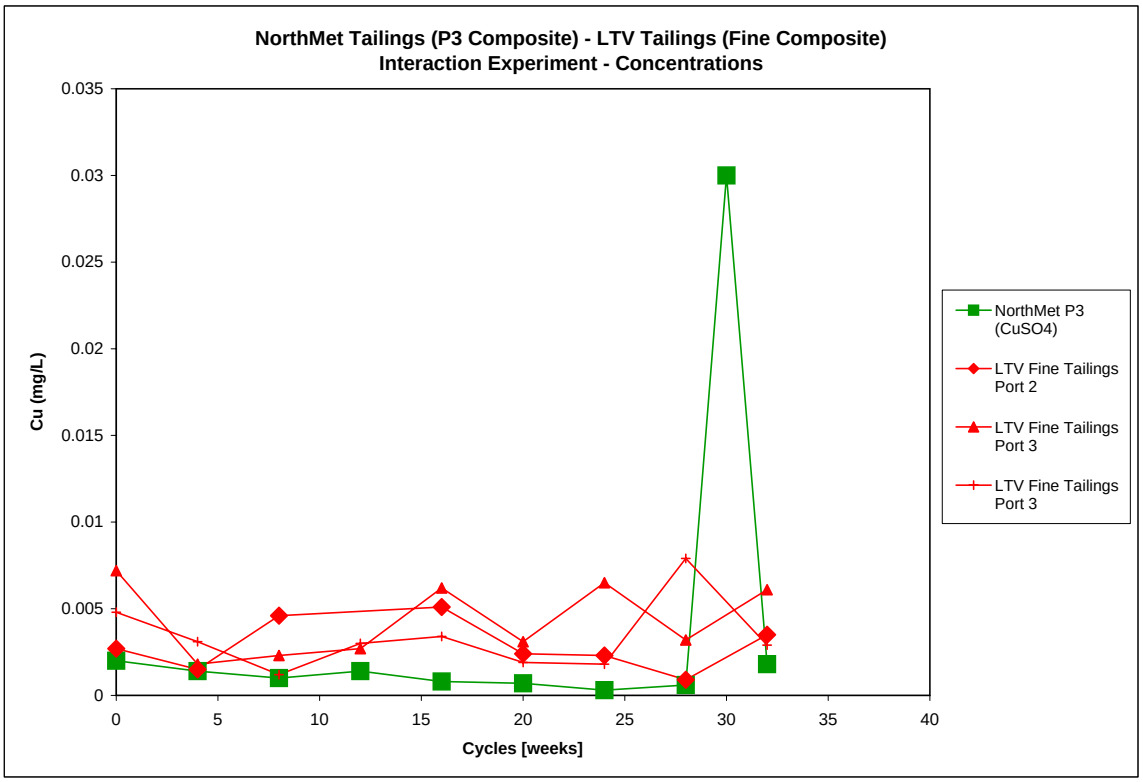
Graph C.3.96



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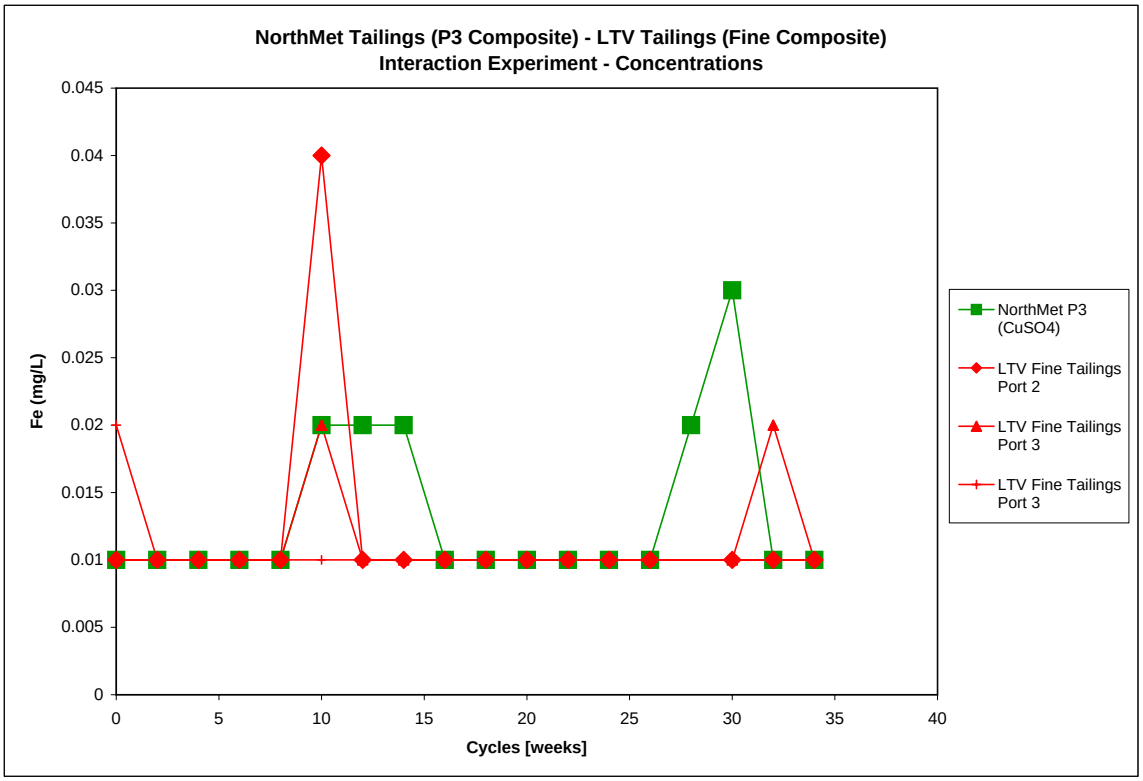
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.97



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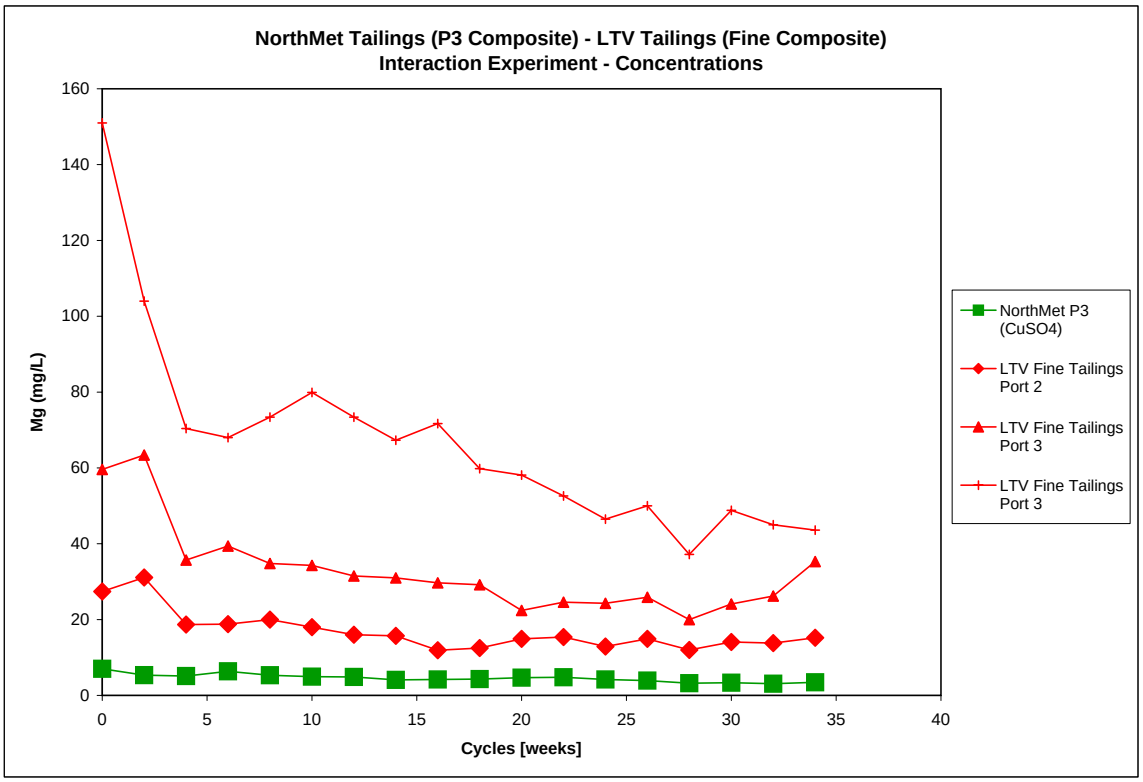
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.98



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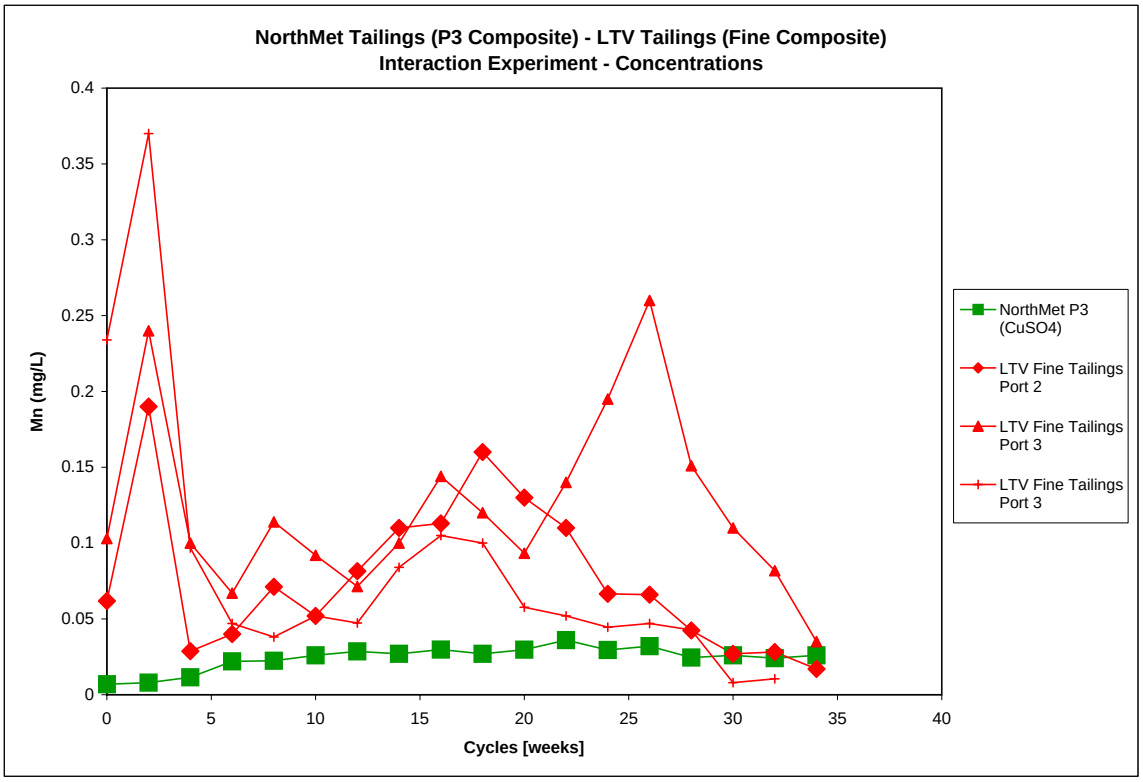
Graph C.3.99



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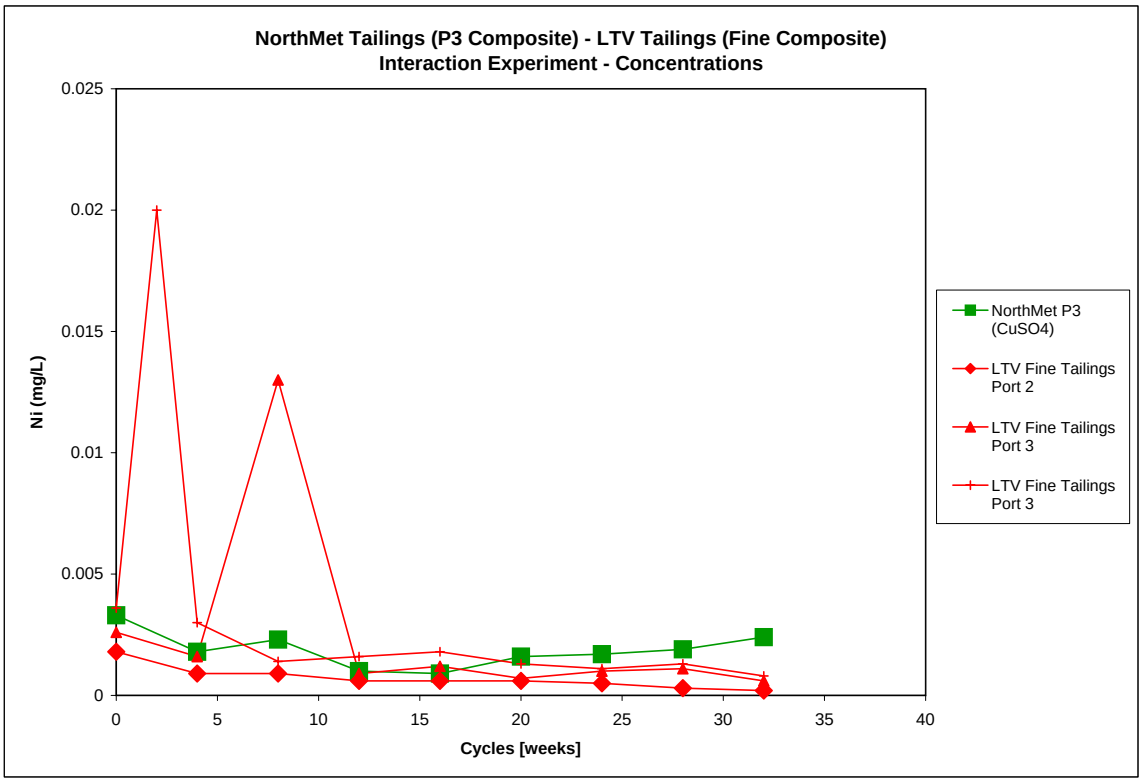
Graph C.3.100



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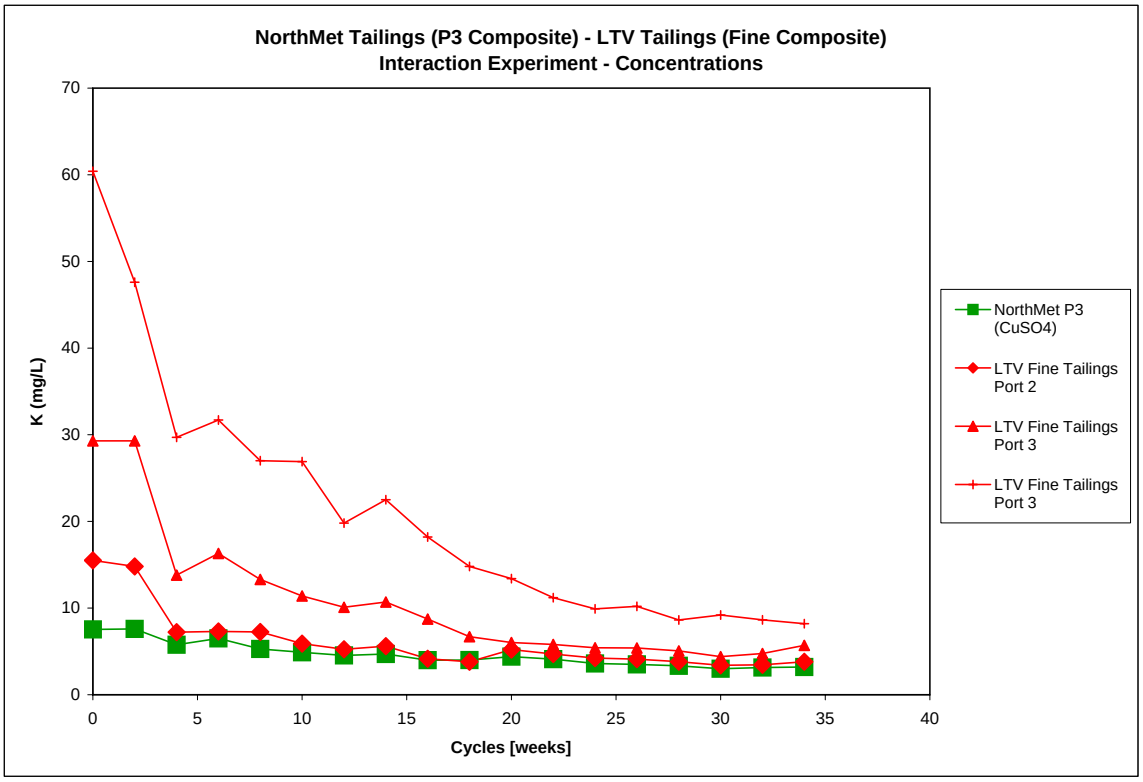
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.101



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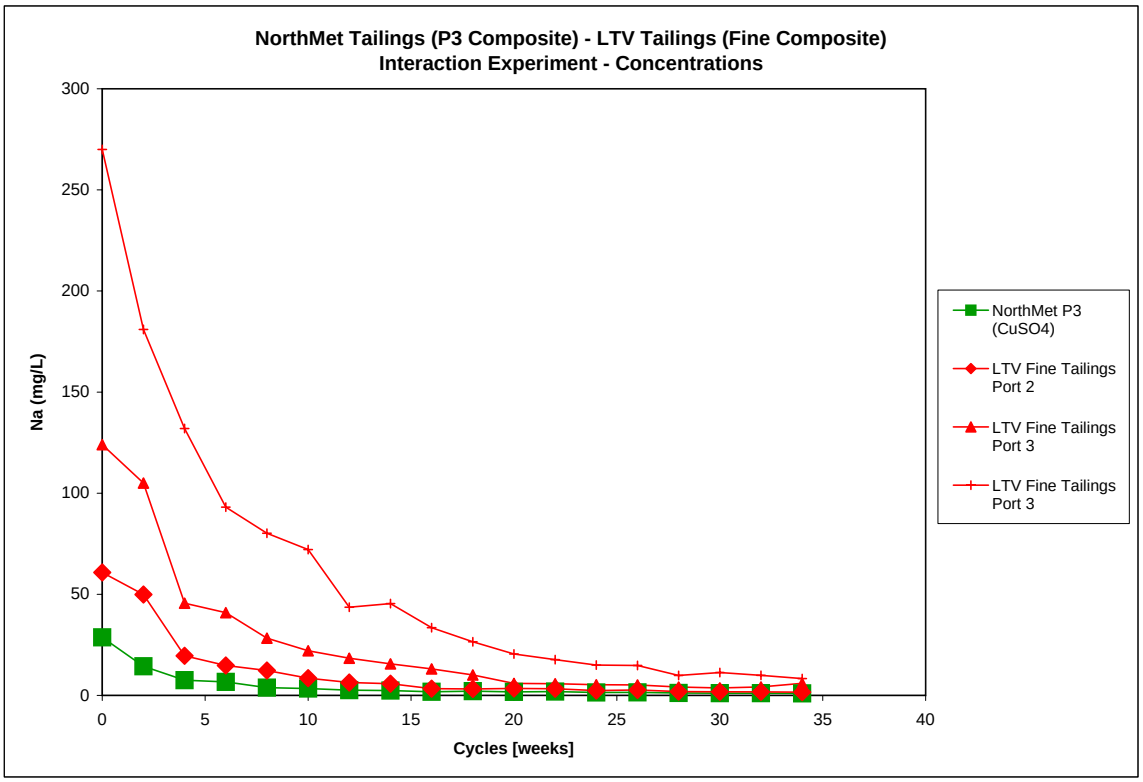
Appendix C.3
LTVSMC Contact Experiment

Graph C.3.102



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Appendix C.4

Low Level Hg Analyses

Sample ID	Test Type	Description	Hg (ng/L)
T1 Cycle 39	ASTM	P1S	4.2
T2 Cycle 39	ASTM	P1 Solid	2.8
T3 Cycle 39	ASTM	P2S	3.5
T4 Cycle 39	ASTM	P3S	2.5
T5 Cycle 17	ASTM	Parcel 2 P2S +100 mesh	3.8
T6 Cycle 17	ASTM	Parcel 2 P2S -100 +200 mesh	3.8
T7 Cycle 17	ASTM	Parcel 2 P2S -200 mesh	3.1
T8 Cycle 17	ASTM	Parcel 1-2 PISCS +100 mesh	<2.0
T9 Cycle 17	ASTM	Parcel 1-2 PISCS -100 +200 mesh	2.4
T10 Cylce 17	ASTM	Parcel 1-2 PISCS -200 mesh	3
T11 Cycle 17	ASTM	Parcel 3 P3S +100 mesh	2.6
T12 Cycle 17	ASTM	Parcel 3 P3S -100 +200 mesh	3.2
T13 Cycle 17	ASTM	Parcel 3 P3S -200 mesh	3
T22 Cycle 11	LTV contact	P1S Composite Port 1 Coarse Stream	3.2
T23 Cycle 11	LTV contact	Coarse Sand Port 2 P1S	2.9
T24 Cycle 11	LTV contact	Coarse Sand Port 3 P1S	3.7
T25 Cycle 11	LTV contact	Coarse Sand Port 4 P1S	3.6
T26 Cycle 11	LTV contact	P1S Composite Port 1 Fine Stream	2.8
T27 Cycle 11	LTV contact	Fine Sand/Slimes Port 2 (P1S)	3.2
T28 Cycle 11	LTV contact	Fine Sand/Slimes Port 3 (P1S)	3.2
T29 Cycle 11	LTV contact	Fine Sand/Slimes Port 4 (P1S)	3.2
T30 Cycle 11	LTV contact	P3S Composite Port 1 Coarse Stream	3.3
T31 Cycle 11	LTV contact	Coarse Sand Port 2 P3S	3.3
T32 Cycle 11	LTV contact	Coarse Sand Port 3 P3S	3.2
T33 Cycle 11	LTV contact	Coarse Sand Port 4 P3S	3.1
T34 Cycle 11	LTV contact	P3S Composite Port 1 Fine Stream	2.8
T35 Cycle 11	LTV contact	Fine Sand/Slimes Port 2 (P3S)	3.2
T36 Cycle 11	LTV contact	Fine Sand/Slimes Port 3 (P3S)	3.2
T37 Cycle 11	LTV contact	Fine Sand/Slimes Port 4 (P3S)	5.4
T38 Cycle 11	LTV contact	Coarse Sand Port 1 Control	3.5
T39 Cycle 11	LTV contact	Coarse Sand Port 2 Control	2.8
T40 Cycle 11	LTV contact	Coarse Sand Port 3 Control	2.9
T41 Cycle 11	LTV contact	Fine Sand/Slimes Port 1 Control	2.3
T42 Cycle 11	LTV contact	Fine Sand/Slimes Port 2 Control	2.8
T43 Cycle 11	LTV contact	Fine Sand/Slimes Port 3 Control	2.6

Appendix D
Modeling Results

Appendix D.1

HYDRUS2D Modeling Results

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Fax: 604.687.5532

Memorandum

To:	File	**Date:**	2007-05-31
cc:	John Chapman Steve Day	**From:**	Michel Noël
Subject:	PolyMet – Tailings impoundment: water transport modelling	**Project #:**	1UP005.001

This memo summarises the modelling that was performed to estimate the infiltration and moisture profiles at the proposed tailings impoundments. The purpose of this water transport modelling was to provide input for the water quantity and quality predictions.

1 Model

The modelling was carried out using Hydrus-2D version 2.008.

2 Material Properties

Two materials were modelled: coarse tailings and fine tailings. The hydraulic properties of the material are summarised in Table 1 below. The unsaturated properties were represented using the van Genuchten (1980) method with an air entry value restricted to -2 cm. This adjustment is to prevent the very steep slope of the derivative of the unsaturated hydraulic conductivity at saturation (Vogel and Cislerova 1988).

Table 1: Hydraulic properties

	θ_r	θ_{sat}	K_{sat} cm s^{-1}	α cm^{-1}	n	ξ	p
Coarse tailings	0.057	0.500	1.200E-03	0.124	2.280	1.0	0.500
Fine tailings	0.034	0.520	2.240E-05	0.016	1.370	1.0	0.500

The unsaturated properties listed above are based on the values reported by Carsel and Parrish (1988). The coarse tailings were assigned the values for loamy sand while the fine tailings was assigned the values for silt. The saturated hydraulic conductivity for both materials was adjusted to the values measured by the laboratory testing.

3 Modelling

3.1 Setup

The geometry of the model consisted of a 1 m wide column with variable heights, namely 15, 30, 45 and 60 m. The simulations were applied for a period of 1 or 100 years. The top boundary was assigned a no flux condition or an infiltration rate of 7.7 or 25 inches per year. The bottom boundary was assigned either a constant pressure or free drainage. The constant pressure was set at -0.1 m, which is equivalent to have a water table 0.1 below the base of the tailings. The free drainage condition corresponds to a unit vertical hydraulic gradient. Table 2 summarises the various simulations that were performed.

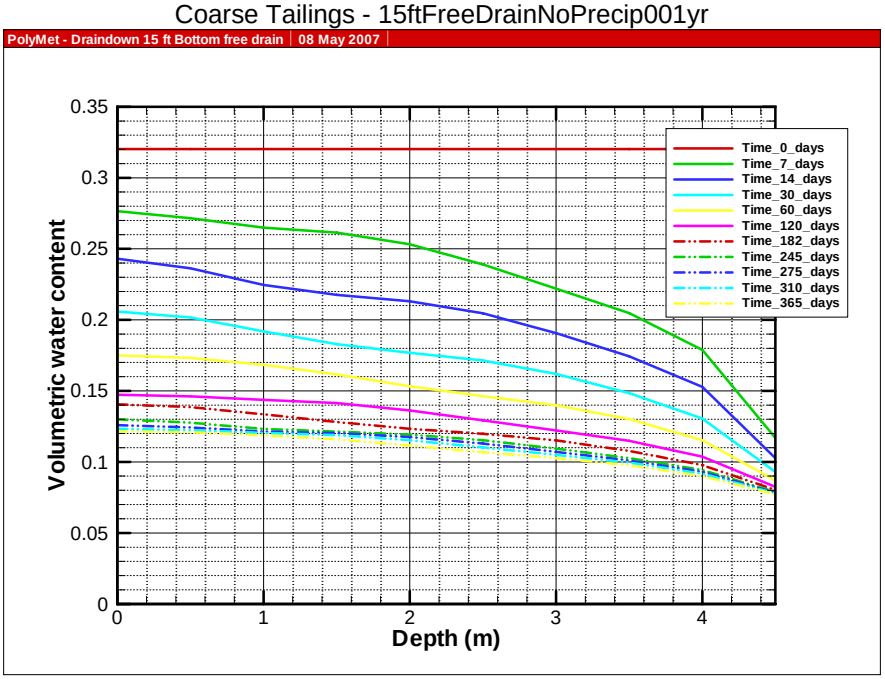
The water content profiles for the individual simulations listed in Table 2 are shown in Section 3.2.

Table 2: Summary of simulated cases

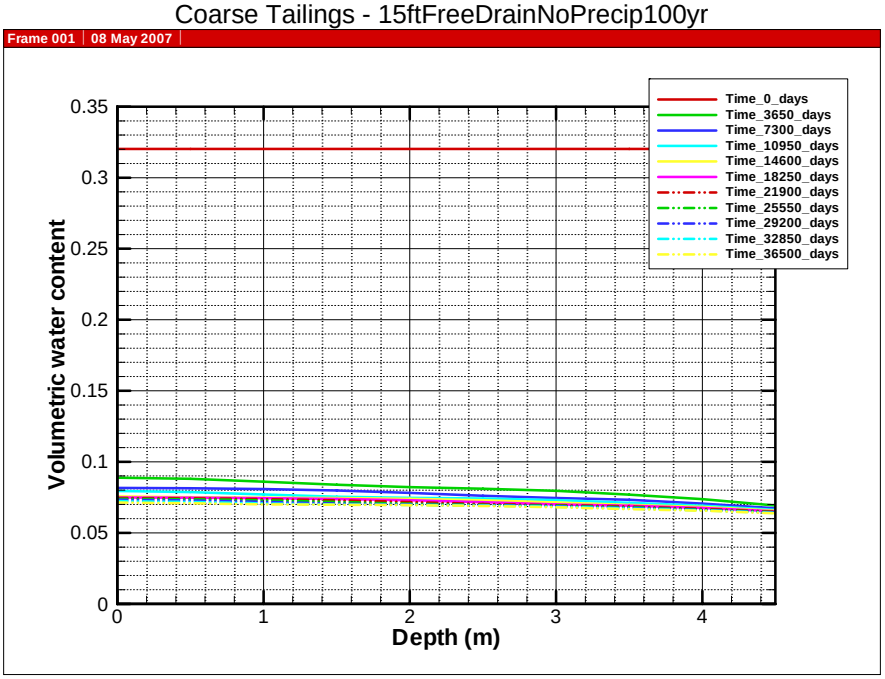
Sheet name	Tailings	Precip	Bottom boundary	Duration	Column height
		in/yr		year	ft
15ftFreeDrainNoPrecip001yr	coarse	0	Free drain	1	15
15ftFreeDrainNoPrecip100yr	coarse	0	Free drain	100	15
15ftFreeDrainPrecip001yr	coarse	7.7	Free drain	1	15
15ftFreeDrainPrecip100yr	coarse	7.7	Free drain	100	15
15ftHeadNoPrecip001yr	coarse	0	Head -0.1 m	1	15
15ftHeadNoPrecip100yr	coarse	0	Head -0.1 m	100	15
15ftHeadPrecip001yr	coarse	7.7	Head -0.1 m	1	15
15ftHeadPrecip100yr	coarse	7.7	Head -0.1 m	100	15
30ftFreeDrainNoPrecip001yr	coarse	0	Free drain	1	30
30ftFreeDrainPrecip001yr	coarse	7.7	Free drain	1	30
45ftFreeDrainNoPrecip001yr	coarse	0	Free drain	1	45
45ftFreeDrainNoPrecip100yr	coarse	0	Free drain	100	45
45ftFreeDrainPrecip001yr	coarse	7.7	Free drain	1	45
45ftFreeDrainPrecip100yr	coarse	7.7	Free drain	100	45
45ftHeadNoPrecip001yr	coarse	0	Head -0.1 m	1	45
45ftHeadNoPrecip100yr	coarse	0	Head -0.1 m	100	45
45ftHeadPrecip001yr	coarse	7.7	Head -0.1 m	1	45
45ftHeadPrecip100yr	coarse	7.7	Head -0.1 m	100	45
FINE15ftFreeDrainNoPrecip001yr	Fine	0	Free drain	1	15
FINE15ftFreeDrainNoPrecip100yrs	Fine	0	Free drain	100	15
FINE15ftFreeDrainPrecip001yr	Fine	7.7	Free drain	1	15
FINE15ftFreeDrainPrecip100yrs	Fine	7.7	Free drain	100	15
FINE30ftFreeDrainNoPrecip001yr	Fine	0	Free drain	1	30
FINE30ftFreeDrainPrecip001yr	Fine	7.7	Free drain	1	30
FINE45ftFreeDrainNoPrecip001yr	Fine	0	Free drain	1	45
FINE45ftFreeDrainNoPrecip100yrs	Fine	0	Free drain	100	45
FINE45ftFreeDrainPrecip001yr	Fine	7.7	Free drain	1	45
FINE45ftFreeDrainPrecip100yrs	Fine	7.7	Free drain	100	45
R25C15ftFreeDrain001yr	coarse	25	Free drain	1	15
R25F15ftFreeDrain001yr	Fine	25	Free drain	1	15
R25C30ftFreeDrain001yr	coarse	25	Free drain	1	30
R25F30ftFreeDrain001yr	Fine	25	Free drain	1	30
R25C60ftFreeDrain001yr	coarse	25	Free drain	1	60
R25F60ftFreeDrain001yr	Fine	25	Free drain	1	60

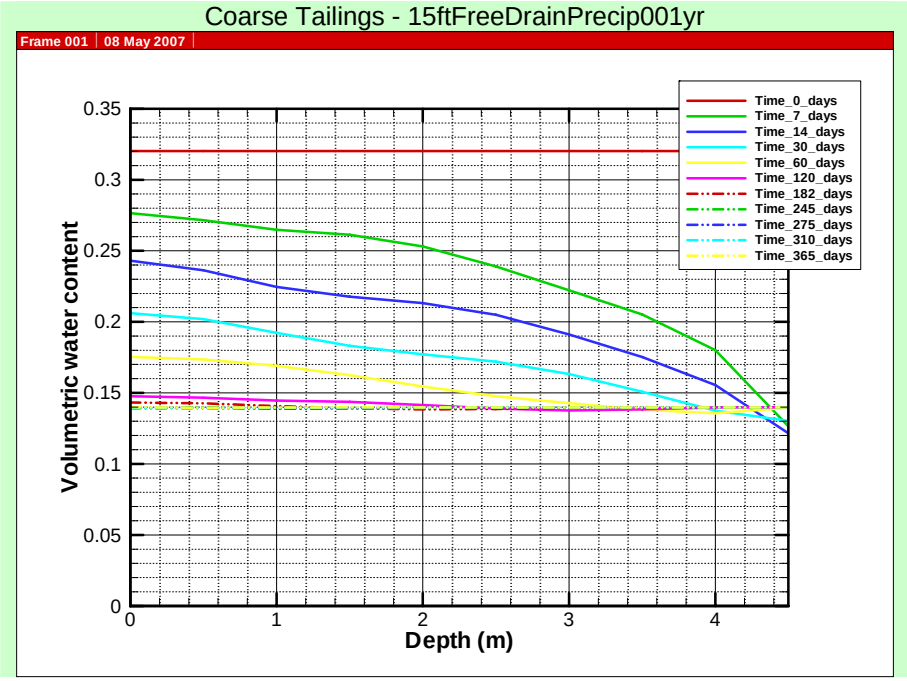
3.2 Results

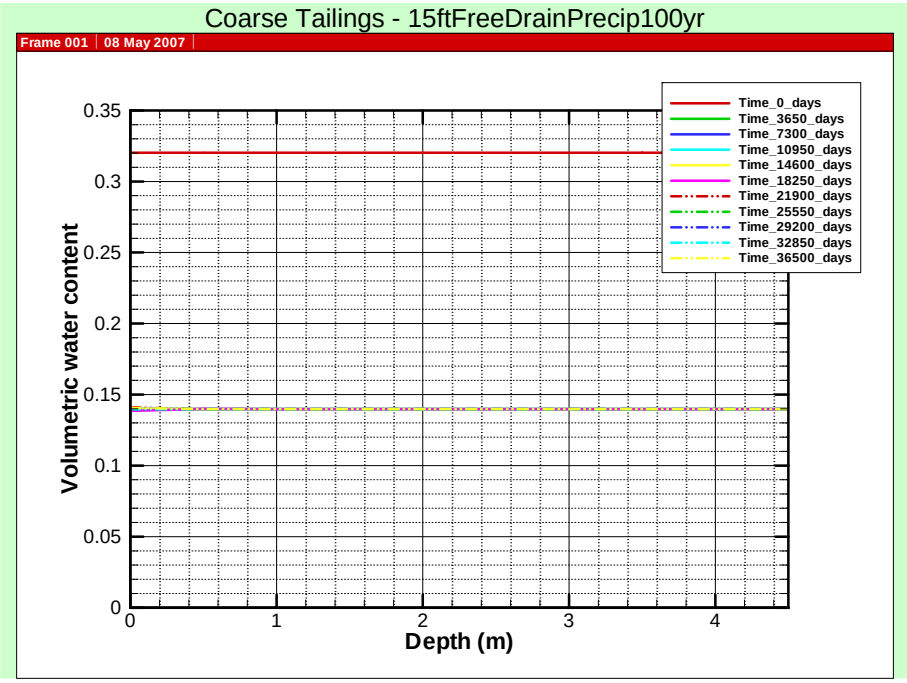
Coarse Tailings - 15ftFreeDrainNoPrecip001yr



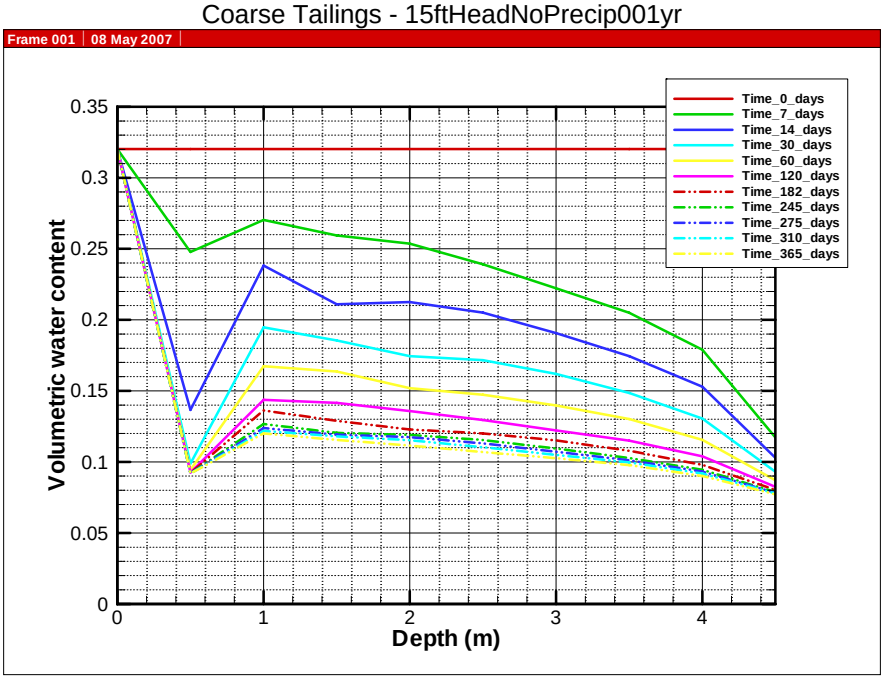
Coarse Tailings - 15ftFreeDrainNoPrecip100yr



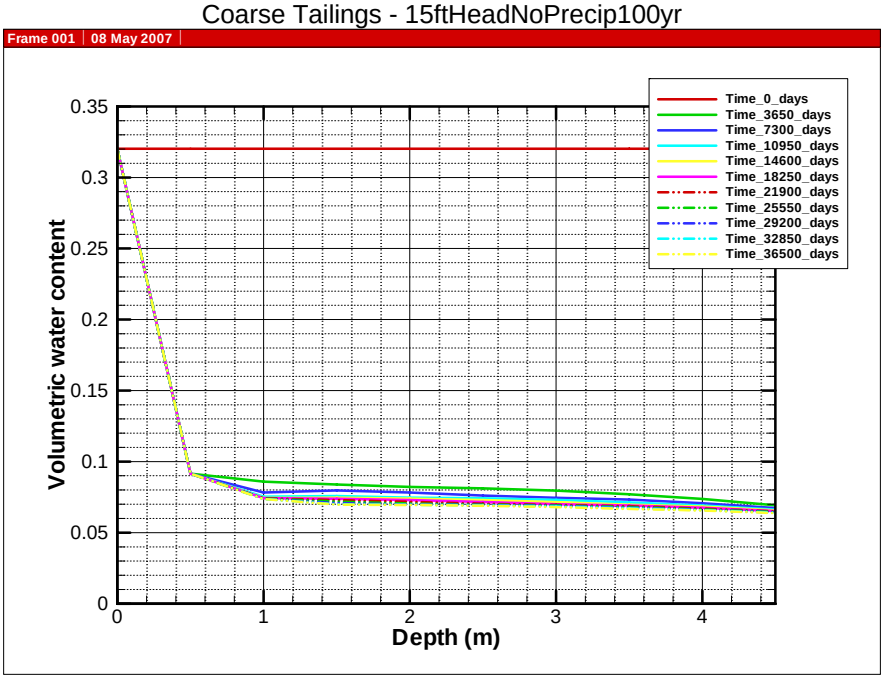

Coarse Tailings - 15ftFreeDrainPrecip001yr
Frame 001 08 May 2007
Volumetric water content
Depth (m)
0
0.05
0.1
0.15
0.2
0.25
0.3
0.35
0
1
2
3
4
Time_0_days
Time_7_days
Time_14_days
Time_30_days
Time_60_days
Time_120_days
Time_182_days
Time_245_days
Time_275_days
Time_310_days
Time_365_days

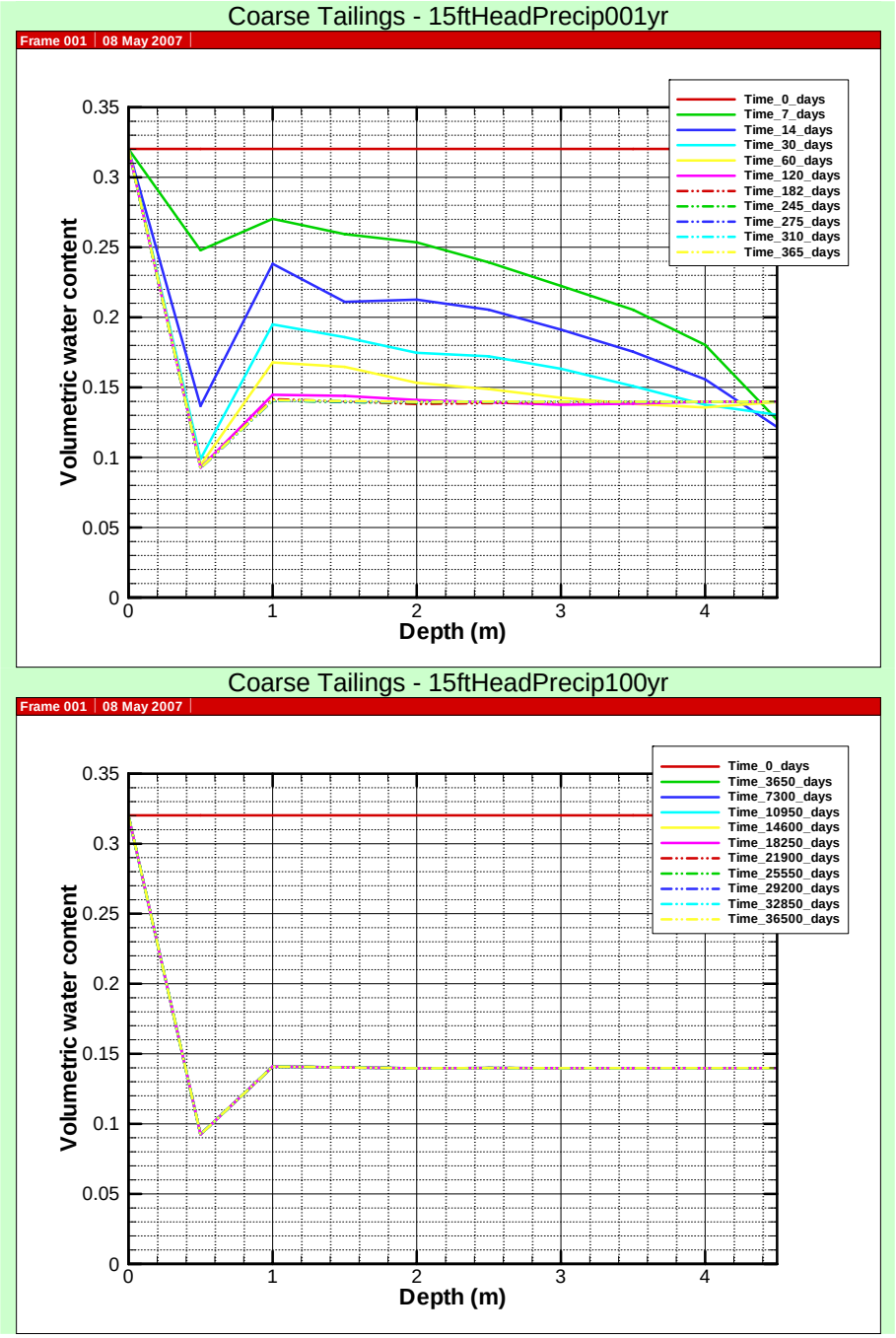

Coarse Tailings - 15ftFreeDrainPrecip100yr
Frame 001 08 May 2007
Volumetric water content
Depth (m)
0
0.05
0.1
0.15
0.2
0.25
0.3
0.35
0
1
2
3
4
Time_0_days
Time_3650_days
Time_7300_days
Time_10950_days
Time_14600_days
Time_18250_days
Time_21900_days
Time_25550_days
Time_29200_days
Time_32850_days
Time_36500_days

Coarse Tailings - 15ftHeadNoPrecip001yr

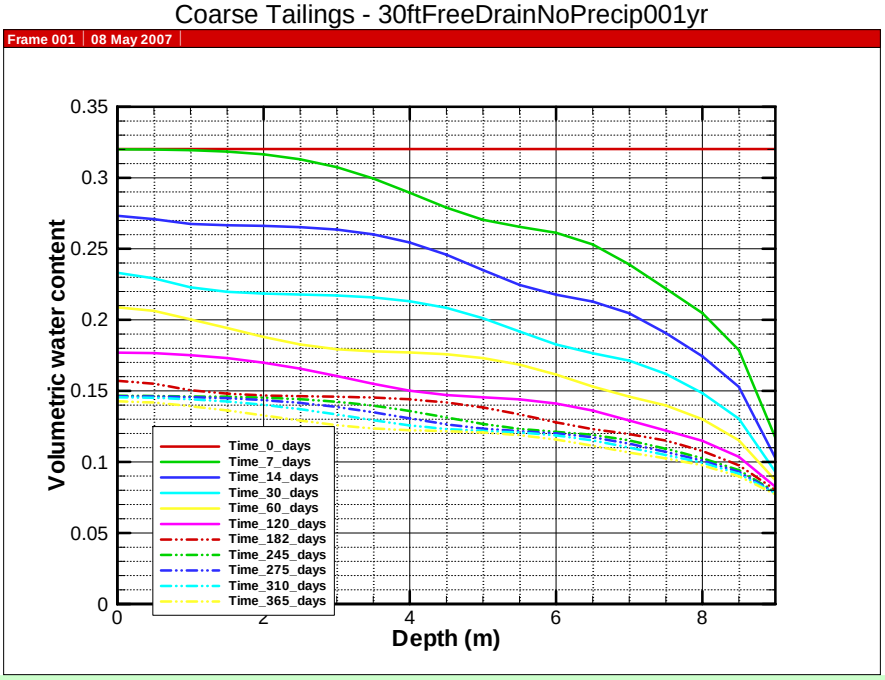


Coarse Tailings - 15ftHeadNoPrecip100yr

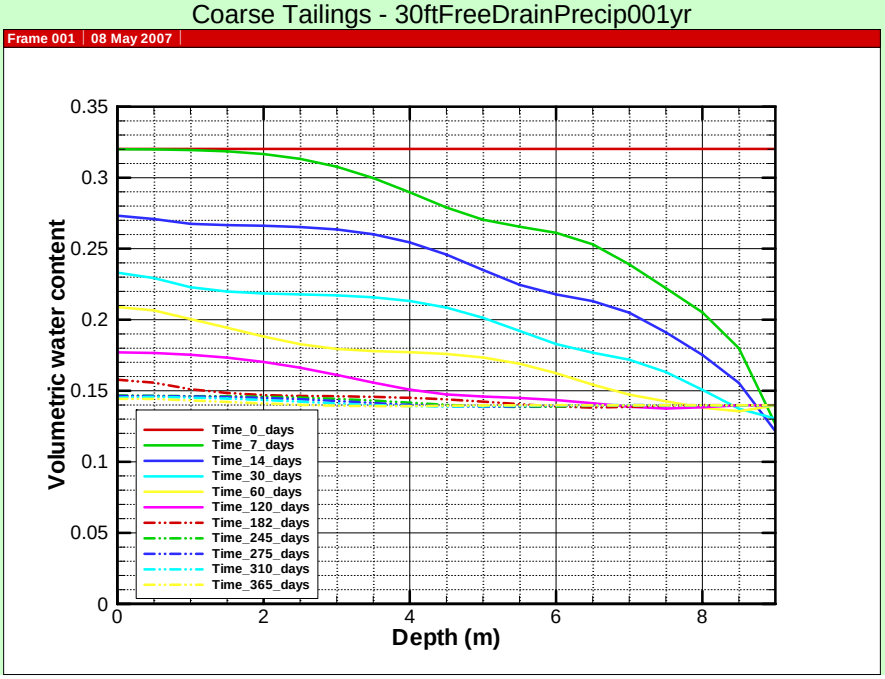



Coarse Tailings - 15ftHeadPrecip001yr
Frame 001 08 May 2007
Volumetric water content
Depth (m)
0
0.05
0.1
0.15
0.2
0.25
0.3
0.35
0
1
2
3
4
Time_0_days
Time_7_days
Time_14_days
Time_30_days
Time_60_days
Time_120_days
Time_182_days
Time_245_days
Time_275_days
Time_310_days
Time_365_days
Coarse Tailings - 15ftHeadPrecip100yr
Frame 001 08 May 2007
Volumetric water content
Depth (m)
Time_0_days
Time_3650_days
Time_7300_days
Time_10950_days
Time_14600_days
Time_18250_days
Time_21900_days
Time_25550_days
Time_29200_days
Time_32850_days
Time_36500_days

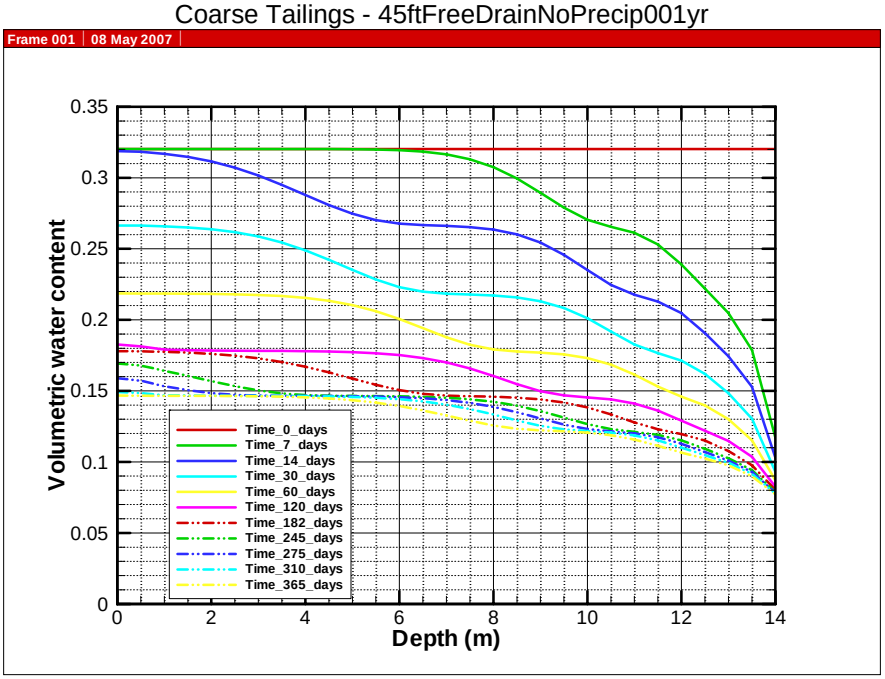
Coarse Tailings - 30ftFreeDrainNoPrecip001yr



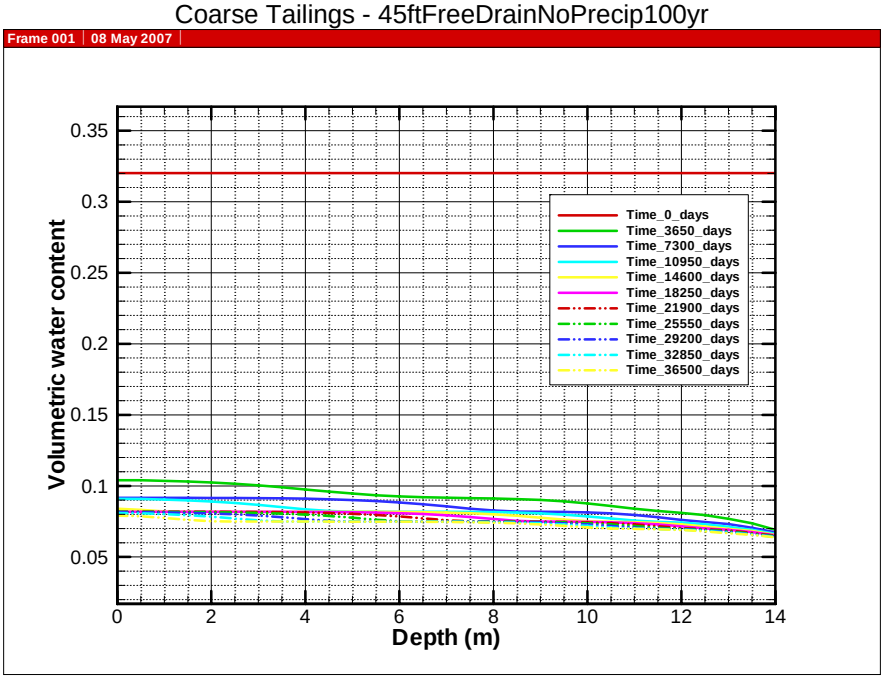
Coarse Tailings - 30ftFreeDrainPrecip001yr

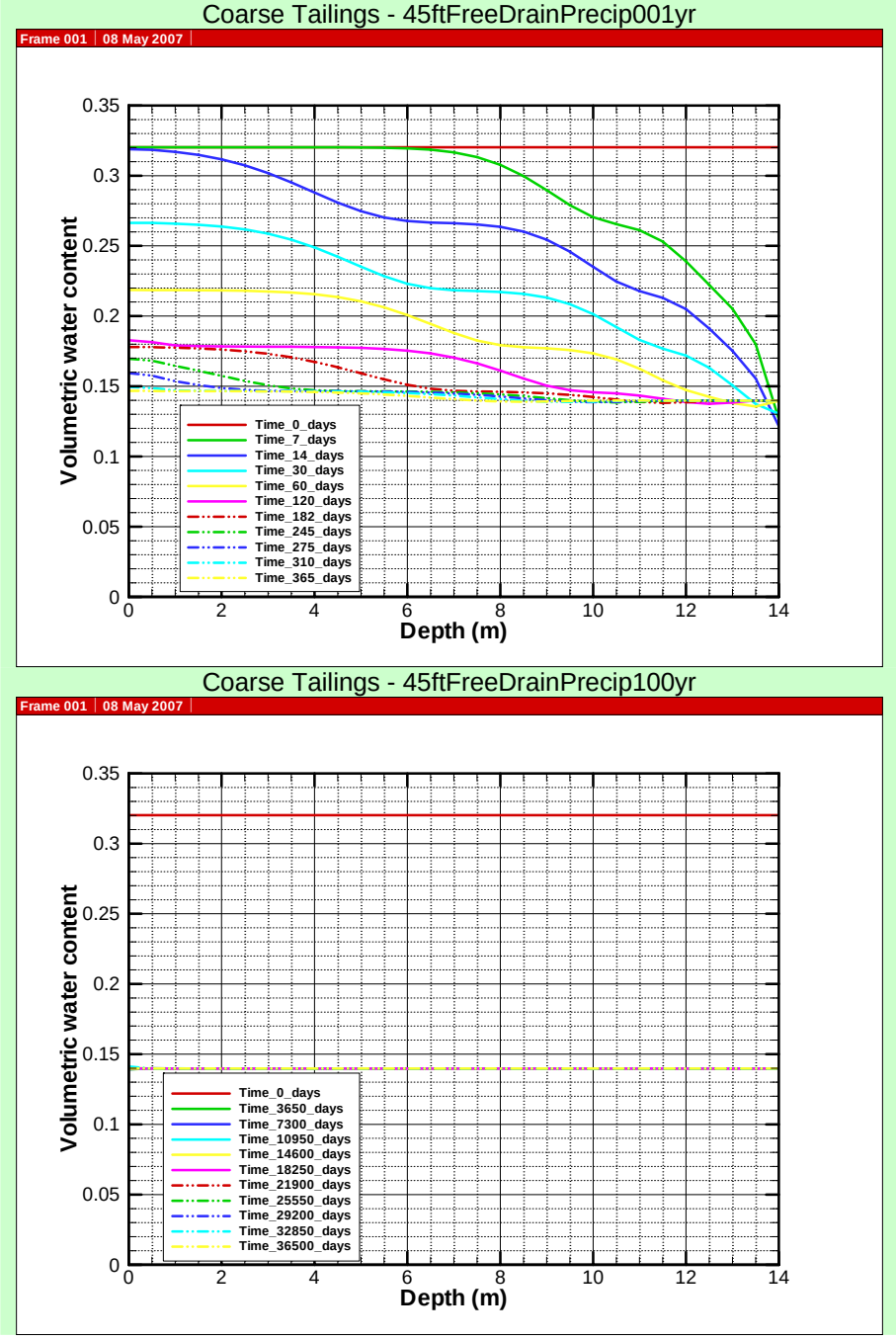


Coarse Tailings - 45ftFreeDrainNoPrecip001yr



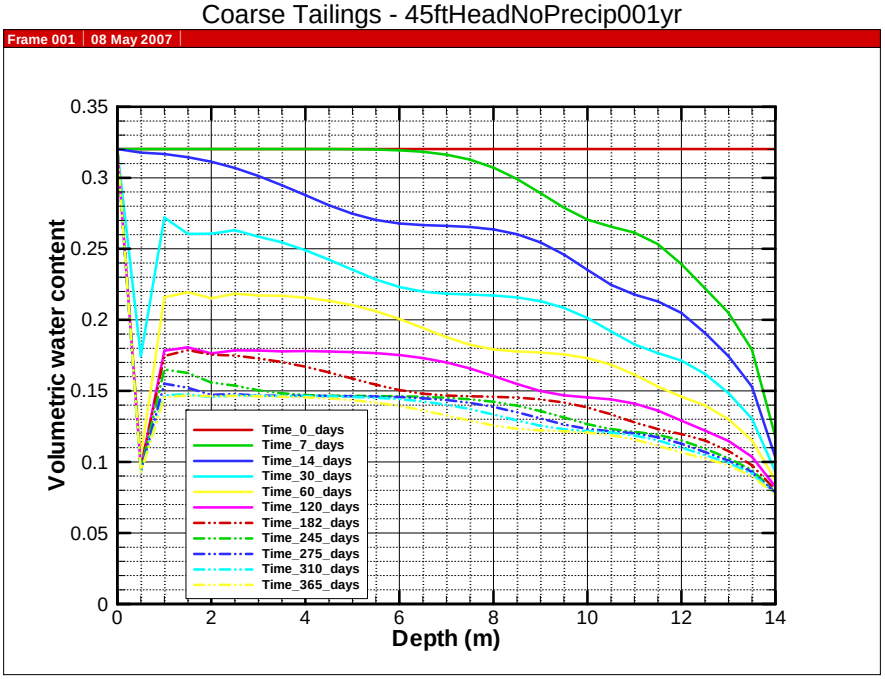
Coarse Tailings - 45ftFreeDrainNoPrecip100yr



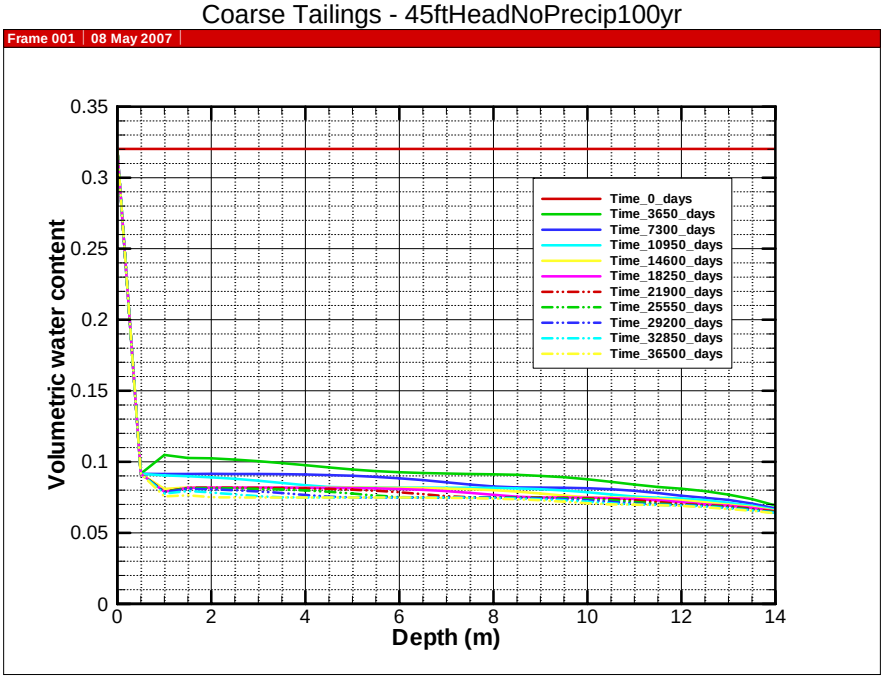

Coarse Tailings - 45ftFreeDrainPrecip001yr
Frame 001 08 May 2007
Volumetric water content
Depth (m)
0.35
0.3
0.25
0.2
0.15
0.1
0.05
0
0
2
4
6
8
10
12
14
Time_0_days
Time_7_days
Time_14_days
Time_30_days
Time_60_days
Time_120_days
Time_182_days
Time_245_days
Time_275_days
Time_310_days
Time_365_days

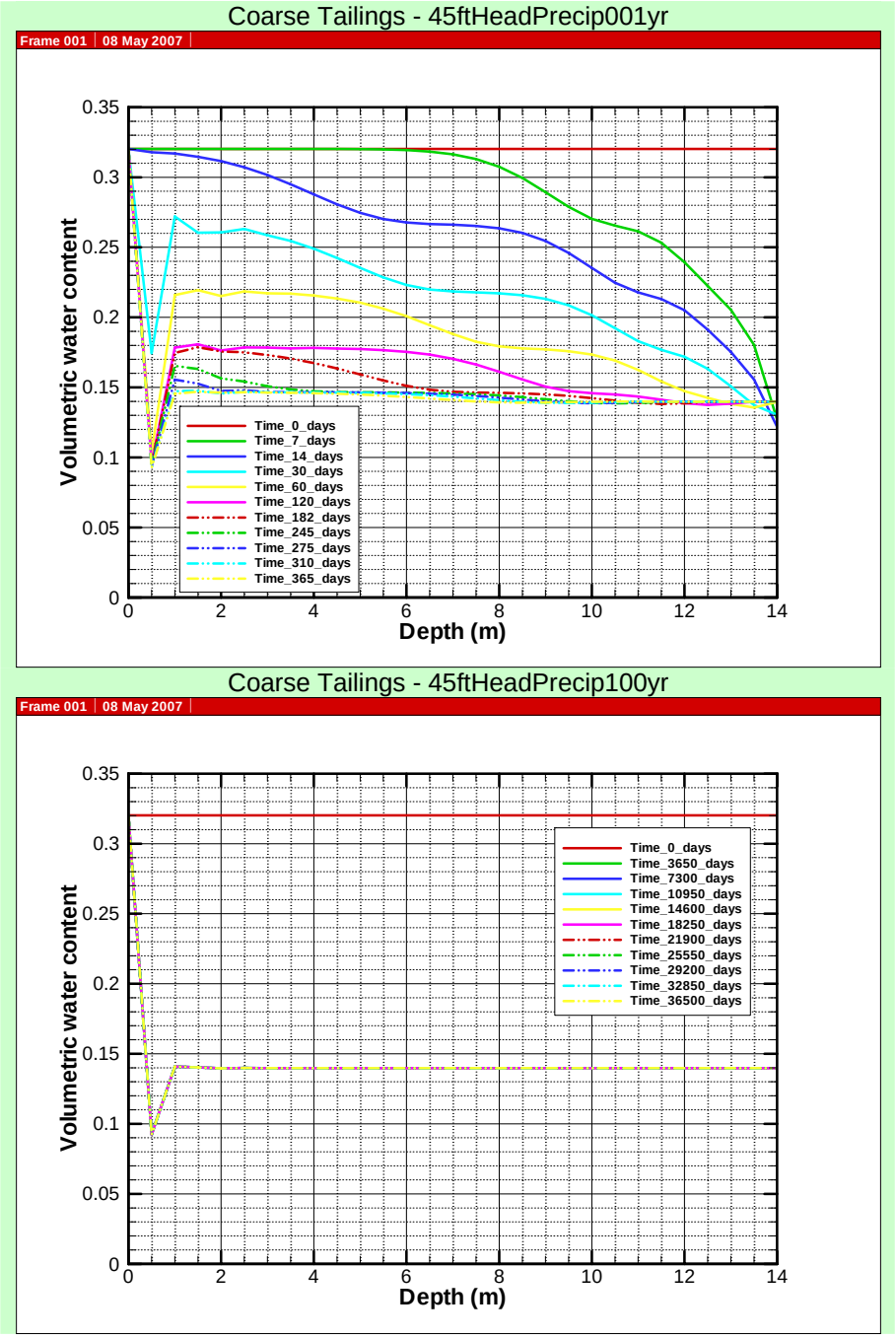
Coarse Tailings - 45ftFreeDrainPrecip100yr
Frame 001 08 May 2007
Volumetric water content
Depth (m)
0.35
0.3
0.25
0.2
0.15
0.1
0.05
0
0
2
4
6
8
10
12
14
Time_0_days
Time_3650_days
Time_7300_days
Time_10950_days
Time_14600_days
Time_18250_days
Time_21900_days
Time_25550_days
Time_29200_days
Time_32850_days
Time_36500_days

Coarse Tailings - 45ftHeadNoPrecip001yr

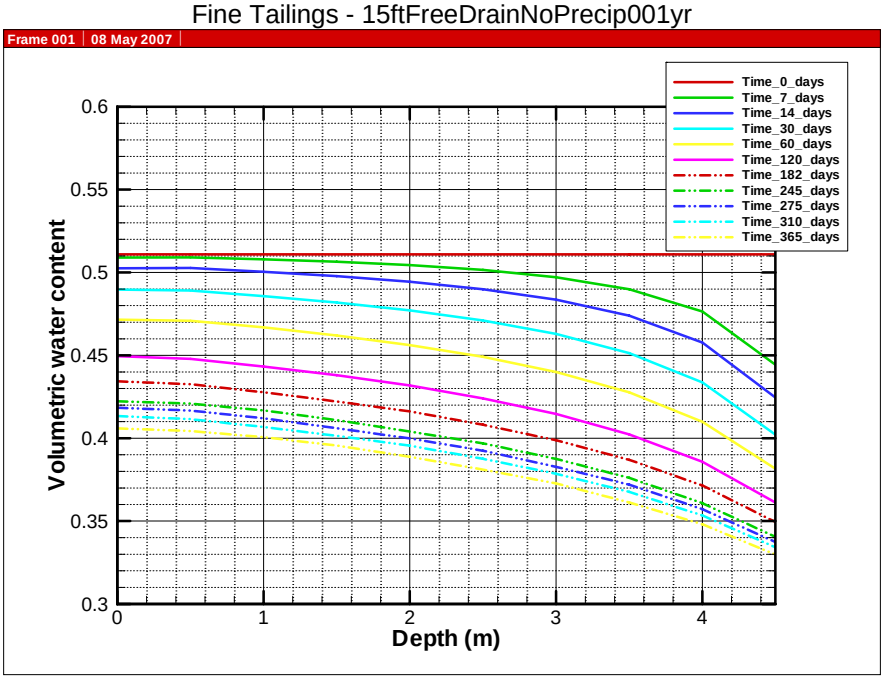


Coarse Tailings - 45ftHeadNoPrecip100yr

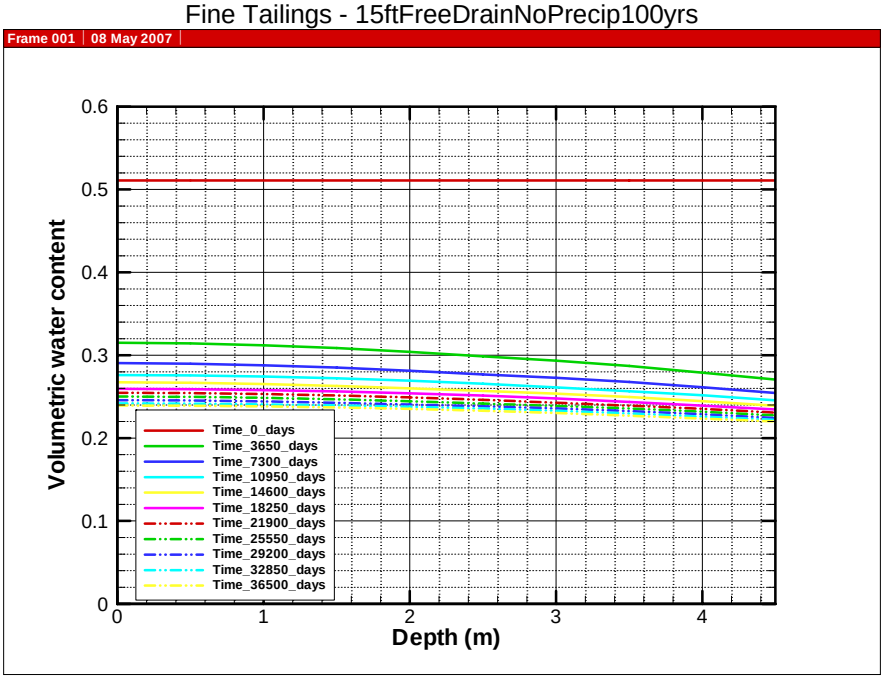



Coarse Tailings - 45ftHeadPrecip001yr
Frame 001 | 08 May 2007
Volumetric water content
Depth (m)
0.35
0.3
0.25
0.2
0.15
0.1
0.05
0
0
2
4
6
8
10
12
14
Time_0_days
Time_7_days
Time_14_days
Time_30_days
Time_60_days
Time_120_days
Time_182_days
Time_245_days
Time_275_days
Time_310_days
Time_365_days
Coarse Tailings - 45ftHeadPrecip100yr
Frame 001 | 08 May 2007
Volumetric water content
Depth (m)
Time_0_days
Time_3650_days
Time_7300_days
Time_10950_days
Time_14600_days
Time_18250_days
Time_21900_days
Time_25550_days
Time_29200_days
Time_32850_days
Time_36500_days

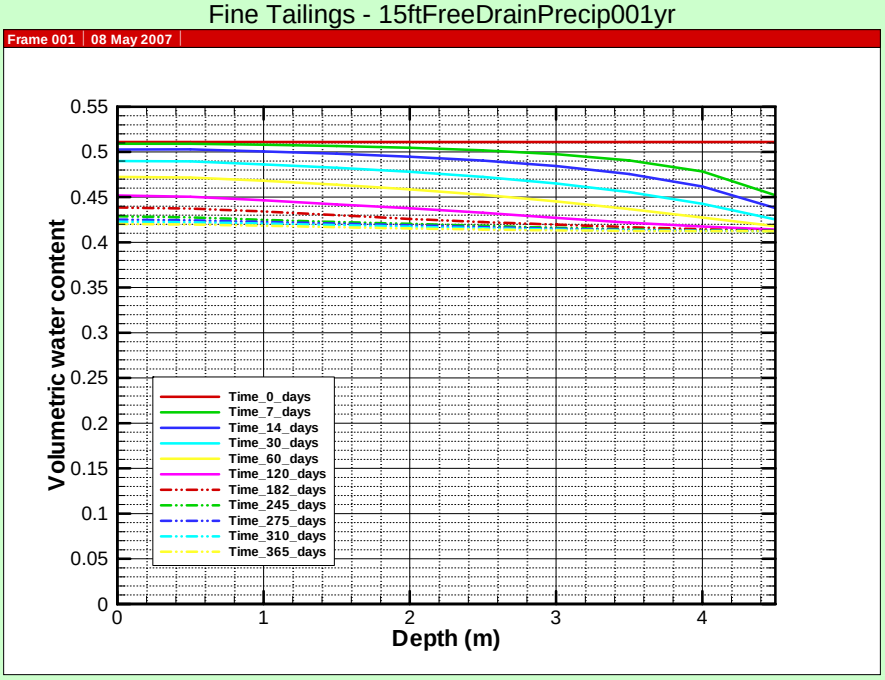
Fine Tailings - 15ftFreeDrainNoPrecip001yr



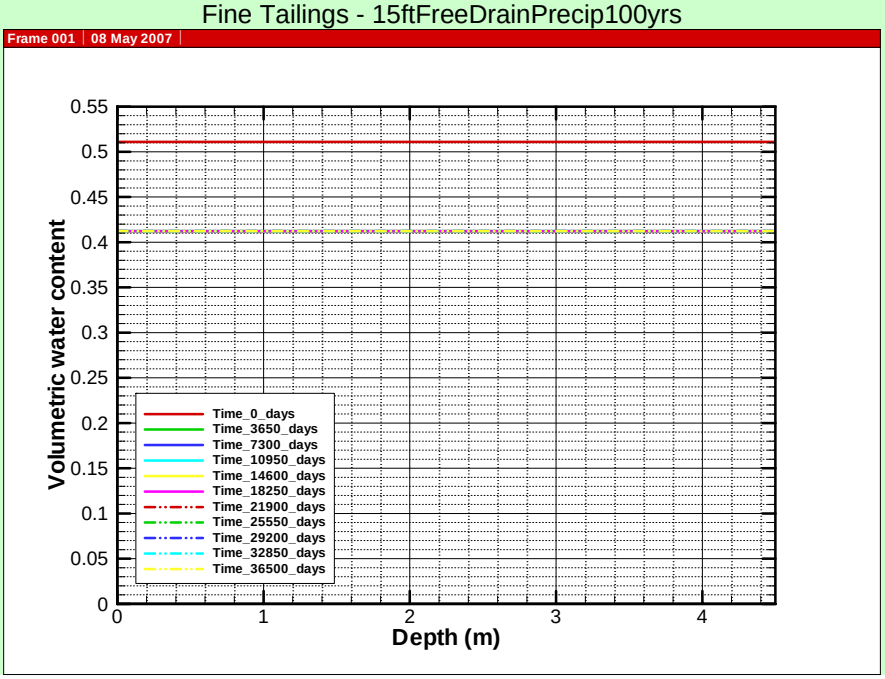
Fine Tailings - 15ftFreeDrainNoPrecip100yrs



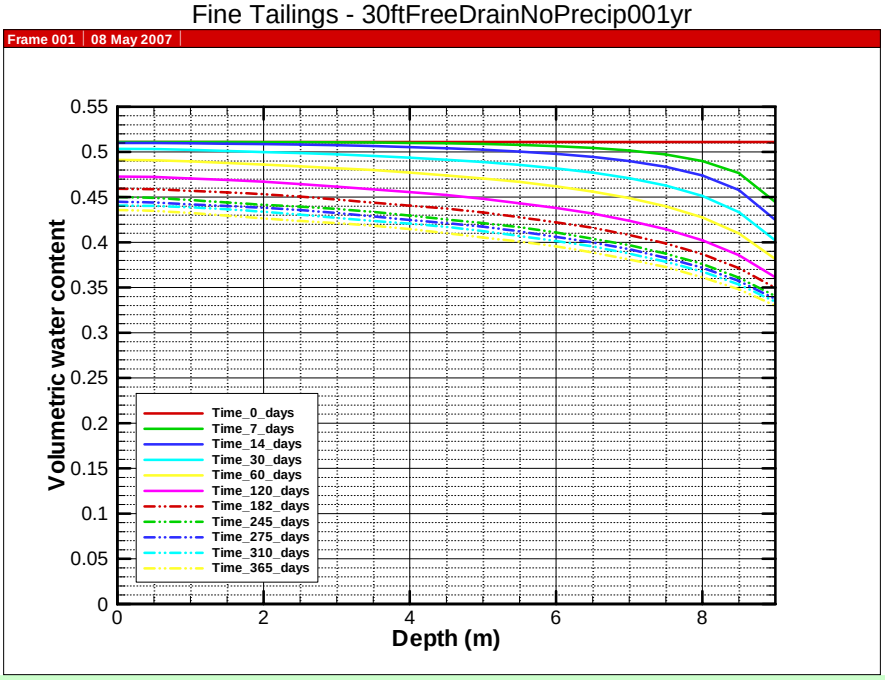
Fine Tailings - 15ftFreeDrainPrecip001yr



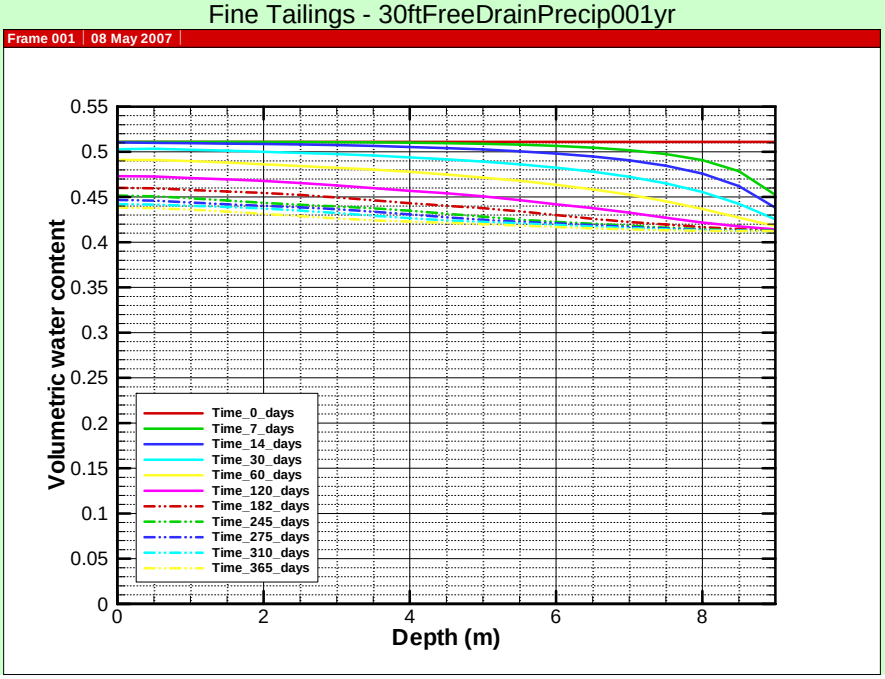
Fine Tailings - 15ftFreeDrainPrecip100yrs



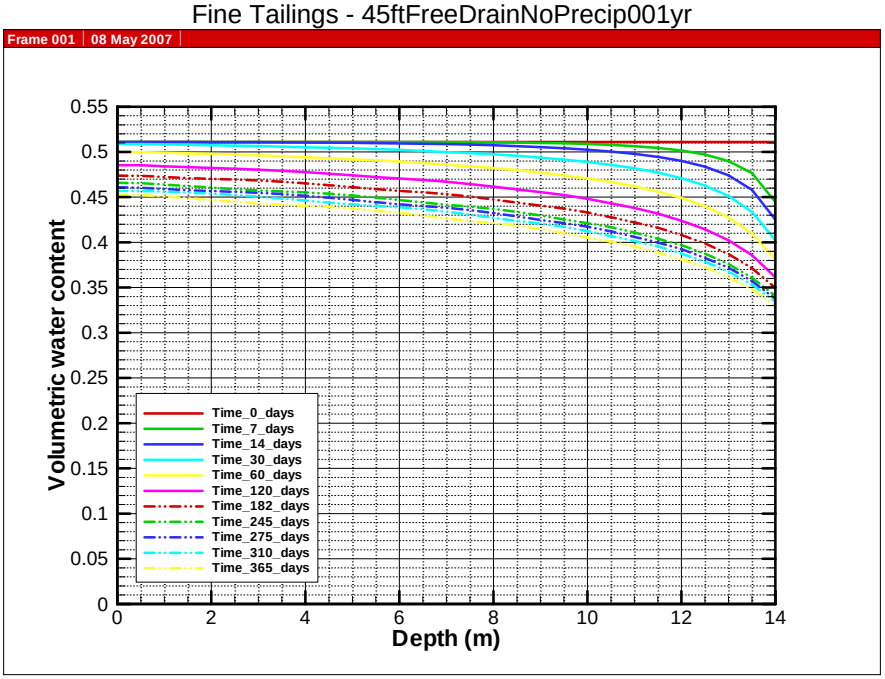
Fine Tailings - 30ftFreeDrainNoPrecip001yr



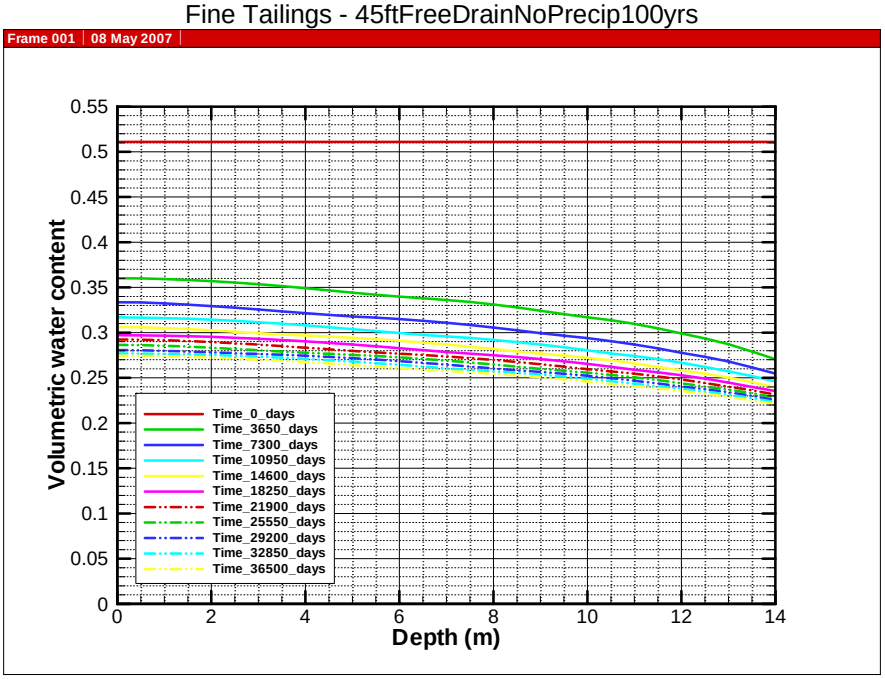
Fine Tailings - 30ftFreeDrainPrecip001yr



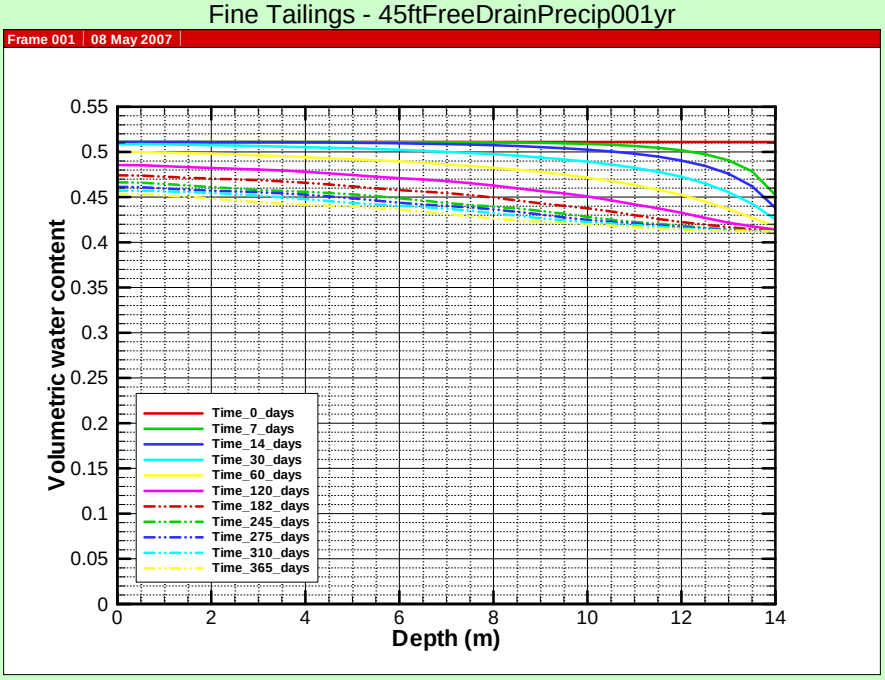
Fine Tailings - 45ftFreeDrainNoPrecip001yr



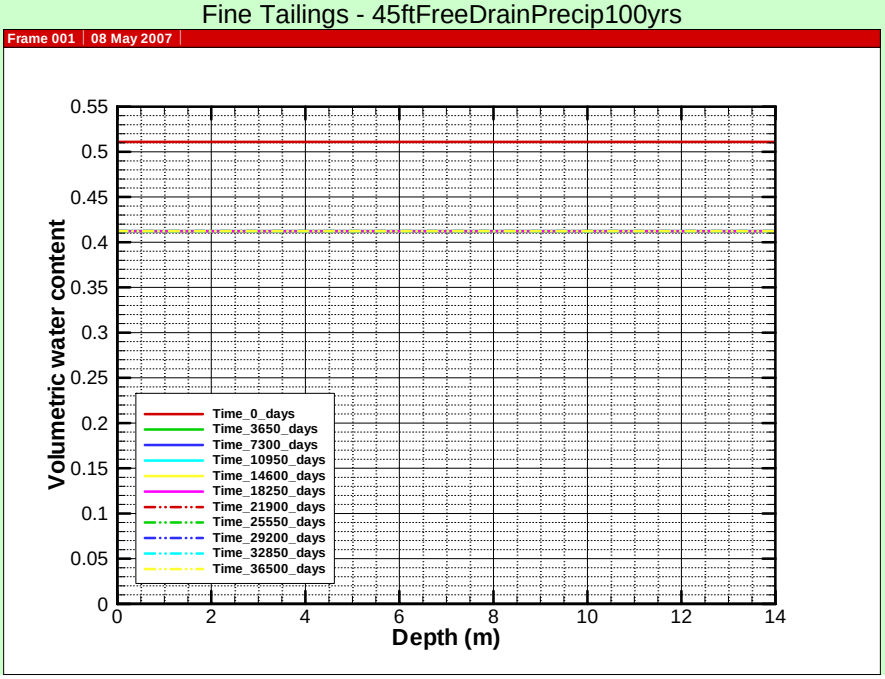
Fine Tailings - 45ftFreeDrainNoPrecip100yrs

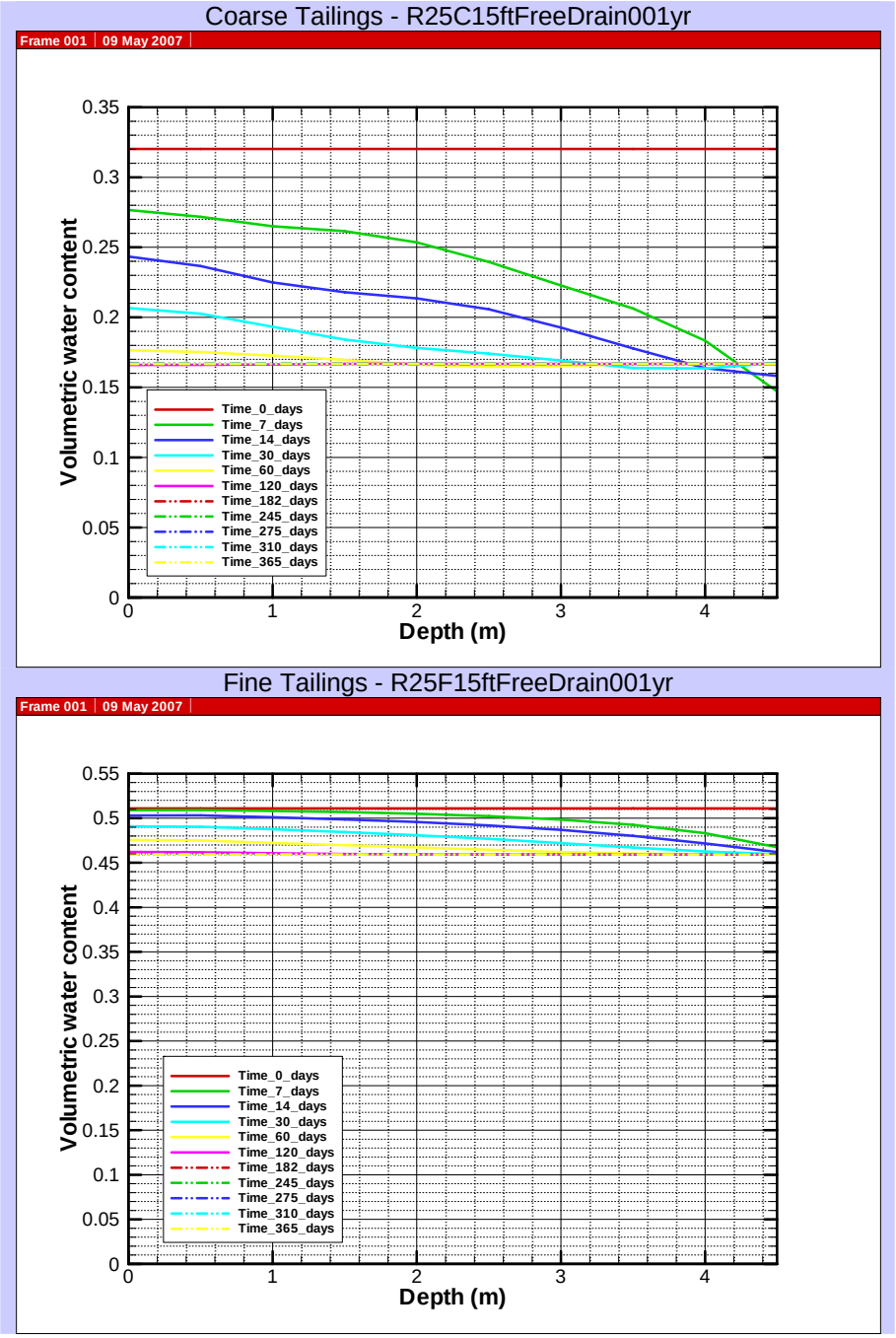


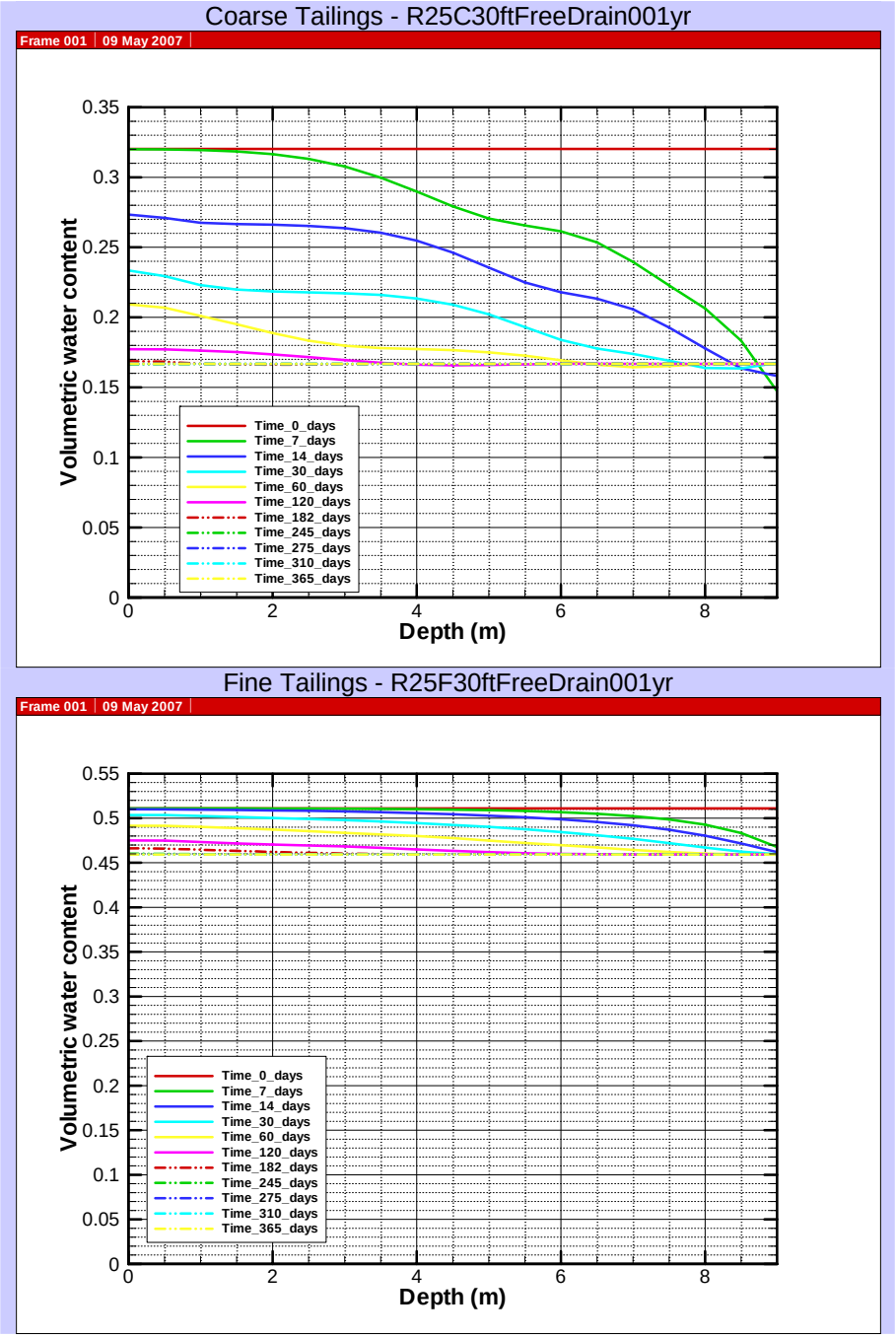
Fine Tailings - 45ftFreeDrainPrecip001yr

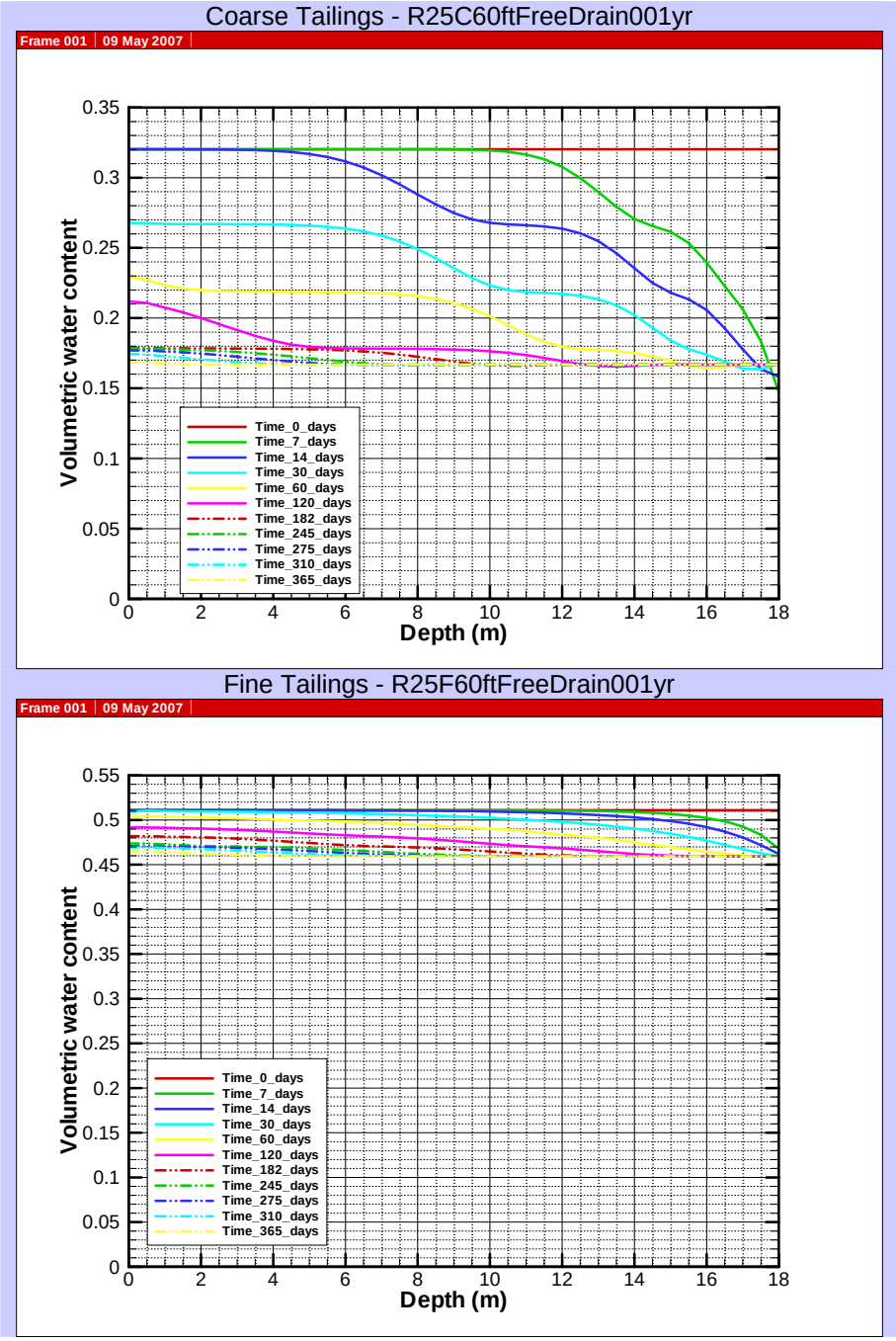


Fine Tailings - 45ftFreeDrainPrecip100yrs




Coarse Tailings - R25C15ftFreeDrain001yr
Frame 001 | 09 May 2007
Volumetric water content
Depth (m)
0.35
0.3
0.25
0.2
0.15
0.1
0.05
0
0
1
2
3
4
Time_0_days
Time_7_days
Time_14_days
Time_30_days
Time_60_days
Time_120_days
Time_182_days
Time_245_days
Time_275_days
Time_310_days
Time_365_days
Fine Tailings - R25F15ftFreeDrain001yr
Frame 001 | 09 May 2007
Volumetric water content
Depth (m)
0.55
0.5
0.45
0.4
0.35
0.3
0.25
0.2
0.15
0.1
0.05
0
0
1
2
3
4
Time_0_days
Time_7_days
Time_14_days
Time_30_days
Time_60_days
Time_120_days
Time_182_days
Time_245_days
Time_275_days
Time_310_days
Time_365_days


Coarse Tailings - R25C30ftFreeDrain001yr
Frame 001 | 09 May 2007
Volumetric water content
Depth (m)
0.35
0.3
0.25
0.2
0.15
0.1
0.05
0
0
2
4
6
8
Time_0_days
Time_7_days
Time_14_days
Time_30_days
Time_60_days
Time_120_days
Time_182_days
Time_245_days
Time_275_days
Time_310_days
Time_365_days
Fine Tailings - R25F30ftFreeDrain001yr
Frame 001 | 09 May 2007
Volumetric water content
Depth (m)
0.55
0.5
0.45
0.4
0.35
0.3
0.25
0.2
0.15
0.1
0.05
0
0
2
4
6
8
Time_0_days
Time_7_days
Time_14_days
Time_30_days
Time_60_days
Time_120_days
Time_182_days
Time_245_days
Time_275_days
Time_310_days
Time_365_days



4 Conclusions

The potential volumetric content of the coarse and fine tailings were assessed for a wide range of conditions, with column heights varied to simulate various lift heights during construction and development of the tailings embankment.

In general, the results show that the coarse tailings would be expected to drain down relatively rapidly (within a period of weeks to months) to a residual saturation of about 35 to 38 % (volumetric water content of about 0.17 to 0.19), with or without precipitation.

The fine tailings in contrast remain relatively saturated and do not drain down as rapidly as the coarse tailings. The results suggest that for the operational period, and for many years after active deposition ceases, the fine tailings will retain volumetric water content of about 0.45, i.e. a saturation of about 90 %. At about 100 years, the level of saturation is expected to decrease to about 0.4, i.e. a saturation of about 80 %.

5 References

Carsel, R.F., and Parrish, R. S. 1998. Developing joint probability distributions of soil water retention characteristics, Water Resour. Res, 24, 755-769.

Simunek, J., Sejna, M. and van Genuchten, M.Th. 1999. The HYDRUS-2D Software package for simulating the two-dimensional movement of water, heat, and multiple solutes in variably-saturated media, Version 2.0. U.S. Salinity Laboratory, Agricultural Research Services, U.S. Department of Agriculture, Riverside, California, USA, 227 pages.

Vogel, T., and M. Císlerová, On the reliability of unsaturated hydraulic conductivity calculated from the moisture retention curve, Transport in Porous Media, 3, 1-15, 1988.

Appendix D.2

Oxygen Transport Modelling Results

Coarse Tailings Oxygen Consumption Rates

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s)																			
			TIME (YEAR)																			
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
0.001	0.002	0.001	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08
0.002	0.007	0.002	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08
0.007	0.023	0.008	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08
0.019	0.062	0.016	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08
0.043	0.141	0.032	7.26E-08	7.27E-08	7.27E-08	7.27E-08	7.27E-08	7.27E-08	7.27E-08	7.27E-08	7.27E-08	7.27E-08	7.27E-08	7.27E-08	7.27E-08	7.27E-08	7.28E-08	7.28E-08	7.28E-08	7.28E-08	7.28E-08	7.28E-08
0.091	0.299	0.064	7.22E-08	7.23E-08	7.24E-08	7.24E-08	7.24E-08	7.24E-08	7.24E-08	7.24E-08	7.24E-08	7.24E-08	7.24E-08	7.24E-08	7.24E-08	7.24E-08	7.24E-08	7.24E-08	7.24E-08	7.24E-08	7.24E-08	7.24E-08
0.181	0.593	0.1152	7.14E-08	7.17E-08	7.18E-08	7.18E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08
0.298	0.979	0.1202	7.04E-08	7.08E-08	7.1E-08	7.11E-08	7.11E-08	7.11E-08	7.11E-08	7.11E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08
0.421	1.381	0.1252	6.93E-08	7E-08	7.02E-08	7.03E-08	7.03E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08
0.549	1.800	0.1302	6.82E-08	6.9E-08	6.93E-08	6.95E-08	6.95E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08
0.681	2.236	0.1352	6.71E-08	6.81E-08	6.85E-08	6.86E-08	6.87E-08	6.88E-08	6.88E-08	6.88E-08	6.88E-08	6.88E-08	6.88E-08	6.88E-08	6.88E-08	6.89E-08	6.89E-08	6.89E-08	6.89E-08	6.89E-08	6.89E-08	6.89E-08
0.819	2.687	0.1402	6.59E-08	6.71E-08	6.76E-08	6.78E-08	6.79E-08	6.79E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08
0.962	3.156	0.1452	6.47E-08	6.62E-08	6.67E-08	6.69E-08	6.7E-08	6.71E-08	6.71E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08
1.110	3.640	0.1502	6.35E-08	6.51E-08	6.57E-08	6.6E-08	6.61E-08	6.62E-08	6.63E-08	6.63E-08	6.63E-08	6.63E-08	6.63E-08	6.63E-08	6.64E-08	6.64E-08	6.64E-08	6.64E-08	6.64E-08	6.64E-08	6.64E-08	6.64E-08
1.262	4.141	0.1552	6.22E-08	6.41E-08	6.48E-08	6.51E-08	6.52E-08	6.53E-08	6.54E-08	6.54E-08	6.54E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08
1.420	4.658	0.1602	6.09E-08	6.3E-08	6.38E-08	6.41E-08	6.43E-08	6.44E-08	6.45E-08	6.45E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08
1.583	5.192	0.1652	5.96E-08	6.2E-08	6.28E-08	6.32E-08	6.34E-08	6.35E-08	6.36E-08	6.36E-08	6.36E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08
1.750	5.742	0.1702	5.83E-08	6.09E-08	6.18E-08	6.22E-08	6.24E-08	6.26E-08	6.26E-08	6.27E-08	6.27E-08	6.27E-08	6.28E-08	6.28E-08	6.28E-08	6.28E-08	6.28E-08	6.28E-08	6.28E-08	6.28E-08	6.28E-08	6.28E-08
1.923	6.309	0.1752	5.69E-08	5.98E-08	6.08E-08	6.12E-08	6.15E-08	6.16E-08	6.17E-08	6.18E-08	6.18E-08	6.18E-08	6.18E-08	6.18E-08	6.19E-08	6.19E-08	6.19E-08	6.19E-08	6.19E-08	6.19E-08	6.19E-08	6.19E-08
2.101	6.892	0.1802	5.55E-08	5.86E-08	5.97E-08	6.02E-08	6.05E-08	6.06E-08	6.07E-08	6.08E-08	6.08E-08	6.09E-08	6.09E-08	6.09E-08	6.09E-08	6.09E-08	6.09E-08	6.09E-08	6.09E-08	6.09E-08	6.09E-08	6.09E-08
2.283	7.491	0.1852	5.41E-08	5.75E-08	5.87E-08	5.92E-08	5.95E-08	5.97E-08	5.98E-08	5.98E-08	5.99E-08	5.99E-08	5.99E-08	5.99E-08	6E-08	6E-08	6E-08	6E-08	6E-08	6E-08	6E-08	6E-08
2.471	8.107	0.1902	5.27E-08	5.63E-08	5.76E-08	5.82E-08	5.85E-08	5.87E-08	5.88E-08	5.89E-08	5.89E-08	5.89E-08	5.9E-08	5.9E-08	5.9E-08	5.9E-08	5.9E-08	5.9E-08	5.9E-08	5.9E-08	5.9E-08	5.9E-08
2.664	8.740	0.1952	5.13E-08	5.51E-08	5.65E-08	5.71E-08	5.75E-08	5.77E-08	5.78E-08	5.79E-08	5.79E-08	5.8E-08	5.8E-08	5.8E-08	5.8E-08	5.8E-08	5.8E-08	5.8E-08	5.8E-08	5.8E-08	5.8E-08	5.8E-08
2.862	9.388	0.2002	4.98E-08	5.4E-08	5.54E-08	5.61E-08	5.65E-08	5.67E-08	5.68E-08	5.69E-08	5.7E-08	5.7E-08	5.7E-08	5.7E-08	5.7E-08	5.7E-08	5.71E-08	5.71E-08	5.71E-08	5.71E-08	5.71E-08	5.71E-08
3.064	10.053	0.2052	4.84E-08	5.28E-08	5.43E-08	5.5E-08	5.54E-08	5.57E-08	5.58E-08	5.59E-08	5.6E-08	5.6E-08	5.6E-08	5.6E-08	5.6E-08	5.61E-08	5.61E-08	5.61E-08	5.61E-08	5.61E-08	5.61E-08	5.61E-08
3.272	10.735	0.2102	4.69E-08	5.16E-08	5.32E-08	5.4E-08	5.44E-08	5.46E-08	5.48E-08	5.49E-08	5.5E-08	5.5E-08	5.5E-08	5.5E-08	5.51E-08	5.51E-08	5.51E-08	5.51E-08	5.51E-08	5.51E-08	5.51E-08	5.51E-08
3.485	11.432	0.2152	4.54E-08	5.03E-08	5.21E-08	5.29E-08	5.34E-08	5.36E-08	5.38E-08	5.39E-08	5.39E-08	5.4E-08	5.4E-08	5.4E-08	5.41E-08	5.41E-08	5.41E-08	5.41E-08	5.41E-08	5.41E-08	5.41E-08	5.41E-08
3.702	12.147	0.2202	4.39E-08	4.91E-08	5.1E-08	5.18E-08	5.23E-08	5.26E-08	5.27E-08	5.29E-08	5.29E-08	5.3E-08	5.3E-08	5.3E-08	5.3E-08	5.31E-08	5.31E-08	5.31E-08	5.31E-08	5.31E-08	5.31E-08	5.31E-08
3.925	12.877	0.2252	4.25E-08	4.79E-08	4.98E-08	5.08E-08	5.13E-08	5.15E-08	5.17E-08	5.18E-08	5.19E-08	5.2E-08	5.2E-08	5.2E-08	5.2E-08	5.2E-08	5.21E-08	5.21E-08	5.21E-08	5.21E-08	5.21E-08	5.21E-08
4.153	13.624	0.2302	4.1E-08	4.67E-08	4.87E-08	4.97E-08	5.02E-08	5.05E-08	5.07E-08	5.08E-08	5.09E-08	5.09E-08	5.1E-08	5.1E-08	5.1E-08	5.1E-08	5.1E-08	5.1E-08	5.11E-08	5.11E-08	5.11E-08	5.11E-08
4.385	14.388	0.2352	3.95E-08	4.54E-08	4.76E-08	4.86E-08	4.91E-08	4.95E-08	4.97E-08	4.98E-08	4.99E-08	4.99E-08	5E-08	5E-08	5E-08	5E-08	5E-08	5E-08	5E-08	5E-08	5E-08	5E-08
4.623	15.168	0.2402	3.8E-08	4.42E-08	4.64E-08	4.75E-08	4.81E-08	4.84E-08	4.86E-08	4.88E-08	4.88E-08	4.89E-08	4.89E-08	4.9E-08	4.9E-08	4.9E-08	4.9E-08	4.9E-08	4.9E-08	4.9E-08	4.9E-08	4.9E-08
4.866	15.964	0.2452	3.65E-08	4.29E-08	4.53E-08	4.64E-08	4.7E-08	4.74E-08	4.76E-08	4.77E-08	4.78E-08	4.79E-08	4.79E-08	4.8E-08	4.8E-08	4.8E-08	4.8E-08	4.8E-08	4.8E-08	4.8E-08	4.8E-08	4.8E-08
5.114	16.777	0.2502	3.5E-08	4.17E-08	4.41E-08	4.53E-08	4.6E-08	4.63E-08	4.66E-08	4.67E-08	4.68E-08	4.69E-08	4.69E-08	4.69E-08	4.7E-08	4.7E-08	4.7E-08	4.7E-08	4.7E-08	4.7E-08	4.7E-08	4.7E-08
5.366	17.606	0.2552	3.36E-08	4.05E-08	4.3E-08	4.42E-08	4.49E-08	4.53E-08	4.55E-08	4.57E-08	4.58E-08	4.58E-08	4.59E-08	4.59E-08	4.59E-08	4.6E-08	4.6E-08	4.6E-08	4.6E-08	4.6E-08	4.6E-08	4.6E-08
5.624	18.451	0.2602	3.21E-08	3.92E-08	4.19E-08	4.31E-08	4.38E-08	4.42E-08	4.45E-08	4.47E-08	4.48E-08	4.48E-08	4.49E-08	4.49E-08	4.49E-08	4.49E-08	4.5E-08	4.5E-08	4.5E-08	4.5E-08	4.5E-08	4.5E-08
5.887	19.313	0.2652	3.07E-08	3.8E-08	4.07E-08	4.21E-08	4.28E-08	4.32E-08	4.35E-08	4.36E-08	4.37E-08	4.38E-08	4.39E-08	4.39E-08	4.39E-08	4.39E-08	4.4E-08	4.4E-08	4.4E-08	4.4E-08	4.4E-08	4.4E-08
6.154	20.191	0.2702	2.93E-08	3.68E-08	3.96E-08	4.1E-08	4.17E-08	4.22E-08	4.24E-08	4.26E-08	4.27E-08	4.28E-08	4.29E-08	4.29E-08	4.29E-08	4.29E-08	4.29E-08	4.3E-08	4.3E-08	4.3E-08	4.3E-08	4.3E-08
6.427	21.086	0.2752	2.79E-08	3.55E-08	3.85E-08	3.99E-08	4.07E-08	4.11E-08	4.14E-08	4.16E-08	4.17E-08	4.18E-08	4.19E-08	4.19E-08	4.19E-08	4.19E-08	4.2E-08	4.2E-08	4.2E-08	4.2E-08	4.2E-08	4.2E-08
6.705	21.997	0.2802	2.65E-08	3.43E-08	3.74E-08	3.88E-08	3.96E-08	4.01E-08	4.04E-08	4.06E-08	4.07E-08	4.08E-08	4.09E-08	4.09E-08	4.09E-08	4.09E-08	4.1E-08	4.1E-08	4.1E-08	4.1E-08	4.1E-08	4.1E-08
6.987	22.925	0.2852	2.51E-08	3.31E-08	3.62E-08	3.78E-08	3.86E-08	3.91E-08	3.94E-08	3.96E-08	3.97E-08	3.98E-08	3.99E-08	3.99E-08	3.99E-08	4E-08	4E-08	4E-08	4E-08	4E-08	4E-08	4E-08
7.275	23.868	0.2902	2.38E-08	3.19E-08	3.51E-08	3.67E-08	3.76E-08	3.81E-08	3.84E-08	3.86E-08	3.87E-08	3.88E-08	3.89E-08	3.89E-08	3.9E-08	3.9E-08	3.9E-08	3.9E-08	3.9E-08	3.9E-08	3.9E-08	3.9E-08
7.568	24.829	0.2952	2.25E-08	3.07E-08	3.4E-08	3.57E-08	3.65E-08	3.71E-08	3.74E-08	3.76E-08	3.78E-08	3.78E-08	3.79E-08	3.79E-08	3.8E-08	3.8E-08	3.8E-08	3.8E-08	3.8E-08	3.8E-08	3.81E-08	3.81E-08
7.866	25.805	0.3002	2.12E-08	2.96E-08	3.29E-08	3.46E-08	3.55E-08	3.61E-08	3.64E-08	3.66E-08	3.68E-08	3.69E-08	3.69E-08	3.7E-08	3.7E-08	3.7E-08	3.71E-08	3.71E-08	3.71E-08	3.71E-08	3.71E-08	3.71E-08
8.168	26.799	0.3052	2E-08	2.84E-08	3.19E-08	3.36E-08	3.45E-08	3.51E-08	3.54E-08	3.57E-08	3.58E-08	3.59E-08	3.6E-08	3.6E-08	3.61E-08	3.61E-08	3.61E-08	3.61E-08	3.61E-08	3.61E-08	3.61E-08	3.61E-08
8.476	27.808	0.3102	1.88E-08	2.73E-08	3.08E-08	3.26E-08	3.35E-08	3.41E-08	3.45E-08	3.47E-08	3.49E-08	3.5E-08	3.5E-08	3.51E-08	3.51E-08	3.51E-08	3.52E-08	3.52E-08	3.52E-08	3.52E-08	3.52E-08	3.52E-08
8.789	28.834	0.3152	1.76E-08	2.62E-08	2.97E-08	3.15E-08	3.25E-08	3.31E-08	3.35E-08	3.38E-08	3.39E-08	3.4E-08	3.41E-08	3.41E-08	3.42E-08	3.42E-08	3.42E-08	3.42E-08	3.42E-08	3.43E-08	3.43E-08	3.43E-08
9.106	29.876	0.3202	1.65E-08	2.51E-08	2.87E-08	3.05E-08	3.16E-08	3.22E-08	3.26E-08	3.28E-08	3.3E-08	3.31E-08	3.32E-08	3.32E-08	3.32E-08	3.33E-08	3.33E-08	3.33E-08	3.33E-08	3.33E-08	3.33E-08	3.33E-08

Coarse Tailings Oxygen Consumption Rates

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s)																			
			TIME (YEAR)																			
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
9.429	30.935	0.3252	1.54E-08	2.4E-08	2.77E-08	2.96E-08	3.06E-08	3.12E-08	3.16E-08	3.19E-08	3.21E-08	3.22E-08	3.22E-08	3.23E-08	3.23E-08	3.24E-08	3.24E-08	3.24E-08	3.24E-08	3.24E-08	3.24E-08	3.24E-08
9.757	32.010	0.3302	1.43E-08	2.29E-08	2.67E-08	2.86E-08	2.97E-08	3.03E-08	3.07E-08	3.1E-08	3.11E-08	3.13E-08	3.13E-08	3.14E-08	3.14E-08	3.15E-08	3.15E-08	3.15E-08	3.15E-08	3.15E-08	3.15E-08	3.15E-08
10.089	33.102	0.3352	1.33E-08	2.19E-08	2.57E-08	2.76E-08	2.87E-08	2.94E-08	2.98E-08	3.01E-08	3.02E-08	3.04E-08	3.04E-08	3.05E-08	3.05E-08	3.06E-08	3.06E-08	3.06E-08	3.06E-08	3.06E-08	3.06E-08	3.06E-08
10.427	34.210	0.3402	1.23E-08	2.08E-08	2.47E-08	2.67E-08	2.78E-08	2.85E-08	2.89E-08	2.92E-08	2.94E-08	2.95E-08	2.96E-08	2.96E-08	2.97E-08	2.97E-08	2.97E-08	2.97E-08	2.97E-08	2.97E-08	2.98E-08	2.98E-08
10.770	35.334	0.3452	1.14E-08	1.98E-08	2.37E-08	2.57E-08	2.69E-08	2.76E-08	2.8E-08	2.83E-08	2.85E-08	2.86E-08	2.87E-08	2.87E-08	2.88E-08	2.88E-08	2.88E-08	2.89E-08	2.89E-08	2.89E-08	2.89E-08	2.89E-08
11.118	36.475	0.3502	1.05E-08	1.89E-08	2.28E-08	2.48E-08	2.6E-08	2.67E-08	2.71E-08	2.74E-08	2.76E-08	2.77E-08	2.78E-08	2.79E-08	2.79E-08	2.8E-08	2.8E-08	2.8E-08	2.8E-08	2.8E-08	2.8E-08	2.8E-08
11.470	37.632	0.3552	9.68E-09	1.79E-08	2.18E-08	2.39E-08	2.51E-08	2.58E-08	2.63E-08	2.66E-08	2.68E-08	2.69E-08	2.7E-08	2.7E-08	2.71E-08	2.71E-08	2.72E-08	2.72E-08	2.72E-08	2.72E-08	2.72E-08	2.72E-08
11.828	38.805	0.3602	8.87E-09	1.7E-08	2.09E-08	2.3E-08	2.42E-08	2.5E-08	2.54E-08	2.57E-08	2.59E-08	2.61E-08	2.62E-08	2.62E-08	2.63E-08	2.63E-08	2.63E-08	2.63E-08	2.64E-08	2.64E-08	2.64E-08	2.64E-08
12.191	39.995	0.3652	8.12E-09	1.61E-08	2E-08	2.21E-08	2.34E-08	2.41E-08	2.46E-08	2.49E-08	2.51E-08	2.52E-08	2.53E-08	2.54E-08	2.55E-08	2.55E-08	2.55E-08	2.55E-08	2.55E-08	2.56E-08	2.56E-08	2.56E-08
12.558	41.202	0.3702	7.4E-09	1.52E-08	1.91E-08	2.13E-08	2.25E-08	2.33E-08	2.38E-08	2.41E-08	2.43E-08	2.44E-08	2.45E-08	2.46E-08	2.47E-08	2.47E-08	2.47E-08	2.47E-08	2.48E-08	2.48E-08	2.48E-08	2.48E-08
12.931	42.425	0.3752	6.73E-09	1.43E-08	1.83E-08	2.04E-08	2.17E-08	2.25E-08	2.3E-08	2.33E-08	2.35E-08	2.37E-08	2.38E-08	2.38E-08	2.39E-08	2.39E-08	2.39E-08	2.4E-08	2.4E-08	2.4E-08	2.4E-08	2.4E-08
13.309	43.664	0.3802	6.1E-09	1.35E-08	1.74E-08	1.96E-08	2.09E-08	2.17E-08	2.22E-08	2.25E-08	2.27E-08	2.29E-08	2.3E-08	2.31E-08	2.31E-08	2.31E-08	2.32E-08	2.32E-08	2.32E-08	2.32E-08	2.32E-08	2.32E-08
13.691	44.919	0.3852	5.51E-09	1.27E-08	1.66E-08	1.88E-08	2.01E-08	2.09E-08	2.14E-08	2.18E-08	2.2E-08	2.21E-08	2.22E-08	2.23E-08	2.23E-08	2.24E-08	2.24E-08	2.24E-08	2.25E-08	2.25E-08	2.25E-08	2.25E-08
14.079	46.191	0.3902	4.96E-09	1.19E-08	1.58E-08	1.8E-08	1.93E-08	2.01E-08	2.06E-08	2.1E-08	2.12E-08	2.14E-08	2.15E-08	2.16E-08	2.16E-08	2.16E-08	2.17E-08	2.17E-08	2.17E-08	2.17E-08	2.17E-08	2.17E-08
14.472	47.480	0.3952	4.45E-09	1.12E-08	1.51E-08	1.73E-08	1.86E-08	1.94E-08	1.99E-08	2.03E-08	2.05E-08	2.06E-08	2.08E-08	2.08E-08	2.09E-08	2.09E-08	2.1E-08	2.1E-08	2.1E-08	2.1E-08	2.1E-08	2.1E-08
14.870	48.784	0.4002	3.98E-09	1.05E-08	1.43E-08	1.65E-08	1.78E-08	1.87E-08	1.92E-08	1.95E-08	1.98E-08	1.99E-08	2E-08	2.01E-08	2.02E-08	2.02E-08	2.02E-08	2.03E-08	2.03E-08	2.03E-08	2.03E-08	2.03E-08
15.272	50.106	0.4052	3.55E-09	9.8E-09	1.36E-08	1.58E-08	1.71E-08	1.79E-08	1.85E-08	1.88E-08	1.91E-08	1.92E-08	1.93E-08	1.94E-08	1.95E-08	1.95E-08	1.96E-08	1.96E-08	1.96E-08	1.96E-08	1.96E-08	1.96E-08
15.680	51.443	0.4102	3.16E-09	9.14E-09	1.29E-08	1.51E-08	1.64E-08	1.72E-08	1.78E-08	1.82E-08	1.84E-08	1.86E-08	1.87E-08	1.87E-08	1.88E-08	1.89E-08	1.89E-08	1.89E-08	1.89E-08	1.89E-08	1.89E-08	1.9E-08
16.093	52.797	0.4152	2.79E-09	8.51E-09	1.22E-08	1.44E-08	1.57E-08	1.66E-08	1.71E-08	1.75E-08	1.77E-08	1.79E-08	1.8E-08	1.81E-08	1.81E-08	1.82E-08	1.82E-08	1.83E-08	1.83E-08	1.83E-08	1.83E-08	1.83E-08
16.510	54.168	0.4202	2.46E-09	7.91E-09	1.15E-08	1.37E-08	1.5E-08	1.59E-08	1.64E-08	1.68E-08	1.71E-08	1.72E-08	1.74E-08	1.74E-08	1.75E-08	1.75E-08	1.76E-08	1.76E-08	1.76E-08	1.76E-08	1.76E-08	1.77E-08
16.933	55.554	0.4252	2.16E-09	7.34E-09	1.09E-08	1.3E-08	1.44E-08	1.52E-08	1.58E-08	1.62E-08	1.64E-08	1.66E-08	1.67E-08	1.68E-08	1.69E-08	1.69E-08	1.7E-08	1.7E-08	1.7E-08	1.7E-08	1.7E-08	1.7E-08
17.361	56.958	0.4302	1.89E-09	6.8E-09	1.03E-08	1.24E-08	1.37E-08	1.46E-08	1.52E-08	1.56E-08	1.58E-08	1.6E-08	1.61E-08	1.62E-08	1.63E-08	1.63E-08	1.63E-08	1.64E-08	1.64E-08	1.64E-08	1.64E-08	1.64E-08
17.793	58.377	0.4352	1.65E-09	6.28E-09	9.66E-09	1.18E-08	1.31E-08	1.4E-08	1.45E-08	1.49E-08	1.52E-08	1.54E-08	1.55E-08	1.56E-08	1.57E-08	1.57E-08	1.57E-08	1.58E-08	1.58E-08	1.58E-08	1.58E-08	1.58E-08
18.231	59.813	0.4402	1.43E-09	5.79E-09	9.08E-09	1.12E-08	1.25E-08	1.34E-08	1.4E-08	1.44E-08	1.46E-08	1.48E-08	1.49E-08	1.5E-08	1.51E-08	1.51E-08	1.52E-08	1.52E-08	1.52E-08	1.52E-08	1.52E-08	1.52E-08
18.674	61.266	0.4452	1.24E-09	5.33E-09	8.53E-09	1.06E-08	1.19E-08	1.28E-08	1.34E-08	1.38E-08	1.4E-08	1.42E-08	1.43E-08	1.44E-08	1.45E-08	1.46E-08	1.46E-08	1.46E-08	1.46E-08	1.47E-08	1.47E-08	1.47E-08
19.122	62.735	0.4502	1.06E-09	4.9E-09	8E-09	1.01E-08	1.14E-08	1.22E-08	1.28E-08	1.32E-08	1.35E-08	1.37E-08	1.38E-08	1.39E-08	1.4E-08	1.4E-08	1.4E-08	1.41E-08	1.41E-08	1.41E-08	1.41E-08	1.41E-08
19.574	64.220	0.4552	9.11E-10	4.49E-09	7.5E-09	9.51E-09	1.08E-08	1.17E-08	1.23E-08	1.27E-08	1.29E-08	1.31E-08	1.33E-08	1.33E-08	1.34E-08	1.35E-08	1.35E-08	1.35E-08	1.36E-08	1.36E-08	1.36E-08	1.36E-08
20.032	65.721	0.4602	7.76E-10	4.1E-09	7.01E-09	8.99E-09	1.03E-08	1.12E-08	1.17E-08	1.21E-08	1.24E-08	1.26E-08	1.27E-08	1.28E-08	1.29E-08	1.29E-08	1.3E-08	1.3E-08	1.3E-08	1.3E-08	1.31E-08	1.31E-08
20.495	67.240	0.4652	6.59E-10	3.74E-09	6.54E-09	8.49E-09	9.78E-09	1.06E-08	1.12E-08	1.16E-08	1.19E-08	1.21E-08	1.22E-08	1.23E-08	1.24E-08	1.24E-08	1.25E-08	1.25E-08	1.25E-08	1.25E-08	1.26E-08	1.26E-08
20.962	68.774	0.4702	5.56E-10	3.4E-09	6.1E-09	8.01E-09	9.28E-09	1.01E-08	1.07E-08	1.11E-08	1.14E-08	1.16E-08	1.17E-08	1.18E-08	1.19E-08	1.19E-08	1.2E-08	1.2E-08	1.2E-08	1.21E-08	1.21E-08	1.21E-08
21.435	70.325	0.4752	4.68E-10	3.09E-09	5.68E-09	7.54E-09	8.8E-09	9.65E-09	1.02E-08	1.06E-08	1.09E-08	1.11E-08	1.12E-08	1.13E-08	1.14E-08	1.15E-08	1.15E-08	1.15E-08	1.16E-08	1.16E-08	1.16E-08	1.16E-08
21.913	71.892	0.4802	3.91E-10	2.8E-09	5.28E-09	7.09E-09	8.34E-09	9.18E-09	9.75E-09	1.02E-08	1.04E-08	1.06E-08	1.08E-08	1.09E-08	1.1E-08	1.1E-08	1.1E-08	1.11E-08	1.11E-08	1.11E-08	1.11E-08	1.11E-08
22.395	73.476	0.4852	3.26E-10	2.53E-09	4.9E-09	6.67E-09	7.89E-09	8.73E-09	9.3E-09	9.72E-09	9.99E-09	1.02E-08	1.03E-08	1.04E-08	1.05E-08	1.06E-08	1.06E-08	1.06E-08	1.07E-08	1.07E-08	1.07E-08	1.07E-08
22.883	75.076	0.4902	2.7E-10	2.28E-09	4.54E-09	6.26E-09	7.46E-09	8.29E-09	8.86E-09	9.28E-09	9.55E-09	9.75E-09	9.89E-09	9.99E-09	1.01E-08	1.01E-08	1.02E-08	1.02E-08	1.02E-08	1.02E-08	1.03E-08	1.03E-08
23.376	76.692	0.4952	2.22E-10	2.05E-09	4.2E-09	5.87E-09	7.05E-09	7.87E-09	8.43E-09	8.85E-09	9.13E-09	9.33E-09	9.47E-09	9.57E-09	9.65E-09	9.71E-09	9.75E-09	9.78E-09	9.81E-09	9.83E-09	9.84E-09	9.85E-09
23.875	78.328	0.5022	1.82E-10	1.83E-09	3.87E-09	5.49E-09	6.65E-09	7.46E-09	8.02E-09	8.44E-09	8.72E-09	8.92E-09	9.06E-09	9.17E-09	9.24E-09	9.3E-09	9.35E-09	9.38E-09	9.4E-09	9.42E-09	9.44E-09	9.45E-09
24.380	79.988	0.5092	1.48E-10	1.64E-09	3.57E-09	5.13E-09	6.27E-09	7.07E-09	7.63E-09	8.04E-09	8.32E-09	8.52E-09	8.66E-09	8.77E-09	8.85E-09	8.91E-09	8.95E-09	8.99E-09	9.01E-09	9.03E-09	9.04E-09	9.05E-09
24.893	81.670	0.5162	1.2E-10	1.46E-09	3.28E-09	4.79E-09	5.9E-09	6.68E-09	7.24E-09	7.66E-09	7.93E-09	8.13E-09	8.28E-09	8.39E-09	8.47E-09	8.53E-09	8.57E-09	8.61E-09	8.63E-09	8.65E-09	8.66E-09	8.67E-09
25.413	83.375	0.5232	9.61E-11	1.29E-09	3E-09	4.46E-09	5.54E-09	6.32E-09	6.87E-09	7.28E-09	7.56E-09	7.76E-09	7.91E-09	8.02E-09	8.1E-09	8.16E-09	8.2E-09	8.24E-09	8.26E-09	8.28E-09	8.3E-09	8.31E-09
25.939	85.103	0.5302	7.66E-11	1.14E-09	2.75E-09	4.15E-09	5.2E-09	5.96E-09	6.51E-09	6.92E-09	7.2E-09	7.4E-09	7.55E-09	7.66E-09	7.74E-09	7.8E-09	7.85E-09	7.88E-09	7.91E-09	7.93E-09	7.94E-09	7.95E-09
26.473	86.854	0.5372	6.07E-11	1.01E-09	2.51E-09	3.85E-09	4.88E-09	5.62E-09	6.16E-09	6.57E-09	6.85E-09	7.05E-09	7.2E-09	7.31E-09	7.39E-09	7.46E-09	7.5E-09	7.54E-09	7.56E-09	7.58E-09	7.6E-09	7.61E-09
27.014	88.628	0.5442	4.78E-11	8.86E-10	2.28E-09	3.57E-09	4.56E-09	5.3E-09	5.83E-09	6.24E-09	6.51E-09	6.72E-09	6.87E-09	6.98E-09	7.06E-09	7.12E-09	7.17E-09	7.2E-09	7.23E-09	7.25E-09	7.27E-09	7.28E-09
27.561	90.425	0.5512	3.73E-11	7.75E-10	2.08E-09	3.3E-09	4.27E-09	4.98E-09	5.51E-09	5.91E-09	6.19E-09	6.39E-09	6.54E-09	6.65E-09	6.74E-09	6.8E-09	6.85E-09	6.88E-09	6.91E-09	6.93E-09	6.95E-09	6.96E-09
28.116	92.244	0.5582	2.89E-11	6.76E-10	1.88E-09	3.05E-09	3.98E-09	4.68E-09	5.2E-09	5.6E-09	5.88E-09	6.08E-09	6.23E-09	6.34E-09	6.43E-09	6.49E-09	6.54E-09	6.58E-09	6.6E-09	6.62E-09	6.64E-09	6.65E-09
28.678	94.087	0.5652	2.23E-11	5.87E-10	1.7E-09	2.81E-09	3.71E-09	4.4E-09	4.91E-09	5.3E-09	5.58E-09	5.78E-09	5.93E-09	6.05E-09	6.13E-09	6.19E-09	6.24E-09	6.28E-09	6.31E-09	6.33E-09	6.34E-09	6.35E-09
29.247	95.953	0.5722	1.7E-11	5.07E-10	1.53E-09	2.58E-09	3.45E-09	4.12E-09	4.62E-09	5.02E-09	5.29E-09	5.49E-09	5.64E-09	5.76E-09	5.84E-09	5.91E-09	5.96E-09	5.99E-09	6.02E-09	6.04E-09	6.06E-09	6.07E-09
29.822	97.842	0.5792	1.29E-11	4.37E-10	1.38E-09	2.37E-09	3.21E-09	3.86E-09	4.35E-09	4.74E-09	5.01E-09	5.22E-09	5.37E-09	5.48E-09	5.57E-09	5.63E-09	5.68E-09	5.72E-09	5.75E-09	5.77E-09	5.79E-09	5.8E-09
30.405	99.754	0.5862	9.68E-12	3.75E-10	1.24E-09	2.17E-09	2.98E-09	3.61E-09	4.09E-09	4.48E-09	4.75E-09	4.95E-09	5.1E-09	5.22E-09	5.3E-09	5.37E-09	5.42E-09	5.46E-09	5.49E-09	5.51E-09	5.52E-09	5.54E-09
30.995	101.688	0.5932	7.22E-12	3.2E-10	1.1E-09	1.99E-09	2.76E-09	3.37E-09	3.85E-09	4.23E-09	4.5E-09	4.7E-09	4.85E-09	4.97E-09	5.05E-09	5.12E-09	5.17E-09	5.21E-09	5.23E-09	5.26E-09	5.27E-09	5.29E-09
31.591	103.646	0.6002	5.34E-12	2.73E-10	9.85E-10	1.81E-09	2.55E-09	3.15E-09	3.61E-09	3.99E-09	4.25E-09	4.45E-09	4.61E-09	4.72E-09	4.81E-09	4.88E-09	4.93E-09	4.97E-09	5E-09	5.02E-09	5.03E-09	5.05E-09
32.195	105.627	0.6072	3.92E-12	2.31E-10	8.76E-10	1.65E-09	2.36E-09	2.93E-09	3.39E-09	3.76E-09	4.02E-09	4.22E-09	4.38E-09	4.49E-09	4.58E-09	4.65E-09	4.7E-09	4.74E-09	4.77E-09	4.79E-09	4.81E-09	4.82E-09

Coarse Tailings Oxygen Consumption Rates

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s)																			
			TIME (YEAR)																			
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
32.806	107.630	0.6142	2.85E-12	1.95E-10	7.77E-10	1.5E-09	2.17E-09	2.73E-09	3.18E-09	3.54E-09	3.8E-09	4E-09	4.16E-09	4.27E-09	4.36E-09	4.43E-09	4.48E-09	4.52E-09	4.55E-09	4.57E-09	4.59E-09	4.6E-09
33.423	109.657	0.6212	2.05E-12	1.63E-10	6.86E-10	1.36E-09	2E-09	2.54E-09	2.98E-09	3.34E-09	3.59E-09	3.79E-09	3.95E-09	4.06E-09	4.15E-09	4.22E-09	4.27E-09	4.31E-09	4.34E-09	4.36E-09	4.38E-09	4.39E-09
34.048	111.706	0.6282	1.47E-12	1.37E-10	6.05E-10	1.23E-09	1.84E-09	2.36E-09	2.78E-09	3.14E-09	3.4E-09	3.59E-09	3.75E-09	3.86E-09	3.95E-09	4.02E-09	4.07E-09	4.11E-09	4.14E-09	4.17E-09	4.18E-09	4.2E-09
34.680	113.779	0.6352	1.04E-12	1.14E-10	5.32E-10	1.11E-09	1.69E-09	2.19E-09	2.61E-09	2.95E-09	3.21E-09	3.41E-09	3.56E-09	3.68E-09	3.77E-09	3.83E-09	3.89E-09	3.93E-09	3.96E-09	3.98E-09	4E-09	4.01E-09
35.319	115.874	0.6422	7.28E-13	9.4E-11	4.66E-10	1E-09	1.55E-09	2.03E-09	2.44E-09	2.78E-09	3.03E-09	3.23E-09	3.38E-09	3.5E-09	3.59E-09	3.66E-09	3.71E-09	3.75E-09	3.78E-09	3.8E-09	3.82E-09	3.83E-09
35.964	117.993	0.6492	5.06E-13	7.74E-11	4.06E-10	9.01E-10	1.42E-09	1.88E-09	2.28E-09	2.61E-09	2.86E-09	3.06E-09	3.21E-09	3.33E-09	3.42E-09	3.49E-09	3.54E-09	3.58E-09	3.61E-09	3.64E-09	3.65E-09	3.67E-09
36.617	120.134	0.6562	3.49E-13	6.34E-11	3.54E-10	8.09E-10	1.3E-09	1.75E-09	2.13E-09	2.46E-09	2.71E-09	2.9E-09	3.05E-09	3.17E-09	3.26E-09	3.33E-09	3.39E-09	3.43E-09	3.46E-09	3.48E-09	3.5E-09	3.51E-09
37.283	122.320	0.6762	2.37E-13	5.16E-11	3.06E-10	7.24E-10	1.18E-09	1.61E-09	1.99E-09	2.31E-09	2.56E-09	2.75E-09	2.9E-09	3.02E-09	3.11E-09	3.18E-09	3.24E-09	3.28E-09	3.31E-09	3.33E-09	3.35E-09	3.36E-09
37.969	124.571	0.6962	1.58E-13	4.16E-11	2.64E-10	6.45E-10	1.08E-09	1.49E-09	1.85E-09	2.18E-09	2.42E-09	2.61E-09	2.76E-09	2.88E-09	2.97E-09	3.04E-09	3.09E-09	3.14E-09	3.17E-09	3.19E-09	3.21E-09	3.22E-09
38.676	126.888	0.7162	1.04E-13	3.32E-11	2.26E-10	5.73E-10	9.8E-10	1.38E-09	1.73E-09	2.05E-09	2.29E-09	2.48E-09	2.63E-09	2.75E-09	2.84E-09	2.91E-09	2.96E-09	3E-09	3.03E-09	3.06E-09	3.08E-09	3.09E-09
39.402	129.271	0.7362	6.67E-14	2.62E-11	1.92E-10	5.08E-10	8.89E-10	1.27E-09	1.61E-09	1.92E-09	2.16E-09	2.35E-09	2.5E-09	2.62E-09	2.71E-09	2.78E-09	2.84E-09	2.88E-09	2.91E-09	2.93E-09	2.95E-09	2.97E-09
40.148	131.719	0.7562	4.21E-14	2.05E-11	1.63E-10	4.48E-10	8.06E-10	1.17E-09	1.51E-09	1.81E-09	2.05E-09	2.24E-09	2.39E-09	2.5E-09	2.6E-09	2.67E-09	2.72E-09	2.76E-09	2.79E-09	2.82E-09	2.84E-09	2.85E-09
40.914	134.233	0.7762	2.61E-14	1.59E-11	1.37E-10	3.95E-10	7.31E-10	1.08E-09	1.41E-09	1.71E-09	1.94E-09	2.13E-09	2.28E-09	2.4E-09	2.49E-09	2.56E-09	2.61E-09	2.66E-09	2.69E-09	2.71E-09	2.73E-09	2.75E-09
41.700	136.812	0.7962	1.59E-14	1.22E-11	1.15E-10	3.49E-10	6.64E-10	1E-09	1.32E-09	1.61E-09	1.84E-09	2.03E-09	2.18E-09	2.3E-09	2.39E-09	2.46E-09	2.52E-09	2.56E-09	2.59E-09	2.62E-09	2.63E-09	2.65E-09
42.507	139.457	0.8162	9.45E-15	9.32E-12	9.69E-11	3.08E-10	6.05E-10	9.28E-10	1.24E-09	1.53E-09	1.76E-09	1.95E-09	2.09E-09	2.21E-09	2.3E-09	2.38E-09	2.43E-09	2.47E-09	2.51E-09	2.53E-09	2.55E-09	2.56E-09
43.333	142.168	0.8362	5.52E-15	7.07E-12	8.16E-11	2.74E-10	5.54E-10	8.66E-10	1.17E-09	1.46E-09	1.68E-09	1.87E-09	2.02E-09	2.14E-09	2.23E-09	2.3E-09	2.36E-09	2.4E-09	2.43E-09	2.46E-09	2.47E-09	2.49E-09
44.179	144.944	0.8562	3.16E-15	5.36E-12	6.91E-11	2.45E-10	5.11E-10	8.14E-10	1.11E-09	1.39E-09	1.62E-09	1.81E-09	1.96E-09	2.07E-09	2.17E-09	2.24E-09	2.29E-09	2.33E-09	2.37E-09	2.39E-09	2.41E-09	2.43E-09
45.045	147.786	0.8762	1.79E-15	4.1E-12	5.95E-11	2.22E-10	4.77E-10	7.72E-10	1.06E-09	1.35E-09	1.57E-09	1.75E-09	1.9E-09	2.02E-09	2.11E-09	2.19E-09	2.24E-09	2.28E-09	2.32E-09	2.34E-09	2.36E-09	2.37E-09
45.931	150.693	0.8962	1.02E-15	3.24E-12	5.25E-11	2.06E-10	4.52E-10	7.41E-10	1.03E-09	1.31E-09	1.53E-09	1.72E-09	1.86E-09	1.98E-09	2.08E-09	2.15E-09	2.2E-09	2.24E-09	2.28E-09	2.3E-09	2.32E-09	2.34E-09
46.838	153.666	0.9162	6.22E-16	2.71E-12	4.8E-11	1.95E-10	4.35E-10	7.21E-10	1.01E-09	1.28E-09	1.51E-09	1.69E-09	1.84E-09	1.96E-09	2.05E-09	2.12E-09	2.18E-09	2.22E-09	2.25E-09	2.28E-09	2.3E-09	2.31E-09
47.616	156.221	0.64134	4.97E-16	2.53E-12	4.65E-11	1.91E-10	4.3E-10	7.13E-10	9.99E-10	1.28E-09	1.5E-09	1.68E-09	1.83E-09	1.95E-09	2.04E-09	2.11E-09	2.17E-09	2.21E-09	2.24E-09	2.27E-09	2.29E-09	2.3E-09

Coarse Tailings Oxygen Consur

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s)																		
			TIME (YEAR)																		
			21	22	24	27	30	33	36	39	42	45	48	51	54	57	60	63	66	69	72	75
0.001	0.002	0.001	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	0	0	0	0	0	0	0
0.002	0.007	0.002	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	0	0	0	0	0	0	0
0.007	0.023	0.008	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	7.3E-08	0	0	0	0	0	0	0
0.019	0.062	0.016	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	7.29E-08	0	0	0	0	0	0	0
0.043	0.141	0.032	7.28E-08	7.28E-08	7.28E-08	7.28E-08	7.28E-08	7.28E-08	7.28E-08	7.28E-08	7.28E-08	7.28E-08	7.28E-08	7.28E-08	7.28E-08	0	0	0	0	0	0	0
0.091	0.299	0.064	7.24E-08	7.24E-08	7.24E-08	7.25E-08	7.25E-08	7.25E-08	7.25E-08	7.25E-08	7.25E-08	7.25E-08	7.25E-08	7.25E-08	7.25E-08	0	0	0	0	0	0	0
0.181	0.593	0.1152	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	7.19E-08	0	0	0	0	0	0	0
0.298	0.979	0.1202	7.12E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08	7.12E-08	0	0	0	0	0	0	0
0.421	1.381	0.1252	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	7.04E-08	0	0	0	0	0	0
0.549	1.800	0.1302	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.96E-08	6.97E-08	0	0	0	0	0	0
0.681	2.236	0.1352	6.89E-08	6.89E-08	6.89E-08	6.89E-08	6.89E-08	6.89E-08	6.89E-08	6.89E-08	6.89E-08	6.89E-08	6.89E-08	6.89E-08	6.89E-08	6.89E-08	0	0	0	0	0	0
0.819	2.687	0.1402	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.8E-08	6.81E-08	0	0	0	0	0	0
0.962	3.156	0.1452	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.72E-08	6.74E-08	0	0	0	0	0
1.110	3.640	0.1502	6.64E-08	6.64E-08	6.64E-08	6.64E-08	6.64E-08	6.64E-08	6.64E-08	6.64E-08	6.64E-08	6.64E-08	6.64E-08	6.64E-08	6.64E-08	6.64E-08	6.65E-08	0	0	0	0	0
1.262	4.141	0.1552	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.55E-08	6.57E-08	0	0	0	0	0
1.420	4.658	0.1602	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.46E-08	6.48E-08	6.48E-08	0	0	0	0
1.583	5.192	0.1652	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.37E-08	6.39E-08	6.39E-08	0	0	0	0
1.750	5.742	0.1702	6.28E-08	6.28E-08	6.28E-08	6.28E-08	6.28E-08	6.28E-08	6.28E-08	6.28E-08	6.28E-08	6.28E-08	6.28E-08	6.28E-08	6.28E-08	6.28E-08	6.3E-08	6.3E-08	0	0	0	0
1.923	6.309	0.1752	6.19E-08	6.19E-08	6.19E-08	6.19E-08	6.19E-08	6.19E-08	6.19E-08	6.19E-08	6.19E-08	6.19E-08	6.19E-08	6.19E-08	6.19E-08	6.19E-08	6.2E-08	6.21E-08	6.25E-08	0	0	0
2.101	6.892	0.1802	6.09E-08	6.09E-08	6.09E-08	6.09E-08	6.09E-08	6.09E-08	6.09E-08	6.09E-08	6.09E-08	6.09E-08	6.09E-08	6.09E-08	6.09E-08	6.1E-08	6.11E-08	6.11E-08	6.16E-08	0	0	0
2.283	7.491	0.1852	6E-08	6E-08	6E-08	6E-08	6E-08	6E-08	6E-08	6E-08	6E-08	6E-08	6E-08	6E-08	6E-08	6E-08	6.01E-08	6.02E-08	6.06E-08	0	0	0
2.471	8.107	0.1902	5.9E-08	5.9E-08	5.9E-08	5.9E-08	5.9E-08	5.9E-08	5.9E-08	5.9E-08	5.9E-08	5.9E-08	5.9E-08	5.9E-08	5.9E-08	5.9E-08	5.92E-08	5.92E-08	5.97E-08	6.01E-08	0	0
2.664	8.740	0.1952	5.8E-08	5.81E-08	5.81E-08	5.81E-08	5.81E-08	5.81E-08	5.81E-08	5.81E-08	5.81E-08	5.81E-08	5.81E-08	5.81E-08	5.81E-08	5.81E-08	5.82E-08	5.82E-08	5.87E-08	5.91E-08	0	0
2.862	9.388	0.2002	5.71E-08	5.71E-08	5.71E-08	5.71E-08	5.71E-08	5.71E-08	5.71E-08	5.71E-08	5.71E-08	5.71E-08	5.71E-08	5.71E-08	5.71E-08	5.71E-08	5.72E-08	5.72E-08	5.77E-08	5.81E-08	0	0
3.064	10.053	0.2052	5.61E-08	5.61E-08	5.61E-08	5.61E-08	5.61E-08	5.61E-08	5.61E-08	5.61E-08	5.61E-08	5.61E-08	5.61E-08	5.61E-08	5.61E-08	5.61E-08	5.62E-08	5.62E-08	5.67E-08	5.71E-08	5.74E-08	0
3.272	10.735	0.2102	5.51E-08	5.51E-08	5.51E-08	5.51E-08	5.51E-08	5.51E-08	5.51E-08	5.51E-08	5.51E-08	5.51E-08	5.51E-08	5.51E-08	5.51E-08	5.51E-08	5.52E-08	5.52E-08	5.57E-08	5.61E-08	5.64E-08	0
3.485	11.432	0.2152	5.41E-08	5.41E-08	5.41E-08	5.41E-08	5.41E-08	5.41E-08	5.41E-08	5.41E-08	5.41E-08	5.41E-08	5.41E-08	5.41E-08	5.41E-08	5.41E-08	5.42E-08	5.42E-08	5.47E-08	5.5E-08	5.54E-08	5.6E-08
3.702	12.147	0.2202	5.31E-08	5.31E-08	5.31E-08	5.31E-08	5.31E-08	5.31E-08	5.31E-08	5.31E-08	5.31E-08	5.31E-08	5.31E-08	5.31E-08	5.31E-08	5.31E-08	5.32E-08	5.32E-08	5.36E-08	5.4E-08	5.44E-08	5.49E-08
3.925	12.877	0.2252	5.21E-08	5.21E-08	5.21E-08	5.21E-08	5.21E-08	5.21E-08	5.21E-08	5.21E-08	5.21E-08	5.21E-08	5.21E-08	5.21E-08	5.21E-08	5.21E-08	5.22E-08	5.22E-08	5.26E-08	5.3E-08	5.33E-08	5.39E-08
4.153	13.624	0.2302	5.11E-08	5.11E-08	5.11E-08	5.11E-08	5.11E-08	5.11E-08	5.11E-08	5.11E-08	5.11E-08	5.11E-08	5.11E-08	5.11E-08	5.11E-08	5.11E-08	5.12E-08	5.12E-08	5.16E-08	5.19E-08	5.23E-08	5.28E-08
4.385	14.388	0.2352	5.01E-08	5.01E-08	5.01E-08	5.01E-08	5.01E-08	5.01E-08	5.01E-08	5.01E-08	5.01E-08	5.01E-08	5.01E-08	5.01E-08	5.01E-08	5.01E-08	5.02E-08	5.02E-08	5.06E-08	5.09E-08	5.13E-08	5.18E-08
4.623	15.168	0.2402	4.9E-08	4.9E-08	4.9E-08	4.9E-08	4.9E-08	4.9E-08	4.9E-08	4.9E-08	4.9E-08	4.9E-08	4.9E-08	4.9E-08	4.9E-08	4.91E-08	4.92E-08	4.92E-08	4.95E-08	4.99E-08	5.02E-08	5.07E-08
4.866	15.964	0.2452	4.8E-08	4.8E-08	4.8E-08	4.8E-08	4.8E-08	4.8E-08	4.8E-08	4.8E-08	4.8E-08	4.8E-08	4.8E-08	4.8E-08	4.8E-08	4.8E-08	4.81E-08	4.82E-08	4.85E-08	4.88E-08	4.92E-08	4.97E-08
5.114	16.777	0.2502	4.7E-08	4.7E-08	4.7E-08	4.7E-08	4.7E-08	4.7E-08	4.7E-08	4.7E-08	4.7E-08	4.7E-08	4.7E-08	4.7E-08	4.7E-08	4.7E-08	4.71E-08	4.71E-08	4.75E-08	4.78E-08	4.81E-08	4.86E-08
5.366	17.606	0.2552	4.6E-08	4.6E-08	4.6E-08	4.6E-08	4.6E-08	4.6E-08	4.6E-08	4.6E-08	4.6E-08	4.6E-08	4.6E-08	4.6E-08	4.6E-08	4.6E-08	4.61E-08	4.61E-08	4.65E-08	4.68E-08	4.71E-08	4.75E-08
5.624	18.451	0.2602	4.5E-08	4.5E-08	4.5E-08	4.5E-08	4.5E-08	4.5E-08	4.5E-08	4.5E-08	4.5E-08	4.5E-08	4.5E-08	4.5E-08	4.5E-08	4.5E-08	4.51E-08	4.51E-08	4.54E-08	4.57E-08	4.61E-08	4.65E-08
5.887	19.313	0.2652	4.4E-08	4.4E-08	4.4E-08	4.4E-08	4.4E-08	4.4E-08	4.4E-08	4.4E-08	4.4E-08	4.4E-08	4.4E-08	4.4E-08	4.4E-08	4.4E-08	4.41E-08	4.41E-08	4.44E-08	4.47E-08	4.5E-08	4.55E-08
6.154	20.191	0.2702	4.3E-08	4.3E-08	4.3E-08	4.3E-08	4.3E-08	4.3E-08	4.3E-08	4.3E-08	4.3E-08	4.3E-08	4.3E-08	4.3E-08	4.3E-08	4.3E-08	4.31E-08	4.31E-08	4.34E-08	4.37E-08	4.4E-08	4.44E-08
6.427	21.086	0.2752	4.2E-08	4.2E-08	4.2E-08	4.2E-08	4.2E-08	4.2E-08	4.2E-08	4.2E-08	4.2E-08	4.2E-08	4.2E-08	4.2E-08	4.2E-08	4.2E-08	4.21E-08	4.21E-08	4.24E-08	4.27E-08	4.3E-08	4.34E-08
6.705	21.997	0.2802	4.1E-08	4.1E-08	4.1E-08	4.1E-08	4.1E-08	4.1E-08	4.1E-08	4.1E-08	4.1E-08	4.1E-08	4.1E-08	4.1E-08	4.1E-08	4.1E-08	4.11E-08	4.11E-08	4.14E-08	4.17E-08	4.19E-08	4.23E-08
6.987	22.925	0.2852	4E-08	4E-08	4E-08	4E-08	4E-08	4E-08	4E-08	4E-08	4E-08	4E-08	4E-08	4E-08	4E-08	4E-08	4.01E-08	4.01E-08	4.04E-08	4.07E-08	4.09E-08	4.13E-08
7.275	23.868	0.2902	3.9E-08	3.9E-08	3.9E-08	3.9E-08	3.9E-08	3.9E-08	3.9E-08	3.9E-08	3.9E-08	3.9E-08	3.9E-08	3.9E-08	3.9E-08	3.9E-08	3.91E-08	3.91E-08	3.94E-08	3.97E-08	3.99E-08	4.03E-08
7.568	24.829	0.2952	3.81E-08	3.81E-08	3.81E-08	3.81E-08	3.81E-08	3.81E-08	3.81E-08	3.81E-08	3.81E-08	3.81E-08	3.81E-08	3.81E-08	3.81E-08	3.81E-08	3.81E-08	3.81E-08	3.84E-08	3.87E-08	3.89E-08	3.93E-08
7.866	25.805	0.3002	3.71E-08	3.71E-08	3.71E-08	3.71E-08	3.71E-08	3.71E-08	3.71E-08	3.71E-08	3.71E-08	3.71E-08	3.71E-08	3.71E-08	3.71E-08	3.71E-08	3.72E-08	3.72E-08	3.74E-08	3.77E-08	3.79E-08	3.83E-08
8.168	26.799	0.3052	3.61E-08	3.61E-08	3.61E-08	3.61E-08	3.62E-08	3.62E-08	3.62E-08	3.62E-08	3.62E-08	3.62E-08	3.62E-08	3.62E-08	3.62E-08	3.62E-08	3.62E-08	3.62E-08	3.65E-08	3.67E-08	3.7E-08	3.73E-08
8.476	27.808	0.3102	3.52E-08	3.52E-08	3.52E-08	3.52E-08	3.52E-08	3.52E-08	3.52E-08	3.52E-08	3.52E-08	3.52E-08	3.52E-08	3.52E-08	3.52E-08	3.52E-08	3.53E-08	3.53E-08	3.55E-08	3.57E-08	3.6E-08	3.63E-08
8.789	28.834	0.3152	3.43E-08	3.43E-08	3.43E-08	3.43E-08	3.43E-08	3.43E-08	3.43E-08	3.43E-08	3.43E-08	3.43E-08	3.43E-08	3.43E-08	3.43E-08	3.43E-08	3.43E-08	3.43E-08	3.46E-08	3.48E-08	3.5E-08	3.54E-08
9.106	29.876	0.3202	3.33E-08	3.33E-08	3.33E-08	3.33E-08	3.33E-08	3.34E-08	3.34E-08	3.34E-08	3.34E-08	3.34E-08	3.34E-08	3.34E-08	3.34E-08	3.34E-08	3.34E-08	3.34E-08	3.36E-08	3.39E-08	3.41E-08	3.44E-08

Coarse Tailings Oxygen Consur

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s)																			
			TIME (YEAR)																			
			21	22	24	27	30	33	36	39	42	45	48	51	54	57	60	63	66	69	72	75
9.429	30.935	0.3252	3.24E-08	3.24E-08	3.24E-08	3.24E-08	3.24E-08	3.24E-08	3.24E-08	3.24E-08	3.24E-08	3.24E-08	3.24E-08	3.24E-08	3.24E-08	3.24E-08	3.25E-08	3.25E-08	3.27E-08	3.29E-08	3.32E-08	3.34E-08
9.757	32.010	0.3302	3.15E-08	3.15E-08	3.15E-08	3.15E-08	3.15E-08	3.15E-08	3.15E-08	3.15E-08	3.15E-08	3.15E-08	3.15E-08	3.15E-08	3.15E-08	3.15E-08	3.15E-08	3.16E-08	3.16E-08	3.18E-08	3.2E-08	3.22E-08	3.25E-08
10.089	33.102	0.3352	3.06E-08	3.06E-08	3.06E-08	3.06E-08	3.06E-08	3.06E-08	3.06E-08	3.06E-08	3.06E-08	3.06E-08	3.06E-08	3.06E-08	3.06E-08	3.07E-08	3.07E-08	3.07E-08	3.09E-08	3.11E-08	3.13E-08	3.16E-08
10.427	34.210	0.3402	2.98E-08	2.98E-08	2.98E-08	2.98E-08	2.98E-08	2.98E-08	2.98E-08	2.98E-08	2.98E-08	2.98E-08	2.98E-08	2.98E-08	2.98E-08	2.98E-08	2.98E-08	2.98E-08	3E-08	3.02E-08	3.04E-08	3.07E-08
10.770	35.334	0.3452	2.89E-08	2.89E-08	2.89E-08	2.89E-08	2.89E-08	2.89E-08	2.89E-08	2.89E-08	2.89E-08	2.89E-08	2.89E-08	2.89E-08	2.89E-08	2.89E-08	2.9E-08	2.9E-08	2.91E-08	2.93E-08	2.95E-08	2.98E-08
11.118	36.475	0.3502	2.8E-08	2.8E-08	2.81E-08	2.81E-08	2.81E-08	2.81E-08	2.81E-08	2.81E-08	2.81E-08	2.81E-08	2.81E-08	2.81E-08	2.81E-08	2.81E-08	2.81E-08	2.81E-08	2.83E-08	2.85E-08	2.87E-08	2.89E-08
11.470	37.632	0.3552	2.72E-08	2.72E-08	2.72E-08	2.72E-08	2.72E-08	2.72E-08	2.72E-08	2.72E-08	2.72E-08	2.72E-08	2.72E-08	2.72E-08	2.72E-08	2.72E-08	2.73E-08	2.73E-08	2.74E-08	2.76E-08	2.78E-08	2.8E-08
11.828	38.805	0.3602	2.64E-08	2.64E-08	2.64E-08	2.64E-08	2.64E-08	2.64E-08	2.64E-08	2.64E-08	2.64E-08	2.64E-08	2.64E-08	2.64E-08	2.64E-08	2.64E-08	2.64E-08	2.64E-08	2.66E-08	2.68E-08	2.7E-08	2.72E-08
12.191	39.995	0.3652	2.56E-08	2.56E-08	2.56E-08	2.56E-08	2.56E-08	2.56E-08	2.56E-08	2.56E-08	2.56E-08	2.56E-08	2.56E-08	2.56E-08	2.56E-08	2.56E-08	2.56E-08	2.56E-08	2.58E-08	2.59E-08	2.61E-08	2.63E-08
12.558	41.202	0.3702	2.48E-08	2.48E-08	2.48E-08	2.48E-08	2.48E-08	2.48E-08	2.48E-08	2.48E-08	2.48E-08	2.48E-08	2.48E-08	2.48E-08	2.48E-08	2.48E-08	2.48E-08	2.48E-08	2.5E-08	2.51E-08	2.53E-08	2.55E-08
12.931	42.425	0.3752	2.4E-08	2.4E-08	2.4E-08	2.4E-08	2.4E-08	2.4E-08	2.4E-08	2.4E-08	2.4E-08	2.4E-08	2.4E-08	2.4E-08	2.4E-08	2.4E-08	2.4E-08	2.41E-08	2.42E-08	2.43E-08	2.45E-08	2.47E-08
13.309	43.664	0.3802	2.32E-08	2.32E-08	2.32E-08	2.32E-08	2.32E-08	2.32E-08	2.32E-08	2.33E-08	2.33E-08	2.33E-08	2.33E-08	2.33E-08	2.33E-08	2.33E-08	2.33E-08	2.33E-08	2.34E-08	2.36E-08	2.37E-08	2.39E-08
13.691	44.919	0.3852	2.25E-08	2.25E-08	2.25E-08	2.25E-08	2.25E-08	2.25E-08	2.25E-08	2.25E-08	2.25E-08	2.25E-08	2.25E-08	2.25E-08	2.25E-08	2.25E-08	2.25E-08	2.25E-08	2.27E-08	2.28E-08	2.3E-08	2.31E-08
14.079	46.191	0.3902	2.17E-08	2.18E-08	2.18E-08	2.18E-08	2.18E-08	2.18E-08	2.18E-08	2.18E-08	2.18E-08	2.18E-08	2.18E-08	2.18E-08	2.18E-08	2.18E-08	2.18E-08	2.18E-08	2.19E-08	2.2E-08	2.22E-08	2.24E-08
14.472	47.480	0.3952	2.1E-08	2.1E-08	2.1E-08	2.1E-08	2.1E-08	2.1E-08	2.1E-08	2.1E-08	2.1E-08	2.1E-08	2.1E-08	2.1E-08	2.1E-08	2.1E-08	2.11E-08	2.11E-08	2.12E-08	2.13E-08	2.15E-08	2.16E-08
14.870	48.784	0.4002	2.03E-08	2.03E-08	2.03E-08	2.03E-08	2.03E-08	2.03E-08	2.03E-08	2.03E-08	2.03E-08	2.03E-08	2.03E-08	2.03E-08	2.03E-08	2.03E-08	2.03E-08	2.04E-08	2.04E-08	2.05E-08	2.06E-08	2.07E-08	2.09E-08
15.272	50.106	0.4052	1.96E-08	1.96E-08	1.96E-08	1.96E-08	1.97E-08	1.97E-08	1.97E-08	1.97E-08	1.97E-08	1.97E-08	1.97E-08	1.97E-08	1.97E-08	1.97E-08	1.97E-08	1.97E-08	1.98E-08	1.99E-08	2E-08	2.02E-08
15.680	51.443	0.4102	1.9E-08	1.9E-08	1.9E-08	1.9E-08	1.9E-08	1.9E-08	1.9E-08	1.9E-08	1.9E-08	1.9E-08	1.9E-08	1.9E-08	1.9E-08	1.9E-08	1.9E-08	1.9E-08	1.91E-08	1.92E-08	1.93E-08	1.95E-08
16.093	52.797	0.4152	1.83E-08	1.83E-08	1.83E-08	1.83E-08	1.83E-08	1.83E-08	1.83E-08	1.83E-08	1.83E-08	1.83E-08	1.83E-08	1.83E-08	1.83E-08	1.83E-08	1.83E-08	1.83E-08	1.84E-08	1.85E-08	1.87E-08	1.88E-08
16.510	54.168	0.4202	1.77E-08	1.77E-08	1.77E-08	1.77E-08	1.77E-08	1.77E-08	1.77E-08	1.77E-08	1.77E-08	1.77E-08	1.77E-08	1.77E-08	1.77E-08	1.77E-08	1.77E-08	1.77E-08	1.78E-08	1.79E-08	1.8E-08	1.82E-08
16.933	55.554	0.4252	1.7E-08	1.7E-08	1.7E-08	1.7E-08	1.71E-08	1.71E-08	1.71E-08	1.71E-08	1.71E-08	1.71E-08	1.71E-08	1.71E-08	1.71E-08	1.71E-08	1.71E-08	1.71E-08	1.72E-08	1.73E-08	1.74E-08	1.75E-08
17.361	56.958	0.4302	1.64E-08	1.64E-08	1.64E-08	1.64E-08	1.64E-08	1.64E-08	1.64E-08	1.64E-08	1.64E-08	1.64E-08	1.64E-08	1.64E-08	1.64E-08	1.64E-08	1.65E-08	1.65E-08	1.65E-08	1.66E-08	1.68E-08	1.69E-08
17.793	58.377	0.4352	1.58E-08	1.58E-08	1.58E-08	1.58E-08	1.58E-08	1.58E-08	1.58E-08	1.58E-08	1.58E-08	1.58E-08	1.58E-08	1.58E-08	1.58E-08	1.58E-08	1.59E-08	1.59E-08	1.59E-08	1.6E-08	1.61E-08	1.63E-08
18.231	59.813	0.4402	1.52E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.54E-08	1.54E-08	1.55E-08	1.57E-08
18.674	61.266	0.4452	1.47E-08	1.47E-08	1.47E-08	1.47E-08	1.47E-08	1.47E-08	1.47E-08	1.47E-08	1.47E-08	1.47E-08	1.47E-08	1.47E-08	1.47E-08	1.47E-08	1.47E-08	1.47E-08	1.48E-08	1.49E-08	1.5E-08	1.51E-08
19.122	62.735	0.4502	1.41E-08	1.41E-08	1.41E-08	1.41E-08	1.42E-08	1.42E-08	1.42E-08	1.42E-08	1.42E-08	1.42E-08	1.42E-08	1.42E-08	1.42E-08	1.42E-08	1.42E-08	1.42E-08	1.42E-08	1.43E-08	1.44E-08	1.45E-08
19.574	64.220	0.4552	1.36E-08	1.36E-08	1.36E-08	1.36E-08	1.36E-08	1.36E-08	1.36E-08	1.36E-08	1.36E-08	1.36E-08	1.36E-08	1.36E-08	1.36E-08	1.36E-08	1.36E-08	1.36E-08	1.37E-08	1.38E-08	1.39E-08	1.4E-08
20.032	65.721	0.4602	1.31E-08	1.31E-08	1.31E-08	1.31E-08	1.31E-08	1.31E-08	1.31E-08	1.31E-08	1.31E-08	1.31E-08	1.31E-08	1.31E-08	1.31E-08	1.31E-08	1.31E-08	1.31E-08	1.32E-08	1.32E-08	1.33E-08	1.34E-08
20.495	67.240	0.4652	1.26E-08	1.26E-08	1.26E-08	1.26E-08	1.26E-08	1.26E-08	1.26E-08	1.26E-08	1.26E-08	1.26E-08	1.26E-08	1.26E-08	1.26E-08	1.26E-08	1.26E-08	1.26E-08	1.27E-08	1.27E-08	1.28E-08	1.29E-08
20.962	68.774	0.4702	1.21E-08	1.21E-08	1.21E-08	1.21E-08	1.21E-08	1.21E-08	1.21E-08	1.21E-08	1.21E-08	1.21E-08	1.21E-08	1.21E-08	1.21E-08	1.21E-08	1.21E-08	1.21E-08	1.22E-08	1.22E-08	1.23E-08	1.24E-08
21.435	70.325	0.4752	1.16E-08	1.16E-08	1.16E-08	1.16E-08	1.16E-08	1.16E-08	1.16E-08	1.16E-08	1.16E-08	1.16E-08	1.16E-08	1.16E-08	1.16E-08	1.16E-08	1.16E-08	1.16E-08	1.17E-08	1.18E-08	1.18E-08	1.19E-08
21.913	71.892	0.4802	1.12E-08	1.12E-08	1.12E-08	1.12E-08	1.12E-08	1.12E-08	1.12E-08	1.12E-08	1.12E-08	1.12E-08	1.12E-08	1.12E-08	1.12E-08	1.12E-08	1.12E-08	1.12E-08	1.12E-08	1.13E-08	1.14E-08	1.15E-08
22.395	73.476	0.4852	1.07E-08	1.07E-08	1.07E-08	1.07E-08	1.07E-08	1.07E-08	1.07E-08	1.07E-08	1.07E-08	1.07E-08	1.07E-08	1.07E-08	1.07E-08	1.07E-08	1.07E-08	1.07E-08	1.08E-08	1.08E-08	1.09E-08	1.1E-08
22.883	75.076	0.4902	1.03E-08	1.03E-08	1.03E-08	1.03E-08	1.03E-08	1.03E-08	1.03E-08	1.03E-08	1.03E-08	1.03E-08	1.03E-08	1.03E-08	1.03E-08	1.03E-08	1.03E-08	1.03E-08	1.03E-08	1.04E-08	1.05E-08	1.05E-08
23.376	76.692	0.4952	9.86E-09	9.86E-09	9.87E-09	9.88E-09	9.88E-09	9.88E-09	9.88E-09	9.88E-09	9.88E-09	9.88E-09	9.88E-09	9.88E-09	9.88E-09	9.88E-09	9.89E-09	9.89E-09	9.92E-09	9.98E-09	1E-08	1.01E-08
23.875	78.328	0.5022	9.45E-09	9.46E-09	9.47E-09	9.47E-09	9.48E-09	9.48E-09	9.48E-09	9.48E-09	9.48E-09	9.48E-09	9.48E-09	9.48E-09	9.48E-09	9.48E-09	9.48E-09	9.49E-09	9.52E-09	9.57E-09	9.63E-09	9.7E-09
24.380	79.988	0.5092	9.06E-09	9.07E-09	9.08E-09	9.08E-09	9.09E-09	9.09E-09	9.09E-09	9.09E-09	9.09E-09	9.09E-09	9.09E-09	9.09E-09	9.09E-09	9.09E-09	9.09E-09	9.09E-09	9.12E-09	9.17E-09	9.23E-09	9.3E-09
24.893	81.670	0.5162	8.68E-09	8.69E-09	8.7E-09	8.7E-09	8.71E-09	8.71E-09	8.71E-09	8.71E-09	8.71E-09	8.71E-09	8.71E-09	8.71E-09	8.71E-09	8.71E-09	8.71E-09	8.71E-09	8.74E-09	8.79E-09	8.85E-09	8.91E-09
25.413	83.375	0.5232	8.32E-09	8.32E-09	8.33E-09	8.34E-09	8.34E-09	8.34E-09	8.34E-09	8.34E-09	8.34E-09	8.34E-09	8.34E-09	8.34E-09	8.34E-09	8.34E-09	8.35E-09	8.35E-09	8.37E-09	8.42E-09	8.47E-09	8.53E-09
25.939	85.103	0.5302	7.96E-09	7.97E-09	7.98E-09	7.98E-09	7.99E-09	7.99E-09	7.99E-09	7.99E-09	7.99E-09	7.99E-09	7.99E-09	7.99E-09	7.99E-09	7.99E-09	7.99E-09	7.99E-09	8.02E-09	8.06E-09	8.11E-09	8.17E-09
26.473	86.854	0.5372	7.62E-09	7.62E-09	7.63E-09	7.64E-09	7.64E-09	7.64E-09	7.64E-09	7.64E-09	7.64E-09	7.64E-09	7.64E-09	7.64E-09	7.64E-09	7.64E-09	7.64E-09	7.65E-09	7.65E-09	7.67E-09	7.71E-09	7.76E-09	7.82E-09
27.014	88.628	0.5442	7.29E-09	7.29E-09	7.3E-09	7.31E-09	7.31E-09	7.31E-09	7.31E-09	7.31E-09	7.31E-09	7.31E-09	7.31E-09	7.31E-09	7.31E-09	7.31E-09	7.32E-09	7.32E-09	7.34E-09	7.38E-09	7.42E-09	7.48E-09
27.561	90.425	0.5512	6.97E-09	6.97E-09	6.98E-09	6.99E-09	6.99E-09	6.99E-09	6.99E-09	6.99E-09	6.99E-09	6.99E-09	6.99E-09	6.99E-09	6.99E-09	7E-09	7E-09	7E-09	7.02E-09	7.05E-09	7.1E-09	7.15E-09
28.116	92.244	0.5582	6.66E-09	6.67E-09	6.68E-09	6.68E-09	6.69E-09	6.69E-09	6.69E-09	6.69E-09	6.69E-09	6.69E-09	6.69E-09	6.69E-09	6.69E-09	6.69E-09	6.69E-09	6.69E-09	6.71E-09	6.74E-09	6.78E-09	6.83E-09
28.678	94.087	0.5652	6.36E-09	6.37E-09	6.38E-09	6.39E-09	6.39E-09	6.39E-09	6.39E-09	6.39E-09	6.39E-09	6.39E-09	6.39E-09	6.39E-09	6.39E-09	6.39E-09	6.39E-09	6.4E-09	6.41E-09	6.44E-09	6.48E-09	6.53E-09
29.247	95.953	0.5722	6.08E-09	6.09E-09	6.1E-09	6.1E-09	6.11E-09	6.11E-09	6.11E-09	6.11E-09	6.11E-09	6.11E-09	6.11E-09	6.11E-09	6.11E-09	6.11E-09	6.11E-09	6.11E-09	6.13E-09	6.16E-09	6.19E-09	6.24E-09
29.822	97.842	0.5792	5.81E-09	5.81E-09	5.82E-09	5.83E-09	5.83E-09	5.84E-09	5.84E-09	5.84E-09	5.84E-09	5.84E-09	5.84E-09	5.84E-09	5.84E-09	5.84E-09	5.84E-09	5.84E-09	5.85E-09	5.88E-09	5.92E-09	5.96E-09
30.405	99.754	0.5862	5.55E-09	5.55E-09	5.56E-09	5.57E-09	5.57E-09	5.57E-09	5.58E-09	5.58E-09	5.58E-09	5.58E-09	5.58E-09	5.58E-09	5.58E-09	5.58E-09	5.58E-09	5.58E-09	5.59E-09	5.62E-09	5.65E-09	5.69E-09
30.995	101.688	0.5932	5.3E-09	5.3E-09	5.31E-09	5.32E-09	5.32E-09	5.32E-09	5.33E-09	5.33E-09	5.33E-09	5.33E-09	5.33E-09	5.33E-09	5.33E-09	5.33E-09	5.33E-09	5.33E-09	5.34E-09	5.36E-09	5.4E-09	5.43E-09
31.591	103.646	0.6002	5.06E-09	5.06E-09	5.07E-09	5.08E-09	5.08E-09	5.09E-09	5.09E-09	5.09E-09	5.09E-09	5.09E-09	5.09E-09	5.09E-09	5.09E-09	5.09E-09	5.09E-09	5.09E-09	5.1E-09	5.12E-09	5.15E-09	5.19E-09
32.195	105.627	0.6072	4.83E-09	4.84E-09	4.85E-09	4.85E-09	4.86E-09	4.86E-09	4.86E-09	4.86E-09	4.86E-09	4.86E-09	4.86E-09	4.86E-09	4.86E-09	4.86E-09	4.86E-09	4.86E-09	4.87E-09	4.89E-09	4.92E-09	4.95E-09

Coarse Tailings Oxygen Consur

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s)																			
			TIME (YEAR)																			
			21	22	24	27	30	33	36	39	42	45	48	51	54	57	60	63	66	69	72	75
32.806	107.630	0.6142	4.61E-09	4.62E-09	4.63E-09	4.64E-09	4.64E-09	4.64E-09	4.64E-09	4.64E-09	4.64E-09	4.64E-09	4.64E-09	4.64E-09	4.64E-09	4.64E-09	4.64E-09	4.65E-09	4.67E-09	4.7E-09	4.73E-09
33.423	109.657	0.6212	4.4E-09	4.41E-09	4.42E-09	4.43E-09	4.43E-09	4.43E-09	4.43E-09	4.43E-09	4.43E-09	4.44E-09	4.44E-09	4.44E-09	4.44E-09	4.44E-09	4.44E-09	4.44E-09	4.44E-09	4.46E-09	4.49E-09	4.52E-09
34.048	111.706	0.6282	4.21E-09	4.21E-09	4.23E-09	4.23E-09	4.24E-09	4.24E-09	4.24E-09	4.24E-09	4.24E-09	4.24E-09	4.24E-09	4.24E-09	4.24E-09	4.24E-09	4.24E-09	4.24E-09	4.25E-09	4.27E-09	4.29E-09	4.32E-09
34.680	113.779	0.6352	4.02E-09	4.03E-09	4.04E-09	4.05E-09	4.05E-09	4.05E-09	4.05E-09	4.05E-09	4.05E-09	4.05E-09	4.05E-09	4.05E-09	4.05E-09	4.05E-09	4.05E-09	4.05E-09	4.06E-09	4.08E-09	4.1E-09	4.13E-09
35.319	115.874	0.6422	3.84E-09	3.85E-09	3.86E-09	3.87E-09	3.87E-09	3.88E-09	3.88E-09	3.88E-09	3.88E-09	3.88E-09	3.88E-09	3.88E-09	3.88E-09	3.88E-09	3.88E-09	3.88E-09	3.88E-09	3.9E-09	3.92E-09	3.95E-09
35.964	117.993	0.6492	3.68E-09	3.69E-09	3.7E-09	3.71E-09	3.71E-09	3.71E-09	3.71E-09	3.71E-09	3.71E-09	3.71E-09	3.71E-09	3.71E-09	3.71E-09	3.71E-09	3.71E-09	3.71E-09	3.72E-09	3.73E-09	3.75E-09	3.78E-09
36.617	120.134	0.6562	3.52E-09	3.53E-09	3.54E-09	3.55E-09	3.55E-09	3.55E-09	3.56E-09	3.56E-09	3.56E-09	3.56E-09	3.56E-09	3.56E-09	3.56E-09	3.56E-09	3.56E-09	3.56E-09	3.56E-09	3.57E-09	3.59E-09	3.62E-09
37.283	122.320	0.6762	3.38E-09	3.38E-09	3.39E-09	3.4E-09	3.41E-09	3.41E-09	3.41E-09	3.41E-09	3.41E-09	3.41E-09	3.41E-09	3.41E-09	3.41E-09	3.41E-09	3.41E-09	3.41E-09	3.41E-09	3.43E-09	3.44E-09	3.47E-09
37.969	124.571	0.6962	3.23E-09	3.24E-09	3.25E-09	3.26E-09	3.27E-09	3.27E-09	3.27E-09	3.27E-09	3.27E-09	3.27E-09	3.27E-09	3.27E-09	3.27E-09	3.27E-09	3.27E-09	3.27E-09	3.27E-09	3.28E-09	3.3E-09	3.32E-09
38.676	126.888	0.7162	3.1E-09	3.11E-09	3.12E-09	3.13E-09	3.13E-09	3.13E-09	3.14E-09	3.14E-09	3.14E-09	3.14E-09	3.14E-09	3.14E-09	3.14E-09	3.14E-09	3.14E-09	3.14E-09	3.14E-09	3.15E-09	3.17E-09	3.19E-09
39.402	129.271	0.7362	2.98E-09	2.99E-09	3E-09	3.01E-09	3.01E-09	3.01E-09	3.01E-09	3.01E-09	3.01E-09	3.01E-09	3.01E-09	3.01E-09	3.01E-09	3.01E-09	3.01E-09	3.01E-09	3.02E-09	3.03E-09	3.04E-09	3.06E-09
40.148	131.719	0.7562	2.86E-09	2.87E-09	2.88E-09	2.89E-09	2.89E-09	2.9E-09	2.9E-09	2.9E-09	2.9E-09	2.9E-09	2.9E-09	2.9E-09	2.9E-09	2.9E-09	2.9E-09	2.9E-09	2.9E-09	2.91E-09	2.92E-09	2.94E-09
40.914	134.233	0.7762	2.76E-09	2.76E-09	2.78E-09	2.78E-09	2.79E-09	2.79E-09	2.79E-09	2.79E-09	2.79E-09	2.79E-09	2.79E-09	2.79E-09	2.79E-09	2.79E-09	2.79E-09	2.79E-09	2.79E-09	2.8E-09	2.82E-09	2.83E-09
41.700	136.812	0.7962	2.66E-09	2.67E-09	2.68E-09	2.69E-09	2.69E-09	2.69E-09	2.69E-09	2.69E-09	2.69E-09	2.69E-09	2.69E-09	2.69E-09	2.69E-09	2.69E-09	2.69E-09	2.7E-09	2.7E-09	2.71E-09	2.72E-09	2.74E-09
42.507	139.457	0.8162	2.57E-09	2.58E-09	2.59E-09	2.6E-09	2.61E-09	2.61E-09	2.61E-09	2.61E-09	2.61E-09	2.61E-09	2.61E-09	2.61E-09	2.61E-09	2.61E-09	2.61E-09	2.61E-09	2.61E-09	2.62E-09	2.63E-09	2.65E-09
43.333	142.168	0.8362	2.5E-09	2.51E-09	2.52E-09	2.53E-09	2.53E-09	2.53E-09	2.53E-09	2.53E-09	2.53E-09	2.53E-09	2.53E-09	2.53E-09	2.53E-09	2.53E-09	2.53E-09	2.53E-09	2.54E-09	2.54E-09	2.56E-09	2.57E-09
44.179	144.944	0.8562	2.44E-09	2.44E-09	2.46E-09	2.46E-09	2.47E-09	2.47E-09	2.47E-09	2.47E-09	2.47E-09	2.47E-09	2.47E-09	2.47E-09	2.47E-09	2.47E-09	2.47E-09	2.47E-09	2.47E-09	2.48E-09	2.49E-09	2.51E-09
45.045	147.786	0.8762	2.39E-09	2.39E-09	2.41E-09	2.41E-09	2.42E-09	2.42E-09	2.42E-09	2.42E-09	2.42E-09	2.42E-09	2.42E-09	2.42E-09	2.42E-09	2.42E-09	2.42E-09	2.42E-09	2.42E-09	2.43E-09	2.44E-09	2.46E-09
45.931	150.693	0.8962	2.35E-09	2.36E-09	2.37E-09	2.38E-09	2.38E-09	2.38E-09	2.38E-09	2.38E-09	2.38E-09	2.38E-09	2.38E-09	2.38E-09	2.38E-09	2.38E-09	2.38E-09	2.38E-09	2.38E-09	2.39E-09	2.4E-09	2.42E-09
46.838	153.666	0.9162	2.32E-09	2.33E-09	2.34E-09	2.35E-09	2.35E-09	2.36E-09	2.36E-09	2.36E-09	2.36E-09	2.36E-09	2.36E-09	2.36E-09	2.36E-09	2.36E-09	2.36E-09	2.36E-09	2.36E-09	2.37E-09	2.38E-09	2.39E-09
47.616	156.221	0.64134	2.31E-09	2.32E-09	2.33E-09	2.34E-09	2.35E-09	2.35E-09	2.35E-09	2.35E-09	2.35E-09	2.35E-09	2.35E-09	2.35E-09	2.35E-09	2.35E-09	2.35E-09	2.35E-09	2.35E-09	2.36E-09	2.37E-09	2.38E-09

Coarse Tailings Oxygen Consur

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s)																		
			TIME (YEAR)																		
			78	81	84	87	90	93	96	99	102	105	108	111	114	117	120	123	126	129	132	135
0.001	0.002	0.001	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.002	0.007	0.002	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.007	0.023	0.008	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.019	0.062	0.016	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.043	0.141	0.032	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.091	0.299	0.064	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.181	0.593	0.1152	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.298	0.979	0.1202	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.421	1.381	0.1252	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.549	1.800	0.1302	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.681	2.236	0.1352	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.819	2.687	0.1402	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.962	3.156	0.1452	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
1.110	3.640	0.1502	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
1.262	4.141	0.1552	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
1.420	4.658	0.1602	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
1.583	5.192	0.1652	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
1.750	5.742	0.1702	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
1.923	6.309	0.1752	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
2.101	6.892	0.1802	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
2.283	7.491	0.1852	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
2.471	8.107	0.1902	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
2.664	8.740	0.1952	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
2.862	9.388	0.2002	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
3.064	10.053	0.2052	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
3.272	10.735	0.2102	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
3.485	11.432	0.2152	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
3.702	12.147	0.2202	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
3.925	12.877	0.2252	5.44E-08	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
4.153	13.624	0.2302	5.33E-08	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
4.385	14.388	0.2352	5.22E-08	5.28E-08	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
4.623	15.168	0.2402	5.12E-08	5.17E-08	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
4.866	15.964	0.2452	5.01E-08	5.06E-08	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
5.114	16.777	0.2502	4.9E-08	4.95E-08	5E-08	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
5.366	17.606	0.2552	4.8E-08	4.85E-08	4.89E-08	4.97E-08	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
5.624	18.451	0.2602	4.69E-08	4.74E-08	4.78E-08	4.86E-08	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
5.887	19.313	0.2652	4.59E-08	4.63E-08	4.67E-08	4.75E-08	4.81E-08	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
6.154	20.191	0.2702	4.48E-08	4.53E-08	4.57E-08	4.64E-08	4.7E-08	4.75E-08	0	0	0	0	0	0	0	0	0	0	0	0	0	0
6.427	21.086	0.2752	4.38E-08	4.42E-08	4.46E-08	4.53E-08	4.59E-08	4.64E-08	0	0	0	0	0	0	0	0	0	0	0	0	0	0
6.705	21.997	0.2802	4.27E-08	4.32E-08	4.36E-08	4.42E-08	4.48E-08	4.53E-08	4.59E-08	0	0	0	0	0	0	0	0	0	0	0	0	0
6.987	22.925	0.2852	4.17E-08	4.21E-08	4.25E-08	4.32E-08	4.37E-08	4.42E-08	4.48E-08	4.53E-08	0	0	0	0	0	0	0	0	0	0	0	0
7.275	23.868	0.2902	4.07E-08	4.11E-08	4.15E-08	4.21E-08	4.26E-08	4.31E-08	4.37E-08	4.42E-08	0	0	0	0	0	0	0	0	0	0	0	0
7.568	24.829	0.2952	3.97E-08	4.01E-08	4.04E-08	4.11E-08	4.16E-08	4.2E-08	4.26E-08	4.31E-08	4.36E-08	4.37E-08	0	0	0	0	0	0	0	0	0	0
7.866	25.805	0.3002	3.86E-08	3.9E-08	3.94E-08	4E-08	4.05E-08	4.09E-08	4.15E-08	4.2E-08	4.25E-08	4.26E-08	4.32E-08	0	0	0	0	0	0	0	0	0
8.168	26.799	0.3052	3.76E-08	3.8E-08	3.84E-08	3.9E-08	3.95E-08	3.99E-08	4.04E-08	4.09E-08	4.14E-08	4.15E-08	4.21E-08	0	0	0	0	0	0	0	0	0
8.476	27.808	0.3102	3.67E-08	3.7E-08	3.74E-08	3.79E-08	3.84E-08	3.88E-08	3.94E-08	3.98E-08	4.03E-08	4.04E-08	4.1E-08	4.14E-08	0	0	0	0	0	0	0	0
8.789	28.834	0.3152	3.57E-08	3.6E-08	3.64E-08	3.69E-08	3.74E-08	3.78E-08	3.83E-08	3.87E-08	3.93E-08	3.93E-08	3.99E-08	4.03E-08	4.1E-08	0	0	0	0	0	0	0
9.106	29.876	0.3202	3.47E-08	3.51E-08	3.54E-08	3.59E-08	3.64E-08	3.68E-08	3.73E-08	3.77E-08	3.82E-08	3.82E-08	3.88E-08	3.92E-08	3.99E-08	4.04E-08	0	0	0	0	0	0

Coarse Tailings Oxygen Consur

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s)																			
			TIME (YEAR)																			
			78	81	84	87	90	93	96	99	102	105	108	111	114	117	120	123	126	129	132	135
9.429	30.935	0.3252	3.38E-08	3.41E-08	3.44E-08	3.49E-08	3.54E-08	3.58E-08	3.62E-08	3.67E-08	3.71E-08	3.72E-08	3.77E-08	3.82E-08	3.88E-08	3.93E-08	3.98E-08	0	0	0	0	0
9.757	32.010	0.3302	3.28E-08	3.31E-08	3.35E-08	3.4E-08	3.44E-08	3.48E-08	3.52E-08	3.56E-08	3.61E-08	3.61E-08	3.67E-08	3.71E-08	3.77E-08	3.82E-08	3.87E-08	3.92E-08	0	0	0	0
10.089	33.102	0.3352	3.19E-08	3.22E-08	3.25E-08	3.3E-08	3.34E-08	3.38E-08	3.42E-08	3.46E-08	3.51E-08	3.51E-08	3.56E-08	3.6E-08	3.66E-08	3.71E-08	3.76E-08	3.81E-08	3.83E-08	0	0	0
10.427	34.210	0.3402	3.1E-08	3.13E-08	3.16E-08	3.2E-08	3.24E-08	3.28E-08	3.32E-08	3.36E-08	3.4E-08	3.41E-08	3.46E-08	3.5E-08	3.55E-08	3.6E-08	3.65E-08	3.7E-08	3.72E-08	3.8E-08	0	0
10.770	35.334	0.3452	3.01E-08	3.04E-08	3.06E-08	3.11E-08	3.15E-08	3.18E-08	3.23E-08	3.26E-08	3.3E-08	3.31E-08	3.36E-08	3.4E-08	3.45E-08	3.5E-08	3.54E-08	3.59E-08	3.61E-08	3.69E-08	3.75E-08	0
11.118	36.475	0.3502	2.92E-08	2.95E-08	2.97E-08	3.02E-08	3.06E-08	3.09E-08	3.13E-08	3.17E-08	3.21E-08	3.21E-08	3.26E-08	3.3E-08	3.35E-08	3.39E-08	3.44E-08	3.49E-08	3.51E-08	3.58E-08	3.64E-08	3.69E-08
11.470	37.632	0.3552	2.83E-08	2.86E-08	2.88E-08	2.93E-08	2.96E-08	3E-08	3.04E-08	3.07E-08	3.11E-08	3.11E-08	3.16E-08	3.2E-08	3.25E-08	3.29E-08	3.34E-08	3.38E-08	3.4E-08	3.47E-08	3.53E-08	3.58E-08
11.828	38.805	0.3602	2.74E-08	2.77E-08	2.8E-08	2.84E-08	2.87E-08	2.91E-08	2.94E-08	2.98E-08	3.01E-08	3.02E-08	3.06E-08	3.1E-08	3.15E-08	3.19E-08	3.23E-08	3.28E-08	3.3E-08	3.37E-08	3.42E-08	3.47E-08
12.191	39.995	0.3652	2.66E-08	2.69E-08	2.71E-08	2.75E-08	2.78E-08	2.82E-08	2.85E-08	2.89E-08	2.92E-08	2.92E-08	2.97E-08	3.01E-08	3.05E-08	3.09E-08	3.13E-08	3.18E-08	3.2E-08	3.26E-08	3.31E-08	3.36E-08
12.558	41.202	0.3702	2.58E-08	2.6E-08	2.63E-08	2.66E-08	2.7E-08	2.73E-08	2.76E-08	2.8E-08	2.83E-08	2.83E-08	2.88E-08	2.91E-08	2.95E-08	2.99E-08	3.04E-08	3.08E-08	3.1E-08	3.16E-08	3.21E-08	3.26E-08
12.931	42.425	0.3752	2.49E-08	2.52E-08	2.54E-08	2.58E-08	2.61E-08	2.64E-08	2.67E-08	2.71E-08	2.74E-08	2.74E-08	2.79E-08	2.82E-08	2.86E-08	2.9E-08	2.94E-08	2.98E-08	3E-08	3.06E-08	3.11E-08	3.15E-08
13.309	43.664	0.3802	2.41E-08	2.44E-08	2.46E-08	2.5E-08	2.53E-08	2.56E-08	2.59E-08	2.62E-08	2.65E-08	2.65E-08	2.7E-08	2.73E-08	2.77E-08	2.81E-08	2.84E-08	2.88E-08	2.9E-08	2.96E-08	3.01E-08	3.05E-08
13.691	44.919	0.3852	2.34E-08	2.36E-08	2.38E-08	2.42E-08	2.45E-08	2.47E-08	2.5E-08	2.54E-08	2.57E-08	2.56E-08	2.61E-08	2.64E-08	2.68E-08	2.71E-08	2.75E-08	2.79E-08	2.81E-08	2.87E-08	2.91E-08	2.95E-08
14.079	46.191	0.3902	2.26E-08	2.28E-08	2.3E-08	2.34E-08	2.36E-08	2.39E-08	2.42E-08	2.45E-08	2.48E-08	2.48E-08	2.52E-08	2.55E-08	2.59E-08	2.62E-08	2.66E-08	2.7E-08	2.71E-08	2.77E-08	2.81E-08	2.86E-08
14.472	47.480	0.3952	2.18E-08	2.21E-08	2.23E-08	2.26E-08	2.29E-08	2.31E-08	2.34E-08	2.37E-08	2.4E-08	2.4E-08	2.44E-08	2.47E-08	2.5E-08	2.54E-08	2.57E-08	2.61E-08	2.62E-08	2.68E-08	2.72E-08	2.76E-08
14.870	48.784	0.4002	2.11E-08	2.13E-08	2.15E-08	2.18E-08	2.21E-08	2.24E-08	2.26E-08	2.29E-08	2.32E-08	2.31E-08	2.36E-08	2.39E-08	2.42E-08	2.45E-08	2.48E-08	2.52E-08	2.53E-08	2.59E-08	2.63E-08	2.67E-08
15.272	50.106	0.4052	2.04E-08	2.06E-08	2.08E-08	2.11E-08	2.13E-08	2.16E-08	2.18E-08	2.21E-08	2.24E-08	2.24E-08	2.27E-08	2.3E-08	2.33E-08	2.37E-08	2.4E-08	2.43E-08	2.45E-08	2.5E-08	2.54E-08	2.58E-08
15.680	51.443	0.4102	1.97E-08	1.99E-08	2.01E-08	2.03E-08	2.06E-08	2.08E-08	2.11E-08	2.14E-08	2.16E-08	2.16E-08	2.2E-08	2.22E-08	2.25E-08	2.29E-08	2.32E-08	2.35E-08	2.36E-08	2.41E-08	2.45E-08	2.49E-08
16.093	52.797	0.4152	1.9E-08	1.92E-08	1.94E-08	1.96E-08	1.99E-08	2.01E-08	2.04E-08	2.06E-08	2.09E-08	2.08E-08	2.12E-08	2.15E-08	2.18E-08	2.21E-08	2.24E-08	2.27E-08	2.28E-08	2.33E-08	2.37E-08	2.4E-08
16.510	54.168	0.4202	1.83E-08	1.85E-08	1.87E-08	1.89E-08	1.92E-08	1.94E-08	1.96E-08	1.99E-08	2.01E-08	2.01E-08	2.04E-08	2.07E-08	2.1E-08	2.13E-08	2.16E-08	2.19E-08	2.2E-08	2.25E-08	2.28E-08	2.31E-08
16.933	55.554	0.4252	1.77E-08	1.78E-08	1.8E-08	1.83E-08	1.85E-08	1.87E-08	1.89E-08	1.92E-08	1.94E-08	1.94E-08	1.97E-08	2E-08	2.02E-08	2.05E-08	2.08E-08	2.11E-08	2.12E-08	2.17E-08	2.2E-08	2.23E-08
17.361	56.958	0.4302	1.7E-08	1.72E-08	1.74E-08	1.76E-08	1.78E-08	1.8E-08	1.82E-08	1.85E-08	1.87E-08	1.87E-08	1.9E-08	1.93E-08	1.95E-08	1.98E-08	2E-08	2.03E-08	2.04E-08	2.09E-08	2.12E-08	2.15E-08
17.793	58.377	0.4352	1.64E-08	1.66E-08	1.67E-08	1.7E-08	1.72E-08	1.74E-08	1.76E-08	1.78E-08	1.8E-08	1.8E-08	1.83E-08	1.85E-08	1.88E-08	1.9E-08	1.93E-08	1.96E-08	1.97E-08	2.01E-08	2.04E-08	2.07E-08
18.231	59.813	0.4402	1.58E-08	1.6E-08	1.61E-08	1.63E-08	1.65E-08	1.67E-08	1.69E-08	1.71E-08	1.74E-08	1.73E-08	1.76E-08	1.79E-08	1.81E-08	1.83E-08	1.86E-08	1.88E-08	1.9E-08	1.94E-08	1.97E-08	2E-08
18.674	61.266	0.4452	1.52E-08	1.54E-08	1.55E-08	1.57E-08	1.59E-08	1.61E-08	1.63E-08	1.65E-08	1.67E-08	1.67E-08	1.7E-08	1.72E-08	1.74E-08	1.77E-08	1.79E-08	1.81E-08	1.83E-08	1.86E-08	1.89E-08	1.92E-08
19.122	62.735	0.4502	1.46E-08	1.48E-08	1.49E-08	1.51E-08	1.53E-08	1.55E-08	1.57E-08	1.59E-08	1.61E-08	1.6E-08	1.63E-08	1.65E-08	1.68E-08	1.7E-08	1.72E-08	1.75E-08	1.76E-08	1.79E-08	1.82E-08	1.85E-08
19.574	64.220	0.4552	1.41E-08	1.42E-08	1.43E-08	1.46E-08	1.47E-08	1.49E-08	1.51E-08	1.53E-08	1.55E-08	1.54E-08	1.57E-08	1.59E-08	1.61E-08	1.63E-08	1.66E-08	1.68E-08	1.69E-08	1.72E-08	1.75E-08	1.78E-08
20.032	65.721	0.4602	1.36E-08	1.37E-08	1.38E-08	1.4E-08	1.42E-08	1.43E-08	1.45E-08	1.47E-08	1.49E-08	1.48E-08	1.51E-08	1.53E-08	1.55E-08	1.57E-08	1.59E-08	1.61E-08	1.62E-08	1.66E-08	1.69E-08	1.71E-08
20.495	67.240	0.4652	1.3E-08	1.32E-08	1.33E-08	1.34E-08	1.36E-08	1.38E-08	1.39E-08	1.41E-08	1.43E-08	1.43E-08	1.45E-08	1.47E-08	1.49E-08	1.51E-08	1.53E-08	1.55E-08	1.56E-08	1.59E-08	1.62E-08	1.64E-08
20.962	68.774	0.4702	1.25E-08	1.26E-08	1.27E-08	1.29E-08	1.31E-08	1.32E-08	1.34E-08	1.36E-08	1.37E-08	1.37E-08	1.39E-08	1.41E-08	1.43E-08	1.45E-08	1.47E-08	1.49E-08	1.5E-08	1.53E-08	1.56E-08	1.58E-08
21.435	70.325	0.4752	1.2E-08	1.21E-08	1.22E-08	1.24E-08	1.26E-08	1.27E-08	1.29E-08	1.3E-08	1.32E-08	1.31E-08	1.34E-08	1.36E-08	1.38E-08	1.39E-08	1.41E-08	1.43E-08	1.44E-08	1.47E-08	1.49E-08	1.52E-08
21.913	71.892	0.4802	1.15E-08	1.17E-08	1.18E-08	1.19E-08	1.21E-08	1.22E-08	1.24E-08	1.25E-08	1.27E-08	1.26E-08	1.29E-08	1.3E-08	1.32E-08	1.34E-08	1.36E-08	1.37E-08	1.38E-08	1.41E-08	1.44E-08	1.46E-08
22.395	73.476	0.4852	1.11E-08	1.12E-08	1.13E-08	1.14E-08	1.16E-08	1.17E-08	1.19E-08	1.2E-08	1.22E-08	1.21E-08	1.23E-08	1.25E-08	1.27E-08	1.28E-08	1.3E-08	1.32E-08	1.33E-08	1.35E-08	1.38E-08	1.4E-08
22.883	75.076	0.4902	1.06E-08	1.07E-08	1.08E-08	1.1E-08	1.11E-08	1.13E-08	1.14E-08	1.15E-08	1.17E-08	1.16E-08	1.18E-08	1.2E-08	1.22E-08	1.23E-08	1.25E-08	1.27E-08	1.27E-08	1.3E-08	1.32E-08	1.34E-08
23.376	76.692	0.4952	1.02E-08	1.03E-08	1.04E-08	1.05E-08	1.07E-08	1.08E-08	1.09E-08	1.11E-08	1.12E-08	1.11E-08	1.13E-08	1.15E-08	1.17E-08	1.18E-08	1.2E-08	1.21E-08	1.22E-08	1.25E-08	1.27E-08	1.29E-08
23.875	78.328	0.5022	9.79E-09	9.88E-09	9.95E-09	1.01E-08	1.02E-08	1.03E-08	1.05E-08	1.06E-08	1.07E-08	1.07E-08	1.09E-08	1.1E-08	1.12E-08	1.13E-08	1.15E-08	1.16E-08	1.17E-08	1.19E-08	1.21E-08	1.23E-08
24.380	79.988	0.5092	9.38E-09	9.47E-09	9.54E-09	9.67E-09	9.8E-09	9.92E-09	1E-08	1.02E-08	1.03E-08	1.02E-08	1.04E-08	1.06E-08	1.07E-08	1.09E-08	1.1E-08	1.12E-08	1.12E-08	1.14E-08	1.16E-08	1.18E-08
24.893	81.670	0.5162	8.99E-09	9.07E-09	9.14E-09	9.26E-09	9.38E-09	9.5E-09	9.61E-09	9.73E-09	9.85E-09	9.81E-09	9.98E-09	1.01E-08	1.03E-08	1.04E-08	1.05E-08	1.07E-08	1.08E-08	1.1E-08	1.12E-08	1.13E-08
25.413	83.375	0.5232	8.61E-09	8.68E-09	8.75E-09	8.87E-09	8.98E-09	9.1E-09	9.2E-09	9.32E-09	9.43E-09	9.39E-09	9.56E-09	9.7E-09	9.83E-09	9.96E-09	1.01E-08	1.02E-08	1.03E-08	1.05E-08	1.07E-08	1.08E-08
25.939	85.103	0.5302	8.24E-09	8.31E-09	8.37E-09	8.49E-09	8.6E-09	8.7E-09	8.81E-09	8.92E-09	9.03E-09	8.99E-09	9.15E-09	9.28E-09	9.41E-09	9.53E-09	9.66E-09	9.79E-09	9.86E-09	1E-08	1.02E-08	1.04E-08
26.473	86.854	0.5372	7.88E-09	7.95E-09	8.01E-09	8.12E-09	8.23E-09	8.33E-09	8.43E-09	8.53E-09	8.64E-09	8.6E-09	8.75E-09	8.88E-09	9E-09	9.12E-09	9.24E-09	9.37E-09	9.43E-09	9.6E-09	9.77E-09	9.92E-09
27.014	88.628	0.5442	7.54E-09	7.61E-09	7.66E-09	7.76E-09	7.87E-09	7.96E-09	8.06E-09	8.16E-09	8.26E-09	8.22E-09	8.36E-09	8.49E-09	8.61E-09	8.72E-09	8.84E-09	8.96E-09	9.02E-09	9.18E-09	9.34E-09	9.48E-09
27.561	90.425	0.5512	7.21E-09	7.27E-09	7.32E-09	7.42E-09	7.52E-09	7.61E-09	7.71E-09	7.8E-09	7.9E-09	7.86E-09	8E-09	8.12E-09	8.23E-09	8.34E-09	8.45E-09	8.56E-09	8.62E-09	8.78E-09	8.93E-09	9.06E-09
28.116	92.244	0.5582	6.89E-09	6.95E-09	7E-09	7.09E-09	7.19E-09	7.28E-09	7.36E-09	7.45E-09	7.55E-09	7.51E-09	7.64E-09	7.76E-09	7.87E-09	7.97E-09	8.07E-09	8.18E-09	8.24E-09	8.38E-09	8.53E-09	8.66E-09
28.678	94.087	0.5652	6.58E-09	6.64E-09	6.69E-09	6.78E-09	6.87E-09	6.95E-09	7.04E-09	7.12E-09	7.21E-09	7.18E-09	7.3E-09	7.41E-09	7.51E-09	7.61E-09	7.71E-09	7.82E-09	7.87E-09	8.01E-09	8.15E-09	8.27E-09
29.247	95.953	0.5722	6.29E-09	6.34E-09	6.39E-09	6.47E-09	6.56E-09	6.64E-09	6.72E-09	6.8E-09	6.89E-09	6.85E-09	6.97E-09	7.08E-09	7.18E-09	7.27E-09	7.37E-09	7.47E-09	7.52E-09	7.65E-09	7.78E-09	7.9E-09
29.822	97.842	0.5792	6.01E-09	6.06E-09	6.1E-09	6.18E-09	6.26E-09	6.34E-09	6.42E-09	6.5E-09	6.58E-09	6.54E-09	6.65E-09	6.76E-09	6.85E-09	6.94E-09	7.04E-09	7.13E-09	7.18E-09	7.3E-09	7.43E-09	7.54E-09
30.405	99.754	0.5862	5.74E-09	5.79E-09	5.82E-09	5.9E-09	5.98E-09	6.05E-09	6.13E-09	6.2E-09	6.28E-09	6.25E-09	6.35E-09	6.45E-09	6.54E-09	6.63E-09	6.72E-09	6.81E-09	6.85E-09	6.97E-09	7.09E-09	7.2E-09
30.995	101.688	0.5932	5.48E-09	5.53E-09	5.56E-09	5.63E-09	5.71E-09	5.78E-09	5.85E-09	5.92E-09	6E-09	5.96E-09	6.06E-09	6.16E-09	6.25E-09	6.33E-09	6.41E-09	6.5E-09	6.54E-09	6.65E-09	6.77E-09	6.88E-09
31.591	103.646	0.6002	5.23E-09	5.28E-09	5.31E-09	5.38E-09	5.45E-09	5.52E-09	5.59E-09	5.65E-09	5.72E-09	5.69E-09	5.79E-09	5.88E-09	5.96E-09	6.04E-09	6.12E-09	6.2E-09	6.24E-09	6.35E-09	6.46E-09	6.56E-09
32.195	105.627	0.6072	4.99E-09	5.04E-09	5.07E-09	5.13E-09	5.2E-09	5.27E-09	5.33E-09	5.4E-09	5.46E-09	5.44E-09	5.52E-09	5.61E-09	5.69E-09	5.77E-09	5.85E-09	5.92E-09	5.96E-09	6.06E-09	6.17E-09	6.27E-09

Coarse Tailings Oxygen Consur

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s)																			
			TIME (YEAR)																			
			78	81	84	87	90	93	96	99	102	105	108	111	114	117	120	123	126	129	132	135
32.806	107.630	0.6142	4.77E-09	4.81E-09	4.84E-09	4.9E-09	4.97E-09	5.03E-09	5.09E-09	5.15E-09	5.22E-09	5.19E-09	5.27E-09	5.36E-09	5.43E-09	5.51E-09	5.58E-09	5.66E-09	5.69E-09	5.79E-09	5.89E-09	5.98E-09
33.423	109.657	0.6212	4.55E-09	4.59E-09	4.62E-09	4.68E-09	4.74E-09	4.8E-09	4.86E-09	4.92E-09	4.98E-09	4.96E-09	5.04E-09	5.12E-09	5.19E-09	5.26E-09	5.33E-09	5.4E-09	5.43E-09	5.52E-09	5.62E-09	5.71E-09
34.048	111.706	0.6282	4.35E-09	4.39E-09	4.41E-09	4.47E-09	4.53E-09	4.59E-09	4.64E-09	4.7E-09	4.76E-09	4.73E-09	4.81E-09	4.89E-09	4.96E-09	5.02E-09	5.09E-09	5.16E-09	5.19E-09	5.28E-09	5.37E-09	5.45E-09
34.680	113.779	0.6352	4.16E-09	4.2E-09	4.22E-09	4.27E-09	4.33E-09	4.38E-09	4.44E-09	4.49E-09	4.55E-09	4.52E-09	4.6E-09	4.67E-09	4.74E-09	4.8E-09	4.86E-09	4.93E-09	4.96E-09	5.04E-09	5.13E-09	5.21E-09
35.319	115.874	0.6422	3.98E-09	4.01E-09	4.03E-09	4.08E-09	4.14E-09	4.19E-09	4.24E-09	4.3E-09	4.35E-09	4.32E-09	4.39E-09	4.46E-09	4.53E-09	4.59E-09	4.65E-09	4.71E-09	4.74E-09	4.82E-09	4.9E-09	4.98E-09
35.964	117.993	0.6492	3.81E-09	3.84E-09	3.86E-09	3.91E-09	3.96E-09	4.01E-09	4.06E-09	4.11E-09	4.16E-09	4.14E-09	4.2E-09	4.27E-09	4.33E-09	4.39E-09	4.45E-09	4.51E-09	4.54E-09	4.61E-09	4.69E-09	4.77E-09
36.617	120.134	0.6562	3.65E-09	3.68E-09	3.7E-09	3.74E-09	3.79E-09	3.84E-09	3.89E-09	3.93E-09	3.98E-09	3.96E-09	4.02E-09	4.09E-09	4.15E-09	4.2E-09	4.26E-09	4.32E-09	4.34E-09	4.41E-09	4.49E-09	4.56E-09
37.283	122.320	0.6762	3.49E-09	3.52E-09	3.54E-09	3.58E-09	3.63E-09	3.68E-09	3.72E-09	3.77E-09	3.82E-09	3.79E-09	3.85E-09	3.92E-09	3.97E-09	4.03E-09	4.08E-09	4.14E-09	4.16E-09	4.23E-09	4.3E-09	4.37E-09
37.969	124.571	0.6962	3.35E-09	3.38E-09	3.39E-09	3.43E-09	3.48E-09	3.53E-09	3.57E-09	3.61E-09	3.66E-09	3.64E-09	3.69E-09	3.75E-09	3.81E-09	3.86E-09	3.91E-09	3.96E-09	3.99E-09	4.05E-09	4.12E-09	4.19E-09
38.676	126.888	0.7162	3.21E-09	3.24E-09	3.25E-09	3.29E-09	3.34E-09	3.38E-09	3.42E-09	3.46E-09	3.51E-09	3.49E-09	3.54E-09	3.6E-09	3.65E-09	3.7E-09	3.75E-09	3.8E-09	3.82E-09	3.88E-09	3.95E-09	4.01E-09
39.402	129.271	0.7362	3.08E-09	3.11E-09	3.12E-09	3.16E-09	3.2E-09	3.24E-09	3.29E-09	3.33E-09	3.37E-09	3.35E-09	3.4E-09	3.45E-09	3.5E-09	3.55E-09	3.6E-09	3.65E-09	3.67E-09	3.72E-09	3.79E-09	3.85E-09
40.148	131.719	0.7562	2.96E-09	2.99E-09	3E-09	3.04E-09	3.08E-09	3.12E-09	3.16E-09	3.2E-09	3.24E-09	3.22E-09	3.26E-09	3.32E-09	3.37E-09	3.41E-09	3.46E-09	3.51E-09	3.53E-09	3.58E-09	3.64E-09	3.7E-09
40.914	134.233	0.7762	2.85E-09	2.88E-09	2.89E-09	2.93E-09	2.97E-09	3E-09	3.04E-09	3.08E-09	3.12E-09	3.1E-09	3.14E-09	3.19E-09	3.24E-09	3.29E-09	3.33E-09	3.38E-09	3.4E-09	3.45E-09	3.51E-09	3.57E-09
41.700	136.812	0.7962	2.76E-09	2.78E-09	2.79E-09	2.82E-09	2.86E-09	2.9E-09	2.93E-09	2.97E-09	3.01E-09	2.99E-09	3.03E-09	3.08E-09	3.13E-09	3.17E-09	3.22E-09	3.26E-09	3.28E-09	3.33E-09	3.38E-09	3.44E-09
42.507	139.457	0.8162	2.67E-09	2.69E-09	2.7E-09	2.73E-09	2.77E-09	2.81E-09	2.84E-09	2.87E-09	2.91E-09	2.89E-09	2.94E-09	2.98E-09	3.03E-09	3.07E-09	3.11E-09	3.15E-09	3.17E-09	3.22E-09	3.27E-09	3.33E-09
43.333	142.168	0.8362	2.59E-09	2.61E-09	2.62E-09	2.65E-09	2.69E-09	2.72E-09	2.76E-09	2.79E-09	2.83E-09	2.81E-09	2.85E-09	2.9E-09	2.94E-09	2.98E-09	3.02E-09	3.06E-09	3.08E-09	3.12E-09	3.18E-09	3.23E-09
44.179	144.944	0.8562	2.53E-09	2.55E-09	2.56E-09	2.59E-09	2.62E-09	2.65E-09	2.69E-09	2.72E-09	2.75E-09	2.74E-09	2.78E-09	2.82E-09	2.87E-09	2.91E-09	2.94E-09	2.98E-09	3E-09	3.04E-09	3.1E-09	3.15E-09
45.045	147.786	0.8762	2.47E-09	2.49E-09	2.5E-09	2.53E-09	2.57E-09	2.6E-09	2.63E-09	2.66E-09	2.7E-09	2.68E-09	2.72E-09	2.76E-09	2.8E-09	2.84E-09	2.88E-09	2.92E-09	2.94E-09	2.98E-09	3.03E-09	3.08E-09
45.931	150.693	0.8962	2.43E-09	2.45E-09	2.46E-09	2.49E-09	2.52E-09	2.56E-09	2.59E-09	2.62E-09	2.65E-09	2.64E-09	2.68E-09	2.72E-09	2.76E-09	2.8E-09	2.84E-09	2.87E-09	2.89E-09	2.93E-09	2.98E-09	3.03E-09
46.838	153.666	0.9162	2.41E-09	2.43E-09	2.44E-09	2.47E-09	2.5E-09	2.53E-09	2.56E-09	2.59E-09	2.63E-09	2.61E-09	2.65E-09	2.69E-09	2.73E-09	2.77E-09	2.81E-09	2.84E-09	2.86E-09	2.9E-09	2.95E-09	3E-09
47.616	156.221	0.64134	2.4E-09	2.42E-09	2.43E-09	2.46E-09	2.49E-09	2.52E-09	2.55E-09	2.58E-09	2.62E-09	2.6E-09	2.64E-09	2.68E-09	2.72E-09	2.76E-09	2.8E-09	2.83E-09	2.85E-09	2.89E-09	2.94E-09	2.99E-09

Coarse Tailings Oxygen Consur

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s) Time (years)									
			138	141	144	147	150	153	156	159	162	165
0.001	0.002	0.001	0	0	0	0	0	0	0	0	0	0
0.002	0.007	0.002	0	0	0	0	0	0	0	0	0	0
0.007	0.023	0.008	0	0	0	0	0	0	0	0	0	0
0.019	0.062	0.016	0	0	0	0	0	0	0	0	0	0
0.043	0.141	0.032	0	0	0	0	0	0	0	0	0	0
0.091	0.299	0.064	0	0	0	0	0	0	0	0	0	0
0.181	0.593	0.1152	0	0	0	0	0	0	0	0	0	0
0.298	0.979	0.1202	0	0	0	0	0	0	0	0	0	0
0.421	1.381	0.1252	0	0	0	0	0	0	0	0	0	0
0.549	1.800	0.1302	0	0	0	0	0	0	0	0	0	0
0.681	2.236	0.1352	0	0	0	0	0	0	0	0	0	0
0.819	2.687	0.1402	0	0	0	0	0	0	0	0	0	0
0.962	3.156	0.1452	0	0	0	0	0	0	0	0	0	0
1.110	3.640	0.1502	0	0	0	0	0	0	0	0	0	0
1.262	4.141	0.1552	0	0	0	0	0	0	0	0	0	0
1.420	4.658	0.1602	0	0	0	0	0	0	0	0	0	0
1.583	5.192	0.1652	0	0	0	0	0	0	0	0	0	0
1.750	5.742	0.1702	0	0	0	0	0	0	0	0	0	0
1.923	6.309	0.1752	0	0	0	0	0	0	0	0	0	0
2.101	6.892	0.1802	0	0	0	0	0	0	0	0	0	0
2.283	7.491	0.1852	0	0	0	0	0	0	0	0	0	0
2.471	8.107	0.1902	0	0	0	0	0	0	0	0	0	0
2.664	8.740	0.1952	0	0	0	0	0	0	0	0	0	0
2.862	9.388	0.2002	0	0	0	0	0	0	0	0	0	0
3.064	10.053	0.2052	0	0	0	0	0	0	0	0	0	0
3.272	10.735	0.2102	0	0	0	0	0	0	0	0	0	0
3.485	11.432	0.2152	0	0	0	0	0	0	0	0	0	0
3.702	12.147	0.2202	0	0	0	0	0	0	0	0	0	0
3.925	12.877	0.2252	0	0	0	0	0	0	0	0	0	0
4.153	13.624	0.2302	0	0	0	0	0	0	0	0	0	0
4.385	14.388	0.2352	0	0	0	0	0	0	0	0	0	0
4.623	15.168	0.2402	0	0	0	0	0	0	0	0	0	0
4.866	15.964	0.2452	0	0	0	0	0	0	0	0	0	0
5.114	16.777	0.2502	0	0	0	0	0	0	0	0	0	0
5.366	17.606	0.2552	0	0	0	0	0	0	0	0	0	0
5.624	18.451	0.2602	0	0	0	0	0	0	0	0	0	0
5.887	19.313	0.2652	0	0	0	0	0	0	0	0	0	0
6.154	20.191	0.2702	0	0	0	0	0	0	0	0	0	0
6.427	21.086	0.2752	0	0	0	0	0	0	0	0	0	0
6.705	21.997	0.2802	0	0	0	0	0	0	0	0	0	0
6.987	22.925	0.2852	0	0	0	0	0	0	0	0	0	0
7.275	23.868	0.2902	0	0	0	0	0	0	0	0	0	0
7.568	24.829	0.2952	0	0	0	0	0	0	0	0	0	0
7.866	25.805	0.3002	0	0	0	0	0	0	0	0	0	0
8.168	26.799	0.3052	0	0	0	0	0	0	0	0	0	0
8.476	27.808	0.3102	0	0	0	0	0	0	0	0	0	0
8.789	28.834	0.3152	0	0	0	0	0	0	0	0	0	0
9.106	29.876	0.3202	0	0	0	0	0	0	0	0	0	0

Coarse Tailings Oxygen Consur

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s) Time (years)									
			138	141	144	147	150	153	156	159	162	165
9.429	30.935	0.3252	0	0	0	0	0	0	0	0	0	0
9.757	32.010	0.3302	0	0	0	0	0	0	0	0	0	0
10.089	33.102	0.3352	0	0	0	0	0	0	0	0	0	0
10.427	34.210	0.3402	0	0	0	0	0	0	0	0	0	0
10.770	35.334	0.3452	0	0	0	0	0	0	0	0	0	0
11.118	36.475	0.3502	0	0	0	0	0	0	0	0	0	0
11.470	37.632	0.3552	3.63E-08	0	0	0	0	0	0	0	0	0
11.828	38.805	0.3602	3.52E-08	3.57E-08	0	0	0	0	0	0	0	0
12.191	39.995	0.3652	3.41E-08	3.46E-08	3.5E-08	3.48E-08	0	0	0	0	0	0
12.558	41.202	0.3702	3.3E-08	3.35E-08	3.39E-08	3.37E-08	3.45E-08	0	0	0	0	0
12.931	42.425	0.3752	3.2E-08	3.24E-08	3.29E-08	3.26E-08	3.34E-08	3.38E-08	3.42E-08	0	0	0
13.309	43.664	0.3802	3.09E-08	3.14E-08	3.18E-08	3.16E-08	3.23E-08	3.28E-08	3.31E-08	3.36E-08	0	0
13.691	44.919	0.3852	2.99E-08	3.04E-08	3.08E-08	3.05E-08	3.12E-08	3.17E-08	3.2E-08	3.25E-08	3.3E-08	0
14.079	46.191	0.3902	2.9E-08	2.94E-08	2.98E-08	2.95E-08	3.02E-08	3.06E-08	3.1E-08	3.15E-08	3.19E-08	3.24E-08
14.472	47.480	0.3952	2.8E-08	2.84E-08	2.88E-08	2.85E-08	2.92E-08	2.96E-08	2.99E-08	3.04E-08	3.09E-08	3.13E-08
14.870	48.784	0.4002	2.7E-08	2.74E-08	2.78E-08	2.76E-08	2.82E-08	2.86E-08	2.89E-08	2.94E-08	2.98E-08	3.02E-08
15.272	50.106	0.4052	2.61E-08	2.65E-08	2.68E-08	2.66E-08	2.72E-08	2.76E-08	2.79E-08	2.84E-08	2.88E-08	2.92E-08
15.680	51.443	0.4102	2.52E-08	2.56E-08	2.59E-08	2.57E-08	2.63E-08	2.67E-08	2.7E-08	2.74E-08	2.78E-08	2.82E-08
16.093	52.797	0.4152	2.43E-08	2.47E-08	2.5E-08	2.48E-08	2.54E-08	2.57E-08	2.6E-08	2.65E-08	2.68E-08	2.72E-08
16.510	54.168	0.4202	2.35E-08	2.38E-08	2.41E-08	2.39E-08	2.44E-08	2.48E-08	2.51E-08	2.55E-08	2.59E-08	2.62E-08
16.933	55.554	0.4252	2.26E-08	2.29E-08	2.32E-08	2.31E-08	2.36E-08	2.39E-08	2.42E-08	2.46E-08	2.5E-08	2.53E-08
17.361	56.958	0.4302	2.18E-08	2.21E-08	2.24E-08	2.22E-08	2.27E-08	2.3E-08	2.34E-08	2.37E-08	2.41E-08	2.44E-08
17.793	58.377	0.4352	2.1E-08	2.13E-08	2.16E-08	2.14E-08	2.19E-08	2.22E-08	2.25E-08	2.29E-08	2.32E-08	2.35E-08
18.231	59.813	0.4402	2.02E-08	2.05E-08	2.08E-08	2.06E-08	2.11E-08	2.14E-08	2.17E-08	2.2E-08	2.23E-08	2.26E-08
18.674	61.266	0.4452	1.95E-08	1.97E-08	2E-08	1.99E-08	2.03E-08	2.06E-08	2.09E-08	2.12E-08	2.15E-08	2.18E-08
19.122	62.735	0.4502	1.87E-08	1.9E-08	1.92E-08	1.91E-08	1.95E-08	1.98E-08	2.01E-08	2.04E-08	2.07E-08	2.09E-08
19.574	64.220	0.4552	1.8E-08	1.83E-08	1.85E-08	1.84E-08	1.88E-08	1.9E-08	1.93E-08	1.96E-08	1.99E-08	2.01E-08
20.032	65.721	0.4602	1.73E-08	1.76E-08	1.78E-08	1.77E-08	1.8E-08	1.83E-08	1.86E-08	1.89E-08	1.91E-08	1.94E-08
20.495	67.240	0.4652	1.67E-08	1.69E-08	1.71E-08	1.7E-08	1.73E-08	1.76E-08	1.79E-08	1.81E-08	1.84E-08	1.86E-08
20.962	68.774	0.4702	1.6E-08	1.62E-08	1.64E-08	1.63E-08	1.66E-08	1.69E-08	1.72E-08	1.74E-08	1.77E-08	1.79E-08
21.435	70.325	0.4752	1.54E-08	1.56E-08	1.58E-08	1.57E-08	1.6E-08	1.62E-08	1.65E-08	1.67E-08	1.7E-08	1.72E-08
21.913	71.892	0.4802	1.48E-08	1.5E-08	1.52E-08	1.5E-08	1.53E-08	1.56E-08	1.58E-08	1.61E-08	1.63E-08	1.65E-08
22.395	73.476	0.4852	1.42E-08	1.44E-08	1.45E-08	1.44E-08	1.47E-08	1.5E-08	1.52E-08	1.54E-08	1.56E-08	1.58E-08
22.883	75.076	0.4902	1.36E-08	1.38E-08	1.4E-08	1.38E-08	1.41E-08	1.44E-08	1.46E-08	1.48E-08	1.5E-08	1.52E-08
23.376	76.692	0.4952	1.3E-08	1.32E-08	1.34E-08	1.33E-08	1.35E-08	1.38E-08	1.4E-08	1.42E-08	1.44E-08	1.46E-08
23.875	78.328	0.5022	1.25E-08	1.27E-08	1.28E-08	1.27E-08	1.3E-08	1.32E-08	1.34E-08	1.36E-08	1.38E-08	1.4E-08
24.380	79.988	0.5092	1.2E-08	1.21E-08	1.23E-08	1.22E-08	1.24E-08	1.26E-08	1.28E-08	1.3E-08	1.32E-08	1.34E-08
24.893	81.670	0.5162	1.15E-08	1.16E-08	1.18E-08	1.17E-08	1.19E-08	1.21E-08	1.23E-08	1.25E-08	1.27E-08	1.28E-08
25.413	83.375	0.5232	1.1E-08	1.11E-08	1.13E-08	1.12E-08	1.14E-08	1.16E-08	1.18E-08	1.2E-08	1.21E-08	1.23E-08
25.939	85.103	0.5302	1.05E-08	1.07E-08	1.08E-08	1.07E-08	1.09E-08	1.11E-08	1.13E-08	1.14E-08	1.16E-08	1.17E-08
26.473	86.854	0.5372	1.01E-08	1.02E-08	1.03E-08	1.02E-08	1.04E-08	1.06E-08	1.08E-08	1.09E-08	1.11E-08	1.12E-08
27.014	88.628	0.5442	9.61E-09	9.75E-09	9.87E-09	9.79E-09	9.98E-09	1.02E-08	1.03E-08	1.05E-08	1.06E-08	1.07E-08
27.561	90.425	0.5512	9.19E-09	9.32E-09	9.44E-09	9.36E-09	9.53E-09	9.7E-09	9.86E-09	1E-08	1.01E-08	1.03E-08
28.116	92.244	0.5582	8.78E-09	8.9E-09	9.02E-09	8.94E-09	9.11E-09	9.27E-09	9.42E-09	9.56E-09	9.69E-09	9.81E-09
28.678	94.087	0.5652	8.39E-09	8.5E-09	8.62E-09	8.54E-09	8.7E-09	8.86E-09	9E-09	9.13E-09	9.26E-09	9.38E-09
29.247	95.953	0.5722	8.01E-09	8.12E-09	8.23E-09	8.16E-09	8.31E-09	8.46E-09	8.59E-09	8.72E-09	8.84E-09	8.95E-09
29.822	97.842	0.5792	7.65E-09	7.76E-09	7.86E-09	7.79E-09	7.93E-09	8.08E-09	8.21E-09	8.33E-09	8.44E-09	8.55E-09
30.405	99.754	0.5862	7.3E-09	7.41E-09	7.51E-09	7.44E-09	7.57E-09	7.71E-09	7.83E-09	7.95E-09	8.06E-09	8.16E-09
30.995	101.688	0.5932	6.97E-09	7.07E-09	7.17E-09	7.1E-09	7.23E-09	7.36E-09	7.48E-09	7.59E-09	7.69E-09	7.79E-09
31.591	103.646	0.6002	6.66E-09	6.75E-09	6.84E-09	6.78E-09	6.9E-09	7.03E-09	7.14E-09	7.25E-09	7.34E-09	7.44E-09
32.195	105.627	0.6072	6.35E-09	6.44E-09	6.53E-09	6.47E-09	6.58E-09	6.71E-09	6.81E-09	6.92E-09	7.01E-09	7.1E-09

Coarse Tailings Oxygen Consur

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s) Time (years)									
			138	141	144	147	150	153	156	159	162	165
32.806	107.630	0.6142	6.07E-09	6.15E-09	6.24E-09	6.17E-09	6.28E-09	6.4E-09	6.51E-09	6.6E-09	6.69E-09	6.78E-09
33.423	109.657	0.6212	5.79E-09	5.87E-09	5.95E-09	5.9E-09	6E-09	6.11E-09	6.21E-09	6.3E-09	6.39E-09	6.48E-09
34.048	111.706	0.6282	5.53E-09	5.61E-09	5.69E-09	5.63E-09	5.73E-09	5.84E-09	5.93E-09	6.02E-09	6.1E-09	6.19E-09
34.680	113.779	0.6352	5.29E-09	5.36E-09	5.44E-09	5.38E-09	5.47E-09	5.58E-09	5.67E-09	5.75E-09	5.83E-09	5.91E-09
35.319	115.874	0.6422	5.05E-09	5.13E-09	5.2E-09	5.14E-09	5.23E-09	5.33E-09	5.42E-09	5.5E-09	5.58E-09	5.65E-09
35.964	117.993	0.6492	4.84E-09	4.9E-09	4.97E-09	4.92E-09	5.01E-09	5.1E-09	5.18E-09	5.26E-09	5.33E-09	5.41E-09
36.617	120.134	0.6562	4.63E-09	4.69E-09	4.76E-09	4.71E-09	4.79E-09	4.88E-09	4.96E-09	5.04E-09	5.11E-09	5.18E-09
37.283	122.320	0.6762	4.43E-09	4.5E-09	4.56E-09	4.51E-09	4.59E-09	4.68E-09	4.75E-09	4.83E-09	4.89E-09	4.96E-09
37.969	124.571	0.6962	4.25E-09	4.31E-09	4.37E-09	4.32E-09	4.4E-09	4.48E-09	4.55E-09	4.62E-09	4.69E-09	4.75E-09
38.676	126.888	0.7162	4.07E-09	4.13E-09	4.19E-09	4.14E-09	4.21E-09	4.29E-09	4.37E-09	4.43E-09	4.5E-09	4.56E-09
39.402	129.271	0.7362	3.91E-09	3.97E-09	4.02E-09	3.98E-09	4.04E-09	4.12E-09	4.19E-09	4.25E-09	4.31E-09	4.38E-09
40.148	131.719	0.7562	3.76E-09	3.81E-09	3.87E-09	3.82E-09	3.89E-09	3.96E-09	4.03E-09	4.09E-09	4.15E-09	4.21E-09
40.914	134.233	0.7762	3.62E-09	3.67E-09	3.72E-09	3.68E-09	3.74E-09	3.81E-09	3.88E-09	3.94E-09	3.99E-09	4.05E-09
41.700	136.812	0.7962	3.49E-09	3.54E-09	3.59E-09	3.55E-09	3.61E-09	3.68E-09	3.74E-09	3.8E-09	3.85E-09	3.91E-09
42.507	139.457	0.8162	3.38E-09	3.43E-09	3.48E-09	3.44E-09	3.49E-09	3.56E-09	3.62E-09	3.68E-09	3.73E-09	3.78E-09
43.333	142.168	0.8362	3.28E-09	3.33E-09	3.38E-09	3.34E-09	3.39E-09	3.46E-09	3.51E-09	3.57E-09	3.62E-09	3.67E-09
44.179	144.944	0.8562	3.2E-09	3.24E-09	3.29E-09	3.25E-09	3.31E-09	3.37E-09	3.43E-09	3.48E-09	3.53E-09	3.58E-09
45.045	147.786	0.8762	3.13E-09	3.18E-09	3.22E-09	3.18E-09	3.24E-09	3.3E-09	3.35E-09	3.41E-09	3.46E-09	3.5E-09
45.931	150.693	0.8962	3.08E-09	3.13E-09	3.17E-09	3.13E-09	3.18E-09	3.24E-09	3.3E-09	3.35E-09	3.4E-09	3.45E-09
46.838	153.666	0.9162	3.05E-09	3.09E-09	3.14E-09	3.1E-09	3.15E-09	3.21E-09	3.26E-09	3.32E-09	3.36E-09	3.41E-09
47.616	156.221	0.64134	3.04E-09	3.08E-09	3.12E-09	3.09E-09	3.14E-09	3.2E-09	3.25E-09	3.3E-09	3.35E-09	3.4E-09

Integrated Coarse Tailings Sulphate generation Rates

	Coarse Tailings Beaches		Embankments	
	Cell 1E SO4	Cell 2E SO4	Cell 1E SO4	Cell 2E SO4
Year	mol/m2/year	mol/m2/year	mol/m2/year	mol/m2/year
0	0.000	1.319	0.000	0.000
1	0.000	1.319	0.000	1.319
2	0.000	1.319	0.000	0.639
3	0.000	1.323	0.000	1.493
4	0.000	1.341	0.000	2.500
5	0.000	1.395	0.000	3.640
6	0.000	1.530	0.000	3.688
7	0.000	1.849	0.000	3.710
8	0.000	2.639	0.000	3.722
9	1.319	4.225	1.319	3.730
10	1.583	7.819	0.639	3.734
11	2.757	11.062	1.493	3.737
12	6.069	11.832	2.500	3.739
13	9.152	12.113	3.640	3.741
14	9.871	12.108	3.688	3.741
15	10.217	11.850	3.710	3.742
16	10.411	11.262	3.722	3.743
17	10.529	10.360	3.730	3.743
18	9.915	8.956	3.734	3.743
19	8.993	6.817	3.737	3.743
20	7.575	6.833	3.739	3.743
21	5.425	6.845	3.741	3.744
22	5.433	6.804	3.741	3.744
24	5.438	6.871	3.742	3.744
27	5.442	6.950	3.743	3.744
30	5.445	7.052	3.743	3.744
33	5.447	7.189	3.743	3.744
36	5.449	7.134	3.743	3.744
39	5.450	7.711	3.743	3.744
42	5.451	8.108	3.744	3.744
45	5.451	8.828	3.744	3.744
48	5.846	9.872	3.744	3.744
51	6.564	11.392	3.744	3.744
54	7.607	11.393	3.744	3.744
57	9.127	11.393	3.744	3.744
60	9.128	11.394	3.744	3.744
63	9.128	11.394	3.744	3.746
66	9.128	11.394	3.744	3.749
69	9.129	11.394	3.744	3.418
72	9.129	11.395	3.744	2.813
75	9.129	11.395	3.744	2.151
78	9.129	11.397	3.744	1.281
81	9.129	11.401	3.744	0.821
84	9.129	11.408	3.744	0.337
87	9.129	11.421	3.746	0.000
90	9.130	11.443	3.749	0.000
93	9.133	11.478	3.418	0.000
96	9.142	11.190	2.813	0.000
99	9.161	10.640	2.151	0.000
102	8.854	10.044	1.281	0.000
105	8.283	9.246	0.821	0.000
108	7.664	8.865	0.337	0.000
111	6.840	8.467	0.000	0.000
114	6.431	8.222	0.000	0.000
117	6.005	8.318	0.000	0.000
120	5.732	8.223	0.000	0.000
123	5.801	8.124	0.000	0.000
126	5.679	7.634	0.000	0.000
129	5.554	7.348	0.000	0.000
132	5.035	6.892	0.000	0.000
135	4.718	6.972	0.000	0.000
138	4.229	6.721	0.000	0.000
141	4.272	6.473	0.000	0.000
144	3.987	6.397	0.000	0.000
147	3.704	6.320	0.000	0.000
150	3.590	6.404	0.000	0.000
153	3.474	6.494	0.000	0.000
156	3.518	6.377	0.000	0.000
159	3.566	6.261	0.000	0.000

Fine Tailings Oxygen Consumption Rates

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s)																			
			TIME (YEAR)																			
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
0.001	0.002	0.001	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07
0.002	0.007	0.002	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07	1.46E-07
0.007	0.023	0.008	1.44E-07	1.44E-07	1.44E-07	1.44E-07	1.44E-07	1.44E-07	1.44E-07	1.44E-07	1.44E-07	1.44E-07	1.44E-07	1.44E-07	1.44E-07	1.44E-07	1.44E-07	1.44E-07	1.44E-07	1.44E-07	1.44E-07	1.44E-07
0.019	0.062	0.016	1.4E-07	1.41E-07	1.41E-07	1.41E-07	1.41E-07	1.41E-07	1.41E-07	1.41E-07	1.41E-07	1.41E-07	1.41E-07	1.41E-07	1.41E-07	1.41E-07	1.41E-07	1.41E-07	1.41E-07	1.41E-07	1.41E-07	1.41E-07
0.043	0.141	0.032	1.32E-07	1.33E-07	1.34E-07	1.34E-07	1.34E-07	1.34E-07	1.34E-07	1.34E-07	1.34E-07	1.34E-07	1.34E-07	1.34E-07	1.34E-07	1.34E-07	1.34E-07	1.34E-07	1.34E-07	1.34E-07	1.34E-07	1.34E-07
0.091	0.299	0.064	1.17E-07	1.2E-07	1.21E-07	1.21E-07	1.21E-07	1.21E-07	1.22E-07	1.22E-07	1.22E-07	1.22E-07	1.22E-07	1.22E-07	1.22E-07	1.22E-07	1.22E-07	1.22E-07	1.22E-07	1.22E-07	1.22E-07	1.22E-07
0.181	0.593	0.1152	9.12E-08	9.78E-08	9.97E-08	1E-07	1.01E-07	1.01E-07	1.01E-07	1.01E-07	1.01E-07	1.01E-07	1.01E-07	1.01E-07	1.01E-07	1.01E-07	1.01E-07	1.01E-07	1.01E-07	1.01E-07	1.01E-07	1.01E-07
0.298	0.979	0.1202	6.47E-08	7.46E-08	7.75E-08	7.86E-08	7.9E-08	7.93E-08	7.94E-08	7.94E-08	7.94E-08	7.95E-08	7.95E-08	7.95E-08	7.95E-08	7.95E-08	7.95E-08	7.95E-08	7.95E-08	7.95E-08	7.95E-08	7.95E-08
0.421	1.381	0.1252	4.35E-08	5.55E-08	5.92E-08	6.07E-08	6.13E-08	6.16E-08	6.18E-08	6.19E-08	6.19E-08	6.19E-08	6.19E-08	6.2E-08	6.2E-08	6.2E-08	6.2E-08	6.2E-08	6.2E-08	6.2E-08	6.2E-08	6.2E-08
0.549	1.800	0.1302	2.75E-08	4.02E-08	4.45E-08	4.63E-08	4.7E-08	4.74E-08	4.76E-08	4.77E-08	4.78E-08	4.78E-08	4.78E-08	4.78E-08	4.78E-08	4.78E-08	4.78E-08	4.78E-08	4.78E-08	4.78E-08	4.78E-08	4.78E-08
0.681	2.236	0.1352	1.62E-08	2.82E-08	3.28E-08	3.47E-08	3.56E-08	3.61E-08	3.63E-08	3.64E-08	3.65E-08	3.65E-08	3.65E-08	3.65E-08	3.65E-08	3.65E-08	3.65E-08	3.65E-08	3.65E-08	3.65E-08	3.65E-08	3.65E-08
0.819	2.687	0.1402	8.85E-09	1.9E-08	2.36E-08	2.57E-08	2.66E-08	2.71E-08	2.73E-08	2.75E-08	2.75E-08	2.76E-08	2.76E-08	2.76E-08	2.76E-08	2.76E-08	2.76E-08	2.76E-08	2.76E-08	2.76E-08	2.76E-08	2.76E-08
0.962	3.156	0.1452	4.43E-09	1.24E-08	1.66E-08	1.86E-08	1.96E-08	2.01E-08	2.04E-08	2.05E-08	2.06E-08	2.06E-08	2.07E-08	2.07E-08	2.07E-08	2.07E-08	2.07E-08	2.07E-08	2.07E-08	2.07E-08	2.07E-08	2.07E-08
1.110	3.640	0.1502	2.03E-09	7.66E-09	1.13E-08	1.32E-08	1.42E-08	1.47E-08	1.5E-08	1.52E-08	1.52E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08
1.262	4.141	0.1552	8.42E-10	4.51E-09	7.48E-09	9.2E-09	1.01E-08	1.06E-08	1.09E-08	1.11E-08	1.11E-08	1.12E-08	1.12E-08	1.12E-08	1.12E-08	1.12E-08	1.12E-08	1.13E-08	1.13E-08	1.13E-08	1.13E-08	1.13E-08
1.420	4.658	0.1602	3.17E-10	2.52E-09	4.77E-09	6.24E-09	7.09E-09	7.57E-09	7.84E-09	7.98E-09	8.07E-09	8.11E-09	8.14E-09	8.15E-09	8.16E-09	8.17E-09	8.17E-09	8.17E-09	8.17E-09	8.17E-09	8.17E-09	8.17E-09
1.583	5.192	0.1652	1.08E-10	1.32E-09	2.93E-09	4.12E-09	4.86E-09	5.3E-09	5.55E-09	5.69E-09	5.77E-09	5.82E-09	5.84E-09	5.86E-09	5.87E-09	5.87E-09	5.87E-09	5.88E-09	5.88E-09	5.88E-09	5.88E-09	5.88E-09
1.750	5.742	0.1702	3.3E-11	6.52E-10	1.73E-09	2.64E-09	3.26E-09	3.64E-09	3.87E-09	4E-09	4.08E-09	4.12E-09	4.15E-09	4.17E-09	4.17E-09	4.18E-09	4.18E-09	4.18E-09	4.18E-09	4.19E-09	4.19E-09	4.19E-09
1.923	6.309	0.1752	9.14E-12	3.01E-10	9.73E-10	1.64E-09	2.13E-09	2.46E-09	2.66E-09	2.78E-09	2.85E-09	2.89E-09	2.92E-09	2.93E-09	2.94E-09	2.94E-09	2.95E-09	2.95E-09	2.95E-09	2.95E-09	2.95E-09	2.95E-09
2.101	6.892	0.1802	2.29E-12	1.29E-10	5.22E-10	9.83E-10	1.36E-09	1.62E-09	1.79E-09	1.9E-09	1.96E-09	2E-09	2.03E-09	2.04E-09	2.05E-09	2.05E-09	2.06E-09	2.06E-09	2.06E-09	2.06E-09	2.06E-09	2.06E-09
2.283	7.491	0.1852	5.17E-13	5.18E-11	2.66E-10	5.67E-10	8.4E-10	1.05E-09	1.19E-09	1.28E-09	1.33E-09	1.37E-09	1.39E-09	1.4E-09	1.41E-09	1.42E-09	1.42E-09	1.42E-09	1.42E-09	1.42E-09	1.42E-09	1.42E-09
2.471	8.107	0.1902	1.06E-13	1.92E-11	1.28E-10	3.14E-10	5.04E-10	6.58E-10	7.7E-10	8.44E-10	8.93E-10	9.24E-10	9.43E-10	9.55E-10	9.63E-10	9.67E-10	9.69E-10	9.71E-10	9.72E-10	9.73E-10	9.73E-10	9.73E-10
2.664	8.740	0.1952	1.96E-14	6.61E-12	5.85E-11	1.66E-10	2.92E-10	4.03E-10	4.89E-10	5.48E-10	5.89E-10	6.15E-10	6.32E-10	6.43E-10	6.49E-10	6.53E-10	6.55E-10	6.57E-10	6.58E-10	6.59E-10	6.59E-10	6.59E-10
2.862	9.388	0.2002	3.32E-15	2.11E-12	2.51E-11	8.41E-11	1.63E-10	2.4E-10	3.03E-10	3.49E-10	3.82E-10	4.04E-10	4.18E-10	4.27E-10	4.33E-10	4.36E-10	4.38E-10	4.4E-10	4.41E-10	4.41E-10	4.42E-10	4.42E-10
3.064	10.053	0.2052	5.13E-16	6.2E-13	1.01E-11	4.06E-11	8.77E-11	1.38E-10	1.83E-10	2.18E-10	2.43E-10	2.61E-10	2.72E-10	2.8E-10	2.85E-10	2.88E-10	2.9E-10	2.91E-10	2.92E-10	2.93E-10	2.93E-10	2.93E-10
3.272	10.735	0.2102	7.34E-17	1.69E-13	3.84E-12	1.86E-11	4.52E-11	7.72E-11	1.08E-10	1.32E-10	1.52E-10	1.66E-10	1.75E-10	1.81E-10	1.86E-10	1.88E-10	1.9E-10	1.91E-10	1.92E-10	1.92E-10	1.93E-10	1.93E-10
3.485	11.432	0.2152	1.07E-17	4.25E-14	1.36E-12	8.08E-12	2.24E-11	4.15E-11	6.14E-11	7.87E-11	9.27E-11	1.03E-10	1.11E-10	1.16E-10	1.19E-10	1.22E-10	1.23E-10	1.24E-10	1.24E-10	1.25E-10	1.25E-10	1.25E-10
3.702	12.147	0.2202	2.58E-18	9.89E-15	4.54E-13	3.33E-12	1.06E-11	2.15E-11	3.39E-11	4.55E-11	5.54E-11	6.31E-11	6.88E-11	7.28E-11	7.56E-11	7.75E-11	7.85E-11	7.94E-11	7.99E-11	8.03E-11	8.05E-11	8.07E-11
3.925	12.877	0.2252	1.68E-18	2.13E-15	1.41E-13	1.3E-12	4.76E-12	1.07E-11	1.81E-11	2.56E-11	3.23E-11	3.78E-11	4.2E-11	4.51E-11	4.73E-11	4.87E-11	4.96E-11	5.03E-11	5.08E-11	5.11E-11	5.13E-11	5.14E-11
4.153	13.624	0.2302	1.65E-18	4.26E-16	4.11E-14	4.77E-13	2.05E-12	5.13E-12	9.37E-12	1.4E-11	1.84E-11	2.22E-11	2.52E-11	2.74E-11	2.91E-11	3.02E-11	3.09E-11	3.15E-11	3.19E-11	3.22E-11	3.23E-11	3.24E-11
4.385	14.388	0.2352	1.65E-18	7.98E-17	1.12E-14	1.65E-13	8.36E-13	2.35E-12	4.67E-12	7.39E-12	1.02E-11	1.27E-11	1.48E-11	1.64E-11	1.76E-11	1.85E-11	1.9E-11	1.95E-11	1.98E-11	2E-11	2.01E-11	2.02E-11
4.623	15.168	0.2402	1.65E-18	1.47E-17	2.84E-15	5.41E-14	3.24E-13	1.03E-12	2.23E-12	3.78E-12	5.47E-12	7.07E-12	8.47E-12	9.61E-12	1.05E-11	1.11E-11	1.15E-11	1.19E-11	1.21E-11	1.23E-11	1.24E-11	1.25E-11
4.866	15.964	0.2452	1.65E-18	3.48E-18	6.76E-16	1.66E-14	1.19E-13	4.31E-13	1.03E-12	1.87E-12	2.85E-12	3.84E-12	4.75E-12	5.51E-12	6.13E-12	6.6E-12	6.89E-12	7.16E-12	7.35E-12	7.48E-12	7.57E-12	7.63E-12
5.114	16.777	0.2502	1.65E-18	1.77E-18	1.51E-16	4.82E-15	4.17E-14	1.72E-13	4.54E-13	8.9E-13	1.44E-12	2.03E-12	2.59E-12	3.1E-12	3.51E-12	3.84E-12	4.05E-12	4.25E-12	4.39E-12	4.49E-12	4.56E-12	4.61E-12
5.366	17.606	0.2552	1.65E-18	1.65E-18	3.22E-17	1.32E-15	1.38E-14	6.55E-14	1.92E-13	4.09E-13	7.02E-13	1.04E-12	1.38E-12	1.7E-12	1.97E-12	2.2E-12	2.34E-12	2.48E-12	2.59E-12	2.66E-12	2.72E-12	2.75E-12
5.624	18.451	0.2602	1.65E-18	1.65E-18	7.21E-18	3.38E-16	4.31E-15	2.37E-14	7.76E-14	1.81E-13	3.31E-13	5.16E-13	7.15E-13	9.09E-13	1.08E-12	1.23E-12	1.33E-12	1.43E-12	1.5E-12	1.56E-12	1.6E-12	1.62E-12
5.888	19.317	0.2677	1.65E-18	1.65E-18	2.39E-18	8.17E-17	1.27E-15	8.12E-15	2.98E-14	7.63E-14	1.5E-13	2.47E-13	3.58E-13	4.72E-13	5.79E-13	6.73E-13	7.37E-13	8.03E-13	8.55E-13	8.94E-13	9.23E-13	9.44E-13
6.159	20.208	0.2752	1.65E-18	1.65E-18	1.65E-18	1.9E-17	3.48E-16	2.61E-15	1.09E-14	3.07E-14	6.49E-14	1.14E-13	1.73E-13	2.37E-13	3E-13	3.58E-13	3.99E-13	4.42E-13	4.76E-13	5.03E-13	5.24E-13	5.39E-13
6.438	21.123	0.2827	1.65E-18	1.65E-18	1.65E-18	4.83E-18	9E-17	7.91E-16	3.73E-15	1.17E-14	2.68E-14	5.01E-14	8.03E-14	1.15E-13	1.51E-13	1.85E-13	2.1E-13	2.37E-13	2.59E-13	2.77E-13	2.91E-13	3.02E-13
6.725	22.063	0.2902	1.65E-18	1.65E-18	1.65E-18	1.99E-18	2.23E-17	2.25E-16	1.21E-15	4.26E-15	1.06E-14	2.11E-14	3.58E-14	5.36E-14	7.31E-14	9.25E-14	1.07E-13	1.24E-13	1.38E-13	1.49E-13	1.59E-13	1.66E-13
7.019	23.027	0.2977	1.65E-18	1.65E-18	1.65E-18	1.65E-18	5.79E-18	6.05E-17	3.7E-16	1.47E-15	3.96E-15	8.5E-15	1.53E-14	2.41E-14	3.42E-14	4.48E-14	5.31E-14	6.28E-14	7.13E-14	7.85E-14	8.44E-14	8.91E-14
7.320	24.016	0.3052	1.65E-18	1.65E-18	1.65E-18	1.65E-18	2.18E-18	1.57E-17	1.07E-16	4.81E-16	1.41E-15	3.27E-15	6.26E-15	1.04E-14	1.54E-14	2.1E-14	2.55E-14	3.09E-14	3.59E-14	4.03E-14	4.39E-14	4.69E-14
7.629	25.030	0.3127	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	4.46E-18	2.95E-17	1.5E-16	4.8E-16	1.2E-15	2.45E-15	4.3E-15	6.69E-15	9.46E-15	1.18E-14	1.48E-14	1.76E-14	2.01E-14	2.23E-14	2.41E-14
7.946	26.068	0.3202	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.94E-18	8.15E-18	4.43E-17	1.55E-16	4.17E-16	9.14E-16	1.7E-15	2.78E-15	4.11E-15	5.31E-15	6.84E-15	8.35E-15	9.77E-15	1.1E-14	1.21E-14
8.269	27.131	0.3277	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	2.76E-18	1.28E-17	4.76E-17	1.39E-16	3.26E-16	6.45E-16	1.11E-15	1.72E-15	2.3E-15	3.06E-15	3.85E-15	4.61E-15	5.31E-15	5.93E-15
8.601	28.218	0.3352	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.66E-18	4.05E-18	1.43E-17	4.41E-17	1.11E-16	2.34E-16	4.27E-16	6.93E-16	9.59E-16	1.32E-15	1.72E-15	2.11E-15	2.49E-15	2.83E-15
8.950	29.364	0.3632	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.84E-18	4.35E-18	1.31E-17	3.48E-17	7.82E-17	1.51E-16	2.58E-16	3.73E-16	5.35E-16	7.17E-16	9.09E-16	1.1E-15	1.28E-15
9.327	30.601	0.3912	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.86E-18	3.92E-18	1.01E-17	2.36E-17	4.84E-17	8.73E-17	1.32E-16	1.98E-16	2.75E-16	3.6E-16	4.49E-16	5.36E-16
9.733	31.931	0.4192	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.75E-18	3.1E-18	6.73E-18	1.42E-17	2.68E-17	4.25E-17	6.66E-17	9.66E-17	1.31E-16	1.69E-16	2.07E-16
10.166	33.352	0.4472	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	2.27E-18	4.12E-18	7.7E-18	1.26E-17	2.05E-17	3.1E-17	4.38E-17	5.84E-17	7.4E-17
10.627	34.865	0.4752	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.73E-18	2.48E-18	3.76E-18	6.01E-18	9.27E-18	1.35E-17	1.86E-17	2.44E-17
11.116	36.470	0.5032	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.67E-18	2.08E-18	2.91E-18	4.09E-18	5.64E-18	7.52E-18
11.633	38.167	0.5312	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.69E-18	2E-18	2.5E-18
12.179	39.956	0.5592	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
12.752	41.836	0.5872	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
13.353	43.809	0.6152	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
13.982	45.873	0.6432	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
14.639	48.029	0.6712	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
15.325	50.277	0.6992	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
16.038	52.617	0.7272	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
16.779	55.049	0.7552	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
17.548	57.573	0.7832	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
18.345	60.188	0.8112	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18

Fine Tailings Oxygen Consumption Rates

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s)																			
			TIME (YEAR)																			
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
19.171	62.895	0.8392	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
20.024	65.695	0.8672	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
20.905	68.586	0.8952	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
21.814	71.569	0.9232	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
22.751	74.643	0.9512	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
23.717	77.810	0.9792	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
24.710	81.069	1.0072	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
25.731	84.419	1.0352	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
26.780	87.861	1.0632	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
27.857	91.395	1.0912	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
28.963	95.021	1.1192	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
30.096	98.739	1.1472	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
31.257	102.549	1.1752	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
32.446	106.450	1.2032	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
33.663	110.444	1.2312	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
34.909	114.529	1.2592	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
36.182	118.706	1.2872	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
37.483	122.975	1.3152	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
38.812	127.336	1.3432	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
40.160	131.760	1.3532	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
41.519	136.216	1.3632	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
42.887	140.704	1.3732	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
44.265	145.226	1.3832	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
45.653	149.781	1.3932	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
47.051	154.368	1.4032	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
47.758	156.686	0.01	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18

Fine Tailings Oxygen Consump

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s)																				
			TIME (YEAR)																				
			21	22	24	27	30	33	36	39	42	45	48	51	54	57	60	63	66	69	72	75	
0.001	0.002	0.001	1.46E-07	1.46E-07	1.46E-07	1.46E-07	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
0.002	0.007	0.002	1.46E-07	1.46E-07	1.46E-07	1.46E-07	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
0.007	0.023	0.008	1.44E-07	1.44E-07	1.44E-07	1.44E-07	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
0.019	0.062	0.016	1.41E-07	1.41E-07	1.41E-07	1.41E-07	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
0.043	0.141	0.032	1.34E-07	1.34E-07	1.34E-07	1.34E-07	1.34E-07	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
0.091	0.299	0.064	1.22E-07	1.22E-07	1.22E-07	1.22E-07	1.22E-07	1.22E-07	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
0.181	0.593	0.1152	1.01E-07	1.01E-07	1.01E-07	1.01E-07	1.01E-07	1.02E-07	1.03E-07	1.04E-07	1.03E-07	0	0	0	0	0	0	0	0	0	0	0	
0.298	0.979	0.1202	7.95E-08	7.95E-08	7.95E-08	7.95E-08	7.95E-08	8E-08	8.13E-08	8.16E-08	8.12E-08	8.61E-08	8.69E-08	8.69E-08	8.69E-08	0	0	0	0	0	0	0	
0.421	1.381	0.1252	6.2E-08	6.2E-08	6.2E-08	6.2E-08	6.2E-08	6.23E-08	6.32E-08	6.36E-08	6.35E-08	6.67E-08	6.77E-08	6.77E-08	6.78E-08	7.37E-08	7.42E-08	7.53E-08	7.43E-08	0	0	0	
0.549	1.800	0.1302	4.78E-08	4.78E-08	4.78E-08	4.78E-08	4.78E-08	4.81E-08	4.86E-08	4.91E-08	4.91E-08	5.12E-08	5.22E-08	5.23E-08	5.23E-08	5.66E-08	5.73E-08	5.78E-08	5.74E-08	6.39E-08	6.44E-08	6.44E-08	
0.681	2.236	0.1352	3.65E-08	3.65E-08	3.65E-08	3.65E-08	3.65E-08	3.67E-08	3.71E-08	3.75E-08	3.75E-08	3.88E-08	3.98E-08	3.99E-08	3.99E-08	4.3E-08	4.37E-08	4.39E-08	4.38E-08	4.86E-08	4.92E-08	4.92E-08	
0.819	2.687	0.1402	2.76E-08	2.76E-08	2.76E-08	2.76E-08	2.76E-08	2.77E-08	2.8E-08	2.83E-08	2.83E-08	2.91E-08	3.01E-08	3.02E-08	3.02E-08	3.24E-08	3.3E-08	3.32E-08	3.32E-08	3.66E-08	3.71E-08	3.72E-08	
0.962	3.156	0.1452	2.07E-08	2.07E-08	2.07E-08	2.07E-08	2.07E-08	2.08E-08	2.09E-08	2.12E-08	2.12E-08	2.16E-08	2.25E-08	2.26E-08	2.26E-08	2.41E-08	2.47E-08	2.48E-08	2.48E-08	2.73E-08	2.78E-08	2.79E-08	
1.110	3.640	0.1502	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.53E-08	1.54E-08	1.55E-08	1.57E-08	1.57E-08	1.59E-08	1.66E-08	1.67E-08	1.68E-08	1.77E-08	1.83E-08	1.84E-08	1.84E-08	2.01E-08	2.06E-08	2.07E-08	
1.262	4.141	0.1552	1.13E-08	1.13E-08	1.13E-08	1.13E-08	1.13E-08	1.13E-08	1.13E-08	1.15E-08	1.15E-08	1.16E-08	1.22E-08	1.23E-08	1.23E-08	1.29E-08	1.34E-08	1.35E-08	1.35E-08	1.46E-08	1.51E-08	1.51E-08	
1.420	4.658	0.1602	8.17E-09	8.17E-09	8.17E-09	8.17E-09	8.17E-09	8.19E-09	8.22E-09	8.35E-09	8.33E-09	8.42E-09	8.82E-09	8.92E-09	8.94E-09	9.27E-09	9.7E-09	9.78E-09	9.82E-09	1.05E-08	1.09E-08	1.1E-08	
1.583	5.192	0.1652	5.88E-09	5.88E-09	5.88E-09	5.88E-09	5.88E-09	5.89E-09	5.91E-09	6E-09	5.98E-09	6.04E-09	6.32E-09	6.41E-09	6.43E-09	6.61E-09	6.96E-09	7.03E-09	7.06E-09	7.51E-09	7.84E-09	7.91E-09	
1.750	5.742	0.1702	4.19E-09	4.19E-09	4.19E-09	4.19E-09	4.19E-09	4.19E-09	4.21E-09	4.27E-09	4.25E-09	4.29E-09	4.48E-09	4.56E-09	4.57E-09	4.67E-09	4.93E-09	5E-09	5.03E-09	5.3E-09	5.56E-09	5.63E-09	
1.923	6.309	0.1752	2.95E-09	2.95E-09	2.95E-09	2.95E-09	2.95E-09	2.95E-09	2.96E-09	3E-09	2.99E-09	3.02E-09	3.15E-09	3.21E-09	3.22E-09	3.28E-09	3.46E-09	3.51E-09	3.54E-09	3.7E-09	3.91E-09	3.96E-09	
2.101	6.892	0.1802	2.06E-09	2.06E-09	2.06E-09	2.06E-09	2.06E-09	2.06E-09	2.07E-09	2.09E-09	2.09E-09	2.11E-09	2.19E-09	2.24E-09	2.25E-09	2.27E-09	2.4E-09	2.44E-09	2.47E-09	2.56E-09	2.71E-09	2.76E-09	
2.283	7.491	0.1852	1.42E-09	1.42E-09	1.42E-09	1.42E-09	1.42E-09	1.42E-09	1.43E-09	1.44E-09	1.44E-09	1.45E-09	1.5E-09	1.54E-09	1.55E-09	1.57E-09	1.65E-09	1.68E-09	1.71E-09	1.75E-09	1.87E-09	1.91E-09	
2.471	8.107	0.1902	9.73E-10	9.74E-10	9.74E-10	9.74E-10	9.74E-10	9.74E-10	9.77E-10	9.85E-10	9.83E-10	9.94E-10	1.02E-09	1.05E-09	1.06E-09	1.07E-09	1.12E-09	1.15E-09	1.17E-09	1.19E-09	1.27E-09	1.3E-09	
2.664	8.740	0.1952	6.59E-10	6.59E-10	6.59E-10	6.59E-10	6.59E-10	6.6E-10	6.61E-10	6.66E-10	6.65E-10	6.72E-10	6.89E-10	7.11E-10	7.19E-10	7.22E-10	7.55E-10	7.72E-10	7.88E-10	8.02E-10	8.53E-10	8.79E-10	
2.862	9.388	0.2002	4.42E-10	4.42E-10	4.42E-10	4.42E-10	4.42E-10	4.42E-10	4.42E-10	4.43E-10	4.46E-10	4.45E-10	4.5E-10	4.6E-10	4.75E-10	4.81E-10	4.84E-10	5.03E-10	5.15E-10	5.27E-10	5.35E-10	5.68E-10	5.88E-10
3.064	10.053	0.2052	2.93E-10	2.93E-10	2.94E-10	2.94E-10	2.94E-10	2.94E-10	2.94E-10	2.96E-10	2.95E-10	2.98E-10	3.04E-10	3.14E-10	3.19E-10	3.21E-10	3.32E-10	3.4E-10	3.49E-10	3.54E-10	3.74E-10	3.89E-10	
3.272	10.735	0.2102	1.93E-10	1.93E-10	1.93E-10	1.93E-10	1.93E-10	1.93E-10	1.93E-10	1.94E-10	1.94E-10	1.96E-10	1.99E-10	2.06E-10	2.09E-10	2.11E-10	2.17E-10	2.22E-10	2.29E-10	2.32E-10	2.44E-10	2.54E-10	
3.485	11.432	0.2152	1.25E-10	1.25E-10	1.26E-10	1.26E-10	1.26E-10	1.26E-10	1.26E-10	1.26E-10	1.26E-10	1.27E-10	1.29E-10	1.33E-10	1.36E-10	1.37E-10	1.4E-10	1.44E-10	1.48E-10	1.51E-10	1.57E-10	1.65E-10	
3.702	12.147	0.2202	8.08E-11	8.08E-11	8.09E-11	8.09E-11	8.09E-11	8.09E-11	8.1E-11	8.13E-11	8.12E-11	8.19E-11	8.29E-11	8.55E-11	8.74E-11	8.82E-11	8.99E-11	9.22E-11	9.53E-11	9.69E-11	1.01E-10	1.06E-10	
3.925	12.877	0.2252	5.15E-11	5.15E-11	5.16E-11	5.16E-11	5.16E-11	5.16E-11	5.17E-11	5.18E-11	5.18E-11	5.22E-11	5.28E-11	5.43E-11	5.56E-11	5.62E-11	5.71E-11	5.85E-11	6.05E-11	6.17E-11	6.37E-11	6.69E-11	
4.153	13.624	0.2302	3.25E-11	3.25E-11	3.26E-11	3.26E-11	3.26E-11	3.26E-11	3.26E-11	3.27E-11	3.27E-11	3.29E-11	3.33E-11	3.41E-11	3.5E-11	3.55E-11	3.59E-11	3.68E-11	3.81E-11	3.89E-11	4E-11	4.2E-11	
4.385	14.388	0.2352	2.03E-11	2.03E-11	2.04E-11	2.04E-11	2.04E-11	2.04E-11	2.04E-11	2.04E-11	2.05E-11	2.05E-11	2.06E-11	2.08E-11	2.12E-11	2.18E-11	2.22E-11	2.24E-11	2.29E-11	2.37E-11	2.42E-11	2.48E-11	2.61E-11
4.623	15.168	0.2402	1.25E-11	1.26E-11	1.26E-11	1.26E-11	1.26E-11	1.26E-11	1.26E-11	1.27E-11	1.27E-11	1.27E-11	1.28E-11	1.31E-11	1.35E-11	1.37E-11	1.38E-11	1.41E-11	1.46E-11	1.5E-11	1.53E-11	1.6E-11	
4.866	15.964	0.2452	7.67E-12	7.69E-12	7.74E-12	7.75E-12	7.75E-12	7.75E-12	7.75E-12	7.76E-12	7.76E-12	7.79E-12	7.86E-12	8E-12	8.22E-12	8.38E-12	8.47E-12	8.62E-12	8.9E-12	9.15E-12	9.34E-12	9.75E-12	
5.114	16.777	0.2502	4.64E-12	4.66E-12	4.69E-12	4.7E-12	4.71E-12	4.71E-12	4.71E-12	4.71E-12	4.71E-12	4.73E-12	4.77E-12	4.84E-12	4.97E-12	5.07E-12	5.13E-12	5.22E-12	5.37E-12	5.53E-12	5.65E-12	5.87E-12	
5.366	17.606	0.2552	2.78E-12	2.79E-12	2.82E-12	2.83E-12	2.83E-12	2.83E-12	2.83E-12	2.83E-12	2.83E-12	2.84E-12	2.86E-12	2.9E-12	2.98E-12	3.04E-12	3.08E-12	3.12E-12	3.21E-12	3.31E-12	3.38E-12	3.51E-12	
5.624	18.451	0.2602	1.64E-12	1.65E-12	1.68E-12	1.68E-12	1.69E-12	1.69E-12	1.69E-12	1.69E-12	1.69E-12	1.69E-12	1.7E-12	1.72E-12	1.76E-12	1.81E-12	1.83E-12	1.85E-12	1.9E-12	1.96E-12	2.01E-12	2.07E-12	
5.888	19.317	0.2677	9.59E-13	9.66E-13	9.84E-13	9.89E-13	9.91E-13	9.91E-13	9.91E-13	9.92E-13	9.92E-13	9.94E-13	9.99E-13	1.01E-12	1.03E-12	1.06E-12	1.07E-12	1.08E-12	1.11E-12	1.15E-12	1.17E-12	1.21E-12	
6.159	20.208	0.2752	5.49E-13	5.55E-13	5.68E-13	5.73E-13	5.74E-13	5.74E-13	5.74E-13	5.74E-13	5.74E-13	5.75E-13	5.78E-13	5.84E-13	5.95E-13	6.1E-13	6.2E-13	6.26E-13	6.42E-13	6.61E-13	6.78E-13	6.96E-13	
6.438	21.123	0.2827	3.09E-13	3.13E-13	3.23E-13	3.26E-13	3.27E-13	3.28E-13	3.28E-13	3.28E-13	3.28E-13	3.28E-13	3.3E-13	3.33E-13	3.38E-13	3.47E-13	3.53E-13	3.56E-13	3.64E-13	3.75E-13	3.85E-13	3.95E-13	
6.725	22.063	0.2902	1.71E-13	1.74E-13	1.81E-13	1.83E-13	1.84E-13	1.84E-13	1.84E-13	1.84E-13	1.84E-13	1.85E-13	1.85E-13	1.87E-13	1.9E-13	1.94E-13	1.98E-13	1.99E-13	2.04E-13	2.1E-13	2.16E-13	2.21E-13	
7.019	23.027	0.2977	9.27E-14	9.45E-14	9.96E-14	1.01E-13	1.02E-13	1.02E-13	1.02E-13	1.02E-13	1.02E-13	1.02E-13	1.03E-13	1.03E-13	1.05E-13	1.07E-13	1.09E-13	1.1E-13	1.13E-13	1.16E-13	1.19E-13	1.22E-13	
7.320	24.016	0.3052	4.92E-14	5.05E-14	5.39E-14	5.52E-14	5.57E-14	5.58E-14	5.58E-14	5.59E-14	5.59E-14	5.59E-14	5.6E-14	5.64E-14	5.7E-14	5.82E-14	5.95E-14	5.99E-14	6.12E-14	6.28E-14	6.46E-14	6.62E-14	
7.629	25.030	0.3127	2.56E-14	2.64E-14	2.87E-14	2.96E-14	2.99E-14	3E-14	3.01E-14	3.01E-14	3.01E-14	3.01E-14	3.02E-14	3.03E-14	3.06E-14	3.12E-14	3.19E-14	3.21E-14	3.28E-14	3.36E-14	3.46E-14	3.55E-14	
7.946	26.068	0.3202	1.3E-14	1.35E-14	1.5E-14	1.56E-14	1.59E-14	1.59E-14	1.6E-14	1.6E-14	1.6E-14	1.6E-14	1.6E-14	1.61E-14	1.62E-14	1.65E-14	1.69E-14	1.7E-14	1.73E-14	1.77E-14	1.83E-14	1.87E-14	
8.269	27.131	0.3277	6.47E-15	6.77E-15	7.7E-15	8.11E-15	8.27E-15	8.33E-15	8.35E-15	8.35E-15	8.35E-15	8.35E-15	8.37E-15	8.4E-15	8.46E-15	8.59E-15	8.78E-15	8.83E-15	9.02E-15	9.23E-15	9.49E-15	9.75E-15	
8.601	28.218	0.3352	3.14E-15	3.31E-15	3.88E-15	4.15E-15	4.26E-15	4.29E-15	4.31E-15	4.31E-15	4.31E-15	4.31E-15	4.32E-15	4.33E-15	4.36E-15	4.42E-15	4.51E-15	4.54E-15	4.64E-15	4.74E-15	4.87E-15	5E-15	
8.950	29.364	0.3632	1.44E-15	1.54E-15	1.87E-15	2.03E-15	2.1E-15	2.13E-15	2.14E-15	2.14E-15	2.14E-15	2.14E-15	2.14E-15	2.15E-15	2.16E-15	2.19E-15	2.23E-15	2.24E-15	2.29E-15	2.34E-15	2.4E-15	2.47E-15	
9.327	30.601	0.3912	6.18E-16	6.68E-16	8.44E-16	9.39E-16	9.82E-16	9.99E-16	1E-15	1.01E-15	1.01E-15	1.01E-15	1.01E-15	1.01E-15	1.02E-15	1.03E-15	1.04E-15	1.05E-15	1.07E-15	1.1E-15	1.12E-15	1.15E-15	
9.733	31.931	0.4192	2.45E-16	2.69E-16	3.57E-16	4.09E-16	4.33E-16	4.44E-16	4.48E-16	4.49E-16	4.49E-16	4.49E-16	4.5E-16	4.51E-16	4.52E-16	4.56E-16	4.63E-16	4.67E-16	4.76E-16	4.86E-16	4.98E-16	5.11E-16	
10.166	33.352	0.4472	8.99E-17	1E-16	1.41E-16	1.67E-16	1.81E-16	1.87E-16	1.89E-16	1.9E-16	1.9E-16	1.9E-16	1.9E-16	1.91E-16	1.91E-16	1.93E-16	1.95E-16	1.97E-16	2E-16	2.05E-16	2.09E-16	2.15E-16	
10.627	34.865	0.4752	3.05E-17	3.48E-17	5.21E-17	6.44E-17	7.12E-17	7.44E-17	7.58E-17	7.63E-17	7.61E-17	7.64E-17	7.65E-17	7.66E-17	7.68E-17	7.72E-17	7.81E-17	7.87E-17	8.01E-17	8.17E-17	8.35E-17	8.57E-17	
11.116	36.470	0.5032	9.65E-18	1.12E-17	1.8E-17	2.32E-17	2.64E-17	2.81E-17	2.88E-17	2.91E-17	2.9E-17	2.92E-17	2.92E-17	2.92E-17	2.93E-17	2.94E-17	2.97E-17	2.99E-17	3.04E-17	3.1E-17	3.17E-17	3.25E-17	
11.633	38.167	0.5312	3.11E-18	3.58E-18	5.87E-18	7.94E-18	9.32E-18	1.01E-17	1.04E-17	1.06E-17	1.05E-17	1.06E-17	1.06E-17	1.07E-17	1.07E-17	1.07E-17	1.08E-17	1.09E-17	1.1E-17	1.12E-17	1.15E-17	1.17E-17	
12.179	39.956	0.5592	1.65E-18	1.65E-18	2.12E-18	2.79E-18	3.28E-18	3.57E-18	3.71E-18	3.77E-18	3.73E-18	3.78E-18	3.8E-18	3.8E-18	3.81E-18	3.82E-18	3.84E-18	3.84E-18	3.9E-18	3.96E-18	4.04E-18	4.13E-18	
12.752	41.836	0.5872	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.66E-18	1.69E-18	
13.353	43.809	0.6152	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	
13.982	45.873	0.6432	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	
14.639	48.029	0.6712	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	
15.325	50.277	0.6992	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	
16.038	52.617	0.7272	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	
16.779	55.049	0.7552	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	
17.548	57.573	0.7832	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	
18.345	60.188	0.8112	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	

Fine Tailings Oxygen Consump

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s)																				
			TIME (YEAR)																				
			21	22	24	27	30	33	36	39	42	45	48	51	54	57	60	63	66	69	72	75	
19.171	62.895	0.8392	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
20.024	65.695	0.8672	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
20.905	68.586	0.8952	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
21.814	71.569	0.9232	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
22.751	74.643	0.9512	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
23.717	77.810	0.9792	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
24.710	81.069	1.0072	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
25.731	84.419	1.0352	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
26.780	87.861	1.0632	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
27.857	91.395	1.0912	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
28.963	95.021	1.1192	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
30.096	98.739	1.1472	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
31.257	102.549	1.1752	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
32.446	106.450	1.2032	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
33.663	110.444	1.2312	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
34.909	114.529	1.2592	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
36.182	118.706	1.2872	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
37.483	122.975	1.3152	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
38.812	127.336	1.3432	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
40.160	131.760	1.3532	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
41.519	136.216	1.3632	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
42.887	140.704	1.3732	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
44.265	145.226	1.3832	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
45.653	149.781	1.3932	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
47.051	154.368	1.4032	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
47.758	156.686	0.01	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18

Fine Tailings Oxygen Consump

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s)																			
			TIME (YEAR)																			
			78	81	84	87	90	93	96	99	102	105	108	111	114	117	120	123	126	129	132	135
0.001	0.002	0.001	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.002	0.007	0.002	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.007	0.023	0.008	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.019	0.062	0.016	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.043	0.141	0.032	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.091	0.299	0.064	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.181	0.593	0.1152	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.298	0.979	0.1202	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.421	1.381	0.1252	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.549	1.800	0.1302	6.44E-08	6.44E-08	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
0.681	2.236	0.1352	4.92E-08	4.92E-08	4.95E-08	5.58E-08	5.65E-08	5.66E-08	5.66E-08	5.66E-08	5.66E-08	0	0	0	0	0	0	0	0	0	0	0
0.819	2.687	0.1402	3.72E-08	3.72E-08	3.73E-08	4.2E-08	4.27E-08	4.28E-08	4.28E-08	4.28E-08	4.28E-08	5.03E-08	5.01E-08	5.02E-08	5.02E-08	5.02E-08	5.02E-08	5.02E-08	0	0	0	0
0.962	3.156	0.1452	2.79E-08	2.79E-08	2.79E-08	3.13E-08	3.19E-08	3.2E-08	3.2E-08	3.2E-08	3.2E-08	3.68E-08	3.75E-08	3.75E-08	3.76E-08	3.76E-08	3.76E-08	3.76E-08	3.77E-08	4.34E-08	4.45E-08	4.48E-08
1.110	3.640	0.1502	2.07E-08	2.07E-08	2.07E-08	2.3E-08	2.36E-08	2.37E-08	2.37E-08	2.37E-08	2.38E-08	2.68E-08	2.78E-08	2.78E-08	2.78E-08	2.78E-08	2.78E-08	2.78E-08	2.79E-08	3.2E-08	3.29E-08	3.32E-08
1.262	4.141	0.1552	1.52E-08	1.52E-08	1.52E-08	1.68E-08	1.73E-08	1.74E-08	1.74E-08	1.74E-08	1.74E-08	1.93E-08	2.03E-08	2.04E-08	2.04E-08	2.04E-08	2.04E-08	2.04E-08	2.04E-08	2.32E-08	2.41E-08	2.43E-08
1.420	4.658	0.1602	1.1E-08	1.1E-08	1.1E-08	1.21E-08	1.25E-08	1.26E-08	1.27E-08	1.27E-08	1.27E-08	1.38E-08	1.47E-08	1.48E-08	1.48E-08	1.48E-08	1.48E-08	1.48E-08	1.48E-08	1.67E-08	1.75E-08	1.77E-08
1.583	5.192	0.1652	7.92E-09	7.92E-09	7.93E-09	8.59E-09	8.99E-09	9.08E-09	9.1E-09	9.11E-09	9.11E-09	9.79E-09	1.06E-08	1.07E-08	1.07E-08	1.07E-08	1.07E-08	1.07E-08	1.07E-08	1.19E-08	1.25E-08	1.27E-08
1.750	5.742	0.1702	5.64E-09	5.64E-09	5.64E-09	6.05E-09	6.38E-09	6.46E-09	6.48E-09	6.48E-09	6.48E-09	6.89E-09	7.49E-09	7.58E-09	7.6E-09	7.6E-09	7.6E-09	7.6E-09	7.6E-09	8.37E-09	8.88E-09	9.03E-09
1.923	6.309	0.1752	3.97E-09	3.98E-09	3.98E-09	4.22E-09	4.48E-09	4.55E-09	4.57E-09	4.57E-09	4.57E-09	4.81E-09	5.26E-09	5.34E-09	5.36E-09	5.36E-09	5.36E-09	5.36E-09	5.36E-09	5.83E-09	6.23E-09	6.35E-09
2.101	6.892	0.1802	2.77E-09	2.78E-09	2.77E-09	2.91E-09	3.11E-09	3.17E-09	3.19E-09	3.19E-09	3.19E-09	3.33E-09	3.64E-09	3.72E-09	3.74E-09	3.74E-09	3.74E-09	3.74E-09	3.74E-09	4.01E-09	4.32E-09	4.43E-09
2.283	7.491	0.1852	1.92E-09	1.92E-09	1.92E-09	1.99E-09	2.14E-09	2.19E-09	2.2E-09	2.2E-09	2.2E-09	2.28E-09	2.5E-09	2.57E-09	2.58E-09	2.58E-09	2.58E-09	2.58E-09	2.58E-09	2.74E-09	2.96E-09	3.05E-09
2.471	8.107	0.1902	1.31E-09	1.31E-09	1.31E-09	1.35E-09	1.45E-09	1.49E-09	1.5E-09	1.51E-09	1.51E-09	1.55E-09	1.69E-09	1.75E-09	1.76E-09	1.77E-09	1.77E-09	1.77E-09	1.77E-09	1.85E-09	2.01E-09	2.08E-09
2.664	8.740	0.1952	8.86E-10	8.88E-10	8.87E-10	9.08E-10	9.75E-10	1.01E-09	1.02E-09	1.02E-09	1.02E-09	1.05E-09	1.13E-09	1.18E-09	1.19E-09	1.2E-09	1.2E-09	1.2E-09	1.2E-09	1.24E-09	1.35E-09	1.4E-09
2.862	9.388	0.2002	5.94E-10	5.96E-10	5.95E-10	6.05E-10	6.48E-10	6.74E-10	6.82E-10	6.84E-10	6.85E-10	6.98E-10	7.52E-10	7.9E-10	8E-10	8.02E-10	8.03E-10	8.03E-10	8.02E-10	8.24E-10	8.97E-10	9.38E-10
3.064	10.053	0.2052	3.94E-10	3.95E-10	3.94E-10	3.99E-10	4.27E-10	4.46E-10	4.52E-10	4.54E-10	4.55E-10	4.61E-10	4.93E-10	5.22E-10	5.3E-10	5.33E-10	5.33E-10	5.33E-10	5.33E-10	5.43E-10	5.89E-10	6.2E-10
3.272	10.735	0.2102	2.59E-10	2.6E-10	2.59E-10	2.62E-10	2.78E-10	2.91E-10	2.97E-10	2.98E-10	2.99E-10	3.02E-10	3.21E-10	3.41E-10	3.48E-10	3.5E-10	3.5E-10	3.5E-10	3.5E-10	3.55E-10	3.83E-10	4.05E-10
3.485	11.432	0.2152	1.68E-10	1.69E-10	1.68E-10	1.7E-10	1.79E-10	1.89E-10	1.93E-10	1.94E-10	1.94E-10	1.96E-10	2.06E-10	2.2E-10	2.26E-10	2.28E-10	2.28E-10	2.28E-10	2.28E-10	2.3E-10	2.46E-10	2.62E-10
3.702	12.147	0.2202	1.08E-10	1.09E-10	1.08E-10	1.09E-10	1.14E-10	1.21E-10	1.24E-10	1.25E-10	1.25E-10	1.26E-10	1.32E-10	1.41E-10	1.45E-10	1.47E-10	1.47E-10	1.47E-10	1.47E-10	1.48E-10	1.57E-10	1.68E-10
3.925	12.877	0.2252	6.87E-11	6.94E-11	6.88E-11	6.95E-11	7.23E-11	7.65E-11	7.88E-11	7.97E-11	7.99E-11	8.03E-11	8.32E-11	8.91E-11	9.23E-11	9.34E-11	9.37E-11	9.38E-11	9.36E-11	9.4E-11	9.9E-11	1.06E-10
4.153	13.624	0.2302	4.33E-11	4.38E-11	4.33E-11	4.38E-11	4.53E-11	4.8E-11	4.96E-11	5.03E-11	5.05E-11	5.06E-11	5.22E-11	5.58E-11	5.81E-11	5.89E-11	5.92E-11	5.92E-11	5.91E-11	5.93E-11	6.19E-11	6.63E-11
4.385	14.388	0.2352	2.7E-11	2.73E-11	2.7E-11	2.74E-11	2.81E-11	2.97E-11	3.09E-11	3.14E-11	3.15E-11	3.16E-11	3.24E-11	3.45E-11	3.61E-11	3.68E-11	3.7E-11	3.7E-11	3.69E-11	3.7E-11	3.84E-11	4.11E-11
4.623	15.168	0.2402	1.66E-11	1.69E-11	1.66E-11	1.69E-11	1.73E-11	1.83E-11	1.9E-11	1.94E-11	1.95E-11	1.95E-11	2E-11	2.12E-11	2.22E-11	2.27E-11	2.29E-11	2.29E-11	2.28E-11	2.29E-11	2.36E-11	2.52E-11
4.866	15.964	0.2452	1.01E-11	1.03E-11	1.02E-11	1.04E-11	1.06E-11	1.11E-11	1.16E-11	1.19E-11	1.2E-11	1.2E-11	1.22E-11	1.28E-11	1.35E-11	1.39E-11	1.4E-11	1.41E-11	1.4E-11	1.4E-11	1.43E-11	1.53E-11
5.114	16.777	0.2502	6.12E-12	6.26E-12	6.13E-12	6.27E-12	6.38E-12	6.68E-12	7E-12	7.18E-12	7.26E-12	7.25E-12	7.36E-12	7.72E-12	8.16E-12	8.41E-12	8.51E-12	8.54E-12	8.47E-12	8.52E-12	8.66E-12	9.17E-12
5.366	17.606	0.2552	3.66E-12	3.75E-12	3.66E-12	3.76E-12	3.82E-12	3.98E-12	4.18E-12	4.3E-12	4.36E-12	4.35E-12	4.41E-12	4.6E-12	4.87E-12	5.04E-12	5.11E-12	5.13E-12	5.08E-12	5.12E-12	5.18E-12	5.46E-12
5.624	18.451	0.2602	2.16E-12	2.23E-12	2.17E-12	2.23E-12	2.27E-12	2.35E-12	2.47E-12	2.55E-12	2.59E-12	2.58E-12	2.62E-12	2.71E-12	2.87E-12	2.98E-12	3.04E-12	3.05E-12	3.02E-12	3.05E-12	3.08E-12	3.22E-12
5.888	19.317	0.2677	1.26E-12	1.3E-12	1.27E-12	1.31E-12	1.33E-12	1.37E-12	1.44E-12	1.49E-12	1.52E-12	1.51E-12	1.54E-12	1.58E-12	1.67E-12	1.74E-12	1.78E-12	1.79E-12	1.77E-12	1.79E-12	1.8E-12	1.87E-12
6.159	20.208	0.2752	7.26E-13	7.51E-13	7.29E-13	7.52E-13	7.67E-13	7.89E-13	8.27E-13	8.6E-13	8.78E-13	8.72E-13	8.87E-13	9.1E-13	9.58E-13	1E-12	1.03E-12	1.04E-12	1.02E-12	1.03E-12	1.04E-12	1.08E-12
6.438	21.123	0.2827	4.11E-13	4.26E-13	4.13E-13	4.27E-13	4.36E-13	4.47E-13	4.68E-13	4.88E-13	4.99E-13	4.96E-13	5.05E-13	5.16E-13	5.42E-13	5.69E-13	5.85E-13	5.92E-13	5.79E-13	5.89E-13	5.94E-13	6.09E-13
6.725	22.063	0.2902	2.29E-13	2.38E-13	2.31E-13	2.39E-13	2.45E-13	2.5E-13	2.61E-13	2.72E-13	2.8E-13	2.77E-13	2.83E-13	2.89E-13	3.02E-13	3.17E-13	3.27E-13	3.32E-13	3.24E-13	3.3E-13	3.34E-13	3.4E-13
7.019	23.027	0.2977	1.26E-13	1.31E-13	1.27E-13	1.32E-13	1.35E-13	1.38E-13	1.43E-13	1.5E-13	1.54E-13	1.53E-13	1.56E-13	1.59E-13	1.66E-13	1.74E-13	1.8E-13	1.84E-13	1.78E-13	1.83E-13	1.85E-13	1.88E-13
7.320	24.016	0.3052	6.84E-14	7.11E-14	6.89E-14	7.14E-14	7.34E-14	7.5E-14	7.76E-14	8.11E-14	8.39E-14	8.3E-14	8.5E-14	8.66E-14	8.97E-14	9.42E-14	9.8E-14	1E-13	9.68E-14	9.93E-14	1.01E-13	1.02E-13
7.629	25.030	0.3127	3.66E-14	3.8E-14	3.69E-14	3.82E-14	3.93E-14	4.02E-14	4.14E-14	4.33E-14	4.49E-14	4.44E-14	4.56E-14	4.65E-14	4.79E-14	5.02E-14	5.24E-14	5.37E-14	5.17E-14	5.32E-14	5.41E-14	5.47E-14
7.946	26.068	0.3202	1.93E-14	2E-14	1.94E-14	2.01E-14	2.07E-14	2.12E-14	2.18E-14	2.28E-14	2.36E-14	2.34E-14	2.41E-14	2.46E-14	2.52E-14	2.64E-14	2.76E-14	2.84E-14	2.72E-14	2.81E-14	2.86E-14	2.9E-14
8.269	27.131	0.3277	1E-14	1.04E-14	1.01E-14	1.04E-14	1.08E-14	1.1E-14	1.13E-14	1.18E-14	1.23E-14	1.22E-14	1.25E-14	1.28E-14	1.31E-14	1.37E-14	1.43E-14	1.48E-14	1.41E-14	1.46E-14	1.49E-14	1.51E-14
8.601	28.218	0.3352	5.14E-15	5.31E-15	5.17E-15	5.35E-15	5.53E-15	5.68E-15	5.82E-15	6.04E-15	6.29E-15	6.23E-15	6.42E-15	6.57E-15	6.72E-15	6.98E-15	7.31E-15	7.57E-15	7.24E-15	7.49E-15	7.67E-15	7.78E-15
8.950	29.364	0.3632	2.53E-15	2.62E-15	2.55E-15	2.63E-15	2.72E-15	2.8E-15	2.87E-15	2.97E-15	3.09E-15	3.07E-15	3.17E-15	3.25E-15	3.32E-15	3.43E-15	3.59E-15	3.73E-15	3.56E-15	3.69E-15	3.79E-15	3.85E-15
9.327	30.601	0.3912	1.18E-15	1.22E-15	1.19E-15	1.23E-15	1.27E-15	1.31E-15	1.34E-15	1.38E-15	1.44E-15	1.43E-15	1.48E-15	1.52E-15	1.55E-15	1.6E-15	1.67E-15	1.74E-15	1.66E-15	1.72E-15	1.77E-15	1.8E-15
9.733	31.931	0.4192	5.25E-16	5.4E-16	5.27E-16	5.44E-16	5.62E-16	5.8E-16	5.95E-16	6.12E-16	6.36E-16	6.33E-16	6.54E-16	6.73E-16	6.89E-16	7.08E-16	7.37E-16	7.69E-16	7.35E-16	7.62E-16	7.86E-16	8.02E-16
10.166	33.352	0.4472	2.21E-16	2.27E-16	2.21E-16	2.28E-16	2.36E-16	2.43E-16	2.5E-16	2.57E-16	2.66E-16	2.65E-16	2.74E-16	2.83E-16	2.9E-16	2.97E-16	3.08E-16	3.22E-16	3.08E-16	3.19E-16	3.3E-16	3.38E-16
10.627	34.865	0.4752	8.8E-17	9.04E-17	8.82E-17	9.08E-17	9.38E-17	9.7E-17	9.97E-17	1.02E-16	1.06E-16	1.06E-16	1.09E-16	1.13E-16	1.16E-16	1.18E-16	1.22E-16	1.28E-16	1.23E-16	1.27E-16	1.31E-16	1.35E-16
11.116	36.470	0.5032	3.33E-17	3.42E-17	3.34E-17	3.44E-17	3.55E-17	3.67E-17	3.78E-17	3.88E-17	4E-17	4E-17	4.13E-17	4.26E-17	4.38E-17	4.49E-17	4.63E-17	4.81E-17	4.64E-17	4.79E-17	4.96E-17	5.11E-17
11.633	38.167	0.5312	1.2E-17	1.23E-17	1.2E-17	1.24E-17	1.28E-17	1.32E-17	1.36E-17	1.4E-17	1.44E-17	1.44E-17	1.48E-17	1.53E-17	1.58E-17	1.62E-17	1.66E-17	1.73E-17	1.67E-17	1.72E-17	1.78E-17	1.84E-17
12.179	39.956	0.5592	4.23E-18	4.33E-18	4.2E-18	4.34E-18	4.47E-18	4.61E-18	4.76E-18	4.89E-18	5.02E-18	4.98E-18	5.17E-18	5.34E-18	5.5E-18	5.64E-18	5.78E-18	5.98E-18	5.77E-18	5.97E-18	6.18E-18	6.38E-18
12.752	41.836	0.5872	1.72E-18	1.75E-18	1.65E-18	1.75E-18	1.79E-18	1.83E-18	1.87E-18	1.91E-18	1.95E-18	1.85E-18	1.99E-18	2.05E-18	2.1E-18	2.14E-18	2.18E-18	2.23E-18	2.09E-18	2.23E-18	2.29E-18	2.36E-18
13.353	43.809	0.6152	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
13.982	45.873	0.6432	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
14.639	48.029	0.6712	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
15.325	50.277	0.6992	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
16.038	52.617	0.7272	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
16.779	55.049	0.7552	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
17.548	57.573	0.7832	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
18.345	60.188	0.8112	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18

Fine Tailings Oxygen Consump

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s)																			
			TIME (YEAR)																			
			78	81	84	87	90	93	96	99	102	105	108	111	114	117	120	123	126	129	132	135
19.171	62.895	0.8392	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
20.024	65.695	0.8672	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
20.905	68.586	0.8952	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
21.814	71.569	0.9232	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
22.751	74.643	0.9512	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
23.717	77.810	0.9792	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
24.710	81.069	1.0072	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
25.731	84.419	1.0352	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
26.780	87.861	1.0632	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
27.857	91.395	1.0912	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
28.963	95.021	1.1192	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
30.096	98.739	1.1472	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
31.257	102.549	1.1752	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
32.446	106.450	1.2032	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
33.663	110.444	1.2312	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
34.909	114.529	1.2592	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
36.182	118.706	1.2872	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
37.483	122.975	1.3152	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
38.812	127.336	1.3432	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
40.160	131.760	1.3532	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
41.519	136.216	1.3632	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
42.887	140.704	1.3732	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
44.265	145.226	1.3832	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
45.653	149.781	1.3932	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
47.051	154.368	1.4032	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
47.758	156.686	0.01	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18

Fine Tailings Oxygen Consump

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s)									
			Time (years)									
			138	141	144	147	150	153	156	159	162	165
0.001	0.002	0.001	0	0	0	0	0	0	0	0	0	0
0.002	0.007	0.002	0	0	0	0	0	0	0	0	0	0
0.007	0.023	0.008	0	0	0	0	0	0	0	0	0	0
0.019	0.062	0.016	0	0	0	0	0	0	0	0	0	0
0.043	0.141	0.032	0	0	0	0	0	0	0	0	0	0
0.091	0.299	0.064	0	0	0	0	0	0	0	0	0	0
0.181	0.593	0.1152	0	0	0	0	0	0	0	0	0	0
0.298	0.979	0.1202	0	0	0	0	0	0	0	0	0	0
0.421	1.381	0.1252	0	0	0	0	0	0	0	0	0	0
0.549	1.800	0.1302	0	0	0	0	0	0	0	0	0	0
0.681	2.236	0.1352	0	0	0	0	0	0	0	0	0	0
0.819	2.687	0.1402	0	0	0	0	0	0	0	0	0	0
0.962	3.156	0.1452	4.48E-08	4.49E-08	4.49E-08	4.52E-08	0	0	0	0	0	0
1.110	3.640	0.1502	3.32E-08	3.32E-08	3.33E-08	3.34E-08	3.62E-08	3.92E-08	4.01E-08	4.03E-08	4.04E-08	4.04E-08
1.262	4.141	0.1552	2.44E-08	2.44E-08	2.44E-08	2.45E-08	2.61E-08	2.86E-08	2.93E-08	2.96E-08	2.96E-08	2.96E-08
1.420	4.658	0.1602	1.77E-08	1.77E-08	1.77E-08	1.78E-08	1.86E-08	2.07E-08	2.13E-08	2.15E-08	2.15E-08	2.15E-08
1.583	5.192	0.1652	1.27E-08	1.27E-08	1.27E-08	1.28E-08	1.32E-08	1.47E-08	1.53E-08	1.54E-08	1.55E-08	1.55E-08
1.750	5.742	0.1702	9.07E-09	9.08E-09	9.08E-09	9.08E-09	9.28E-09	1.04E-08	1.08E-08	1.1E-08	1.1E-08	1.1E-08
1.923	6.309	0.1752	6.39E-09	6.4E-09	6.4E-09	6.4E-09	6.48E-09	7.26E-09	7.61E-09	7.72E-09	7.76E-09	7.77E-09
2.101	6.892	0.1802	4.46E-09	4.46E-09	4.47E-09	4.47E-09	4.5E-09	5.01E-09	5.29E-09	5.38E-09	5.41E-09	5.42E-09
2.283	7.491	0.1852	3.08E-09	3.08E-09	3.09E-09	3.09E-09	3.1E-09	3.42E-09	3.64E-09	3.71E-09	3.74E-09	3.75E-09
2.471	8.107	0.1902	2.1E-09	2.11E-09	2.11E-09	2.11E-09	2.12E-09	2.31E-09	2.47E-09	2.53E-09	2.56E-09	2.56E-09
2.664	8.740	0.1952	1.42E-09	1.43E-09	1.43E-09	1.43E-09	1.43E-09	1.55E-09	1.66E-09	1.71E-09	1.73E-09	1.73E-09
2.862	9.388	0.2002	9.53E-10	9.57E-10	9.59E-10	9.59E-10	9.61E-10	1.02E-09	1.11E-09	1.14E-09	1.16E-09	1.16E-09
3.064	10.053	0.2052	6.32E-10	6.35E-10	6.36E-10	6.37E-10	6.38E-10	6.71E-10	7.28E-10	7.57E-10	7.68E-10	7.72E-10
3.272	10.735	0.2102	4.14E-10	4.17E-10	4.18E-10	4.18E-10	4.19E-10	4.36E-10	4.74E-10	4.95E-10	5.04E-10	5.07E-10
3.485	11.432	0.2152	2.69E-10	2.71E-10	2.72E-10	2.72E-10	2.73E-10	2.81E-10	3.05E-10	3.21E-10	3.27E-10	3.3E-10
3.702	12.147	0.2202	1.73E-10	1.75E-10	1.75E-10	1.75E-10	1.76E-10	1.8E-10	1.94E-10	2.05E-10	2.1E-10	2.12E-10
3.925	12.877	0.2252	1.1E-10	1.11E-10	1.12E-10	1.12E-10	1.12E-10	1.14E-10	1.23E-10	1.3E-10	1.34E-10	1.35E-10
4.153	13.624	0.2302	6.91E-11	7.02E-11	7.06E-11	7.06E-11	7.07E-11	7.15E-11	7.65E-11	8.16E-11	8.43E-11	8.54E-11
4.385	14.388	0.2352	4.3E-11	4.38E-11	4.41E-11	4.41E-11	4.42E-11	4.46E-11	4.73E-11	5.06E-11	5.25E-11	5.33E-11
4.623	15.168	0.2402	2.64E-11	2.71E-11	2.73E-11	2.73E-11	2.74E-11	2.75E-11	2.9E-11	3.1E-11	3.23E-11	3.29E-11
4.866	15.964	0.2452	1.61E-11	1.65E-11	1.67E-11	1.67E-11	1.68E-11	1.69E-11	1.76E-11	1.88E-11	1.97E-11	2.02E-11
5.114	16.777	0.2502	9.7E-12	1E-11	1.01E-11	1.01E-11	1.02E-11	1.02E-11	1.06E-11	1.13E-11	1.19E-11	1.22E-11
5.366	17.606	0.2552	5.78E-12	5.99E-12	6.09E-12	6.07E-12	6.12E-12	6.14E-12	6.3E-12	6.72E-12	7.1E-12	7.31E-12
5.624	18.451	0.2602	3.41E-12	3.55E-12	3.62E-12	3.6E-12	3.64E-12	3.65E-12	3.73E-12	3.96E-12	4.19E-12	4.33E-12
5.888	19.317	0.2677	1.99E-12	2.07E-12	2.12E-12	2.11E-12	2.14E-12	2.15E-12	2.18E-12	2.3E-12	2.44E-12	2.53E-12
6.159	20.208	0.2752	1.14E-12	1.19E-12	1.22E-12	1.22E-12	1.24E-12	1.24E-12	1.26E-12	1.32E-12	1.4E-12	1.46E-12
6.438	21.123	0.2827	6.43E-13	6.76E-13	6.96E-13	6.92E-13	7.04E-13	7.09E-13	7.15E-13	7.44E-13	7.91E-13	8.28E-13
6.725	22.063	0.2902	3.58E-13	3.77E-13	3.9E-13	3.87E-13	3.95E-13	3.98E-13	4.01E-13	4.15E-13	4.4E-13	4.62E-13
7.019	23.027	0.2977	1.96E-13	2.07E-13	2.15E-13	2.13E-13	2.18E-13	2.2E-13	2.22E-13	2.28E-13	2.41E-13	2.54E-13
7.320	24.016	0.3052	1.06E-13	1.12E-13	1.17E-13	1.15E-13	1.19E-13	1.2E-13	1.21E-13	1.24E-13	1.3E-13	1.37E-13
7.629	25.030	0.3127	5.66E-14	5.96E-14	6.23E-14	6.17E-14	6.36E-14	6.46E-14	6.51E-14	6.62E-14	6.93E-14	7.32E-14
7.946	26.068	0.3202	2.98E-14	3.13E-14	3.28E-14	3.25E-14	3.35E-14	3.42E-14	3.45E-14	3.5E-14	3.64E-14	3.84E-14
8.269	27.131	0.3277	1.55E-14	1.62E-14	1.7E-14	1.68E-14	1.74E-14	1.78E-14	1.8E-14	1.82E-14	1.88E-14	1.99E-14
8.601	28.218	0.3352	7.93E-15	8.27E-15	8.68E-15	8.62E-15	8.94E-15	9.16E-15	9.28E-15	9.37E-15	9.64E-15	1.01E-14
8.950	29.364	0.3632	3.92E-15	4.06E-15	4.26E-15	4.24E-15	4.4E-15	4.53E-15	4.6E-15	4.64E-15	4.75E-15	4.97E-15
9.327	30.601	0.3912	1.83E-15	1.89E-15	1.98E-15	1.97E-15	2.05E-15	2.12E-15	2.16E-15	2.18E-15	2.22E-15	2.31E-15
9.733	31.931	0.4192	8.14E-16	8.37E-16	8.74E-16	8.72E-16	9.07E-16	9.37E-16	9.57E-16	9.69E-16	9.84E-16	1.02E-15
10.166	33.352	0.4472	3.43E-16	3.51E-16	3.65E-16	3.66E-16	3.8E-16	3.93E-16	4.03E-16	4.09E-16	4.15E-16	4.27E-16
10.627	34.865	0.4752	1.37E-16	1.4E-16	1.45E-16	1.45E-16	1.51E-16	1.56E-16	1.61E-16	1.64E-16	1.66E-16	1.7E-16
11.116	36.470	0.5032	5.21E-17	5.31E-17	5.47E-17	5.5E-17	5.69E-17	5.91E-17	6.09E-17	6.22E-17	6.3E-17	6.43E-17
11.633	38.167	0.5312	1.88E-17	1.92E-17	1.97E-17	1.98E-17	2.04E-17	2.12E-17	2.19E-17	2.25E-17	2.28E-17	2.32E-17
12.179	39.956	0.5592	6.54E-18	6.66E-18	6.81E-18	6.82E-18	7.06E-18	7.31E-18	7.57E-18	7.77E-18	7.9E-18	8.03E-18
12.752	41.836	0.5872	2.41E-18	2.45E-18	2.49E-18	2.42E-18	2.55E-18	2.63E-18	2.71E-18	2.77E-18	2.82E-18	2.86E-18
13.353	43.809	0.6152	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
13.982	45.873	0.6432	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
14.639	48.029	0.6712	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
15.325	50.277	0.6992	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
16.038	52.617	0.7272	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
16.779	55.049	0.7552	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
17.548	57.573	0.7832	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
18.345	60.188	0.8112	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18

Fine Tailings Oxygen Consump

DEPTH (m)	Depth (ft)	Cell Sizes (m)	Oxygen Consumption (mol/m3/s)									
			Time (years)									
			138	141	144	147	150	153	156	159	162	165
19.171	62.895	0.8392	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
20.024	65.695	0.8672	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
20.905	68.586	0.8952	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
21.814	71.569	0.9232	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
22.751	74.643	0.9512	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
23.717	77.810	0.9792	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
24.710	81.069	1.0072	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
25.731	84.419	1.0352	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
26.780	87.861	1.0632	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
27.857	91.395	1.0912	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
28.963	95.021	1.1192	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
30.096	98.739	1.1472	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
31.257	102.549	1.1752	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
32.446	106.450	1.2032	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
33.663	110.444	1.2312	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
34.909	114.529	1.2592	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
36.182	118.706	1.2872	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
37.483	122.975	1.3152	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
38.812	127.336	1.3432	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
40.160	131.760	1.3532	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
41.519	136.216	1.3632	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
42.887	140.704	1.3732	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
44.265	145.226	1.3832	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
45.653	149.781	1.3932	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
47.051	154.368	1.4032	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18
47.758	156.686	0.01	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18	1.65E-18

Integrated Fine Tailings Sulphate generation Rates

	FIne Tailings Beaches	
	Cell 1E SO4	Cell 2E SO4
Year	mol/m2/year	mol/m2/year
1	0.000	0.136
2	0.000	0.170
3	0.000	0.186
4	0.000	0.130
5	0.000	0.136
6	0.000	0.170
7	0.000	0.186
8	0.000	0.130
9	0.239	0.136
10	0.005	0.170
11	0.003	0.186
12	0.005	0.130
13	0.005	0.136
14	0.005	0.170
15	0.005	0.186
16	0.239	0.130
17	0.005	0.136
18	0.003	0.170
19	0.005	0.186
20	0.005	0.130
21	0.005	0.136
22	0.005	0.170
25	0.239	0.186
28	0.005	0.130
31	0.003	0.000
34	0.005	0.000
37	0.005	0.000
40	0.005	0.000
43	0.005	0.000
46	0.239	0.000
49	0.005	0.239
52	0.003	0.005
55	0.005	0.003
58	0.187	0.005
61	0.846	0.005
64	0.859	0.005
67	0.861	0.005
70	0.862	0.239
73	0.862	0.005
76	0.862	0.003
79	0.862	0.005
82	0.862	0.005
85	0.862	0.005
88	0.812	0.005
91	0.761	0.239
94	0.666	0.005
97	0.669	0.003
100	0.676	0.005
103	0.540	0.005
106	0.543	0.005
109	0.546	0.005
112	0.566	0.239
115	0.442	0.005
118	0.445	0.003
121	0.452	0.005
124	0.480	0.187
127	0.360	0.846
130	0.361	0.859
133	0.361	0.861
136	0.361	0.862
139	0.272	0.862
142	0.298	0.862
145	0.302	0.862
148	0.302	0.862
151	0.302	0.862
154	0.303	0.812
157	0.312	0.761
160	0.249	0.666

Appendix D.3

Tailings Source Pore Water Concentrations

CELL 2E EMBANKMENT SOURCE CONCENTRATIONS

Year	SO4 mg/L	Sb mg/L	As mg/L	Cu mg/L	Ni mg/L	Zn mg/L	Co mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Ag mg/L	B mg/L	Be mg/L	Cd mg/L	Pb mg/L	Se mg/L	Tl mg/L
1	823	0.047	0.132	0.075	0.280	0.613	0.018	155	86	68	94	0.003	0.272	0.009	0.002	0.006	0.010	0.0030
2	667	0.039	0.109	0.059	0.217	0.482	0.014	142	71	64	73	0.003	0.262	0.007	0.002	0.006	0.008	0.0028
3	741	0.043	0.120	0.067	0.247	0.544	0.016	148	78	66	83	0.003	0.267	0.008	0.002	0.006	0.009	0.0029
4	894	0.050	0.143	0.082	0.309	0.673	0.019	160	93	70	103	0.003	0.277	0.010	0.003	0.006	0.011	0.0031
5	1090	0.060	0.173	0.102	0.388	0.837	0.024	175	111	75	129	0.004	0.290	0.012	0.003	0.007	0.014	0.0033
6	1214	0.067	0.192	0.115	0.438	0.941	0.027	185	123	79	146	0.005	0.299	0.014	0.004	0.007	0.015	0.0035
7	1273	0.070	0.201	0.121	0.462	0.990	0.029	189	129	80	154	0.005	0.303	0.015	0.004	0.008	0.016	0.0036
8	1360	0.074	0.214	0.130	0.498	1.063	0.031	196	137	83	165	0.005	0.309	0.016	0.004	0.008	0.017	0.0037
9	1549	0.084	0.243	0.149	0.574	1.222	0.035	211	155	88	190	0.006	0.321	0.019	0.004	0.009	0.020	0.0040
10	1755	0.095	0.274	0.170	0.658	1.394	0.040	226	175	93	218	0.006	0.335	0.021	0.005	0.009	0.023	0.0042
11	1780	0.096	0.278	0.172	0.668	1.416	0.041	228	177	94	221	0.006	0.337	0.022	0.005	0.009	0.023	0.0043
12	1791	0.096	0.280	0.173	0.672	1.425	0.041	229	178	94	223	0.006	0.337	0.022	0.005	0.009	0.023	0.0043
13	1841	0.099	0.287	0.179	0.693	1.467	0.042	233	183	95	229	0.007	0.341	0.023	0.005	0.010	0.024	0.0044
14	1786	0.096	0.279	0.173	0.670	1.421	0.041	229	178	94	222	0.006	0.337	0.022	0.005	0.009	0.023	0.0043
15	1809	0.097	0.282	0.175	0.679	1.440	0.041	231	180	94	225	0.007	0.339	0.022	0.005	0.009	0.023	0.0043
16	1854	0.100	0.289	0.180	0.698	1.478	0.043	234	184	96	231	0.007	0.342	0.023	0.005	0.010	0.024	0.0044
17	1862	0.100	0.291	0.181	0.701	1.484	0.043	235	185	96	232	0.007	0.342	0.023	0.005	0.010	0.024	0.0044
18	1903	0.102	0.297	0.185	0.718	1.519	0.044	238	189	97	238	0.007	0.345	0.023	0.005	0.010	0.025	0.0044
19	1922	0.103	0.300	0.187	0.726	1.535	0.044	239	191	97	240	0.007	0.346	0.024	0.005	0.010	0.025	0.0045
20	1943	0.104	0.303	0.189	0.734	1.552	0.045	241	193	98	243	0.007	0.348	0.024	0.006	0.010	0.025	0.0045
21	1965	0.105	0.306	0.191	0.743	1.571	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
22	1967	0.105	0.307	0.191	0.743	1.572	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
24	1967	0.105	0.307	0.191	0.744	1.573	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
27	1968	0.105	0.307	0.192	0.744	1.573	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
30	1968	0.105	0.307	0.192	0.744	1.573	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
33	1968	0.105	0.307	0.192	0.744	1.574	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
36	1968	0.105	0.307	0.192	0.744	1.574	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
39	1969	0.105	0.307	0.192	0.744	1.574	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
42	1969	0.105	0.307	0.192	0.744	1.574	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
45	1969	0.105	0.307	0.192	0.744	1.574	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
48	1969	0.105	0.307	0.192	0.744	1.574	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
51	1969	0.105	0.307	0.192	0.744	1.574	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
54	1969	0.105	0.307	0.192	0.744	1.574	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
57	1969	0.105	0.307	0.192	0.744	1.574	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
60	1969	0.105	0.307	0.192	0.744	1.574	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
63	1969	0.105	0.307	0.192	0.744	1.574	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
66	1769	0.091	0.270	0.180	0.717	1.484	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
69	1756	0.090	0.268	0.179	0.712	1.473	0.042	136	168	46	234	0.006	0.118	0.024	0.005	0.006	0.024	0.0024
72	1719	0.088	0.262	0.175	0.697	1.442	0.042	133	164	45	229	0.006	0.116	0.023	0.005	0.006	0.023	0.0023
75	1654	0.085	0.252	0.168	0.671	1.387	0.040	128	158	44	220	0.006	0.111	0.022	0.004	0.006	0.022	0.0022
78	1552	0.080	0.236	0.158	0.629	1.302	0.038	120	148	41	207	0.005	0.104	0.021	0.004	0.005	0.021	0.0021
81	1429	0.073	0.218	0.145	0.580	1.199	0.035	110	137	38	190	0.005	0.096	0.019	0.004	0.005	0.019	0.0019
84	1282	0.066	0.195	0.130	0.520	1.075	0.031	99	123	34	171	0.004	0.086	0.017	0.003	0.004	0.017	0.0017
87	1108	0.057	0.169	0.113	0.449	0.930	0.027	86	106	29	148	0.004	0.074	0.015	0.003	0.004	0.015	0.0015
90	930	0.048	0.142	0.095	0.377	0.780	0.023	72	89	25	124	0.003	0.063	0.013	0.003	0.003	0.013	0.0013
93	767	0.039	0.117	0.078	0.311	0.643	0.019	59	73	20	102	0.003	0.052	0.010	0.002	0.003	0.010	0.0010
96	628	0.032	0.096	0.064	0.255	0.527	0.015	48	60	17	84	0.002	0.042	0.008	0.002	0.002	0.008	0.0008
99	515	0.026	0.078	0.052	0.209	0.432	0.012	40	49	14	69	0.002	0.035	0.007	0.001	0.002	0.007	0.0007
102	423	0.022	0.065	0.043	0.172	0.355	0.010	33	40	11	56	0.001	0.028	0.006	0.001	0.001	0.006	0.0006
105	356	0.018	0.054	0.036	0.144	0.298	0.009	27	34	9	47	0.001	0.024	0.005	0.001	0.001	0.005	0.0005
108	290	0.015	0.044	0.030	0.118	0.243	0.007	22	28	8	39	0.001	0.020	0.004	0.001	0.001	0.004	0.0004
111	241	0.012	0.037	0.024	0.098	0.202	0.006	19	23	6	32	0.001	0.016	0.003	0.001	0.001	0.003	0.0003
114	187	0.010	0.029	0.019	0.076	0.157	0.005	14	18	5	25	0.001	0.013	0.003	0.001	0.001	0.003	0.0003
117	125	0.006	0.019	0.013	0.051	0.105	0.003	10	12	3	17	0.000	0.008	0.002	0.000	0.000	0.002	0.0002
120	92	0.005	0.014	0.009	0.037	0.077	0.002	7	9	2	12	0.000	0.006	0.001	0.000	0.000	0.001	0.0001
123	56	0.003	0.009	0.006	0.023	0.047	0.001	4	5	1	8	0.000	0.004	0.001	0.000	0.000	0.001	0.0001
126	24	0.001	0.004	0.002	0.010	0.020	0.001	2	2	1	3	0.000	0.002	0.000	0.000	0.000	0.000	0.0000
129	16	0.001	0.002	0.002	0.006	0.013	0.000	1	1	0	2	0.000	0.001	0.000	0.000	0.000	0.000	0.0000
132	6	0.000	0.001	0.001	0.003	0.005	0.000	0	1	0	1	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
135	0	0.000	0.000	0.000	0.000	0.000	0.000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
138	0	0.000	0.000	0.000	0.000	0.000	0.000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
141	0	0.000	0.000	0.000	0.000	0.000	0.000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
144	0	0.000	0.000	0.000	0.000	0.000	0.000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
147	0	0.000	0.000	0.000	0.000	0.000	0.000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
150	0	0.000	0.000	0.000	0.000	0.000	0.000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000

CELL 2E COARSE BEACH SOURCE CONCENTRATIONS

Year	SO4 mg/L	Sb mg/L	As mg/L	Cu mg/L	Ni mg/L	Zn mg/L	Co mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Ag mg/L	B mg/L	Be mg/L	Cd mg/L	Pb mg/L	Se mg/L	Tl mg/L
1	332	0.022	0.057	0.025	0.081	0.201	0.006	117	39	55	28	0.002	0.239	0.002	0.001	0.004	0.004	0.0023
2	332	0.022	0.057	0.025	0.081	0.201	0.006	117	39	55	28	0.002	0.239	0.002	0.001	0.004	0.004	0.0023
3	332	0.022	0.057	0.025	0.081	0.201	0.006	117	39	55	28	0.002	0.239	0.002	0.001	0.004	0.004	0.0023
4	332	0.022	0.058	0.025	0.081	0.201	0.006	117	39	55	28	0.002	0.239	0.002	0.001	0.004	0.004	0.0023
5	334	0.022	0.058	0.025	0.082	0.203	0.006	117	39	56	29	0.002	0.240	0.002	0.001	0.004	0.004	0.0023
6	340	0.022	0.059	0.026	0.084	0.208	0.006	117	40	56	29	0.002	0.240	0.002	0.001	0.004	0.004	0.0023
7	353	0.023	0.061	0.027	0.089	0.219	0.006	118	41	56	31	0.002	0.241	0.003	0.001	0.005	0.004	0.0024
8	385	0.024	0.066	0.030	0.102	0.246	0.007	121	44	57	35	0.002	0.243	0.003	0.001	0.005	0.004	0.0024
9	725	0.042	0.117	0.065	0.240	0.531	0.015	147	77	66	81	0.003	0.266	0.008	0.002	0.006	0.009	0.0029
10	1,040	0.058	0.165	0.097	0.368	0.795	0.023	171	107	74	123	0.004	0.287	0.012	0.003	0.007	0.013	0.0033
11	1,755	0.095	0.274	0.170	0.658	1.395	0.040	227	175	93	218	0.006	0.335	0.021	0.005	0.009	0.023	0.0042
12	2,401	0.128	0.373	0.236	0.919	1.936	0.056	276	237	110	304	0.009	0.378	0.030	0.007	0.011	0.031	0.0051
13	2,554	0.135	0.396	0.251	0.982	2.065	0.060	288	251	114	324	0.009	0.389	0.032	0.007	0.012	0.033	0.0053
14	2,610	0.138	0.404	0.257	1.004	2.111	0.061	292	257	116	332	0.009	0.393	0.033	0.007	0.012	0.034	0.0054
15	2,609	0.138	0.404	0.257	1.004	2.111	0.061	292	257	116	332	0.009	0.392	0.033	0.007	0.012	0.034	0.0054
16	2,557	0.136	0.397	0.251	0.983	2.068	0.060	288	252	114	325	0.009	0.389	0.032	0.007	0.012	0.034	0.0053
17	2,440	0.130	0.379	0.240	0.936	1.970	0.057	279	241	111	309	0.009	0.381	0.031	0.007	0.012	0.032	0.0052
18	2,261	0.120	0.351	0.221	0.863	1.819	0.052	266	223	106	285	0.008	0.369	0.028	0.006	0.011	0.030	0.0049
19	1,982	0.106	0.309	0.193	0.750	1.585	0.046	244	197	99	248	0.007	0.350	0.024	0.006	0.010	0.026	0.0045
20	1,556	0.084	0.244	0.150	0.577	1.228	0.035	211	156	88	191	0.006	0.322	0.019	0.004	0.009	0.020	0.0040
21	1,559	0.085	0.244	0.150	0.578	1.231	0.035	211	156	88	192	0.006	0.322	0.019	0.004	0.009	0.020	0.0040
22	3,434	0.181	0.530	0.341	1.339	2.803	0.081	356	336	137	442	0.012	0.448	0.044	0.010	0.015	0.045	0.0065
24	3,414	0.180	0.527	0.339	1.331	2.787	0.080	355	334	137	439	0.012	0.447	0.044	0.009	0.015	0.045	0.0065
27	3,446	0.181	0.532	0.342	1.343	2.813	0.081	357	337	138	443	0.012	0.449	0.044	0.010	0.015	0.045	0.0065
30	3,483	0.183	0.538	0.346	1.359	2.844	0.082	360	340	139	448	0.012	0.451	0.045	0.010	0.015	0.046	0.0066
33	3,532	0.186	0.545	0.351	1.378	2.885	0.083	364	345	140	455	0.012	0.454	0.045	0.010	0.015	0.047	0.0066
36	3,596	0.189	0.555	0.357	1.404	2.939	0.085	369	351	142	463	0.013	0.459	0.046	0.010	0.015	0.047	0.0067
39	3,571	0.188	0.551	0.355	1.394	2.917	0.084	367	349	141	460	0.012	0.457	0.046	0.010	0.015	0.047	0.0067
42	3,843	0.202	0.592	0.382	1.504	3.146	0.091	388	375	148	496	0.013	0.475	0.050	0.011	0.016	0.051	0.0070
45	4,031	0.211	0.621	0.401	1.580	3.303	0.095	402	393	153	521	0.014	0.488	0.052	0.011	0.017	0.053	0.0073
48	4,371	0.229	0.673	0.436	1.718	3.588	0.103	428	425	162	566	0.015	0.511	0.057	0.012	0.018	0.058	0.0078
51	4,864	0.254	0.748	0.486	1.918	4.002	0.115	467	472	175	632	0.017	0.544	0.063	0.013	0.020	0.065	0.0084
54	5,582	0.291	0.857	0.559	2.210	4.605	0.133	522	541	194	728	0.019	0.592	0.073	0.015	0.022	0.074	0.0094
57	5,582	0.291	0.857	0.559	2.210	4.605	0.133	522	541	194	728	0.019	0.592	0.073	0.015	0.022	0.074	0.0094
60	5,583	0.291	0.857	0.559	2.210	4.605	0.133	522	541	194	728	0.019	0.592	0.073	0.015	0.022	0.074	0.0094
63	5,583	0.291	0.857	0.559	2.210	4.605	0.133	522	541	194	728	0.019	0.592	0.073	0.015	0.022	0.074	0.0094
66	5,383	0.276	0.820	0.548	2.183	4.515	0.130	416	515	142	717	0.018	0.362	0.072	0.014	0.018	0.072	0.0072
69	5,383	0.276	0.820	0.548	2.183	4.515	0.130	416	515	142	717	0.018	0.362	0.072	0.014	0.018	0.072	0.0072
72	5,383	0.276	0.820	0.548	2.183	4.515	0.130	416	515	142	717	0.018	0.362	0.072	0.014	0.018	0.072	0.0072
75	5,383	0.276	0.820	0.548	2.183	4.515	0.130	416	515	142	717	0.018	0.362	0.072	0.014	0.018	0.072	0.0072
78	5,384	0.276	0.820	0.548	2.183	4.515	0.130	416	515	142	717	0.018	0.362	0.072	0.014	0.018	0.072	0.0072
81	5,385	0.276	0.820	0.548	2.183	4.516	0.130	416	515	142	717	0.018	0.362	0.072	0.014	0.018	0.072	0.0072
84	5,386	0.276	0.821	0.548	2.184	4.518	0.130	416	515	142	717	0.018	0.362	0.072	0.014	0.018	0.072	0.0072
87	5,390	0.276	0.821	0.548	2.186	4.520	0.130	416	516	142	718	0.018	0.362	0.072	0.014	0.018	0.072	0.0072
90	5,396	0.276	0.822	0.549	2.188	4.526	0.131	417	516	142	719	0.018	0.363	0.073	0.015	0.018	0.073	0.0073
93	5,406	0.277	0.824	0.550	2.192	4.534	0.131	417	517	143	720	0.018	0.363	0.073	0.015	0.018	0.073	0.0073
96	5,423	0.278	0.826	0.552	2.199	4.548	0.131	419	519	143	722	0.018	0.364	0.073	0.015	0.018	0.073	0.0073
99	5,287	0.271	0.805	0.538	2.144	4.434	0.128	408	506	139	704	0.018	0.355	0.071	0.014	0.018	0.071	0.0071
102	5,027	0.258	0.766	0.511	2.038	4.216	0.122	388	481	133	670	0.017	0.338	0.068	0.014	0.017	0.068	0.0068
105	4,745	0.243	0.723	0.483	1.924	3.980	0.115	366	454	125	632	0.016	0.319	0.064	0.013	0.016	0.064	0.0064
108	4,368	0.224	0.665	0.444	1.771	3.663	0.106	337	418	115	582	0.015	0.294	0.059	0.012	0.015	0.059	0.0059
111	4,188	0.215	0.638	0.426	1.698	3.513	0.101	323	401	110	558	0.014	0.281	0.056	0.011	0.014	0.056	0.0056
114	4,000	0.205	0.609	0.407	1.622	3.355	0.097	309	383	106	533	0.013	0.269	0.054	0.011	0.014	0.054	0.0054
117	3,885	0.199	0.592	0.395	1.575	3.258	0.094	300	372	102	517	0.013	0.261	0.052	0.010	0.013	0.052	0.0052
120	3,930	0.201	0.599	0.400	1.594	3.296	0.095	303	376	104	523	0.013	0.264	0.053	0.011	0.013	0.053	0.0053
123	3,885	0.199	0.592	0.395	1.575	3.258	0.094	300	372	103	517	0.013	0.261	0.052	0.010	0.013	0.052	0.0052
126	3,838	0.197	0.585	0.391	1.556	3.219	0.093	296	367	101	511	0.013	0.258	0.052	0.010	0.013	0.052	0.0052
129	3,607	0.185	0.549	0.367	1.462	3.025	0.087	278	345	95	480	0.012	0.242	0.048	0.010	0.012	0.048	0.0048
132	3,471	0.178	0.529	0.353	1.408	2.912	0.084	268	332	92	462	0.012	0.233	0.047	0.009	0.012	0.047	0.0047
135	3,256	0.167	0.496	0.331	1.320	2.731	0.079	251	311	86	434	0.011	0.219	0.044	0.009	0.011	0.044	0.0044
138	3,294	0.169	0.502	0.335	1.336	2.762	0.080	254	315	87	439	0.011	0.221	0.044	0.009	0.011	0.044	0.0044
141	3,175	0.163	0.484	0.323	1.288	2.663	0.077	245	304	84	423	0.011	0.213	0.043	0.009	0.011	0.043	0.0043
144	3,058	0.157	0.466	0.311	1.240	2.565	0.074	236	293	81	407	0.010	0.206	0.041	0.008	0.010	0.041	0.0041
147	3,022	0.155	0.460	0.308	1.225	2.535	0.073	233	289	80	403	0.010	0.203	0.041	0.008	0.010	0.041	0.0041
150	2,986	0.153	0.455	0.304	1.211	2.504	0.072	231	286	79	398	0.010	0.201	0.040	0.008	0.010	0.040	0.0040

CELL 2E FINES BEACH SOURCE CONCENTRATIONS

Year	SO4 mg/L	Sb mg/L	As mg/L	Cu mg/L	Ni mg/L	Zn mg/L	Co mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Ag mg/L	B mg/L	Be mg/L	Cd mg/L	Pb mg/L	Se mg/L	Tl mg/L
1	230	0.016	0.047	0.014	0.028	0.104	0.003	107	30	54	14	0.001	0.233	0.001	0.001	0.004	0.002	0.0022
2	238	0.017	0.050	0.015	0.029	0.107	0.003	107	31	54	15	0.001	0.234	0.001	0.001	0.004	0.002	0.0022
3	241	0.017	0.051	0.015	0.029	0.108	0.003	107	31	54	16	0.001	0.235	0.001	0.001	0.004	0.002	0.0022
4	229	0.016	0.047	0.014	0.028	0.103	0.003	107	30	54	14	0.001	0.233	0.001	0.001	0.004	0.002	0.0022
5	230	0.016	0.047	0.014	0.028	0.104	0.003	107	30	54	14	0.001	0.233	0.001	0.001	0.004	0.002	0.0022
6	238	0.017	0.050	0.015	0.029	0.107	0.003	107	31	54	15	0.001	0.234	0.001	0.001	0.004	0.002	0.0022
7	241	0.017	0.051	0.015	0.029	0.108	0.003	107	31	54	16	0.001	0.235	0.001	0.001	0.004	0.002	0.0022
8	229	0.016	0.047	0.014	0.028	0.103	0.003	107	30	54	14	0.001	0.233	0.001	0.001	0.004	0.002	0.0022
9	245	0.017	0.052	0.016	0.029	0.110	0.003	107	31	54	16	0.001	0.235	0.001	0.001	0.004	0.002	0.0022
10	256	0.018	0.056	0.017	0.029	0.115	0.003	107	33	55	17	0.001	0.236	0.001	0.001	0.004	0.003	0.0022
11	262	0.018	0.057	0.017	0.030	0.117	0.003	107	33	55	18	0.001	0.236	0.001	0.001	0.004	0.003	0.0022
12	243	0.017	0.051	0.016	0.029	0.109	0.003	107	31	54	16	0.001	0.235	0.001	0.001	0.004	0.002	0.0022
13	245	0.017	0.052	0.016	0.029	0.110	0.003	107	31	54	16	0.001	0.235	0.001	0.001	0.004	0.002	0.0022
14	256	0.018	0.056	0.017	0.029	0.115	0.003	107	33	55	17	0.001	0.236	0.001	0.001	0.004	0.003	0.0022
15	262	0.018	0.057	0.017	0.030	0.117	0.003	107	33	55	18	0.001	0.236	0.001	0.001	0.004	0.003	0.0022
16	243	0.017	0.051	0.016	0.029	0.109	0.003	107	31	54	16	0.001	0.235	0.001	0.001	0.004	0.002	0.0022
17	245	0.017	0.052	0.016	0.029	0.110	0.003	107	31	54	16	0.001	0.235	0.001	0.001	0.004	0.002	0.0022
18	256	0.018	0.056	0.017	0.029	0.115	0.003	107	33	55	17	0.001	0.236	0.001	0.001	0.004	0.003	0.0022
19	262	0.018	0.057	0.017	0.030	0.117	0.003	107	33	55	18	0.001	0.236	0.001	0.001	0.004	0.003	0.0022
20	243	0.017	0.051	0.016	0.029	0.109	0.003	107	31	54	16	0.001	0.235	0.001	0.001	0.004	0.002	0.0022
21	245	0.017	0.052	0.016	0.029	0.110	0.003	107	31	54	16	0.001	0.235	0.001	0.001	0.004	0.002	0.0022
22	280	0.019	0.063	0.019	0.030	0.125	0.003	108	35	56	20	0.001	0.238	0.002	0.001	0.004	0.003	0.0022
24	288	0.019	0.066	0.020	0.031	0.129	0.003	108	36	57	21	0.001	0.239	0.002	0.001	0.004	0.003	0.0023
27	262	0.018	0.057	0.017	0.030	0.117	0.003	107	33	55	18	0.001	0.236	0.001	0.001	0.004	0.003	0.0022
30	200	0.015	0.037	0.012	0.027	0.091	0.003	106	26	52	11	0.001	0.231	0.001	0.001	0.004	0.002	0.0022
33	200	0.015	0.037	0.012	0.027	0.091	0.003	106	26	52	11	0.001	0.231	0.001	0.001	0.004	0.002	0.0022
36	200	0.015	0.037	0.012	0.027	0.091	0.003	106	26	52	11	0.001	0.231	0.001	0.001	0.004	0.002	0.0022
39	200	0.015	0.037	0.012	0.027	0.091	0.003	106	26	52	11	0.001	0.231	0.001	0.001	0.004	0.002	0.0022
42	200	0.015	0.037	0.012	0.027	0.091	0.003	106	26	52	11	0.001	0.231	0.001	0.001	0.004	0.002	0.0022
45	200	0.015	0.037	0.012	0.027	0.091	0.003	106	26	52	11	0.001	0.231	0.001	0.001	0.004	0.002	0.0022
48	313	0.021	0.074	0.022	0.032	0.140	0.003	108	39	58	24	0.002	0.241	0.002	0.001	0.004	0.003	0.0023
51	202	0.015	0.038	0.012	0.027	0.092	0.003	106	27	52	11	0.001	0.231	0.001	0.001	0.004	0.002	0.0022
54	201	0.015	0.038	0.012	0.027	0.091	0.003	106	26	52	11	0.001	0.231	0.001	0.001	0.004	0.002	0.0022
57	202	0.015	0.038	0.012	0.027	0.092	0.003	106	27	52	11	0.001	0.231	0.001	0.001	0.004	0.002	0.0022
60	202	0.015	0.038	0.012	0.027	0.092	0.003	106	27	52	11	0.001	0.231	0.001	0.001	0.004	0.002	0.0022
63	202	0.015	0.038	0.012	0.027	0.092	0.003	106	27	52	11	0.001	0.231	0.001	0.001	0.004	0.002	0.0022
66	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.00002	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
69	113	0.0058	0.0368	0.0105	0.0045	0.0491	0.0009	2	13	6	13	0.000	0.011	0.001	0.000	0.000	0.001	0.0001
72	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.0000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
75	1	0.00006	0.00040	0.00011	0.00005	0.00053	0.00001	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
78	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.0000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
81	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.0000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
84	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.0000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
87	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.0000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
90	113	0.0058	0.0368	0.0105	0.0045	0.0491	0.0009	2	13	6	13	0.000	0.011	0.001	0.000	0.000	0.001	0.0001
93	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.0000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
96	1	0.0001	0.0004	0.0001	0.0000	0.0005	0.0000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
99	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.0000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
102	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.0000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
105	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.0000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
108	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.0000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
111	113	0.0058	0.0368	0.0105	0.0045	0.0491	0.0009	2	13	6	13	0.000	0.011	0.001	0.000	0.000	0.001	0.0001
114	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.0000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
117	1	0.0001	0.0004	0.0001	0.0000	0.0005	0.0000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
120	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.0000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
123	88	0.0045	0.0287	0.0082	0.0035	0.0383	0.0007	2	10	5	10	0.000	0.008	0.001	0.000	0.000	0.001	0.0001
126	399	0.0205	0.1299	0.0372	0.0158	0.1734	0.0031	7	45	22	46	0.001	0.038	0.005	0.001	0.001	0.005	0.0005
129	406	0.0208	0.1320	0.0378	0.0161	0.1762	0.0032	7	46	22	47	0.001	0.039	0.005	0.001	0.001	0.005	0.0005
132	407	0.0208	0.1323	0.0379	0.0161	0.1766	0.0032	7	46	22	47	0.001	0.039	0.005	0.001	0.001	0.005	0.0005
135	407	0.0209	0.1324	0.0379	0.0161	0.1767	0.0032	7	46	22	47	0.001	0.039	0.005	0.001	0.001	0.005	0.0005
138	407	0.0209	0.1324	0.0379	0.0161	0.1767	0.0032	7	46	22	47	0.001	0.039	0.005	0.001	0.001	0.005	0.0005
141	407	0.021	0.132	0.038	0.016	0.177	0.003	7	46	22	47	0.001	0.039	0.005	0.001	0.001	0.005	0.0005
144	407	0.021	0.132	0.038	0.016	0.177	0.003	7	46	22	47	0.001	0.039	0.005	0.001	0.001	0.005	0.0005
147	407	0.021	0.132	0.038	0.016	0.177	0.003	7	46	22	47	0.001	0.039	0.005	0.001	0.001	0.005	0.0005
150	407	0.021	0.132	0.038	0.016	0.177	0.003	7	46	22	47	0.001	0.039	0.005	0.001	0.001	0.005	0.0005

CELL 1E EMBANKMENT SOURCE CONCENTRATIONS

Year	SO4 mg/L	Sb mg/L	As mg/L	Cu mg/L	Ni mg/L	Zn mg/L	Co mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Ag mg/L	B mg/L	Be mg/L	Cd mg/L	Pb mg/L	Se mg/L	Tl mg/L
1																		
2																		
3																		
4																		
5																		
6																		
7																		
8																		
9	823	0.047	0.132	0.075	0.280	0.613	0.018	155	86	68	94	0.003	0.272	0.009	0.002	0.006	0.010	0.0030
10	502	0.030	0.083	0.042	0.150	0.344	0.010	130	55	60	51	0.002	0.251	0.005	0.002	0.005	0.006	0.0026
11	906	0.051	0.145	0.083	0.313	0.682	0.020	161	94	71	105	0.004	0.278	0.010	0.003	0.006	0.011	0.0031
12	1136	0.063	0.180	0.107	0.407	0.876	0.025	179	116	77	136	0.004	0.294	0.013	0.003	0.007	0.014	0.0034
13	1298	0.071	0.205	0.123	0.472	1.012	0.029	191	131	81	157	0.005	0.304	0.015	0.004	0.008	0.017	0.0036
14	1488	0.081	0.234	0.143	0.549	1.171	0.034	206	149	86	182	0.005	0.317	0.018	0.004	0.008	0.019	0.0039
15	1421	0.077	0.223	0.136	0.522	1.114	0.032	201	143	84	173	0.005	0.313	0.017	0.004	0.008	0.018	0.0038
16	1481	0.080	0.232	0.142	0.546	1.165	0.034	205	149	86	181	0.005	0.317	0.018	0.004	0.008	0.019	0.0039
17	1618	0.088	0.253	0.156	0.602	1.280	0.037	216	162	89	200	0.006	0.326	0.020	0.005	0.009	0.021	0.0041
18	1744	0.094	0.273	0.169	0.653	1.385	0.040	226	174	93	216	0.006	0.334	0.021	0.005	0.009	0.023	0.0042
19	1903	0.102	0.297	0.185	0.718	1.519	0.044	238	189	97	238	0.007	0.345	0.023	0.005	0.010	0.025	0.0044
20	1924	0.103	0.300	0.187	0.726	1.537	0.044	240	191	97	240	0.007	0.346	0.024	0.005	0.010	0.025	0.0045
21	1943	0.104	0.303	0.189	0.734	1.553	0.045	241	193	98	243	0.007	0.348	0.024	0.006	0.010	0.025	0.0045
22	1963	0.105	0.306	0.191	0.742	1.569	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
24	1965	0.105	0.306	0.191	0.743	1.571	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
27	1966	0.105	0.306	0.191	0.743	1.572	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
30	1967	0.105	0.307	0.191	0.744	1.573	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
33	1968	0.105	0.307	0.191	0.744	1.573	0.045	243	195	99	246	0.007	0.349	0.024	0.006	0.010	0.026	0.0045
36	1768	0.091	0.269	0.180	0.717	1.483	0.043	136	169	47	235	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
39	1768	0.091	0.269	0.180	0.717	1.483	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
42	1768	0.091	0.269	0.180	0.717	1.483	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
45	1768	0.091	0.269	0.180	0.717	1.483	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
48	1769	0.091	0.269	0.180	0.717	1.483	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
51	1769	0.091	0.269	0.180	0.717	1.483	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
54	1769	0.091	0.269	0.180	0.717	1.483	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
57	1769	0.091	0.269	0.180	0.717	1.483	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
60	1769	0.091	0.269	0.180	0.717	1.483	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
63	1769	0.091	0.269	0.180	0.717	1.483	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
66	1769	0.091	0.269	0.180	0.717	1.483	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
69	1769	0.091	0.269	0.180	0.717	1.483	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
72	1769	0.091	0.269	0.180	0.717	1.483	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
75	1769	0.091	0.269	0.180	0.717	1.483	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
78	1769	0.091	0.269	0.180	0.717	1.483	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
81	1769	0.091	0.269	0.180	0.717	1.483	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
84	1769	0.091	0.269	0.180	0.717	1.484	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
87	1769	0.091	0.270	0.180	0.717	1.484	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
90	1770	0.091	0.270	0.180	0.718	1.484	0.043	137	169	47	236	0.006	0.119	0.024	0.005	0.006	0.024	0.0024
93	1712	0.088	0.261	0.174	0.694	1.436	0.041	132	164	45	228	0.006	0.115	0.023	0.005	0.006	0.023	0.0023
96	1606	0.082	0.245	0.163	0.651	1.347	0.039	124	154	42	214	0.005	0.108	0.022	0.004	0.005	0.022	0.0022
99	1491	0.076	0.227	0.152	0.605	1.251	0.036	115	143	39	199	0.005	0.100	0.020	0.004	0.005	0.020	0.0020
102	1294	0.066	0.197	0.132	0.525	1.086	0.031	100	124	34	172	0.004	0.087	0.017	0.003	0.004	0.017	0.0017
105	1132	0.058	0.172	0.115	0.459	0.949	0.027	87	108	30	151	0.004	0.076	0.015	0.003	0.004	0.015	0.0015
108	957	0.049	0.146	0.097	0.388	0.803	0.023	74	92	25	128	0.003	0.064	0.013	0.003	0.003	0.013	0.0013
111	732	0.037	0.111	0.074	0.297	0.614	0.018	56	70	19	97	0.002	0.049	0.010	0.002	0.002	0.010	0.0010
114	580	0.030	0.088	0.059	0.235	0.487	0.014	45	55	15	77	0.002	0.039	0.008	0.002	0.002	0.008	0.0008
117	417	0.021	0.064	0.042	0.169	0.350	0.010	32	40	11	56	0.001	0.028	0.006	0.001	0.001	0.006	0.0006
120	239	0.012	0.036	0.024	0.097	0.200	0.006	18	23	6	32	0.001	0.016	0.003	0.001	0.001	0.003	0.0003
123	163	0.008	0.025	0.017	0.066	0.136	0.004	13	16	4	22	0.001	0.011	0.002	0.000	0.001	0.002	0.0002
126	82	0.004	0.012	0.008	0.033	0.068	0.002	6	8	2	11	0.000	0.005	0.001	0.000	0.000	0.001	0.0001
129	19	0.001	0.003	0.002	0.008	0.016	0.000	1	2	1	3	0.000	0.001	0.000	0.000	0.000	0.000	0.0000
132	12	0.001	0.002	0.001	0.005	0.010	0.000	1	1	0	2	0.000	0.001	0.000	0.000	0.000	0.000	0.0000
135	5	0.000	0.001	0.001	0.002	0.004	0.000	0	0	0	1	0.000	0.000	0.000	0.000	0.000	0.000	0.0000
138	0	0.000	0.000	0.000	0.000	0.000	0.000	0	0	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.0000

CELL 1E Coarse Beach CONCENTRATIONS

Year	SO4 mg/L	Sb mg/L	As mg/L	Cu mg/L	Ni mg/L	Zn mg/L	Co mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Ag mg/L	B mg/L	Be mg/L	Cd mg/L	Pb mg/L	Se mg/L	Tl mg/L
1																		
2																		
3																		
4																		
5																		
6																		
7																		
8																		
9	462	0.0283	0.0774	0.0383	0.1336	0.3106	0.0089	127	51	59	46	0.0020	0.2482	0.0041	0.0015	0.0049	0.0054	0.0025
10	515	0.0310	0.0854	0.0437	0.1548	0.3546	0.0102	131	56	60	53	0.0022	0.2517	0.0048	0.0017	0.0050	0.0061	0.0026
11	749	0.0430	0.1210	0.0674	0.2496	0.5505	0.0158	149	79	66	84	0.0030	0.2674	0.0079	0.0023	0.0058	0.0092	0.0029
12	1407	0.0767	0.2213	0.1345	0.5167	1.1030	0.0318	200	142	84	172	0.0052	0.3117	0.0168	0.0041	0.0081	0.0181	0.0038
13	2020	0.1081	0.3147	0.1969	0.7654	1.6174	0.0466	247	200	100	253	0.0073	0.3529	0.0250	0.0057	0.0101	0.0263	0.0046
14	2164	0.1155	0.3365	0.2114	0.8234	1.7374	0.0501	258	214	104	272	0.0078	0.3625	0.0269	0.0061	0.0106	0.0282	0.0048
15	2232	0.1190	0.3470	0.2184	0.8513	1.7951	0.0517	263	221	106	282	0.0080	0.3672	0.0279	0.0063	0.0108	0.0291	0.0049
16	2271	0.1210	0.3529	0.2224	0.8669	1.8274	0.0527	266	224	107	287	0.0081	0.3698	0.0284	0.0064	0.0110	0.0297	0.0049
17	2295	0.1222	0.3565	0.2248	0.8765	1.8472	0.0532	268	227	107	290	0.0082	0.3713	0.0287	0.0064	0.0111	0.0300	0.0050
18	2172	0.1159	0.3379	0.2123	0.8269	1.7447	0.0503	259	215	104	274	0.0078	0.3631	0.0270	0.0061	0.0106	0.0283	0.0048
19	1989	0.1065	0.3099	0.1937	0.7526	1.5910	0.0458	245	197	99	249	0.0072	0.3508	0.0246	0.0056	0.0100	0.0259	0.0046
20	1707	0.0921	0.2669	0.1649	0.6381	1.3542	0.0390	223	170	92	212	0.0062	0.3318	0.0208	0.0049	0.0091	0.0221	0.0042
21	1279	0.0701	0.2018	0.1214	0.4647	0.9956	0.0287	190	130	80	155	0.0048	0.3031	0.0150	0.0037	0.0076	0.0163	0.0036
22	2767	0.1464	0.4284	0.2728	1.0679	2.2431	0.0647	305	272	120	353	0.0098	0.4031	0.0350	0.0077	0.0126	0.0363	0.0056
24	2769	0.1465	0.4288	0.2731	1.0690	2.2453	0.0647	305	272	120	353	0.0098	0.4032	0.0351	0.0077	0.0127	0.0364	0.0056
27	2771	0.1466	0.4291	0.2733	1.0697	2.2469	0.0648	305	272	120	353	0.0098	0.4034	0.0351	0.0077	0.0127	0.0364	0.0056
30	2772	0.1467	0.4293	0.2734	1.0703	2.2481	0.0648	305	272	120	353	0.0098	0.4035	0.0351	0.0077	0.0127	0.0364	0.0056
33	2773	0.1467	0.4295	0.2735	1.0707	2.2489	0.0648	305	272	120	354	0.0098	0.4035	0.0351	0.0077	0.0127	0.0364	0.0056
36	2574	0.1319	0.3922	0.2619	1.0438	2.1590	0.0623	199	246	68	343	0.0087	0.1730	0.0346	0.0069	0.0087	0.0346	0.0035
39	2575	0.1319	0.3923	0.2620	1.0441	2.1595	0.0623	199	246	68	343	0.0087	0.1731	0.0346	0.0069	0.0087	0.0346	0.0035
42	2575	0.1319	0.3923	0.2620	1.0442	2.1599	0.0623	199	246	68	343	0.0087	0.1731	0.0346	0.0069	0.0087	0.0346	0.0035
45	2576	0.1320	0.3924	0.2621	1.0444	2.1601	0.0623	199	246	68	343	0.0087	0.1731	0.0346	0.0069	0.0087	0.0346	0.0035
48	2762	0.1415	0.4208	0.2810	1.1200	2.3165	0.0668	213	264	73	368	0.0093	0.1856	0.0371	0.0074	0.0093	0.0371	0.0037
51	3101	0.1589	0.4725	0.3156	1.2576	2.6011	0.0750	239	297	82	413	0.0104	0.2084	0.0417	0.0083	0.0105	0.0417	0.0042
54	3594	0.1842	0.5475	0.3657	1.4574	3.0143	0.0870	277	344	95	479	0.0121	0.2416	0.0483	0.0097	0.0121	0.0483	0.0048
57	4312	0.2209	0.6569	0.4388	1.7485	3.6166	0.1043	333	412	114	574	0.0145	0.2898	0.0580	0.0116	0.0146	0.0580	0.0058
60	4312	0.2210	0.6570	0.4388	1.7486	3.6168	0.1043	333	413	114	574	0.0145	0.2898	0.0580	0.0116	0.0146	0.0580	0.0058
63	4313	0.2210	0.6570	0.4388	1.7487	3.6170	0.1043	333	413	114	574	0.0145	0.2898	0.0580	0.0116	0.0146	0.0580	0.0058
66	4313	0.2210	0.6570	0.4388	1.7488	3.6171	0.1043	333	413	114	574	0.0145	0.2899	0.0580	0.0116	0.0146	0.0580	0.0058
69	4313	0.2210	0.6570	0.4389	1.7488	3.6171	0.1043	333	413	114	574	0.0145	0.2899	0.0580	0.0116	0.0146	0.0580	0.0058
72	4313	0.2210	0.6570	0.4389	1.7488	3.6172	0.1043	333	413	114	574	0.0145	0.2899	0.0580	0.0116	0.0146	0.0580	0.0058
75	4313	0.2210	0.6570	0.4389	1.7488	3.6172	0.1043	333	413	114	574	0.0145	0.2899	0.0580	0.0116	0.0146	0.0580	0.0058
78	4313	0.2210	0.6570	0.4389	1.7488	3.6172	0.1043	333	413	114	574	0.0145	0.2899	0.0580	0.0116	0.0146	0.0580	0.0058
81	4313	0.2210	0.6570	0.4389	1.7488	3.6172	0.1043	333	413	114	574	0.0145	0.2899	0.0580	0.0116	0.0146	0.0580	0.0058
84	4313	0.2210	0.6570	0.4389	1.7488	3.6172	0.1043	333	413	114	574	0.0145	0.2899	0.0580	0.0116	0.0146	0.0580	0.0058
87	4313	0.2210	0.6571	0.4389	1.7489	3.6173	0.1044	333	413	114	574	0.0145	0.2899	0.0580	0.0116	0.0146	0.0580	0.0058
90	4313	0.2210	0.6571	0.4389	1.7490	3.6176	0.1044	333	413	114	575	0.0145	0.2899	0.0580	0.0116	0.0146	0.0580	0.0058
93	4315	0.2211	0.6574	0.4391	1.7497	3.6190	0.1044	333	413	114	575	0.0145	0.2900	0.0580	0.0116	0.0146	0.0580	0.0058
96	4319	0.2213	0.6580	0.4395	1.7515	3.6226	0.1045	333	413	114	575	0.0145	0.2903	0.0581	0.0116	0.0146	0.0581	0.0058
99	4328	0.2218	0.6593	0.4404	1.7549	3.6298	0.1047	334	414	114	576	0.0145	0.2909	0.0582	0.0116	0.0146	0.0582	0.0058
102	4183	0.2143	0.6373	0.4257	1.6963	3.5085	0.1012	323	400	110	557	0.0141	0.2811	0.0562	0.0112	0.0141	0.0562	0.0056
105	3913	0.2005	0.5962	0.3982	1.5868	3.2820	0.0947	302	374	103	521	0.0132	0.2630	0.0526	0.0105	0.0132	0.0526	0.0053
108	3621	0.1855	0.5516	0.3684	1.4681	3.0366	0.0876	280	346	96	482	0.0122	0.2433	0.0487	0.0097	0.0122	0.0487	0.0049
111	3231	0.1656	0.4923	0.3288	1.3103	2.7102	0.0782	249	309	85	430	0.0109	0.2172	0.0434	0.0087	0.0109	0.0434	0.0043
114	3038	0.1557	0.4629	0.3092	1.2320	2.5482	0.0735	235	291	80	405	0.0102	0.2042	0.0408	0.0082	0.0103	0.0408	0.0041
117	2837	0.1454	0.4322	0.2887	1.1504	2.3794	0.0686	219	271	75	378	0.0095	0.1907	0.0381	0.0076	0.0096	0.0381	0.0038
120	2708	0.1387	0.4125	0.2755	1.0980	2.2711	0.0655	209	259	71	361	0.0091	0.1820	0.0364	0.0073	0.0091	0.0364	0.0036
123	2740	0.1404	0.4175	0.2789	1.1112	2.2985	0.0663	212	262	72	365	0.0092	0.1842	0.0368	0.0074	0.0093	0.0368	0.0037
126	2683	0.1375	0.4087	0.2730	1.0879	2.2501	0.0649	207	257	71	357	0.0090	0.1803	0.0361	0.0072	0.0091	0.0361	0.0036
129	2624	0.1344	0.3997	0.2670	1.0639	2.2006	0.0635	203	251	69	349	0.0088	0.1763	0.0353	0.0071	0.0089	0.0353	0.0035
132	2379	0.1219	0.3624	0.2421	0.9646	1.9951	0.0576	184	228	63	317	0.0080	0.1599	0.0320	0.0064	0.0080	0.0320	0.0032
135	2229	0.1142	0.3396	0.2268	0.9039	1.8696	0.0539	172	213	59	297	0.0075	0.1498	0.0300	0.0060	0.0075	0.0300	0.0030
138	1998	0.1024	0.3044	0.2033	0.8102	1.6758	0.0483	154	191	53	266	0.0067	0.1343	0.0269	0.0054	0.0067	0.0269	0.0027

CELL 1E FINEBeach CONCENTRATIONS

Year	SO4 mg/L	Sb mg/L	As mg/L	Cu mg/L	Ni mg/L	Zn mg/L	Co mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Ag mg/L	B mg/L	Be mg/L	Cd mg/L	Pb mg/L	Se mg/L	Tl mg/L
1																		
2																		
3																		
4																		
5																		
6																		
7																		
8																		
9	279	0.0189	0.0632	0.0190	0.0303	0.1250	0.0032	107.793	35.282	56.382	19.920	0.0014	0.2381	0.0015	0.0010	0.0042	0.0028	0.0022
10	202	0.0149	0.0379	0.0118	0.0272	0.0912	0.0026	106.467	26.490	52.096	11.008	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0022
11	201	0.0149	0.0377	0.0117	0.0272	0.0909	0.0026	106.452	26.393	52.049	10.910	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
12	202	0.0149	0.0380	0.0118	0.0272	0.0913	0.0026	106.467	26.492	52.097	11.010	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0022
13	202	0.0149	0.0380	0.0118	0.0272	0.0913	0.0026	106.467	26.492	52.097	11.010	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0022
14	202	0.0149	0.0380	0.0118	0.0272	0.0913	0.0026	106.467	26.492	52.097	11.010	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0022
15	202	0.0149	0.0380	0.0118	0.0272	0.0913	0.0026	106.467	26.492	52.097	11.010	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0022
16	279	0.0189	0.0632	0.0190	0.0303	0.1250	0.0032	107.793	35.282	56.382	19.920	0.0014	0.2381	0.0015	0.0010	0.0042	0.0028	0.0022
17	202	0.0149	0.0379	0.0118	0.0272	0.0912	0.0026	106.467	26.490	52.096	11.008	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0022
18	201	0.0149	0.0377	0.0117	0.0272	0.0909	0.0026	106.452	26.393	52.049	10.910	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
19	202	0.0149	0.0380	0.0118	0.0272	0.0913	0.0026	106.467	26.492	52.097	11.010	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0022
20	202	0.0149	0.0380	0.0118	0.0272	0.0913	0.0026	106.467	26.492	52.097	11.010	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0022
21	202	0.0149	0.0380	0.0118	0.0272	0.0913	0.0026	106.467	26.492	52.097	11.010	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0022
22	202	0.0150	0.0382	0.0118	0.0273	0.0916	0.0026	106.480	26.574	52.137	11.094	0.0012	0.2308	0.0006	0.0008	0.0040	0.0019	0.0022
24	313	0.0207	0.0742	0.0221	0.0317	0.1396	0.0034	108.370	39.101	58.244	23.791	0.0015	0.2414	0.0019	0.0011	0.0043	0.0032	0.0023
27	202	0.0150	0.0382	0.0118	0.0273	0.0916	0.0026	106.479	26.572	52.136	11.092	0.0012	0.2308	0.0006	0.0008	0.0040	0.0019	0.0022
30	201	0.0149	0.0378	0.0117	0.0272	0.0910	0.0026	106.458	26.435	52.069	10.952	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0022
33	202	0.0150	0.0382	0.0118	0.0273	0.0916	0.0026	106.480	26.574	52.137	11.094	0.0012	0.2308	0.0006	0.0008	0.0040	0.0019	0.0022
36	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.0000	0.042	0.277	0.135	0.281	0.0000	0.0002	0.0000	0.0000	0.0000	0.0000	0.0000
39	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.0000	0.042	0.277	0.135	0.281	0.0000	0.0002	0.0000	0.0000	0.0000	0.0000	0.0000
42	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.0000	0.042	0.277	0.135	0.281	0.0000	0.0002	0.0000	0.0000	0.0000	0.0000	0.0000
45	113	0.0058	0.0368	0.0105	0.0045	0.0491	0.0009	1.932	12.804	6.242	12.978	0.0003	0.0108	0.0014	0.0003	0.0004	0.0014	0.0001
48	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.0000	0.042	0.275	0.134	0.279	0.0000	0.0002	0.0000	0.0000	0.0000	0.0000	0.0000
51	1	0.0001	0.0004	0.0001	0.00005	0.0005	0.00001	0.021	0.138	0.067	0.139	0.0000	0.0001	0.0000	0.0000	0.0000	0.0000	0.0000
54	2	0.0001	0.0008	0.0002	0.0001	0.0011	0.00002	0.042	0.277	0.135	0.281	0.0000	0.0002	0.0000	0.0000	0.0000	0.0000	0.0000
57	88	0.0045	0.0287	0.0082	0.0035	0.0383	0.0007	1.506	9.983	4.866	10.119	0.0003	0.0084	0.0011	0.0002	0.0003	0.0011	0.0001
60	399	0.0205	0.1299	0.0372	0.0158	0.1734	0.0031	6.822	45.212	22.040	45.827	0.0012	0.0381	0.0049	0.0010	0.0012	0.0049	0.0005
63	406	0.0208	0.1320	0.0378	0.0161	0.1762	0.0032	6.931	45.935	22.393	46.560	0.0013	0.0387	0.0050	0.0010	0.0013	0.0050	0.0005
66	407	0.0208	0.1323	0.0379	0.0161	0.1766	0.0032	6.949	46.050	22.449	46.677	0.0013	0.0388	0.0050	0.0010	0.0013	0.0050	0.0005
69	407	0.0209	0.1324	0.0379	0.0161	0.1767	0.0032	6.952	46.071	22.459	46.699	0.0013	0.0388	0.0050	0.0010	0.0013	0.0050	0.0005
72	407	0.0209	0.1324	0.0379	0.0161	0.1767	0.0032	6.952	46.076	22.461	46.703	0.0013	0.0388	0.0050	0.0010	0.0013	0.0050	0.0005
75	407	0.0209	0.1324	0.0379	0.0161	0.1767	0.0032	6.953	46.077	22.462	46.704	0.0013	0.0388	0.0050	0.0010	0.0013	0.0050	0.0005
78	407	0.0209	0.1324	0.0379	0.0161	0.1767	0.0032	6.953	46.077	22.462	46.705	0.0013	0.0388	0.0050	0.0010	0.0013	0.0050	0.0005
81	407	0.0209	0.1324	0.0379	0.0161	0.1767	0.0032	6.953	46.078	22.462	46.705	0.0013	0.0388	0.0050	0.0010	0.0013	0.0050	0.0005
84	407	0.0209	0.1324	0.0379	0.0161	0.1767	0.0032	6.954	46.084	22.465	46.712	0.0013	0.0388	0.0050	0.0010	0.0013	0.0050	0.0005
87	384	0.0197	0.1248	0.0357	0.0152	0.1666	0.0030	6.553	43.429	21.171	44.020	0.0012	0.0366	0.0047	0.0009	0.0012	0.0047	0.0005
90	360	0.0184	0.1169	0.0335	0.0143	0.1561	0.0028	6.140	40.691	19.836	41.245	0.0011	0.0343	0.0044	0.0009	0.0011	0.0044	0.0004
93	315	0.0161	0.1023	0.0293	0.0125	0.1366	0.0025	5.375	35.625	17.367	36.110	0.0010	0.0300	0.0039	0.0008	0.0010	0.0039	0.0004
96	316	0.0162	0.1027	0.0294	0.0125	0.1371	0.0025	5.395	35.755	17.430	36.242	0.0010	0.0301	0.0039	0.0008	0.0010	0.0039	0.0004
99	319	0.0164	0.1039	0.0297	0.0127	0.1386	0.0025	5.455	36.152	17.623	36.644	0.0010	0.0305	0.0039	0.0008	0.0010	0.0039	0.0004
102	255	0.0131	0.0829	0.0237	0.0101	0.1107	0.0020	4.354	28.854	14.066	29.247	0.0008	0.0243	0.0031	0.0006	0.0008	0.0031	0.0003
105	257	0.0131	0.0834	0.0239	0.0102	0.1114	0.0020	4.382	29.044	14.158	29.439	0.0008	0.0245	0.0032	0.0006	0.0008	0.0032	0.0003
108	258	0.0132	0.0839	0.0240	0.0102	0.1120	0.0020	4.406	29.200	14.234	29.597	0.0008	0.0246	0.0032	0.0006	0.0008	0.0032	0.0003
111	267	0.0137	0.0870	0.0249	0.0106	0.1161	0.0021	4.567	30.270	14.756	30.682	0.0008	0.0255	0.0033	0.0007	0.0008	0.0033	0.0003
114	209	0.0107	0.0680	0.0194	0.0083	0.0907	0.0016	3.569	23.654	11.531	23.976	0.0006	0.0199	0.0026	0.0005	0.0007	0.0026	0.0003
117	210	0.0108	0.0684	0.0196	0.0083	0.0913	0.0016	3.593	23.811	11.607	24.135	0.0006	0.0201	0.0026	0.0005	0.0007	0.0026	0.0003
120	213	0.0109	0.0694	0.0199	0.0085	0.0926	0.0017	3.643	24.145	11.770	24.473	0.0007	0.0203	0.0026	0.0005	0.0007	0.0026	0.0003
123	227	0.0116	0.0737	0.0211	0.0090	0.0983	0.0018	3.869	25.644	12.501	25.993	0.0007	0.0216	0.0028	0.0006	0.0007	0.0028	0.0003
126	170	0.0087	0.0553	0.0158	0.0067	0.0738	0.0013	2.905	19.254	9.386	19.516	0.0005	0.0162	0.0021	0.0004	0.0005	0.0021	0.0002
129	170	0.0087	0.0554	0.0159	0.0068	0.0740	0.0013	2.910	19.283	9.400	19.545	0.0005	0.0163	0.0021	0.0004	0.0005	0.0021	0.0002
132	170	0.0087	0.0554	0.0159	0.0068	0.0740	0.0013	2.910	19.288	9.403	19.551	0.0005	0.0163	0.0021	0.0004	0.0005	0.0021	0.0002
135	171	0.0087	0.0555	0.0159	0.0068	0.0741	0.0013	2.915	19.316	9.416	19.579	0.0005	0.0163	0.0021	0.0004	0.0005	0.0021	0.0002
138	128	0.0066	0.0417	0.0119	0.0051	0.0557	0.0010	2.191	14.520	7.078	14.718	0.0004	0.0122	0.0016	0.0003	0.0004	0.0016	0.0002

Appendix D.4

Seepage Water Quality Results

Cell 2E Seepage Concentrations in Collection Drains

Year	SO4 mg/L	Sb mg/L	As mg/L	Cu mg/L	Ni mg/L	Zn mg/L	Co mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Ag mg/L	B mg/L	Be mg/L	Cd mg/L	Pb mg/L	Se mg/L	Tl mg/L
1	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
2	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
3	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
4	196	0.0046	0.0069	0.0043	0.0089	0.0266	0.0017	73	77	94	16	0.0011	0.2336	0.0004	0.0003	0.0069	0.0021	0.0020
5	205	0.0058	0.0104	0.0058	0.0139	0.0387	0.0020	76	75	91	17	0.0011	0.2340	0.0005	0.0003	0.0067	0.0022	0.0020
6	211	0.0065	0.0126	0.0067	0.0170	0.0463	0.0021	78	73	89	17	0.0011	0.2342	0.0006	0.0004	0.0066	0.0023	0.0021
7	215	0.0071	0.0142	0.0073	0.0192	0.0517	0.0023	79	72	88	18	0.0011	0.2344	0.0007	0.0004	0.0065	0.0023	0.0021
8	258	0.0165	0.0422	0.0148	0.0400	0.1171	0.0033	109	29	53	15	0.0013	0.2327	0.0010	0.0009	0.0041	0.0023	0.0022
9	260	0.0166	0.0427	0.0151	0.0411	0.1195	0.0034	109	30	53	15	0.0013	0.2329	0.0010	0.0009	0.0041	0.0023	0.0022
10	262	0.0168	0.0432	0.0155	0.0425	0.1222	0.0035	109	30	53	16	0.0013	0.2331	0.0010	0.0009	0.0041	0.0023	0.0022
11	267	0.0171	0.0440	0.0160	0.0444	0.1263	0.0036	110	30	53	17	0.0013	0.2335	0.0011	0.0009	0.0041	0.0024	0.0022
12	312	0.0196	0.0510	0.0197	0.0578	0.1565	0.0045	120	35	58	21	0.0015	0.2519	0.0015	0.0011	0.0045	0.0029	0.0024
13	295	0.0186	0.0482	0.0180	0.0513	0.1426	0.0041	117	33	56	19	0.0014	0.2475	0.0013	0.0010	0.0044	0.0027	0.0023
14	278	0.0176	0.0454	0.0164	0.0450	0.1291	0.0037	114	31	55	17	0.0013	0.2424	0.0011	0.0010	0.0043	0.0025	0.0023
15	269	0.0170	0.0439	0.0154	0.0415	0.1215	0.0034	112	30	54	16	0.0013	0.2396	0.0010	0.0009	0.0042	0.0023	0.0023
16	262	0.0167	0.0428	0.0148	0.0391	0.1163	0.0033	111	30	54	15	0.0013	0.2374	0.0009	0.0009	0.0042	0.0023	0.0022
17	260	0.0165	0.0424	0.0145	0.0379	0.1139	0.0032	111	29	54	15	0.0013	0.2370	0.0009	0.0009	0.0042	0.0022	0.0022
18	258	0.0164	0.0421	0.0143	0.0372	0.1124	0.0032	110	29	54	14	0.0013	0.2363	0.0009	0.0009	0.0041	0.0022	0.0022
19	257	0.0163	0.0419	0.0142	0.0368	0.1115	0.0032	110	29	54	14	0.0013	0.2360	0.0009	0.0009	0.0041	0.0022	0.0022
20	257	0.0163	0.0419	0.0142	0.0366	0.1111	0.0031	110	29	53	14	0.0012	0.2358	0.0009	0.0009	0.0041	0.0022	0.0022
21	257	0.0163	0.0419	0.0142	0.0366	0.1112	0.0031	110	29	53	14	0.0012	0.2357	0.0009	0.0009	0.0041	0.0022	0.0022
22	257	0.0163	0.0419	0.0142	0.0367	0.1113	0.0031	110	29	53	14	0.0012	0.2356	0.0009	0.0009	0.0041	0.0022	0.0022
23	257	0.0163	0.0419	0.0142	0.0368	0.1115	0.0032	110	29	53	14	0.0013	0.2355	0.0009	0.0009	0.0041	0.0022	0.0022
24	258	0.0164	0.0421	0.0143	0.0371	0.1122	0.0032	110	29	54	14	0.0013	0.2360	0.0009	0.0009	0.0041	0.0022	0.0022
25	256	0.0162	0.0417	0.0141	0.0361	0.1101	0.0031	110	29	53	14	0.0012	0.2358	0.0008	0.0009	0.0041	0.0022	0.0022
26	253	0.0161	0.0412	0.0138	0.0349	0.1076	0.0030	109	29	53	14	0.0012	0.2353	0.0008	0.0009	0.0041	0.0021	0.0022
27	251	0.0160	0.0409	0.0136	0.0341	0.1060	0.0030	109	28	53	13	0.0012	0.2350	0.0008	0.0009	0.0041	0.0021	0.0022
28	250	0.0159	0.0406	0.0134	0.0336	0.1047	0.0030	109	28	53	13	0.0012	0.2347	0.0008	0.0009	0.0041	0.0021	0.0022
29	249	0.0158	0.0404	0.0133	0.0331	0.1037	0.0029	109	28	53	13	0.0012	0.2345	0.0007	0.0009	0.0041	0.0021	0.0022
30	248	0.0158	0.0403	0.0132	0.0327	0.1029	0.0029	109	28	53	13	0.0012	0.2343	0.0007	0.0009	0.0041	0.0020	0.0022
31	247	0.0157	0.0401	0.0131	0.0324	0.1022	0.0029	109	28	53	13	0.0012	0.2341	0.0007	0.0009	0.0041	0.0020	0.0022
32	246	0.0157	0.0400	0.0130	0.0321	0.1016	0.0029	108	28	53	13	0.0012	0.2340	0.0007	0.0009	0.0041	0.0020	0.0022
33	246	0.0156	0.0398	0.0130	0.0318	0.1010	0.0029	108	28	53	12	0.0012	0.2338	0.0007	0.0009	0.0041	0.0020	0.0022
34	245	0.0156	0.0397	0.0129	0.0316	0.1005	0.0028	108	28	53	12	0.0012	0.2337	0.0007	0.0009	0.0041	0.0020	0.0022
35	245	0.0156	0.0396	0.0128	0.0314	0.1000	0.0028	108	28	53	12	0.0012	0.2335	0.0007	0.0009	0.0041	0.0020	0.0022
36	244	0.0155	0.0395	0.0128	0.0312	0.0996	0.0028	108	28	53	12	0.0012	0.2334	0.0007	0.0009	0.0041	0.0020	0.0022
37	244	0.0155	0.0394	0.0127	0.0311	0.0992	0.0028	108	27	53	12	0.0012	0.2332	0.0007	0.0008	0.0041	0.0020	0.0022
38	243	0.0155	0.0393	0.0127	0.0309	0.0989	0.0028	108	27	53	12	0.0012	0.2331	0.0007	0.0008	0.0041	0.0020	0.0022
39	243	0.0154	0.0393	0.0126	0.0308	0.0986	0.0028	108	27	53	12	0.0012	0.2330	0.0007	0.0008	0.0040	0.0020	0.0022
40	243	0.0154	0.0392	0.0126	0.0306	0.0983	0.0028	108	27	53	12	0.0012	0.2329	0.0007	0.0008	0.0040	0.0020	0.0022
41	242	0.0154	0.0391	0.0126	0.0305	0.0980	0.0028	108	27	53	12	0.0012	0.2328	0.0007	0.0008	0.0040	0.0020	0.0022
42	242	0.0154	0.0391	0.0125	0.0304	0.0978	0.0028	108	27	53	12	0.0012	0.2328	0.0007	0.0008	0.0040	0.0020	0.0022
43	242	0.0154	0.0391	0.0125	0.0303	0.0976	0.0028	108	27	53	12	0.0012	0.2327	0.0006	0.0008	0.0040	0.0019	0.0022
44	242	0.0154	0.0390	0.0125	0.0302	0.0974	0.0028	108	27	53	12	0.0012	0.2326	0.0006	0.0008	0.0040	0.0019	0.0022
45	241	0.0153	0.0390	0.0125	0.0301	0.0972	0.0027	108	27	53	12	0.0012	0.2326	0.0006	0.0008	0.0040	0.0019	0.0022
46	241	0.0153	0.0390	0.0125	0.0301	0.0972	0.0027	108	27	53	12	0.0012	0.2325	0.0006	0.0008	0.0040	0.0019	0.0022
47	242	0.0154	0.0390	0.0125	0.0302	0.0974	0.0028	108	27	53	12	0.0012	0.2325	0.0006	0.0008	0.0040	0.0019	0.0022
48	242	0.0154	0.0391	0.0126	0.0304	0.0978	0.0028	108	27	53	12	0.0012	0.2325	0.0007	0.0008	0.0040	0.0020	0.0022
49	243	0.0154	0.0393	0.0127	0.0308	0.0986	0.0028	108	27	53	12	0.0012	0.2326	0.0007	0.0008	0.0040	0.0020	0.0022
50	244	0.0155	0.0395	0.0128	0.0312	0.0995	0.0028	108	28	53	12	0.0012	0.2327	0.0007	0.0008	0.0040	0.0020	0.0022

Cell 2E Seepage Concentrations in Collection Drains (cont.)

Year	SO4 mg/L	Sb mg/L	As mg/L	Cu mg/L	Ni mg/L	Zn mg/L	Co mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Ag mg/L	B mg/L	Be mg/L	Cd mg/L	Pb mg/L	Se mg/L	Tl mg/L
51	245	0.0156	0.0397	0.0129	0.0317	0.1007	0.0028	108	28	53	12	0.0012	0.2328	0.0007	0.0009	0.0040	0.0020	0.0022
52	247	0.0157	0.0400	0.0131	0.0324	0.1020	0.0029	108	28	53	13	0.0012	0.2329	0.0007	0.0009	0.0041	0.0020	0.0022
53	249	0.0157	0.0403	0.0133	0.0331	0.1036	0.0029	108	28	53	13	0.0012	0.2330	0.0007	0.0009	0.0041	0.0020	0.0022
54	251	0.0159	0.0406	0.0135	0.0340	0.1053	0.0030	108	28	53	13	0.0012	0.2333	0.0008	0.0009	0.0041	0.0021	0.0022
55	253	0.0159	0.0409	0.0136	0.0345	0.1066	0.0030	109	28	53	13	0.0012	0.2334	0.0008	0.0009	0.0041	0.0021	0.0022
56	254	0.0160	0.0411	0.0138	0.0352	0.1080	0.0031	109	29	53	14	0.0012	0.2336	0.0008	0.0009	0.0041	0.0021	0.0022
57	256	0.0161	0.0414	0.0140	0.0359	0.1095	0.0031	109	29	53	14	0.0012	0.2338	0.0008	0.0009	0.0041	0.0021	0.0022
58	258	0.0162	0.0417	0.0142	0.0367	0.1111	0.0031	109	29	53	14	0.0012	0.2339	0.0009	0.0009	0.0041	0.0022	0.0022
59	260	0.0163	0.0420	0.0144	0.0376	0.1129	0.0032	109	29	53	14	0.0013	0.2341	0.0009	0.0009	0.0041	0.0022	0.0022
60	262	0.0164	0.0423	0.0146	0.0385	0.1147	0.0032	109	29	53	15	0.0013	0.2343	0.0009	0.0009	0.0041	0.0022	0.0022
61	265	0.0166	0.0426	0.0148	0.0394	0.1166	0.0033	110	29	53	15	0.0013	0.2344	0.0010	0.0009	0.0041	0.0023	0.0022
62	267	0.0167	0.0429	0.0151	0.0404	0.1185	0.0034	110	30	53	15	0.0013	0.2346	0.0010	0.0009	0.0041	0.0023	0.0022
63	269	0.0168	0.0432	0.0153	0.0413	0.1204	0.0034	110	30	53	16	0.0013	0.2347	0.0010	0.0009	0.0041	0.0023	0.0022
64	271	0.0169	0.0435	0.0155	0.0422	0.1222	0.0035	110	30	53	16	0.0013	0.2348	0.0010	0.0009	0.0041	0.0023	0.0022
65	273	0.0170	0.0438	0.0157	0.0430	0.1239	0.0035	110	30	53	16	0.0013	0.2349	0.0011	0.0009	0.0041	0.0024	0.0022
66	275	0.0171	0.0441	0.0159	0.0438	0.1254	0.0036	110	30	53	16	0.0013	0.2350	0.0011	0.0009	0.0042	0.0024	0.0022
67	276	0.0171	0.0443	0.0160	0.0444	0.1266	0.0036	110	31	53	17	0.0013	0.2350	0.0011	0.0009	0.0042	0.0024	0.0022
68	278	0.0172	0.0444	0.0161	0.0449	0.1276	0.0036	110	31	54	17	0.0013	0.2350	0.0011	0.0009	0.0042	0.0024	0.0022
69	278	0.0172	0.0445	0.0162	0.0452	0.1282	0.0036	111	31	54	17	0.0013	0.2350	0.0011	0.0009	0.0042	0.0024	0.0022
70	279	0.0172	0.0445	0.0162	0.0453	0.1284	0.0037	111	31	54	17	0.0013	0.2350	0.0011	0.0009	0.0042	0.0024	0.0022
71	278	0.0172	0.0445	0.0162	0.0453	0.1283	0.0036	110	31	54	17	0.0013	0.2350	0.0011	0.0009	0.0042	0.0024	0.0022
72	278	0.0172	0.0444	0.0162	0.0450	0.1277	0.0036	110	31	53	17	0.0013	0.2349	0.0011	0.0009	0.0042	0.0024	0.0022
73	277	0.0171	0.0442	0.0160	0.0446	0.1268	0.0036	110	31	53	17	0.0013	0.2348	0.0011	0.0009	0.0042	0.0024	0.0022
74	275	0.0171	0.0439	0.0159	0.0440	0.1255	0.0036	110	30	53	16	0.0013	0.2346	0.0011	0.0009	0.0041	0.0024	0.0022
75	273	0.0170	0.0436	0.0157	0.0432	0.1240	0.0035	110	30	53	16	0.0013	0.2345	0.0011	0.0009	0.0041	0.0024	0.0022
76	271	0.0168	0.0433	0.0155	0.0424	0.1222	0.0035	110	30	53	16	0.0013	0.2343	0.0010	0.0009	0.0041	0.0023	0.0022
77	269	0.0167	0.0429	0.0152	0.0414	0.1203	0.0034	110	30	53	16	0.0013	0.2341	0.0010	0.0009	0.0041	0.0023	0.0022
78	266	0.0166	0.0426	0.0150	0.0405	0.1182	0.0034	109	30	53	15	0.0013	0.2339	0.0010	0.0009	0.0041	0.0023	0.0022
79	264	0.0165	0.0422	0.0147	0.0394	0.1161	0.0033	109	29	53	15	0.0013	0.2336	0.0009	0.0009	0.0041	0.0022	0.0022
80	262	0.0163	0.0418	0.0145	0.0384	0.1140	0.0032	109	29	53	15	0.0013	0.2334	0.0009	0.0009	0.0041	0.0022	0.0022
81	259	0.0162	0.0414	0.0142	0.0375	0.1120	0.0032	109	29	53	14	0.0012	0.2332	0.0009	0.0009	0.0041	0.0022	0.0022
82	257	0.0161	0.0411	0.0140	0.0365	0.1101	0.0031	109	29	53	14	0.0012	0.2330	0.0008	0.0009	0.0041	0.0021	0.0022
83	255	0.0160	0.0407	0.0138	0.0357	0.1082	0.0031	108	28	53	14	0.0012	0.2328	0.0008	0.0009	0.0041	0.0021	0.0022
84	253	0.0159	0.0404	0.0136	0.0348	0.1065	0.0030	108	28	53	13	0.0012	0.2326	0.0008	0.0009	0.0041	0.0021	0.0022
85	251	0.0158	0.0401	0.0134	0.0341	0.1050	0.0030	108	28	53	13	0.0012	0.2325	0.0008	0.0009	0.0041	0.0021	0.0022
86	249	0.0157	0.0399	0.0132	0.0334	0.1035	0.0029	108	28	53	13	0.0012	0.2323	0.0007	0.0009	0.0040	0.0020	0.0022
87	248	0.0156	0.0396	0.0131	0.0328	0.1023	0.0029	108	28	53	13	0.0012	0.2322	0.0007	0.0009	0.0040	0.0020	0.0022
88	246	0.0155	0.0394	0.0129	0.0322	0.1011	0.0029	108	28	52	12	0.0012	0.2321	0.0007	0.0009	0.0040	0.0020	0.0022
89	245	0.0155	0.0392	0.0128	0.0317	0.1001	0.0028	108	27	52	12	0.0012	0.2320	0.0007	0.0008	0.0040	0.0020	0.0022
90	244	0.0154	0.0391	0.0127	0.0313	0.0992	0.0028	107	27	52	12	0.0012	0.2319	0.0007	0.0008	0.0040	0.0020	0.0022
91	243	0.0154	0.0389	0.0126	0.0309	0.0984	0.0028	107	27	52	12	0.0012	0.2318	0.0007	0.0008	0.0040	0.0020	0.0022
92	242	0.0153	0.0388	0.0125	0.0306	0.0977	0.0028	107	27	52	12	0.0012	0.2317	0.0006	0.0008	0.0040	0.0019	0.0022
93	242	0.0153	0.0387	0.0124	0.0303	0.0971	0.0027	107	27	52	12	0.0012	0.2316	0.0006	0.0008	0.0040	0.0019	0.0022
94	241	0.0153	0.0386	0.0124	0.0301	0.0966	0.0027	107	27	52	12	0.0012	0.2316	0.0006	0.0008	0.0040	0.0019	0.0022
95	240	0.0152	0.0385	0.0123	0.0298	0.0962	0.0027	107	27	52	12	0.0012	0.2315	0.0006	0.0008	0.0040	0.0019	0.0022
96	240	0.0152	0.0384	0.0123	0.0297	0.0958	0.0027	107	27	52	12	0.0012	0.2315	0.0006	0.0008	0.0040	0.0019	0.0022
97	240	0.0152	0.0384	0.0122	0.0295	0.0954	0.0027	107	27	52	12	0.0012	0.2315	0.0006	0.0008	0.0040	0.0019	0.0022
98	239	0.0152	0.0383	0.0122	0.0294	0.0952	0.0027	107	27	52	12	0.0012	0.2314	0.0006	0.0008	0.0040	0.0019	0.0022
99	239	0.0151	0.0383	0.0122	0.0292	0.0949	0.0027	107	27	52	12	0.0012	0.2314	0.0006	0.0008	0.0040	0.0019	0.0022
100	239	0.0151	0.0382	0.0121	0.0291	0.0947	0.0027	107	27	52	11	0.0012	0.2314	0.0006	0.0008	0.0040	0.0019	0.0022

Cell 2E Concentrations in Percolate to Groundwater

Year	SO4 mg/L	Sb mg/L	As mg/L	Cu mg/L	Ni mg/L	Zn mg/L	Co mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Ag mg/L	B mg/L	Be mg/L	Cd mg/L	Pb mg/L	Se mg/L	Tl mg/L
1	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
2	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
3	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
4	186	0.0034	0.0033	0.0029	0.0039	0.0142	0.0014	70	80	96	15	0.0010	0.2331	0.0002	0.0002	0.0070	0.0020	0.0020
5	188	0.0036	0.0040	0.0031	0.0048	0.0165	0.0014	70	79	96	15	0.0010	0.2332	0.0003	0.0002	0.0070	0.0021	0.0020
6	188	0.0037	0.0042	0.0032	0.0051	0.0173	0.0015	71	79	96	15	0.0010	0.2332	0.0003	0.0002	0.0070	0.0021	0.0020
7	189	0.0037	0.0043	0.0033	0.0053	0.0177	0.0015	71	79	96	15	0.0010	0.2333	0.0003	0.0002	0.0070	0.0021	0.0020
8	238	0.0151	0.0382	0.0122	0.0293	0.0950	0.0027	107	27	52	12	0.0012	0.2309	0.0006	0.0008	0.0040	0.0019	0.0022
9	238	0.0151	0.0382	0.0122	0.0293	0.0949	0.0027	107	27	52	12	0.0012	0.2309	0.0006	0.0008	0.0040	0.0019	0.0022
10	238	0.0152	0.0383	0.0122	0.0295	0.0953	0.0027	107	27	52	12	0.0012	0.2310	0.0006	0.0008	0.0040	0.0019	0.0022
11	239	0.0152	0.0384	0.0123	0.0298	0.0959	0.0027	107	27	52	12	0.0012	0.2310	0.0006	0.0008	0.0040	0.0019	0.0022
12	246	0.0156	0.0394	0.0128	0.0318	0.1005	0.0028	108	28	53	12	0.0012	0.2338	0.0007	0.0009	0.0041	0.0020	0.0022
13	243	0.0154	0.0390	0.0126	0.0308	0.0983	0.0028	108	27	53	12	0.0012	0.2331	0.0007	0.0008	0.0040	0.0020	0.0022
14	241	0.0153	0.0386	0.0123	0.0298	0.0963	0.0027	108	27	52	12	0.0012	0.2323	0.0006	0.0008	0.0040	0.0019	0.0022
15	239	0.0152	0.0384	0.0122	0.0293	0.0951	0.0027	107	27	52	12	0.0012	0.2319	0.0006	0.0008	0.0040	0.0019	0.0022
16	238	0.0151	0.0382	0.0121	0.0289	0.0943	0.0027	107	27	52	11	0.0012	0.2316	0.0006	0.0008	0.0040	0.0019	0.0022
17	238	0.0151	0.0381	0.0120	0.0288	0.0940	0.0027	107	27	52	11	0.0012	0.2315	0.0006	0.0008	0.0040	0.0019	0.0022
18	237	0.0151	0.0381	0.0120	0.0286	0.0937	0.0027	107	27	52	11	0.0012	0.2314	0.0006	0.0008	0.0040	0.0019	0.0022
19	237	0.0151	0.0381	0.0120	0.0286	0.0936	0.0026	107	27	52	11	0.0012	0.2314	0.0006	0.0008	0.0040	0.0019	0.0022
20	237	0.0151	0.0380	0.0120	0.0285	0.0935	0.0026	107	27	52	11	0.0012	0.2313	0.0006	0.0008	0.0040	0.0019	0.0022
21	237	0.0151	0.0380	0.0120	0.0285	0.0935	0.0026	107	27	52	11	0.0012	0.2313	0.0006	0.0008	0.0040	0.0019	0.0022
22	237	0.0151	0.0380	0.0120	0.0285	0.0935	0.0026	107	27	52	11	0.0012	0.2313	0.0006	0.0008	0.0040	0.0019	0.0022
23	237	0.0151	0.0380	0.0120	0.0286	0.0935	0.0026	107	27	52	11	0.0012	0.2313	0.0006	0.0008	0.0040	0.0019	0.0022
24	237	0.0151	0.0381	0.0120	0.0286	0.0936	0.0026	107	27	52	11	0.0012	0.2314	0.0006	0.0008	0.0040	0.0019	0.0022
25	237	0.0151	0.0380	0.0120	0.0285	0.0933	0.0026	107	27	52	11	0.0012	0.2313	0.0006	0.0008	0.0040	0.0019	0.0022
26	237	0.0150	0.0379	0.0119	0.0283	0.0930	0.0026	107	27	52	11	0.0012	0.2313	0.0006	0.0008	0.0040	0.0019	0.0022
27	236	0.0150	0.0379	0.0119	0.0282	0.0927	0.0026	107	27	52	11	0.0012	0.2312	0.0006	0.0008	0.0040	0.0019	0.0022
28	236	0.0150	0.0379	0.0119	0.0281	0.0926	0.0026	107	27	52	11	0.0012	0.2312	0.0006	0.0008	0.0040	0.0019	0.0022
29	236	0.0150	0.0378	0.0119	0.0280	0.0924	0.0026	107	27	52	11	0.0012	0.2311	0.0006	0.0008	0.0040	0.0019	0.0022
30	236	0.0150	0.0378	0.0118	0.0280	0.0923	0.0026	107	27	52	11	0.0012	0.2311	0.0006	0.0008	0.0040	0.0019	0.0022
31	236	0.0150	0.0378	0.0118	0.0279	0.0922	0.0026	107	27	52	11	0.0012	0.2311	0.0006	0.0008	0.0040	0.0019	0.0022
32	236	0.0150	0.0378	0.0118	0.0279	0.0921	0.0026	107	27	52	11	0.0012	0.2311	0.0006	0.0008	0.0040	0.0019	0.0022
33	236	0.0150	0.0377	0.0118	0.0278	0.0920	0.0026	107	27	52	11	0.0012	0.2310	0.0006	0.0008	0.0040	0.0019	0.0022
34	236	0.0150	0.0377	0.0118	0.0278	0.0919	0.0026	107	26	52	11	0.0012	0.2310	0.0006	0.0008	0.0040	0.0019	0.0022
35	235	0.0150	0.0377	0.0118	0.0278	0.0919	0.0026	107	26	52	11	0.0012	0.2310	0.0006	0.0008	0.0040	0.0018	0.0022
36	235	0.0150	0.0377	0.0118	0.0278	0.0918	0.0026	107	26	52	11	0.0012	0.2310	0.0006	0.0008	0.0040	0.0018	0.0022
37	235	0.0149	0.0377	0.0118	0.0277	0.0918	0.0026	107	26	52	11	0.0012	0.2310	0.0006	0.0008	0.0040	0.0018	0.0022
38	235	0.0149	0.0377	0.0118	0.0277	0.0917	0.0026	107	26	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0018	0.0022
39	235	0.0149	0.0377	0.0118	0.0277	0.0917	0.0026	107	26	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0018	0.0022
40	235	0.0149	0.0377	0.0118	0.0277	0.0916	0.0026	107	26	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0018	0.0022
41	235	0.0149	0.0376	0.0118	0.0277	0.0916	0.0026	107	26	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0018	0.0022
42	235	0.0149	0.0376	0.0118	0.0276	0.0916	0.0026	107	26	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0018	0.0022
43	235	0.0149	0.0376	0.0118	0.0276	0.0915	0.0026	107	26	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0018	0.0022
44	235	0.0149	0.0376	0.0117	0.0276	0.0915	0.0026	107	26	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0018	0.0022
45	235	0.0149	0.0376	0.0117	0.0276	0.0915	0.0026	107	26	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0018	0.0022
46	235	0.0149	0.0376	0.0117	0.0276	0.0915	0.0026	107	26	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0018	0.0022
47	235	0.0149	0.0376	0.0117	0.0276	0.0915	0.0026	107	26	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0018	0.0022
48	235	0.0149	0.0376	0.0118	0.0276	0.0916	0.0026	107	26	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0018	0.0022
49	235	0.0149	0.0377	0.0118	0.0277	0.0917	0.0026	107	26	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0018	0.0022
50	235	0.0149	0.0377	0.0118	0.0277	0.0918	0.0026	107	26	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0018	0.0022

Cell 2E Concentrations in Percolate to Groundwater (cont.)

Year	SO4 mg/L	Sb mg/L	As mg/L	Cu mg/L	Ni mg/L	Zn mg/L	Co mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Ag mg/L	B mg/L	Be mg/L	Cd mg/L	Pb mg/L	Se mg/L	Tl mg/L
51	236	0.0150	0.0377	0.0118	0.0278	0.0920	0.0026	107	26	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0019	0.0022
52	236	0.0150	0.0378	0.0118	0.0279	0.0922	0.0026	107	27	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0019	0.0022
53	236	0.0150	0.0378	0.0119	0.0280	0.0924	0.0026	107	27	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0019	0.0022
54	236	0.0150	0.0379	0.0119	0.0281	0.0926	0.0026	107	27	52	11	0.0012	0.2310	0.0006	0.0008	0.0040	0.0019	0.0022
55	237	0.0150	0.0379	0.0119	0.0282	0.0928	0.0026	107	27	52	11	0.0012	0.2310	0.0006	0.0008	0.0040	0.0019	0.0022
56	237	0.0150	0.0379	0.0119	0.0283	0.0930	0.0026	107	27	52	11	0.0012	0.2310	0.0006	0.0008	0.0040	0.0019	0.0022
57	237	0.0150	0.0380	0.0120	0.0284	0.0932	0.0026	107	27	52	11	0.0012	0.2310	0.0006	0.0008	0.0040	0.0019	0.0022
58	237	0.0151	0.0380	0.0120	0.0285	0.0935	0.0026	107	27	52	11	0.0012	0.2311	0.0006	0.0008	0.0040	0.0019	0.0022
59	238	0.0151	0.0381	0.0120	0.0287	0.0937	0.0027	107	27	52	11	0.0012	0.2311	0.0006	0.0008	0.0040	0.0019	0.0022
60	238	0.0151	0.0381	0.0121	0.0288	0.0940	0.0027	107	27	52	11	0.0012	0.2311	0.0006	0.0008	0.0040	0.0019	0.0022
61	238	0.0151	0.0381	0.0121	0.0289	0.0943	0.0027	107	27	52	11	0.0012	0.2311	0.0006	0.0008	0.0040	0.0019	0.0022
62	239	0.0151	0.0382	0.0121	0.0291	0.0945	0.0027	107	27	52	11	0.0012	0.2311	0.0006	0.0008	0.0040	0.0019	0.0022
63	239	0.0151	0.0382	0.0122	0.0292	0.0948	0.0027	107	27	52	12	0.0012	0.2312	0.0006	0.0008	0.0040	0.0019	0.0022
64	239	0.0151	0.0383	0.0122	0.0293	0.0951	0.0027	107	27	52	12	0.0012	0.2312	0.0006	0.0008	0.0040	0.0019	0.0022
65	240	0.0152	0.0383	0.0122	0.0295	0.0953	0.0027	107	27	52	12	0.0012	0.2312	0.0006	0.0008	0.0040	0.0019	0.0022
66	240	0.0152	0.0384	0.0122	0.0296	0.0955	0.0027	107	27	52	12	0.0012	0.2312	0.0006	0.0008	0.0040	0.0019	0.0022
67	240	0.0152	0.0384	0.0123	0.0297	0.0957	0.0027	107	27	52	12	0.0012	0.2312	0.0006	0.0008	0.0040	0.0019	0.0022
68	240	0.0152	0.0384	0.0123	0.0297	0.0959	0.0027	107	27	52	12	0.0012	0.2312	0.0006	0.0008	0.0040	0.0019	0.0022
69	240	0.0152	0.0384	0.0123	0.0298	0.0959	0.0027	107	27	52	12	0.0012	0.2312	0.0006	0.0008	0.0040	0.0019	0.0022
70	240	0.0152	0.0384	0.0123	0.0298	0.0960	0.0027	107	27	52	12	0.0012	0.2312	0.0006	0.0008	0.0040	0.0019	0.0022
71	240	0.0152	0.0384	0.0123	0.0298	0.0960	0.0027	107	27	52	12	0.0012	0.2312	0.0006	0.0008	0.0040	0.0019	0.0022
72	240	0.0152	0.0384	0.0123	0.0297	0.0959	0.0027	107	27	52	12	0.0012	0.2312	0.0006	0.0008	0.0040	0.0019	0.0022
73	240	0.0152	0.0384	0.0123	0.0297	0.0957	0.0027	107	27	52	12	0.0012	0.2312	0.0006	0.0008	0.0040	0.0019	0.0022
74	240	0.0152	0.0383	0.0122	0.0296	0.0956	0.0027	107	27	52	12	0.0012	0.2312	0.0006	0.0008	0.0040	0.0019	0.0022
75	240	0.0152	0.0383	0.0122	0.0295	0.0953	0.0027	107	27	52	12	0.0012	0.2311	0.0006	0.0008	0.0040	0.0019	0.0022
76	239	0.0151	0.0382	0.0122	0.0294	0.0951	0.0027	107	27	52	12	0.0012	0.2311	0.0006	0.0008	0.0040	0.0019	0.0022
77	239	0.0151	0.0382	0.0121	0.0292	0.0948	0.0027	107	27	52	11	0.0012	0.2311	0.0006	0.0008	0.0040	0.0019	0.0022
78	239	0.0151	0.0381	0.0121	0.0291	0.0945	0.0027	107	27	52	11	0.0012	0.2310	0.0006	0.0008	0.0040	0.0019	0.0022
79	238	0.0151	0.0381	0.0121	0.0289	0.0942	0.0027	107	27	52	11	0.0012	0.2310	0.0006	0.0008	0.0040	0.0019	0.0022
80	238	0.0151	0.0380	0.0120	0.0288	0.0939	0.0027	107	27	52	11	0.0012	0.2310	0.0006	0.0008	0.0040	0.0019	0.0022
81	238	0.0151	0.0380	0.0120	0.0287	0.0936	0.0026	107	27	52	11	0.0012	0.2310	0.0006	0.0008	0.0040	0.0019	0.0022
82	237	0.0150	0.0379	0.0120	0.0285	0.0933	0.0026	107	27	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0019	0.0022
83	237	0.0150	0.0379	0.0119	0.0284	0.0931	0.0026	107	27	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0019	0.0022
84	237	0.0150	0.0378	0.0119	0.0283	0.0928	0.0026	107	27	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0019	0.0022
85	236	0.0150	0.0378	0.0119	0.0282	0.0926	0.0026	107	27	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0019	0.0022
86	236	0.0150	0.0377	0.0119	0.0281	0.0924	0.0026	107	27	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0019	0.0022
87	236	0.0150	0.0377	0.0118	0.0280	0.0922	0.0026	107	26	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0019	0.0022
88	236	0.0150	0.0377	0.0118	0.0279	0.0920	0.0026	107	26	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0019	0.0022
89	236	0.0149	0.0377	0.0118	0.0278	0.0919	0.0026	107	26	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0018	0.0022
90	235	0.0149	0.0376	0.0118	0.0278	0.0918	0.0026	107	26	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0018	0.0022
91	235	0.0149	0.0376	0.0118	0.0277	0.0916	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0022
92	235	0.0149	0.0376	0.0117	0.0277	0.0916	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0022
93	235	0.0149	0.0376	0.0117	0.0276	0.0915	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0022
94	235	0.0149	0.0376	0.0117	0.0276	0.0914	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0022
95	235	0.0149	0.0376	0.0117	0.0276	0.0913	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0022
96	235	0.0149	0.0375	0.0117	0.0275	0.0913	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0022
97	235	0.0149	0.0375	0.0117	0.0275	0.0912	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0022
98	235	0.0149	0.0375	0.0117	0.0275	0.0912	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0022
99	235	0.0149	0.0375	0.0117	0.0275	0.0911	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0022
100	235	0.0149	0.0375	0.0117	0.0275	0.0911	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0022

Cell 1E Underdrain Seepage Collection Concentrations

Year	SO4 mg/L	Sb mg/L	As mg/L	Cu mg/L	Ni mg/L	Zn mg/L	Co mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Ag mg/L	B mg/L	Be mg/L	Cd mg/L	Pb mg/L	Se mg/L	Tl mg/L
1																		
2																		
3																		
4																		
5																		
6																		
7																		
8																		
9	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
10	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
11	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
12	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
13	234	0.0149	0.0374	0.0116	0.0272	0.0905	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
14	234	0.0149	0.0374	0.0116	0.0272	0.0905	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
15	234	0.0149	0.0374	0.0116	0.0272	0.0905	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
16	238	0.0151	0.0380	0.0121	0.0289	0.0941	0.0027	107	27	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0019	0.0021
17	269	0.0169	0.0436	0.0157	0.0436	0.1244	0.0035	110	30	53	16	0.0013	0.2333	0.0011	0.0009	0.0041	0.0024	0.0018
18	302	0.0188	0.0491	0.0194	0.0584	0.1550	0.0044	112	34	54	21	0.0014	0.2357	0.0016	0.0010	0.0042	0.0029	0.0016
19	359	0.0218	0.0580	0.0254	0.0821	0.2041	0.0058	117	39	56	29	0.0016	0.2397	0.0024	0.0012	0.0044	0.0036	0.0015
20	479	0.0280	0.0765	0.0377	0.1312	0.3057	0.0088	126	51	59	45	0.0020	0.2478	0.0040	0.0015	0.0049	0.0053	0.0014
21	646	0.0367	0.1023	0.0549	0.1998	0.4476	0.0129	139	67	63	68	0.0026	0.2592	0.0063	0.0020	0.0054	0.0075	0.0013
22	785	0.0438	0.1234	0.0691	0.2562	0.5642	0.0162	150	80	67	86	0.0031	0.2685	0.0081	0.0023	0.0059	0.0094	0.0013
23	893	0.0493	0.1399	0.0801	0.3000	0.6548	0.0188	158	91	70	100	0.0034	0.2758	0.0096	0.0026	0.0063	0.0109	0.0013
24	976	0.0536	0.1525	0.0885	0.3336	0.7244	0.0208	165	99	72	111	0.0037	0.2814	0.0107	0.0028	0.0065	0.0120	0.0013
25	1026	0.0561	0.1601	0.0936	0.3538	0.7661	0.0220	169	103	73	118	0.0039	0.2847	0.0114	0.0030	0.0067	0.0127	0.0013
26	1031	0.0564	0.1609	0.0941	0.3559	0.7704	0.0222	169	104	73	119	0.0039	0.2851	0.0114	0.0030	0.0067	0.0127	0.0013
27	1007	0.0552	0.1573	0.0915	0.3453	0.7490	0.0216	168	102	73	115	0.0038	0.2864	0.0111	0.0029	0.0067	0.0124	0.0013
28	975	0.0539	0.1526	0.0868	0.3244	0.7097	0.0204	176	99	78	109	0.0037	0.3097	0.0103	0.0029	0.0070	0.0118	0.0016
29	887	0.0496	0.1393	0.0768	0.2828	0.6266	0.0180	177	91	80	95	0.0035	0.3221	0.0089	0.0026	0.0069	0.0105	0.0018
30	677	0.0385	0.1067	0.0557	0.1995	0.4527	0.0130	156	70	72	68	0.0027	0.2966	0.0062	0.0021	0.0060	0.0077	0.0019
31	512	0.0298	0.0810	0.0391	0.1346	0.3168	0.0091	139	54	65	46	0.0022	0.2749	0.0041	0.0016	0.0053	0.0055	0.0020
32	403	0.0240	0.0638	0.0280	0.0908	0.2254	0.0064	128	43	61	32	0.0018	0.2622	0.0026	0.0013	0.0049	0.0040	0.0021
33	320	0.0195	0.0509	0.0199	0.0593	0.1588	0.0045	118	35	57	21	0.0015	0.2476	0.0016	0.0011	0.0044	0.0029	0.0021
34	262	0.0164	0.0418	0.0144	0.0380	0.1134	0.0032	110	29	53	14	0.0013	0.2349	0.0009	0.0009	0.0041	0.0022	0.0021
35	250	0.0157	0.0399	0.0132	0.0335	0.1038	0.0029	108	28	53	13	0.0012	0.2325	0.0007	0.0009	0.0041	0.0020	0.0021
36	247	0.0155	0.0394	0.0129	0.0323	0.1012	0.0029	108	28	52	12	0.0012	0.2320	0.0007	0.0009	0.0040	0.0020	0.0021
37	245	0.0154	0.0391	0.0127	0.0314	0.0994	0.0028	107	27	52	12	0.0012	0.2318	0.0007	0.0008	0.0040	0.0020	0.0021
38	243	0.0153	0.0388	0.0126	0.0308	0.0982	0.0028	107	27	52	12	0.0012	0.2316	0.0007	0.0008	0.0040	0.0019	0.0021
39	242	0.0153	0.0386	0.0124	0.0304	0.0972	0.0028	107	27	52	12	0.0012	0.2314	0.0006	0.0008	0.0040	0.0019	0.0021
40	241	0.0152	0.0385	0.0123	0.0300	0.0965	0.0027	107	27	52	12	0.0012	0.2313	0.0006	0.0008	0.0040	0.0019	0.0021
41	240	0.0152	0.0384	0.0123	0.0297	0.0959	0.0027	107	27	52	12	0.0012	0.2312	0.0006	0.0008	0.0040	0.0019	0.0021
42	240	0.0152	0.0383	0.0122	0.0295	0.0954	0.0027	107	27	52	12	0.0012	0.2312	0.0006	0.0008	0.0040	0.0019	0.0021
43	239	0.0151	0.0382	0.0122	0.0293	0.0950	0.0027	107	27	52	12	0.0012	0.2311	0.0006	0.0008	0.0040	0.0019	0.0021
44	239	0.0151	0.0382	0.0121	0.0291	0.0946	0.0027	107	27	52	11	0.0012	0.2311	0.0006	0.0008	0.0040	0.0019	0.0021
45	238	0.0151	0.0381	0.0121	0.0290	0.0943	0.0027	107	27	52	11	0.0012	0.2310	0.0006	0.0008	0.0040	0.0019	0.0021
46	238	0.0151	0.0380	0.0120	0.0289	0.0940	0.0027	107	27	52	11	0.0012	0.2310	0.0006	0.0008	0.0040	0.0019	0.0021
47	238	0.0151	0.0380	0.0120	0.0288	0.0938	0.0027	107	27	52	11	0.0012	0.2310	0.0006	0.0008	0.0040	0.0019	0.0021
48	238	0.0150	0.0380	0.0120	0.0287	0.0936	0.0026	107	27	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0019	0.0021
49	237	0.0150	0.0379	0.0120	0.0286	0.0934	0.0026	107	27	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0019	0.0021
50	237	0.0150	0.0379	0.0120	0.0285	0.0932	0.0026	107	27	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0019	0.0021

Cell 1E Underdrain Seepage Collection Concentrations (cont.)

Year	SO4 mg/L	Sb mg/L	As mg/L	Cu mg/L	Ni mg/L	Zn mg/L	Co mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Ag mg/L	B mg/L	Be mg/L	Cd mg/L	Pb mg/L	Se mg/L	Tl mg/L
51	237	0.0150	0.0379	0.0119	0.0284	0.0931	0.0026	107	27	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0019	0.0021
52	237	0.0150	0.0378	0.0119	0.0284	0.0930	0.0026	107	27	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0019	0.0021
53	237	0.0150	0.0378	0.0119	0.0283	0.0928	0.0026	107	27	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0019	0.0021
54	237	0.0150	0.0378	0.0119	0.0282	0.0927	0.0026	107	27	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0019	0.0021
55	236	0.0150	0.0378	0.0119	0.0282	0.0926	0.0026	107	27	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0019	0.0021
56	236	0.0150	0.0378	0.0119	0.0281	0.0925	0.0026	107	27	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0019	0.0021
57	236	0.0150	0.0378	0.0119	0.0281	0.0925	0.0026	107	27	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0019	0.0021
58	236	0.0150	0.0377	0.0118	0.0281	0.0924	0.0026	107	27	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0019	0.0021
59	236	0.0150	0.0377	0.0118	0.0280	0.0923	0.0026	107	27	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0019	0.0021
60	236	0.0150	0.0377	0.0118	0.0280	0.0922	0.0026	107	26	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0019	0.0021
61	236	0.0150	0.0377	0.0118	0.0280	0.0922	0.0026	107	26	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0019	0.0021
62	236	0.0150	0.0377	0.0118	0.0279	0.0921	0.0026	107	26	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0019	0.0021
63	236	0.0150	0.0377	0.0118	0.0279	0.0921	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0019	0.0021
64	236	0.0149	0.0377	0.0118	0.0279	0.0920	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
65	236	0.0149	0.0377	0.0118	0.0279	0.0920	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
66	236	0.0149	0.0377	0.0118	0.0278	0.0919	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
67	236	0.0149	0.0376	0.0118	0.0278	0.0919	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
68	236	0.0149	0.0376	0.0118	0.0278	0.0918	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
69	235	0.0149	0.0376	0.0118	0.0278	0.0918	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
70	235	0.0149	0.0376	0.0118	0.0278	0.0917	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
71	235	0.0149	0.0376	0.0118	0.0277	0.0917	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
72	235	0.0149	0.0376	0.0118	0.0277	0.0917	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
73	235	0.0149	0.0376	0.0118	0.0277	0.0916	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
74	235	0.0149	0.0376	0.0118	0.0277	0.0916	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
75	235	0.0149	0.0376	0.0118	0.0277	0.0916	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
76	235	0.0149	0.0376	0.0117	0.0277	0.0916	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
77	235	0.0149	0.0376	0.0117	0.0277	0.0915	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
78	235	0.0149	0.0376	0.0117	0.0276	0.0915	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
79	235	0.0149	0.0376	0.0117	0.0276	0.0915	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
80	235	0.0149	0.0376	0.0117	0.0276	0.0915	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
81	235	0.0149	0.0376	0.0117	0.0276	0.0914	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
82	235	0.0149	0.0376	0.0117	0.0276	0.0914	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
83	235	0.0149	0.0376	0.0117	0.0276	0.0914	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0021
84	235	0.0149	0.0376	0.0117	0.0276	0.0914	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0021
85	235	0.0149	0.0376	0.0117	0.0276	0.0914	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0021
86	235	0.0149	0.0375	0.0117	0.0276	0.0913	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0021
87	235	0.0149	0.0375	0.0117	0.0276	0.0913	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0021
88	235	0.0149	0.0375	0.0117	0.0276	0.0913	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0021
89	235	0.0149	0.0375	0.0117	0.0275	0.0913	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0021
90	235	0.0149	0.0375	0.0117	0.0275	0.0913	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0021
91	235	0.0149	0.0375	0.0117	0.0275	0.0913	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0021
92	235	0.0149	0.0375	0.0117	0.0275	0.0913	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0021
93	235	0.0149	0.0375	0.0117	0.0275	0.0912	0.0026	107	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0021
94	235	0.0149	0.0375	0.0117	0.0275	0.0912	0.0026	107	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0021
95	235	0.0149	0.0375	0.0117	0.0275	0.0912	0.0026	107	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0021
96	235	0.0149	0.0375	0.0117	0.0275	0.0912	0.0026	107	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0021
97	235	0.0149	0.0375	0.0117	0.0275	0.0912	0.0026	107	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0021
98	235	0.0149	0.0375	0.0117	0.0275	0.0912	0.0026	107	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0021
99	235	0.0149	0.0375	0.0117	0.0275	0.0912	0.0026	107	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0021
100	235	0.0149	0.0375	0.0117	0.0275	0.0912	0.0026	107	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0021

Cell 1E Percolate Concentrations at Barrier Wall

Year	SO4 mg/L	Sb mg/L	As mg/L	Cu mg/L	Ni mg/L	Zn mg/L	Co mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Ag mg/L	B mg/L	Be mg/L	Cd mg/L	Pb mg/L	Se mg/L	Tl mg/L
1	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
2	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
3	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
4	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
5	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
6	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
7	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
8	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
9	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
10	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
11	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
12	183	0.0030	0.0021	0.0024	0.0022	0.0101	0.0013	69	81	97	15	0.0010	0.2330	0.0002	0.0002	0.0071	0.0020	0.0020
13	234	0.0149	0.0374	0.0116	0.0272	0.0905	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
14	234	0.0149	0.0374	0.0116	0.0272	0.0905	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
15	234	0.0149	0.0374	0.0116	0.0272	0.0905	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
16	235	0.0149	0.0376	0.0117	0.0275	0.0913	0.0026	107	26	52	11	0.0012	0.2309	0.0005	0.0008	0.0040	0.0018	0.0021
17	244	0.0154	0.0390	0.0126	0.0307	0.0981	0.0028	108	27	53	12	0.0012	0.2338	0.0006	0.0008	0.0041	0.0020	0.0021
18	252	0.0159	0.0405	0.0134	0.0339	0.1052	0.0030	110	28	53	13	0.0012	0.2362	0.0008	0.0009	0.0041	0.0021	0.0021
19	265	0.0167	0.0426	0.0147	0.0391	0.1160	0.0033	111	30	54	15	0.0013	0.2383	0.0009	0.0009	0.0042	0.0022	0.0021
20	292	0.0180	0.0466	0.0174	0.0497	0.1381	0.0039	114	32	55	18	0.0014	0.2409	0.0013	0.0010	0.0043	0.0026	0.0021
21	328	0.0199	0.0522	0.0211	0.0641	0.1681	0.0048	117	36	56	23	0.0015	0.2441	0.0017	0.0011	0.0044	0.0031	0.0020
22	357	0.0214	0.0566	0.0240	0.0759	0.1925	0.0055	119	38	57	27	0.0016	0.2462	0.0021	0.0012	0.0045	0.0035	0.0020
23	380	0.0226	0.0601	0.0263	0.0851	0.2115	0.0060	121	41	57	30	0.0017	0.2477	0.0024	0.0012	0.0046	0.0038	0.0020
24	397	0.0234	0.0627	0.0281	0.0921	0.2260	0.0065	122	42	58	32	0.0017	0.2489	0.0027	0.0013	0.0047	0.0040	0.0020
25	407	0.0240	0.0643	0.0291	0.0963	0.2347	0.0067	123	43	58	34	0.0018	0.2497	0.0028	0.0013	0.0047	0.0041	0.0020
26	408	0.0240	0.0645	0.0293	0.0968	0.2357	0.0067	123	43	58	34	0.0018	0.2498	0.0028	0.0013	0.0047	0.0042	0.0020
27	403	0.0238	0.0637	0.0287	0.0945	0.2311	0.0066	123	43	58	33	0.0017	0.2498	0.0028	0.0013	0.0047	0.0041	0.0020
28	394	0.0234	0.0623	0.0276	0.0899	0.2220	0.0063	124	42	59	32	0.0017	0.2524	0.0026	0.0013	0.0047	0.0040	0.0021
29	374	0.0223	0.0592	0.0254	0.0810	0.2039	0.0058	123	40	59	29	0.0017	0.2531	0.0023	0.0012	0.0047	0.0037	0.0021
30	329	0.0199	0.0522	0.0209	0.0634	0.1670	0.0048	118	36	57	23	0.0015	0.2465	0.0017	0.0011	0.0045	0.0031	0.0021
31	293	0.0180	0.0467	0.0174	0.0498	0.1382	0.0039	114	32	55	18	0.0014	0.2410	0.0013	0.0010	0.0043	0.0026	0.0021
32	270	0.0168	0.0430	0.0151	0.0405	0.1188	0.0034	111	30	54	15	0.0013	0.2375	0.0010	0.0009	0.0042	0.0023	0.0021
33	252	0.0158	0.0402	0.0134	0.0339	0.1049	0.0030	109	28	53	13	0.0012	0.2343	0.0008	0.0009	0.0041	0.0021	0.0021
34	240	0.0152	0.0383	0.0122	0.0295	0.0953	0.0027	107	27	52	12	0.0012	0.2316	0.0006	0.0008	0.0040	0.0019	0.0021
35	237	0.0150	0.0379	0.0120	0.0285	0.0933	0.0026	107	27	52	11	0.0012	0.2311	0.0006	0.0008	0.0040	0.0019	0.0021
36	237	0.0150	0.0378	0.0119	0.0282	0.0928	0.0026	107	27	52	11	0.0012	0.2310	0.0006	0.0008	0.0040	0.0019	0.0021
37	236	0.0150	0.0378	0.0119	0.0281	0.0924	0.0026	107	27	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0019	0.0021
38	236	0.0150	0.0377	0.0118	0.0279	0.0921	0.0026	107	26	52	11	0.0012	0.2309	0.0006	0.0008	0.0040	0.0019	0.0021
39	236	0.0149	0.0377	0.0118	0.0278	0.0919	0.0026	107	26	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0018	0.0021
40	236	0.0149	0.0376	0.0118	0.0278	0.0918	0.0026	107	26	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0018	0.0021
41	235	0.0149	0.0376	0.0118	0.0277	0.0917	0.0026	107	26	52	11	0.0012	0.2308	0.0006	0.0008	0.0040	0.0018	0.0021
42	235	0.0149	0.0376	0.0117	0.0277	0.0915	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
43	235	0.0149	0.0376	0.0117	0.0276	0.0915	0.0026	107	26	52	11	0.0012	0.2307	0.0006	0.0008	0.0040	0.0018	0.0021
44	235	0.0149	0.0376	0.0117	0.0276	0.0914	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0021
45	235	0.0149	0.0375	0.0117	0.0276	0.0913	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0022
46	235	0.0149	0.0375	0.0117	0.0275	0.0913	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0022
47	235	0.0149	0.0375	0.0117	0.0275	0.0912	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0022
48	235	0.0149	0.0375	0.0117	0.0275	0.0912	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0022
49	235	0.0149	0.0375	0.0117	0.0275	0.0911	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0022
50	235	0.0149	0.0375	0.0117	0.0274	0.0911	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0022

Cell 1E Percolate Concentrations at Barrier Wall (cont.)

Year	SO4 mg/L	Sb mg/L	As mg/L	Cu mg/L	Ni mg/L	Zn mg/L	Co mg/L	Ca mg/L	Mg mg/L	Na mg/L	K mg/L	Ag mg/L	B mg/L	Be mg/L	Cd mg/L	Pb mg/L	Se mg/L	Tl mg/L
51	235	0.0149	0.0375	0.0117	0.0274	0.0911	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0022
52	235	0.0149	0.0375	0.0117	0.0274	0.0910	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0022
53	235	0.0149	0.0375	0.0117	0.0274	0.0910	0.0026	107	26	52	11	0.0012	0.2307	0.0005	0.0008	0.0040	0.0018	0.0022
54	235	0.0149	0.0375	0.0117	0.0274	0.0910	0.0026	107	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
55	235	0.0149	0.0375	0.0117	0.0274	0.0910	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
56	234	0.0149	0.0375	0.0117	0.0274	0.0909	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
57	234	0.0149	0.0375	0.0117	0.0274	0.0909	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
58	234	0.0149	0.0375	0.0117	0.0274	0.0909	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
59	234	0.0149	0.0375	0.0117	0.0273	0.0909	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
60	234	0.0149	0.0375	0.0117	0.0273	0.0909	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
61	234	0.0149	0.0375	0.0117	0.0273	0.0909	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
62	234	0.0149	0.0375	0.0117	0.0273	0.0909	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
63	234	0.0149	0.0375	0.0117	0.0273	0.0908	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
64	234	0.0149	0.0375	0.0117	0.0273	0.0908	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
65	234	0.0149	0.0375	0.0117	0.0273	0.0908	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
66	234	0.0149	0.0375	0.0117	0.0273	0.0908	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
67	234	0.0149	0.0374	0.0117	0.0273	0.0908	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
68	234	0.0149	0.0374	0.0117	0.0273	0.0908	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
69	234	0.0149	0.0374	0.0117	0.0273	0.0908	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
70	234	0.0149	0.0374	0.0117	0.0273	0.0908	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
71	234	0.0149	0.0374	0.0117	0.0273	0.0908	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
72	234	0.0149	0.0374	0.0117	0.0273	0.0908	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
73	234	0.0149	0.0374	0.0117	0.0273	0.0908	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
74	234	0.0149	0.0374	0.0117	0.0273	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
75	234	0.0149	0.0374	0.0116	0.0273	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
76	234	0.0149	0.0374	0.0116	0.0273	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
77	234	0.0149	0.0374	0.0116	0.0273	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
78	234	0.0149	0.0374	0.0116	0.0273	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
79	234	0.0149	0.0374	0.0116	0.0273	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
80	234	0.0149	0.0374	0.0116	0.0273	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
81	234	0.0149	0.0374	0.0116	0.0273	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
82	234	0.0149	0.0374	0.0116	0.0273	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
83	234	0.0149	0.0374	0.0116	0.0273	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
84	234	0.0149	0.0374	0.0116	0.0273	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
85	234	0.0149	0.0374	0.0116	0.0273	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
86	234	0.0149	0.0374	0.0116	0.0273	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
87	234	0.0149	0.0374	0.0116	0.0273	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
88	234	0.0149	0.0374	0.0116	0.0272	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
89	234	0.0149	0.0374	0.0116	0.0272	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
90	234	0.0149	0.0374	0.0116	0.0272	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
91	234	0.0149	0.0374	0.0116	0.0272	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
92	234	0.0149	0.0374	0.0116	0.0272	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
93	234	0.0149	0.0374	0.0116	0.0272	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
94	234	0.0149	0.0374	0.0116	0.0272	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
95	234	0.0149	0.0374	0.0116	0.0272	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
96	234	0.0149	0.0374	0.0116	0.0272	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
97	234	0.0149	0.0374	0.0116	0.0272	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
98	234	0.0149	0.0374	0.0116	0.0272	0.0907	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
99	234	0.0149	0.0374	0.0116	0.0272	0.0906	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022
100	234	0.0149	0.0374	0.0116	0.0272	0.0906	0.0026	106	26	52	11	0.0012	0.2306	0.0005	0.0008	0.0040	0.0018	0.0022

Appendix D.5

Process Water Pond Concentrations

SRK Consulting
Project NorthMet
Number 1UP005.001
Date 7/17/2007 16:37
G:\PolyMet Mining\1UP005.01_Northmet_project_2004\Water_Quality_Predictions\Operational_Tailings\Model\[PolyMet Water Balance_1E-2E_v27.7.xls]Summary
Operational Tailings Pond Chemistry Predictions

Year	Month	Days/Month	Active Cell	Alkalinity	Chloride	Fluoride	Hardness	Nitrate + Nitrite, as N	Nitrogen, Ammonia as N	Sulfate	Aluminum	Antimony	Arsenic	Barium	Beryllium	Boron	Cadmium	Calcium	Chromium	Cobalt	Copper	Iron	Lead	Magnesium
Fixed Standards				#N/A	230	2	#N/A	#N/A	#N/A	250	0.125	0.031	0.148	2	0.004	0.5	f(H)	#N/A	f(H)	0.005	f(H)	0.3	f(H)	#N/A
	Jan-01	31		3.945075	0.218203	0.029773	5.156219	0.000736	0.000736	3.261290649	0.001852629	0.000205	0.000476	0.000798	8.1911E-06	0.0036	1.73119E-05	2.813844	4.13462E-05	1.9679E-05	8.97333E-05	0.002863	4.45569E-05	0.600919342
	Feb-01	28		13.63536	2.310071	0.058932	13.54723	0.005537	0.005537	9.072334377	0.008013894	0.000485	0.001162	0.001736	1.59447E-05	0.011783	3.51952E-05	6.002242	8.9267E-05	4.65768E-05	0.00020271	0.004	8.54772E-05	1.525753728
	Mar-01	31		23.24376	3.691047	0.096733	24.32412	0.010854	0.010854	15.34508124	0.01	0.000818	0.001756	0.00366	2.87035E-05	0.020543	5.14765E-05	9.350448	0.000200868	9.35517E-05	0.000408972	0.004	0.000135454	2.80828166
	Apr-01	30		28.48159	4.408889	0.128476	31.70562	0.01451	0.01451	20.94350833	0.01	0.001126	0.002282	0.005365	4.14477E-05	0.027212	6.64298E-05	12.26035	0.000297774	0.000132642	0.000626857	0.004	0.000184292	3.591592596
	May-01	31		35.72447	4.682754	0.166703	41.52043	0.017659	0.017659	25.43353477	0.01	0.001403	0.002732	0.00725	5.4173E-05	0.033262	8.03914E-05	15.20342	0.000409774	0.000178698	0.000811779	0.004	0.000232365	4.859327366
1	Jun-01	30	1E	42.62871	4.92575	0.203898	51.97791	0.02208	0.02208	30.5815172	0.01	0.001711	0.003133	0.009731	6.93183E-05	0.040385	9.10308E-05	17.89931	0.000561008	0.000234418	0.001071008	0.004	0.000279811	6.169362096
	Jul-01	31		47.8589	5.144675	0.238316	60.88993	0.026335	0.026335	35.84163719	0.01	0.002021	0.003527	0.012191	8.45068E-05	0.047406	0.000102331	20.56792	0.000710496	0.000285098	0.001343365	0.004	0.000328609	7.20958963
	Aug-01	31		57.284	5.133295	0.27777	74.11839	0.031192	0.031192	39.83408493	0.01	0.002282	0.003772	0.014996	9.90842E-05	0.054369	0.000107849	22.81179	0.000888161	0.000350569	0.001580164	0.004	0.000367056	9.057111669
	Sep-01	30		61.4488	5.123965	0.302575	80.28009	0.032957	0.032957	42.53058307	0.01	0.002457	0.004051	0.016265	0.000107642	0.057963	0.000116548	24.66791	0.000964365	0.000380573	0.001705056	0.004	0.000398508	9.859361543
	Oct-01	31		63.30855	5.248693	0.326068	84.56693	0.034613	0.034613	46.95852832	0.01	0.002707	0.004437	0.017351	0.000118187	0.061634	0.000127658	26.7572	0.00102021	0.00041174	0.001884789	0.004	0.00043771	10.26254118
	Nov-01	30		64.1564	5.29722	0.343659	87.45716	0.035697	0.035697	50.62140249	0.01	0.002913	0.004757	0.018114	0.000126719	0.064215	0.000136206	28.34643	0.001056751	0.000436479	0.002033254	0.004	0.000468672	10.49803002
	Dec-01	31		65.94049	5.376848	0.364799	90.8323	0.036493	0.036493	53.74076007	0.01	0.003098	0.005106	0.018861	0.000134423	0.067006	0.000148015	30.33809	0.001094815	0.000455883	0.002146046	0.004	0.000503442	10.83046585
	Jan-02	31		67.33521	5.450092	0.384551	94.63569	0.038214	0.038214	57.85952425	0.01	0.003336	0.005414	0.019947	0.00014446	0.070398	0.000160008	32.49285	0.001150423	0.000484324	0.002309389	0.004	0.000540431	11.14709647
	Feb-02	28		68.33157	5.526707	0.400954	98.12323	0.040509	0.040509	61.45378476	0.01	0.003547	0.005679	0.021326	0.000154259	0.074283	0.0001692	34.28849	0.001230398	0.000510212	0.002490131	0.004	0.000572742	11.41479698
	Mar-02	31		69.49525	5.60916	0.418779	101.8043	0.042781	0.042781	65.19681312	0.01	0.003762	0.005924	0.022701	0.000164144	0.078199	0.000178999	36.1722	0.001309025	0.000537262	0.002668639	0.004	0.000606639	11.71433453
	Apr-02	30		79.24993	5.487356	0.446067	113.6846	0.046614	0.046614	66.0554103	0.01	0.003858	0.005909	0.024853	0.00017181	0.082851	0.000178627	37.32414	0.001454037	0.000585409	0.002765351	0.004	0.000620525	13.54485824
	May-02	31		81.19909	5.515659	0.462119	117.3938	0.048205	0.048205	68.58121369	0.01	0.004011	0.006095	0.025882	0.000178969	0.085755	0.000186616	38.87006	0.001513684	0.000606987	0.002879729	0.004	0.000646464	13.94539648
2	Jun-02	30	1E	84.0175	5.584893	0.482601	122.1282	0.049811	0.049811	71.7789667	0.01	0.004203	0.006368	0.026902	0.000187373	0.088913	0.000196284	40.73866	0.001569444	0.000634417	0.003006118	0.004	0.000677594	14.48436482
	Jul-02	31		89.64307	5.47224	0.499441	128.9785	0.051572	0.051572	72.53643934	0.01	0.004271	0.006378	0.027917	0.000191815	0.091058	0.000196602	41.45544	0.001635265	0.000664069	0.003053151	0.004	0.000686763	15.55032447
	Aug-02	31		92.29176	5.483193	0.515021	133.1264	0.052987	0.052987	74.66293825	0.01	0.004402	0.006532	0.028797	0.000197952	0.093496	0.000202456	42.7242	0.001686138	0.000686694	0.003144996	0.004	0.000707824	16.06955001
	Sep-02	30		93.39816	5.566662	0.531919	135.6895	0.053618	0.053618	77.77804962	0.01	0.004583	0.006841	0.029331	0.000204994	0.09568	0.000213894	44.58797	0.001708241	0.000704409	0.003244909	0.004	0.000739096	16.2710788
	Oct-02	31		94.6392	5.666038	0.550748	138.6729	0.054416	0.054416	81.49203534	0.01	0.004794	0.007171	0.02992	0.000213097	0.09808	0.000225628	46.55716	0.00173129	0.000727038	0.003365203	0.004	0.000773296	16.51509353
	Nov-02	30		94.89901	5.73259	0.563322	140.5185	0.055357	0.055357	84.39863688	0.01	0.004956	0.007406	0.030547	0.000219765	0.100217	0.000233698	47.94801	0.0017614	0.000744005	0.003479942	0.004	0.000799034	16.61040091
	Dec-02	31		95.29264	5.80112	0.575901	142.292	0.056202	0.056202	86.94150272	0.01	0.005101	0.007622	0.031166	0.00022578	0.102343	0.000242464	49.3952	0.001792578	0.000757753	0.003577685	0.004	0.000824563	16.70825042
	Jan-03	31		95.22683	5.846767	0.585671	143.784	0.057298	0.057298	89.58549554	0.01	0.005254	0.007784	0.031847	0.000232051	0.104426	0.000251851	50.92002	0.001826309	0.000771855	0.003678373	0.004	0.000849462	16.73233763
	Feb-03	28		94.41573	5.853973	0.589412	144.1339	0.058161	0.058161	91.24602679	0.01	0.00535	0.007888	0.032373	0.000236306	0.105847	0.000257121	51.77896	0.001854699	0.000779355	0.003758465	0.004	0.000864378	16.63366913
	Mar-03	31		93.14757	5.831739	0.590702	143.7981	0.058739	0.058739	92.50305678	0.01	0.005419	0.007918	0.03274	0.000239287	0.106728	0.000261112	52.37638	0.001873466	0.000783538	0.00381757	0.004	0.000875346	16.46262742
	Apr-03	30		102.6669	5.597802	0.600124	152.9479	0.06075	0.06075	89.14516581	0.01	0.005279	0.007579	0.033734	0.000236778	0.107737	0.000249928	51.58867	0.001956656	0.000806432	0.003726122	0.004	0.000852648	18.16148573
	May-03	31		102.6731	5.651367	0.610334	153.9405	0.061051	0.061051	91.35254837	0.01	0.005409	0.007779	0.034059	0.00024158	0.109155	0.00026061	53.16528	0.001967555	0.000814543	0.003786474	0.004	0.00087722	18.14782484
3	Jun-03	30	1E	104.8211	5.67301	0.623637	157.1347	0.061793	0.061793	93.25889598	0.01	0.005528	0.007946	0.034558	0.000246458	0.110789	0.000268114	54.51267	0.001991982	0.000831327	0.003842566	0.004	0.000897606	18.52975651
	Jul-03	31		106.8554	5.673362	0.6353	160.0154	0.062334	0.062334	94.80721942	0.01	0.005625	0.008071	0.034938	0.000250381	0.112027	0.000274436	55.65136	0.00200964	0.000846071	0.00388186	0.004	0.000914551	18.89273292
	Aug-03	31		107.7883	5.75308	0.649703	162.3506	0.063105	0.063105	97.68466139	0.01	0.005792	0.008311	0.035483	0.000256849	0.114083	0.000285173	57.39736	0.00203298	0.000862577	0.003972485	0.004	0.000943201	19.06353863
	Sep-03	30		108.4447	5.819274	0.662003	164.129	0.063603	0.063603	100.1076677	0.01	0.005935	0.008538	0.035893	0.000262284	0.115759	0.000295387	58.99731	0.002049115	0.00087482	0.004044065	0.004	0.000968792	19.17075376
	Oct-03	31		108.1375	5.819993	0.665884	164.6424	0.064126	0.064126	101.3629188	0.01	0.006003	0.008594	0.036213	0.000265177	0.116642	0.000298524	59.54002	0.002065061	0.000882013	0.004097364	0.004	0.000979498	19.15942376
	Nov-03	30		107.7324	5.891129	0.673592	165.532	0.065093	0.065093	103.5689302	0.01	0.006125	0.008747	0.036811	0.000270369	0.118533	0.000305246	60.64885	0.002096466	0.000892485	0.004192712	0.004	0.000999961	19.12395687
	Dec-03	31		107.8557	5.957231	0.683386	166.6821	0.06563	0.06563	105.6672283	0.01	0.006246	0.008924	0.037233	0.000275096	0.120132	0.000314241	62.02976	0.002115281	0.000901473	0.00426053	0.004	0.001022646	19.14416099
	Jan-04	31		107.0147	5.979004	0.687112	166.8007	0.066192	0.066192	107.3703183	0.01	0.006343	0.009003	0.037577	0.000278823	0.121201	0.000321651	63.11132	0.002129488	0.000907072	0.004316484	0.004	0.001039321	19.00853009
	Feb-04	28		106.3411	6.011151	0.690332	167.0446	0.066903	0.066903	108.8459881	0.01	0.006432	0.009106	0.037999	0.000282554	0.122452	0.000327772	64.041	0.002151317	0.000912378	0.004379016	0.004	0.00105388	18.89939276
	Mar-04	31		105.72	6.042511	0.694274	167.3334	0.067557	0.067557	110.3910105	0.01	0.00652	0.00918	0.038396	0.000286242	0.123653	0.00033438	65.02407	0.002170787	0.000918186	0.004436871	0.004	0.001069278	18.80083769
	Apr-04	30		108.9994	6.019254	0.701493	170.9876	0.068671	0.068671	110.2936629	0.01	0.006535	0.00915	0.038994	0.000287965	0.124937	0.000335694	65.52373	0.002212009	0.000930134	0.004441317	0.004	0.001072841	19.36705605
	May-04	31		110.289	6.152512	0.718411	173.8119	0.069792	0.069792	113.503138	0.01	0.00673	0.009429	0.039764	0.000295724	0.127771	0.000351358	67.96438	0.002249072	0.000946444	0.004538297	0.004	0.001109109	19.55016473
4	Jun-04	30	1E	111.6437	6.149824	0.723681	175.7029	0.070502	0.070502	114.1234107	0.01	0.006775	0.009459	0.040162	0.000298097	0.12877	0.000354362	68.56028	0.002273298	0.000955187	0.004564556	0.004	0.001116838	19.78973961
	Jul-04	31		112.7116	6.212253	0.731609	177.8401	0.071852	0.071852	115.5793572	0.01	0.006864	0.009532	0.040922	0.000302582	0.13086	0.000359859	69.5747	0.002319461	0.00096713	0.004636345	0.004	0.001132544	19.98953821
	Aug-04	31		114.7119	6.106567	0.727333	179.4545	0.07253	0.07253	113.6064399	0.01	0.006762	0.009298	0.041175	0.000299965	0.13064	0.000351376	68.54393	0.002346789	0.000969116	0.004596002	0.004	0.001114252	20.37543925
	Sep-04	30		113.8374	6.027043	0.720233	178.0984	0.072166	0.072166	112.4076729	0.01	0.006697	0.009181	0.040967	0.000297561	0.129716	0.000348731	68.01852	0.002337698	0.000960612	0.004555986	0.004	0.001104392	20.21989779
	Oct-04	31		113.0818	6.067178	0.725631	178.1619	0.072375	0.072375	114.2396998	0.01	0.0068	0.009315	0.041155	0.000301377	0.130608	0.000356471	69.11221	0.002341372	0.000966324	0.004610005	0.004	0.001122316	20.08562188
	Nov-04	30		112.8537	6.123808	0.733779	178.9375	0.072749	0.072749	116.4487613	0.01	0.006926	0.009498	0.04144	0.000306178	0.131855	0.000365324	70.43288	0.002350039	0.000975666	0.004677145	0.004	0.001143865	20.04161981
	Dec-04	31		112.242	6.148634	0.738169	178.7839	0.072623	0.072623	117.7488612	0.01	0.007003	0.009613	0.041471	0.000308744	0.132331	0.000373019	71.45699	0.002344923	0.000977415	0.004701508	0.004	0.001159091	19.90785941

SRK Consulting
Project NorthMet
Number 1UP005.001
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G:\PolyMet Mining\1UP005.01_Northmet_project_2004\Water_Quality_Predictions\Operational_Tailings\Model\[PolyMet Water Balance_1E-2E_v27.7.xls]Summary
Operational Tailings Pond Chemistry Predictions

Year	Month	Days/Month	Active Cell	Alkalinity	Chloride	Fluoride	Hardness	Nitrate + Nitrite, as N	Nitrogen, Ammonia as N	Sulfate	Aluminum	Antimony	Arsenic	Barium	Beryllium	Boron	Cadmium	Calcium	Chromium	Cobalt	Copper	Iron	Lead	Magnesium
Fixed Standards				#N/A	230	2	#N/A	#N/A	#N/A	250	0.125	0.031	0.148	2	0.004	0.5	f(H)	#N/A	f(H)	0.005	f(H)	0.3	f(H)	#N/A
	Jan-05	31		112.1747	6.252429	0.743479	183.0941	0.075014	0.075014	122.6844468	0.01	0.007202	0.009805	0.04202	0.000313075	0.134998	0.000379941	72.5489	0.002356724	0.00100227	0.004754181	0.004	0.001307614	19.77317775
	Feb-05	28		111.3951	6.306468	0.743143	185.4168	0.076665	0.076665	125.8875309	0.01	0.007323	0.009929	0.042329	0.000315332	0.13659	0.000382805	72.97157	0.002361474	0.001016727	0.004786712	0.004	0.001413941	19.54851796
	Mar-05	31		111.1933	6.388642	0.746905	188.5251	0.078546	0.078546	129.602796	0.01	0.007474	0.010068	0.042798	0.000318863	0.13872	0.000388339	73.83003	0.002374547	0.001035119	0.004832246	0.004	0.001524792	19.42186606
	Apr-05	30		113.8074	6.208763	0.735227	190.9399	0.078882	0.078882	126.7397019	0.01	0.007291	0.009733	0.042482	0.000311071	0.137078	0.000373091	71.70365	0.002365071	0.001034034	0.004688115	0.004	0.001543479	19.82786889
	May-05	31		114.3502	6.34648	0.74634	195.0142	0.080863	0.080863	131.425795	0.01	0.007507	0.009998	0.043129	0.00031721	0.140039	0.000384813	73.51066	0.002385516	0.001057079	0.004760686	0.004	0.001660218	19.80815131
5	Jun-05	30	1E	117.2764	6.203065	0.74081	197.7996	0.081084	0.081084	129.4484868	0.01	0.007385	0.009777	0.042892	0.000311976	0.138858	0.000374187	72.1026	0.002377281	0.001060602	0.004651266	0.004	0.001674349	20.27835153
	Jul-05	31		119.481	6.227848	0.747003	201.8574	0.08223	0.08223	131.2412446	0.01	0.007459	0.009852	0.043154	0.000313214	0.140128	0.000378114	72.79255	0.002384599	0.001075289	0.004632507	0.004	0.001749383	20.55480322
	Aug-05	31		120.5164	6.324034	0.758416	205.7757	0.083646	0.083646	135.2843343	0.01	0.007651	0.010098	0.043582	0.0003186	0.142364	0.000388483	74.40899	0.002392879	0.001097386	0.004686553	0.004	0.001846069	20.63889285
	Sep-05	30		120.5404	6.390604	0.765362	208.164	0.084625	0.084625	138.6219886	0.01	0.007804	0.010302	0.043818	0.000322778	0.143883	0.000396265	75.56777	0.002391785	0.001113519	0.004733411	0.004	0.001924765	20.57216668
	Oct-05	31		119.9637	6.354768	0.76162	208.3631	0.084826	0.084826	139.1241581	0.01	0.007808	0.010266	0.043684	0.000321997	0.143602	0.000394303	75.22078	0.002379503	0.001116122	0.004718695	0.004	0.001955132	20.45209751
	Nov-05	30		119.812	6.452232	0.769287	210.7668	0.086148	0.086148	142.7113955	0.01	0.007976	0.010482	0.04412	0.000327063	0.145658	0.000402794	76.51528	0.002391667	0.001133035	0.004788268	0.004	0.002034655	20.36334063
	Dec-05	31		119.5076	6.509826	0.774265	212.5181	0.086782	0.086782	145.5885673	0.01	0.008104	0.010654	0.044187	0.000329837	0.146754	0.000410966	77.6228	0.00238039	0.001143494	0.004805059	0.004	0.002113286	20.2119279
	Jan-06	31		118.7284	6.54857	0.776186	213.4395	0.087442	0.087442	147.836886	0.01	0.0082	0.010758	0.044299	0.000332191	0.14762	0.000415476	78.23124	0.002376424	0.001152175	0.004835053	0.004	0.002170286	20.03052294
	Feb-06	28		118.285	6.601249	0.778748	214.6618	0.088303	0.088303	149.9623976	0.01	0.008296	0.010894	0.044545	0.000334977	0.148812	0.000419839	78.87475	0.002383071	0.001161259	0.004875777	0.004	0.002223555	19.90646499
	Mar-06	31		117.697	6.646898	0.780656	215.6802	0.089091	0.089091	151.9836718	0.01	0.008382	0.010981	0.044742	0.000337263	0.149831	0.000423901	79.43995	0.00238643	0.001169485	0.004906933	0.004	0.002276523	19.76018406
	Apr-06	30		120.5038	6.56848	0.778622	217.9873	0.089418	0.089418	150.0520261	0.01	0.00828	0.010812	0.044776	0.000333789	0.149401	0.000416688	78.55643	0.002397865	0.001170877	0.004834231	0.004	0.002271005	20.22556342
	May-06	31		119.9836	6.526555	0.773214	217.5969	0.089398	0.089398	149.6152547	0.01	0.008241	0.010728	0.04461	0.000331849	0.148848	0.000413596	78.01568	0.002388217	0.001167968	0.004802313	0.004	0.002283137	20.11569
6	Jun-06	30	1E	120.6682	6.541823	0.775702	219.305	0.089937	0.089937	150.6998283	0.01	0.008287	0.010785	0.044715	0.000332769	0.149474	0.000416092	78.40122	0.002389397	0.001174917	0.004799034	0.004	0.002323746	20.17739639
	Jul-06	31		122.657	6.480806	0.774796	221.0738	0.089936	0.089936	149.6102212	0.01	0.008226	0.010686	0.04458	0.000329991	0.148951	0.000412838	77.97234	0.002384047	0.001175551	0.004727797	0.004	0.002333254	20.47242224
	Aug-06	31		123.7414	6.453873	0.77475	222.4191	0.090274	0.090274	149.4217808	0.01	0.008209	0.010636	0.044632	0.000329316	0.148997	0.000410602	77.72023	0.002388459	0.001179741	0.004708196	0.004	0.00234513	20.64665595
	Sep-06	30		123.1335	6.469164	0.77502	222.7754	0.090601	0.090601	150.6620185	0.01	0.008261	0.010698	0.044658	0.000330597	0.149383	0.000412683	77.97856	0.002384084	0.001184395	0.004725462	0.004	0.002377743	20.51433269
	Oct-06	31		122.6087	6.409971	0.770045	222.0636	0.09027	0.09027	150.0643463	0.01	0.008218	0.010617	0.04439	0.000328784	0.14848	0.000408841	77.34217	0.002368454	0.0011818	0.004698187	0.004	0.002375693	20.43121892
	Nov-06	30		121.4887	6.458886	0.772789	222.5088	0.09068	0.09068	152.4769393	0.01	0.008328	0.01077	0.044437	0.00033173	0.149281	0.000414862	78.13241	0.002359528	0.001189443	0.004737398	0.004	0.002427996	20.19045039
	Dec-06	31		121.0569	6.507633	0.777586	223.7061	0.090941	0.090941	155.0166703	0.01	0.008444	0.010941	0.044385	0.000334202	0.15002	0.00042316	79.22105	0.002341613	0.001197302	0.004747834	0.004	0.002493869	20.02285982
	Jan-07	31		120.4306	6.544854	0.780905	224.5019	0.091297	0.091297	157.1995954	0.01	0.008543	0.011069	0.044408	0.000336756	0.150699	0.000428779	79.974	0.002331763	0.001205444	0.004775106	0.004	0.002543174	19.86701533
	Feb-07	28		119.7033	6.575413	0.781988	224.8359	0.091643	0.091643	158.7054902	0.01	0.008612	0.011183	0.044463	0.000338675	0.151273	0.000432264	80.43236	0.002328271	0.001210247	0.004802197	0.004	0.00257863	19.70503567
	Mar-07	31		119.5246	6.637732	0.78674	226.2244	0.092406	0.092406	160.9469445	0.01	0.008716	0.01131	0.044707	0.000341798	0.152513	0.00043818	81.29512	0.002333822	0.001220143	0.004841267	0.004	0.002629613	19.62652936
	Apr-07	30		125.2746	6.464916	0.782969	229.1701	0.092182	0.092182	155.346027	0.01	0.00845	0.010919	0.04467	0.000334017	0.150875	0.000421275	79.21493	0.002358891	0.001214974	0.004699183	0.004	0.002548849	20.66487727
	May-07	31		125.3158	6.565967	0.790738	231.3103	0.093098	0.093098	158.4268758	0.01	0.008596	0.011125	0.044956	0.000337889	0.152636	0.000431855	80.7114	0.002362161	0.001226139	0.004732566	0.004	0.002626852	20.5639675
7	Jun-07	30	1E	125.9711	6.637514	0.798662	233.6995	0.093926	0.093926	160.989799	0.01	0.008722	0.011305	0.045233	0.000341607	0.154128	0.000439595	81.88527	0.002367996	0.00123925	0.004768682	0.004	0.002686582	20.60680387
	Jul-07	31		126.8027	6.675923	0.802674	235.4408	0.094691	0.094691	162.0256575	0.01	0.008771	0.011352	0.045534	0.000343371	0.155163	0.000442007	82.32607	0.002383823	0.001247304	0.00478864	0.004	0.002715181	20.72358282
	Aug-07	31		128.2184	6.545152	0.792941	234.6459	0.094248	0.094248	157.8343799	0.01	0.008561	0.011019	0.045384	0.000337486	0.153457	0.00042725	80.29901	0.002395688	0.001236377	0.004715411	0.004	0.00264062	21.05515142
	Sep-07	30		127.5676	6.620481	0.796622	235.5059	0.094887	0.094887	160.0171217	0.01	0.008663	0.011167	0.045579	0.000340206	0.154672	0.000434369	81.26296	0.002397897	0.001242255	0.004746542	0.004	0.002695359	20.87020111
	Oct-07	31		126.505	6.616424	0.794707	234.8274	0.094855	0.094855	160.5620883	0.01	0.00868	0.011175	0.045492	0.000340635	0.154546	0.000434505	81.2	0.002389604	0.001242722	0.004758442	0.004	0.002706926	20.68845729
	Nov-07	30		124.1003	6.615207	0.789393	232.7494	0.094651	0.094651	161.0774242	0.01	0.008693	0.011185	0.045324	0.000340821	0.154196	0.000434305	80.99096	0.002376284	0.001238554	0.004778787	0.004	0.002715401	20.28174548
	Dec-07	31		123.087	6.651235	0.790356	232.7192	0.094725	0.094725	162.5856459	0.01	0.008757	0.011279	0.045244	0.000341966	0.154555	0.000439678	81.61912	0.002362375	0.001239853	0.0047826	0.004	0.002757915	20.04057454
	Jan-08	31		122.8876	6.719016	0.796157	234.1204	0.095428	0.095428	165.0341279	0.01	0.008874	0.011431	0.045478	0.000345542	0.155837	0.000446519	82.61468	0.002366223	0.001250489	0.004827305	0.004	0.002809461	19.95911754
	Feb-08	28		122.1125	6.751503	0.7967	234.1328	0.095749	0.095749	166.2726716	0.01	0.008936	0.011542	0.045563	0.000347508	0.156401	0.00044956	82.99939	0.00236694	0.001254462	0.004861883	0.004	0.002833938	19.78266258
	Mar-08	31		121.8888	6.819087	0.800784	235.1963	0.096523	0.096523	168.2707622	0.01	0.009036	0.011675	0.045851	0.000350754	0.157684	0.000455336	83.81894	0.002377754	0.001263682	0.004910386	0.004	0.002874441	19.67375282
	Apr-08	30		127.5655	6.644811	0.796186	237.5116	0.09614	0.09614	162.1885592	0.01	0.008758	0.011284	0.045817	0.00034292	0.155896	0.000437777	81.6273	0.002406352	0.001256783	0.004778238	0.004	0.00277212	20.69010007
	May-08	31		126.8877	6.728701	0.800328	238.3499	0.096883	0.096883	164.6528422	0.01	0.008881	0.011461	0.046066	0.000346541	0.157283	0.000445816	82.70224	0.002411951	0.001265468	0.004829124	0.004	0.00282432	20.46776716
8	Jun-08	30	1E	128.121	6.615731	0.793426	237.7032	0.096108	0.096108	161.6520797	0.01	0.008741	0.011285	0.045719	0.000342064	0.155583	0.000437052	81.40762	0.002403654	0.001257583	0.004760477	0.004	0.002771152	20.66686858
	Jul-08	31		128.9961	6.753795	0.805577	241.0555	0.097396	0.097396	165.5252883	0.01	0.008938	0.011579	0.046193	0.000347667	0.158093	0.000451234	83.46839	0.002416299	0.001274824	0.004813651	0.004	0.002862222	20.65390644
	Aug-08	31		130.3341	6.820229	0.81371	243.6653	0.098399	0.098399	167.5999667	0.01	0.009054	0.011744	0.046611	0.000351775	0.15971	0.000457265	84.44859	0.002436127	0.001290806	0.004868339	0.004	0.002900132	20.79826682
	Sep-08	30		130.0372	6.928606	0.820118	245.1563	0.099603	0.099603	170.4835552	0.01	0.009205	0.011963	0.047112	0.000357006	0.161754	0.000465492	85.65415	0.002458319	0.001304675	0.004954513	0.004	0.002951555	20.65453775
	Oct-08	31		128.8043	6.964006	0.818513	244.4033	0.100036	0.100036	171.3805668	0.01	0.009253	0.012023	0.047285	0.000358966	0.162334	0.000467004	85.78501	0.002468176	0.001307806	0.005006807	0.004	0.00296	20.39609768
	Nov-08	30		126.4553	6.964519	0.811352	241.8834	0.099963	0.099963	171.3140273	0.01	0.00925	0.012022	0.047231	0.000359168	0.162058	0.000465318	85.35073	0.002467502	0.001303041	0.005043958	0.004	0.00294779	19.96763553
	Dec-08	31		124.963	6.979112	0.807955	240.5968	0.099822	0.099822	171.7793055	0.01	0.009269	0.012062	0.047097	0.000359165	0.161943	0.000467681	85.48255	0.002456394	0.001299371	0.005047233	0.004	0.00296249	19.63588035

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Operational Tailings Pond Chemistry Predictions

Year	Month	Days/Month	Active Cell	Alkalinity	Chloride	Fluoride	Hardness	Nitrate + Nitrite, as N	Nitrogen, Ammonia as N	Sulfate	Aluminum	Antimony	Arsenic	Barium	Beryllium	Boron	Cadmium	Calcium	Chromium	Cobalt	Copper	Iron	Lead	Magnesium
Fixed Standards				#N/A	230	2	#N/A	#N/A	#N/A	250	0.125	0.031	0.148	2	0.004	0.5	f(H)	#N/A	f(H)	0.005	f(H)	0.3	f(H)	#N/A
9	Jan-09	31	1E	124.963	6.979112	0.807955	240.5968	0.099822	0.099822	171.7793055	0.01	0.009269	0.012062	0.047097	0.000359165	0.161943	0.000467681	85.48255	0.002456394	0.001299371	0.005047233	0.004	0.00296249	19.63588035
	Feb-09	28		124.963	6.979112	0.807955	240.5968	0.099822	0.099822	171.779305	0.01	0.009269	0.012062	0.047097	0.000359165	0.161943	0.000467681	85.48255	0.002456394	0.001299371	0.005047233	0.004	0.00296249	19.63588035
	Mar-09	31		124.963	6.979112	0.807955	240.5968	0.099822	0.099822	171.7793055	0.01	0.009269	0.012062	0.047097	0.000359165	0.161943	0.000467681	85.48255	0.002456394	0.001299371	0.005047233	0.004	0.00296249	19.63588035
	Apr-09	30		124.963	6.979112	0.807955	240.5968	0.099822	0.099822	171.7793055	0.01	0.009269	0.012062	0.047097	0.000359165	0.161943	0.000467681	85.48255	0.002456394	0.001299371	0.005047233	0.004	0.00296249	19.63588035
	May-09	31		124.963	6.979112	0.807955	240.5968	0.099822	0.099822	171.7793055	0.01	0.009269	0.012062	0.047097	0.000359165	0.161943	0.000467681	85.48255	0.002456394	0.001299371	0.005047233	0.004	0.00296249	19.63588035
	Jun-09	30		124.963	6.979112	0.807955	240.5968	0.099822	0.099822	171.7793055	0.01	0.009269	0.012062	0.047097	0.000359165	0.161943	0.000467681	85.48255	0.002456394	0.001299371	0.005047233	0.004	0.00296249	19.63588035
	Jul-09	31		124.963	6.979112	0.807955	240.5968	0.099822	0.099822	171.7793055	0.01	0.009269	0.012062	0.047097	0.000359165	0.161943	0.000467681	85.48255	0.002456394	0.001299371	0.005047233	0.004	0.00296249	19.63588035
	Aug-09	31		124.963	6.979112	0.807955	240.5968	0.099822	0.099822	171.7793055	0.01	0.009269	0.012062	0.047097	0.000359165	0.161943	0.000467681	85.48255	0.002456394	0.001299371	0.005047233	0.004	0.00296249	19.63588035
	Sep-09	30		124.963	6.979112	0.807955	240.5968	0.099822	0.099822	171.7793055	0.01	0.009269	0.012062	0.047097	0.000359165	0.161943	0.000467681	85.48255	0.002456394	0.001299371	0.005047233	0.004	0.00296249	19.63588035
	Oct-09	31		124.963	6.979112	0.807955	240.5968	0.099822	0.099822	171.7793055	0.01	0.009269	0.012062	0.047097	0.000359165	0.161943	0.000467681	85.48255	0.002456394	0.001299371	0.005047233	0.004	0.00296249	19.63588035
	Nov-09	30		124.963	6.979112	0.807955	240.5968	0.099822	0.099822	171.7793055	0.01	0.009269	0.012062	0.047097	0.000359165	0.161943	0.000467681	85.48255	0.002456394	0.001299371	0.005047233	0.004	0.00296249	19.63588035
	Dec-09	31		124.963	6.979112	0.807955	240.5968	0.099822	0.099822	171.77930547	0.01	0.009269	0.012062	0.047097	0.000359165	0.161943	0.000467681	85.48255	0.002456394	0.001299371	0.005047233	0.004	0.00296249	19.63588035
1	Jan-01	31	2E	52.98832	15.47936	0.048311	34.8064	0.034056	0.034056	26.17050273	0.01	0.000938	0.002226	0.002548	9.32819E-06	0.043076	3.54614E-05	7.922838	0.000124717	9.18674E-05	0.000312272	0.004	4.91792E-05	3.659993001
	Feb-01	28		41.90682	11.10256	0.089404	39.32969	0.034303	0.034303	31.69530103	0.01	0.001396	0.002241	0.007425	4.2938E-05	0.044832	3.69856E-05	9.654153	0.000424038	0.000185749	0.000981141	0.004	0.000126933	4.003542644
	Mar-01	31		37.28139	8.37097	0.129903	46.10867	0.036242	0.036242	37.03453087	0.01	0.001798	0.002379	0.011605	7.021E-05	0.048858	4.22867E-05	11.81883	0.00068188	0.000265629	0.001509759	0.004	0.000195413	4.704923251
	Apr-01	30		35.9201	7.076689	0.158603	50.1364	0.035545	0.035545	41.44428359	0.01	0.002091	0.002702	0.012748	8.48873E-05	0.049475	5.07643E-05	13.65288	0.000734675	0.000313508	0.001746992	0.004	0.000240879	5.14577018
	May-01	31		37.67813	6.333112	0.195812	59.37146	0.041276	0.041276	47.01434653	0.01	0.002451	0.002937	0.017042	0.000107344	0.05821	5.86097E-05	16.22488	0.001009224	0.000379625	0.002204407	0.004	0.000300507	6.132645589
	Jun-01	30		40.97544	6.076559	0.230617	69.04737	0.047886	0.047886	52.41421036	0.01	0.002787	0.003174	0.021297	0.000128036	0.067872	6.59966E-05	18.72946	0.001284141	0.000441398	0.002635472	0.004	0.00035495	7.180680722
	Jul-01	31		44.90485	5.877592	0.257423	74.14511	0.046535	0.046535	55.62445207	0.01	0.002986	0.003592	0.020958	0.000134432	0.067752	7.71634E-05	20.64955	0.001241697	0.000471967	0.002666774	0.004	0.000386191	7.920220201
	Aug-01	31		49.44958	5.593391	0.28156	80.85918	0.047992	0.047992	57.40026373	0.01	0.003121	0.003763	0.022248	0.000142124	0.070575	8.28364E-05	22.13049	0.001322192	0.000503825	0.002769275	0.004	0.000411508	8.885061108
	Sep-01	30		54.13037	5.363239	0.30332	85.05209	0.044835	0.044835	58.84893319	0.01	0.003225	0.004117	0.020854	0.000143128	0.067873	9.20244E-05	23.50122	0.001212713	0.000522714	0.002669522	0.004	0.00042859	9.646932258
	Oct-01	31		57.26722	5.346958	0.32562	89.91195	0.045589	0.045589	62.00929393	0.01	0.003415	0.004429	0.021483	0.000150865	0.070047	0.000100554	25.1857	0.001240979	0.000551969	0.00277411	0.004	0.00045807	10.26235447
	Nov-01	30		57.50882	5.318758	0.337896	92.90504	0.048611	0.048611	64.51904057	0.01	0.003563	0.004537	0.023363	0.000159866	0.074492	0.000104205	26.25016	0.001362735	0.000574248	0.002976223	0.004	0.000482354	10.49171114
	Dec-01	31		59.42522	5.353086	0.357464	96.77351	0.049815	0.049815	67.88435944	0.01	0.003754	0.004807	0.024198	0.000168019	0.077163	0.000112422	27.85378	0.001405201	0.000601063	0.003108577	0.004	0.000512336	10.91078788
2	Jan-02	31	2E	57.97508	5.478644	0.364317	100.1805	0.057438	0.057438	70.74819797	0.01	0.003913	0.004682	0.028446	0.000182025	0.086702	0.000110654	28.58354	0.001695604	0.000629011	0.003503972	0.004	0.000538094	11.00351449
	Feb-02	28		59.5543	5.523151	0.380602	103.2425	0.058236	0.058236	73.55423817	0.01	0.004076	0.004944	0.029034	0.000188673	0.088782	0.000118719	30.05113	0.001723258	0.000649647	0.003604951	0.004	0.000564075	11.32231281
	Mar-02	31		61.02391	5.580732	0.397213	106.4855	0.05949	0.05949	76.58978398	0.01	0.004251	0.005191	0.029861	0.000196173	0.091408	0.000126993	31.6001	0.001766777	0.000672086	0.003728192	0.004	0.000591882	11.63934333
	Apr-02	30		66.8149	5.631834	0.42251	114.2684	0.062032	0.062032	79.10437127	0.01	0.004418	0.005393	0.031369	0.00020472	0.095494	0.000133858	33.3613	0.001860126	0.000708071	0.003849618	0.004	0.000619378	12.70929823
	May-02	31		72.18724	5.60487	0.450912	119.9584	0.060164	0.060164	82.28720438	0.01	0.004614	0.005848	0.030559	0.000209603	0.094816	0.000147791	35.59733	0.001783855	0.000736902	0.003822065	0.004	0.000651231	13.56737071
	Jun-02	30		76.18578	5.686504	0.475532	125.8046	0.061476	0.061476	85.86132688	0.01	0.004826	0.006172	0.03139	0.000218291	0.097708	0.000157991	37.60753	0.001824155	0.000769966	0.003937725	0.004	0.000684654	14.29969208
	Jul-02	31		78.14414	5.633855	0.482908	128.9819	0.06347	0.06347	86.20636436	0.01	0.004861	0.00613	0.032589	0.000221948	0.100259	0.000158361	38.06089	0.001906829	0.000782973	0.004012038	0.004	0.000692638	14.7276002
	Aug-02	31		83.74015	5.610476	0.510305	135.3258	0.062731	0.062731	89.16811031	0.01	0.005048	0.006508	0.032363	0.000227698	0.100712	0.000170247	40.15166	0.001873271	0.000814304	0.004019461	0.004	0.000722433	15.66400054
	Sep-02	30		87.78041	5.62625	0.534669	140.5179	0.062554	0.062554	92.44465366	0.01	0.005244	0.006867	0.032425	0.000234332	0.101831	0.000181532	42.13643	0.001859559	0.000843344	0.004070033	0.004	0.000753887	16.36282971
	Oct-02	31		86.06705	5.706687	0.534013	141.1314	0.066401	0.066401	93.61140485	0.01	0.005303	0.006757	0.034523	0.000240561	0.106453	0.000179485	42.25701	0.002004489	0.000852602	0.004266102	0.004	0.000763953	16.23696346
	Nov-02	30		84.80577	5.727879	0.534609	141.2648	0.06833	0.06833	94.52421523	0.01	0.005352	0.006729	0.035634	0.000244473	0.10894	0.000179932	42.51737	0.002079624	0.000857622	0.004378367	0.004	0.000772893	16.1172876
	Dec-02	31		85.18787	5.780552	0.54474	143.1757	0.06972	0.06972	96.6860728	0.01	0.005473	0.006864	0.036488	0.000250206	0.111342	0.000185143	43.53835	0.002129715	0.000872345	0.004488686	0.004	0.000793627	16.2524609
3	Jan-03	31	2E	83.42285	5.822273	0.542899	142.844	0.071993	0.071993	97.35560395	0.01	0.005504	0.006788	0.037758	0.000253844	0.114112	0.000184127	43.58537	0.00221776	0.000875023	0.004610588	0.004	0.000800306	16.05590637
	Feb-03	28		83.30053	5.806675	0.548323	143.3022	0.071977	0.071977	98.530096	0.01	0.005573	0.006903	0.037865	0.000256375	0.114621	0.000189032	44.31728	0.002218974	0.000880015	0.004643621	0.004	0.000813003	16.04869931
	Mar-03	31		83.52702	5.761069	0.555781	143.8142	0.071107	0.071107	99.88017889	0.01	0.005654	0.007067	0.037542	0.000258402	0.114246	0.000196197	45.29078	0.002187864	0.000885134	0.004643196	0.004	0.000827825	16.07210419
	Apr-03	30		89.6778	5.775172	0.580414	151.0012	0.071984	0.071984	101.97922	0.01	0.005799	0.007304	0.038132	0.00026431	0.116288	0.000204998	47.11193	0.002216763	0.000915884	0.004678227	0.004	0.000852701	17.1261504
	May-03	31		95.29421	5.811004	0.611976	158.0444	0.07184	0.07184	106.1781789	0.01	0.006052	0.007752	0.038214	0.000272786	0.117749	0.00022024	49.7821	0.002198735	0.000953982	0.004734226	0.004	0.000893472	18.07928053
	Jun-03	30		98.71216	5.85176	0.631561	162.75	0.072588	0.072588	108.8481007	0.01	0.006215	0.007999	0.038725	0.000279129	0.119663	0.000229565	51.50962	0.002220019	0.000978932	0.004801729	0.004	0.000920557	18.68241802
	Jul-03	31		100.0481	5.899679	0.640094	165.401	0.074364	0.074364	110.2457366	0.01	0.0063	0.008048	0.03972	0.000283858	0.122169	0.000232709	52.3017	0.002283327	0.000993523	0.004892283	0.004	0.000935002	18.97151135
	Aug-03	31		101.8503	5.968842	0.653741	168.7408	0.075883	0.075883	112.6209444	0.01	0.006439	0.008205	0.040602	0.000290155	0.124737	0.000239334	53.62059	0.002333402	0.001013652	0.004994315	0.004	0.000957783	19.32990133
	Sep-03	30		100.2086	5.995093	0.649211	167.9677	0.077578	0.077578	112.6155909	0.01	0.006433	0.008091	0.041516	0.000291803	0.126577	0.000237219	53.41488	0.002399123	0.001011879	0.005070788	0.004	0.000958721	19.12113517
	Oct-03	31		96.79127	5.947629	0.633127	164.4551	0.078475	0.078475	110.7016667	0.01	0.006318	0.007815	0.041981	0.000288962	0.126845	0.000230237	52.23677	0.002442756	0.00099362	0.005081421	0.004	0.000942911	18.58764575
	Nov-03	30		98.21394	5.938553	0.646166	166.4527	0.077864	0.077864	112.770522	0.01	0.006441	0.008047	0.041792	0.000292625	0.127033	0.000239752	53.66926	0.002417252	0.001006059	0.005096809	0.004	0.000964758	18.81098504
	Dec-03	31		97.59964	5.974833	0.649005	166.7518	0.078847	0.078847	113.8286555	0.01	0.006497	0.008077	0.04237	0.000295604	0.128518	0.000242338	54.14648	0.002453118	0.00101083	0.005164711	0.004	0.000975809	18.7497236

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Operational Tailings Pond Chemistry Predictions

Year	Month	Days/Month	Active Cell	Alkalinity	Chloride	Fluoride	Hardness	Nitrate + Nitrite, as N	Nitrogen, Ammonia as N	Sulfate	Aluminum	Antimony	Arsenic	Barium	Beryllium	Boron	Cadmium	Calcium	Chromium	Cobalt	Copper	Iron	Lead	Magnesium
Fixed Standards				#N/A	230	2	#N/A	#N/A	#N/A	250	0.125	0.031	0.148	2	0.004	0.5	f(H)	#N/A	f(H)	0.005	f(H)	0.3	f(H)	#N/A
4	Jan-04	31	2E	94.55573	5.977845	0.636502	163.9883	0.08007	0.08007	112.8322238	0.01	0.006435	0.007893	0.042986	0.000295402	0.129441	0.000237449	53.32594	0.002504675	0.001000202	0.005216492	0.004	0.000965702	18.27222533
	Feb-04	28		94.9227	5.991028	0.643952	164.9834	0.080057	0.080057	114.4433358	0.01	0.006532	0.008059	0.043061	0.000299146	0.130071	0.000244494	54.36482	0.002502054	0.001009136	0.005256115	0.004	0.000981595	18.32017987
	Mar-04	31		94.78446	6.013277	0.648979	165.5626	0.080418	0.080418	115.845813	0.01	0.006615	0.008168	0.043314	0.00030269	0.130988	0.000250229	55.22054	0.00251315	0.001016182	0.005306059	0.004	0.000995238	18.29640152
	Apr-04	30		97.02591	6.112808	0.664007	169.4188	0.082156	0.082156	118.4471146	0.01	0.006771	0.008356	0.044293	0.000310008	0.133914	0.000258212	56.75756	0.002569605	0.001039021	0.005418682	0.004	0.001020122	18.70928284
	May-04	31		100.2384	6.256966	0.688918	174.9149	0.083927	0.083927	122.8809644	0.01	0.007031	0.008715	0.045341	0.000320827	0.137545	0.000272348	59.2945	0.002621276	0.001073501	0.005570131	0.004	0.001061995	19.28831241
	Jun-04	30		96.8487	6.315393	0.670003	172.0594	0.086852	0.086852	120.7846057	0.01	0.006901	0.008339	0.04676	0.000319603	0.139887	0.000260299	57.62636	0.00273588	0.001059377	0.00565712	0.004	0.001041119	18.81009451
	Jul-04	31		99.50966	6.391447	0.688527	176.1094	0.087703	0.087703	123.7677906	0.01	0.007082	0.008604	0.04732	0.000326754	0.142047	0.000271971	59.61199	0.002759476	0.001082562	0.005737742	0.004	0.001071554	19.26031906
	Aug-04	31		98.90892	6.303175	0.680869	174.7223	0.087157	0.087157	122.0474672	0.01	0.006991	0.008459	0.047041	0.000323322	0.140912	0.000269441	59.05808	0.00274763	0.001070681	0.005672165	0.004	0.001059638	19.12867336
	Sep-04	30		102.9218	6.201553	0.701921	178.4792	0.084632	0.084632	124.1961395	0.01	0.007137	0.008816	0.045905	0.000325927	0.139158	0.000285258	61.2871	0.00265188	0.00108697	0.005587663	0.004	0.001087075	19.70090734
	Oct-04	31		104.6356	6.195565	0.713257	180.7185	0.084388	0.084388	125.8505683	0.01	0.00724	0.008987	0.045864	0.000329341	0.139551	0.000293403	62.55101	0.002639223	0.001099378	0.005601079	0.004	0.001105343	19.97174029
	Nov-04	30		103.5606	6.192287	0.709543	179.8448	0.084841	0.084841	125.6766297	0.01	0.007228	0.008927	0.046102	0.00032955	0.139937	0.00029269	62.41218	0.002657368	0.001096204	0.005620721	0.004	0.001104336	19.80475494
	Dec-04	31		102.2214	6.190535	0.705225	178.6901	0.085259	0.085259	125.4587188	0.01	0.007212	0.008858	0.046325	0.000329583	0.140285	0.000291814	62.22624	0.002674914	0.00109177	0.005639934	0.004	0.001103383	19.59421975
5	Jan-05	31	2E	99.84668	6.189428	0.694386	176.2961	0.085941	0.085941	124.4441922	0.01	0.007149	0.008705	0.046639	0.000328712	0.140593	0.000287414	61.45528	0.002704579	0.00108197	0.005663939	0.004	0.001093001	19.20778151
	Feb-05	28		99.14057	6.169728	0.693205	176.0108	0.085751	0.085751	125.0574115	0.01	0.007176	0.008752	0.046464	0.000329174	0.140379	0.000290126	61.70807	0.002689083	0.001082063	0.005656225	0.004	0.00111449	19.04069316
	Mar-05	31		99.81299	6.200163	0.701571	178.1967	0.086112	0.086112	127.6357875	0.01	0.007307	0.008944	0.04653	0.000333106	0.141297	0.000298261	62.90723	0.002679968	0.00109643	0.005686868	0.004	0.001166614	19.09690662
	Apr-05	30		99.90505	6.165261	0.69823	178.8472	0.086338	0.086338	127.6611969	0.01	0.007293	0.008904	0.046436	0.000331765	0.141129	0.000297458	62.71065	0.002672612	0.001096622	0.005649602	0.004	0.001198042	19.05261868
	May-05	31		106.1685	6.314251	0.738622	189.2953	0.087933	0.087933	135.6038871	0.01	0.007726	0.009563	0.04711	0.000346085	0.145063	0.000321887	66.83144	0.002677018	0.001155886	0.005779113	0.004	0.001323939	20.03416052
	Jun-05	30		105.4073	6.259947	0.728268	188.7138	0.088236	0.088236	134.4472598	0.01	0.007641	0.009398	0.047009	0.000342312	0.144505	0.000316111	65.85871	0.002675519	0.001148192	0.005722304	0.004	0.001343977	19.85675191
	Jul-05	31		110.5878	6.303443	0.755373	196.7632	0.08887	0.08887	139.6145018	0.01	0.007915	0.009828	0.047103	0.000350062	0.146355	0.000332216	68.52605	0.00265258	0.001188906	0.005748956	0.004	0.001448984	20.62919502
	Aug-05	31		112.7268	6.3807	0.766637	201.2535	0.090627	0.090627	142.5563308	0.01	0.008058	0.009985	0.047776	0.000355071	0.148742	0.000337865	69.62921	0.002684753	0.001212487	0.005811371	0.004	0.001516916	20.96040264
	Sep-05	30		111.0371	6.391573	0.756028	200.2148	0.091798	0.091798	141.9065762	0.01	0.007997	0.009819	0.048157	0.000353232	0.149343	0.000331577	68.66944	0.002716135	0.001206435	0.005819181	0.004	0.001533885	20.67232342
	Oct-05	31		108.9957	6.302976	0.741695	197.8582	0.091196	0.091196	140.338101	0.01	0.007885	0.009638	0.047604	0.000347663	0.147644	0.000325636	67.47791	0.002684406	0.001190683	0.005729742	0.004	0.001548236	20.26095568
	Nov-05	30		111.4749	6.320333	0.757611	202.61	0.091316	0.091316	144.3201634	0.01	0.008083	0.009956	0.047409	0.00035235	0.148453	0.000337856	69.32816	0.002647592	0.001215971	0.005731553	0.004	0.001639282	20.57073768
	Dec-05	31		109.759	6.33753	0.74976	201.6643	0.092232	0.092232	144.277277	0.01	0.008056	0.009852	0.047688	0.000351448	0.149006	0.000334038	68.70851	0.002668626	0.001212137	0.005746426	0.004	0.001661017	20.27475034
6	Jan-06	31	2E	106.7196	6.327526	0.733678	198.5858	0.092838	0.092838	142.7494	0.01	0.007949	0.009633	0.047831	0.000348194	0.148807	0.0003259	67.32189	0.002688638	0.001197283	0.005741087	0.004	0.001658327	19.75954838
	Feb-06	28		106.711	6.357856	0.736684	199.9977	0.093421	0.093421	144.6149184	0.01	0.008033	0.009746	0.047938	0.000350489	0.149616	0.000329984	67.90339	0.002686714	0.001206689	0.005763366	0.004	0.001708061	19.7036428
	Mar-06	31		105.9208	6.386776	0.735313	200.3542	0.094131	0.094131	145.7540923	0.01	0.008075	0.009775	0.048117	0.000351667	0.150341	0.000331208	68.03801	0.002694118	0.00121069	0.0057856	0.004	0.001745617	19.53400434
	Apr-06	30		107.4622	6.485069	0.745013	204.0627	0.096003	0.096003	148.4571897	0.01	0.00821	0.009922	0.048905	0.00035696	0.152924	0.00033631	69.04918	0.002736518	0.001231559	0.005866509	0.004	0.001802024	19.77763121
	May-06	31		109.556	6.499625	0.756824	207.946	0.096244	0.096244	151.4000386	0.01	0.008355	0.010151	0.048806	0.000360448	0.153655	0.000345283	70.41547	0.00271265	0.001251248	0.005864796	0.004	0.001874551	20.03888788
	Jun-06	30		112.482	6.593622	0.774839	213.6035	0.09778	0.09778	155.5949924	0.01	0.008571	0.010447	0.049389	0.000367677	0.156247	0.000356046	72.28875	0.002731566	0.001282521	0.005940075	0.004	0.001956884	20.4813249
	Jul-06	31		112.3346	6.596625	0.771565	213.8673	0.098415	0.098415	155.3841119	0.01	0.008548	0.010377	0.049594	0.000367054	0.156647	0.000353392	71.91612	0.002747695	0.001282407	0.005939545	0.004	0.001970266	20.44771265
	Aug-06	31		113.0918	6.581531	0.772545	215.1734	0.098609	0.098609	155.7900076	0.01	0.00856	0.010385	0.049549	0.000366822	0.156715	0.000353938	71.99518	0.002741902	0.001286638	0.005917106	0.004	0.001997733	20.53388235
	Sep-06	30		114.3364	6.542322	0.777003	217.1056	0.098176	0.098176	156.9135256	0.01	0.008609	0.010483	0.049155	0.0003669	0.156237	0.000357989	72.52296	0.002706558	0.001294167	0.005871086	0.004	0.002040748	20.66479144
	Oct-06	31		112.9794	6.411756	0.765035	214.5447	0.096519	0.096519	154.8177866	0.01	0.008483	0.010333	0.048187	0.000360552	0.153483	0.000353144	71.44416	0.002647514	0.001276429	0.005748085	0.004	0.002034232	20.36840959
	Nov-06	30		115.0433	6.371465	0.776479	217.9392	0.095746	0.095746	157.5366061	0.01	0.008617	0.010581	0.047571	0.000362621	0.152986	0.000362838	72.79183	0.002587325	0.001293216	0.005702159	0.004	0.002104953	20.6014548
	Dec-06	31		113.8436	6.361934	0.770926	216.8817	0.095907	0.095907	157.2099969	0.01	0.008586	0.01051	0.047568	0.000361506	0.152839	0.000360134	72.30839	0.002589489	0.001288717	0.005698647	0.004	0.002110648	20.39579516
7	Jan-07	31	2E	110.4692	6.338051	0.752641	212.46	0.096068	0.096068	154.480297	0.01	0.008428	0.010229	0.047642	0.00035713	0.152113	0.000349484	70.5466	0.002611397	0.001267004	0.005689361	0.004	0.002070455	19.86687668
	Feb-07	28		109.6019	6.335102	0.749921	211.9696	0.096097	0.096097	154.808986	0.01	0.008435	0.010239	0.04756	0.000357053	0.152026	0.00034969	70.47366	0.002604424	0.001265958	0.00568786	0.004	0.002087274	19.69106277
	Mar-07	31		109.2949	6.375886	0.751745	212.7358	0.096782	0.096782	156.1678284	0.01	0.008496	0.010305	0.047799	0.00035927	0.15295	0.000352151	70.84729	0.002615077	0.001272798	0.005724602	0.004	0.002118243	19.61888909
	Apr-07	30		110.3417	6.453577	0.755904	214.8943	0.098467	0.098467	157.0955722	0.01	0.008544	0.010322	0.0486	0.000362369	0.15503	0.000352208	71.10808	0.002668334	0.001284072	0.005792187	0.004	0.002136414	19.81938358
	May-07	31		116.2509	6.566143	0.790488	224.8824	0.099947	0.099947	164.1772922	0.01	0.008917	0.010887	0.049081	0.000374012	0.158347	0.000373234	74.61707	0.002663634	0.001337472	0.005875061	0.004	0.00227268	20.70336913
	Jun-07	30		114.3221	6.663188	0.780128	222.9416	0.102192	0.102192	162.5344308	0.01	0.008827	0.010643	0.050301	0.000374318	0.160486	0.000363113	73.3456	0.002762534	0.001329784	0.00597592	0.004	0.002232728	20.49803762
	Jul-07	31		113.748	6.732227	0.777152	222.8494	0.103691	0.103691	162.303705	0.01	0.00881	0.010553	0.051058	0.000375479	0.162088	0.000359481	72.96011	0.00281935	0.00133022	0.00603794	0.004	0.002228151	20.44845634
	Aug-07	31		112.2383	6.633665	0.763734	219.7641	0.102495	0.102495	159.4555611	0.01	0.008652	0.01034	0.05043	0.000369165	0.159893	0.000352379	71.63202	0.002789182	0.001309167	0.005939749	0.004	0.002198359	20.16639077
	Sep-07	30		117.1378	6.66762	0.790787	227.5704	0.1027	0.1027	164.8676031	0.01	0.008934	0.010807	0.050274	0.000376581	0.161324	0.000370388	74.46101	0.002747068	0.001348388	0.00594588	0.004	0.002315355	20.85219113
	Oct-07	31		113.6904	6.58352	0.768868	221.908	0.101844	0.101844	160.6048496	0.01	0.0087	0.010448	0.049909	0.000368945	0.159201	0.000357255	72.22979	0.002744703	0.001316554	0.005876433	0.004	0.002249298	20.30953262
	Nov-07	30		113.0888	6.486383	0.764469	220.7043	0.100221	0.100221	160.0015501	0.01	0.008658	0.010441	0.048983	0.000365327	0.157021	0.000357919	72.01039	0.002680682	0.001307635	0.005779042	0.004	0.002261389	20.12587043
	Dec-07	31		113.8388	6.486951	0.770946	222.4675	0.10006	0.10006	161.8229698	0.01	0.008745	0.010586	0.048777	0.00036722	0.157149	0.000363773	72.82326	0.002655959	0.001317571	0.005772558	0.004	0.002306516	20.19162492

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Project NorthMet
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Operational Tailings Pond Chemistry Predictions

Year	Month	Days/Month	Active Cell	Alkalinity	Chloride	Fluoride	Hardness	Nitrate + Nitrite, as N	Nitrogen, Ammonia as N	Sulfate	Aluminum	Antimony	Arsenic	Barium	Beryllium	Boron	Cadmium	Calcium	Chromium	Cobalt	Copper	Iron	Lead	Magnesium
Fixed Standards				#N/A	230	2	#N/A	#N/A	#N/A	250	0.125	0.031	0.148	2	0.004	0.5	f(H)	#N/A	f(H)	0.005	f(H)	0.3	f(H)	#N/A
8	Jan-08	31	2E	111.4517	6.498444	0.757195	218.9701	0.10051	0.10051	160.2382441	0.01	0.008687	0.010521	0.049018	0.000366416	0.157168	0.000358597	71.79759	0.00268709	0.001308414	0.005827092	0.004	0.002262	19.64581263
	Feb-08	28		109.8203	6.489014	0.748227	216.6242	0.100424	0.100424	159.4579378	0.01	0.008664	0.010529	0.048943	0.000365705	0.156767	0.000356871	71.27681	0.002689493	0.001302481	0.005846881	0.004	0.002242002	19.23017144
	Mar-08	31		109.777	6.538841	0.750279	217.3075	0.101116	0.101116	161.0483703	0.01	0.008764	0.010702	0.049199	0.000369237	0.157831	0.000361659	71.88614	0.002702568	0.001313598	0.005912413	0.004	0.002263392	19.07130463
	Apr-08	30		110.1999	6.606682	0.751037	218.1603	0.102458	0.102458	161.6127029	0.01	0.008812	0.010766	0.049857	0.000372244	0.159488	0.00036221	72.04484	0.002749723	0.001322652	0.005991721	0.004	0.002262799	19.06116564
	May-08	31		116.3174	6.728973	0.785699	228.2817	0.104026	0.104026	169.1036708	0.01	0.009226	0.011429	0.050348	0.000385057	0.162991	0.000385794	75.82129	0.002745439	0.00138037	0.006095331	0.004	0.002400654	19.85361143
	Jun-08	30		113.9004	6.695772	0.767957	223.9298	0.104034	0.104034	165.5529716	0.01	0.009051	0.011162	0.050451	0.0003804	0.162121	0.000374496	73.98968	0.002774454	0.001358737	0.006088348	0.004	0.002330394	19.44115342
	Jul-08	31		119.8545	6.855789	0.80242	234.0616	0.106308	0.106308	172.9742446	0.01	0.009461	0.011788	0.051342	0.00039415	0.166468	0.00039641	77.6299	0.002799854	0.001417451	0.006230158	0.004	0.002460619	20.24984248
	Aug-08	31		114.867	6.854077	0.772104	226.1416	0.107076	0.107076	166.9935913	0.01	0.009155	0.011262	0.051986	0.000387247	0.16597	0.000375089	74.3661	0.002882193	0.001378995	0.006265061	0.004	0.002332268	19.52598828
	Sep-08	30		115.1923	6.903322	0.774023	227.0041	0.107913	0.107913	167.7920024	0.01	0.009206	0.011335	0.052373	0.000389569	0.167111	0.000377045	74.68527	0.002907413	0.00138628	0.006313986	0.004	0.00234363	19.51600502
	Oct-08	31		112.5571	6.852205	0.757995	222.6644	0.107297	0.107297	164.8506026	0.01	0.009053	0.011115	0.052122	0.000384588	0.165661	0.000368717	73.19275	0.00290671	0.001363974	0.00627687	0.004	0.002292564	19.05460027
	Nov-08	30		111.3089	6.768706	0.750353	220.4209	0.105825	0.105825	163.7193646	0.01	0.008992	0.011084	0.051321	0.000380779	0.163597	0.000368073	72.74093	0.002855429	0.001351467	0.006197572	0.004	0.002286864	18.74346365
	Dec-08	31		111.8294	6.757081	0.755024	221.5606	0.105333	0.105333	165.3303359	0.01	0.009079	0.011261	0.050948	0.000382253	0.163325	0.000374811	73.55934	0.00282065	0.001359527	0.006183559	0.004	0.002325831	18.69729241
9	Jan-09	31	2E	117.1357	6.816208	0.780604	230.4416	0.102684	0.102684	168.7226514	0.01	0.009161	0.011567	0.049089	0.000370138	0.162613	0.000421775	79.49609	0.002642109	0.00132513	0.005608006	0.004	0.002670624	18.97389023
	Feb-09	28		116.5187	6.767126	0.783436	230.7396	0.102839	0.102839	169.8066709	0.01	0.009205	0.01158	0.049156	0.000371772	0.162785	0.000423574	79.73029	0.00264211	0.001330419	0.005633585	0.004	0.002699205	18.8836711
	Mar-09	31		116.0134	6.688562	0.784763	230.7375	0.102698	0.102698	170.2278502	0.01	0.009214	0.011532	0.049086	0.000372055	0.162436	0.000423703	79.69638	0.002635463	0.001332675	0.005634504	0.004	0.002718118	18.82059621
	Apr-09	30		116.2992	6.666081	0.79334	232.6504	0.103212	0.103212	172.5492556	0.01	0.009324	0.011638	0.049295	0.000375891	0.163308	0.000429074	80.52944	0.002639074	0.001347269	0.005682255	0.004	0.002768391	18.86256635
	May-09	31		112.3647	6.52673	0.772636	226.5368	0.102067	0.102067	168.2609386	0.01	0.009082	0.011227	0.048873	0.000368169	0.160792	0.000415508	78.24488	0.002632398	0.001314182	0.005613675	0.004	0.002700332	18.30389446
	Jun-09	30		113.0866	6.484477	0.782157	228.7572	0.102209	0.102209	170.6106114	0.01	0.009196	0.011358	0.048873	0.000371628	0.161231	0.000421589	79.15578	0.002620523	0.001329652	0.005640887	0.004	0.002752764	18.40081688
	Jul-09	31		115.5956	6.600039	0.805477	235.0692	0.104846	0.104846	176.0780195	0.01	0.009477	0.011668	0.050112	0.000382597	0.165386	0.000434591	81.46674	0.002681534	0.001369414	0.005801525	0.004	0.002851982	18.81157774
	Aug-09	31		111.3461	6.513596	0.780874	228.222	0.104368	0.104368	170.7364682	0.01	0.009185	0.011168	0.050093	0.000374237	0.163556	0.000417317	78.73472	0.002709523	0.001330711	0.005753886	0.004	0.002757903	18.24191012
	Sep-09	30		110.0731	6.395564	0.776634	226.52	0.103173	0.103173	170.1135609	0.01	0.00914	0.011096	0.049483	0.000371772	0.161771	0.000415698	78.27356	0.00266936	0.00132328	0.005703396	0.004	0.002758535	18.02580983
	Oct-09	31		110.9493	6.373987	0.788231	229.2381	0.103495	0.103495	173.0832509	0.01	0.009285	0.011272	0.049554	0.000376292	0.162567	0.000423572	79.4647	0.002659297	0.00134231	0.005744241	0.004	0.002820839	18.13781291
	Nov-09	30		109.3064	6.270901	0.781712	226.9524	0.102447	0.102447	172.0003795	0.01	0.009217	0.011162	0.049042	0.000373344	0.160892	0.000420418	78.78583	0.002628716	0.001331501	0.005698446	0.004	0.002810815	17.87393668
	Dec-09	31		109.9788	6.248506	0.791658	229.2431	0.102715	0.102715	174.5882608	0.01	0.009343	0.011311	0.0491	0.000377256	0.161558	0.000427215	79.80747	0.002619625	0.001347867	0.005734237	0.004	0.002865581	17.95770327
10	Jan-10	31	Combined	108.3513	6.215057	0.788097	229.2884	0.102731	0.102731	172.7602791	0.01	0.009251	0.011093	0.049218	0.00037624	0.16102	0.000429363	78.68031	0.002638414	0.001383094	0.005944104	0.004	0.002840681	17.70701802
	Feb-10	28		108.4224	6.177843	0.79552	232.0441	0.102632	0.102632	173.5936486	0.01	0.009299	0.011121	0.049161	0.000378765	0.160881	0.000438854	78.83954	0.002630403	0.001430337	0.00613927	0.004	0.002867239	17.6752656
	Mar-10	31		109.1634	6.171488	0.80835	236.4112	0.103097	0.103097	175.6079738	0.01	0.009407	0.011207	0.049359	0.000383645	0.161619	0.000451908	79.52254	0.002634193	0.001489441	0.006378481	0.004	0.002914746	17.74877483
	Apr-10	30		108.1457	6.087481	0.804563	236.2502	0.102286	0.102286	173.7009248	0.01	0.009311	0.01103	0.049013	0.000381202	0.160068	0.000453242	78.4895	0.002619421	0.001514961	0.00649905	0.004	0.002891356	17.57651816
	May-10	31		110.5789	6.167805	0.829528	244.1113	0.104186	0.104186	178.3798136	0.01	0.009563	0.011293	0.049888	0.00039169	0.163105	0.000473026	80.3931	0.002657941	0.001594874	0.006821272	0.004	0.002983451	17.92142079
	Jun-10	30		111.0178	6.246307	0.83839	247.6506	0.105941	0.105941	179.4880131	0.01	0.009628	0.011285	0.050826	0.000396673	0.165335	0.000480621	80.68189	0.002720093	0.00164239	0.007077241	0.004	0.003004262	18.01446208
	Jul-10	31		108.9717	6.273587	0.827503	245.6749	0.106728	0.106728	176.2797048	0.01	0.009465	0.010959	0.051404	0.000393952	0.16558	0.000474387	79.02748	0.002779114	0.001649618	0.007216922	0.004	0.00294343	17.76336021
	Aug-10	31		109.0514	6.290485	0.832992	248.02	0.107442	0.107442	176.7348135	0.01	0.009493	0.010928	0.051795	0.000396565	0.166396	0.000480516	79.05379	0.00280513	0.001687576	0.007403858	0.004	0.002956423	17.77663192
	Sep-10	30		105.869	6.106752	0.812618	242.5974	0.104705	0.104705	171.7410917	0.01	0.009229	0.010575	0.050512	0.000386705	0.161929	0.000471451	76.67107	0.002739022	0.001670221	0.007341059	0.004	0.002878316	17.25449269
	Oct-10	31		107.8499	6.109937	0.833682	249.0232	0.105286	0.105286	175.7282467	0.01	0.009441	0.010825	0.050689	0.000394414	0.163206	0.000489778	78.29847	0.002730821	0.001738724	0.007578518	0.004	0.002963	17.49508489
	Nov-10	30		107.252	6.073564	0.834182	249.6273	0.105013	0.105013	175.3609676	0.01	0.009422	0.010763	0.05057	0.000394389	0.162659	0.000493317	77.975	0.002724331	0.001762284	0.007687324	0.004	0.002963453	17.38235808
	Dec-10	31		105.8202	6.065632	0.82773	248.3715	0.105104	0.105104	173.5161908	0.01	0.009327	0.010573	0.050707	0.000392464	0.16232	0.000490796	76.98738	0.002744196	0.001769552	0.007772214	0.004	0.002931263	17.18127751
11	Jan-11	31	Combined	105.6038	6.047887	0.830809	249.7282	0.105135	0.105135	173.7523999	0.01	0.00934	0.010547	0.050725	0.000393753	0.162259	0.000495847	76.9464	0.002744446	0.001797332	0.007895744	0.004	0.002941907	17.12962157
	Feb-11	28		105.5642	6.034304	0.834559	251.2314	0.1052	0.1052	174.1909969	0.01	0.009365	0.010556	0.050754	0.000395355	0.162308	0.000501027	77.0177	0.002744523	0.001823929	0.008011597	0.004	0.002955582	17.10376886
	Mar-11	31		105.479	6.012289	0.837448	252.5227	0.105158	0.105158	174.3446436	0.01	0.009375	0.010531	0.050734	0.000396365	0.162154	0.000505542	76.9676	0.00274238	0.001849167	0.008118858	0.004	0.002964668	17.07427353
	Apr-11	30		107.4537	6.10683	0.857439	258.7791	0.107098	0.107098	178.0626595	0.01	0.009574	0.01073	0.051672	0.000405249	0.165155	0.00052028	78.48322	0.00279085	0.001910341	0.008381851	0.004	0.003035327	17.37601391
	May-11	31		104.0097	6.054597	0.832105	251.9139	0.106161	0.106161	172.2486907	0.01	0.009268	0.010278	0.051404	0.000395737	0.162927	0.000502857	75.7983	0.002802577	0.001866571	0.008288911	0.004	0.002926121	16.90843569
	Jun-11	30		106.9167	6.18038	0.859198	260.2489	0.108678	0.108678	177.4507481	0.01	0.009546	0.010577	0.0526	0.000407592	0.166911	0.000522215	77.981	0.002861715	0.001942896	0.00860503	0.004	0.003023756	17.34982409
	Jul-11	31		104.6466	6.107301	0.841747	255.5585	0.107518	0.107518	173.2352549	0.01	0.009324	0.01026	0.052145	0.000400249	0.164682	0.000510767	76.0497	0.002851255	0.001915025	0.008529987	0.004	0.002948575	17.0302371
	Aug-11	31		106.761	6.170812	0.86223	261.854	0.108974	0.108974	177.051538	0.01	0.009527	0.010478	0.052812	0.000408644	0.167089	0.000526336	77.63088	0.002879476	0.00197627	0.008769204	0.004	0.003023952	17.33582793
	Sep-11	30		105.2582	6.115074	0.852576	259.2605	0.108119	0.108119	174.6931369	0.01	0.009402	0.010299	0.052451	0.00040449	0.165566	0.000521056	76.50559	0.002866144	0.001965591	0.008746642	0.004	0.002984258	17.10917606
	Oct-11	31		106.3253	6.136516	0.86494	263.0823	0.108776	0.108776	176.9192131	0.01	0.00952	0.010418	0.052742	0.00040937	0.166702	0.000531422	77.38431	0.002875575	0.002007423	0.008909025	0.004	0.003030925	17.2513255
	Nov-11	30		104.6281	6.132606	0.853511	260.0568	0.108672	0.108672	174.2976169	0.01	0.009382	0.010203	0.052799	0.000405561	0.166091	0.00052354	76.14223	0.00289384	0.001989914	0.008893344	0.004	0.002980287	17.02721049
	Dec-11	31		104.6161	6.123808	0.856531	261.1133	0.108703	0.108703	174.6402606	0.01	0.0094	0.010201	0.052817	0.000406641	0.166134	0.000527164	76.20446	0.002893581	0.002008115	0.008970876	0.004	0.002990957	17.0144998

SRK Consulting
Project NorthMet
Number 1UP005.001
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G:\PolyMet Mining\1UP005.01_Northmet_project_2004\Water_Quality_Predictions\Operational_Tailings\Model\[PolyMet Water Balance_1E-2E_v27.7.xls]Summary
Operational Tailings Pond Chemistry Predictions

Year	Month	Days/Month	Active Cell	Alkalinity	Chloride	Fluoride	Hardness	Nitrate + Nitrite, as N	Nitrogen, Ammonia as N	Sulfate	Aluminum	Antimony	Arsenic	Barium	Beryllium	Boron	Cadmium	Calcium	Chromium	Cobalt	Copper	Iron	Lead	Magnesium
Fixed Standards				#N/A	230	2	#N/A	#N/A	#N/A	250	0.125	0.031	0.148	2	0.004	0.5	f(H)	#N/A	f(H)	0.005	f(H)	0.3	f(H)	#N/A
12	Jan-12	31	Combined	104.5105	6.117288	0.858344	260.3393	0.108426	0.108426	174.2476256	0.01	0.009375	0.010225	0.052856	0.000407692	0.166162	0.000521553	75.63224	0.002903662	0.002003761	0.008985472	0.004	0.002968673	17.09633977
	Feb-12	28		104.3204	6.128721	0.858821	259.4106	0.10843	0.10843	173.683296	0.01	0.009344	0.010237	0.053034	0.000408611	0.166478	0.000515334	75.03617	0.002924267	0.001996783	0.00900049	0.004	0.002942805	17.16527255
	Mar-12	31		103.1227	6.094963	0.851144	255.9846	0.107603	0.107603	171.4315717	0.01	0.009222	0.010132	0.052824	0.000405824	0.165462	0.000503604	73.69097	0.002925734	0.001970144	0.008935201	0.004	0.002886466	17.08198484
	Apr-12	30		104.6867	6.183906	0.865354	259.1715	0.109043	0.109043	173.5453571	0.01	0.009334	0.0103	0.053692	0.000412766	0.168043	0.000505716	74.26113	0.002982056	0.001994973	0.009079736	0.004	0.00290762	17.43316373
	May-12	31		103.0247	6.25446	0.852328	254.7149	0.109799	0.109799	170.3081613	0.01	0.009164	0.010075	0.054356	0.000409886	0.168803	0.000488244	72.54207	0.003051113	0.001953807	0.009027759	0.004	0.002823102	17.34031059
	Jun-12	30		103.1476	6.320136	0.854135	254.4744	0.11071	0.11071	170.0234352	0.01	0.009149	0.010071	0.055	0.000412051	0.170259	0.000482232	72.11665	0.003103218	0.001949459	0.009074778	0.004	0.002799356	17.47820153
	Jul-12	31		103.6843	6.407612	0.859609	255.2174	0.111984	0.111984	170.3877485	0.01	0.009168	0.010102	0.055831	0.000415693	0.172321	0.000478104	71.96496	0.003165582	0.001952448	0.00915362	0.004	0.002787325	17.68773517
	Aug-12	31		98.54485	6.122563	0.81448	241.2491	0.106831	0.106831	160.517314	0.01	0.008639	0.00952	0.053456	0.000394605	0.164378	0.000445324	67.57469	0.003047745	0.001839776	0.00868507	0.004	0.002608081	16.92863164
	Sep-12	30		99.99996	6.102452	0.829139	244.2369	0.106547	0.106547	162.7530242	0.01	0.008753	0.009734	0.053375	0.000399615	0.164796	0.000450881	68.22336	0.003034619	0.001868121	0.008776492	0.004	0.002644142	17.19120627
	Oct-12	31		101.0765	6.118041	0.840707	246.4531	0.10677	0.10677	164.4210407	0.01	0.008837	0.009886	0.053588	0.000404189	0.165745	0.000453498	68.62599	0.003045482	0.001888379	0.008865034	0.004	0.002665532	17.42101625
	Nov-12	30		98.88264	6.090031	0.823481	240.8914	0.105961	0.105961	160.5861376	0.01	0.008633	0.009642	0.053366	0.000397853	0.164296	0.000437613	66.7846	0.003051923	0.001841604	0.008732523	0.004	0.002584073	17.16384927
	Dec-12	31		98.71451	6.075523	0.824123	240.1582	0.105589	0.105589	160.1915088	0.01	0.008608	0.009647	0.053295	0.000398027	0.164065	0.000433773	66.35922	0.003052256	0.00183706	0.008730309	0.004	0.002568712	17.19715288
13	Jan-13	31	Combined	97.97082	6.025975	0.816911	238.6109	0.104843	0.104843	159.5150775	0.01	0.008609	0.009723	0.052847	0.000397197	0.16288	0.000436253	66.14103	0.003028179	0.00184122	0.008754056	0.004	0.002546826	16.82160171
	Feb-13	28		98.38857	6.011402	0.819729	239.7985	0.104751	0.104751	160.68832	0.01	0.008705	0.009925	0.052719	0.000400294	0.162912	0.000444038	66.67623	0.003017629	0.001866035	0.008854584	0.004	0.002559123	16.65681129
	Mar-13	31		98.43571	5.993703	0.819363	240.1361	0.104579	0.104579	161.2765767	0.01	0.00877	0.010074	0.052557	0.000402241	0.162709	0.00045	66.97301	0.003007593	0.001884211	0.008934054	0.004	0.002559942	16.42894664
	Apr-13	30		99.81594	6.029009	0.827099	242.9852	0.105405	0.105405	163.168874	0.01	0.008907	0.010304	0.052905	0.000407616	0.164028	0.000459288	67.8569	0.003027526	0.001918596	0.009082863	0.004	0.002582053	16.44194949
	May-13	31		98.9366	6.065429	0.818002	241.1313	0.105966	0.105966	162.0027717	0.01	0.008877	0.010279	0.053185	0.000407007	0.164399	0.000457292	67.44498	0.003056987	0.001912079	0.00911245	0.004	0.002546245	16.15017966
	Jun-13	30		98.18919	6.011771	0.809228	239.1064	0.10514	0.10514	160.6798687	0.01	0.008834	0.01028	0.052726	0.000404642	0.163024	0.000456466	66.97102	0.003033742	0.001905975	0.009089069	0.004	0.002516185	15.85212766
	Jul-13	31		101.8352	6.184776	0.839148	248.1873	0.108343	0.108343	167.1889745	0.01	0.009219	0.010808	0.054245	0.000420826	0.16822	0.000479457	69.71726	0.003116152	0.001993624	0.009480185	0.004	0.002614002	16.21801342
	Aug-13	31		100.2958	6.182732	0.824024	244.5294	0.108217	0.108217	164.6556022	0.01	0.009108	0.010674	0.054198	0.000416821	0.167482	0.000472703	68.73848	0.003127685	0.00196919	0.009426758	0.004	0.002557812	15.8651607
	Sep-13	30		100.2143	6.164111	0.822662	244.4708	0.107989	0.107989	164.863087	0.01	0.009144	0.010771	0.05403	0.000417746	0.167165	0.00047636	68.86531	0.003117849	0.001979785	0.009473637	0.004	0.0025547	15.68152681
	Oct-13	31		100.539	6.160141	0.825028	245.3737	0.108009	0.108009	165.7712573	0.01	0.009219	0.010919	0.053987	0.000420235	0.167325	0.000482308	69.27807	0.003114017	0.001998875	0.009554871	0.004	0.002563505	15.55071654
	Nov-13	30		99.39205	6.152094	0.814671	242.772	0.107782	0.107782	164.1060439	0.01	0.009149	0.010849	0.053882	0.00041759	0.166683	0.000478359	68.62822	0.003117066	0.00198346	0.009523572	0.004	0.0025259	15.26597419
	Dec-13	31		99.62532	6.149667	0.81633	243.4582	0.107811	0.107811	164.8357421	0.01	0.009211	0.010974	0.053852	0.000419677	0.166823	0.000483362	68.96319	0.003114593	0.00199945	0.009593722	0.004	0.00253208	15.14123122
14	Jan-14	31	Combined	99.58272	6.132032	0.815801	243.732	0.107625	0.107625	165.1723963	0.01	0.00925	0.011051	0.053688	0.000420227	0.166527	0.000488241	69.19916	0.00310288	0.002012147	0.009641731	0.004	0.002536499	14.96427782
	Feb-14	28		99.73448	6.114831	0.816872	244.4018	0.107453	0.107453	165.7810823	0.01	0.009302	0.011158	0.053533	0.000421354	0.166329	0.000493733	69.53736	0.003090457	0.002027422	0.009697238	0.004	0.002546162	14.82821695
	Mar-14	31		99.91652	6.097573	0.818321	245.1804	0.107293	0.107293	166.4808666	0.01	0.009359	0.011263	0.053378	0.000422607	0.166145	0.000499752	69.91508	0.003077617	0.002044349	0.009757554	0.004	0.002557442	14.68805829
	Apr-14	30		100.1738	6.078931	0.81945	245.894	0.10712	0.10712	167.0097051	0.01	0.009407	0.011355	0.053225	0.000423557	0.165927	0.000504934	70.23075	0.003065507	0.002059048	0.009807214	0.004	0.002565962	14.57568578
	May-14	31		100.9434	6.176033	0.825136	248.1666	0.108804	0.108804	168.5405196	0.01	0.00951	0.011469	0.054048	0.000428215	0.168219	0.00051112	70.96216	0.00311877	0.002082801	0.009947988	0.004	0.002583703	14.59360617
	Jun-14	30		99.49078	6.113605	0.811073	244.5268	0.107729	0.107729	165.859909	0.01	0.009375	0.011302	0.053503	0.000422183	0.166276	0.000504424	69.93931	0.003092657	0.002055058	0.009834549	0.004	0.002537639	14.30558722
	Jul-14	31		102.0848	6.209894	0.832198	251.0804	0.109632	0.109632	170.4984093	0.01	0.009652	0.011686	0.054358	0.00043296	0.169438	0.0005228	71.96458	0.00313439	0.002121018	0.010109746	0.004	0.002612081	14.51055664
	Aug-14	31		102.5476	6.28228	0.835284	252.5134	0.110885	0.110885	171.4369823	0.01	0.00972	0.011757	0.05497	0.000436057	0.171091	0.000526938	72.43835	0.003175016	0.002136854	0.010209966	0.004	0.002621317	14.49841316
	Sep-14	30		102.2422	6.292683	0.832346	252.0449	0.111067	0.111067	171.1288555	0.01	0.009716	0.011753	0.055043	0.000435731	0.171181	0.000527478	72.37644	0.003182563	0.002137193	0.010227389	0.004	0.002612882	14.37286263
	Oct-14	31		100.1686	6.153248	0.81482	247.0512	0.108695	0.108695	167.7645398	0.01	0.009537	0.011554	0.053825	0.000427045	0.167498	0.000519492	71.02238	0.00311114	0.002100836	0.010045406	0.004	0.002560829	13.97705963
	Nov-14	30		100.5036	6.14787	0.81769	248.1208	0.10871	0.10871	168.6394014	0.01	0.009599	0.01166	0.053775	0.00042879	0.167606	0.000525003	71.44487	0.003104673	0.002117481	0.010106951	0.004	0.002575216	13.9050753
	Dec-14	31		100.5429	6.129595	0.818127	248.3846	0.10846	0.10846	168.9470308	0.01	0.009628	0.011721	0.053609	0.000429202	0.167307	0.000528539	71.6274	0.003092515	0.002126629	0.010136803	0.004	0.002580429	13.79607173
15	Jan-15	31	Combined	100.9407	6.13026	0.823209	248.6083	0.107983	0.107983	169.1970835	0.01	0.009681	0.011881	0.053747	0.000430616	0.167771	0.000525642	71.35345	0.003103617	0.00213068	0.010208879	0.004	0.002573841	13.814416
	Feb-15	28		100.5341	6.120926	0.821119	246.9553	0.107308	0.107308	168.0684961	0.01	0.009651	0.011919	0.053769	0.000429084	0.167612	0.00051786	70.55085	0.003112228	0.002115753	0.010207173	0.004	0.002544483	13.75760308
	Mar-15	31		100.2411	6.113099	0.820206	245.5577	0.106648	0.106648	167.135622	0.01	0.009634	0.011967	0.053814	0.000427953	0.167549	0.000510654	69.80782	0.003121598	0.002103595	0.010217176	0.004	0.002518054	13.71109792
	Apr-15	30		98.99606	6.109968	0.809953	241.7498	0.106005	0.106005	164.2912978	0.01	0.009503	0.01183	0.053875	0.000423058	0.16703	0.000496009	68.32465	0.003140134	0.002065023	0.010133962	0.004	0.002458238	13.60196296
	May-15	31		99.3306	6.137516	0.813832	241.8933	0.106013	0.106013	164.3785266	0.01	0.009541	0.011942	0.054225	0.000424448	0.167931	0.000492282	68.04121	0.003166688	0.00206586	0.010200653	0.004	0.002448225	13.64343576
	Jun-15	30		99.87708	6.194572	0.818849	242.5295	0.106534	0.106534	164.7118596	0.01	0.009591	0.012057	0.054825	0.00042677	0.169473	0.000488977	67.88973	0.00320994	0.002069169	0.010287314	0.004	0.002440967	13.73372879
	Jul-15	31		98.24161	6.124334	0.804686	237.6338	0.104882	0.104882	161.1041712	0.01	0.009412	0.011865	0.054304	0.00041922	0.167402	0.000473543	66.14436	0.003189875	0.002022877	0.010133573	0.004	0.002374155	13.54876267
	Aug-15	31		100.3505	6.220202	0.823542	242.1197	0.106154	0.106154	164.2591754	0.01	0.009627	0.01222	0.055268	0.000427781	0.170513	0.000480329	67.1336	0.003247105	0.002063876	0.010370807	0.004	0.002413645	13.7967032
	Sep-15	30		95.94128	6.071292	0.786543	230.9532	0.103102	0.103102	156.3059487	0.01	0.009185	0.011637	0.053976	0.000410046	0.165511	0.000450634	63.67743	0.003189983	0.001959126	0.009966079	0.004	0.002278368	13.29907696
	Oct-15	31		97.39129	6.075378	0.800442	233.8363	0.102895	0.102895	158.5021857	0.01	0.009344	0.011948	0.054143	0.00041523	0.16653	0.000456145	64.27406	0.003194438	0.001990077	0.010119909	0.004	0.002308178	13.41189581
	Nov-15	30		96.58102	6.036936	0.794878	231.462	0.101881	0.101881	156.9112643	0.01	0.009275	0.011908	0.053873	0.000412006	0.165544	0.000448435	63.38362	0.003183402	0.001969553	0.010066696	0.004	0.002276088	13.29817911
	Dec-15	31		96.8474	6.034359	0.798434	231.556	0.101484	0.101484	157.0582634	0.01	0.009309	0.012018	0.053936	0.000412864	0.165792	0.000446476	63.18768	0.003189076	0.001971961	0.0101129	0.004	0.002271508	13.30445318

SRK Consulting
Project NorthMet
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G:\PolyMet Mining\1UP005.01_Northmet_project_2004\Water_Quality_Predictions\Operational_Tailings\Model\[PolyMet Water Balance_1E-2E_v27.7.xls]Summary
Operational Tailings Pond Chemistry Predictions

Year	Month	Days/Month	Active Cell	Alkalinity	Chloride	Fluoride	Hardness	Nitrate + Nitrite, as N	Nitrogen, Ammonia as N	Sulfate	Aluminum	Antimony	Arsenic	Barium	Beryllium	Boron	Cadmium	Calcium	Chromium	Cobalt	Copper	Iron	Lead	Magnesium
Fixed Standards				#N/A	230	2	#N/A	#N/A	#N/A	250	0.125	0.031	0.148	2	0.004	0.5	f(H)	#N/A	f(H)	0.005	f(H)	0.3	f(H)	#N/A
	Jan-16	31		95.67725	6.03705	0.789179	228.289	0.101119	0.101119	154.7588005	0.01	0.009193	0.011875	0.053979	0.000408846	0.165347	0.000435175	62.05265	0.003203805	0.001939232	0.010030617	0.004	0.002223751	13.19729661
	Feb-16	28		95.50749	6.030708	0.788537	227.3646	0.100705	0.100705	154.1933269	0.01	0.00918	0.011913	0.053974	0.000408231	0.16525	0.0004309	61.62047	0.0032078	0.001931123	0.010030026	0.004	0.002207207	13.16737862
	Mar-16	31		95.737	6.045751	0.7914	227.3668	0.100634	0.100634	154.27661	0.01	0.009206	0.011997	0.054168	0.000409259	0.165776	0.000428403	61.42956	0.003223274	0.001931326	0.010071244	0.004	0.002199839	13.18871245
	Apr-16	30		96.54198	6.103766	0.798097	228.6232	0.101309	0.101309	155.0861427	0.01	0.009275	0.012127	0.054744	0.000412443	0.16738	0.000427731	61.55973	0.003262889	0.001940496	0.010162977	0.004	0.002202355	13.30648576
	May-16	31		96.4154	6.135617	0.797426	227.8527	0.101522	0.101522	154.5254949	0.01	0.009259	0.012127	0.055061	0.00041234	0.168014	0.000422665	61.15013	0.003289549	0.00193118	0.010174125	0.004	0.002183611	13.31599426
16	Jun-16	30	Combined	97.03358	6.193719	0.802686	228.7729	0.102205	0.102205	155.1105141	0.01	0.009312	0.012228	0.055625	0.000415006	0.169525	0.000421319	61.20507	0.003329074	0.001937083	0.010252355	0.004	0.002182802	13.41516311
	Jul-16	31		97.65193	6.270855	0.807918	229.7531	0.103194	0.103194	155.7078623	0.01	0.009365	0.012314	0.056347	0.000417949	0.171399	0.000419599	61.27152	0.003379735	0.001942412	0.010337318	0.004	0.002180996	13.52949267
	Aug-16	31		95.44873	6.236769	0.788553	224.1551	0.102364	0.102364	151.6052491	0.01	0.00913	0.011969	0.056023	0.000409392	0.169604	0.000403639	59.5502	0.003374694	0.001886505	0.010134038	0.004	0.002108901	13.32213441
	Sep-16	30		95.32555	6.209515	0.788233	223.3808	0.101683	0.101683	151.1624502	0.01	0.009121	0.012009	0.055837	0.000408575	0.169104	0.000400775	59.1998	0.003364824	0.001881113	0.010125898	0.004	0.002097362	13.28260388
	Oct-16	31		95.73559	6.221013	0.792603	223.9208	0.101629	0.101629	151.6390408	0.01	0.009166	0.012117	0.055995	0.000410226	0.16964	0.000400333	59.20508	0.003375724	0.001886995	0.010179473	0.004	0.002098524	13.31698294
	Nov-16	30		95.08318	6.187326	0.787874	222.0484	0.100851	0.100851	150.4205489	0.01	0.009107	0.012072	0.055726	0.000407623	0.168737	0.000395012	58.57477	0.003362927	0.00187087	0.010125819	0.004	0.002075093	13.22476767
	Dec-16	31		95.1057	6.170188	0.789019	221.7039	0.100347	0.100347	150.301994	0.01	0.009116	0.01213	0.055625	0.000407568	0.168514	0.000393222	58.35753	0.003357726	0.001869527	0.010136344	0.004	0.0020687	13.20281629
	Jan-17	31		94.1357	6.165149	0.780944	219.2133	0.100056	0.100056	148.660414	0.01	0.009026	0.01202	0.05555	0.00040609	0.167962	0.000385707	57.56223	0.003362491	0.00184741	0.010070866	0.004	0.002034213	13.12461177
	Feb-17	28		94.27098	6.171539	0.782424	219.2014	0.099974	0.099974	148.8111934	0.01	0.009047	0.012102	0.055619	0.000408384	0.168226	0.000384421	57.45318	0.003370128	0.001849783	0.010104013	0.004	0.00202911	13.14658178
	Mar-17	31		94.35075	6.181951	0.783516	219.0698	0.099948	0.099948	148.8795067	0.01	0.009063	0.012164	0.05572	0.000410622	0.168537	0.000382674	57.30299	0.003380508	0.001850945	0.010134517	0.004	0.002022053	13.16793011
	Apr-17	30		95.02854	6.225338	0.788937	220.1653	0.100465	0.100465	149.6999556	0.01	0.009124	0.012289	0.05612	0.000414942	0.169753	0.000382953	57.46547	0.00340928	0.001861646	0.010215156	0.004	0.002025633	13.27435292
	May-17	31		95.05557	6.251543	0.789183	219.8924	0.1007	0.1007	149.5878483	0.01	0.009127	0.012318	0.056345	0.000416926	0.170306	0.00038041	57.27732	0.003429086	0.001859725	0.010238734	0.004	0.002015348	13.30598607
17	Jun-17	30	Combined	95.86489	6.3163	0.795789	221.3849	0.101569	0.101569	150.6700596	0.01	0.009203	0.012453	0.056927	0.000421991	0.172005	0.000381237	57.55347	0.003469524	0.001873128	0.010339325	0.004	0.002022251	13.43829871
	Jul-17	31		93.57884	6.186869	0.775141	215.5055	0.099369	0.099369	146.509512	0.01	0.008957	0.012131	0.055761	0.000412707	0.168253	0.000368336	55.86986	0.003405516	0.001821188	0.010086552	0.004	0.001957676	13.16580442
	Aug-17	31		94.61259	6.237977	0.784006	217.4876	0.100017	0.100017	148.0034526	0.01	0.009059	0.012317	0.056241	0.000418415	0.169824	0.00037091	56.28715	0.003437004	0.001840821	0.01020465	0.004	0.001972193	13.30337889
	Sep-17	30		94.29034	6.24325	0.781407	216.516	0.099949	0.099949	147.4033011	0.01	0.00903	0.012297	0.056273	0.000418652	0.169788	0.000367463	55.94417	0.003444305	0.00183262	0.010190523	0.004	0.00195676	13.28427884
	Oct-17	31		94.35888	6.249606	0.782391	216.4019	0.099889	0.099889	147.4585516	0.01	0.009042	0.012347	0.056332	0.000420371	0.169995	0.000366206	55.83065	0.00345116	0.001833553	0.010213119	0.004	0.001951602	13.29777054
	Nov-17	30		93.94018	6.223822	0.77928	215.1887	0.099325	0.099325	146.7543646	0.01	0.009007	0.012333	0.056102	0.000419833	0.169323	0.000363152	55.44017	0.003440039	0.001824988	0.010182138	0.004	0.001936772	13.24233302
	Dec-17	31		94.07234	6.215255	0.780955	215.1948	0.099029	0.099029	146.9301949	0.01	0.009027	0.012405	0.056041	0.000421532	0.169281	0.000362688	55.36751	0.00343774	0.00182815	0.010205729	0.004	0.001934725	13.24799541
	Jan-18	31		93.21285	6.205676	0.77449	214.0077	0.098728	0.098728	145.8050836	0.01	0.00897	0.012305	0.055954	0.000421411	0.168797	0.000359617	54.85932	0.003436293	0.001820901	0.010192898	0.004	0.001922659	13.17372583
	Feb-18	28		93.46962	6.204365	0.777665	215.1151	0.098542	0.098542	146.4262397	0.01	0.009024	0.01241	0.055985	0.000425215	0.169072	0.000362079	54.97692	0.003437029	0.001836828	0.010275862	0.004	0.001936494	13.19944262
	Mar-18	31		93.74771	6.208037	0.781157	216.3285	0.09843	0.09843	147.1063333	0.01	0.009082	0.01251	0.056062	0.000429319	0.169461	0.000364676	55.10929	0.003440719	0.001853911	0.010365926	0.004	0.001951302	13.23172474
	Apr-18	30		93.94474	6.206523	0.783474	217.2435	0.098259	0.098259	147.5406995	0.01	0.009123	0.012587	0.056091	0.000432641	0.169676	0.000366532	55.16708	0.003441903	0.001867383	0.010436995	0.004	0.001962286	13.25765447
	May-18	31		94.78638	6.247341	0.791538	219.7102	0.098747	0.098747	149.0754788	0.01	0.009233	0.012757	0.056501	0.000439114	0.171068	0.000371202	55.62741	0.003466132	0.00189479	0.010584673	0.004	0.00198812	13.36988255
18	Jun-18	30	Combined	94.32449	6.217852	0.787587	218.9497	0.098159	0.098159	148.2812617	0.01	0.009196	0.012711	0.05626	0.00043903	0.17033	0.000369529	55.26068	0.003452752	0.001891429	0.010571169	0.004	0.001981124	13.3220043
	Jul-18	31		94.9694	6.290234	0.793281	220.9363	0.099182	0.099182	149.3720343	0.01	0.009273	0.012804	0.056915	0.000444696	0.172165	0.000372094	55.6055	0.003495934	0.001910707	0.010697583	0.004	0.001997924	13.44617234
	Aug-18	31		95.33846	6.341922	0.796756	222.2737	0.09988	0.09988	150.0478824	0.01	0.009325	0.012864	0.057382	0.000449003	0.173455	0.000373674	55.79536	0.003527361	0.001924512	0.010791793	0.004	0.002009201	13.5277853
	Sep-18	30		94.33643	6.28909	0.788727	220.3314	0.098935	0.098935	148.5406975	0.01	0.009241	0.012747	0.056913	0.000446494	0.171988	0.000370028	55.17244	0.003500238	0.001910362	0.01072263	0.004	0.001991637	13.40410259
	Oct-18	31		93.92383	6.215475	0.78633	219.6465	0.097611	0.097611	148.0343712	0.01	0.009223	0.012763	0.056303	0.000446265	0.170477	0.000370133	54.86932	0.003459739	0.001911685	0.010707956	0.004	0.00199105	13.31174148
	Nov-18	30		94.51063	6.198321	0.792528	221.2877	0.097165	0.097165	149.1506739	0.01	0.009307	0.012931	0.056209	0.000450698	0.170604	0.000374504	55.16043	0.003450075	0.001934021	0.01080575	0.004	0.00201273	13.34973568
	Dec-18	31		93.75308	6.188197	0.786514	219.9757	0.096922	0.096922	148.0657303	0.01	0.009246	0.012824	0.056102	0.000449447	0.170076	0.000371375	54.7192	0.003446877	0.001923357	0.010768722	0.004	0.001998978	13.27899561
	Jan-19	31		93.65551	6.172259	0.785307	219.2806	0.096578	0.096578	148.0991121	0.01	0.009246	0.012895	0.055902	0.000454059	0.169693	0.000370022	54.53708	0.003439802	0.001924685	0.010773258	0.004	0.001987183	13.29342901
	Feb-19	28		93.81704	6.172747	0.786234	219.2337	0.096499	0.096499	148.5075292	0.01	0.00927	0.013001	0.055859	0.000459302	0.169772	0.000369794	54.51842	0.003441652	0.00193078	0.010805166	0.004	0.001981923	13.33693393
	Mar-19	31		93.72103	6.166013	0.784984	218.5917	0.096313	0.096313	148.5235551	0.01	0.009268	0.013058	0.05574	0.000463638	0.169563	0.000368321	54.35044	0.003439803	0.001931428	0.010810488	0.004	0.001970228	13.3594344
	Apr-19	30		94.74319	6.225948	0.792884	220.5135	0.097176	0.097176	150.2344126	0.01	0.009373	0.013268	0.056233	0.000473012	0.171235	0.000371215	54.8137	0.003475199	0.001954404	0.010937999	0.004	0.001981915	13.53459727
	May-19	31		94.19988	6.288307	0.78693	219.0753	0.098181	0.098181	149.4331796	0.01	0.009313	0.013165	0.056668	0.000475314	0.172053	0.000365844	54.45263	0.003514212	0.001939755	0.010909232	0.004	0.00195483	13.57077374
19	Jun-19	30	Combined	95.57407	6.414755	0.79765	221.9825	0.100122	0.100122	151.7250192	0.01	0.009451	0.013388	0.057731	0.000486646	0.175187	0.000369428	55.17015	0.003587378	0.001968213	0.011089526	0.004	0.001972489	13.82720563
	Jul-19	31		94.00017	6.371305	0.782575	217.9101	0.099463	0.099463	149.0498496	0.01	0.009277	0.013139	0.057245	0.000482473	0.173375	0.000360208	54.13322	0.003567193	0.001931001	0.010916211	0.004	0.001923729	13.69157273
	Aug-19	31		93.91174	6.331788	0.78105	217.1191	0.098759	0.098759	148.8689023	0.01	0.009266	0.013193	0.056864	0.000485362	0.172513	0.000359061	53.91204	0.003546706	0.001930641	0.010899648	0.004	0.001913033	13.68768369
	Sep-19	30		94.13502	6.318651	0.782727	217.1978	0.09845	0.09845	149.3366439	0.01	0.009295	0.013305	0.056715	0.000490115	0.172363	0.000359575	53.92419	0.00354026	0.001938366	0.01093047	0.004	0.001911275	13.72337108
	Oct-19	31		92.71168	6.20851	0.770411	213.4824	0.096656	0.096656	147.1275254	0.01	0.009156	0.013161	0.055689	0.000486132	0.169439	0.000353362	52.98947	0.003479964	0.001910739	0.01076901	0.004	0.001874649	13.5324968
	Nov-19	30		92.90069	6.210963	0.771765	213.5703	0.096613	0.096613	147.5423555	0.01	0.009181	0.013251	0.055671	0.000490563	0.169568	0.000353507	53.00768	0.003482409	0.001916904	0.010800069	0.004	0.001872045	13.57464605
	Dec-19	31		92.56577	6.185171	0.768739	212.4763	0.096139	0.096139	147.1222563	0.01	0.009153	0.013255	0.055396	0.000492226	0.168868	0.000351534	52.73254	0.003469139	0.001911926	0.010771473	0.004	0.001858597	13.54569079
	Jan-20	31		92.07396	6.135853	0.764219	210.6872	0.095289	0.095289	146.8796814	0.01	0.009125	0.013323	0.054865	0.000499447	0.167628	0.000349022	52.41936	0.003443042	0.001910037	0.01076834	0.004	0.001837178	13.53037789
	Feb-20	28		91.9128	6.097221	0.762528	209.6933	0.094598	0.094598	147.097552	0.01	0.00913	0.013442	0.054449	0.000507196	0.166783	0.000348082	52.28814	0.003422509	0.001914533	0.010794471	0.004	0.001824373	13.54547354
	Mar-20	31		92.39584	6.103063	0.766228	210.1441	0.094597	0.094597	148.3917806	0.01	0.009199	0.013653	0.054423	0.000519142	0.167137	0.000349533	52.52452	0.003427062	0.00193307	0.010900432	0.004	0.001823692	13.66164536
	Apr-20	30		92.63718	6.11086	0.766508	209.9159	0.0947	0.0947	148.9182541	0.01	0.009221	0.013766	0.054419	0.000528159	0.167322	0.000348721	52.55312	0.003434371	0.001940758	0.010956504	0.004	0.001813338	13.7674415
	May-20	31		92.81562	6.186383	0.766929	209.963	0.095893	0.095893	149.599512	0.01	0.009246	0.013834	0.054969	0.000537787	0.168806	0.000347204	52.68223	0.003479613	0.001947017	0.011038831	0.004	0.001802459	13.90483583
20	Jun-20	30	Combined	91.74475	6.149824	0.75659	207.0294	0.095345	0.095345	148.0692697	0.01	0.009139	0.013716	0.054552	0.000538855	0.167444	0.000341109	52.03577	0.00346196	0.00192583	0.010951201	0.004	0.001767249	13.83556761
	Jul-20	31		92.83857	6.195135	0.76512	208.8726	0.095972	0.095972	150.220762	0.01	0.009262	0.013998	0.05489	0.000553104	0.168873	0.000344739	52.60072	0.003488826	0.001955549	0.011118873	0.004	0.001778725	14.03871643
	Aug-20	31		93.53583	6.267371	0.770232	210.0403	0.097071	0.097071	151.7419276	0.01	0.009344	0.014173	0.055439	0.000564978	0.170597	0.000346053	53.0004	0.00353137	0.001974501	0.011252124	0.004	0.001781089	14.21827657
	Sep-20	30		92.6235	6.245537	0.761898	207.6557	0.096738	0.096738	150.5800487	0.01	0.00926	0.014083	0.055156	0.000566433	0.169641	0.000341109	52.4938	0.003521116	0.001957832	0.011189162	0.004	0.001752491	14.1603537
	Oct-20	31		92.75255	6.238135	0.762686	207.4275	0.096542	0.096542	151.1862027	0.01	0.009289	0.014209	0.055019	0.000574684	0.169543	0.000341219	52.53516	0.003517989	0.001967026	0.011244031	0.004	0.001746535	14.21878099
	Nov-20	30		90.64118	6.142768	0.744517	202.4224	0.09507	0.09507	148.018914	0.01	0.009082	0.01392	0.054087	0.000568052	0.166526	0.000331848	51.35423	0.003466114	0.001924053	0.011032507	0.004	0.001696067	13.97614947
	Dec-20	31		91.00656	6.105637	0.74769	202.6192	0.09436	0.09436	149.0470101	0.01	0.00914	0.014125	0.053713	0.000577468	0.166009	0.000333828	51.49906	0.003445402	0.001940877	0.011106601	0.004	0.001697548	14.03257204

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Operational Tailings Pond Chemistry Predictions

Year	Month	Days/Month	Active Cell	Manganese	Mercury	Nickel	Potassium	Selenium	Silver	Sodium	Thallium	Zinc	Acidity	Hardness
Fixed Standards				0.05	#N/A	f(H)	#N/A	0.005	0.001	#N/A	0.00028	f(H)		
1	Jan-01	31	1E	0.005312893	3.33254E-07	0.000149	0.208027046	3.6E-05	1.5E-05	0.514499	2.69E-05	0.000635	0.017988	9.489987
	Feb-01	28		0.010129039	6.87263E-07	0.000457	1.105801681	0.000147	2.83E-05	4.515716	5E-05	0.001433	0.055274	21.24701
	Mar-01	31		0.032325032	1.35223E-06	0.000957	1.760765101	0.000242	6.05E-05	7.232725	7.17E-05	0.002239	0.066308	34.87454
	Apr-01	30		0.051615228	2.01698E-06	0.001467	2.158948058	0.000306	8.95E-05	8.676682	9.35E-05	0.002898	0.066308	45.35466
	May-01	31		0.07575041	2.69887E-06	0.001873	2.405889654	0.000356	0.000123	9.358559	0.000114	0.00357	0.066308	57.91099
	Jun-01	30		0.110768359	3.54919E-06	0.002441	2.629374155	0.000409	0.000168	9.95379	0.000129	0.004183	0.066308	70.02501
	Jul-01	31		0.145210528	4.35202E-06	0.003002	2.831233047	0.00046	0.000212	10.45024	0.000145	0.004786	0.066308	80.96066
	Aug-01	31		0.188959027	5.23035E-06	0.00351	2.949363189	0.000503	0.000263	10.61927	0.000152	0.005317	0.066308	94.15932
	Sep-01	30		0.20581146	5.68931E-06	0.003766	3.040426058	0.000528	0.000286	10.70919	0.000165	0.005733	0.066308	102.0895
	Oct-01	31		0.215965985	6.32498E-06	0.004294	3.208357028	0.000558	0.000304	11.08603	0.000183	0.006153	0.066308	108.9589
	Nov-01	30		0.221977193	6.87319E-06	0.004767	3.328725658	0.00058	0.000315	11.30809	0.000198	0.006454	0.066308	113.8911
	Dec-01	31		0.227871837	7.26247E-06	0.005043	3.458409223	0.000604	0.000329	11.54982	0.000217	0.006877	0.066308	120.2259
2	Jan-02	31	1E	0.238408057	7.82417E-06	0.005524	3.591480829	0.000633	0.000347	11.83423	0.000233	0.007328	0.066308	126.9024
	Feb-02	28		0.256155472	8.31579E-06	0.005914	3.691710718	0.000662	0.000371	12.02127	0.000245	0.00772	0.066308	132.4819
	Mar-02	31		0.273375035	8.80923E-06	0.006323	3.798177861	0.000691	0.000395	12.23551	0.000257	0.008132	0.066308	138.4121
	Apr-02	30		0.31145812	9.2626E-06	0.006435	3.805987899	0.000715	0.000436	12.15158	0.000252	0.008465	0.066308	148.8178
	May-02	31		0.324527411	9.61377E-06	0.006688	3.880680184	0.000737	0.000455	12.27413	0.000263	0.008812	0.066308	154.3213
	Jun-02	30		0.335793995	1.00919E-05	0.007042	4.002646872	0.000762	0.000472	12.5437	0.000276	0.009215	0.066308	161.1991
	Jul-02	31		0.352823078	1.04197E-05	0.007187	4.012161884	0.000773	0.000491	12.48786	0.000275	0.009396	0.066308	167.3733
	Aug-02	31		0.364122356	1.07719E-05	0.007434	4.081191452	0.000791	0.000506	12.61464	0.000283	0.009677	0.066308	172.674
	Sep-02	30		0.366149276	1.11606E-05	0.007755	4.209173244	0.000812	0.000515	12.90321	0.0003	0.010063	0.066308	178.1511
	Oct-02	31		0.368077008	1.16532E-05	0.00819	4.358894352	0.000836	0.000523	13.26432	0.000318	0.010457	0.066308	184.066
	Nov-02	30		0.373053221	1.20325E-05	0.008531	4.456591279	0.000854	0.000533	13.4798	0.00033	0.010736	0.066308	187.9267
	Dec-02	31		0.378357284	1.23226E-05	0.00878	4.545691407	0.000873	0.000544	13.67128	0.000342	0.011044	0.066308	191.9383
3	Jan-03	31	1E	0.384572947	1.26131E-05	0.009058	4.619019253	0.000892	0.000555	13.83975	0.000353	0.011375	0.066308	195.8399
	Feb-03	28		0.390572765	1.27975E-05	0.009233	4.646847983	0.000903	0.000565	13.87187	0.000359	0.011561	0.066308	197.5759
	Mar-03	31		0.394536425	1.29082E-05	0.009375	4.650045263	0.00091	0.000571	13.8416	0.000362	0.011689	0.066308	198.3619
	Apr-03	30		0.420450189	1.28186E-05	0.009007	4.525162778	0.000904	0.000593	13.48221	0.000341	0.011624	0.066308	203.3887
	May-03	31		0.420397187	1.30156E-05	0.009189	4.607327344	0.00092	0.000598	13.66985	0.000355	0.011965	0.066308	207.2642
	Jun-03	30		0.424584096	1.32972E-05	0.009394	4.684891403	0.000935	0.000606	13.8521	0.000365	0.012259	0.066308	212.196
	Jul-03	31		0.427396457	1.35335E-05	0.009568	4.746274122	0.000946	0.000613	13.99263	0.000373	0.012506	0.066308	216.5293
	Aug-03	31		0.43021369	1.3874E-05	0.009868	4.859032617	0.000967	0.000621	14.27732	0.000387	0.012875	0.066308	221.5863
	Sep-03	30		0.431431864	1.41424E-05	0.010097	4.95669466	0.000984	0.000628	14.51387	0.000401	0.013217	0.066308	226.0175
	Oct-03	31		0.434597277	1.42966E-05	0.010257	4.980966694	0.000991	0.000633	14.56035	0.000405	0.013326	0.066308	227.3242
	Nov-03	30		0.440667454	1.45337E-05	0.010482	5.048705328	0.001008	0.000643	14.72896	0.000414	0.013562	0.066308	229.9434
	Dec-03	31		0.4430163	1.47317E-05	0.010662	5.127130409	0.001024	0.00065	14.92169	0.000426	0.013863	0.066308	233.4702
4	Jan-04	31	1E	0.445017064	1.48693E-05	0.010827	5.165954771	0.001035	0.000656	15.02572	0.000433	0.014101	0.066308	235.6092
	Feb-04	28		0.449258339	1.50006E-05	0.010959	5.202273279	0.001046	0.000664	15.11138	0.000439	0.014319	0.066308	237.4785
	Mar-04	31		0.45280218	1.5119E-05	0.011101	5.239777255	0.001058	0.000671	15.20798	0.000446	0.014554	0.066308	239.5245
	Apr-04	30		0.463729426	1.51655E-05	0.011052	5.241357145	0.001066	0.000683	15.21528	0.000444	0.014722	0.066308	243.1007
	May-04	31		0.469307267	1.54577E-05	0.011295	5.378816704	0.001095	0.000697	15.58468	0.000463	0.015289	0.066308	249.9406
	Jun-04	30		0.475064912	1.55491E-05	0.01135	5.398260571	0.001103	0.000704	15.63119	0.000465	0.015455	0.066308	252.4125
	Jul-04	31		0.485778658	1.56976E-05	0.011465	5.446426678	0.001119	0.000718	15.76896	0.00047	0.015723	0.066308	255.7645
	Aug-04	31		0.495730434	1.55492E-05	0.011278	5.353786934	0.00111	0.000725	15.52425	0.000456	0.015555	0.066308	254.782
	Sep-04	30		0.494645716	1.5367E-05	0.011133	5.287446624	0.001102	0.000722	15.3258	0.000451	0.015469	0.066308	252.8317
	Oct-04	31		0.493517841	1.55242E-05	0.011322	5.347482134	0.001113	0.000725	15.47541	0.000461	0.015717	0.066308	255.0065
	Nov-04	30		0.493265448	1.57497E-05	0.011556	5.428779146	0.001127	0.000729	15.6819	0.000472	0.016012	0.066308	258.1189
	Dec-04	31		0.490065843	1.58226E-05	0.011655	5.475837274	0.001135	0.000729	15.79342	0.000482	0.016248	0.066308	260.1223

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Operational Tailings Pond Chemistry Predictions

Year	Month	Days/Month	Active Cell	Manganese	Mercury	Nickel	Potassium	Selenium	Silver	Sodium	Thallium	Zinc	Acidity	Hardness
Fixed Standards				0.05	#N/A	f(H)	#N/A	0.005	0.001	#N/A	0.00028	f(H)		
5	Jan-05	31	1E	0.489167906	1.59264E-05	0.011965	5.59321688	0.00116	0.00074	17.13285	0.000504	0.01914	0.066308	262.291
	Feb-05	28		0.488078021	1.59486E-05	0.012149	5.654759903	0.001175	0.000747	18.05687	0.000517	0.021243	0.066308	262.4206
	Mar-05	31		0.488508704	1.60132E-05	0.012377	5.741100657	0.001195	0.000756	19.04606	0.000534	0.023414	0.066308	264.0402
	Apr-05	30		0.49001714	1.56127E-05	0.012017	5.602282551	0.001174	0.000753	19.11334	0.000518	0.024153	0.066308	260.4083
	May-05	31		0.490790695	1.57933E-05	0.012317	5.746694881	0.001204	0.000766	20.19717	0.000542	0.026377	0.066308	264.8334
	Jun-05	30		0.491372335	1.55605E-05	0.012089	5.658770378	0.001189	0.000763	20.27123	0.000531	0.026932	0.066308	263.257
	Jul-05	31		0.491516812	1.55435E-05	0.012122	5.713636514	0.001201	0.000769	20.94989	0.000542	0.028501	0.066308	266.1152
	Aug-05	31		0.489655316	1.57528E-05	0.012422	5.846131448	0.001225	0.000777	21.86413	0.000563	0.030334	0.066308	270.4923
	Sep-05	30		0.486179665	1.59226E-05	0.012693	5.947329438	0.001242	0.000781	22.59279	0.000579	0.031815	0.066308	273.1074
	Oct-05	31		0.483026412	1.58685E-05	0.012736	5.9358274	0.00124	0.000779	22.82669	0.00058	0.032465	0.066308	271.748
	Nov-05	30		0.482818277	1.60554E-05	0.013017	6.045040725	0.001261	0.000787	23.56792	0.000597	0.03394	0.066308	274.6109
	Dec-05	31		0.476821185	1.61113E-05	0.013189	6.130519606	0.001275	0.000789	24.27225	0.000614	0.035457	0.066308	276.7497
6	Jan-06	31	1E	0.473814466	1.6177E-05	0.013365	6.185761129	0.001285	0.000791	24.77721	0.000625	0.036542	0.066308	277.5205
	Feb-06	28		0.473645036	1.626E-05	0.013516	6.243972862	0.001297	0.000795	25.25228	0.000635	0.037561	0.066308	278.6147
	Mar-06	31		0.472776802	1.63025E-05	0.013657	6.293367888	0.001308	0.000799	25.72051	0.000645	0.038586	0.066308	279.4222
	Apr-06	30		0.477981258	1.61171E-05	0.013417	6.227919137	0.001299	0.000802	25.64715	0.000635	0.038689	0.066308	279.1341
	May-06	31		0.476233437	1.59765E-05	0.013345	6.191111546	0.001294	0.0008	25.70637	0.000633	0.039021	0.066308	277.3334
	Jun-06	30		0.475477953	1.59714E-05	0.013382	6.22266647	0.001301	0.000802	26.06686	0.000639	0.039864	0.066308	278.5488
	Jul-06	31		0.475115876	1.58074E-05	0.013207	6.186805259	0.001295	0.000801	26.12895	0.000636	0.040216	0.066308	278.6934
	Aug-06	31		0.476749283	1.57621E-05	0.013175	6.174361539	0.001294	0.000803	26.22823	0.000634	0.040532	0.066308	278.7817
	Sep-06	30		0.474580293	1.57944E-05	0.013279	6.202845465	0.001299	0.000803	26.51052	0.00064	0.041165	0.066308	278.8814
	Oct-06	31		0.471480357	1.57177E-05	0.013257	6.165475279	0.00129	0.000799	26.4578	0.000636	0.041185	0.066308	276.9523
	Nov-06	30		0.466799733	1.58212E-05	0.013465	6.234168568	0.001301	0.000799	26.92843	0.000648	0.042147	0.066308	277.9322
	Dec-06	31		0.459220398	1.58762E-05	0.013624	6.315354615	0.001313	0.000798	27.52231	0.000664	0.043406	0.066308	279.9573
7	Jan-07	31	1E	0.454499338	1.5968E-05	0.013811	6.379101896	0.001323	0.000798	27.97197	0.000676	0.044327	0.066308	281.1937
	Feb-07	28		0.452151285	1.60291E-05	0.013931	6.421380806	0.00133	0.000799	28.2839	0.000683	0.044997	0.066308	281.6701
	Mar-07	31		0.451410341	1.61188E-05	0.014096	6.490153916	0.001343	0.000804	28.75666	0.000695	0.045959	0.066308	283.4986
	Apr-07	30		0.463793036	1.58106E-05	0.01356	6.328011988	0.001319	0.000806	28.02315	0.000666	0.04472	0.066308	282.5841
	May-07	31		0.46111967	1.58767E-05	0.013712	6.432155449	0.001339	0.000812	28.73975	0.000685	0.046212	0.066308	285.9007
	Jun-07	30		0.459889275	1.60029E-05	0.013888	6.523094841	0.001355	0.000817	29.30783	0.000699	0.047347	0.066308	289.0043
	Jul-07	31		0.463085135	1.60388E-05	0.013942	6.558100447	0.001364	0.000823	29.58124	0.000704	0.047922	0.066308	290.5842
	Aug-07	31		0.471056134	1.57982E-05	0.013613	6.410868767	0.001341	0.000821	28.87013	0.00068	0.046653	0.066308	286.8936
	Sep-07	30		0.469208823	1.58296E-05	0.013719	6.479163037	0.001355	0.000825	29.35981	0.000693	0.047694	0.066308	288.5364
	Oct-07	31		0.46676449	1.58356E-05	0.013797	6.483677419	0.001355	0.000824	29.44893	0.000695	0.047908	0.066308	287.6314
	Nov-07	30		0.463098935	1.58139E-05	0.013874	6.477678909	0.001353	0.00082	29.48674	0.000696	0.048042	0.066308	285.4364
	Dec-07	31		0.457675281	1.57893E-05	0.01394	6.520457308	0.00136	0.000819	29.85159	0.000707	0.04886	0.066308	286.0105
8	Jan-08	31	1E	0.456209666	1.59097E-05	0.014135	6.599844551	0.001375	0.000823	30.33882	0.000719	0.04981	0.066308	288.1579
	Feb-08	28		0.455254794	1.59664E-05	0.014246	6.633378723	0.001381	0.000825	30.54857	0.000725	0.050312	0.066308	288.3912
	Mar-08	31		0.456149796	1.60438E-05	0.014402	6.692774056	0.001394	0.000831	30.91818	0.000735	0.051152	0.066308	289.9867
	Apr-08	30		0.469936438	1.5745E-05	0.013858	6.515736913	0.001367	0.000832	29.98683	0.000703	0.04956	0.066308	288.7038
	May-08	31		0.468827054	1.58094E-05	0.014028	6.588587328	0.001382	0.000838	30.45087	0.000717	0.050646	0.066308	290.4695
	Jun-08	30		0.469853341	1.5626E-05	0.0138	6.486050127	0.001363	0.000832	29.92764	0.000702	0.049879	0.066308	288.0604
	Jul-08	31		0.468648914	1.57507E-05	0.014007	6.624421779	0.001391	0.000842	30.77818	0.000726	0.051696	0.066308	293.1461
	Aug-08	31		0.471839162	1.59034E-05	0.014199	6.697689487	0.001406	0.00085	31.1518	0.000736	0.052548	0.066308	296.1846
	Sep-08	30		0.474876031	1.60601E-05	0.014441	6.786851943	0.001426	0.00086	31.62863	0.000749	0.053612	0.066308	298.5995
	Oct-08	31		0.476975626	1.60904E-05	0.014561	6.796762293	0.00143	0.000863	31.67917	0.000751	0.053895	0.066308	297.8623
	Nov-08	30		0.477390359	1.60469E-05	0.014615	6.765477492	0.001427	0.000862	31.51197	0.000748	0.053784	0.066308	295.0161
	Dec-08	31		0.473931198	1.59587E-05	0.014619	6.764532597	0.001428	0.00086	31.59661	0.000752	0.054182	0.066308	293.9795

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Operational Tailings Pond Chemistry Predictions

Year	Month	Days/Month	Active Cell	Manganese	Mercury	Nickel	Potassium	Selenium	Silver	Sodium	Thallium	Zinc	Acidity	Hardness
Fixed Standards				0.05	#N/A	f(H)	#N/A	0.005	0.001	#N/A	0.00028	f(H)		
9	Jan-09	31	1E	0.473931198	1.59587E-05	0.014619	6.764532597	0.001428	0.00086	31.59661	0.000752	0.054182	0.066308	293.9795
	Feb-09	28		0.473931198	1.59587E-05	0.014619	6.764532597	0.001428	0.00086	31.59661	0.000752	0.054182	0.066308	293.9795
	Mar-09	31		0.473931198	1.59587E-05	0.014619	6.764532597	0.001428	0.00086	31.59661	0.000752	0.054182	0.066308	293.9795
	Apr-09	30		0.473931198	1.59587E-05	0.014619	6.764532597	0.001428	0.00086	31.59661	0.000752	0.054182	0.066308	293.9795
	May-09	31		0.473931198	1.59587E-05	0.014619	6.764532597	0.001428	0.00086	31.59661	0.000752	0.054182	0.066308	293.9795
	Jun-09	30		0.473931198	1.59587E-05	0.014619	6.764532597	0.001428	0.00086	31.59661	0.000752	0.054182	0.066308	293.9795
	Jul-09	31		0.473931198	1.59587E-05	0.014619	6.764532597	0.001428	0.00086	31.59661	0.000752	0.054182	0.066308	293.9795
	Aug-09	31		0.473931198	1.59587E-05	0.014619	6.764532597	0.001428	0.00086	31.59661	0.000752	0.054182	0.066308	293.9795
	Sep-09	30		0.473931198	1.59587E-05	0.014619	6.764532597	0.001428	0.00086	31.59661	0.000752	0.054182	0.066308	293.9795
	Oct-09	31		0.473931198	1.59587E-05	0.014619	6.764532597	0.001428	0.00086	31.59661	0.000752	0.054182	0.066308	293.9795
	Nov-09	30		0.473931198	1.59587E-05	0.014619	6.764532597	0.001428	0.00086	31.59661	0.000752	0.054182	0.066308	293.9795
	Dec-09	31		0.473931198	1.59587E-05	0.014619	6.764532597	0.001428	0.00086	31.59661	0.000752	0.054182	0.066308	293.9795
1	Jan-01	31	2E	0.005461201	5.67732E-07	0.001522	5.892935149	0.000666	1.28E-05	28.99714	1.83E-05	0.00244	0.066308	34.8194
	Feb-01	28		0.084035362	2.62216E-06	0.002962	4.452691215	0.000562	0.000105	20.98175	3.09E-05	0.00253	0.066308	40.55068
	Mar-01	31		0.15001162	4.21098E-06	0.004049	3.597193731	0.000516	0.000184	16.0001	4.49E-05	0.002839	0.066308	48.83522
	Apr-01	30		0.160575272	5.31743E-06	0.004923	3.30578381	0.000493	0.000202	13.93188	6.4E-05	0.003073	0.066308	55.22309
	May-01	31		0.229355951	6.37468E-06	0.0056	3.112091102	0.000525	0.000282	12.44344	7.48E-05	0.00365	0.066308	65.69827
	Jun-01	30		0.298176875	7.28356E-06	0.006165	3.074666999	0.000574	0.000362	11.81072	8.32E-05	0.004257	0.066308	76.257
	Jul-01	31		0.282797368	7.9607E-06	0.006701	3.194858495	0.000578	0.000352	11.88761	0.000105	0.004573	0.066308	84.08865
	Aug-01	31		0.302000753	8.40307E-06	0.006905	3.174155424	0.00059	0.000377	11.48628	0.000113	0.004906	0.066308	91.75229
	Sep-01	30		0.270051759	8.86741E-06	0.007256	3.281102768	0.000578	0.000348	11.59348	0.000132	0.005093	0.066308	98.30585
	Oct-01	31		0.27404779	9.42612E-06	0.007683	3.398294893	0.000596	0.000357	11.78442	0.000147	0.005419	0.066308	105.0392
	Nov-01	30		0.304401936	9.76868E-06	0.007913	3.397379247	0.000621	0.000392	11.602	0.000151	0.005683	0.066308	108.6375
	Dec-01	31		0.31221127	1.02858E-05	0.008341	3.510290357	0.000642	0.000406	11.81195	0.000165	0.006001	0.066308	114.3612
2	Jan-02	31	2E	0.388481721	1.0526E-05	0.008452	3.433863693	0.000693	0.000488	11.47277	0.000156	0.006305	0.066308	116.5626
	Feb-02	28		0.392703344	1.0945E-05	0.008791	3.540017059	0.000711	0.000497	11.6882	0.000168	0.006599	0.066308	121.5344
	Mar-02	31		0.401051206	1.13787E-05	0.009146	3.644539312	0.000732	0.000511	11.90206	0.000181	0.006919	0.066308	126.7018
	Apr-02	30		0.423045085	1.18548E-05	0.009403	3.749575174	0.000761	0.000538	12.13032	0.000189	0.007335	0.066308	135.4969
	May-02	31		0.398179279	1.24835E-05	0.009912	3.957666421	0.000768	0.000519	12.63957	0.000214	0.007724	0.066308	144.6042
	Jun-02	30		0.404927263	1.30724E-05	0.010362	4.118040998	0.000794	0.000533	13.03919	0.000229	0.008137	0.066308	152.6308
	Jul-02	31		0.426709028	1.31469E-05	0.010329	4.083983173	0.000806	0.000556	12.86588	0.000227	0.00829	0.066308	155.5223
	Aug-02	31		0.413774192	1.37429E-05	0.01078	4.259726848	0.000818	0.000549	13.28147	0.000247	0.008681	0.066308	164.5897
	Sep-02	30		0.406110628	1.43215E-05	0.011245	4.426611946	0.000834	0.000547	13.68636	0.000266	0.009054	0.066308	172.4151
	Oct-02	31		0.444608063	1.43655E-05	0.01126	4.373514326	0.000858	0.000588	13.5113	0.000259	0.009157	0.066308	172.1979
	Nov-02	30		0.464159235	1.4414E-05	0.011295	4.346401629	0.000871	0.000609	13.39008	0.000258	0.00925	0.066308	172.3546
	Dec-02	31		0.475369841	1.46822E-05	0.01152	4.404751348	0.000889	0.000624	13.51577	0.000265	0.009473	0.066308	175.457
3	Jan-03	31	2E	0.498761891	1.46689E-05	0.011506	4.361648385	0.000903	0.000649	13.37194	0.000261	0.009532	0.066308	174.7654
	Feb-03	28		0.497617988	1.48071E-05	0.011632	4.394682037	0.000909	0.00065	13.41221	0.000268	0.009677	0.066308	176.561
	Mar-03	31		0.487401704	1.49846E-05	0.011807	4.448509122	0.000911	0.000643	13.49946	0.000279	0.009854	0.066308	179.0849
	Apr-03	30		0.492193392	1.54224E-05	0.012059	4.574730866	0.00093	0.000653	13.8183	0.000291	0.010253	0.066308	187.9641
	May-03	31		0.482223454	1.61581E-05	0.01265	4.79721483	0.000951	0.000651	14.39451	0.000315	0.010766	0.066308	198.5452
	Jun-03	30		0.484719415	1.65921E-05	0.012978	4.921680861	0.00097	0.000658	14.71397	0.000328	0.011125	0.066308	205.3353
	Jul-03	31		0.500317497	1.67772E-05	0.013098	4.955020043	0.000986	0.000677	14.80036	0.000331	0.011325	0.066308	208.5002
	Aug-03	31		0.511214142	1.71092E-05	0.013358	5.039850844	0.001007	0.000692	15.02383	0.000339	0.011618	0.066308	213.2641
	Sep-03	30		0.529262935	1.70137E-05	0.013277	4.986094306	0.001015	0.000711	14.88109	0.000333	0.011619	0.066308	211.892
	Oct-03	31		0.543430419	1.66167E-05	0.012953	4.842815031	0.001009	0.000722	14.48343	0.000321	0.011422	0.066308	206.7586
	Nov-03	30		0.53385954	1.69142E-05	0.013218	4.940878205	0.001017	0.000717	14.71299	0.000335	0.011695	0.066308	211.25
	Dec-03	31		0.542469193	1.6995E-05	0.013299	4.955947402	0.001028	0.000727	14.75449	0.000338	0.011811	0.066308	212.188

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Operational Tailings Pond Chemistry Predictions

Year	Month	Days/Month	Active Cell	Manganese	Mercury	Nickel	Potassium	Selenium	Silver	Sodium	Thallium	Zinc	Acidity	Hardness
Fixed Standards				0.05	#N/A	f(H)	#N/A	0.005	0.001	#N/A	0.00028	f(H)		
4	Jan-04	31	2E	0.557904966	1.6722E-05	0.013117	4.85970007	0.001028	0.000741	14.4705	0.000328	0.011751	0.066308	208.1767
	Feb-04	28		0.55523669	1.69108E-05	0.013312	4.925334486	0.001037	0.000742	14.60777	0.000337	0.01201	0.066308	210.9648
	Mar-04	31		0.556625744	1.70462E-05	0.013465	4.970585809	0.001047	0.000747	14.70232	0.000344	0.012237	0.066308	213.0009
	Apr-04	30		0.568918205	1.73958E-05	0.013745	5.075227585	0.001071	0.000764	14.98492	0.000354	0.012622	0.066308	218.533
	May-04	31		0.577825578	1.80049E-05	0.014264	5.266437419	0.001105	0.000781	15.50772	0.000373	0.013197	0.066308	227.2424
	Jun-04	30		0.611836448	1.75709E-05	0.013887	5.103499319	0.001109	0.000812	15.10891	0.000351	0.012962	0.066308	221.1144
	Jul-04	31		0.614531464	1.79744E-05	0.014222	5.239631067	0.001131	0.000821	15.47082	0.000366	0.013413	0.066308	227.9189
	Aug-04	31		0.612958249	1.76987E-05	0.013978	5.160272699	0.001121	0.000817	15.23766	0.000361	0.013336	0.066308	225.9958
	Sep-04	30		0.583611058	1.80756E-05	0.014299	5.31093708	0.001122	0.000793	15.58518	0.000384	0.013773	0.066308	233.9093
	Oct-04	31		0.578033666	1.83099E-05	0.014507	5.391791376	0.00113	0.000792	15.7872	0.000395	0.014051	0.066308	238.1758
	Nov-04	30		0.583255702	1.82284E-05	0.014457	5.361729759	0.001132	0.000797	15.70656	0.000393	0.014055	0.066308	237.1424
	Dec-04	31		0.588365333	1.81263E-05	0.014397	5.328452729	0.001132	0.000802	15.62346	0.000391	0.01405	0.066308	235.8123
5	Jan-05	31	2E	0.597833087	1.78857E-05	0.014241	5.25196665	0.00113	0.00081	15.40783	0.000382	0.013981	0.066308	232.2994
	Feb-05	28		0.59347182	1.78474E-05	0.014276	5.262418929	0.001131	0.000807	15.55105	0.000387	0.014459	0.066308	232.2422
	Mar-05	31		0.587886707	1.80322E-05	0.014523	5.354473026	0.001144	0.000808	16.04156	0.000402	0.015448	0.066308	235.464
	Apr-05	30		0.586085204	1.78789E-05	0.014441	5.335313419	0.001143	0.000807	16.26106	0.000403	0.0162	0.066308	234.7915
	May-05	31		0.578546279	1.88012E-05	0.01532	5.677139799	0.001189	0.000816	17.65562	0.000444	0.018379	0.066308	249.107
	Jun-05	30		0.579796459	1.84822E-05	0.015088	5.599291471	0.001181	0.000817	17.72837	0.000439	0.01897	0.066308	245.9512
	Jul-05	31		0.567917225	1.90205E-05	0.015625	5.82397773	0.001207	0.000817	18.83468	0.000469	0.020911	0.066308	255.7817
	Aug-05	31		0.573804855	1.92401E-05	0.015871	5.923080406	0.001228	0.000829	19.50323	0.000481	0.02221	0.066308	259.8957
	Sep-05	30		0.583626351	1.89784E-05	0.015697	5.851876162	0.001228	0.000838	19.5602	0.000474	0.022665	0.066308	256.3168
	Oct-05	31		0.577125389	1.85844E-05	0.015431	5.755941907	0.001213	0.00083	19.55723	0.000469	0.023132	0.066308	251.6525
	Nov-05	30		0.562964168	1.89034E-05	0.015815	5.912571832	0.001229	0.000825	20.45732	0.000493	0.024845	0.066308	257.5414
	Dec-05	31		0.56927627	1.87212E-05	0.015727	5.869998457	0.00123	0.000832	20.58083	0.00049	0.025334	0.066308	254.7781
6	Jan-06	31	2E	0.576964047	1.83558E-05	0.015481	5.763661312	0.001223	0.000837	20.42644	0.000479	0.025448	0.066308	249.2
	Feb-06	28		0.574736125	1.84184E-05	0.015624	5.817716186	0.001232	0.000839	20.87264	0.000488	0.026438	0.066308	250.4201
	Mar-06	31		0.575914867	1.83853E-05	0.015681	5.834647555	0.001238	0.000843	21.18101	0.000493	0.027213	0.066308	250.0577
	Apr-06	30		0.584776045	1.85895E-05	0.015906	5.925011386	0.00126	0.000858	21.74131	0.000503	0.028289	0.066308	253.5819
	May-06	31		0.575208547	1.87987E-05	0.016171	6.040445872	0.001273	0.000856	22.44946	0.000521	0.029697	0.066308	258.0642
	Jun-06	30		0.575993704	1.91857E-05	0.016577	6.202090694	0.001299	0.000866	23.29748	0.000541	0.031223	0.066308	264.5565
	Jul-06	31		0.580845579	1.90686E-05	0.016499	6.176952536	0.0013	0.000871	23.38362	0.000539	0.031562	0.066308	263.4889
	Aug-06	31		0.578949538	1.90241E-05	0.016493	6.186702267	0.001301	0.000871	23.62107	0.000542	0.032166	0.066308	264.0407
	Sep-06	30		0.568187345	1.90545E-05	0.016577	6.232571322	0.001302	0.000863	24.01715	0.000552	0.033053	0.066308	265.8956
	Oct-06	31		0.554512574	1.87104E-05	0.01632	6.142957374	0.001281	0.000847	23.85548	0.000547	0.033092	0.066308	261.9856
	Nov-06	30		0.535288683	1.89135E-05	0.016594	6.257936403	0.001287	0.000834	24.54478	0.000567	0.034462	0.066308	266.3054
	Dec-06	31		0.536547787	1.87886E-05	0.016532	6.225076858	0.001284	0.000835	24.55062	0.000564	0.034615	0.066308	264.2535
7	Jan-07	31	2E	0.545843947	1.83973E-05	0.016212	6.085599539	0.001271	0.000839	24.06191	0.000547	0.033972	0.066308	257.6834
	Feb-07	28		0.543815809	1.83382E-05	0.016219	6.084227374	0.001271	0.000838	24.18222	0.000549	0.034344	0.066308	256.778
	Mar-07	31		0.545514994	1.83894E-05	0.01633	6.120667979	0.00128	0.000842	24.46265	0.000554	0.034948	0.066308	257.4127
	Apr-07	30		0.559116683	1.84633E-05	0.016387	6.1477313	0.001293	0.000858	24.65267	0.000555	0.035316	0.066308	258.8881
	May-07	31		0.550170141	1.91726E-05	0.017111	6.445994424	0.001333	0.000865	26.0923	0.000593	0.037814	0.066308	271.2765
	Jun-07	30		0.579164753	1.89848E-05	0.0169	6.34949481	0.001338	0.000889	25.6776	0.000574	0.037074	0.066308	267.2608
	Jul-07	31		0.595034247	1.89153E-05	0.016829	6.321687124	0.001346	0.000904	25.61816	0.000568	0.037009	0.066308	266.0954
	Aug-07	31		0.589830468	1.85334E-05	0.016487	6.206690914	0.001326	0.000894	25.24686	0.000557	0.036589	0.066308	261.6227
	Sep-07	30		0.572386812	1.90349E-05	0.017026	6.443361014	0.001351	0.00089	26.4557	0.000591	0.03878	0.066308	271.4998
	Oct-07	31		0.576457589	1.85241E-05	0.016553	6.258209933	0.001326	0.000885	25.71375	0.000569	0.037649	0.066308	263.7025
	Nov-07	30		0.559664923	1.83599E-05	0.016466	6.237273281	0.001313	0.000868	25.77302	0.000573	0.037967	0.066308	262.3996
	Dec-07	31		0.551054474	1.84679E-05	0.016636	6.308265227	0.00132	0.000864	26.20887	0.000585	0.038822	0.066308	264.6973

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Operational Tailings Pond Chemistry Predictions

Year	Month	Days/Month	Active Cell	Manganese	Mercury	Nickel	Potassium	Selenium	Silver	Sodium	Thallium	Zinc	Acidity	Hardness
Fixed Standards				0.05	#N/A	f(H)	#N/A	0.005	0.001	#N/A	0.00028	f(H)		
8	Jan-08	31	2E	0.561666865	1.8233E-05	0.016526	6.204053204	0.001313	0.000871	25.67263	0.000573	0.038524	0.066308	259.8933
	Feb-08	28		0.563441264	1.80747E-05	0.016477	6.145388069	0.001308	0.000872	25.39305	0.000568	0.038559	0.066308	256.8842
	Mar-08	31		0.565571481	1.81557E-05	0.016666	6.182533146	0.001317	0.000877	25.55926	0.000574	0.039305	0.066308	257.7499
	Apr-08	30		0.577960808	1.81899E-05	0.016737	6.185287678	0.001327	0.000891	25.53392	0.000573	0.039588	0.066308	258.104
	May-08	31		0.568733242	1.89075E-05	0.017526	6.488661941	0.001369	0.000899	26.95694	0.000614	0.042416	0.066308	270.7826
	Jun-08	30		0.580600184	1.85309E-05	0.017171	6.32819437	0.001353	0.000903	26.19612	0.000592	0.041362	0.066308	264.5177
	Jul-08	31		0.579574019	1.92592E-05	0.017951	6.624317541	0.001399	0.000918	27.55817	0.00063	0.044004	0.066308	276.9235
	Aug-08	31		0.608719659	1.86632E-05	0.017344	6.350550304	0.001376	0.000933	26.21325	0.00059	0.041788	0.066308	265.8055
	Sep-08	30		0.61479243	1.86987E-05	0.017416	6.36947023	0.001384	0.000941	26.30898	0.000592	0.042176	0.066308	266.5603
	Oct-08	31		0.617842753	1.83391E-05	0.017102	6.237092541	0.001368	0.000938	25.73122	0.000577	0.041397	0.066308	260.9396
	Nov-08	30		0.605074636	1.81248E-05	0.01697	6.188530217	0.001353	0.000924	25.60303	0.000577	0.041471	0.066308	258.5324
	Dec-08	31		0.593906765	1.81903E-05	0.017129	6.247731407	0.001357	0.000917	25.95393	0.000589	0.042373	0.066308	260.3834
9	Jan-09	31	2E	0.534976858	1.69161E-05	0.015826	6.49238962	0.001393	0.000891	29.07584	0.000672	0.048476	0.066308	276.3265
	Feb-09	28		0.534389714	1.69509E-05	0.015971	6.515266968	0.001397	0.000891	29.41664	0.000676	0.048766	0.066308	276.5392
	Mar-09	31		0.532704772	1.69264E-05	0.016057	6.51523564	0.001395	0.00089	29.6654	0.000677	0.048879	0.066308	276.1951
	Apr-09	30		0.531783352	1.70828E-05	0.016331	6.587524793	0.001406	0.000893	30.23669	0.000687	0.049576	0.066308	278.4453
	May-09	31		0.534975478	1.6596E-05	0.015904	6.399002525	0.001378	0.000886	29.55194	0.000664	0.048106	0.066308	270.4491
	Jun-09	30		0.529771217	1.67704E-05	0.016193	6.478614681	0.001387	0.000885	30.14204	0.000675	0.048878	0.066308	273.1195
	Jul-09	31		0.541031208	1.72403E-05	0.016757	6.670754215	0.001425	0.000907	31.25087	0.000697	0.050456	0.066308	280.5729
	Aug-09	31		0.554535038	1.66738E-05	0.016181	6.442121655	0.001399	0.000909	30.29285	0.000666	0.048548	0.066308	271.4155
	Sep-09	30		0.544729042	1.6561E-05	0.016168	6.407980854	0.001387	0.000897	30.31449	0.000665	0.048416	0.066308	269.3762
	Oct-09	31		0.539357274	1.67875E-05	0.016518	6.510149664	0.001399	0.000897	31.00147	0.00068	0.049382	0.066308	272.8075
	Nov-09	30		0.532583056	1.66313E-05	0.016445	6.456901119	0.001386	0.000888	30.90716	0.000675	0.049066	0.066308	270.0287
	Dec-09	31		0.527845338	1.6824E-05	0.016751	6.544677459	0.001396	0.000888	31.51228	0.000688	0.049904	0.066308	272.9211
10	Jan-10	31	Combined	0.535120599	1.65939E-05	0.016901	6.446350073	0.001389	0.000891	30.9621	0.000676	0.050313	0.066308	269.0786
	Feb-10	28		0.532388157	1.66295E-05	0.017317	6.459146118	0.001391	0.00089	30.98997	0.00068	0.051545	0.066308	269.3451
	Mar-10	31		0.531682367	1.67676E-05	0.017874	6.512944465	0.001401	0.000893	31.22631	0.000688	0.053202	0.066308	271.3508
	Apr-10	30		0.529879348	1.65619E-05	0.017973	6.424879765	0.001387	0.000887	30.74903	0.000679	0.053487	0.066308	268.0658
	May-10	31		0.535814705	1.69564E-05	0.018793	6.579118578	0.001418	0.000902	31.47191	0.000699	0.05593	0.066308	274.2322
	Jun-10	30		0.551697284	1.70307E-05	0.019144	6.597309284	0.001433	0.00092	31.49033	0.000699	0.056943	0.066308	275.3353
	Jul-10	31		0.571078368	1.67063E-05	0.018956	6.452743599	0.001426	0.000932	30.69404	0.000677	0.056326	0.066308	270.1763
	Aug-10	31		0.57786341	1.67172E-05	0.019239	6.450930882	0.001432	0.00094	30.64028	0.000676	0.057151	0.066308	270.2965
	Sep-10	30		0.565276441	1.62209E-05	0.018909	6.25360062	0.001393	0.000917	29.6615	0.000656	0.056159	0.066308	262.2058
	Oct-10	31		0.559291799	1.65489E-05	0.019658	6.387756863	0.001412	0.000919	30.31741	0.000675	0.05841	0.066308	267.2543
	Nov-10	30		0.558121136	1.64792E-05	0.019827	6.358877771	0.001408	0.000916	30.15575	0.000673	0.058905	0.066308	265.9837
	Dec-10	31		0.565523937	1.6277E-05	0.019762	6.27272317	0.001401	0.00092	29.6941	0.000662	0.058682	0.066308	262.6933
11	Jan-11	31	Combined	0.565561975	1.62607E-05	0.019989	6.267491053	0.001401	0.00092	29.64584	0.000662	0.059359	0.066308	262.3786
	Feb-11	28		0.565305843	1.62762E-05	0.020217	6.272975204	0.001403	0.00092	29.64592	0.000664	0.06004	0.066308	262.45
	Mar-11	31		0.564753565	1.62597E-05	0.02042	6.267305223	0.001403	0.00092	29.59737	0.000664	0.060648	0.066308	262.2036
	Apr-11	30		0.57424743	1.65711E-05	0.021032	6.391625116	0.001431	0.000937	30.17082	0.000679	0.062473	0.066308	267.225
	May-11	31		0.583312936	1.60192E-05	0.020368	6.167215315	0.001402	0.000934	29.02794	0.000648	0.060455	0.066308	258.6052
	Jun-11	30		0.594192553	1.64696E-05	0.021163	6.346590945	0.001439	0.000955	29.86995	0.00067	0.06283	0.066308	265.8648
	Jul-11	31		0.595658652	1.60706E-05	0.020727	6.186381699	0.001415	0.000948	29.0605	0.000649	0.061517	0.066308	259.7334
	Aug-11	31		0.599591702	1.63902E-05	0.021367	6.316956261	0.00144	0.00096	29.68061	0.000666	0.063436	0.066308	264.9341
	Sep-11	30		0.59844182	1.61542E-05	0.021172	6.224323778	0.001425	0.000954	29.21324	0.000655	0.062847	0.066308	261.1951
	Oct-11	31		0.598894024	1.63261E-05	0.021601	6.297223179	0.001438	0.000959	29.56013	0.000665	0.064135	0.066308	263.9714
	Nov-11	30		0.606500566	1.60727E-05	0.021306	6.192925743	0.001427	0.000961	29.01876	0.00065	0.06323	0.066308	259.9517
	Dec-11	31		0.60196196	1.60774E-05	0.021464	6.19819105	0.001428	0.000961	29.03634	0.000652	0.063703	0.066308	260.0546

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Operational Tailings Pond Chemistry Predictions

Year	Month	Days/Month	Active Cell	Manganese	Mercury	Nickel	Potassium	Selenium	Silver	Sodium	Thallium	Zinc	Acidity	Hardness
Fixed Standards				0.05	#N/A	f(H)	#N/A	0.005	0.001	#N/A	0.00028	f(H)		
12	Jan-12	31	Combined	0.609559748	1.61437E-05	0.021493	6.207214308	0.001426	0.000962	28.91992	0.000648	0.063004	0.066308	258.9644
	Feb-12	28		0.615873915	1.61928E-05	0.021469	6.207617309	0.001425	0.000965	28.76558	0.000643	0.062234	0.066308	257.7616
	Mar-12	31		0.618729285	1.6082E-05	0.021227	6.146741926	0.001411	0.000962	28.3232	0.000631	0.060801	0.066308	254.0642
	Apr-12	30		0.632022221	1.63793E-05	0.02155	6.244220058	0.00143	0.000978	28.62649	0.000637	0.06104	0.066308	256.9313
	May-12	31		0.653995245	1.61793E-05	0.021054	6.139340604	0.001424	0.000992	27.94872	0.000615	0.058929	0.066308	252.2622
	Jun-12	30		0.668486479	1.6247E-05	0.021017	6.145529681	0.00143	0.001005	27.82439	0.000609	0.058193	0.066308	251.7688
	Jul-12	31		0.685082802	1.63689E-05	0.021059	6.17619737	0.001442	0.001021	27.81958	0.000606	0.057684	0.066308	252.2528
	Aug-12	31		0.66314583	1.55187E-05	0.019819	5.837680074	0.001369	0.000978	26.15229	0.000566	0.05373	0.066308	238.1806
	Sep-12	30		0.657597091	1.57988E-05	0.020247	5.942819962	0.001376	0.000976	26.55226	0.000577	0.054373	0.066308	240.8788
	Oct-12	31		0.659066684	1.60246E-05	0.020554	6.023742994	0.001385	0.000979	26.82637	0.000584	0.054667	0.066308	242.8286
	Nov-12	30		0.664671807	1.57206E-05	0.020019	5.893183447	0.001366	0.000976	26.1157	0.000565	0.052746	0.066308	237.1783
	Dec-12	31		0.665299296	1.57422E-05	0.020021	5.894426713	0.001362	0.000975	26.02969	0.000562	0.052266	0.066308	236.2545
13	Jan-13	31	Combined	0.659736986	1.56144E-05	0.020079	5.839070137	0.001353	0.000969	25.68047	0.00056	0.052621	0.066308	234.1649
	Feb-13	28		0.65597238	1.56814E-05	0.020387	5.858753605	0.001356	0.000968	25.67925	0.000567	0.053604	0.066308	234.8214
	Mar-13	31		0.652940955	1.56814E-05	0.020609	5.853249133	0.001356	0.000966	25.56369	0.00057	0.054373	0.066308	234.6238
	Apr-13	30		0.656651471	1.58306E-05	0.020987	5.90110504	0.001368	0.000974	25.67879	0.000578	0.055548	0.066308	236.8816
	May-13	31		0.665894491	1.56738E-05	0.020858	5.829203313	0.001366	0.000981	25.25293	0.00057	0.05536	0.066308	234.6536
	Jun-13	30		0.661082878	1.55121E-05	0.020781	5.760803524	0.001354	0.000974	24.86824	0.000565	0.055306	0.066308	232.2451
	Jul-13	31		0.677184382	1.60881E-05	0.021783	5.972892563	0.0014	0.001003	25.71557	0.000591	0.058127	0.066308	240.5993
	Aug-13	31		0.682873429	1.58131E-05	0.021447	5.857211112	0.001388	0.001005	25.11398	0.000577	0.057357	0.066308	236.7064
	Sep-13	30		0.680192899	1.57945E-05	0.021578	5.845581463	0.001387	0.001003	24.99414	0.000579	0.057834	0.066308	236.267
	Oct-13	31		0.678447607	1.58344E-05	0.021811	5.859342489	0.00139	0.001003	24.98932	0.000584	0.058587	0.066308	236.758
	Nov-13	30		0.680981885	1.5642E-05	0.021606	5.780636652	0.001381	0.001003	24.57384	0.000575	0.058141	0.066308	233.9656
	Dec-13	31		0.679757976	1.56694E-05	0.021799	5.789583899	0.001383	0.001003	24.5537	0.000579	0.058776	0.066308	234.2876
14	Jan-14	31	Combined	0.676363157	1.56398E-05	0.021931	5.778440616	0.001382	0.001001	24.48953	0.000582	0.059394	0.066308	234.1479
	Feb-14	28		0.672494007	1.56501E-05	0.022101	5.781432337	0.001382	0.000999	24.48427	0.000586	0.060082	0.066308	234.4313
	Mar-14	31		0.668447237	1.56582E-05	0.02229	5.785341434	0.001383	0.000996	24.48885	0.00059	0.060835	0.066308	234.7965
	Apr-14	30		0.664738146	1.56616E-05	0.022447	5.786285424	0.001383	0.000994	24.47791	0.000594	0.061487	0.066308	235.1213
	May-14	31		0.67764655	1.57603E-05	0.022657	5.81817065	0.0014	0.001011	24.58522	0.000598	0.062266	0.066308	237.019
	Jun-14	30		0.673204077	1.54805E-05	0.022302	5.70931268	0.001381	0.001001	24.09593	0.000587	0.061476	0.066308	233.283
	Jul-14	31		0.680066019	1.58621E-05	0.023045	5.853281402	0.001411	0.001018	24.70116	0.000607	0.063731	0.066308	239.177
	Aug-14	31		0.690133663	1.5912E-05	0.023173	5.867528998	0.001423	0.00103	24.7368	0.000609	0.064257	0.066308	240.3085
	Sep-14	30		0.692492823	1.58494E-05	0.023145	5.841099613	0.001422	0.001032	24.60386	0.000607	0.064341	0.066308	239.6375
	Oct-14	31		0.676541006	1.55013E-05	0.022742	5.712200233	0.001392	0.00101	24.04886	0.000596	0.063383	0.066308	234.6319
	Nov-14	30		0.674016531	1.55442E-05	0.022932	5.728946154	0.001395	0.001009	24.11188	0.0006	0.064067	0.066308	235.3893
	Dec-14	31		0.67048891	1.55303E-05	0.023036	5.726642927	0.001394	0.001007	24.09576	0.000603	0.06451	0.066308	235.3959
15	Jan-15	31	Combined	0.673550918	1.56538E-05	0.023175	5.752100793	0.001393	0.001008	24.17602	0.000601	0.06418	0.066308	234.7882
	Feb-15	28		0.677189963	1.56495E-05	0.023071	5.728986833	0.001385	0.001007	24.04178	0.000593	0.063253	0.066308	232.5529
	Mar-15	31		0.680881344	1.56604E-05	0.023005	5.712284409	0.001378	0.001007	23.94035	0.000586	0.062397	0.066308	230.5086
	Apr-15	30		0.688784131	1.54986E-05	0.022588	5.628626678	0.001364	0.001007	23.54289	0.000569	0.060639	0.066308	226.3608
	May-15	31		0.696102122	1.55979E-05	0.022656	5.645784486	0.001365	0.001013	23.58568	0.000565	0.060207	0.066308	225.8247
	Jun-15	30		0.707644332	1.57213E-05	0.022732	5.670820059	0.001371	0.001023	23.65652	0.000562	0.059828	0.066308	225.8185
	Jul-15	31		0.705857871	1.54721E-05	0.022239	5.560445637	0.001347	0.001013	23.15921	0.000544	0.057969	0.066308	220.7047
	Aug-15	31		0.718530951	1.58529E-05	0.022777	5.683219989	0.001369	0.00103	23.65357	0.000554	0.058818	0.066308	224.192
	Sep-15	30		0.710794489	1.51733E-05	0.021573	5.416713678	0.001318	0.001005	22.49418	0.000518	0.055213	0.066308	213.5253
	Oct-15	31		0.710190048	1.54525E-05	0.022035	5.507055587	0.001326	0.001007	22.86522	0.000527	0.055901	0.066308	215.4774
	Nov-15	30		0.708910754	1.5367E-05	0.021852	5.462300854	0.001314	0.001001	22.65588	0.000518	0.054974	0.066308	212.7889
	Dec-15	31		0.710496881	1.54433E-05	0.021944	5.479089854	0.001312	0.001002	22.71028	0.000518	0.05475	0.066308	212.3261

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Project NorthMet
Number 1UP005.001
Date 7/17/2007 16:37
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Operational Tailings Pond Chemistry Predictions

Year	Month	Days/Month	Active Cell	Manganese	Mercury	Nickel	Potassium	Selenium	Silver	Sodium	Thallium	Zinc	Acidity	Hardness
Fixed Standards				0.05	#N/A	f(H)	#N/A	0.005	0.001	#N/A	0.00028	f(H)		
16	Jan-16	31	Combined	0.716746019	1.5284E-05	0.021579	5.407327602	0.001302	0.001002	22.36582	0.000504	0.053382	0.066308	209.0546
	Feb-16	28		0.718485258	1.52902E-05	0.021533	5.398188258	0.001298	0.001002	22.29296	0.0005	0.052868	0.066308	207.8537
	Mar-16	31		0.722733134	1.53572E-05	0.021586	5.41142478	0.001298	0.001005	22.31713	0.000498	0.052574	0.066308	207.4655
	Apr-16	30		0.732762802	1.54996E-05	0.02172	5.450305744	0.001307	0.001015	22.44331	0.000498	0.052506	0.066308	208.2747
	May-16	31		0.740582291	1.55004E-05	0.021631	5.438843593	0.001307	0.001021	22.36031	0.000492	0.051899	0.066308	207.2924
	Jun-16	30		0.750784987	1.56157E-05	0.021721	5.468506369	0.001315	0.001031	22.44917	0.000491	0.051747	0.066308	207.8375
	Jul-16	31		0.763966782	1.57296E-05	0.021791	5.497358597	0.001325	0.001044	22.53376	0.000489	0.05155	0.066308	208.4737
	Aug-16	31		0.766478572	1.53691E-05	0.02111	5.357221746	0.001304	0.001038	21.91381	0.00047	0.04961	0.066308	203.3278
	Sep-16	30		0.764366334	1.53714E-05	0.021101	5.350681098	0.001298	0.001034	21.86532	0.000467	0.049267	0.066308	202.2913
	Oct-16	31		0.766992403	1.54636E-05	0.021219	5.375930925	0.0013	0.001037	21.94878	0.000468	0.049221	0.066308	202.446
	Nov-16	30		0.764790265	1.53821E-05	0.021068	5.33985034	0.00129	0.001031	21.7777	0.000462	0.048575	0.066308	200.4947
	Dec-16	31		0.763632961	1.54103E-05	0.021105	5.343570788	0.001287	0.001029	21.77558	0.000461	0.048363	0.066308	199.8626
17	Jan-17	31	Combined	0.766618697	1.52711E-05	0.020869	5.303746906	0.00128	0.001028	21.50103	0.000451	0.047516	0.066308	197.5575
	Feb-17	28		0.768714465	1.53187E-05	0.020949	5.328791546	0.001282	0.00103	21.5078	0.000449	0.047421	0.066308	197.3759
	Mar-17	31		0.771646926	1.53529E-05	0.021014	5.350670665	0.001284	0.001033	21.50029	0.000446	0.047274	0.066308	197.0892
	Apr-17	30		0.778792994	1.54638E-05	0.021178	5.400317515	0.001293	0.001041	21.60566	0.000446	0.047377	0.066308	197.9324
	May-17	31		0.784381	1.54737E-05	0.021184	5.414269477	0.001296	0.001046	21.56818	0.000442	0.04713	0.066308	197.5934
	Jun-17	30		0.794407119	1.56091E-05	0.021369	5.471382377	0.001308	0.001057	21.70775	0.000442	0.047296	0.066308	198.8265
	Jul-17	31		0.781132621	1.52052E-05	0.020779	5.337611462	0.001277	0.001036	21.08984	0.000426	0.045763	0.066308	193.5066
	Aug-17	31		0.788395547	1.53801E-05	0.021057	5.410014552	0.00129	0.001046	21.29925	0.000429	0.046142	0.066308	195.1134
	Sep-17	30		0.791030959	1.5336E-05	0.020983	5.402416563	0.001288	0.001047	21.19301	0.000424	0.045769	0.066308	194.1795
	Oct-17	31		0.792970641	1.53581E-05	0.021034	5.419765232	0.001289	0.001048	21.18978	0.000422	0.045667	0.066308	193.9519
	Nov-17	30		0.790756539	1.5302E-05	0.020973	5.408402635	0.001284	0.001045	21.07775	0.000418	0.045337	0.066308	192.75
	Dec-17	31		0.79013034	1.53357E-05	0.021062	5.430221772	0.001284	0.001044	21.099	0.000418	0.045329	0.066308	192.5921
18	Jan-18	31	Combined	0.791008747	1.51839E-05	0.020949	5.413088878	0.001281	0.001043	20.98844	0.000409	0.045122	0.066308	191.0192
	Feb-18	28		0.790885362	1.52265E-05	0.021154	5.462330424	0.001285	0.001044	21.14105	0.000409	0.045582	0.066308	191.4183
	Mar-18	31		0.791530763	1.52671E-05	0.021371	5.514104355	0.00129	0.001046	21.30582	0.000408	0.046069	0.066308	191.8812
	Apr-18	30		0.791695354	1.5288E-05	0.02154	5.55548237	0.001293	0.001047	21.43058	0.000407	0.046453	0.066308	192.132
	May-18	31		0.797070903	1.54192E-05	0.021874	5.63785098	0.001306	0.001055	21.71564	0.000409	0.047193	0.066308	193.7418
	Jun-18	30		0.794377432	1.53126E-05	0.021827	5.631109474	0.001301	0.001051	21.65235	0.000404	0.047122	0.066308	192.6303
	Jul-18	31		0.805176127	1.53955E-05	0.022027	5.692685156	0.001316	0.001064	21.8533	0.000403	0.047587	0.066308	194.0011
	Aug-18	31		0.813184443	1.54364E-05	0.022168	5.737712412	0.001326	0.001073	21.99234	0.000401	0.047921	0.066308	194.8105
	Sep-18	30		0.807408503	1.52577E-05	0.021995	5.698902338	0.001315	0.001065	21.81182	0.000394	0.047573	0.066308	192.748
	Oct-18	31		0.797199359	1.51805E-05	0.022051	5.702772569	0.001307	0.001053	21.80017	0.000392	0.047706	0.066308	191.6121
	Nov-18	30		0.793821705	1.52722E-05	0.02236	5.769023351	0.001311	0.001052	22.02812	0.000395	0.04838	0.066308	192.4944
	Dec-18	31		0.794059365	1.51363E-05	0.02221	5.741165523	0.001306	0.00105	21.89413	0.000389	0.048083	0.066308	191.1029
19	Jan-19	31	Combined	0.792476067	1.51312E-05	0.022324	5.784381041	0.001308	0.001049	21.80983	0.000387	0.04808	0.066308	190.7082
	Feb-19	28		0.792902439	1.51713E-05	0.022482	5.838177121	0.001312	0.00105	21.79118	0.000386	0.048201	0.066308	190.8513
	Mar-19	31		0.792707474	1.51648E-05	0.022574	5.877046903	0.001315	0.001051	21.70719	0.000383	0.04817	0.066308	190.5143
	Apr-19	30		0.800997929	1.53349E-05	0.022924	5.981843647	0.001332	0.001062	21.87792	0.000386	0.048701	0.066308	192.3905
	May-19	31		0.812373677	1.52454E-05	0.022747	5.974932239	0.001338	0.001072	21.64632	0.000378	0.048142	0.066308	191.6389
	Jun-19	30		0.830242652	1.54738E-05	0.023125	6.096361167	0.001364	0.001095	21.89049	0.00038	0.048754	0.066308	194.4835
	Jul-19	31		0.827346147	1.51933E-05	0.022692	6.015858708	0.00135	0.001088	21.41038	0.000369	0.047679	0.066308	191.3395
	Aug-19	31		0.822274865	1.51666E-05	0.022774	6.044961572	0.001348	0.001082	21.32568	0.000367	0.047669	0.066308	190.7719
	Sep-19	30		0.820395817	1.52053E-05	0.022953	6.098054334	0.001351	0.001081	21.33596	0.000367	0.04787	0.066308	190.9491
	Oct-19	31		0.806373352	1.49707E-05	0.022697	6.0394136	0.001332	0.001064	20.96092	0.000361	0.047172	0.066308	187.8326
	Nov-19	30		0.806893105	1.50042E-05	0.022839	6.085848993	0.001336	0.001065	20.96285	0.00036	0.047312	0.066308	188.0515
	Dec-19	31		0.803914081	1.4951E-05	0.022842	6.096525672	0.001333	0.001062	20.84481	0.000358	0.047168	0.066308	187.2462
20	Jan-20	31	Combined	0.797628997	1.48798E-05	0.022984	6.159272616	0.001334	0.001056	20.64992	0.000356	0.047164	0.066308	186.4022
	Feb-20	28		0.792377604	1.48666E-05	0.023195	6.234795416	0.001337	0.001053	20.54279	0.000355	0.04733	0.066308	186.137
	Mar-20	31		0.792980046	1.49533E-05	0.023584	6.35940686	0.001351	0.001057	20.57594	0.000357	0.047844	0.066308	187.2046
	Apr-20	30		0.794896664	1.49737E-05	0.023797	6.444577412	0.001361	0.001061	20.50768	0.000357	0.048029	0.066308	187.7113
	May-20	31		0.806804294	1.50029E-05	0.023945	6.525143593	0.001378	0.001075	20.44295	0.000354	0.048103	0.066308	188.5987
	Jun-20	30		0.803689271	1.48186E-05	0.02376	6.508554849	0.001373	0.001071	20.09557	0.000348	0.047523	0.066308	186.7015
	Jul-20	31		0.809498931	1.49972E-05	0.024268	6.663686464	0.001394	0.001081	20.26274	0.000352	0.048305	0.066308	188.9464
	Aug-20	31		0.819977365	1.51135E-05	0.024596	6.781042703	0.001414	0.001096	20.33541	0.000353	0.04875	0.066308	190.682
	Sep-20	30		0.818491683	1.49678E-05	0.024457	6.772335747	0.001411	0.001093	20.05486	0.000348	0.04829	0.066308	189.1803
	Oct-20	31		0.817518637	1.4988E-05	0.024694	6.854668505	0.001419	0.001094	20.02168	0.000348	0.048564	0.066308	189.5239
	Nov-20	30		0.806496085	1.46419E-05	0.024207	6.749967203	0.001398	0.001079	19.48781	0.000338	0.047452	0.066308	185.5804
	Dec-20	31		0.800490636	1.47049E-05	0.024577	6.855887354	0.001404	0.001075	19.52739	0.000341	0.047982	0.066308	186.1738

Attachment 2-15

SRK Consulting. RS82 - Update on Use of Kinetic Test Data for Water Quality Predictions – DRAFT 02. Memorandum to Kim Lapakko, MDNR. February 2, 2009.

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Memo

To:	Kim Lapakko, MDNR	**Date:**	February 2, 2009
cc:	Stuart Arkley, MDNR Jim Scott, PolyMet Miguel Wong, Barr	**From:**	Stephen Day
Subject:	RS82 - Update on Use of Kinetic Test Data for Water Quality Predictions – DRAFT 02[1]	**Project #:**	1UP005.001

1 Introduction

First drafts ("Draft 01") of RS31 (Open Pit Water Quality), RS42 (Waste Rock and Lean Ore Stockpile Water Quality), RS46 (Tailings) and RS65 (Hydrometallurgical Residues) were prepared on the basis of dissolution test data available after about 1 year of testwork (Table 1). The water quality estimates presented in these reports formed the basis for assessment of project water quality effects evaluated in RS74 (Draft 01). The dissolution tests have continued and subsequent data used to develop updated water quality estimates for input into Draft 02 of RS74 delivered for agency review in September 2008 as shown in Table 1. Further explanation of the data ranges used is provided below.

The purpose of this memorandum is to describe the recent dissolution data and the effect on the water quality predictions presented in RS74 Draft 02. Charts of the data attached and electronic data files are available.

[1] This version replaces version dated December 12, 2008

Table 1: Testwork Ranges (in weeks) Used as Basis for RS74

Material Type	Project Facility											
	Waste Rock			Open Pits			Tailings			Hydrometallurgical Residue Cells		
	RS74 Draft 01	RS74 Draft 02	Project Website	RS74 Draft 01	RS74 Draft 02	Project Website	RS74 Draft 01	RS74 Draft 02	Project Website	RS74 Draft 01	RS74 Draft 02	Project Website
	RS42 Draft 01	RS82 Report	Data to October 08	RS31 Draft 01	RS82 Report	Data to October 08	RS46 Draft 01	RS82 Report	Data to October 08	RS65 Draft 01	RS82 Report	Data to October 08
Waste Rock and Lean Ore	About 60	About 130	About 160	About 60	About 130	About 160	About 60	About 130	About 160	Not Applicable	Not Applicable	Not Applicable
Ore	Not Applicable	Not Applicable	Not Applicable	Not Applicable	Not Applicable	Not Applicable	60	60	About 160	Not Applicable	Not Applicable	Not Applicable
POM Tailings	Not Applicable	Not Applicable	Not Applicable	Not Applicable	Not Applicable	Not Applicable	About 60	95	About 130	Not Applicable	Not Applicable	Not Applicable
Hydrometallurgical Residues	Not Applicable	Not Applicable	Not Applicable	Not Applicable	Not Applicable	Not Applicable	Not Applicable	Not Applicable	Not Applicable	36	About 115	About 130

1. The geochemical behavior of ore exposures was evaluated using results for waste rock and lean ore samples because the three ore samples were process composites.

2 Data Presentation

Electronic updates to the data graphing and calculations are available on Barr Engineering's NorthMet Project website. These data shown are the most recent for which quality assurance checks are complete (Table 1). For example, waste rock data are now available for 3 years. Reporting of some recent metals data has been delayed due to an instrument failure at the analytical laboratory.

The descriptions of data in the following sections are with reference to the trends as observed at the time the data were used to generate inputs into RS74 (Draft 02) but graphs show the data available as of October 2008.

Selected graphs are provided for illustration in the following sections.

3 Waste Rock, Lean Ore and Ore Kinetic Tests

3.1 Status of Program

Description of the waste rock, lean ore and ore kinetic test program, including characteristics of the samples being tested can be found in RS42 (Draft 01). The test program consists mainly of conventional humidity cells with additional tests in other configurations to evaluate dissolution effects for size fractions.

Most of the tests had been running between 60 and 70 weeks when the data were interpreted for use in RS42 and RS31 (Draft 01). A few lean ore tests were started later due to modification of the program and had about 55 weeks of data for use in the reports. RS74 (Draft 02) was based on about 2.5 years of data as shown in Table 1.

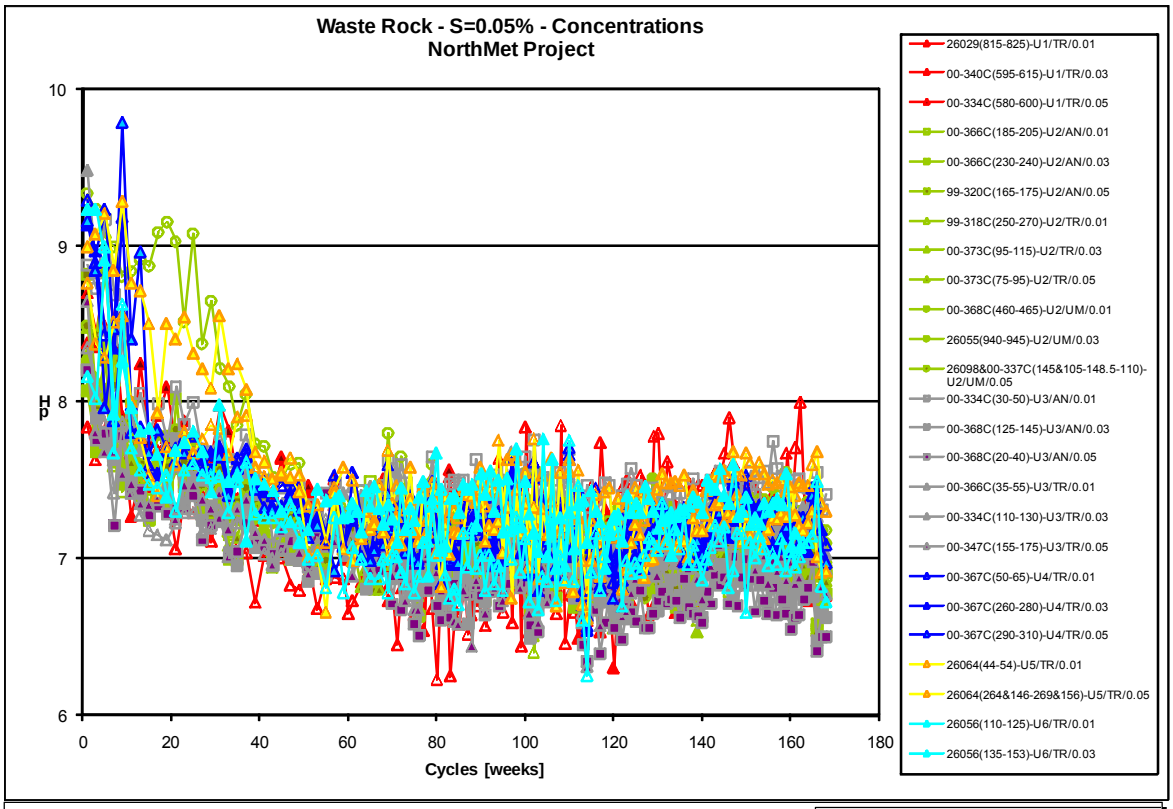
All tests originally started are continuing for the durations shown in Table 1. None of the tests have been terminated and the analytical program is unchanged from that developed in consultation with the MDNR.

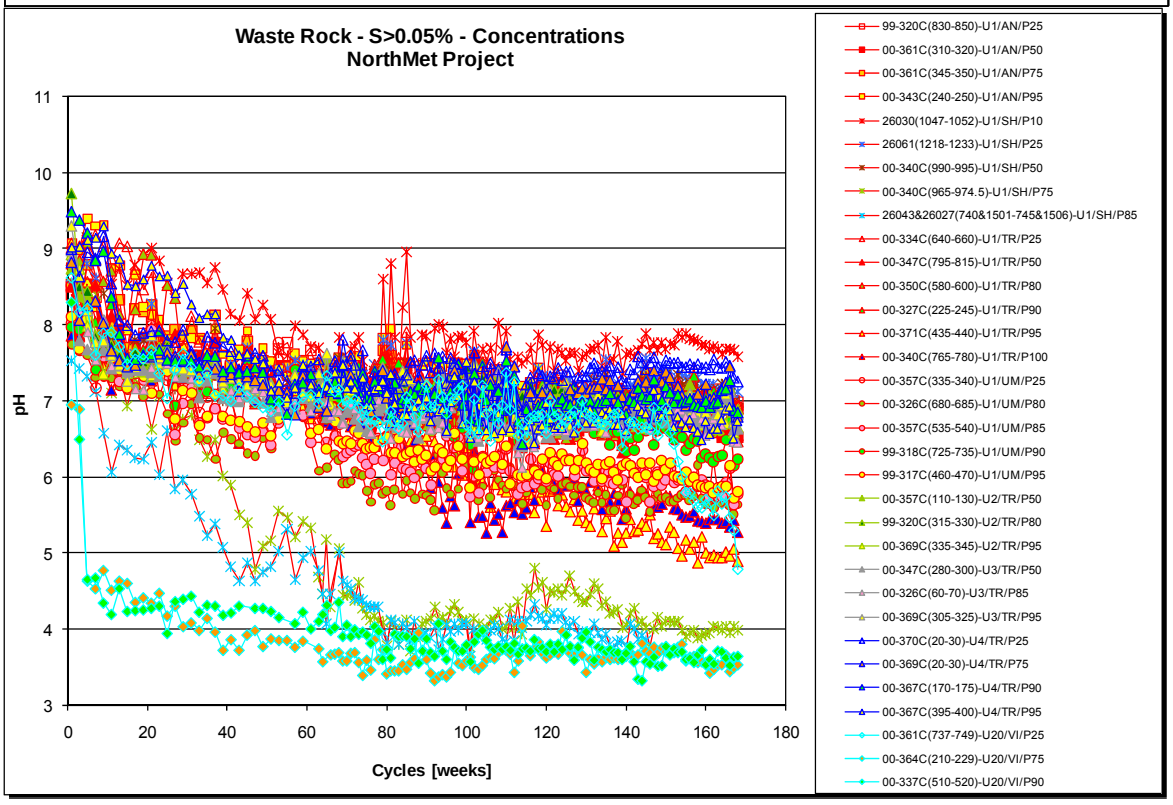
3.2 Description of Results

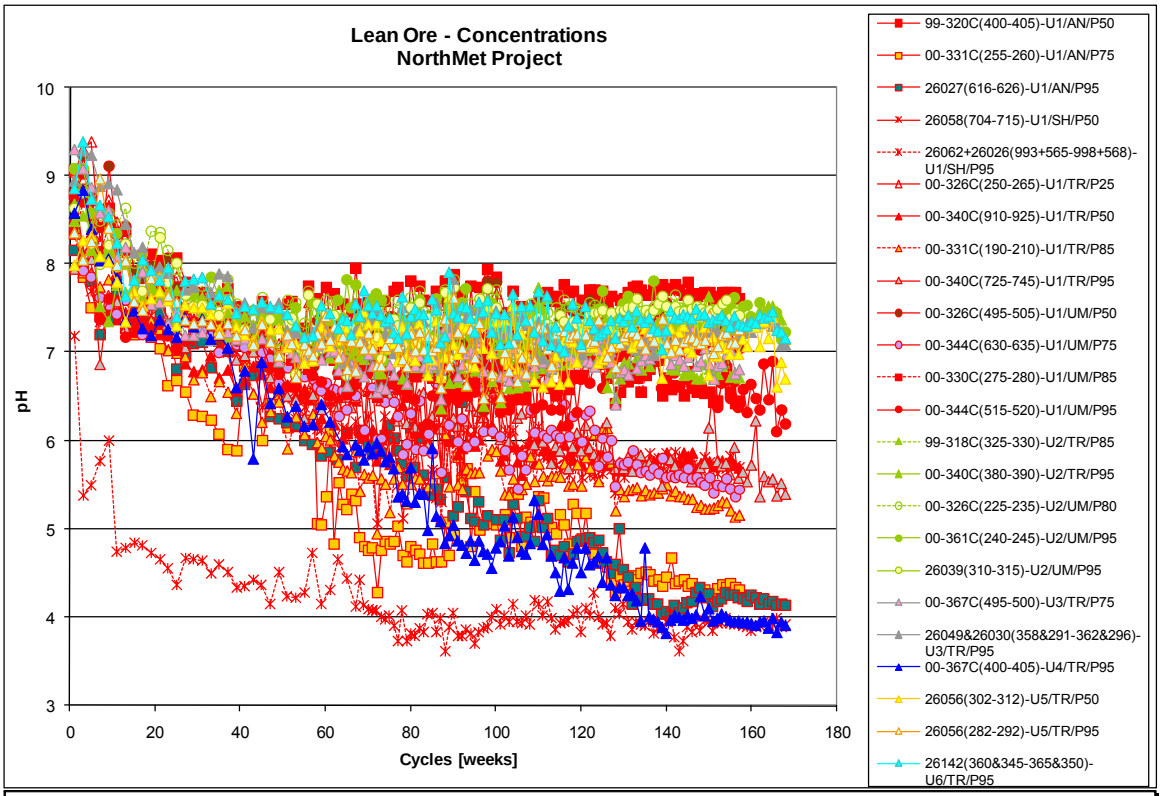
For consistency with previous reporting, graphs are provided in the electronic data package in the same four groups as RS42 (waste rock samples with sulfur less than or equal to 0.05%, waste rock samples with sulfur greater than 0.05%, lean ore samples and the three composite samples). Description of major trends as a comparison to trends observed after about 60 weeks of testing are provided below for each group.

3.2.1 Waste Rock S≤0.05% Group

The earlier test data showed declining pH from initial values above 8 (Figure 1).


Waste Rock - S=0.05% - Concentrations
NorthMet Project
Hp
Cycles [weeks]
26029(815-825)-U1/TR/0.01
00-340C(595-615)-U1/TR/0.03
00-334C(580-600)-U1/TR/0.05
00-366C(185-205)-U2/AN/0.01
00-366C(230-240)-U2/AN/0.03
99-320C(165-175)-U2/AN/0.05
99-318C(250-270)-U2/TR/0.01
00-373C(95-115)-U2/TR/0.03
00-373C(75-95)-U2/TR/0.05
00-368C(460-465)-U2/UM/0.01
26055(940-945)-U2/UM/0.03
26098&00-337C(145&105-148.5-110)-U2/UM/0.05
00-334C(30-50)-U3/AN/0.01
00-368C(125-145)-U3/AN/0.03
00-368C(20-40)-U3/AN/0.05
00-366C(35-55)-U3/TR/0.01
00-334C(110-130)-U3/TR/0.03
00-347C(155-175)-U3/TR/0.05
00-367C(50-65)-U4/TR/0.01
00-367C(260-280)-U4/TR/0.03
00-367C(290-310)-U4/TR/0.05
26064(44-54)-U5/TR/0.01
26064(264&146-269&156)-U5/TR/0.05
26056(110-125)-U6/TR/0.01
26056(135-153)-U6/TR/0.03


Waste Rock - S>0.05% - Concentrations
NorthMet Project
pH
Cycles [weeks]
99-320C(830-850)-U1/AN/P25
00-361C(310-320)-U1/AN/P50
00-361C(345-350)-U1/AN/P75
00-343C(240-250)-U1/AN/P95
26030(1047-1052)-U1/SH/P10
26061(1218-1233)-U1/SH/P25
00-340C(990-995)-U1/SH/P50
00-340C(965-974.5)-U1/SH/P75
26043&26027(740&1501-745&1506)-U1/SH/P85
00-334C(640-660)-U1/TR/P25
00-347C(795-815)-U1/TR/P50
00-350C(580-600)-U1/TR/P80
00-327C(225-245)-U1/TR/P90
00-371C(435-440)-U1/TR/P95
00-340C(765-780)-U1/TR/P100
00-357C(335-340)-U1/UM/P25
00-326C(680-685)-U1/UM/P80
00-357C(535-540)-U1/UM/P85
99-318C(725-735)-U1/UM/P90
99-317C(460-470)-U1/UM/P95
00-357C(110-130)-U2/TR/P50
99-320C(315-330)-U2/TR/P80
00-369C(335-345)-U2/TR/P95
00-347C(280-300)-U3/TR/P50
00-326C(60-70)-U3/TR/P85
00-369C(305-325)-U3/TR/P95
00-370C(20-30)-U4/TR/P25
00-369C(20-30)-U4/TR/P75
00-367C(170-175)-U4/TR/P90
00-367C(395-400)-U4/TR/P95
00-361C(737-749)-U20/VI/P25
00-364C(210-229)-U20/VI/P75
00-337C(510-520)-U20/VI/P90



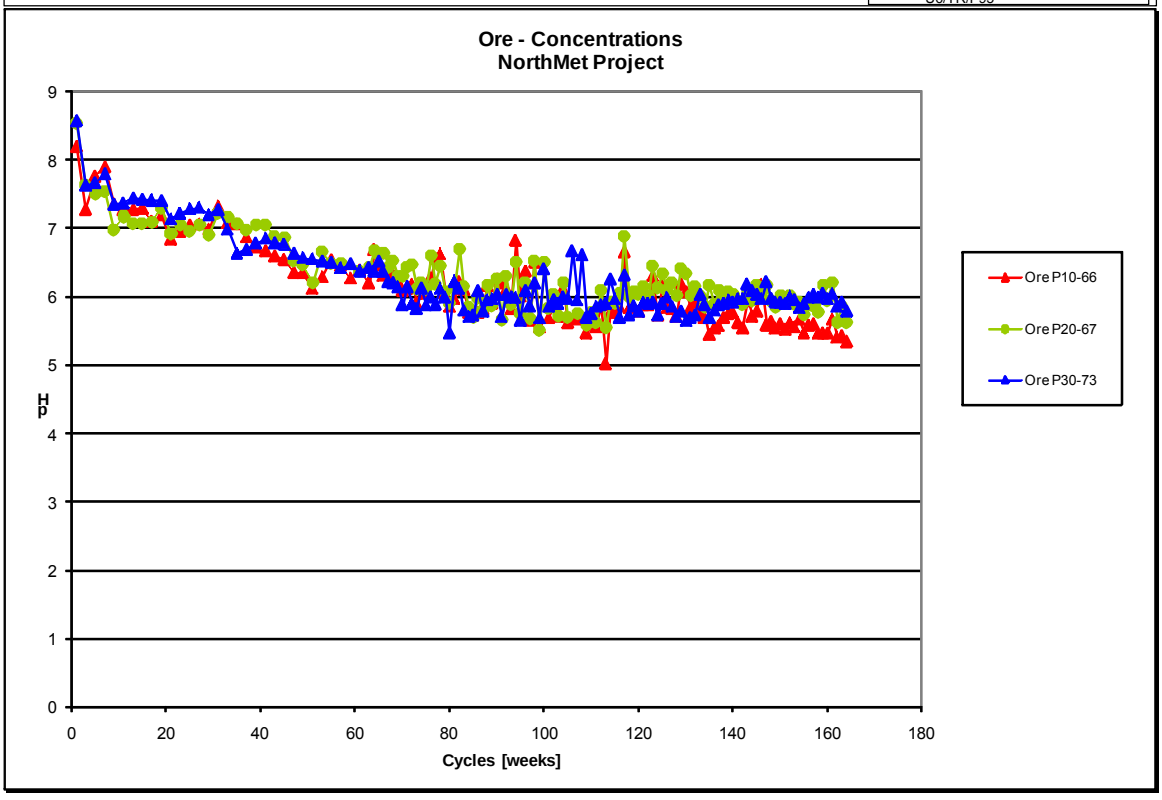


Figure 1: pH Trends for Rock Humidity Cells

The main feature of subsequent data has been that the variability of pH readings has increased but downward or upward trends are not apparent. Leachate pHs have remained between 6.5 and 7.5.

Other parameters have mostly shown stable flat or decreasing trends. Sulfate release has shown generally decreasing trends. Metal leaching also became erratic as pHs became more variable. For example, at about week 40, copper release was typically in the range 0.0002 to 0.001 mg/kg/week but this range has expanded to release rates near 0.002 mg/kg/week. This does not appear to be part of clear upward trends but reflects subtle differences in metal leaching due to variations in pH.

3.2.2 Waste Rock S>0.05% Group

Leachates from this group are now showing a clear separation into three pH sub-groups (Figure 1), which was only partially apparent after about 1 year of testing:

- pH 3.4 to 4.7 – Four samples of Virginia Formation and sedimentary hornfels rock with concentrations of sulfur above 1%.
- 5<pH<6.2 – Five samples of Duluth Complex rock showing downward trending pH.
- pH>6.4 – Balance of Duluth Complex and Sedimentary Honfels rock samples.

The lowest pH sub-group showed increasing metal leaching trends in the early test data. These tests represent the expected performance of Category 4 waste rock.

A feature of the intermediate pH sub-group was orders-of-magnitude increases in copper, nickel, cadmium, manganese, cobalt and zinc leaching shown by the tests. This began to be apparent in the first year of testing and was reflected in the RS74 (Draft 01) deterministic modeling as 95th percentile rates, and it has become clearer as testing has progressed. Increasing sulfate release is not necessarily linked to decrease in pH. Three of the five samples in this sub-group have shown increase in sulfate release as metal release has accelerated though the increases were within an order of magnitude. Samples in the intermediate pH sub-group have sulfur concentrations greater than or equal to 0.2% and therefore correspond to Category 3 and 4 Duluth Complex waste rock and lean ore. As shown in Figure 2, the lowest sulfur concentrations in this sub-group are ultramafic rocks with sulfur concentrations of 0.2% and 0.3%. These two samples were classified as Category 3 based on their Cu/S ratio. Troctolitic rock had sulfur concentrations of more than 0.8%.

The highest pH sub-group showed generally lower metal leaching rates but a number showed slightly increasing nickel leaching trends without pH depression or increase in sulfate release. In several cases, nickel leaching reached a stable rate below 0.001 mg/kg/week compared to stable rates above 0.1 mg/kg/week for some samples in the intermediate pH group. Two samples in this group containing higher sulfur concentrations (0.7 to 0.8%) sulphur show increasing nickel and cobalt (but not copper leaching) accompanied by sulfate leaching. Due to the sulfur concentrations in the samples, it might be expected that these tests will show pH depression and that the accelerating sulfate and metal release is a pre-cursor to these conditions.

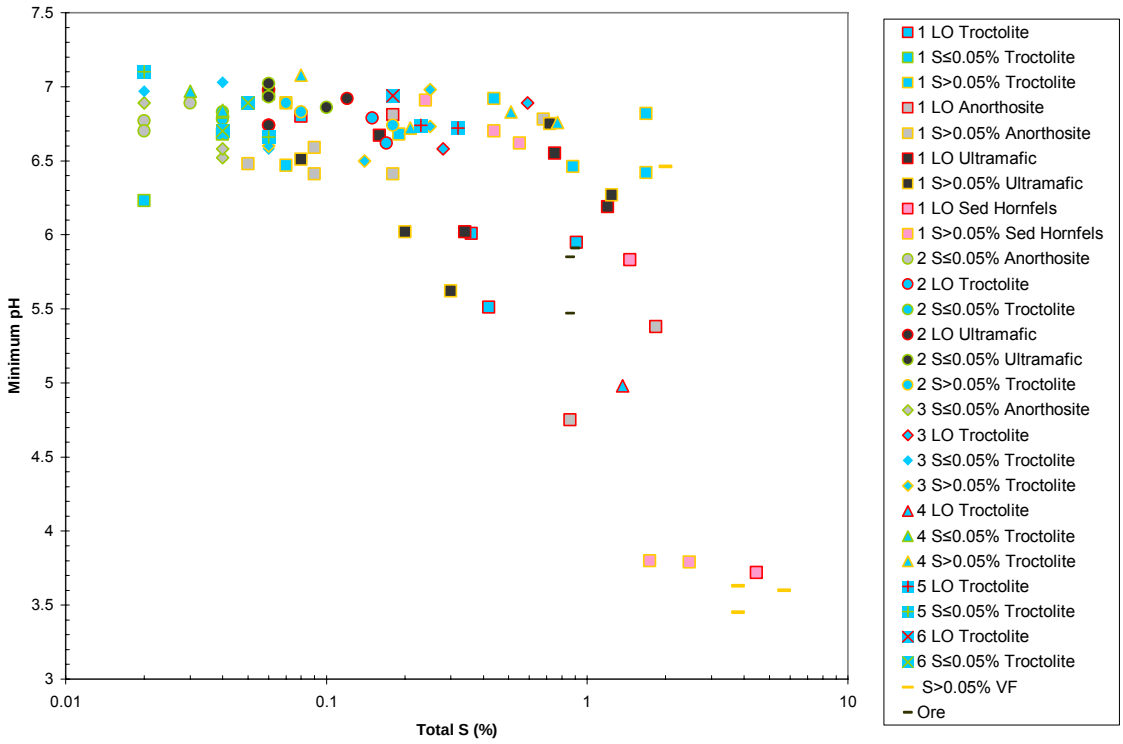


Figure 2: Solids Sulfur Concentrations Compared to Minimum pH for All Sample Groups

3.2.3 Lean Ore Group

Leachate pHs for the lean ore samples fall into four sub-groups (Figure 1) which have become apparent as testing has proceeded beyond the initial year of data:

- pH Steady at approximately 4 – One sample of sedimentary hornfels.
- pH declining below 5 – Three samples of Duluth Complex with sulfur concentrations above 0.8%.
- pH steady near 6 – Four samples of Duluth Complex Rock with sulfur concentratins between 0.3 and 0.9%.
- pH Between 6.5 and 7.8 – Balance of tests.

All tests in the middle two sub-groups have shown accelerated leaching of cobalt and nickel with some also showing accelerated cadmium, manganese and zinc leaching. This trend was apparent in the data used for RS74 (Draft 01) and has continued. Dissolution rates appear to have reached peaks in some cases, and manganese release is declining. Some samples have also shown accelerated copper leaching. However, unlike the waste rock tests, only one of the tests with leachate pH below 5 has shown accelerated release of sulfate.

3.2.4 Ore

The three ore samples have shown very similar leaching results comparable to the third group of lean ore tests. Leachate pHs are currently stable near 6 (Figure 1) and sulfate release has not shown clear trends. Release of cadmium, copper, cobalt, manganese, nickel and zinc all accelerated as pH declined. Except for copper, release rates have stabilized, and for manganese are clearly declining.

3.2.5 Comparison Tests

Comparative test (MDNR reactor with ASTM humidity cells) results were not directly used in the water quality estimates in RS42. These tests have continued to show the same comparative results as reported in RS42:

- The MDNR reactor tests produced lower pH leachates than the ASTM test when pHs of both tests are near neutral. Sulfate concentrations were higher for the humidity cells but nickel concentrations were greater for the MDNR reactor tests.
- Acidic leachates on a Virginia Formation sample had lower pH for the ASTM test than the MDNR reactor. Metal and sulfate release were both greater for the ASTM test compared to the MDNR reactor.

As discussed in RS42, the difference in results is interpreted to be due to the difference in water to rock ratios in the tests.

3.2.6 Duplicate Testwork

All seven duplicate humidity tests have continued. Six of these duplicates showed good reproducibility of the majority of parameters. The lean ore duplicate test showed good reproducibility for pH, conductivity and total ion concentrations until about week 110. Prior to this, there was a shift in cation mix (calcium, magnesium and sodium) that was different for the two samples. At week 110, one sample showed markedly more dilute leachate. At the same time, the laboratory observed the leachate drained rapidly from this sample and recommended that both samples be gently agitated to encourage even distribution of flow. The result was that concentrations increased in the sample with relatively dilute leachate and then became similar to the other sample in the pair. The reason for this behavior is not known. Rate of leachate drainage appears to be an important consideration and this is being monitored periodically for other tests.

3.3 Interpretation

3.3.1 Effect on Sulfur Management Criteria

Page 66 of RS42 described the development of sulfur management criteria for the separation of Categories 1/2 and 3. The main inputs to the method were the rate of alkalinity generation from weathering of silicates in rocks containing negligible sulfur, and the relationship between sulfur content and oxidation rate. The criteria were calculated for the condition when oxidation rate was balanced by alkalinity release rate and hence to the critical sulfur content.

Alkalinity release was estimated based on six humidity cells. The continuing alkalinity trend is shown in Figure 3. The dashed line at 48 weeks is the time when the data were interpreted for RS42 (Draft 01). The continuing trend indicates that alkalinity leaching has stabilized for the four cells with lowest alkalinity release. As the method was based on the lower alkalinity values, the input alkalinity release used in the calculation would not be different with the more recent data.

Figure 4 compares average sulfate release for the earlier and more recent data. The diagonal line indicates equal values. As shown, average sulfate release rates at lower releases have decreased slightly. At higher levels, the difference is not apparent. As the sulfur criterion was derived from the lower rates, the comparison indicates a lower sulfate release rate would be input into the calculations.

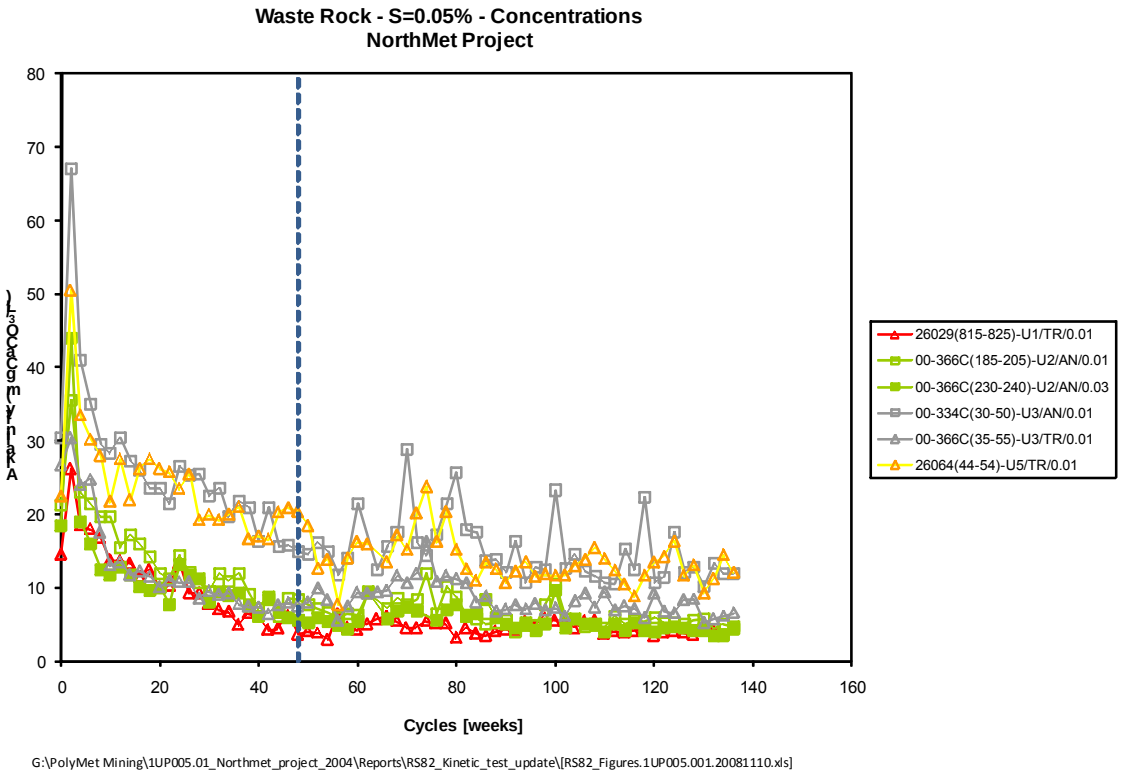


Figure 3: Alkalinity Concentrations in Leachates from Low Sulfur Cells

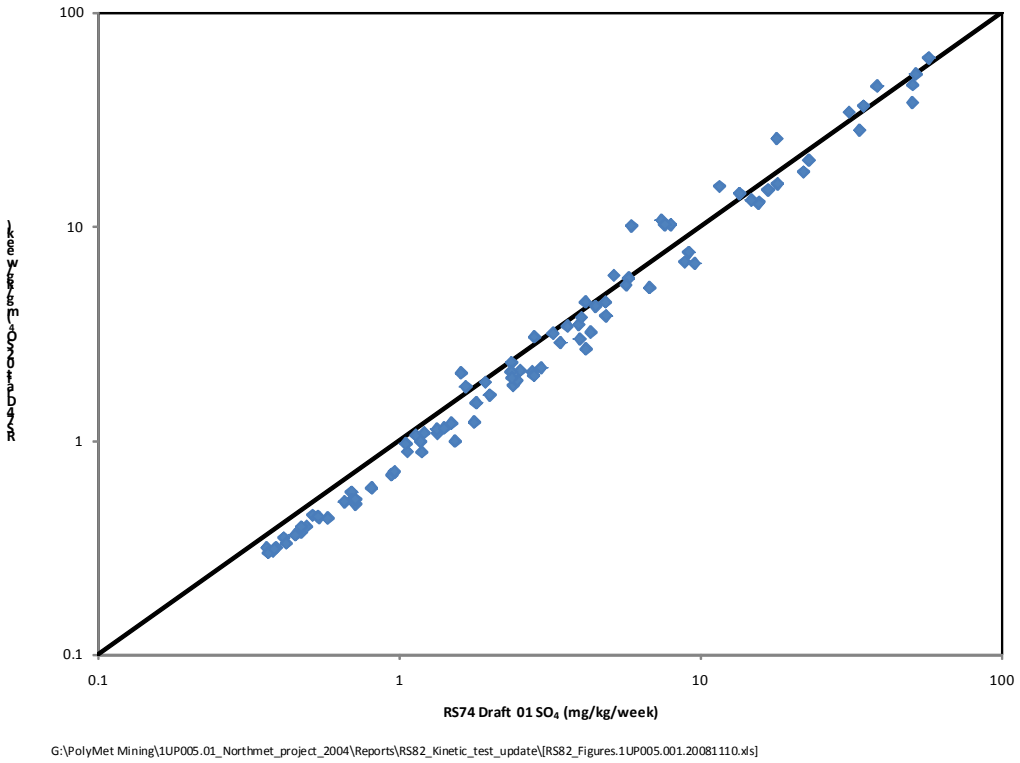


Figure 4: Comparison of Average Sulfate Release Rates for Data Used in RS74 Draft 01 and 02

Overall, the more recent data would result in a slightly higher sulfur criterion than calculated for RS42 Draft 01 because the alkalinity rate has not changed but oxidation rates have decreased slightly. This indicates that a slightly higher sulfur concentration would define the separation between Categories 1/2 and 3 as defined for the Proposed Action.

3.3.2 Effect on Dissolution Rates Used to Estimate Water Quality

The data ranges shown in Table 1 have been used to estimate 95th percentile release rates for input into water chemistry from waste rock and lean ore stockpiles (Table 2). The table shows original rates as provided in RS42 for the Proposed Action and used in RS74 (Draft 01). Revised rates were calculated for use in RS74 (Draft 02), based on the extended data range shown in Table 1. Subsequent checks of the calculations have been used to calculate updated rates for the Proposed Action, and for the Reasonable Alternative 1 using the data range shown for RS74 (Draft 02) in Table 1. Metal release rates are greater than reported in RS42 using the more recent data due to the observed increase in release indicated by the humidity cells.

For the Proposed Action, the differences between the rates in "RS74 (Draft 02)" and the "Update" (i.e. this document) are small changes (almost entirely decreases) in the rates applied to Category 1/2, Category 3 and Category 4 (Virginia Formation), and increases in Category 4. For the first group, this results in minor decreases to the rates applied to majority (>99%) of waste rock. The increase in Category 4 rates applies only to the small Category 4 Lean Ore Surge Pile not the Category 4 Waste Rock Stockpile composed mainly of Virginia Formation. In general, all these differences (decreases or increases) will have negligible effects on the majority of water chemistry estimates due to the consideration of solubility limits on actual release.

For the Reasonable Alternative 1 the differences between the rates in "RS74 (Draft 02)" and the "Update" are small decreases in the rates applied to Category 2 and small increases in the rates applied to Category 3 compared to the Proposed Action due to the re-classification of rock with higher sulfur concentrations from Category 2 to 3.

Further discussion on the implications of these increases to the water quality estimates is provided in Section 3.5.

Table 2: Weighted Average 95th Percentile Dissolution Rates (in mg/kg/week) for Waste Rock and Lean Ore Stockpiles for Proposed Project and RA1

Waste Category	Total Acidity	Alkalinity	F	Cl	SO_4	Al	Sb	As	Ba	Be	B	Cd	Ca	Cr	Co	Cu	Fe	Pb	Mn	Hg	Mo	Ni	Se	Ag	Tl	V	Zn
Proposed Action (RS74 Version 01)																											
2	1.3	4.4	0.027	0.11	2.3	0.087	-	0.0044	0.0088	-	0.0039	0.00002	0.98	0.00011	0.000053	0.00085	0.015	0.000063	0.00096	0.0000045	0.000027	0.00024	0.00011	0.000025	0.00001	-	0.0013
3	1.4	5.3	0.024	0.14	11	0.052	-	0.0068	0.0081	-	0.0054	0.000028	2.8	0.00011	0.0059	0.0084	0.011	0.000069	0.023	0.0000045	0.000043	0.07	0.0002	0.000031	0.00001	-	0.004
4	1.4	11	0.03	0.11	11	0.063	-	0.0044	0.0075	-	0.016	0.000021	1.8	0.00013	0.000086	0.00078	0.03	0.000059	0.0033	0.0000045	0.00014	0.0009	0.00042	0.000096	0.00001	-	0.00069
4 - Virginia	24	0.075	0.034	0.11	50	0.37	-	0.00071	0.0052	-	0.021	0.0032	1.6	0.00012	0.039	0.0048	9.5	0.0011	0.12	0.000011	0.000026	0.56	0.0006	0.000029	0.000012	-	0.6
Ore	1.4	5.2	0.041	0.11	23	0.017	-	0.0014	0.0059	-	0.011	0.000022	7.3	0.0001	0.0028	0.0053	0.0074	0.000076	0.022	0.0000098	0.000034	0.057	0.00012	0.000025	0.00001	-	0.0021
Proposed Action (RS74 Version 02)																											
2	1.4	3.3	0.025	0.1	1.9	0.063	0.0015	0.0033	0.0056	0.00011	0.0027	0.00002	0.85	0.0001	0.000053	0.0009	0.0095	0.000041	0.0012	0.0000045	0.000042	0.00034	0.0001	0.000026	0.0000099	0.0013	0.0012
3	1.8	3.7	0.024	0.12	9.9	0.036	0.0015	0.0034	0.0085	0.00011	0.0034	0.000064	2.2	0.0001	0.017	0.098	0.04	0.000056	0.028	0.0000045	0.000036	0.21	0.00027	0.0001	0.000011	0.00079	0.011
4	1.5	7.2	0.031	0.1	11	0.042	0.0015	0.0024	0.0054	0.000099	0.02	0.000029	2.6	0.00012	0.00064	0.001	0.019	0.000044	0.0085	0.0000045	0.000088	0.0077	0.00042	0.000062	0.00001	0.0051	0.0013
4 - Virginia	26	0.17	0.033	0.1	50	0.5	0.0002	0.00054	0.0042	0.00057	0.016	0.0032	1.1	0.00012	0.039	0.007	9.7	0.0006	0.088	0.0000099	0.000024	0.48	0.00051	0.000031	0.000012	0.000097	0.51
Ore	2.1	2.8	0.031	0.11	20	0.0081	0.0015	0.00077	0.0063	0.000098	0.011	0.00007	5.5	0.000098	0.037	0.059	0.006	0.000054	0.086	0.0000098	0.000029	0.62	0.00023	0.000025	0.000014	0.00011	0.015
Proposed Action (Update)																											
2	1.4	3.4	0.025	0.1	1.9	0.063	0.0025	0.0026	0.0056	0.00011	0.0027	0.00002	1.9	0.0001	0.000053	0.0009	0.0093	0.000031	0.0011	0.0000045	0.000042	0.00034	0.0001	0.000026	0.0000098	0.0013	0.0012
3	1.8	3.9	0.024	0.1	8.4	0.037	0.0025	0.0019	0.0088	0.00011	0.003	0.000066	3.7	0.000097	0.017	0.098	0.01	0.000053	0.028	0.0000045	0.000032	0.21	0.00024	0.000041	0.000011	0.00029	0.011
4	1.9	2.8	0.03	0.14	17	0.022	0.0025	0.0054	0.0064	0.00012	0.012	0.000043	5.2	0.00011	0.019	0.008	0.1	0.00019	0.049	0.0000045	0.000045	0.14	0.00033	0.00015	0.000011	0.0012	0.0073
4 - Virginia	26	0.16	0.033	0.1	50	0.51	0.002	0.00054	0.0038	0.00056	0.015	0.0031	2.2	0.00012	0.039	0.0071	9.7	0.0006	0.087	0.0000099	0.000024	0.47	0.0005	0.000026	0.000012	0.000097	0.49
Ore	2.1	1.9	0.031	0.11	20	0.0081	0.0025	0.00077	0.0063	0.000098	0.011	0.00007	5.5	0.000098	0.037	0.059	0.006	0.000054	0.086	0.0000098	0.000029	0.62	0.00023	0.000025	0.000014	0.00011	0.015
Reasonable Alternative 1																											
2	1.4	3.3	0.025	0.1	1.3	0.063	0.0025	0.00035	0.0056	0.00011	0.0013	0.00002	1.8	0.0001	0.000053	0.00088	0.0067	0.000031	0.0008	0.0000045	0.000041	0.00034	0.000098	0.000024	0.0000098	0.00021	0.0012
3	1.8	3.5	0.025	0.1	8.3	0.043	0.0021	0.0033	0.0086	0.00011	0.0046	0.000065	3.6	0.0001	0.017	0.096	0.012	0.000052	0.027	0.0000045	0.000032	0.21	0.00024	0.000042	0.000011	0.0016	0.011
4	1.9	2.8	0.03	0.14	17	0.022	0.0021	0.0054	0.0064	0.00012	0.012	0.000043	5.2	0.00011	0.019	0.008	0.1	0.00019	0.049	0.0000045	0.000045	0.14	0.00033	0.00015	0.000011	0.0012	0.0073
4 - Virginia	26	0.16	0.033	0.1	50	0.51	0.002	0.00054	0.0038	0.00056	0.015	0.0031	2.2	0.00012	0.039	0.0071	9.7	0.0006	0.087	0.0000099	0.000024	0.47	0.0005	0.000026	0.000012	0.000097	0.49
Ore	2.1	1.9	0.031	0.11	20	0.0081	0.0025	0.00077	0.0063	0.000098	0.011	0.00007	5.5	0.000098	0.037	0.059	0.006	0.000054	0.086	0.0000098	0.000029	0.62	0.00023	0.000025	0.000014	0.00011	0.015

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3.4 Comparison with MDNR Testwork Trends

The ongoing waste rock and lean ore tests show similar trends to the first two or three years of testing in the MDNR's long term tests on Duluth Complex rocks. The main features of this early stage are:

- Decreasing pH (greater for higher sulfur concentrations).
- Decreasing or stable sulfate release rates.
- Increasing cobalt, nickel and zinc release (for higher sulfur concentrations).
- Beginning of increases in copper release.

In the MDNR's testwork, peak cobalt, nickel and zinc concentrations seen at this stage were the highest at any point in the testwork. Peak copper concentrations were observed later (5 years) as pH dropped to between 3 and 4. As leachates from the PolyMet tests are only recently showing pHs below 5, copper concentrations have only shown initial increases.

3.5 Implications for Water Quality Estimates

95th percentile dissolution rates in Table 2 have been used to produce new water quality predictions for waste rock and lean ore stockpiles and pit walls for RS74 (Draft 02).

The method used to estimate stockpile water chemistry using these dissolution rates was provided in Section 8 of RS42 (Draft 01). In summary, the method used weighted average 95th percentile dissolution rates indicated by the NorthMet Project humidity cells as available at the time (Table 1). As the testwork was at an early stage, the dissolution rates were expected to increase for the Category 3 and 4 waste rock types based on the MDNR's long term testwork. Therefore, the MDNR's data were used to estimate the increase in oxidation rates that would be expected to occur as the testwork proceeded and pH decreased. The factor by which rates increased in the MDNR testwork was applied to the early rates observed for the NorthMet Project tests to estimate rates at lower pHs.

This same approach has been carried through to develop the inputs used in RS74 Draft 02. However, as presented above, many of the higher sulfur samples are now showing behavior transitional to the longer term effects seen in the MDNR tests particularly with respect to increasing leaching of elements that become more mobile as pH decreases below 7 (for example, cadmium, cobalt, manganese, nickel and zinc). Therefore, the 95th percentile rates from the NorthMet Project tests for this period reflect both initial and transitional conditions and the application of the factors to represent decreases in pH may have resulted in over-stating release rates for these elements. As release rates increase, this result will continue. This concern does not apply to copper, which is not expected to show increases in release until leachate pH drops further. The application of the factors based on MDNR testwork to copper does not currently result in over-statement of later release rates though this may occur as the tests proceed and copper release rates increase.

4 Tailings Kinetic Tests

4.1 Status of Program

Description of the tailings kinetic test program, including characteristics of the samples being tested can be found in RS46 (Draft 01). The test program on NorthMet Project tailings consists of conventional humidity cells with parallel tests using the MDNR reactor configuration on bulk tailings and tailings size fractions. A separate program is evaluating the interaction of NorthMet Project bulk tailings leachate with the LTVSMC tailings.

All tests continue to operate as described in RS46, however, the following issue occurred for the P2 and P3 bulk tailings samples in humidity cells. The filter cloth in the cells was found to be partially damaged in November 2006 and repairs were performed. Following re-start of the test, arsenic concentrations increased from about 0.003 mg/L before the repair to an immediate peak of 0.1 mg/L. Arsenic concentrations then declined reaching pre-repair levels in April 2008.

The filter cloth in the cells containing all four bulk samples was found to be partially damaged in April and May 2008 allowing the leachate to drain too quickly. The samples were transferred to new cells. Following these repairs, leachate chemistry re-stabilized quickly returning to pre-repair levels shortly within a few cycles. The reason for the increase in arsenic when the cells were repaired in November 2006 is unknown. Arsenic concentrations from P2 and P3 humidity cell tests have not been used in subsequent calculations. Leachate data from the MDNR Reactor tests on the samples were used instead. These tests showed a uniform trend consistent with other tests.

All other tests continue to operate smoothly.

All tests originally started are continuing. None of the tests have been terminated and the analytical program is unchanged from that developed in consultation with the MDNR.

4.2 Description of Results

4.2.1 NorthMet Project Tailings

All humidity cell tests showed an initial decline in pH (Figure 5) which was apparent in the data reported in RS46. The decline was greatest for the two coarser particles size fractions and least for the -200 mesh fraction samples. The trend for bulk tailings was between the coarse and fine fractions. The range in leachate pHs following the main initial decline was 6.0 to 7.8. Leachate pHs remain somewhat erratic but no decline to lower pHs has been apparent. Variation reflects whether copper sulfate was used in the pilot mineral processing circuit to improve recovery of sulfide minerals during flotation. Higher pH leachates are shown by samples with lower sulfur contents produced by the use of copper sulfate.

Sulfate release has generally declined as the testwork has continued (Figure 5). Major ions have followed sulfate and are generally decreasing. Alkalinity leaching from the two coarser particle sizes declined sharply and then stabilized, with the stable period starting at the end of the RS46 reporting period. In contrast, alkalinity from the -200 mesh samples has continued to decline. Bulk tailings with higher sulfur contents have shown the similar trends to the two coarser particle sizes whereas bulk tailings with lower sulfur contents were closer to the -200 mesh samples.

The main distinctive feature of leachates has been the increase in leaching of cobalt, manganese and nickel from the two coarse tailings fractions. Zinc also shows a similar though more erratic trend. This positive trend was apparent in the RS46 Draft 01 reporting period but has been confirmed by continued monitoring. The bulk tailings also showed increasing nickel but at an order of magnitude below the comparable coarse fraction samples. The -200 mesh samples have remained relatively stable. The degree to which these elements leach is related to sulfur content. The two coarse fraction samples produced without using copper sulfate have shown greatest nickel leaching (at about 0.1 mg/kg/week) whereas the same fractions produced using copper sulfate have shown peak leaching near 0.02 mg/kg/week.

The coarse fractions have shown sustained elevated nickel leaching for about a year. One sample (P2 -100+200 mesh) is showing decreasing nickel release.

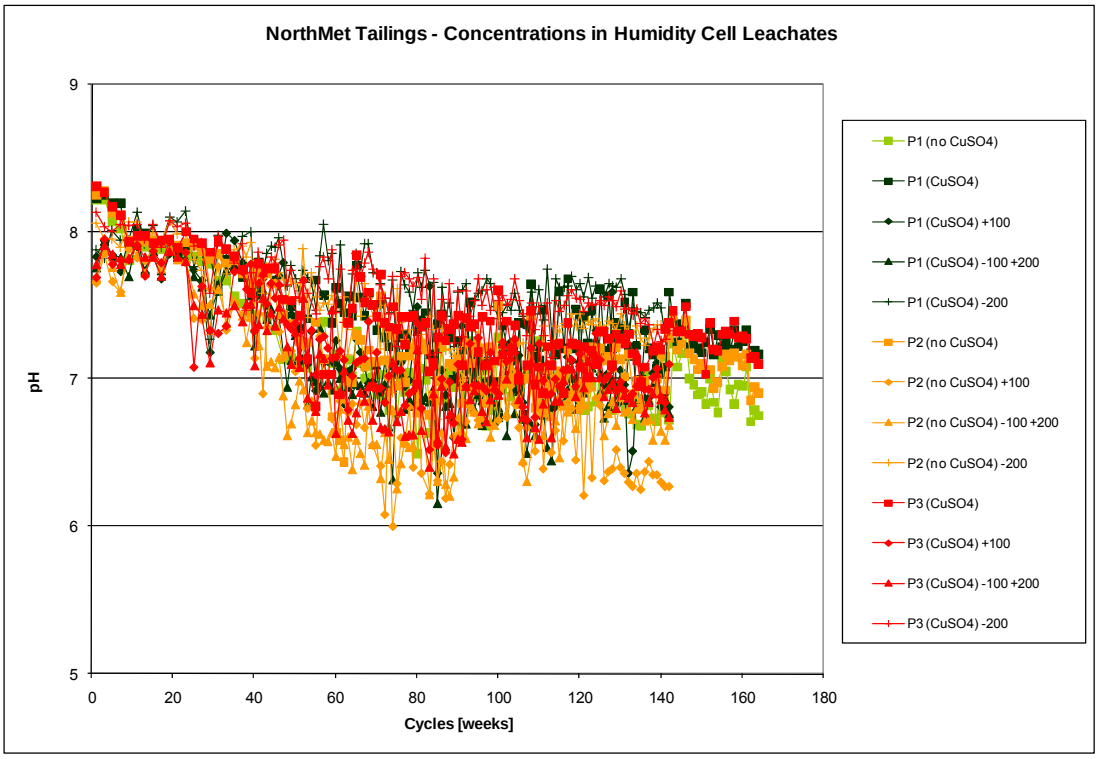
Samples tested using the MDNR reactor configuration have shown similar trends with respect to pH and release of cobalt and nickel. The sample of P1 (with copper sulfate) -200 mesh tailings showed an increase in pH, alkalinity and base cations (mainly calcium) at about 110 cycles. Likewise increases in arsenic and vanadium were observed. About 140 cycles, alkalinity had returned to previous levels. No changes in test conditions were identified to account for these trends.

4.2.2 LTVSMC Tailings

These experiments have largely shown the same effects as documented in RS46, which include:

- More basic leachates from LTVSMC tailings when compared to the PolyMet tailings.
- High alkalinity for LTVSMC tailings indicating partial pressures of CO_2 exceeding atmospheric ($10^{-3.5}$ atm).
- Stable leaching of alkalinity, calcium and magnesium for both NorthMet and LTVSMC tailings but with highest concentrations for the latter.
- Steady sulfate and arsenic release for NorthMet Project tailings but declining sulfate and arsenic for LTVSMC tailings.
- Strong removal of vanadium leached from NorthMet Project tailings by LTVSMC tailings.
- Removal of nickel leached from NorthMet Project tailings by LTVSMC tailings in three of the four tests (P1 to LTVSMC fine, P3 to LTVSMC coarse and fine).

A new finding as the tests have continued is removal of arsenic and manganese from NorthMet Project tailings by the LTVSMC tailings for all tests. These trends have become apparent as leaching of arsenic and manganese from the LTVSMC tailings themselves have decreased.



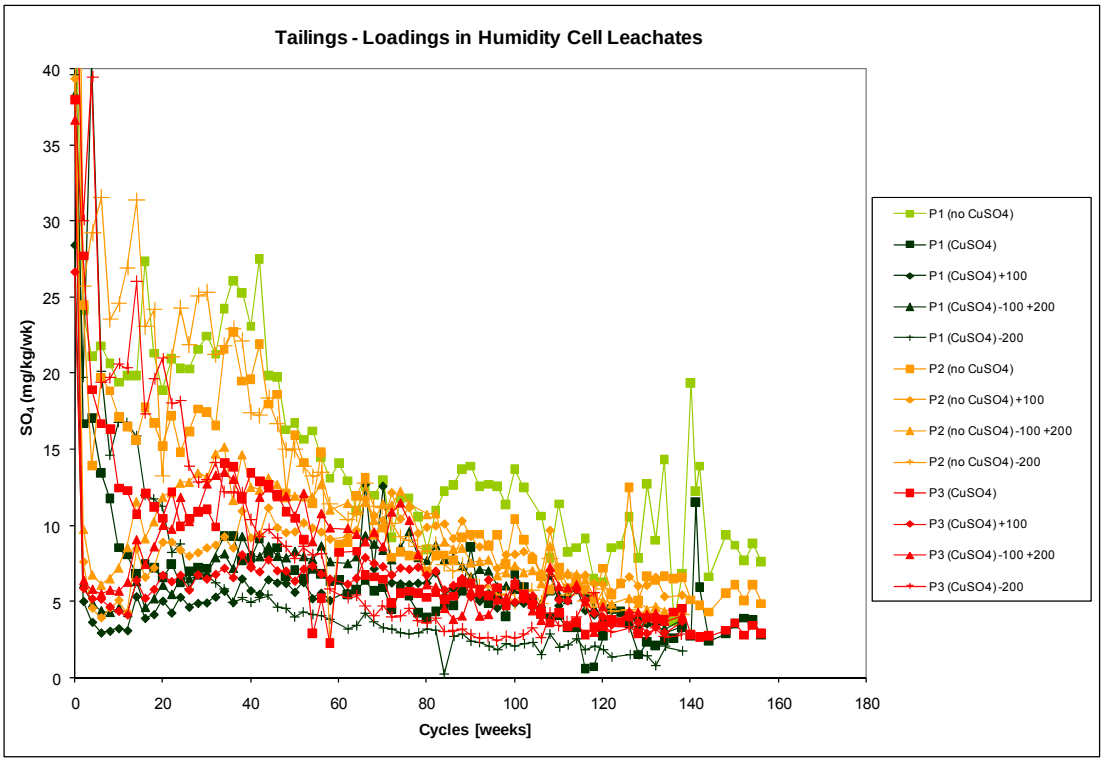


Figure 5: Leachate pH and Sulfate Loadings for Tailings Humidity Cells

4.3 Interpretation

4.3.1 Effect on Sulfur Management Criterion

Sulfur management criteria were developed in RS46 (Section 6.3.1) using the data presented in RS46. These criteria were up dated using the recent data and presented in a memorandum (dated June 18, 2008) in response to an Agency Geochemical Model Validation workplan request for RS46. This re-calculation showed the critical sulfur concentrations for bulk and coarse tailings were between 0.14% and 0.17%. These concentrations were slightly lower than those presented in RS46.

4.3.2 Effect on Molar Release Rate Ratios

Data obtained to week 95 was used to calculate average release rates used in Draft 2 of RS74. The increase in release of nickel and cobalt without a matching increase in sulfate release resulted in increases in the molar release ratios for these parameters for the coarse tailings compared to the ratios used in RS74 Draft 01 (Table 3).

Table 3: Comparison of Molar Release Ratios for Draft 01 and Draft 02 of RS74

Parameter	Coarse Tailings			Fine Tailings		
	Draft 01	Draft 02	Draft01/ Draft02	Draft 01	Draft 02	Draft01/ Draft02
F	-	$1.51x10^{-2}$	-	-	$2.57x10^{-2}$	-
SO_4	1	1	1	1	1	1
Al	-	$1.81x10^{-2}$	-	-	$5.18x10^{-2}$	-
Sb	$4.00x10^{-5}$	$2.53x10^{-5}$	0.6	$4.00x10^{-5}$	$7.17x10^{-5}$	1.8
As	$2.00x10^{-4}$	$1.52x10^{-4}$	0.8	$2.00x10^{-4}$	$6.13x10^{-4}$	3.1
Ba	-	$7.68x10^{-5}$	-	-	$9.04x10^{-5}$	-
Be	$1.43x10^{-4}$	$1.28x10^{-4}$	0.9	$1.43x10^{-4}$	$2.11x10^{-4}$	1.5
B	$5.97x10^{-4}$	$1.83x10^{-3}$	3.1	$5.97x10^{-4}$	$4.52x10^{-3}$	7.6
Cd	$2.30x10^{-6}$	$2.51x10^{-6}$	1.1	$2.30x10^{-6}$	$3.70x10^{-6}$	1.6
Cr	-	$2.73x10^{-5}$	-	-	$3.87x10^{-5}$	-
Co	$3.90x10^{-5}$	$1.33x10^{-4}$	3.4	$3.90x10^{-5}$	$1.61x10^{-5}$	0.4
Cu	$1.50x10^{-4}$	$2.07x10^{-4}$	1.4	$1.50x10^{-4}$	$4.10x10^{-4}$	2.7
Fe	-	$1.17x10^{-3}$	-	-	$8.52x10^{-3}$	-
Pb	$1.56x10^{-6}$	$1.40x10^{-6}$	0.9	$1.56x10^{-6}$	$2.93x10^{-6}$	1.9
Mn	-	$4.08x10^{-3}$	-	-	$9.65x10^{-4}$	-
Mo	-	$3.76x10^{-6}$	-	-	$2.15x10^{-5}$	-
Ni	$6.60x10^{-4}$	$2.58x10^{-3}$	3.9	$6.60x10^{-4}$	$1.18x10^{-4}$	0.2
Se	$1.63x10^{-5}$	$1.79x10^{-5}$	1.1	$1.63x10^{-5}$	$2.69x10^{-5}$	1.7
Ag	$2.99x10^{-6}$	$2.68x10^{-6}$	0.9	$2.99x10^{-6}$	$4.40x10^{-6}$	1.5
Na	$1.10x10^{-1}$	$9.95x10^{-2}$	0.9	$1.10x10^{-1}$	$3.04x10^{-1}$	2.8
Tl	$6.31x10^{-7}$	$6.25x10^{-7}$	1.0	$6.31x10^{-7}$	$9.28x10^{-7}$	1.5
Sn	-	$5.29x10^{-6}$	-	-	$4.72x10^{-5}$	-
V	-	$2.60x10^{-5}$	-	-	$2.92x10^{-4}$	-
Zn	$1.20x10^{-3}$	$4.59x10^{-4}$	0.4	$1.20x10^{-3}$	$2.37x10^{-4}$	0.2

4.4 Comparison with MDNR Testwork Trends

4.4.1 ASTM Humidity Cells

Comparison of data for the NorthMet Project tailings samples tested in humidity cells with samples of Babbitt Deposit bulk tailings tested in ASTM humidity cells by MDNR showed some similar trends:

- Leachates for both Babbitt Deposit and NorthMet Project showed declining pH from initial values near 8 to values near 7. The decline was more rapid for the NorthMet Project but appears to have stabilized or be increasing slightly.
- Both datasets showed overall declining sulfate release trends. The NorthMet Project tests showed a modest increase after about 30 weeks.
- Similar nickel release rates with a tendency to increasing nickel release as the test proceeded. The increase occurred earlier for the NorthMet Project tailing samples.

4.4.2 MDNR Reactor Tests

For the MDNR reactor tests, strong differences were apparent for the Babbitt Deposit and NorthMet Project tailings.

Leachate pH from the MDNR's experiments with Babbitt Deposit tailings showed a rapid 1 pH unit decrease less than 1 year after starting followed by relatively stable pH near 6.5. The NorthMet Project tailings also showed decreasing pH but the decrease was more uniform. Both tests have shown a tendency to increasing pH recently. Differences in nickel leaching were very pronounced. The Babbitt Deposit tests showed a two order-of-magnitude increase in nickel release beginning after about 6 months, peaking after a year and then declining. The NorthMet Project bulk tailings samples have not shown the same increase though nickel release is on a slow increasing trend. The reason for the differences in results is not known. The results for the MDNR tests more closely resemble those for the NorthMet Project coarser particle size tests. This suggests that the Babbitt Project samples may have been coarser than the NorthMet Project samples making the Babbitt Project samples more susceptible to decrease in pH induced by the low pH of the deionized water used in the test.

4.5 Implications for Water Quality Estimates

The oxidation rates and molar rate ratios have been incorporated into water chemistry estimates in RS74 Draft 02.

5 Hydrometallurgical Residue Kinetic Tests

5.1 Status of Program

Most of program initiated in February 2006 (as described in RS65) is continuing. The exception is that five of the shake flask experiments had to be stopped. Three tests were stopped after 89 leach cycles because the samples accidentally become cross-contaminated during sampling when the solids were returned to the shake flasks:

- #3 - Leach Residue from concentrate produced without copper sulfate
- #7- Fe /Al Residue
- #8 - Leach residue from concentrate produced with copper sulfate

The following tests were stopped recently because the test charge had been completely removed by leaching:

- #1 - Raffinate Neutralization Residue
- #9 – Gypsum

Continuing shake flask tests are Combined Residues (with and without gypsum) and magnesium residue. Tests in MDNR reactors and humidity cell configurations are all continuing.

5.2 Description of Results

5.2.1 Shake Flask Tests

A strong control on leachate chemistry was the complete leaching of gypsum which resulted in decrease in sulfate concentrations. All of the non-leach residues have now shown this result (Figure 6). In the case of the Mg residue this resulted in sharp increase in pH to near 10 (Figure 6). The combined residue with gypsum has shown a downward trend in pH recently. In contrast, the combined residue that did not contain the gypsum residue showed increasing pH after gypsum in the other residues was depleted.

Changes in pHs and sulfate resulted in some changes in trace element leaching. Increasing pH resulted in increases in leaching of aluminum and boron. Increases in cadmium, cobalt, copper, manganese, nickel and zinc were apparent as pH decreased to 4.7 for the combined residue. Increases in barium leaching occurred as sulfate decreased.

5.2.2 MDNR Reactor Tests

The MDNR reactor tests have shown similar trends to the shake flasks though only the combined residues (no gypsum) and magnesium residue have shown complete removal of sulfate. Leachate pHs have either remained stable or increased. Strongest increases were shown for the magnesium residue.

As a result of the lack of major pH changes, trace element leaching as largely decreased or remained stable.

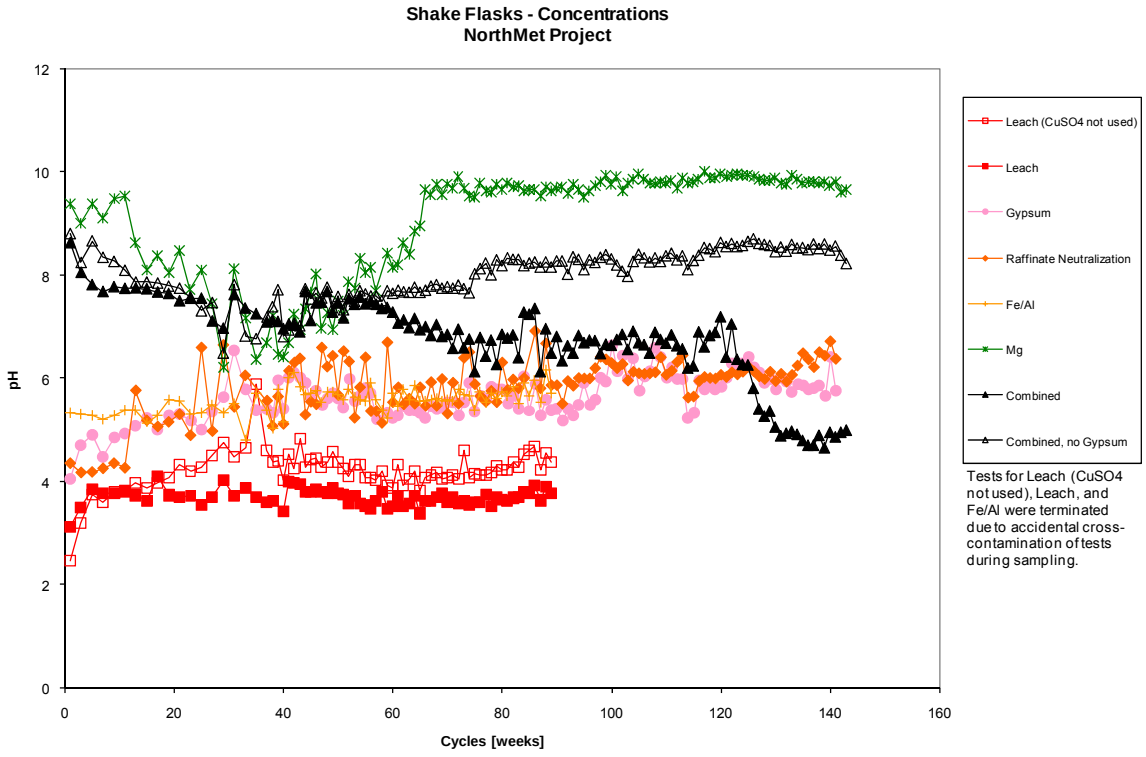
5.2.3 Humidity Cell Tests

The humidity cell tests showed the most stable pH trends of the three types with either stable or increasing trends. Leachate pHs for the magnesium residues were very similar to the combined residues after two years of testing. For these tests, only the leach residues showed decrease in sulfate. Magnesium residues continued to leach at sulfate concentrations around 5000 mg/L. Other residues leached at stable levels of near 1500 mg/L.

Trace element concentrations remained stable or decreased.

5.3 Discussion

The testwork demonstrates common results namely that leachate chemistry is relatively stable until gypsum is completely leached which results in decreasing sulfate concentrations which in turn may trigger changes in pH and trace element leaching. Changes in pH occur because removal of gypsum may allow other minerals to start to dissolve. Examples of minerals that may show accelerated dissolution after removal of gypsum include calcium hydroxide, calcium carbonate and barium sulfate. The three different tests show different results because the liquid to solid ratio varies from 3:1 (shake flask) to 1:2 (humidity cell) with the three different test protocols. Complete leaching of gypsum occurs first in the shake flask tests, then MDNR reactors and finally humidity cells.



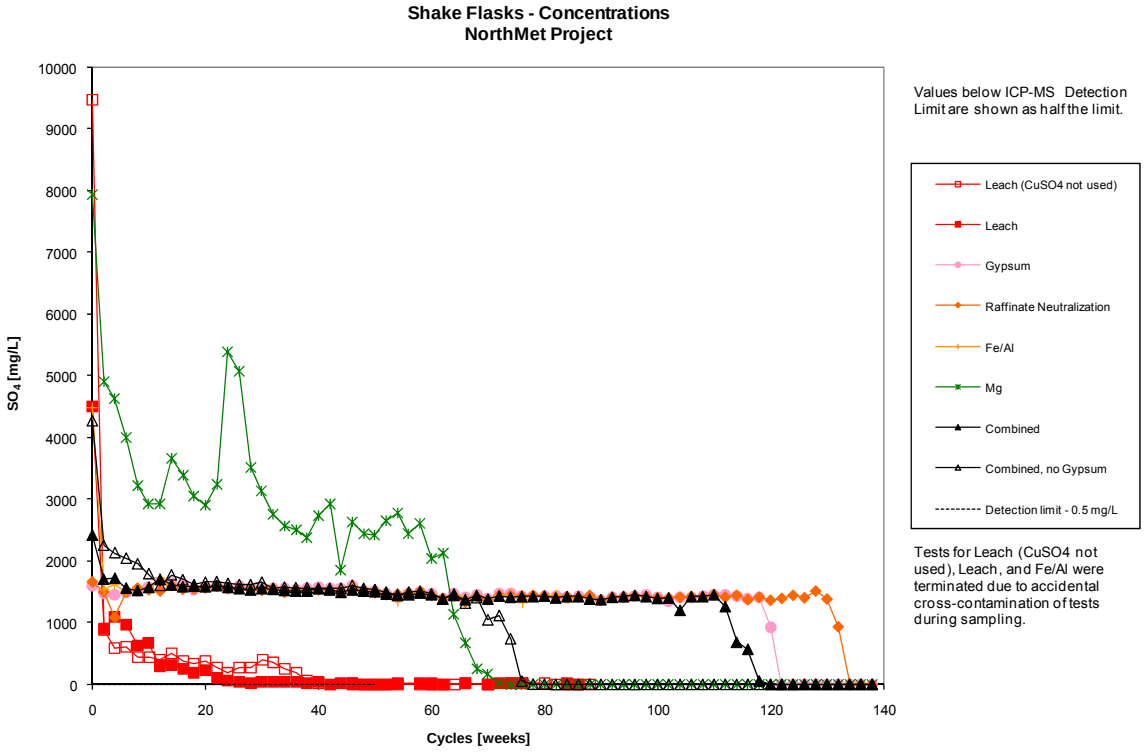


Figure 6: Leachate pH and Sulfate Concentrations for Hydrometallurgical Residues in Sequential Shake Flask Tests

5.4 Implications for Water Quality Estimates

The pore water quality estimates presented in RS65 were based on maximum concentrations observed in the first 36 weeks of testing of humidity cells (dominantly) containing combined residues. Subsequent changes in leachate chemistry are related to the continued depletion of gypsum which in concept could trigger changes in pH. However, the number of pore water volumes flushed in the tests far exceeds pore water displacement that will occur in the lined residue cells; therefore, these longer term effects (such as decrease in sulfate) seen in the tests do not represent site conditions. The main value of the testwork is in understanding the solubility of the residues under different pH conditions.

The ongoing results do not indicate a need to update water chemistry predictions provided in RS65.

6 Conclusions

The following are concluded for the waste rock and lean ore program:

- Three years of NorthMet Project waste rock kinetic tests are showing dissolution trends consistent with longer term testwork conducted by the MDNR. For higher sulfur samples (Category 3 and 4), this is reflected in expected declining pH and related accelerated leaching of several elements including nickel and cobalt that began to be apparent in first year of testing but is now demonstrated.
- The longer term trend in accelerated leaching of copper shown by the MDNR's testwork is not yet readily apparent for the NorthMet Project tests. This latter effect was anticipated and reflected in the RS42 Draft 01 predictions (used in RS74, Draft 01) by applying increases in rates shown by MDNR's long term testwork. The same approach was used to develop predictions that were used as inputs into RS42 Draft 02.
- For lower sulfur samples, alkalinity leaching is stable and oxidation rates have decreased slightly. This indicates that sulfur-based waste rock management criteria do not need to be modified based on new data.

For the tailings:

- Continued monitoring has confirmed accelerated leaching of cobalt and nickel as pH has declined slightly.
- In contrast, bulk tailings have shown only smaller increases in metal leaching similar to results for the MDNR Babbitt Project tailings samples.
- Sulfur-based waste management criteria have been re-evaluated using the more recent data and the criteria have as a result decreased.
- Changes in metal leaching rates have resulted in increased metal to sulfate molar release rates. These have been used for recent oxidation modelling and are reflected in RS74 Draft 02.

Tests on hydrometallurgical residues have undergone far more pore water extractions than will occur under site conditions in the lined disposal cells. The result of extensive flushing is the complete depletion of gypsum which in some cases resulted in pH changes. However, the results do not affect the pore water predictions in RS65 or RS74 Draft 02

Attachment 2-16

MDNR. Recommended Kinetic Test Modifications for PolyMet's NorthMet Project. Memorandum to PolyMet. April 22, 2009 (updated May 11, 2009).

From:	Chev Kellogg
To:	jr.scott@frontiernet.net; Day, Stephen
Cc:	Jamie Schrenzel; Jennifer Engstrom; Kim Lapakko; Michael Olson; Mike Berndt; Paul Eger
Subject:	Preliminary PolyMet kinetic test cell modification attached
Date:	Thursday, April 23, 2009 9:39:00 AM
Attachments:	KT Sample Selection_090422.pdf DNR kinetic test modifications.doc

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Jim,

Attached is the document you received yesterday concerning modifications to the kinetic test cell program. Please keep in mind that the waste rock cell recommendations are preliminary. The final list will be delivered to you by 15 May 2009.

Sincerely,

Chev H. Kellogg, Ph.D.

Minnesota Department of Natural Resources
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phone: 651-259-5428
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Recommended Kinetic Test Modifications for PolyMet's NorthMet Project

Chev Kellogg
Mike Olson
Kim Lapakko

11 May 2009

This memo contains revised recommendations for continued kinetic cell testing (see Table 1) and sampling frequency (see next section). The forty waste rock humidity cells recommended for continuation has been reduced when compared to the preliminary list (Kellogg et al. 2009). The revised list covers the ranges of release rates for all elements analyzed, while focusing on the rock types and waste categories predicted to be most common.

MNDNR agrees with the SRK recommendations to continue all MDNR reactors, tailings humidity cells, and LTV tailings humidity cells. We also agree that hydrometallurgical humidity cells can be discontinued and analyzed as proposed (Day 2009).

The three ore humidity cells should be continued. MNDNR is concerned that exposed ore in pit walls will not be adequately characterized if ore samples are discontinued now. The ore humidity cells show increasing release rates for many elements. These increasing release rates, combined with the fact that ore will remain perpetually exposed in the west pit lake walls, justifies a longer record for the ore humidity cells.

<u>Reduced sampling frequency</u>:

As stated in the earlier memo (Kellogg et al. 2009), all waste rock, ore, tailings, and LTV tailings humidity cells can immediately be sampled at the reduced frequency proposed by SRK (Day 2009).

Response to SRK proposal on modified sampling frequency in response to pH change:

MNDNR has an alternative suggestion to that proposed by SRK (Corriveau 2009) that will reduce sample analysis while giving a more complete representation of release rates during deviations from typical pH.

To quantify the large changes in release rates expected for rapid changes in pH, MNDNR agrees with the SRK recommendation for a pH criterion of greater than a pH 0.4 unit departure from stable average conditions (Corriveau 2009), after the definition of "stable average conditions" is agreed to by MNDNR. To define stable average pH, MNDNR suggests preparation of a memo including the stable average pH for each cell in Table 1, as well as the time period used to define this pH. MNDNR can then evaluate each stable pH. If PolyMet determines that a change in a sample's stable average pH is required, a request to define a new stable pH can be sent to MNDNR for verification.

In contrast to the SRK recommendations of bi-weekly sampling, MNDNR proposes that if leachate deviates from stable average conditions by $\geq$ 0.4 pH, then a composite sample of leachate from the most recent four weeks (week of pH exceedance plus prior three weeks) be analyzed to determine elemental release rates. SRK will design a method for compositing samples that retains some of the weekly leachate in case a need to analyze a specific week arises. This method will give an average release rate for the 4-week period including the pH deviation from average conditions, while reducing the number of samples analyzed. This composite method will be used during scheduled elemental analyses if the pH deviates from stable average conditions by $\geq$ 0.4 pH. The purpose of preferring a 4-week composite sample to biweekly sampling is to ensure that any increase in elemental release is quantified for accurate determination of acidification factors for future uncertainty analyses.

Hydrometallurgical test cells can be discontinued and analyzed as proposed (Day 2009). If there is any untested hydrometallurgical residue from the pilot plant run, a plan should be developed to test these in a way that is more representative of the conditions they will experience in the hydrometallurgical cells during full-scale operations at the site.

Rationale for MNDNR test cell recommendations:

The waste rock humidity cell test modifications proposed by MNDNR differ from those proposed by SRK. SRK's recommended modifications (Day 2009) were based primarily on the range of sulfur contents and waste rock category. MNDNR recommendations initially took sulfate, Ni and As into account for each waste rock category. Nickel and arsenic were included because of pronounced changes in solubility of these elements at circumneutral pH. Rock type and unit have significant differences in regressions of sulfate release versus %S (Kellogg 2008), so these factors were also included in decisions about which kinetic tests to continue. A complete list of factors included in the recommendations follows.

Considerations for proposed kinetic test modifications from MN DNR (roughly in order of importance to revised decisions):

1. Waste rock category
2. Ranges of typical sulfate release rates in each waste rock category (see Figure 1 for example from Cat 1)
3. Nickel release behavior (increasing, stable, stable after increase; see Figures 2-4 for examples)
4. Arsenic release rates (only three samples had higher release rates; humidity cells 5, 6, and 7 are all recommended for continuation)
5. Rock type (more cells retained in rock types predicted to be majority of waste rock; some ultramafic retained to help prediction of unit 7 which has no test cells)
6. Rock unit (units 4-6 were underrepresented initially so we recommend retaining most of these; also aids in prediction of unit 7)
7. Test cells with release rates exceeding 95% confidence intervals of elements (from RS 82, draft 2) predicted to exceed water quality standards (RS 74, A & B)
8. Test cells with release rates exceeding 95% confidence intervals of elements (from RS 82, draft 2) not predicted to exceed water quality standards (RS 74, A & B)

9. Retain humidity cells with corresponding MDNR reactors (if there were duplicate humidity cells, only one of the pair was retained)

Waste rock categories 1 and 2 are estimated to be 83% of waste rock (PolyMet 2008). Because of the dominance of these waste categories, and the possibility of Cat 1 rock remaining on the surface (RA-1), Cat 1 waste rock cells were retained at a higher rate than other waste rock categories. However, since the proposed action retains higher %S waste categories on the surface, it was necessary to retain representative samples from these categories also.

The importance of rock types in predicting leachate was proven statistically by MNDNR (Kellogg 2008). The importance of these differences varies among waste rock categories and elements in leachate. For sulfate release, there is little difference among rock types in Cat 1, while Cat 2 shows greater differences with anorthitic troctolite higher than troctolite (Figure 5). Differences among rock type with Ni release are even more pronounced (Figure 6). Ultramafic generally had higher release rates than most other rock types.

Certain rock types (anorthitic troctolite, augite troctolite, troctolite) were retained preferentially to other rock types due to higher mass of these rock types in waste rock (Day 2006). Several ultramafic samples were retained to help account for the main difference between units 4-6 and unit 7. Unit 7, although approximately 5.5% of predicted waste rock, has no humidity test cells and retention of representative samples of ultramafic rock and cells from units 4-6 are necessary to characterize this unit.

The low number of kinetic test cells in rock units 4-6 causes increased uncertainty about predictions waste rock leachate chemistry. Because of the high percentage of waste rock predicted to originate from these units (approximately 33%), as well as their importance in predicting leachate from unit 7, an effort was made to retain humidity cells from these units. Differences in sulfate release among units 4, 5 and 6 were more pronounced as waste rock category increased from Cat1 to Cat 3 (Figure 7). However, Ni release varied among units even in Cat 1 waste rock (Figure 8). These differences raise doubts about the ability to predict unit 7 behavior from units 4-6, and emphasize the importance of retaining the unit 4-6 cells recommended by MNDNR. Retention of cells from units 4-6 in Cat 1/2 waste rock is particularly important since the majority of unit 7 waste rock is in this waste rock category (PolyMet 2008).

References

Corriveau, M. 2009. pH Criteria for Modification to Kinetic Test Program - NorthMet Project, 28 April 2009.

Day, S. 2006. Waste rock and lean ore geochemical characterization plan, 5 May 2006.

Day, S. 2009. Modification to Kinetic Test Program memo, 9 Feb 2009.

Kellogg, C. 2008. Critique of PolyMet Analysis of Sulfate Release as a Function of Rock Type and Unit, 12 June 2008.

Kellogg, C., Olson, M., Lappako, K. 2009. Recommended Kinetic Test Modifications, 22 April 2009.

PolyMet. 2008. NorthMet waste rock chemistry model validation plan – sampling program 2.1, 15 April 2008.

Table 1. MNDNR Recommendations for SRK Kinetic Test Sampling

Waste Rock Category	SO_4 range (mg/kg/wk)	HCT ID#	Nickel trend	Recent Ni release weeks 130-168 (mg/kg/wk)	Rock Type	Rock Unit	S%	Min pH	Notes
1	less than 0.5	56	Stable	0.000062	Anor Troc	5	0.02	6.78	Above P95 Al rate
		37	Increasing	0.000323	Anor Troc	3	0.04	6.48	
		14	Increasing	0.000215	Aug Troc	1	0.06	6.66	
		51	Stable	0.000107	Troc	4	0.04	6.63	
		61	Stable	0.000411	Ultramafic	2	0.06	6.93	Above P95 Fe, Ca rates
		40	Stable	0.000060	Anor Troc	3	0.02	6.89	Above P95 Al, Ca rates
	0.5-1.5	3	Increasing	0.000465	Anor Troc	1	0.05	6.48	
		78	Stable	0.000419	Troc	6	0.05	6.89	
		33	Increasing	0.000226	Troc	2	0.07	6.6	
		15	Stable	0.000302	Aug Troc	1	0.07	6.6	Keep due to size fraction sample
	greater than 1.5	1	Increasing	0.001434	Anor Troc	1	0.09	6.41	Above P95 Co, Mn rates
		59	Increasing	0.000171	Troc	5	0.06	6.62	Above P95 As, Ca rate
		104	Stable	0.001304	Ultra	2	0.12	6.92	Above P95 Co, Fe, Se rates
2	less than 3.5	17	Increasing	0.000663	Aug Troc	1	0.19	6.09	Above P95 SO4, Ni, Cu, Mn, Zn rates
		34	Increasing	0.000629	Troc	2	0.18	6.57	Above P95 SO4, Ni, Cu, Mn, Cr rates
		53	Stable after inc	0.000349	Troc	4	0.21	6.44	Above P95 Cu, Mn, Zn rates
		48	Increasing	0.000181	Troc	3	0.25	6.54	Above P95 Cu rate
	greater than 3.5	2	Increasing	0.001312	Anor Troc	1	0.18	6.39	Above P95 SO4, Ni, Cu, Co, Mn rates
		5	Stable	0.000215	Sed Horn	1	0.24	6.87	Greater As release. Above P95 SO4, As, Ca, Zn rates
3	less than 4	**47**	Stable	0.001069	Troc	3	0.14	6.12	
		23	Increasing	0.327461	Ultra	1	0.2	5.63	High nickel. Above P95 Ni, Cu, Co, Mn, Zn rates
	greater than 4	**54**	Increasing	0.000854	Aug Troc	4	0.51	6.44	
		6	Stable	0.000270	Sed Horn	1	0.44	6.68	Greater As release Above P95 Mo rate
		7	Increasing	0.000178	Sed Horn	1	0.55	6.51	Greater As release Above P95 Mo rate
3LO	less than 4	103	Stable	0.000520	Anor Troc	1	0.18	6.81	
		96	Increasing	0.007392	Troc	2	0.17	6.36	
		80	Increasing	0.000222	Other Troc	6	0.18	6.94	Above P95 Al rate
	4-8	**77**	Stable	0.004980	Anor Troc	5	0.23	6.61	
		105	Increasing	0.013614	Anor Troc	3	0.28	6.4	
		100	Increasing	0.489076	Other Troc	1	0.36	5.48	High nickel. Above P95 Ni, Cu, Co, Mn, Cd, Zn rates
	greater than 8	**102**	Stable after inc	0.686973	Aug Troc	1	0.42	4.95	Only sample in range, high nickel. Above P95 SO4, Ni, Cu, Co, Al, Fe, Mn, Tl, Ba, Cd, Se, Zn rates
4	less than 25	10	Increasing	0.216502	Aug Troc	1	1.68	5.27	Above P95 Cu, Mn, Ba rates
		19	Increasing	0.553215	Other Troc	1	0.88	4.88	Above P95 Ni, Co, Mn, Se rates
	greater than 25	62	Increasing	0.007929	Sed Horn	VF	2	4.78	Above P95 Ca rate
		11	Stable after inc	0.270563	Sed Horn	1	2.47	3.64	Above P95 SO4 rate
		63	Stable	0.488788	Sed Horn	VF	3.79	3.36	Above P95 Ni, Cu, Co, F, Al, Be, Tl, Cd, Pb, Se, Zn rates
4LO	less than 30	93	Stable after inc	1.014068	Other Troc	1	0.86	4.23	Above P95 Ni, Cu, Co, Al, Be, Fe, Tl, Cd, Pb, Se, Zn rates
	greater than 30	65	Stable after inc	0.842457	Anor Troc	1	1.83	4	Above P95 SO4, Ni, Co, Al, Fe, Mn, Tl, Ba, Cd, Pb, Se, Zn rates
		106	Increasing	1.116882	Sed Horn	1	1.46	5.06	Above P95 SO4, Ni, F, Mn, Tl, B, Cd, Ca, Se, Zn rates
		68	Stable	0.533640	Sed Horn	1	4.46	3.61	Above P95 SO4, Ni, Co, Al, Fe, Tl, Cd, Zn rates

Bold: Indicates cells recommended by SRK for continuation
P95: Calculated 95% percentile for humidity cell release rates

Figure 1. Example SO_4 Range for Cat 1 Samples

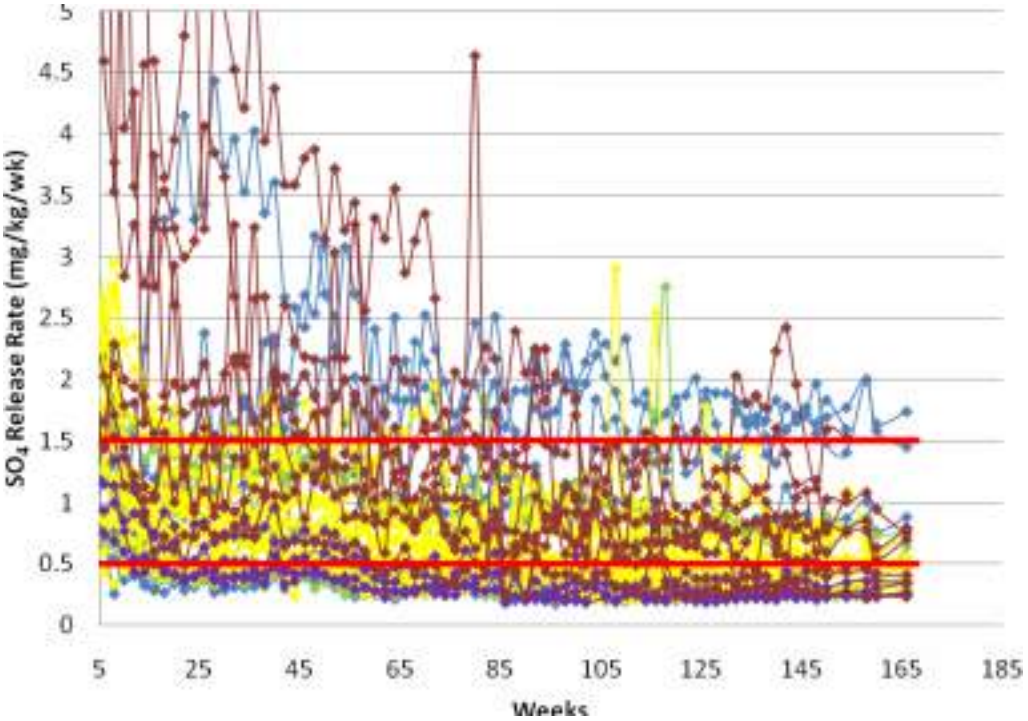


Figure 2. Stable Nickel Trend (Cat 1 Anor Troc: Ni, pH vs. time – HCT ID #56)

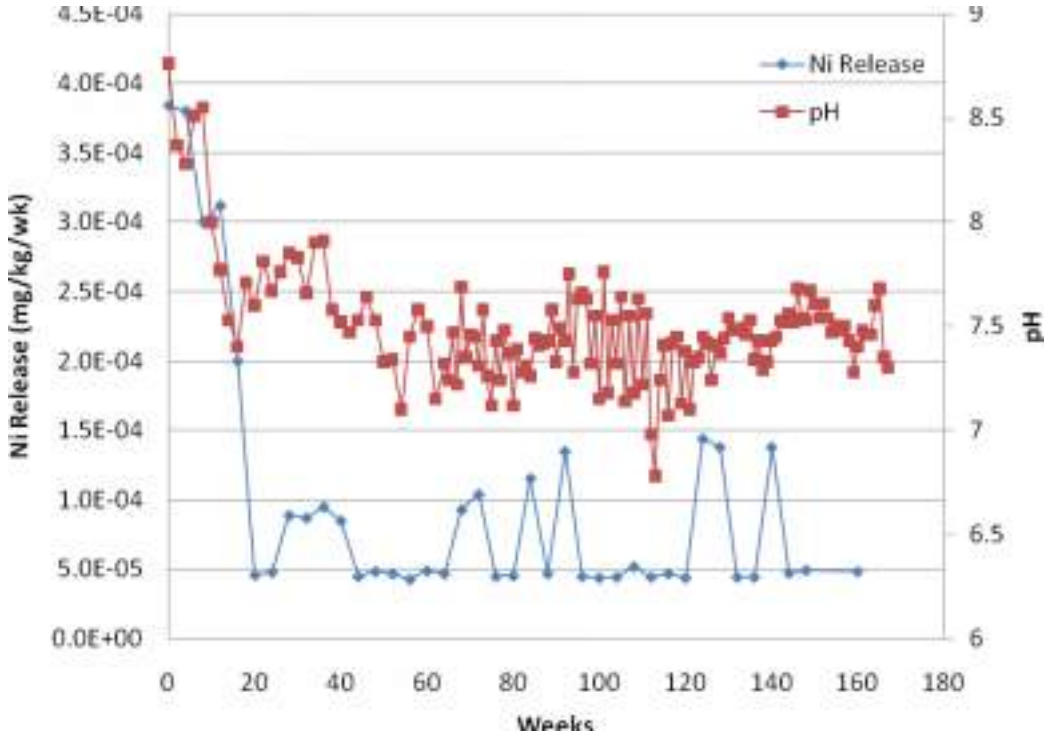


Figure 3. Increasing Nickel Trend (Cat 2 Anor Troc: Ni, pH vs. time – HCT ID #2)

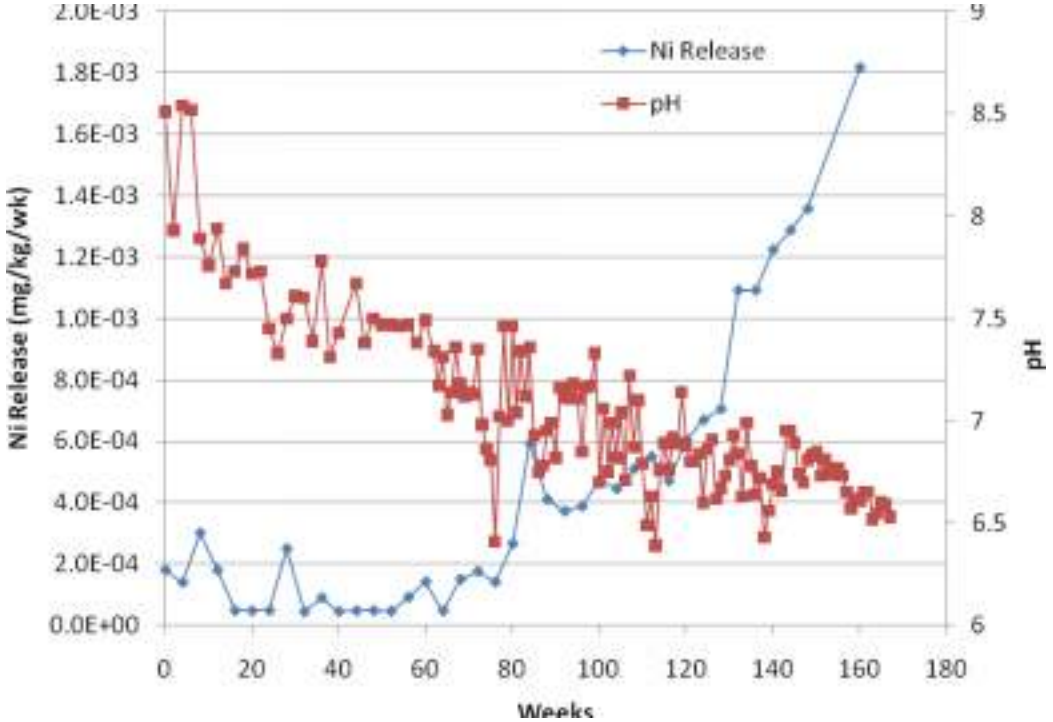


Figure 4. Stable After Increasing Nickel Trend
(Cat 3LO Aug Troc: Ni, pH vs. time – HCT ID #102)

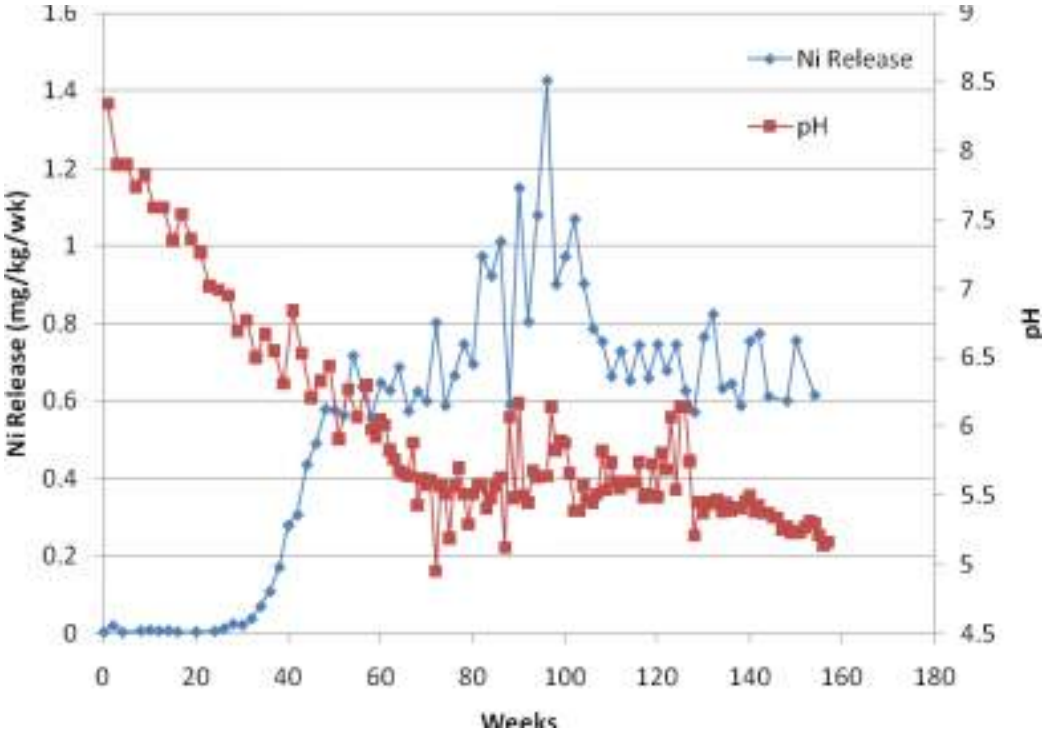


Figure 5. Average SO_4 release rate vs. S% by rock type

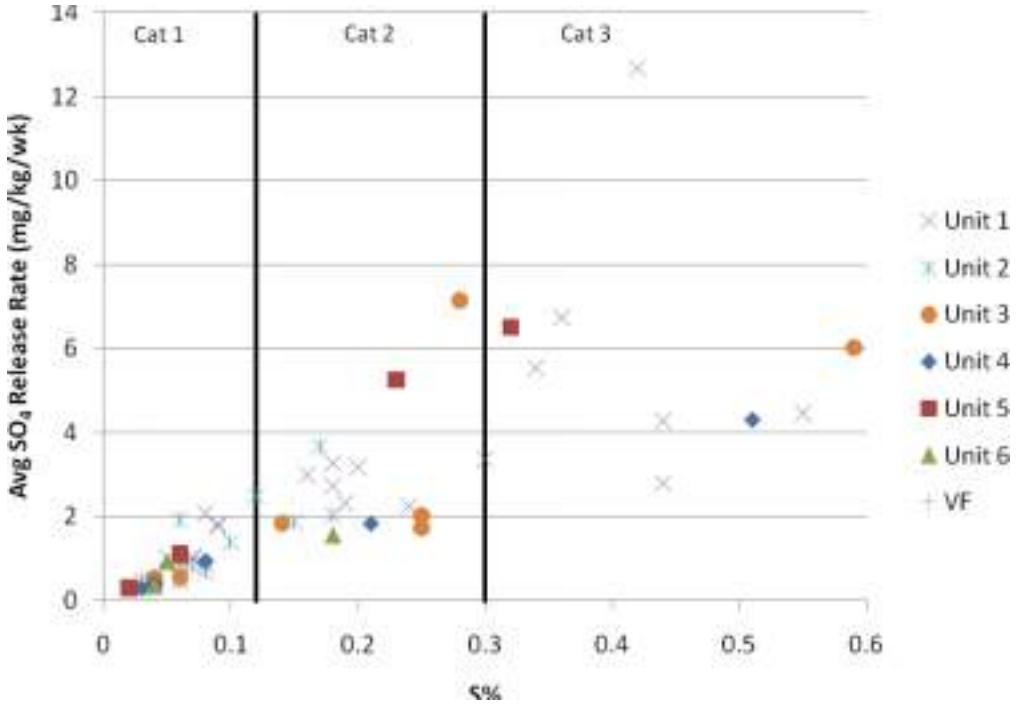


Figure 6. Average Ni release rate vs. S% by rock type

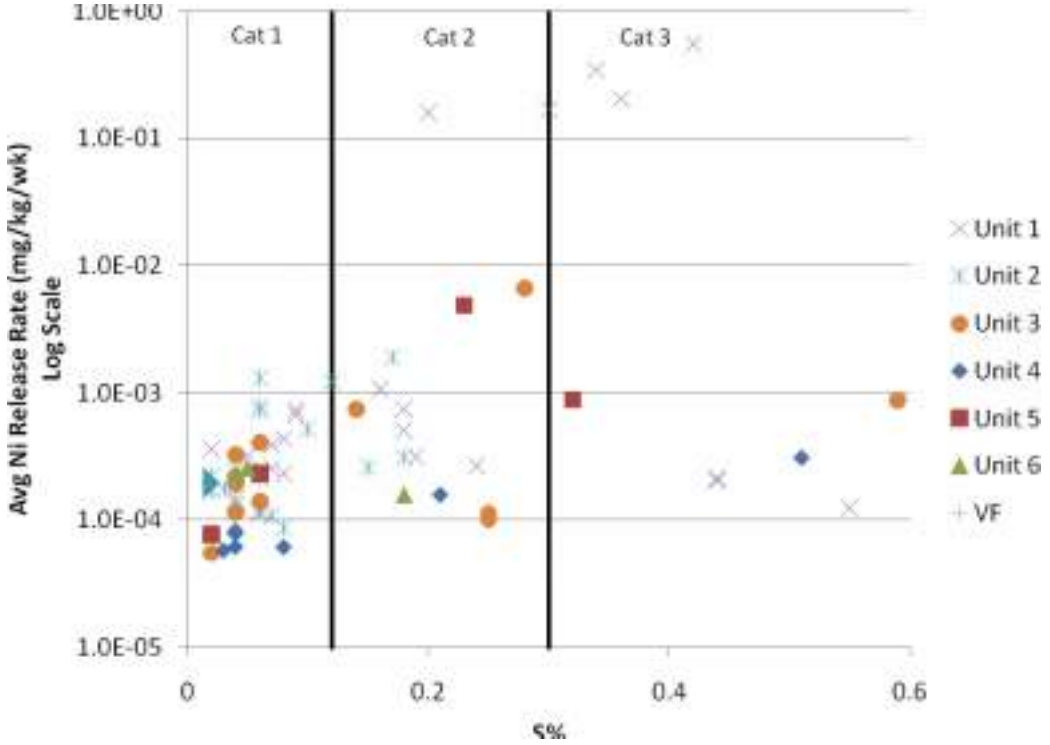


Figure 7. Average SO_4 release rate vs. S% for rock units 4-6

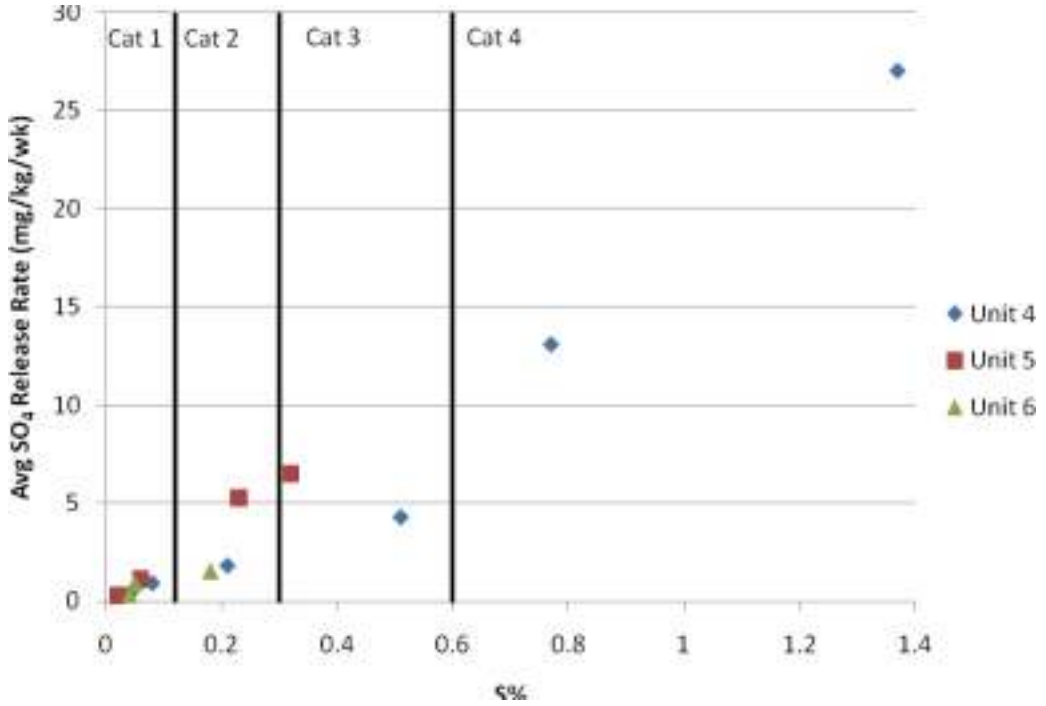
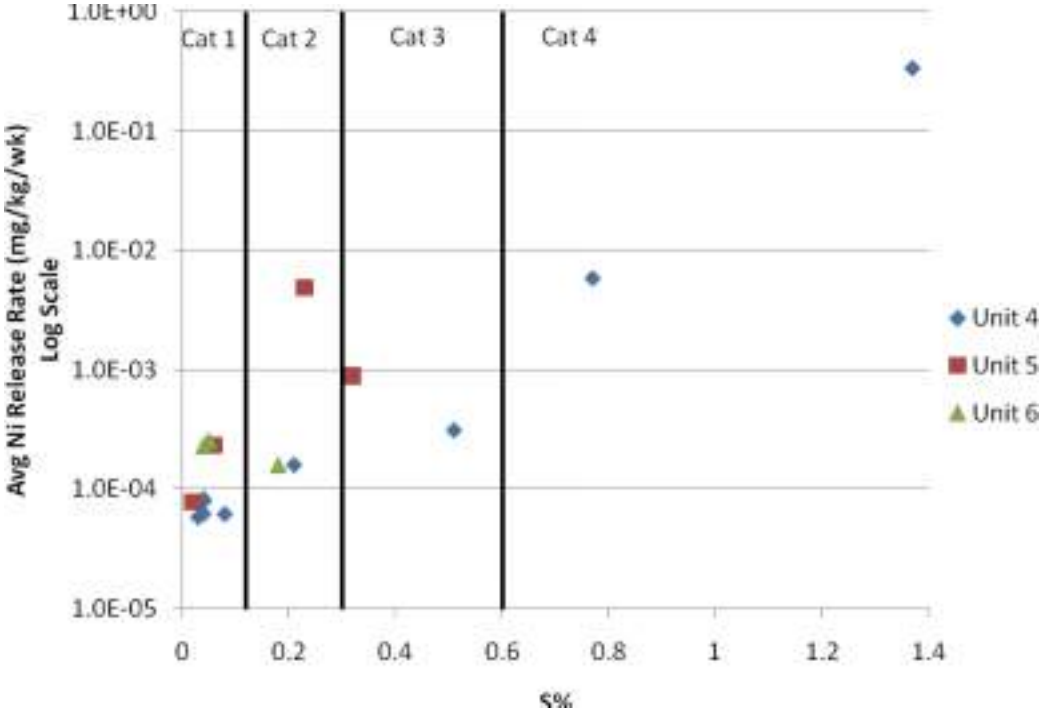


Figure 8. Average Ni release rate vs. S% for rock units 4-6



Attachment 2-17

SRK Consulting. pH Criteria for Modification to Kinetic Test Program - NorthMet Project. Memorandum to Jim Scott, PolyMet Mining. April 28, 2009.

SRK Consulting (Canada) Inc.
Suite 2200 – 1066 West Hastings Street
Vancouver, B.C. V6E 3X2
Canada

vancouver@srk.com
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Tel: 604.681.4196
Fax: 604.687.5532

Memo

To:	Jim Scott, PolyMet	**Date:**	April 28, 2009
cc:	Stephen Day	**From:**	Madeleine Corriveau
Subject:	pH Criteria for Modification to Kinetic Test Program - NorthMet Project DRAFT	**Project #:**	1UP005.001

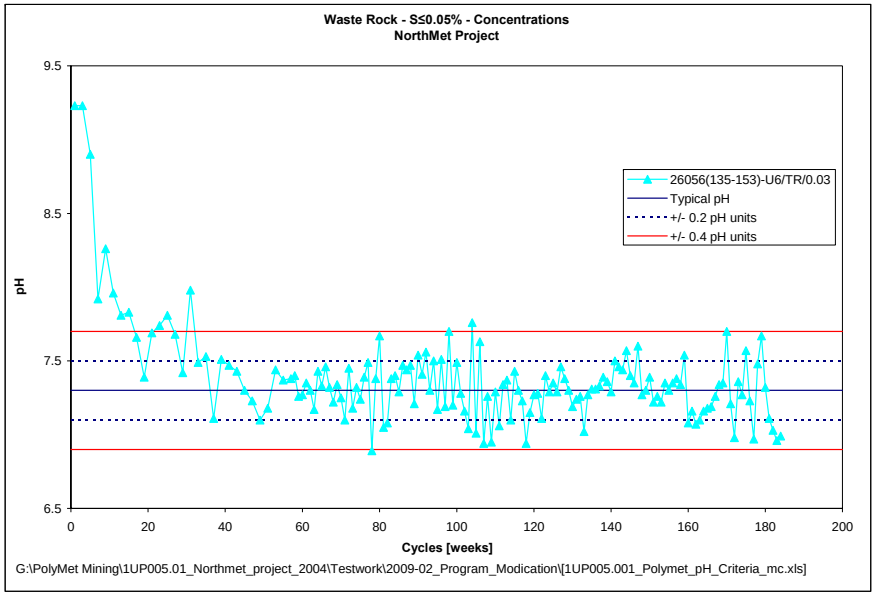
SRK provided recommendations for modifications to the NorthMet Project kinetic test program on February 9, 2009 ("Modification to Kinetic Test Program"). MDNR , in "Recommended Kinetic Test Modifications" (April 22, 2009), agreed that all waste rock, ore, tailings, and LTV tailings humidity cells can immediately be sampled at the reduced frequency proposed by SRK. However, MDNR required that weekly leachate be retained and stored for all cells and if any cell has a pH unit change of ≥0.2 during a 4-week period, all weekly samples for the four week period should be analyzed.

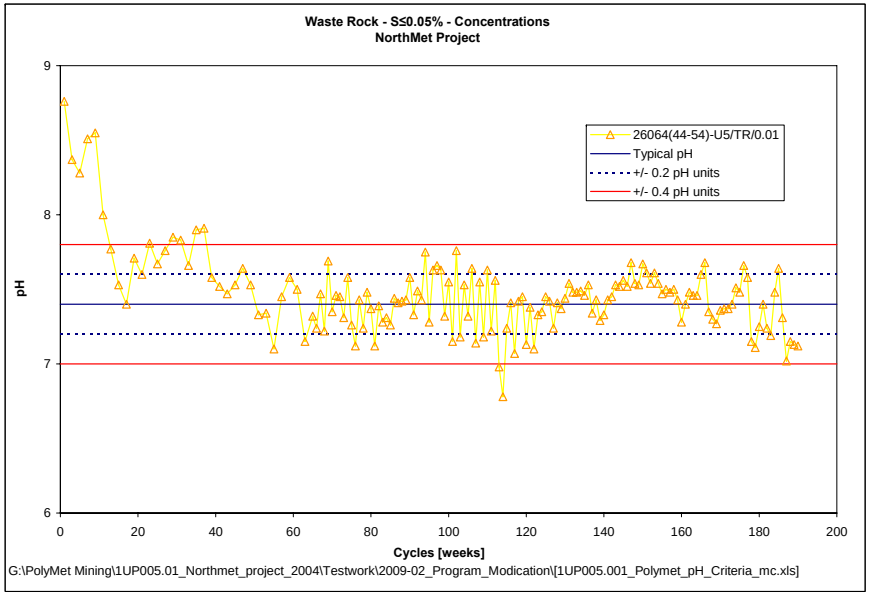
SRK has noted that this pH criterion will frequently trigger re-analysis through simple variation in leachate pH. Figure 1 provides examples of cells with relatively stable pH trends (neither consistently upwards or downwards) that often have pH changes greater than ±0.2 within a 4-week period. Variability for stable leachate pH is typically within ±0.4 pH units.

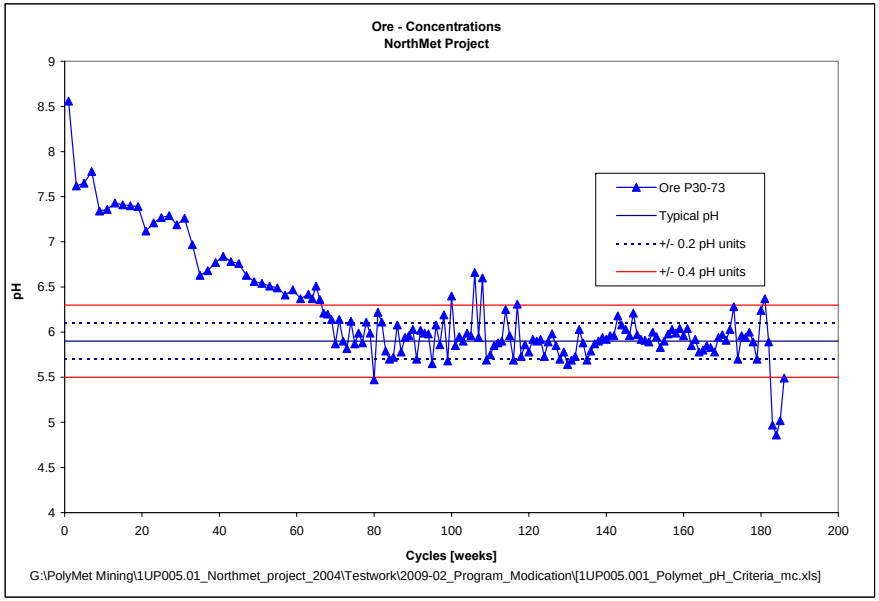
SRK understands MDNR indicated the 0.2 pH criterion to quantify large changes in release rates expected as pH changes rapidly. Figure 2 presents examples of cells that have demonstrated rapid changes in pH. Using the cells in Figure 2 as an example, rapid changes in pH move outside the ±0.4 pH unit range over a 4-week period and would trigger analysis of archived leachates. SRK also proposes that re-analysis be bi-weekly rather than weekly as required by MDNR because the existing program to date is based on bi-weekly analysis and has provided good definition of metal release trends.

Therefore, in order to avoid triggering analysis due to leachate variability, but still capture changes in release rates due to rapid changes in pH, SRK proposes:

- Bi-weekly analysis of archived leachates for tests showing a greater than pH 0.4 unit departure from stable average conditions.







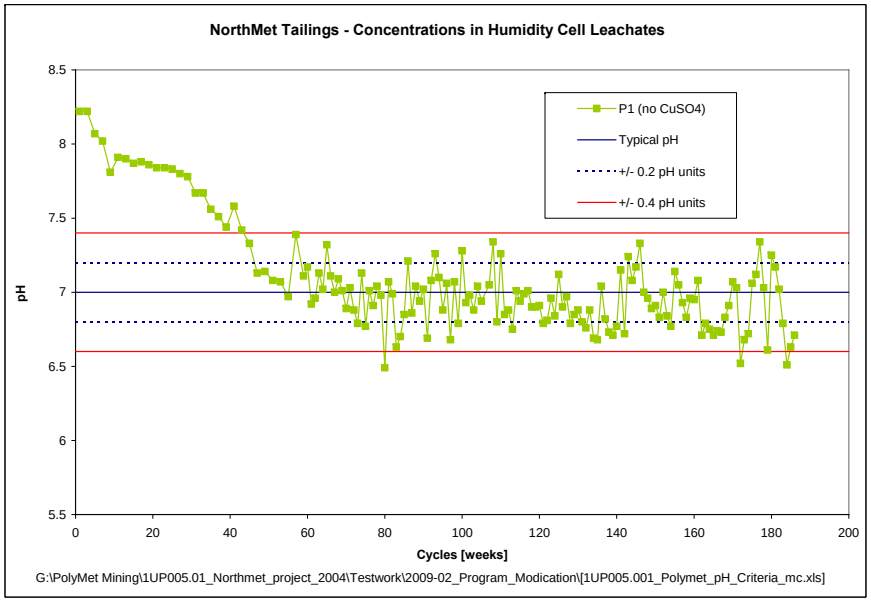


Figure 1: Examples of Humidity Cells with Relatively Stable Leachate pH

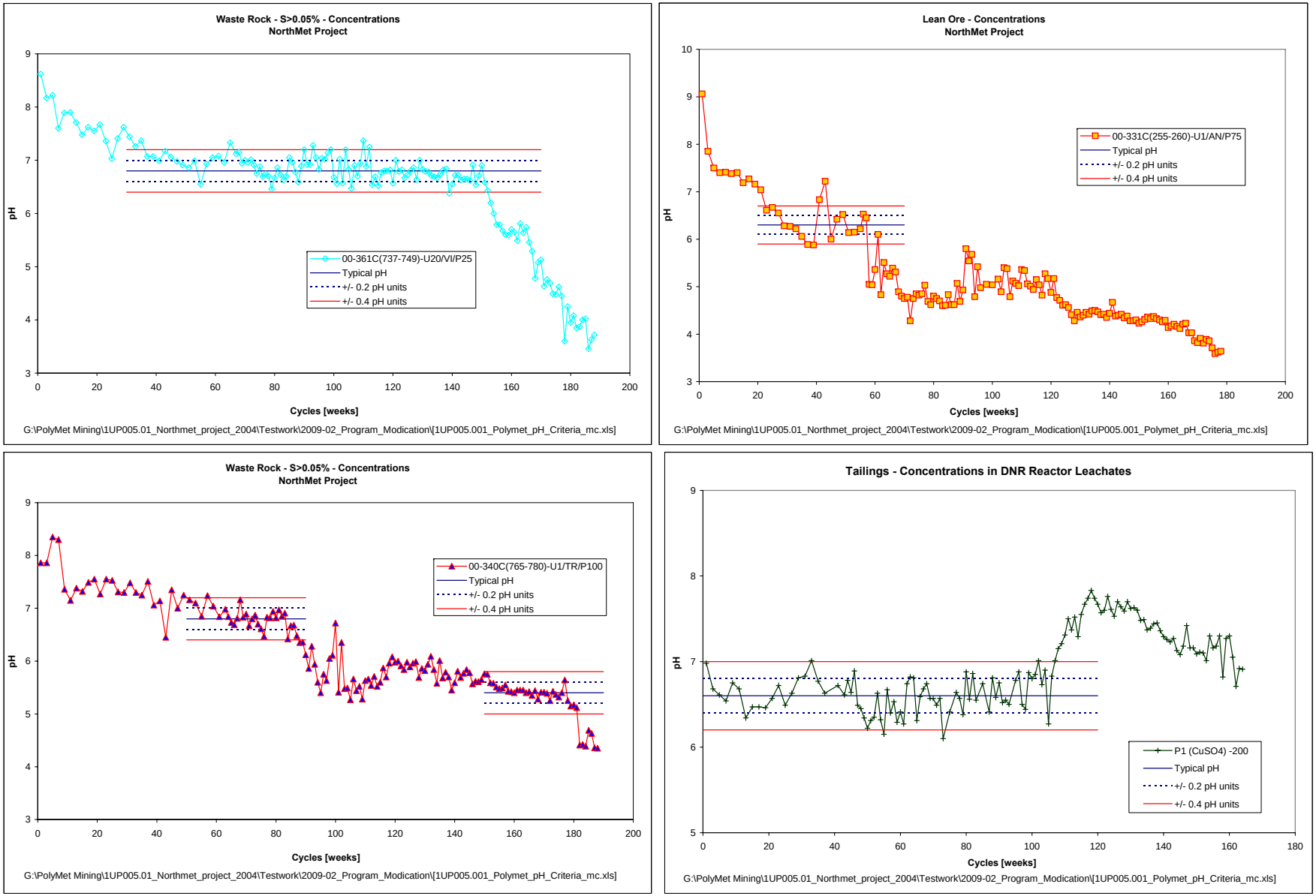


Figure 2: Examples of Humidity Cells with Changes in Leachate pH

Attachment 2-18

SRK Consulting. Update on Tailings Humidity Cell Test Data – NorthMet Project – DRAFT. Memorandum to Peter Hinck, Barr Engineering. December 23, 2011.

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Memo

To:	Peter Hinck	**Date:**	December 23, 2011
Company:	Barr	**From:**	Laura Donkervoort Stephen Day
Copy to:	Jim Scott, PolyMet	**Project #:**	1UP005.001
Subject:	Update on Tailings Humidity Cell Test Data, NorthMet Project – DRAFT		

1 Introduction

Kinetic geochemical testing of samples of tailings produced by pilot plant processing of NorthMet Project ores began in 2005 when the first process testing occurred. Subsequently, PolyMet has repeated similar pilot plant testing three times to optimize the flotation process and evaluate recovery of concentrate for different ore composites. Each time a pilot plant was operated, the resulting tailings samples have been submitted for the same geochemical test procedures including solids analysis for sulfur and trace element content and kinetic testing on whole samples and size fractions.

Fundamentally, the flowsheet of the flotation process has not changed except for copper sulfate not being used in the very first runs of Pilot Plant 1. Due to the improvement in recovery of sulfide minerals (and consequent reduction in sulfide content of tailings), copper sulfate has been used in all subsequent pilot plants. As a result, it is appropriate to consider all tailings results as a single dataset rather than individual datasets for each pilot plant run.

Separate tailings testwork has been initiated on four samples of LTVSMC tailings which will be used for construction of tailings embankments. As these tests are being performed on completely different materials, they are described separately at the end of this memorandum.

The purpose of the memorandum is to provide an update on tailings humidity cell data and their influence on inputs to water quality prediction models. A previous update (RS82 - Update on Use of Kinetic Test Data for Water Quality Predictions, February 2, 2009) was prepared with approximately 3 years data from NorthMet Project pilot plant samples generated in 2005 and 2006. Since that time, an additional three years of data have been collected for these tests and two additional data sets have been generated from tailings produced by the two additional pilot plants run in 2008 and 2009.

Due to the change in conceptualization of the overall modeling approach from the use of metal to sulfate ratios of release rates to the use mainly of sulfate release rates and metal to sulfur ratios in solids to predict pore water chemistry, this update report focuses mainly on a comparison of trends in pH and sulfate between the tests.

2 NorthMet Project Tailings Kinetic Tests

2.1 Status of Program

Table 1 lists all tailings humidity cells. The test program consists of conventional humidity cells with parallel tests using the MDNR reactor configuration on bulk tailings and tailings size fractions. All cells are currently running smoothly however, some repairs have occurred in addition to those stated in the RS82 report.

The filter fabrics in T55 and T62 humidity cells were found to be torn and allowing the breakthrough of solids into the leachate collection chamber. The cells were dismantled and repaired in September 2009. All solids were recovered and returned to the respective cells. Following these repairs, leachate chemistry re-stabilized within 2 weeks, returning to pre-repair levels. Arsenic levels were slower to decline, taking 7 weeks to return to pre-repair levels. This type of occurrence has been previously observed in the RS82 report.

All tests originally started are continuing and none of the tests have been terminated. Laboratory and analytical methods are unchanged from that developed in consultation with the MDNR and are consistent between all tests included in this update report.

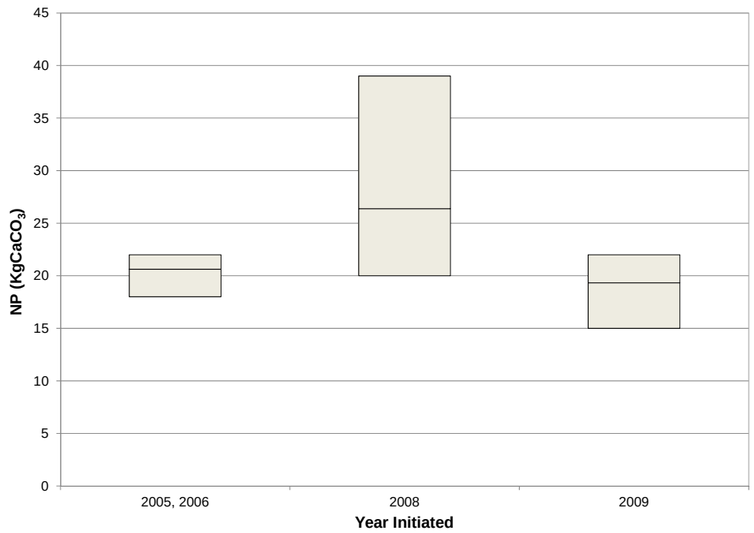
Table 1: Tailings Humidity Cells Used as Basis for Update Report

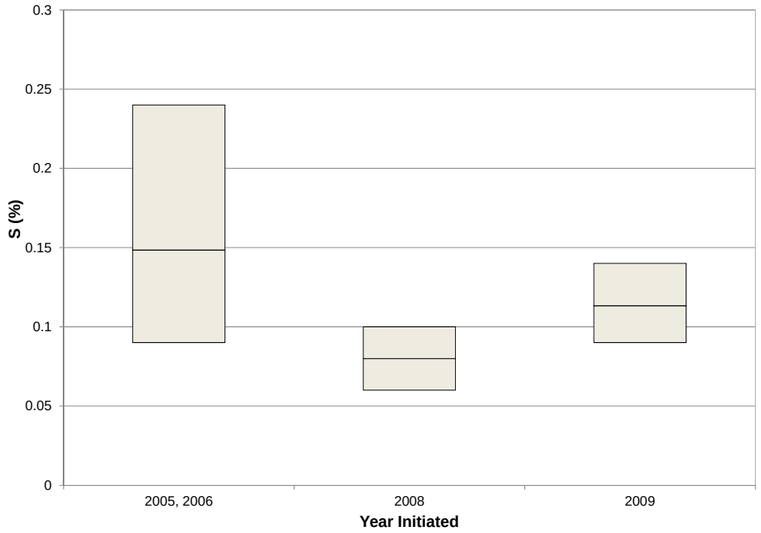
HCT ID	Fraction	HCT Full ID	Total Sulfur %	Initial Start Date	Total Weeks
T1	Whole	P1 (CuSO4)	0.1	9/8/2005	304
T2	Whole	P1 (no CuSO4)	0.23	9/8/2005	304
T3	Whole	P2 (no CuSO4)	0.2	9/8/2005	304
T4	Whole	P3 (CuSO4)	0.15	9/8/2005	304
T5	+100	Parcel 2 P2S +100 mesh	0.15	2/10/2006	282
T6	-100+200	Parcel 2 P2S -100 +200 mesh	0.17	2/10/2006	282
T7	-200	Parcel 2 P2S -200 mesh	0.24	2/10/2006	282
T8	+100	Parcel 1-2 PISCS +100 mesh	0.11	2/10/2006	282
T9	-100+200	Parcel 1-2 PISCS -100 +200 mesh	0.1	2/10/2006	282
T10	-200	Parcel 1-2 PISCS -200 mesh	0.09	2/10/2006	282
T11	+100	Parcel 3 P3S +100 mesh	0.11	2/10/2006	282
T12	-100+200	Parcel 3 P3S -100 +200 mesh	0.14	2/10/2006	282
T13	-200	Parcel 3 P3S -200 mesh	0.14	2/10/2006	282
T52	Whole	Pilot Plant 2 (0.30 Cu feed) Composite Bulk Tailings (as received)	0.07	7/8/2008	156
T53	+100	Pilot Plant 2 (0.30 Cu feed) Composite (+100 mesh)	0.08	7/8/2008	156
T54	-100+200	Pilot Plant 2 (0.30 Cu feed) Composite (-100 + 200 mesh)	0.06	7/8/2008	156
T55	-200	Pilot Plant 2 (0.30 Cu feed) Composite (-200 mesh)	0.09	7/8/2008	156
T56	Whole	Pilot Plant 3 (0.25 Cu feed) Composite Bulk Tailings (as received)	0.08	7/8/2008	156
T57	+100	Pilot Plant 3 (0.25 Cu feed) Composite (+100 mesh)	0.1	7/8/2008	156
T58	-100+200	Pilot Plant 3 (0.25 Cu feed) Composite (-100 + 200 mesh)	0.08	7/8/2008	156
T59	-200	Pilot Plant 3 (0.25 Cu feed) Composite (-200 mesh)	0.08	7/8/2008	156
T60	Whole	SCAV TAILS 9/30/09 1600	0.09	11/24/2009	84
T61	+100	SCAV TAILS 9/30/09 1600 +100	0.1	11/24/2009	84
T62	-100+200	SCAV TAILS 9/30/09 1600 -100+200	0.09	11/24/2009	84
T63	-200	SCAV TAILS 9/30/09 1600 -200	0.11	11/24/2009	84
T64	Whole	SCAV TAILS 10/1/09 09.00	0.13	11/24/2009	84
T65	+100	SCAV TAILS 10/1/09 09.00 +100	0.11	11/24/2009	84
T66	-100+200	SCAV TAILS 10/1/09 09.00 -100+200	0.14	11/24/2009	84
T67	-200	SCAV TAILS 10/1/09 09.00 -200	0.14	11/24/2009	84
T68	Whole	SCAV TAILS 10/1/09 17.00	0.12	11/24/2009	84
T69	+100	SCAV TAILS 10/1/09 17.00 +100	0.1	11/24/2009	84
T70	-100+200	SCAV TAILS 10/1/09 17.00 -100+200	0.1	11/24/2009	84
T71	-200	SCAV TAILS 10/1/09 17.00 -200	0.13	11/24/2009	84

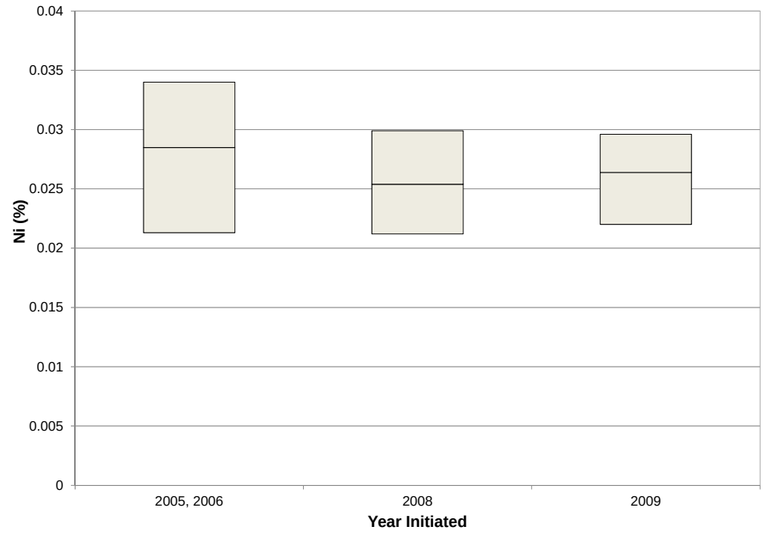
2.2 Comparison of Solids Data

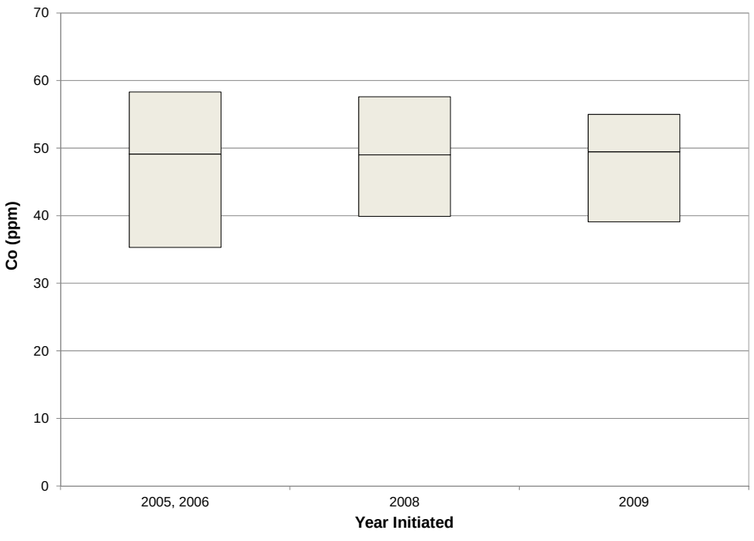
Solids data was compared between the three sets of humidity cells, (those initiated in 2005 & 2006, 2008 and 2009). Figure 1 shows the minimum, average and maximum values for key parameters.

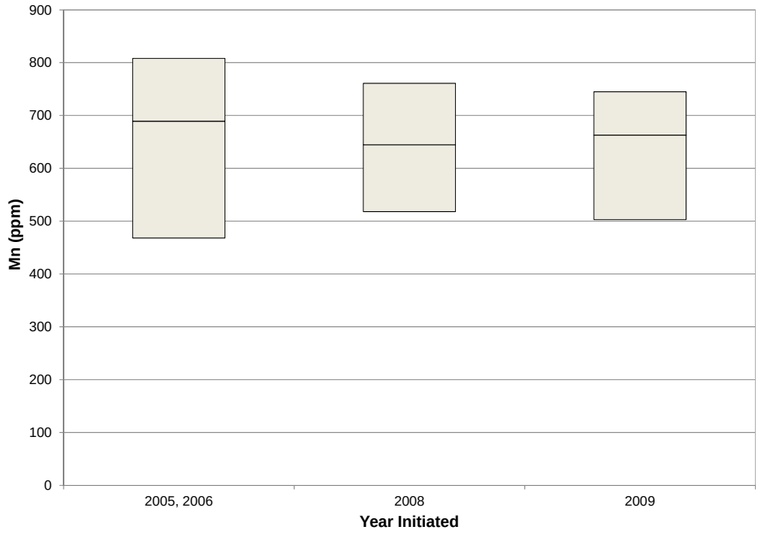
Sulfur concentrations showed the greatest range in the early dataset due to evaluation of the use of copper sulfate to optimize recovery of sulfide minerals by flotation. Subsequent pilot plants have used copper sulfate. Metal concentrations were very similar between the three data sets with the exception of copper, which showed slightly higher average and maximum values in the cells initiated 2005 and 2006.











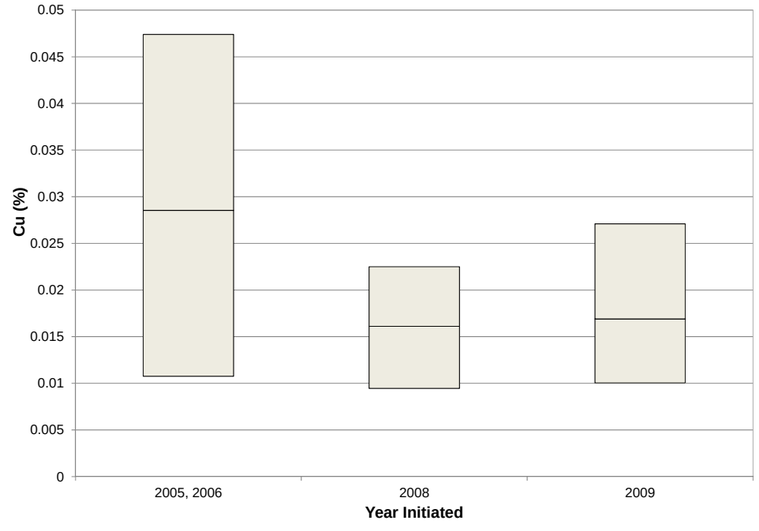


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Figure 1: Comparison of Solids data using Minimum, Average and Maximum Values – Key Results

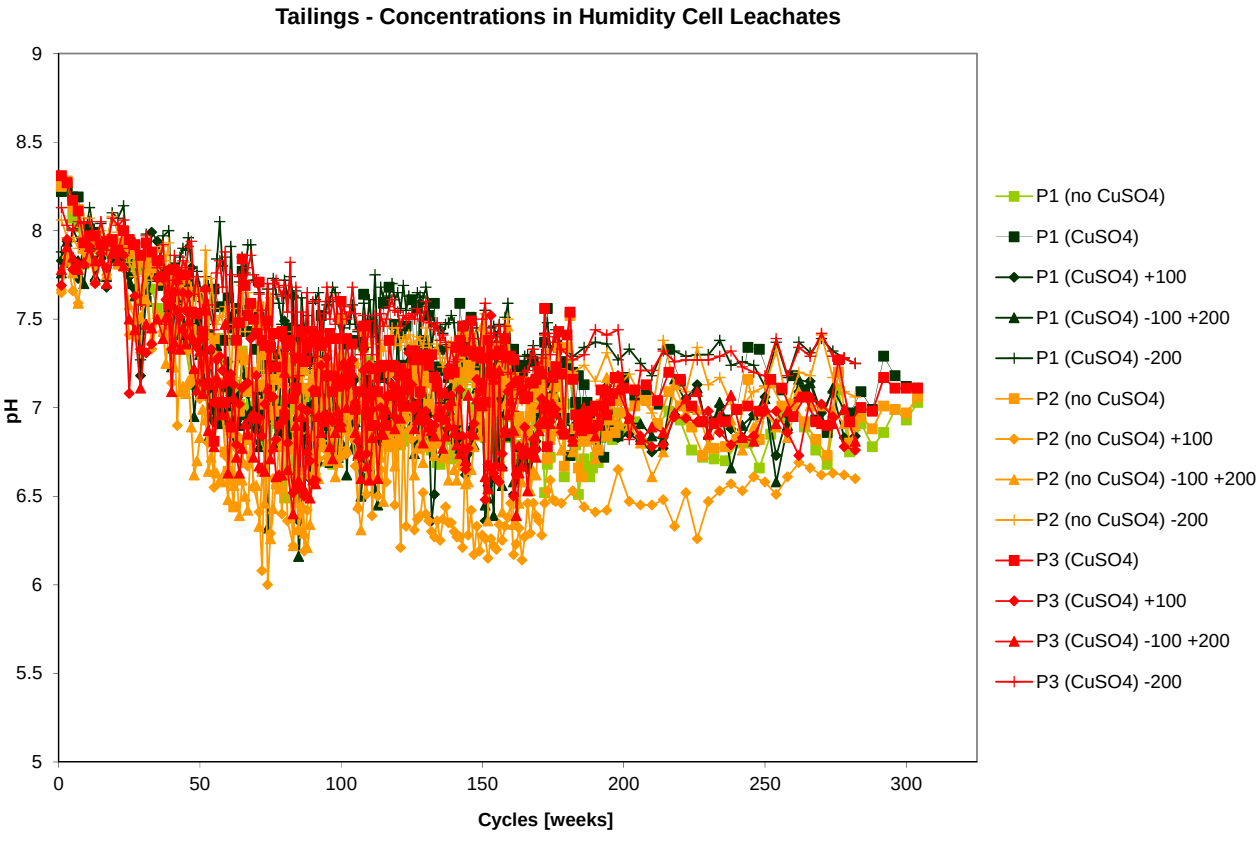
2.3 Description of Humidity Cell Results

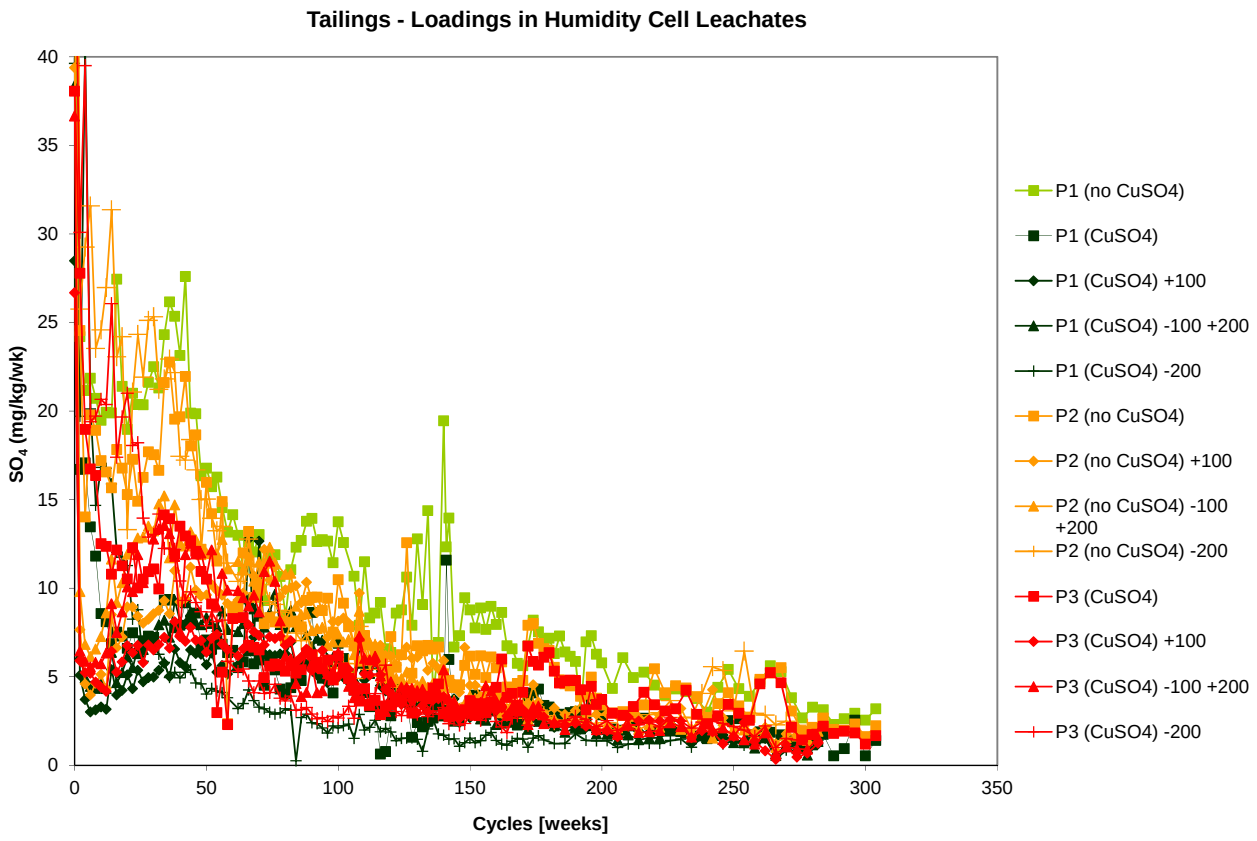
2.3.1 Tests Started in 2005 & 2006

All humidity cell tests showed an initial decline in pH (Figure 2). The decline was greatest for the two coarser particles size fractions and least for the -200 mesh fraction samples. The trend for bulk tailings was between the coarse and fine fractions. The range in leachate pHs following the main initial decline was 6.0 to 7.8. Leachate pHs remain somewhat erratic but no decline to lower pHs has been apparent and pH appears to be stabilizing with time. Variation reflects whether copper sulfate was used in the pilot mineral processing circuit to improve recovery of sulfide minerals during flotation. Higher pH leachates are generally associated with samples with lower sulfur contents produced by the use of copper sulfate.

Sulfate release has shown a decline from week zero to 200. After week 200, sulfate release has generally stabilized (Figure 2). Major ions have followed sulfate and were generally decreasing followed by stabilization after week 200. Alkalinity leaching from the two coarser particle sizes declined sharply and then stabilized, with the stable period starting at approximately week 75. In contrast, alkalinity from the -200 mesh samples continued to decline until stabilization at approximately week 125. Bulk tailings with higher sulfur contents have shown the similar trends to the two coarser particle sizes whereas bulk tailings with lower sulfur contents were closer to the -200 mesh samples with all bulk tailings samples having values between the coarse and fine fractions.

The main distinctive feature of leachates has been the increase in leaching of nickel (Figure 2) cobalt and manganese from the two coarse tailings fractions. This positive trend was apparent in the RS46 Draft 01 reporting period and was confirmed by the RS82 report. Since the RS82 report leachates have shown a decrease in these metals following their peak leaching rates at approximately week 100. However, values remain elevated relative to the fine size fraction and bulk samples. In addition, an increase in leaching of these metals in bulk samples was observed at approximately week 140 followed by a general decline but at an order of magnitude below the comparable coarse fraction samples. Copper leaching rates are stable at low values (<0.002 mg/kg/week).





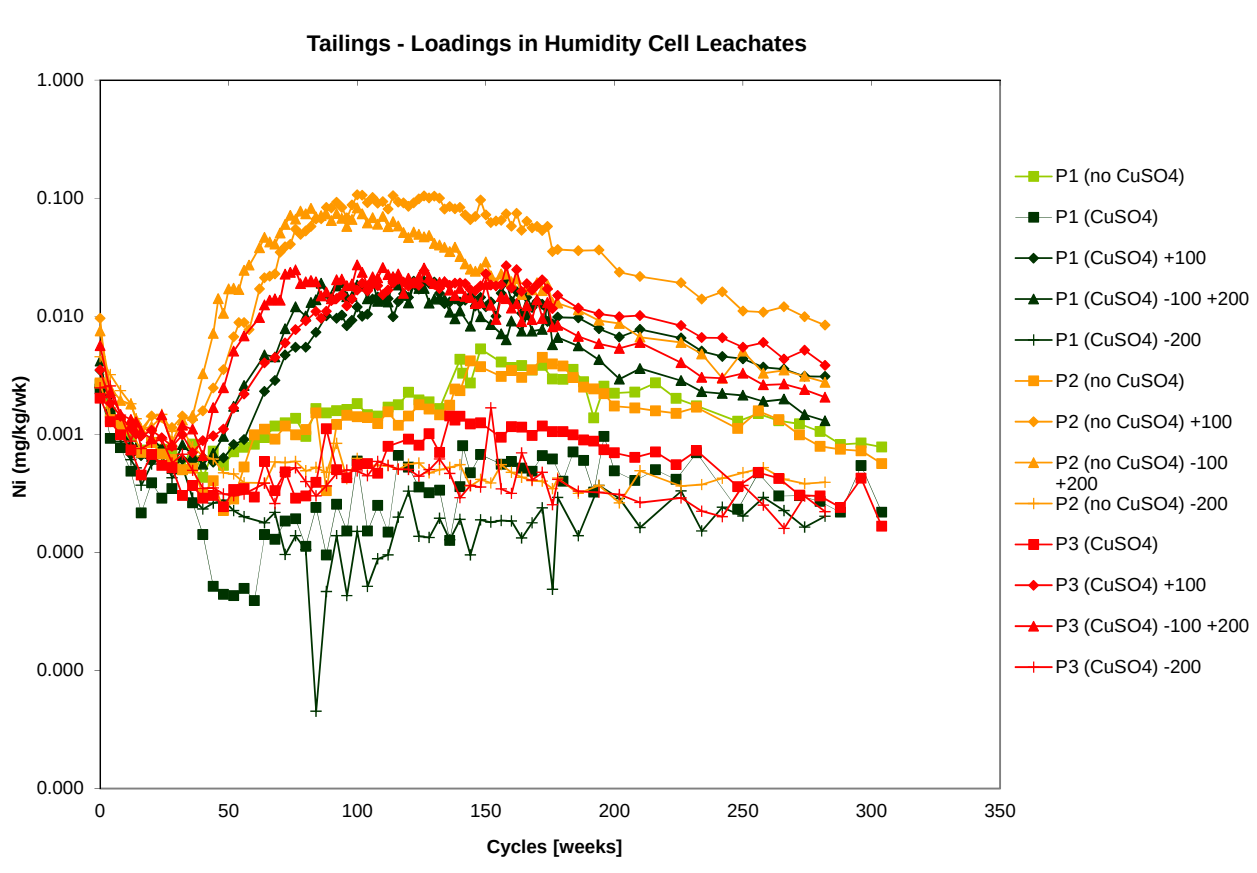


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Figure 2: Leachate pH, Sulfate and Nickel Loadings for 2005 & 2006 NorthMet Project Tailings

2.3.2 Tests Started in 2008

All humidity cell tests showed a slight initial decline in pH (Figure 3). The typical range in leachate pHs following the main initial decline was 7.25 to 7.78. pH does not appear to be related to fraction size as all pH measurements fluctuate in range of values. Leachate pHs remain somewhat erratic but no decline to lower pHs has been apparent, pH appears to be stabilizing with time.

Sulfate release initially declined sharply followed by a gradual decline (Figure 3). Major ions have followed sulfate but with calcium having a larger declining trend. Alkalinity has shown a consistent decreasing trend and is highest in the coarse particle size (+100) and lowest in the medium (+100-200) particle size.

For all fraction sizes, nickel showed an initial decline until week 80 at which time nickel release increased slightly and stabilized at levels slightly above the lowest release rates (Figure 3). The exception to this is the +100 mesh samples which showed an increase in nickel release at weeks 80 to 140 followed by a decrease to the lowest values of the test period. Copper release rates were more erratic but remained at low levels throughout the testing period in the range of 0.0002 mg/kg/week to 0.002 mg/kg/week.

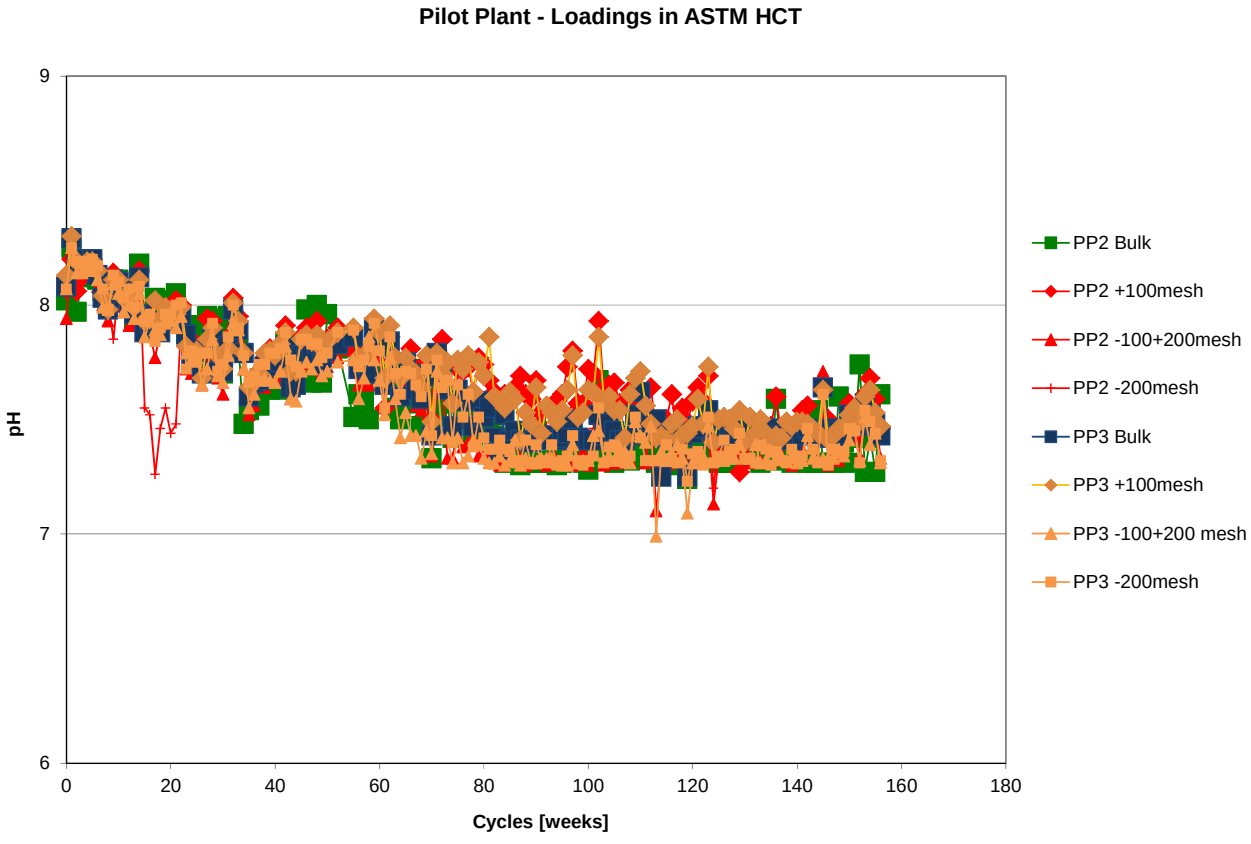
Results from these tests are distinctively different from the similar tests started in 2005 and 2006. Leachate pHs have not decreased to the same degree when comparing the same test period (first three years). As a result, accompanying order-of-magnitude increases in nickel concentrations have not been observed. Sulfate release for the 2008 tests also stabilized earlier.

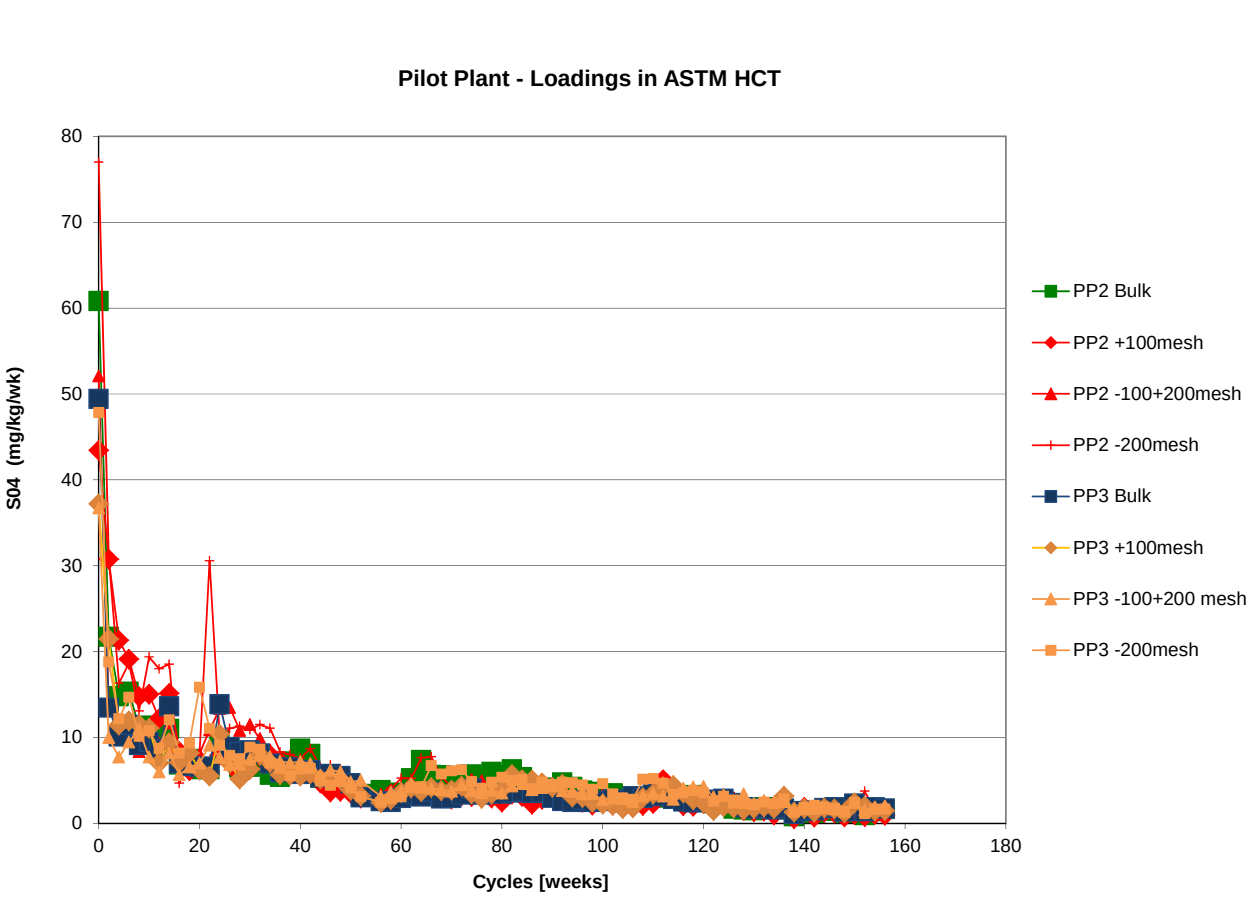
2.3.3 Tests Started in 2009

All humidity cell tests showed generally stable pH (Figure 4). However, a slight decline has occurred after 50 weeks of measurement in the -100 +200 mesh samples. The general trend is highest pHs are reported for the fine fraction and lowest for the -100 +200 mesh fraction with bulk tailings measurements between the coarse and fine fractions. The range in leachate pHs was 7.1 to 8.0.

Sulfate release initially declined sharply followed by a gradual decline (Figure 4). Major ions have followed sulfate but show a larger declining trend. Alkalinity showed an initial stable period from weeks 4 to 36 followed by a declining trend. This trend is most apparent in the coarse fraction.

The slight pH decrease at week 50 in the -200 +100 mesh tailings fractions was also accompanied by a slight rise in nickel (Figure4), cobalt and manganese release rates. All other size fractions have shown stable or decreasing release rates. Copper release rates have remained low for all cells throughout the test period in the range of 0.0001 mg/kg/week to 0.001 mg/kg/week.





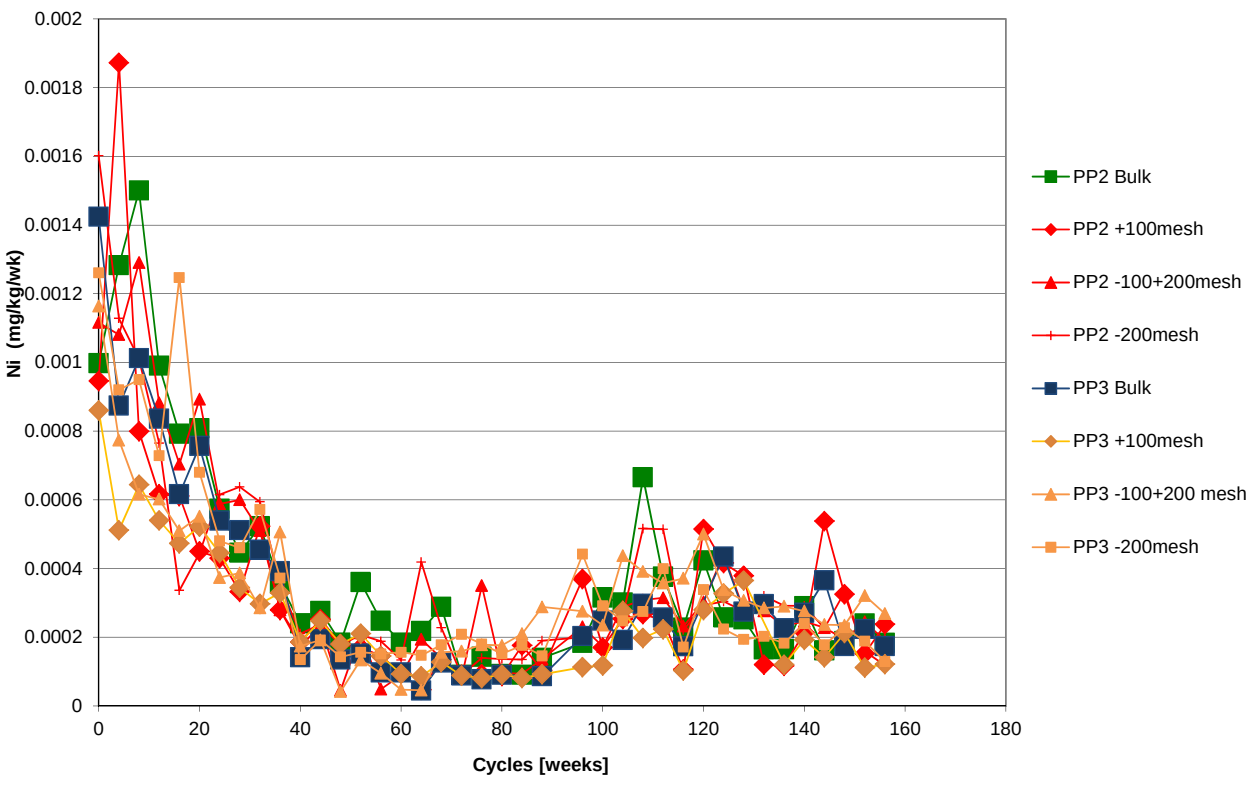


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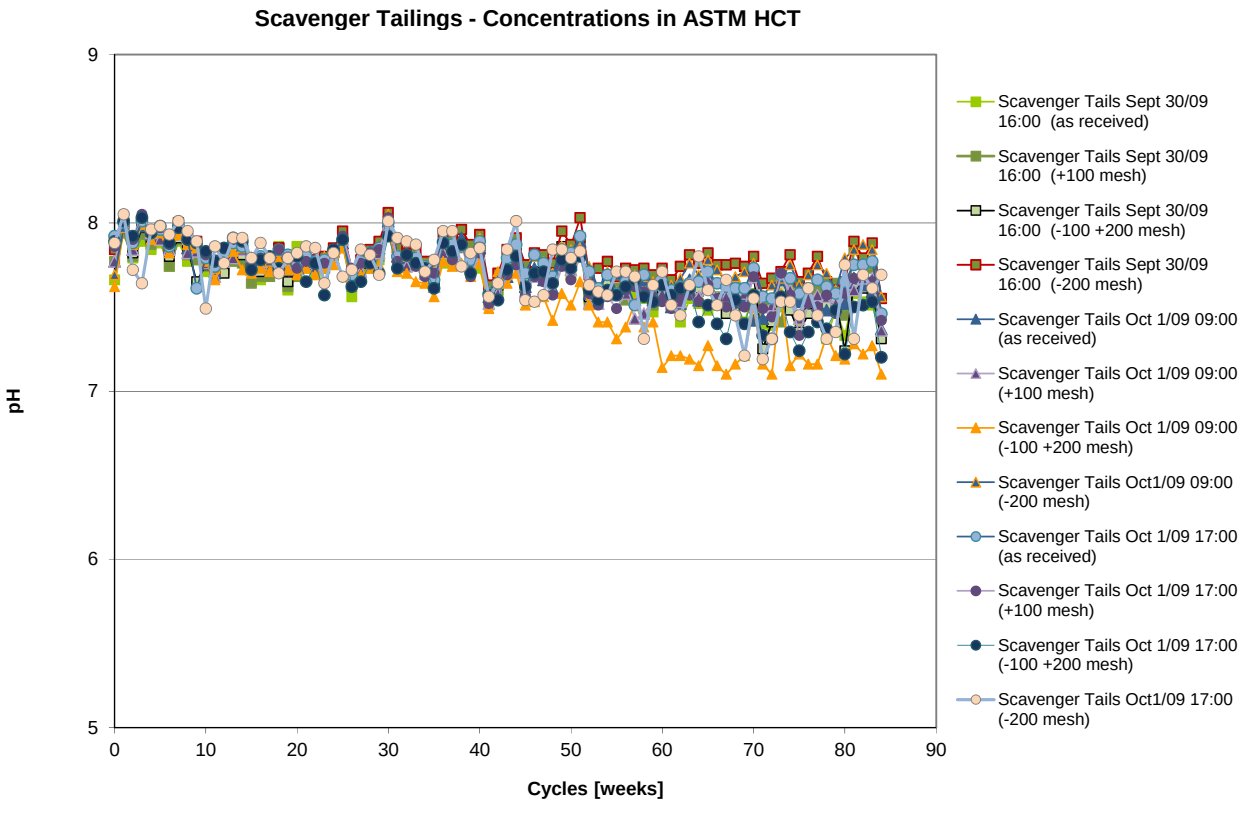
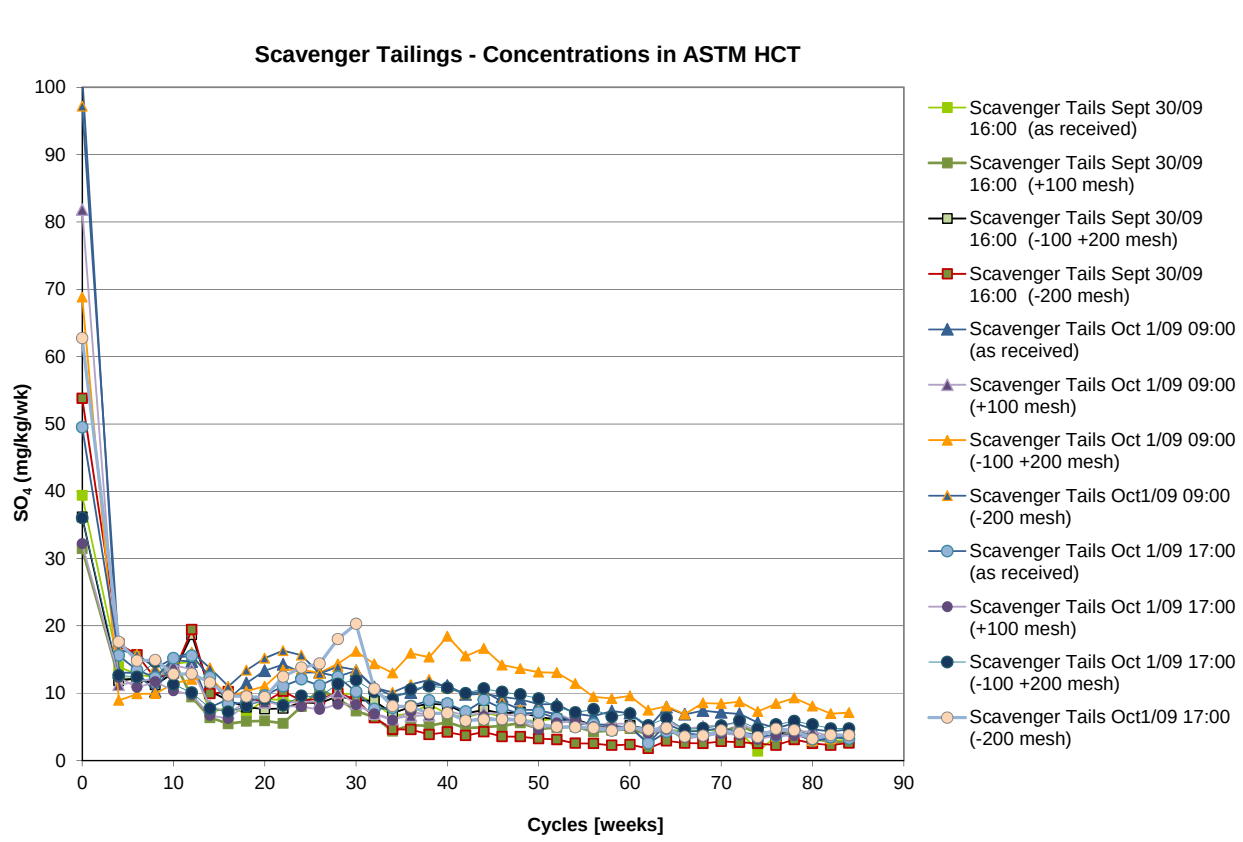
Figure 3: Leachate pH, Sulfate and Nickel Loadings for NorthMet 2008 Pilot Plant





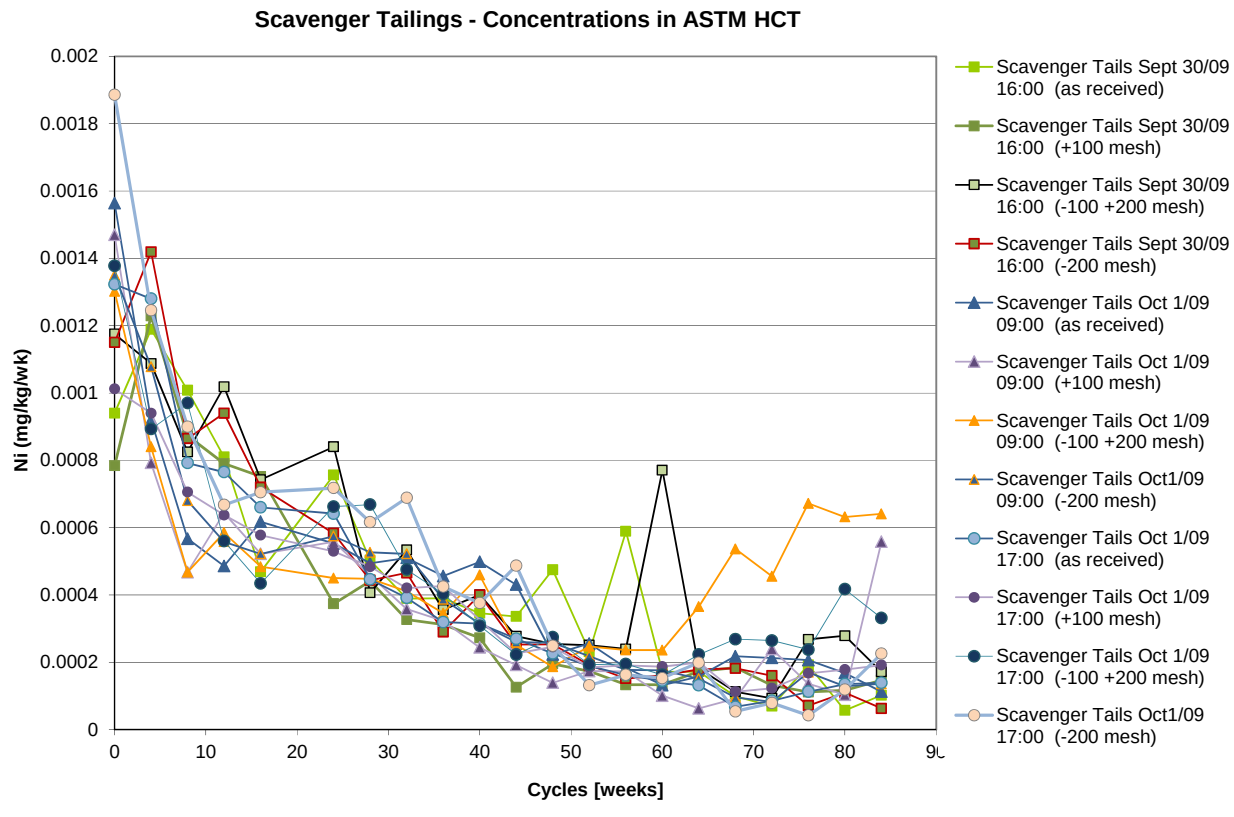


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Figure 4: Leachate pH, Sulfate and Nickel Loadings for NorthMet 2009 Scavenger Tailings

2.4 Discussion

The primary inputs to water quality modeling for the tailings are sulfide oxidation rates indicated by sulfate release from humidity cells. For the purpose of modeling, the sulfide oxidation reaction is assumed to be zero order, that is, the oxidation rate is independent of the amount of reactant (sulfide) remaining. This assumption was adopted to simplify the implementation of sulfide oxidation in the tailings water quality model and is conservative from the standpoint of estimating maximum expected concentrations of sulfate and other parameters. The current extended dataset provides an opportunity to evaluate the assumption and compare tailings produce by several metallurgical test programs.

Figure 5 shows sulfate release for all whole tailings samples being tested. Lines on the graph show the three different programs as different symbols and color-coding illustrates different initial sulfur concentrations. Sulfur concentrations exceeding 0.2% are shown as red; concentrations between 0.1 and 0.2% are shown as amber; and concentrations less than 0.1% are shown in green.

The overall distribution of rates shows that oxidation rates are correlated with sulfur content. Highest rates are associated with higher initial sulfur concentrations, and lowest rates are shown by samples with lowest sulfur concentrations.

Samples with sulfur concentrations exceeding 0.2% were only generated in the first pilot plant run when copper sulfate was not used at first to improve sulfide mineral flotation. Comparison between programs for the middle and lower sulfur ranges shows that there is little difference between results from the programs. In the middle sulfur range, oxidation rates for the initial pilot plant run samples were slightly higher than those in the same range produced in 2009, but the 2009 samples also had slightly lower initial sulfur concentrations. The same effect is seen in the lower sulfur range. The first pilot plant sample had the highest concentration of the four samples in this range and it appears yielded the highest sulfate release rates in that sulfur range. As a group, the testwork performed at various times shows similar relationships between sulfur content and sulfate release.

All tests showed that sulfate release declined over time. Mass balance indicates that the first and second sample set have been depleted of between 30% and 40% of the initial sulfur content indicating that the decline in sulfate release may be due to reduction in available oxidation rates. This implies the sulfide oxidation reaction is non-zero order and that the use of initial oxidation rates in modeling remains conservative.

Differences in pH trends and consequent release of nickel have been observed in the testwork. The initial dataset showed that pH decreased to a level at which nickel release could increase due to increased solubility of secondary nickel. This trend has not been observed to the same degree in subsequent testwork. The strong sensitivity of the test procedures to pH buffering by silicate mineral weathering has been observed previously and may be the reason for the observed differences. The 2008 tests suggest the tailings have greater buffering capacity than the 2005/2006 tests. Use of the first dataset to define sulfur concentrations which could result in significant metal leaching due to pH depression that could occur appears to be conservative.

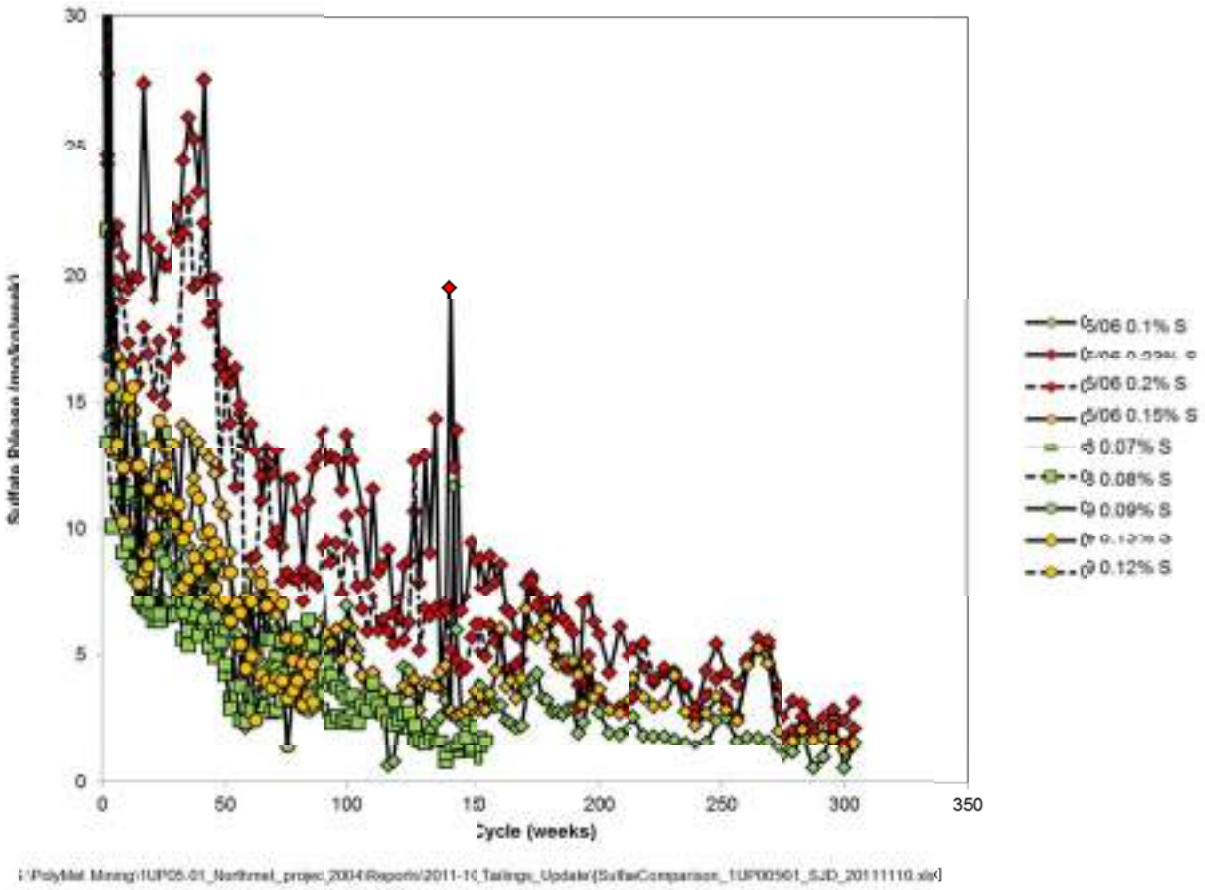

Figure 5: Comparison of Sulfate Leaching Rates as a Function of Sulfur Content

2.5 Conclusions

Review of ongoing and new humidity cell data shows that the assumptions used for oxidation rates in the water quality modelling are conservative when assuming the reaction order for sulfide oxidation, and that oxidation rates shown by tailings produced at different times are comparable.

Differences in pH trends are apparent between tailings produced by the different rounds of metallurgical testwork. However, the differences do not indicate a need to re-visit assumptions about sulfur management criteria (tailings to contain less than 0.2% sulfur). Ongoing testwork on tailings containing more than 0.2% sulfur for six years has shown stable pH as sulfur content has depleted to 40% of the starting composition.

3 LTVSMC Tailings Samples

3.1 Source of Samples

Twenty-nine samples of coarse LTVSMC tailings were obtained as part of a drilling program supervised by Barr Engineering to evaluate the geotechnical characteristics of the tailings. Ten drill holes were sampled in five different potential borrow areas (Figure 6).

Figure 6: LTVSMC Drilling Locations

3.2 Characteristics of Samples

Selected characteristics of the samples are shown in Table 2. Overall characteristics are similar to those for samples collected in an earlier drilling program and previously reported in RS46. Sulfur and trace element contents are very low. LTVSMC tailings samples are classified as non-acid generating due to the presence of carbonate minerals.

Results for the 29 samples were used to generate a sub-set of samples for humidity cell testing. In order to obtain sufficient sample for humidity cell testing (1 kg), composites were produced to represent coarse tailings in the four main borrow areas:

- 2E North Embankment
- 1E and 2E Separator
- 1E South Beach
- 2W North Embankment

Characteristics of the resulting composites are shown in Table 3. The sample from 2E North Embankment was tested in duplicate.

Table 2: Characteristics of LTVSMC Tailings

Statistic	As mg/kg	Co mg/kg	Cu mg/kg	Ni mg/kg	Carbonate %, C	Total S %, S
P5	14.88	5.94	5.28	2.78	0.638	<0.01
P50	25.5	7.9	7.3	4.1	1.45	0.03
P95	55.96	10.62	21.44	7.14	2.66	0.08

Source: \\Van-svr0.van.na.srk.ad\ge_projects\PolyMet Mining\1UP005.01_Northmet_project_2004\Testwork\2011-04_LTV_Static_Compilation\[LTV_Static_Compilation_1UP005001_SJD_20110428.xlsx]

Table 3: Characteristics of LTVSMC Composite Tailings Samples Tested in Humidity Cells

HCT ID	As mg/kg	Co mg/kg	Cu mg/kg	Ni mg/kg	Total S %
2E North Embankment Composite	28.7	8	7.8	4.1	0.03
2E North Embankment Composite (Duplicate)	22.7	6.9	7.7	3.8	0.03
1E and 2E Separator Composite	31.9	9.8	16.1	5.1	0.04
1E South Beach Composite	21.1	8.4	20.1	7	0.01
2W North Embankment Composite	46.9	8.1	8.5	4.6	0.06

Source: \\Van-svr0.van.na.srk.ad\ge_projects\PolyMet Mining\1UP005.01_Northmet_project_2004\Reports\2011-10_Tailings_Update\Re-Issue\[LTVSMC_Tailings_Data_1UP005.01_ld_rev00.xlsx

3.3 Humidity Cell Procedure

Shake flask testing of the samples prior to humidity cell testing indicated that the tailings were slightly oxidized due to weathering under field conditions. Since the purpose of humidity cells testing is to understand sulfide mineral oxidation rates, the samples were initially flushed with de-ionized water to remove soluble oxidation products. Following this procedure, the flushed samples were tested using the same procedures as used for NorthMet Project tailings.

3.4 Results

Selected humidity cell results from about 65 weeks of testing are shown in Figures 7 and 8.

Leachate pH values from all tests have been consistently between 7.3 and 8.1 and result mainly from leaching carbonate minerals. One of the 2E North Embankment samples showed consistently lower pH than the other tests. Likewise alkalinity, Ca and Mg were not comparable for the duplicate pair. All other parameters showed good reproducibility for the pair implying that the differences are due to heterogeneity in the carbonate content which did not affect elements associated with sulfide minerals.

Initial differences in sulfate release were observed. Higher release rates were observed for the two samples containing higher sulfur concentrations (2W North Embankment, and 1E and 2E Separator) but after about 25 weeks, differences disappeared and all samples yielded comparable low sulfate release. Leaching of trace elements occurred at very low rates. Arsenic, cobalt, manganese, molybdenum, selenium showed an initial decline like sulfate but most elements, including copper, nickel and zinc indicated no trend.

3.5 Conclusions

LTVSMC tailings leachate chemistry is dominated mainly by the buffering effects from dissolution of carbonate minerals. Initial differences in sulfate and some trace element release may be linked to differences in sulfide content but after about 25 weeks similar weathering behavior was shown regardless of sulfur content. Due to low sulfur content and expected long term effects of carbonate buffering, the tests have proceeded sufficiently to provide rates for input into water quality modeling. Release rates derived from these tests will be the average observed release over the 65-week testing period, which is conservative with respect to sulfate release.

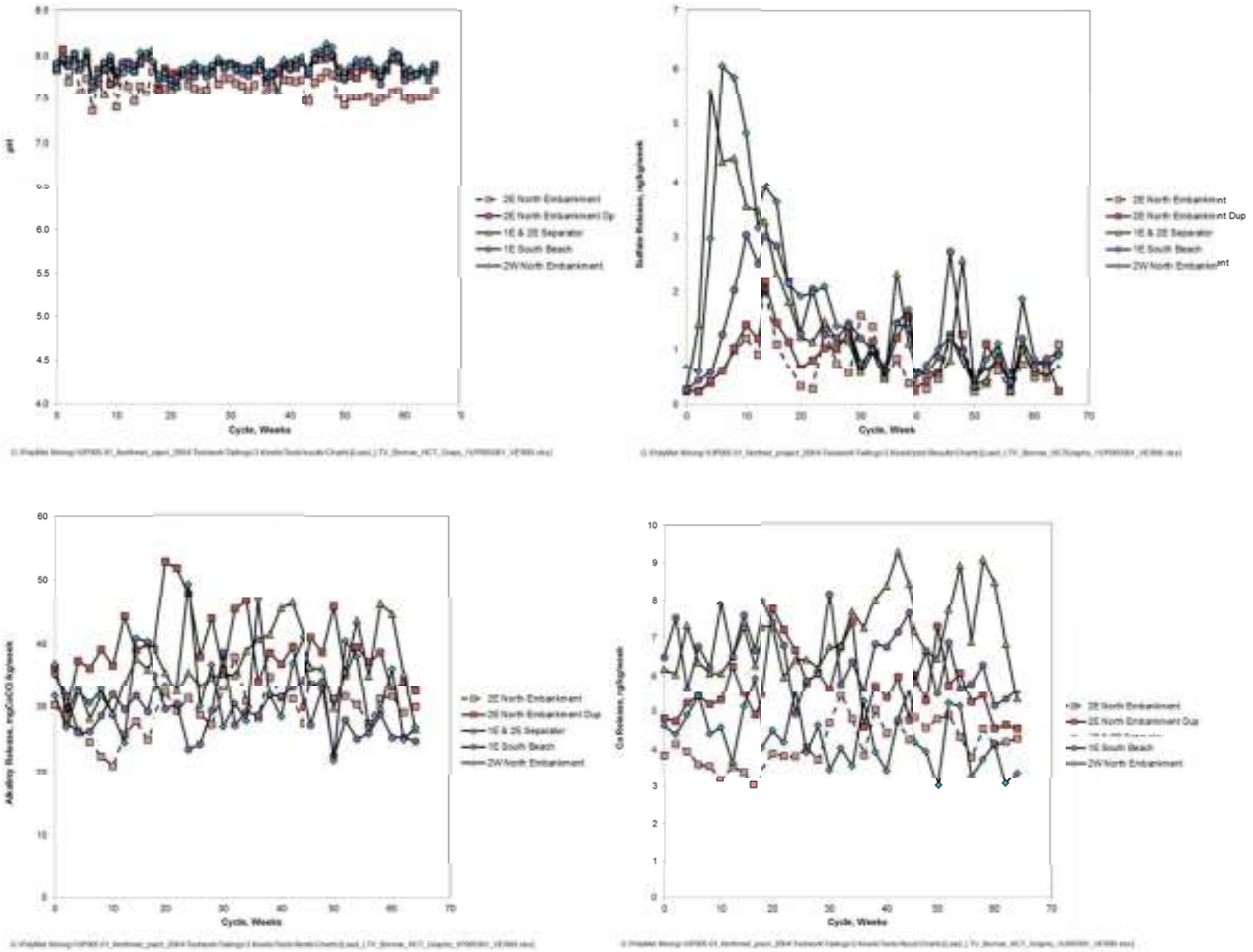

Figure 7: Major Ion Results for LTVSMC Humidity Cells

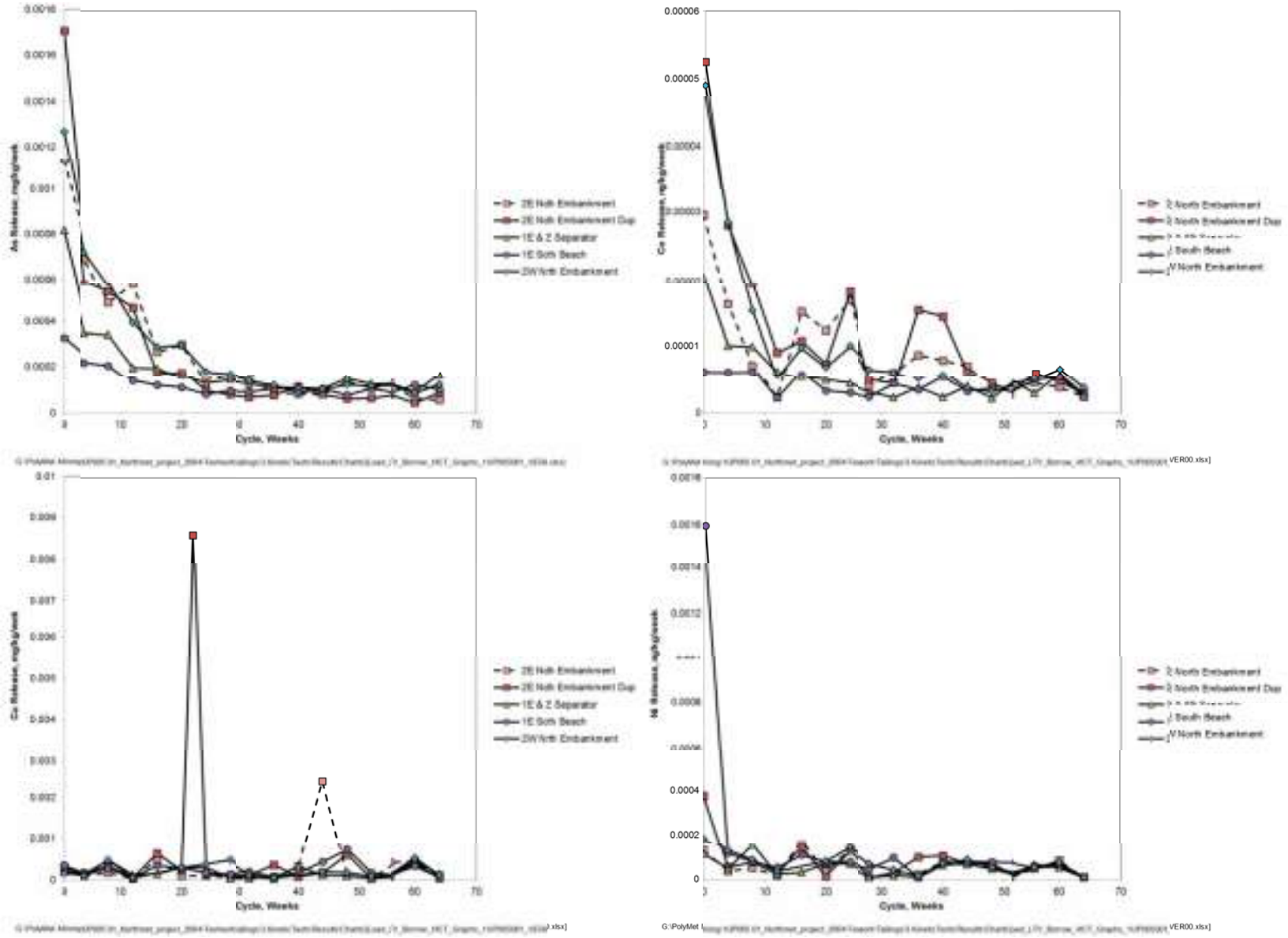

Figure 8: Trace Element Results for LTVSMC Humidity Cells

SRK Consulting (Canada) Inc.

Laura Donkervoort, M.Sc.
Staff Consultant (Geochemistry)

Stephen Day, M.Sc., P.Geo.
Corporate Consultant (Geochemistry)

Attachment 2-19

SRK Consulting. Update on Kinetic Test Data, NorthMet Project – DRAFT. Memorandum to Jim Scott, PolyMet Mining. October 19, 2012 (updated January 25, 2015).

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Memo

To:	Jim Scott	**Date:**	October 19, 2012
Company:	PolyMet Mining	**From:**	Stephen Day
Copy to:		**Project #:**	1UP005.001
Subject:	Update on Kinetic Test Data, NorthMet Project – DRAFT		

1 Introduction

Kinetic tests (mainly humidity cells) were initiated for the NorthMet Project in 2004 on samples of rock, simulated tailings and hydrometallurgical process residues according to overall program designs prepared following discussion between PolyMet Mining Inc. (PolyMet), Minnesota Department of Natural Resources (MDNR) and SRK Consulting (Canada) Inc. (SRK). Results from these tests have been used at various junctures of the project to develop waste management plans and evaluate project environmental effects. In 2009, the program was modified in consultation with MDNR to stop some tests and modify the frequency of analysis of leachates based on trend interpretation (SRK 2009).

The purpose of this memorandum is to provide an update on the progress of the testwork and transmit release rates as it relates to water quality source term predictions in the Environmental Impact Statement (EIS).

2 Data Presentation

Electronic updates to the data graphing and calculations are available on Barr Engineering's NorthMet Project website. The data period covered by the testwork now includes up to 6.5 years because some waste rock humidity cells were initiated in August 2005. The cutoff for interpreting data for this update was the end of January to early February 2012. Due to reduced analytical frequency for most tests (SRK 2009), complete QA review and update of the database occurs quarterly. Data collected subsequent to February 2012 completed QA after this update was prepared and are not included in this review.

3 Waste Rock, Lean Ore and Ore Kinetic Tests

3.1 Status of Program

The original waste rock humidity cell program consisted of 92 tests distributed according to the four initially defined classifications shown in Table 1. Waste rock was subsequently classified into three categories. The current program consists of 43 tests with most ongoing tasks in the mid-sulfur range for waste rock and lean ore (19 tests). Table 2 shows all samples and the testwork duration used to calculate rates for use in water quality predictions.

The primary uses of the data in the waste characterization program are to:

- Understand long term performance of wastes and inform definition of waste management categories based on sulfur content; and
- Provide leaching rates for use in source term geochemical predictions.

Table 1. Sample Categories

Initial Classification	Waste Category	Totals	Stopped	Continuing
S≤0.05%	1	27	20	7
S>0.05%	1	10	6	4
	2/3	14	3	11
	4	13	8	5
Lean Ore	1	4	3	1
	2/3	13	5	8
	4	8	4	4
Ore	Not applicable	3	0	3
Totals	All	92	49	43

Source: G:\PolyMet Mining\1UP005.01_Northmet_project_2004\Testwork\WR_KT\Characteristics\Humidity_Cell_Table\[1UP005.001_WR_Outcomes_ver01.xlsx]Testwork Tallies

Results are described below in the context of these two objectives. Each heading describes results according to the waste categories (1, 2/3 and 4). Charts for indicator parameters are shown to indicate major features (Figures 1 to 4). Overall tendency toward acidic conditions is shown by pH. Sulfate indicates sulfide mineral oxidation rates. Nickel is shown because as observed in this test program and testwork performed previously by DNR it (along with cobalt) tends to respond first to declining pH. Copper is shown because it leaches more rapidly as pH declines further. Arsenic provides an indication of how a heavy oxyanion is affected by changes in pH.

In order to obtain average leaching rates, typical trends have been recognized and average rates calculated for common qualitative leaching features or “conditions” as summarized in Table 3. In these definitions, “stable” means typically neither clearly increasing or decreasing. Due to variability in the testwork results, the definitions are not strict and require some flexibility in interpretation. For example, the transition from Condition 1 to 2 may not occur exactly at pH 7.

Condition 2 is typically accompanied by increasing release of cobalt and nickel which may continue into Condition 3. Lower pH (less than 6) typically occurs during Condition 3 and 4. In some cases, pH recovery is observed during Condition 4 as oxidation rates decrease.

Table 2. List of Rock Humidity Cells

HCT ID	Comment	Original Waste Type	Geological Unit	Rock Type	Category	Sample ID	S %	Initial Date	Data Record Length weeks	Duration Used in Current Modeling[1] weeks
1	9 Dup	RWR	1	Anorthositic	1	99-320C(830-850)	0.09	8/8/2005	337	284
3		RWR	1	Anorthositic	1	00-361C(345-350)	0.05	8/8/2005	337	284
26		NRWR	2	Anorthositic	1	00-366C(185-205)	0.02	8/9/2005	198	198
27		NRWR	2	Anorthositic	1	00-366C(230-240)	0.02	8/9/2005	198	198
28		NRWR	2	Anorthositic	1	99-320C(165-175)	0.03	8/9/2005	198	198
40		NRWR	3	Anorthositic	1	00-334C(30-50)	0.02	8/9/2005	337	284
41	37 Dup	NRWR	3	Anorthositic	1	00-368C(125-145)	0.04	8/10/2005	337	284
42		NRWR	3	Anorthositic	1	00-368C(20-40)	0.04	8/10/2005	198	198
13		NRWR	1	Troctolitic	1	00-340C(595-615)	0.04	8/10/2005	198	198
14		NRWR	1	Troctolitic	1	00-334C(580-600)	0.06	8/15/2005	336	284
15		RWR	1	Troctolitic	1	00-334C(640-660)	0.07	8/8/2005	337	284
16		RWR	1	Troctolitic	1	00-347C(795-815)	0.07	8/8/2005	198	198
29		NRWR	2	Troctolitic	1	99-318C(250-270)	0.04	8/9/2005	198	198
30		NRWR	2	Troctolitic	1	00-373C(95-115)	0.04	8/9/2005	198	198
31		NRWR	2	Troctolitic	1	00-373C(75-95)	0.06	8/9/2005	198	198
32		RWR	2	Troctolitic	1	00-357C(110-130)	0.08	8/9/2005	198	198
33		RWR	2	Troctolitic	1	99-320C(315-330)	0.07	8/9/2005	337	284
43		NRWR	3	Troctolitic	1	00-366C(35-55)	0.02	8/10/2005	198	198
44		NRWR	3	Troctolitic	1	00-334C(110-130)	0.04	8/10/2005	198	198
45		NRWR	3	Troctolitic	1	00-347C(155-175)	0.06	8/10/2005	198	198
46		RWR	3	Troctolitic	1	00-347C(280-300)	0.06	8/10/2005	198	198
49		NRWR	4	Troctolitic	1	00-367C(50-65)	0.03	8/10/2005	198	198
50		NRWR	4	Troctolitic	1	00-367C(260-280)	0.04	8/10/2005	198	198
51	57 Dup	NRWR	4	Troctolitic	1	00-367C(290-310)	0.04	8/10/2005	337	284
52		RWR	4	Troctolitic	1	00-370C(20-30)	0.08	8/10/2005	198	198
56		NRWR	5	Troctolitic	1	26064(44-54)	0.02	8/10/2005	337	284
59		NRWR	5	Troctolitic	1	26064(264+146-269+156)	0.06	8/10/2005	337	284
60		NRWR	6	Troctolitic	1	26056(110-125)	0.04	8/10/2005	198	198
74		NRWR	1	Troctolitic	1	26029(815-825)	0.02	9/8/2005	194	194
78		NRWR	6	Troctolitic	1	26056(135-153)	0.05	9/22/2005	331	278
99		LeanOre	1	Troctolitic	1	00-326C(250-265)	0.08	10/28/2005	187	186
21		RWR	1	Ultramafic	1	00-357C(335-340)	0.08	8/9/2005	198	198
35		NRWR	2	Ultramafic	1	00-368C(460-465)	0.06	8/9/2005	198	198
36		NRWR	2	Ultramafic	1	26055(940-945)	0.06	8/9/2005	198	198
39		NRWR	2	Ultramafic	1	26098+00-337C	0.1	8/9/2005	198	198
72	61 Dup	LeanOre	2	Ultramafic	1	00-361C(240-245)	0.06	8/11/2005	337	284
101		LeanOre	2	Ultramafic	1	26039(310-315)	0.06	10/28/2005	187	186
2		RWR	1	Anorthositic	2/3	00-361C(310-320)	0.18	8/8/2005	337	284
103		LeanOre	1	Anorthositic	2/3	99-320C(400-405)	0.18	10/28/2005	326	273
5		RWR	1	Sed Honfels	2/3	26030(1047-1052)	0.24	8/8/2005	337	284
6		RWR	1	Sed Honfels	2/3	26061(1218-1233)	0.44	8/8/2005	337	284
7		RWR	1	Sed Honfels	2/3	00-340C(990-995)	0.55	8/8/2005	337	284
17		RWR	1	Troctolitic	2/3	00-350C(580-600)	0.19	8/8/2005	337	284
18		RWR	1	Troctolitic	2/3	00-327C(225-245)	0.44	8/8/2005	198	198
34		RWR	2	Troctolitic	2/3	00-369C(335-345)	0.18	8/9/2005	337	284
47		RWR	3	Troctolitic	2/3	00-326C(60-70)	0.14	8/10/2005	337	284
48	38 Dup	RWR	3	Troctolitic	2/3	00-369C(305-325)	0.25	8/10/2005	198	198
53		RWR	4	Troctolitic	2/3	00-369C(20-30)	0.21	8/10/2005	337	284
54		RWR	4	Troctolitic	2/3	00-367C(170-175)	0.51	8/10/2005	337	284
71		LeanOre	2	Troctolitic	2/3	00-340C(380-390)	0.15	8/11/2005	198	198
75		LeanOre	3	Troctolitic	2/3	26049+26030	0.59	8/11/2005	198	198
77		LeanOre	5	Troctolitic	2/3	26056(302-312)	0.23	8/11/2005	337	284
80		LeanOre	6	Troctolitic	2/3	26142(360+345-365+350)	0.18	8/11/2005	337	284
96		LeanOre	2	Troctolitic	2/3	99-318C(325-330)	0.17	10/28/2005	326	273
98		LeanOre	5	Troctolitic	2/3	26056(282-292)	0.32	10/28/2005	187	186
100		LeanOre	1	Troctolitic	2/3	00-340C(910-925)	0.36	10/28/2005	326	273
102		LeanOre	1	Troctolitic	2/3	00-331C(190-210)	0.42	10/28/2005	326	273
105		LeanOre	3	Troctolitic	2/3	00-367C(495-500)	0.28	10/28/2005	326	273
22		RWR	1	Ultramafic	2/3	00-326C(680-685)	0.30	8/9/2005	198	198
23		RWR	1	Ultramafic	2/3	00-357C(535-540)	0.2	8/9/2005	337	284
94		LeanOre	1	Ultramafic	2/3	00-344C(630-635)	0.34	10/28/2005	187	186
95		LeanOre	1	Ultramafic	2/3	00-326C(495-505)	0.16	10/28/2005	187	186

[1] Waste Characterization Data Package Version 9 (July 3 2012).

HCT ID	Comment	Original Waste Type	Geological Unit	Rock Type	Category	Sample ID	S %	Initial Date	Data Record Length weeks	Duration Used in Current Modeling[1] weeks
104		LeanOre	2	Ultramafic	2/3	00-326C(225-235)	0.12	10/28/2005	326	273
4		RWR	1	Anorthositic	4	00-343C(240-250)	0.68	8/8/2005	198	198
65		LeanOre	1	Anorthositic	4	26027(616-626)	1.83	8/11/2005	337	284
93		LeanOre	1	Anorthositic	4	00-331C(255-260)	0.86	10/28/2005	326	273
8		RWR	1	Sed Honfels	4	00-340C(965-974.5)	1.74	8/8/2005	198	198
11		RWR	1	Sed Honfels	4	26043+26027	2.47	8/8/2005	337	284
68		LeanOre	1	Sed Honfels	4	26062+26026	4.46	8/11/2005	337	284
106		LeanOre	1	Sed Honfels	4	26058(704-715)	1.46	10/28/2005	326	273
19		RWR	1	Troctolitic	4	00-371C(435-440)	0.88	8/8/2005	337	284
20	10 Dup	RWR	1	Troctolitic	4	00-340C(765-780)	1.68	8/8/2005	337	284
55		RWR	4	Troctolitic	4	00-367C(395-400)	0.77	8/10/2005	198	198
69		LeanOre	1	Troctolitic	4	00-340C(725-745)	0.91	8/11/2005	198	198
76		LeanOre	4	Troctolitic	4	00-367(400-405)	1.37	8/11/2005	198	198
24		RWR	1	Ultramafic	4	99-318C(725-735)	0.72	8/9/2005	198	198
25		RWR	1	Ultramafic	4	99-317C(460-470)	1.24	8/9/2005	198	198
70		LeanOre	1	Ultramafic	4	00-344C(515-520)	1.2	8/11/2005	198	198
97		LeanOre	1	Ultramafic	4	00-330C(275-280)	0.75	10/28/2005	187	186
62		RWR	20	Virginia	4	00-361C(737-749)	2	8/11/2005	337	284
63	58 Dup	RWR	20	Virginia	4	00-364C(210-229)	3.79	8/11/2005	198	198
64		RWR	20	Virginia	4	00-337C(510-520)	5.68	8/11/2005	198	198
66		Ore			-	P10	0.86	9/8/2005	333	268
67		Ore			-	P20	0.9	9/8/2005	333	268
73		Ore			-	P30	0.86	9/8/2005	333	268

Source: G:\PolyMet Mining\1UP005.01_Northmet_project_2004\Reports\2012-06_RS82_Kinetic Update\Tables\[RateCompilation_1UP005001_SJD_VER03_20110204_NoLinks.xlsx]

Table 3. Qualitative Definition of Leaching Conditions

Condition	Trend Features
1	pH>7, stable SO_4
2	pH<7, typically stable SO_4
3	SO_4 sharply increasing and unstable
4	SO_4 peaked and decreasing

3.2 Description of Results by Waste-Rock Caregory

3.2.1 Category 1

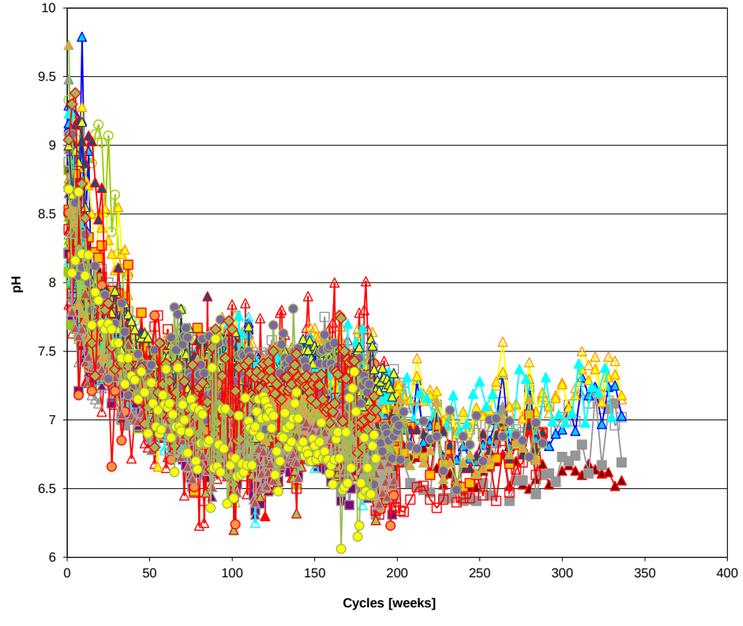
Results for pH, sulfate, nickel, copper and arsenic are shown in Figure 1.

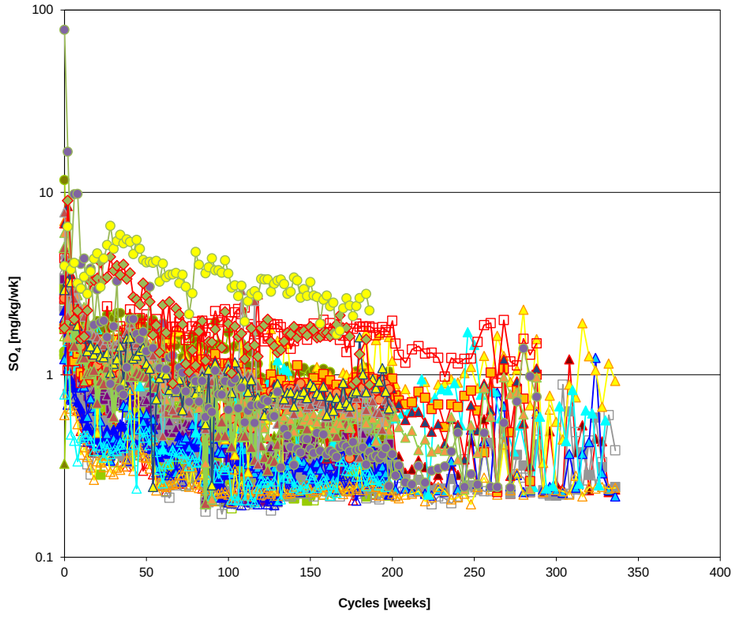
All Category 1 humidity cells have yielded pH above 6 throughout the program. Generally, pHs declined as the tests proceeded. Initial pHs in most cases were above 8 but declined rapidly and have typically fluctuated between 6.5 and 7.5. There is no indication for the 12 continuing tests that pHs are on a declining trend. In a few cases, pHs have recovered slightly. At the same time, the alkalinity trend is stable.

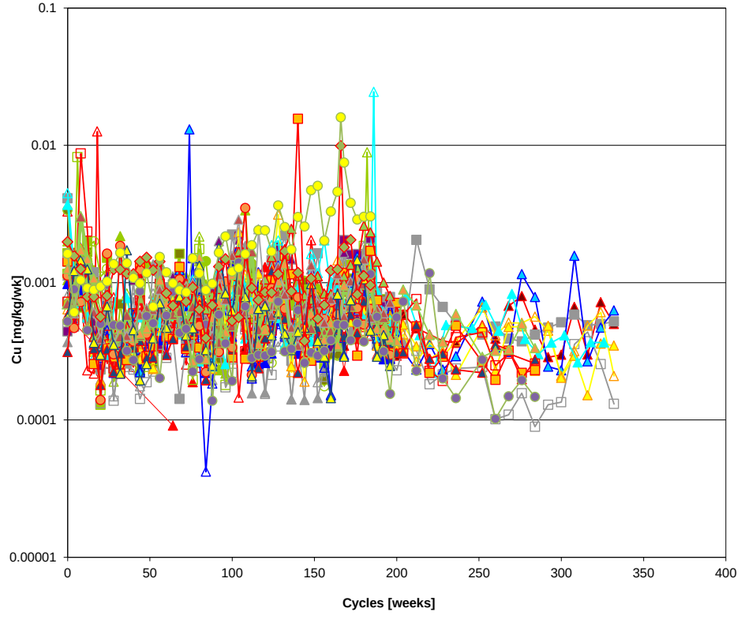
Sulfate leaching rates have been low throughout the program with most continuing tests showing rates below 1 mg/kg/week. At these low rates, variability in trends is apparent but no clear upward trends have been apparent.

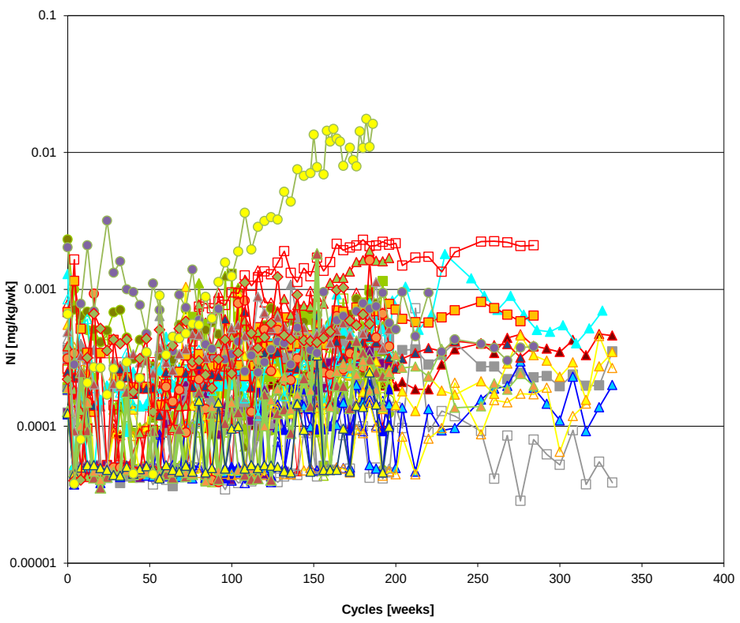
One humidity cell (26039-310-315) showed pronounced accelerated nickel leaching as pH decreased below 7. This sample is ultramafic rock and classified as lean ore (nickel content of 0.1%). A few other samples showed increased nickel leaching in response to pH but peak rates were an order-of-magnitude below this sample. Similar results were obtained from cobalt. In contrast, copper only showed a modest increase in leaching rates for sample 26039-310-315.

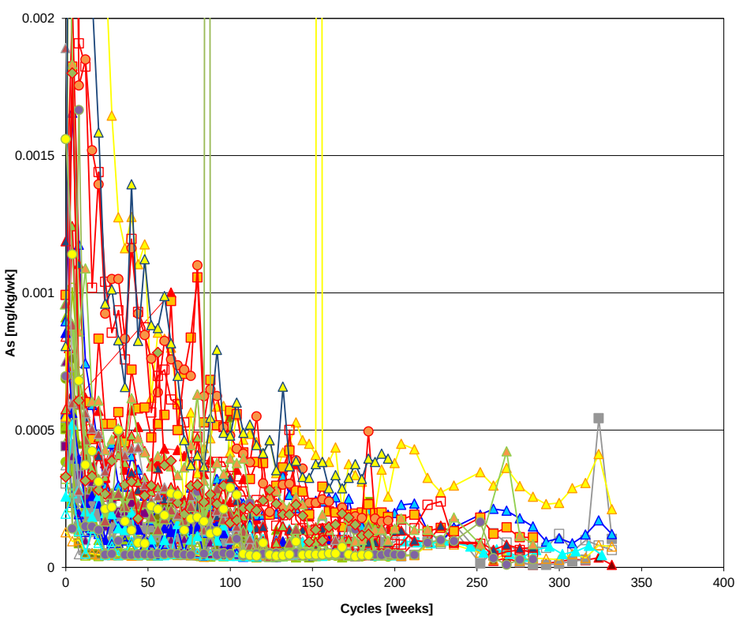
Arsenic showed no increase in leaching rates due to pH changes. Arsenic leaching has either stabilized or decreased steadily.











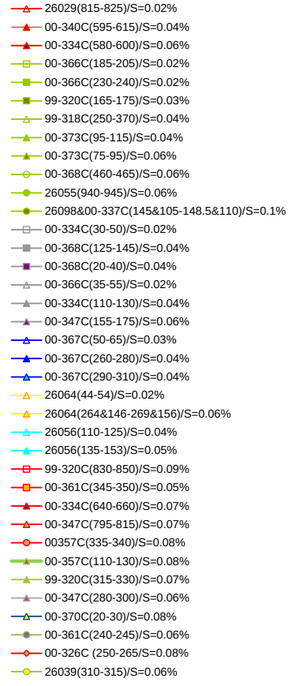


Figure 1. Leaching Trends for Category 1 Samples

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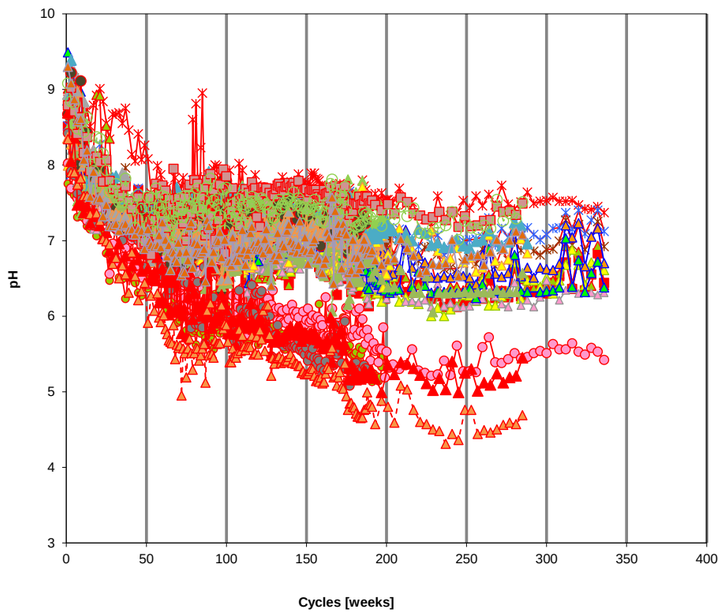

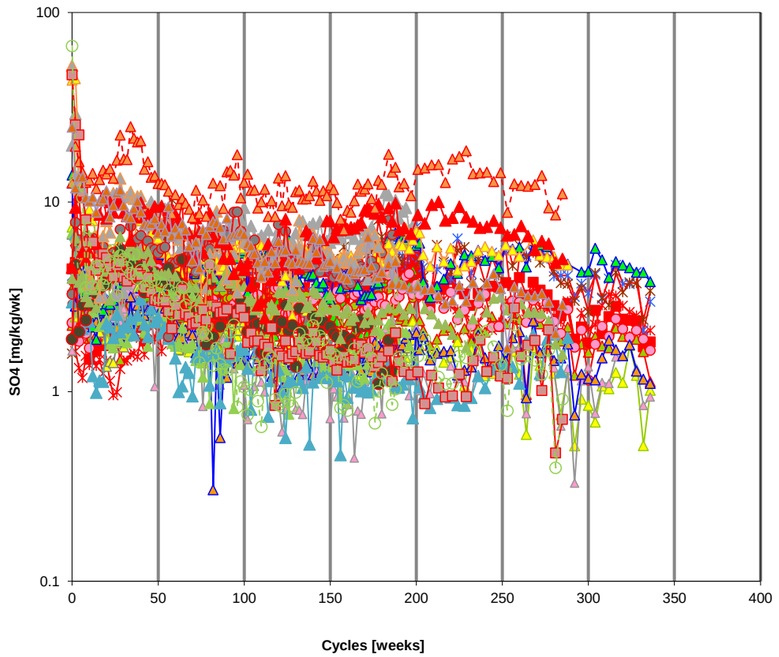

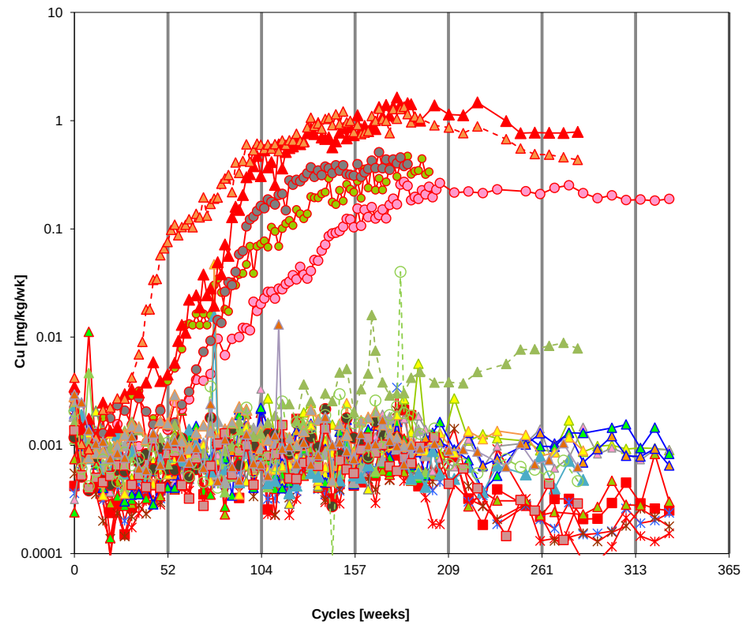

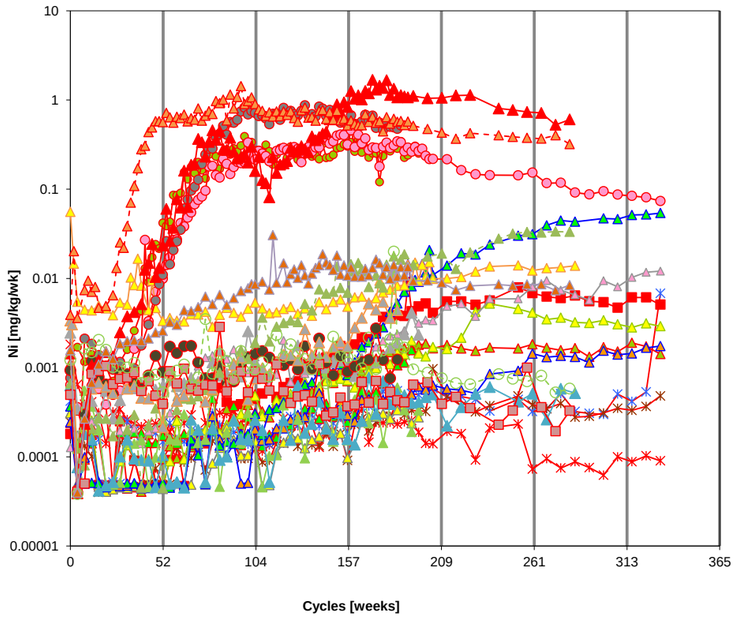

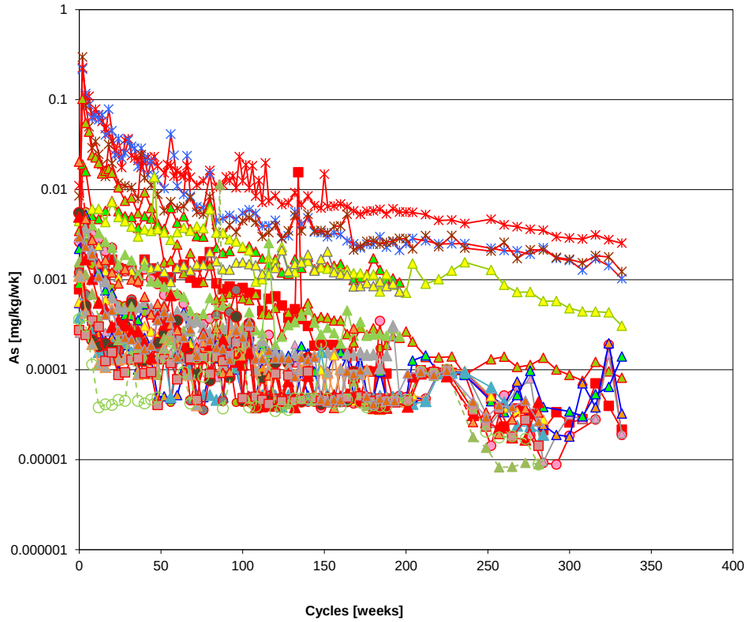

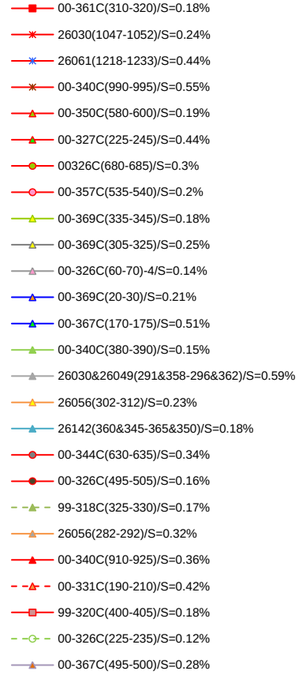

Figure 2. Leaching Trends for Category 2/3 Samples

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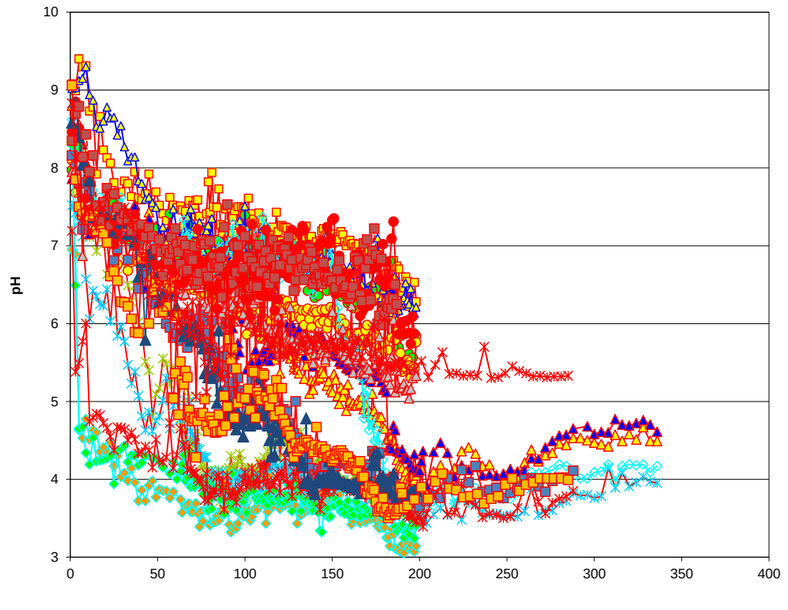

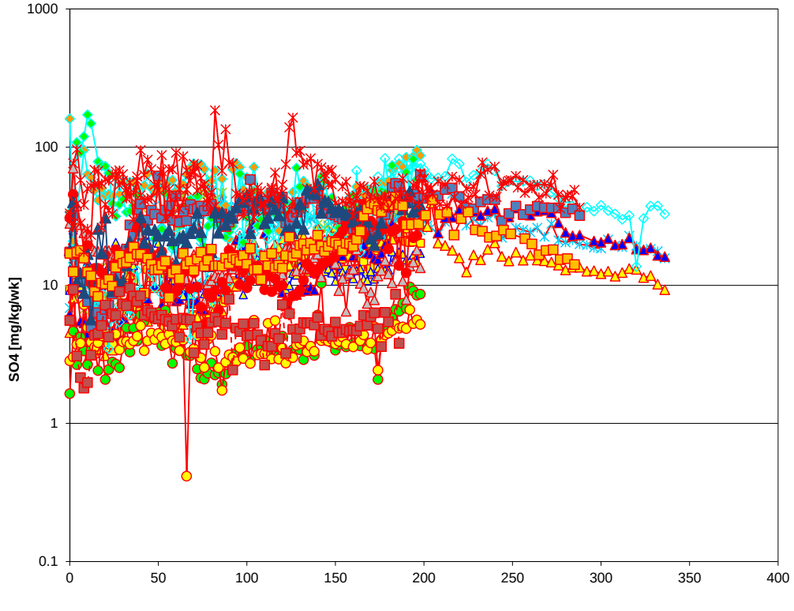

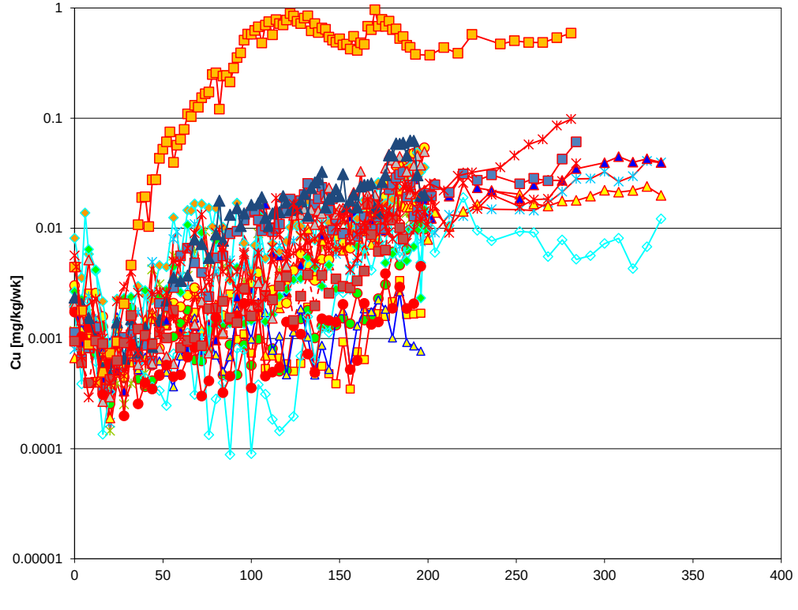

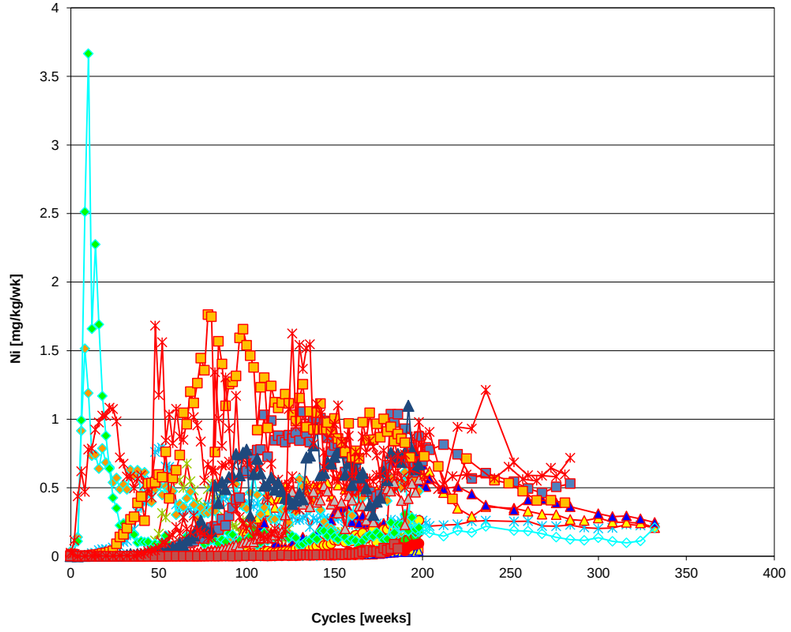

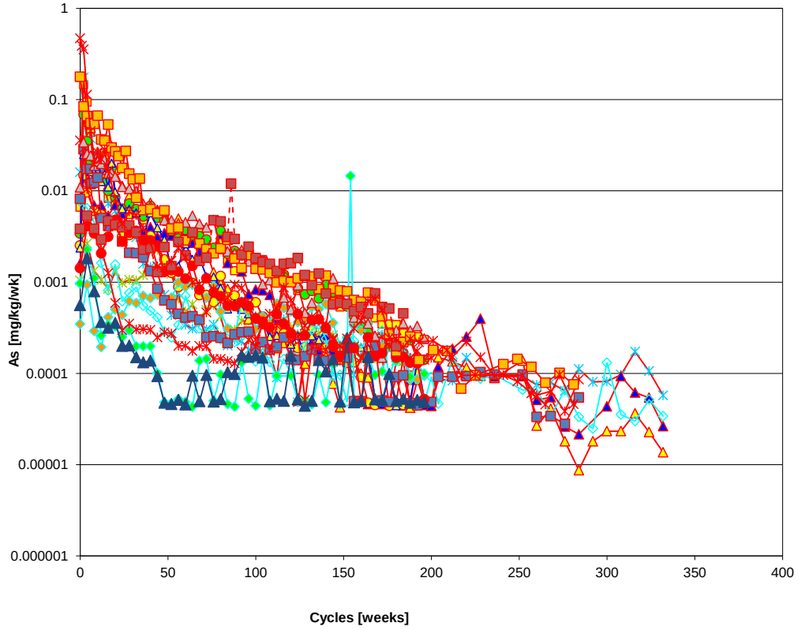

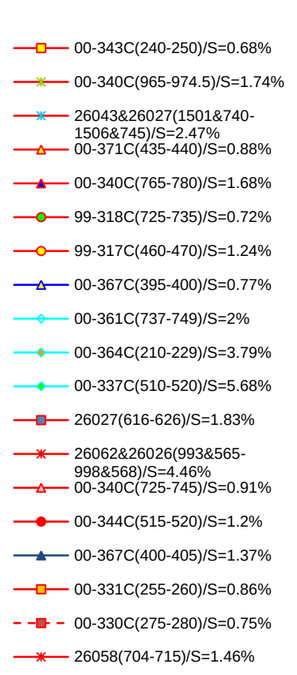

Figure 3. Leaching Trends for Category 4 Samples

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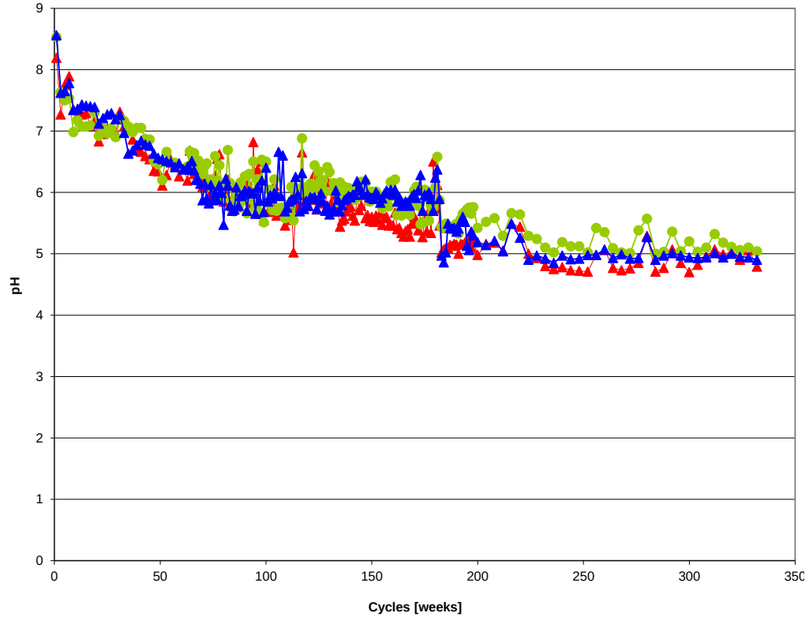

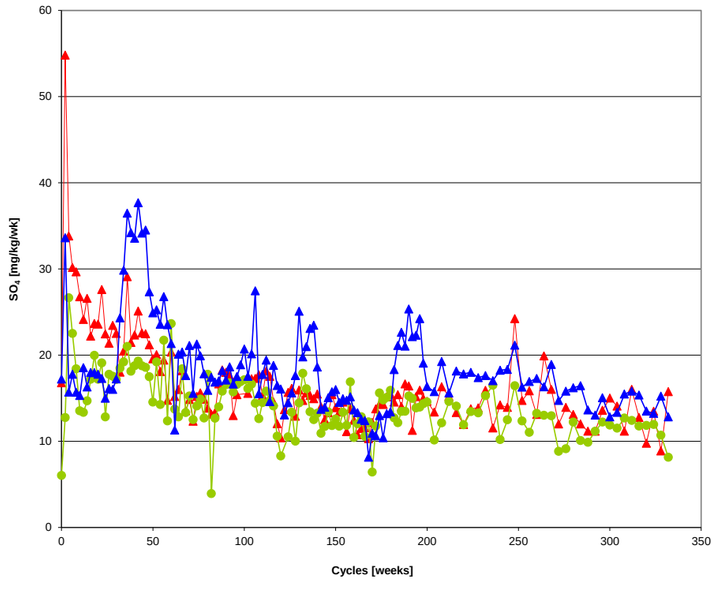

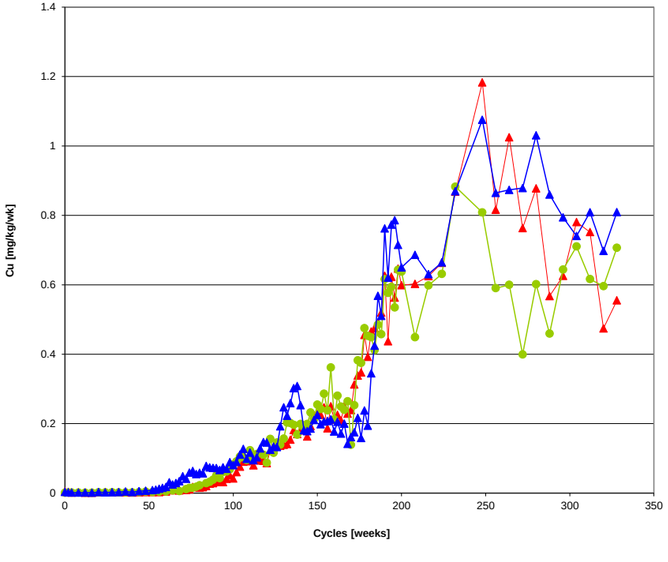

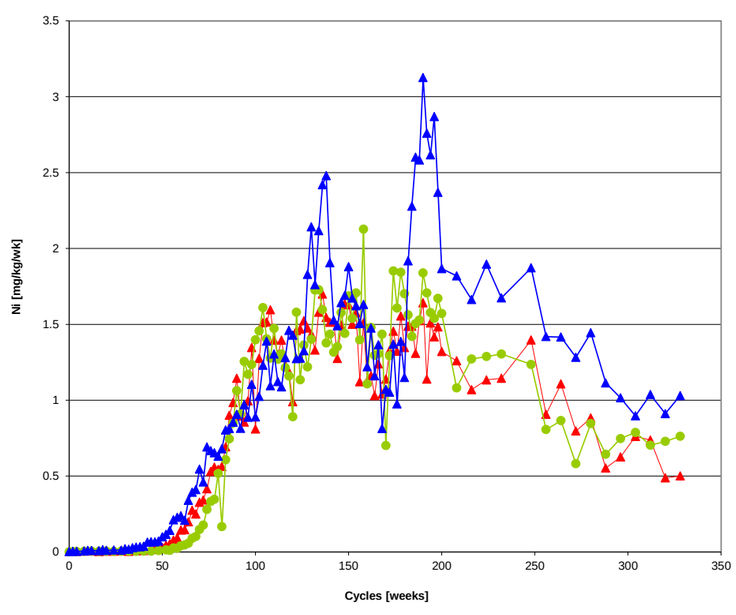

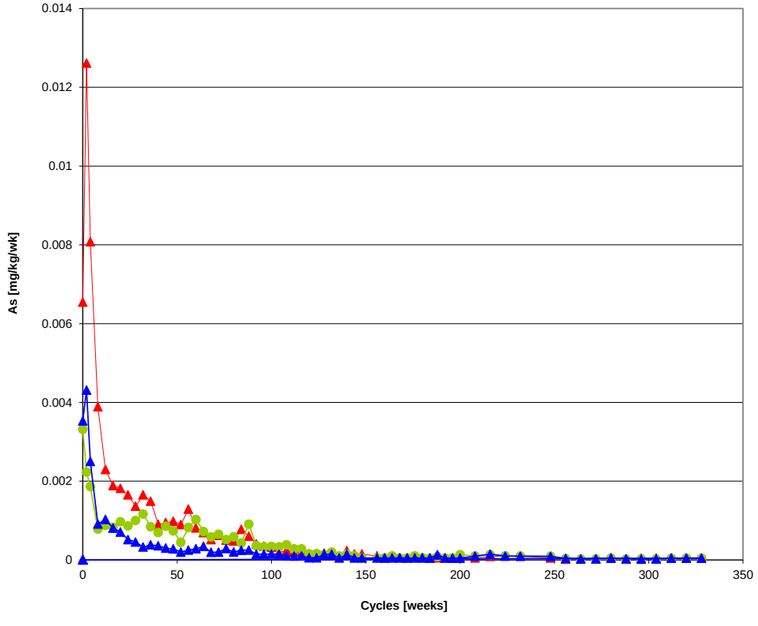

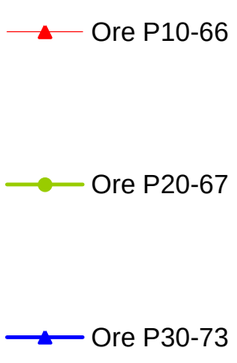

Figure 4. Leaching Trends for Ore Composite Samples

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Testing on Category 1 samples has confirmed to date that alkalinity generated by weathering of silicate minerals is able to result in pH above the pH of de-ionized water. This indicates that the alkalinity is consuming acidity introduced by both the dissolved CO_2, introduced by the deionized water (pH 5 to 6), and acidity produced by oxidation of sulfide minerals (locally pH<4). The steady maintenance of pHs above 6 shows that this is a steady long term mechanism for preventing pH depression below 6 in humidity cells.

3.2.2 Category 2/3

Results for pH, sulfate, nickel, copper and arsenic are shown in Figure 2.

Five samples (including three of the continuing 19 samples) showed pH depression below 6 while all other samples have shown stable pHs above 6 with similar trends to the Category 1 samples. Only one of the six samples has shown pH below 5. The sulfur content of the five samples varied from 0.2% to 0.42%. The sample with 0.2% sulfur showed the highest pH in this set at near 5.5. The sulfur contents of the other four samples were 0.3% and higher.

Sulfate release for Category 2/3 samples varied between 1 and 10 mg/kg/week with the highest rates being observed for samples containing higher sulfur contents. While sulfate release has varied, there was a lack of consistent upward trends.

A number of samples showed upwards trends in nickel and cobalt were related to pH decrease below 7. For the five samples showing distinctive pH depression, nickel release increased steeply during the first year of testing and subsequently showed gentle decline. Other samples showed slower nickel increase later in the program due to slower decline in pH. The same five samples also showed increase in copper leaching though the increase was less rapid than nickel and decline occurred later or not at all. One other sample (99-318C-325-330) showed accelerated copper leaching. Leachate pH was at the low end of those samples not showing substantial pH depression. It is a lean ore sample containing 0.1% copper.

Arsenic leaching trends were similar to Category 1. Rates have trended downwards. Higher rates are associated with samples yielding higher leachate pHs.

3.2.3 Category 4

Results for pH, sulfate, nickel, copper and arsenic are shown in Figure 3. Eight tests are continuing, two of which are Duluth Complex troctolite samples (00-371C(435-440) and 00-340C(765-780)), two are Duluth Complex Anorthositic Troctolites (00-331C (255-260) and 26027(616-626)), three are sedimentary hornfels samples (26058 (704-715), 26043&26027(740&1501-745&1506)) and DDH-26062&26026(993&565-998&568)-68) and one is Virginia Formation (00-361C(737-749)).

All tests have shown pH depression to some degree with pHs below 4. Of the continuing tests, one sedimentary hornfels sample (26058-704-715) showed the least depression with stable pHs near 5.3. The other samples showed leachate pHs near or below 4 and have recently shown pH recovery from minimum values. The two troctolite samples have shown pHs above 4.5 following a steep decline to pH near 4 earlier in the program. The other two Duluth Complex samples, Virginia Formation sample and other sedimentary hornfels samples have shown weaker pH recovery. Stabilization at pHs between 4 and 5 are indicative of buffering by reaction of acidity with alumino-silicate minerals resulting in formation of secondary aluminum minerals.

Some increases in sulfate release have been observed as pH decreased. Peak sulfate release rates have typically been less than a factor of five times lowest levels earlier in the test and peaks are not sharp but erratic. Decline in sulfate release has been observed following the peak. These declines are rapid at first then lessen as shown by straight lines on the log axis plot.

Due to the consistent pH depression below 7, all samples showed accelerated nickel leaching followed by declining leaching after reaching a subdued peak. All continuing tests have shown this decline parallels the sulfate release trend implying that nickel release is derived from sulfide minerals. Cobalt showed similar trends.

Upward trends in copper release were also apparent for all tests but as observed for acidifying Category 2/3 samples, the increase in copper release occurred later than nickel. Also, peaks followed by downward trends in copper release were less apparent for nickel and most ongoing tests have shown increase in copper release rather than a decrease paralleling nickel and sulfate release.

As with all other tests, arsenic release followed a declining trend.

3.2.4 Ore Composites

Results for pH, sulfate, nickel, copper and arsenic are shown in Figure 4.

Leachate pH trends for the three samples are nearly identical and show pH has been stable near 5 for over 2 years. Likewise, sulfate release has been similar for all three tests and has not shown consistent upward or downward trends. As pH decreased, nickel release accelerated, reaching peak rates after about three years then declining. Composite P30 showed a marked separate pH after 4 years that was not apparent in the other tests. A sulfate release peak was observed about the same time though was less pronounced compared to the other two tests.

Copper release also increased as the tests progressed but the upward trend was delayed compared to nickel and reached peak values after more than 4 years of testing. Copper release has subsequently declined.

3.3 Comparison Tests

Five samples tested as four size fractions continue to be tested in parallel with the humidity cell program. These tests are comparing differences in the weathering characteristics of the samples as well as differences in testing configuration (ASTM Humidity Cell and DNR Reactor) Samples are classified as Category 1 (two samples), Category 2/3 (two samples) and Category 4 (1 samples).

Both Category 1 samples have yielded leachate pHs near 7 in all size fractions and flat sulfate release trends. Sulfate release continues to be a function of particle size with lowest release (when detected) coming from the coarser size fractions. Similar findings apply to metal release. Nickel release is greater for the finer size fractions.

Consistent with the conventional humidity cells for Category 2/3 samples, lower pHs were observed but due to the sulfur content near 0.2% for both samples, pHs have remained above 6 and the samples continue to generate alkalinity particularly from the finer size fractions. Sulfate release was variable for both samples and no trends were apparent. Difference between sulfate releases in size fractions was most apparent for one sample (00-361C-310-320) with the fine fractions releasing higher sulfate than the coarse fraction. Both samples showed accelerated nickel leaching from all fractions as pH decreased below 7. Differences between size fractions were strongest for 00-361C-310-320. Temporal trends in copper release were not apparent but copper release rates were greater for the finer size fractions.

Virginia Formation rock classified as Category 4 showed sharp decline in pH to near and below 3.5 in all size fractions. Peak sulfate release was observed around 4 years for the two fine fractions. Peak sulfate release rates for the coarsest fractions have not observed because rates began increasing after six years. These differences in timing were not linked to greater pH depression for the fine fractions, in fact, the finer fractions yielded higher pHs.

These tests have shown that:

- Testing of different size fractions to date has shown that samples generating acidic leachate did so regardless of size fraction tested and protocol used.
- Rates of sulfate and metal release are generally higher for the finer size fractions presumably reflecting differences in surface area.
- For one sample that became strongly acidic, the finer fractions showed more rapid sulfate release earlier than the coarse fractions presumably reflecting faster oxidation and greater liberation of sulfide minerals.

3.4 Duplicate Testwork

Following observation of good reproducibility of trends previously (SRK 2009), all seven standard humidity cell tests being run as duplicates were stopped as part of reduction of the program in consultation with the DNR. Duplicate DNR reactor tests on two samples, and duplicate humidity cells containing size fractions for two samples have continued for a total of four duplicates (eight tests).

The two humidity cell-style duplicates are continuing to show strong reproducibility for the major ions and trace elements of interest (arsenic cobalt, copper, nickel). Weak reproducibility was apparent for barium (one sample).

The DNR-style reactors showed good reproducibility for one sample and poor reproducibility of pH for the other sample. The difference in pH (near 6 compared to 6.5) appeared to be due to differences in oxidation rate (faster for lower pH) which also resulted in greater leaching rates for cobalt and nickel.

3.5 Interpretation

3.5.1 Effect on Sulfur Management Criteria

Figure 5 shows minimum leachate pH as a function of total sulfur content for all humidity cell tests. Lowest pHs are observed for samples containing highest sulfur concentrations (Virginia Formation and Sedimentary Hornfels). The lowest sulfur concentration showing pH depression below 5 (i.e. below the range of pH of deionized water used in the test) was 0.36%. Samples with sulfur content below the Category 1 threshold of 0.12% have shown leachate pH above 6 indicating the threshold is appropriate based on current data.

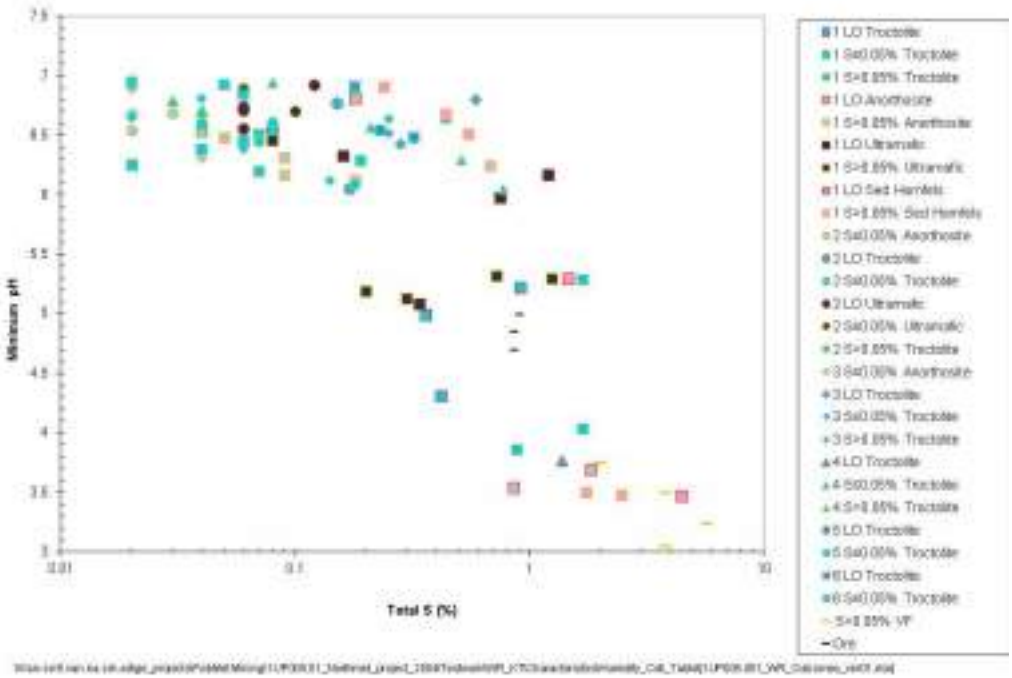

Figure 5. Minimum Leachate pH as a Function of Total Sulfur Content

3.5.2 Effect on Dissolution Rates Used to Estimate Water Quality

The condition definitions shown in Table 3 were used to assess trends for each test.

An example for one test showing all four conditions is provided in Figure 6. Condition 1 lasted for 39 weeks and was marked higher pHs and relatively stable SO_4 following an initial flush in the first few weeks. As pHs declined below 7, sulfate in this case become more variable and leaching of nickel accelerated. Condition 3 began at week 164 and was marked by steep decline in pH to 3.5 and rapidly increasing sulfate release paralleled by nickel. Condition 4 began at week 194 when sulfate release peaked and then began to declining.

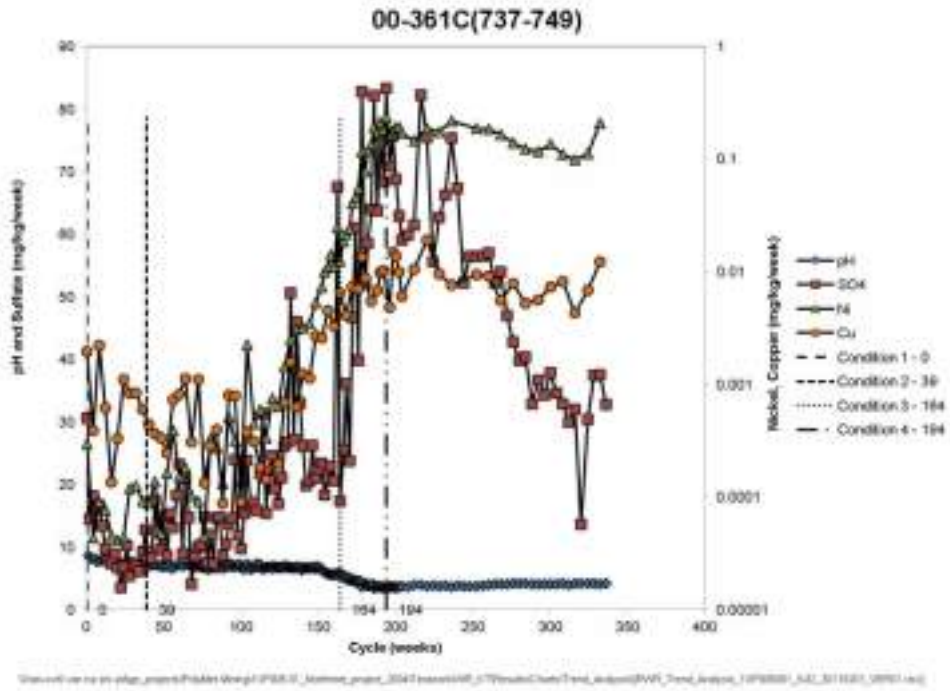

Figure 6. Example of Condition Assignment for a Waste Rock Humidity Cell Test

Resulting rates for each condition have been provided to Barr Engineering and include data collected to late January and early February 2012. These may be used as the basis for various inputs to source term predictions for the water quality model. However, the updated rate table primarily adds a few rates for samples that transitioned to acidic conditions. The majority of samples leaching under non-acidic conditions continue to show stable rates that have not changed significantly. As a result of this finding, there is likely little incremental value in modifying the rates used as inputs to the water quality modeling.

4 Tailings Kinetic Tests

4.1 Status of Program

A review of the tailings program was provided recently (SRK 2011). All tests have continued since findings were reported resulting in about six months of additional data. As this represents less than 15% additional time for most tests (with the exception of tests started in 2009), this memorandum provides a limited review to indicate whether any significant changes have occurred since the previous review.

4.2 Description of Results

4.2.1 NorthMet Project Tailings

Table 3 lists all tailings humidity cells. The test program consists of conventional humidity cells with parallel tests using the MDNR reactor configuration on bulk tailings and tailings size fractions.

Tests started in 2005 and 2006 (Pilot Plant 1) have continued to the same trends reported by SRK (2011) including stable or increasing pH, decreasing sulfate and stable or decreasing metal concentrations.

Tests started in 2008 (Pilot Plants 2 and 3) have reached stable pH with no indication of trends below pH 7 as shown in some samples produced by Pilot Plant 1. Sulfate release has stabilized for all tests. Nickel and cobalt release have not shown the same trends as Pilot Plant 1 samples but have shown stable concentrations in leachates below 0.002 mg/L and 0.0001 mg/L, respectively. PP2 +100 mesh showed an arsenic spike to 0.08 mg/L at week 168 following repair of the cell to address rapid drainage of water during the leach cycle.

Tests started in 2009 (scavenger tailings samples) have shown downwards trends in pH with lowest pHs typically above 7. At the same time, sulfate release has also trended downward and for some tests has stabilized. Nickel and cobalt release trended downward initially for all tests. Upwards trends for both elements were most apparent for samples Oct 1/09 09:00 (-100+200 mesh) which also showed some pHs below 7.

An arsenic spike resembling those for the 2008 samples discussed above was also apparent for two cells. These spikes were correlated with a large number of spikes in other parameters including aluminum, iron and silicon. It appears the cell malfunctioned perhaps allowing solids to be present in the leachates and thereby causing an increase in the dissolved fraction due to the presence of colloidal matter.

Table 4: Tailings Humidity Cells Used as Basis for Update Report

HCT ID	Fraction	HCT Full ID	Total Sulfur (%)	Start Date	Total Duration weeks	Duration Used for Current Modeling[2] weeks
T1	Whole	P1 (CuSO4)	0.1	9/8/2005	333	--
T2	Whole	P1 (no CuSO4)	0.23	9/8/2005	333	--
T3	Whole	P2 (no CuSO4)	0.2	9/8/2005	333	--
T4	Whole	P3 (CuSO4)	0.15	9/8/2005	333	--
T5	100	Parcel 2 P2S +100 mesh	0.15	2/10/2006	311	--
T6	100	Parcel 2 P2S -100 +200 mesh	0.17	2/10/2006	311	--
T7	-200	Parcel 2 P2S -200 mesh	0.24	2/10/2006	311	--
T8	100	Parcel 1-2 PISCS +100 mesh	0.11	2/10/2006	311	271
T9	100	Parcel 1-2 PISCS -100 +200 mesh	0.1	2/10/2006	311	271
T10	-200	Parcel 1-2 PISCS -200 mesh	0.09	2/10/2006	311	271
T11	100	Parcel 3 P3S +100 mesh	0.11	2/10/2006	311	271
T12	100	Parcel 3 P3S -100 +200 mesh	0.14	2/10/2006	311	271
T13	-200	Parcel 3 P3S -200 mesh	0.14	2/10/2006	311	271
T52	Whole	Pilot Plant 2 (0.30 Cu feed) Composite Bulk Tailings (as rec'd)	0.07	7/8/2008	185	--
T53	100	Pilot Plant 2 (0.30 Cu feed) Composite (+100 mesh)	0.08	7/8/2008	185	146
T54	100	Pilot Plant 2 (0.30 Cu feed) Composite (-100 + 200 mesh)	0.06	7/8/2008	185	146
T55	-200	Pilot Plant 2 (0.30 Cu feed) Composite (-200 mesh)	0.09	7/8/2008	185	146
T56	Whole	Pilot Plant 3 (0.25 Cu feed) Composite Bulk Tailings (as rec'd)	0.08	7/8/2008	185	--
T57	100	Pilot Plant 3 (0.25 Cu feed) Composite (+100 mesh)	0.1	7/8/2008	185	146
T58	100	Pilot Plant 3 (0.25 Cu feed) Composite (-100 + 200 mesh)	0.08	7/8/2008	185	146
T59	-200	Pilot Plant 3 (0.25 Cu feed) Composite (-200 mesh)	0.08	7/8/2008	185	146
T60	Whole	SCAV TAILS 9/30/09 1600	0.09	11/24/2009	113	--
T61	100	SCAV TAILS 9/30/09 1600 +100	0.1	11/24/2009	113	74
T62	100	SCAV TAILS 9/30/09 1600 -100+200	0.09	11/24/2009	113	74
T63	-200	SCAV TAILS 9/30/09 1600 -200	0.11	11/24/2009	113	74
T64	Whole	SCAV TAILS 10/1/09 09.00	0.13	11/24/2009	113	--
T65	100	SCAV TAILS 10/1/09 09.00 +100	0.11	11/24/2009	113	74
T66	100	SCAV TAILS 10/1/09 09.00 -100+200	0.14	11/24/2009	113	74
T67	-200	SCAV TAILS 10/1/09 09.00 -200	0.14	11/24/2009	113	74
T68	Whole	SCAV TAILS 10/1/09 17.00	0.12	11/24/2009	113	--
T69	100	SCAV TAILS 10/1/09 17.00 +100	0.1	11/24/2009	113	74
T70	100	SCAV TAILS 10/1/09 17.00 -100+200	0.1	11/24/2009	113	74
T71	-200	SCAV TAILS 10/1/09 17.00 -200	0.13	11/24/2009	113	74

[2] Waste Characterization Data Package Version 9 (July 3 2012)

4.2.2 LTVSMC Tailings

Four samples of LTVSMC tailings are being tested in humidity cells. No significant changes in leachate chemistry have been observed since the previous update (SRK 2011). Leachate pHs have continued to vary in a narrow band between 7.4 and 8 with variable but non-trending alkalinity and sulfate.

Other parameters have continued to show no upward trend or downward trends. Isolated data spikes (for example, arsenic) are observed but do not represent significant trends.

4.3 Interpretation

Ongoing tailings testwork has shown no indication of development of pHs below that of the deionized water used to leach the samples (Figure 7) indicating that weathering of silicates minerals in the samples is continuing to generate alkalinity to offset acidity from sulfide oxidation and the deionized water even in the tests that have been operating for 7 years. The longer term tests yielded pH minima in their trends which are responsible for the lower pHs shown in Figure 7. Upward trends in pH are observed in most cases so that recent pHs are higher than the lowest values.

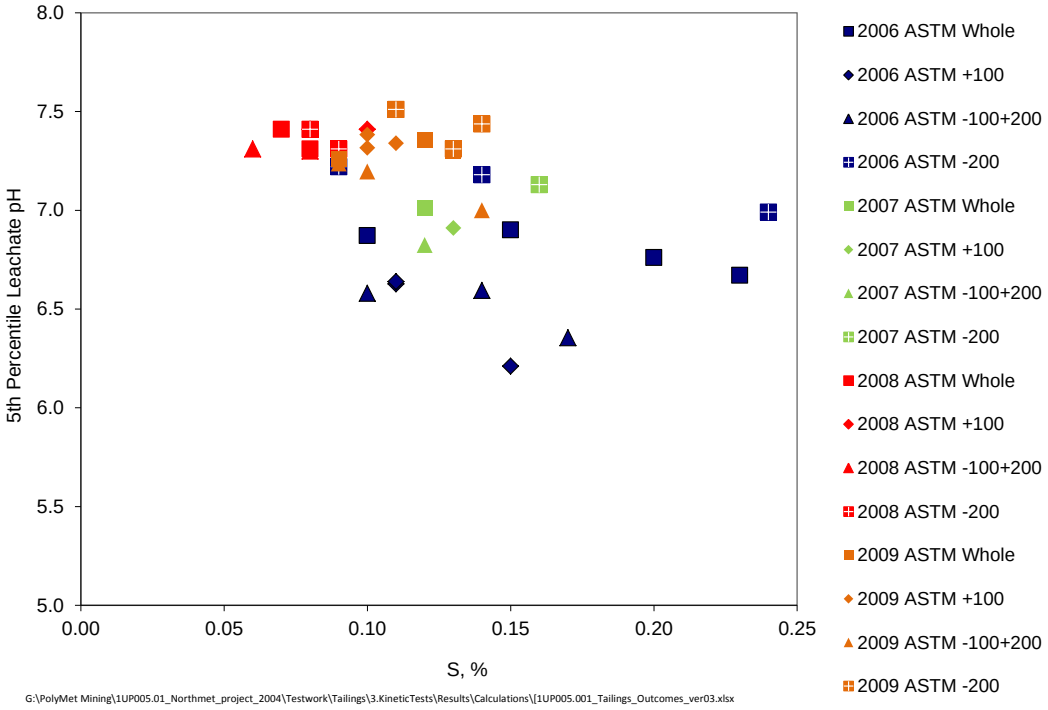

Figure 7. 5th Percentile pH as a Function of Total Sulfur Content for Tailings Humidity Cells

Correlations between initial total sulfur content and average release rates are apparent as have been observed previously (Figure 8). However, this figure suggests different relationships for the 2006 and 2007 datasets compared to the 2008 and 2009 datasets apparently implying that the earlier samples are less reactive. However, the differences are probably caused by significant depletion of sulfide as the tests have progressed with a correlated decrease in release rates, reducing the calculated average release rates. For example, between 30% and 78% of the initial sulfur has been depleted from the earlier tests compared to less than 30% for the more recent tests. For the oldest tests, remaining sulfur content varies from 0.04% to 0.12%. The general downward trend in sulfate release has resulted in progressively lower average sulfate release. The downward trend in sulfate release (and therefore acid generation rate) also provides a reasonable explanation for upward trends in pH.

The implication of results from the older tests is that the potential for long term pH depression below 5 is shown to be very low because silicate weathering was able to offset acid generation when sulfur content was higher than the current depleted levels.

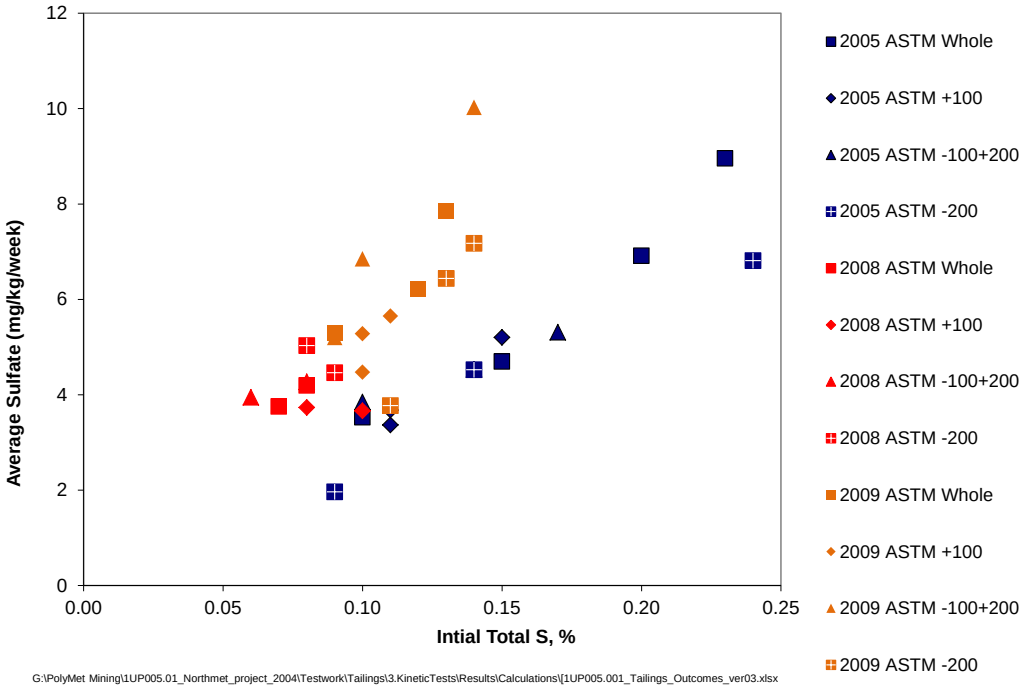

Figure 8. Average Sulfate Release as a Function of Initial Total Sulfur Content

The long term downward trend in sulfate release rates (Figure 9) indicates that average sulfate release rates for individual tests as used as input into the tailings water quality models are a function of time and will decrease as the test duration increases. To address this finding and provide sulfate release rates that are consistent with the assumed zero order reaction rate law for sulfide oxidation in the water quality model, relationships between sulfate release and sulfur depletion were examined to estimate sulfide oxidation rates at the initial sulfur content. The concept is to obtain the initial rates at t=0 by regressing depleted sulfur content against sulfate release rates to obtain equations of the form:

$$\frac{dM_{SO_4}}{dt} = a_1 M_S + a_0$$

Graphs of sulfate release rate as a function of sulfur remaining showed three different relationships. The most common relationship (15 tests) was for the slope of the equation to be steeper at the start than in the longer term. Other trends were (1) initially lower rates followed by a peak then long term decay (about 6 tests); and (2) consistent slope for the duration of the test (12 tests).

The first type of relationship is commonly observed in humidity cell results and is interpreted as initial flushing of stored sulfate oxidation products followed by sulfide oxidation. The initial part of the trend therefore does not represent sulfide oxidation and should not be included in the regression equations to obtain the oxidation rate at t=0. The preferred approach is to use the data for each test on its own merit to evaluate the break point when leaching of stored oxidation products is complete and the long term decay trend is well-established. LAM MDNR disagreed with this approach and required that the regression equations be developed excluding the first 5 weeks of data based on the following commentary in the ASTM humidity cell method D5744-96, (Section 11.2):

> "In the testing of mining wastes, cation and anion loadings are commonly high in the Week 0 leachate due to the dissolution of pre-existing soluble oxidation salts present in the sample prior to sample collection. The average number of weekly accelerated weathering cycles required to flush these pre-existing salts ranges from 3 to 5 weeks. Oxidation products observed during these 3 to 5 weeks are principally from pre-existing salts, while those

products observed after this period are considered to be solely a function of the accelerated weathering procedure."

SRK does not agree with the use of a fixed initial flushing period because it is unlikely that this flushing process can be generalized. In reality, the period required to fully flush stored oxidation products depends on the types, solubility and quantity of the oxidation products, and the physical and mineralogical characteristics of the samples. Tailings samples also contain entrained process water which is expected to be soaked into the finer particles leading to a protracted initial flushing trend. Nonetheless, to advance the process of acceptance, the LAM MDNR requested method was adopted to calculate oxidation rates at t=0.

Graphs for all tests showing regression relationships using the MDNR method are provided in Attachment A

Figure 9 shows initial sulfate release (determined from the regression equation using the initial sulfide content of the sample) as a function of initial total S content of the samples for comparison with Figure 8. Initial sulfate release calculated using the MDNR and SRK methods is shown. The difference in rates between tailings samples generated by different pilot plants implied by Figure 8 is reduced by both methods but the DNR method appears to result in lower rates for the 2005 samples and a weaker correlation with sulfur content than the SRK method which eliminates differences between the different datasets. The similarity of rates implies that the tailings generated at different times have similar reactivity. Correlation between initial sulfur content and initial sulfate release is apparent when considering the entire data range though weaker at lower initial sulfur contents.

For the purpose of water quality modeling, the distribution of sulfate release rates will be based on the initial sulfate rates (Figure 9) calculated using the MDNR method as agreed This method will result in the use of sulfide oxidation rates in the model that are nearly always higher than would actually be expected. Application of a single initial rate will result in predicted sulfate and trace element concentrations that are likely to be much higher than would really occur as sulfide content depletes over time. The approach is therefore conservative for predicting water quality.

For the purpose of modeling other parameters the following recommendations are made for rates:

- For parameters based on a solid ratio to sulfur content, rates should be calculated based on the ratios previously specified.
- For parameters based on average rate to average sulfate rate ratios (Ca, K, Mg, Na, Se), rates should be calculated based on the average rate ratio as previously specified. For the major ions, downward trends are apparent that parallel decrease in sulfate indicating that ratios of average rates are appropriate.

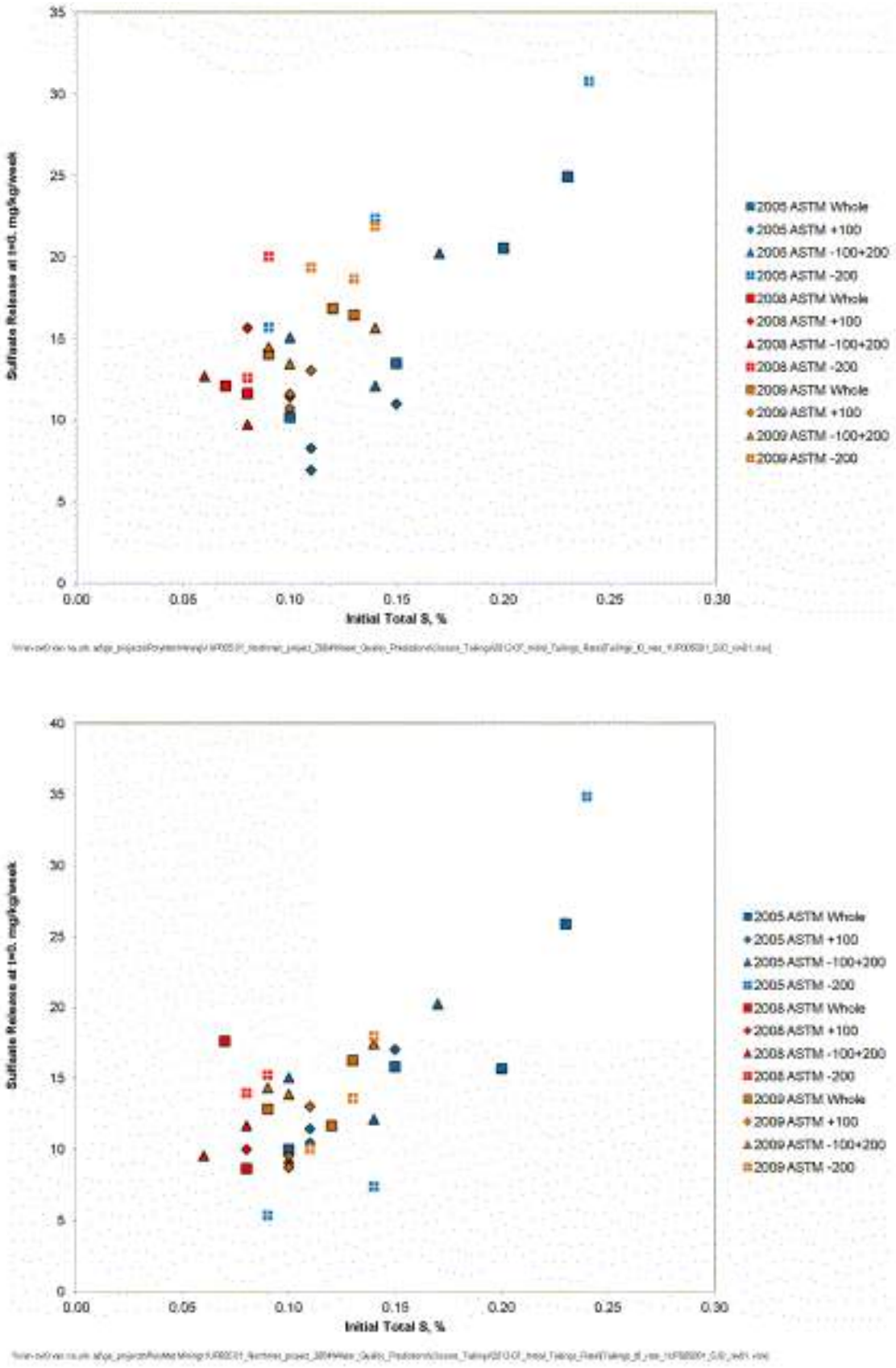

Figure 9. Calculated Initial Sulfate Release as a Function of Initial Total Sulfur Content Using MDNR (top) and SRK (bottom) methods.

5 Hydrometallurgical Residue Kinetic Tests

As agreed with MNDR, hydrometallurgical kinetic tests were stopped shortly after reported by SRK (2009).

6 Conclusions

Review of data generated by up to nearly eight years of testing indicates that:

- Waste rock classification into three categories using sulfur content to address potential for pH acidification appears to be robust. In particular, samples classified as Category 1 have not shown pH depression below that of the deionized water used to leach the samples.
- Sulfide oxidation and metal leaching trends for waste rock samples show well defined phases as pH depression occurs. These trends have been used to calculate release rates for use in development of source terms. Ongoing testwork has resulted in some additional rate information for later phases.
- Tailings humidity cells have not shown pH depression below that of the deionized water indicating that weathering of silicates provides alkalinity to offset acidity from sulfide oxidation.
- Long term trends in tailings humidity cells show declining oxidation rates which are presumably linked to depletion of the sulfide minerals and indicates that sulfide oxidation is following a non-zero order reaction rate law. However, to be consistent with the simplified modeling approach used in GoldSim®, which assumes a zero order reaction rate law, trends in sulfide oxidation rates have been used to back-calculate reaction rates at the initial sulfur content for use in modeling.
- Other than addressing this recommendation, updating the rates used as inputs to tailings water quality modeling is not recommended.

7 References

SRK Consulting. 2009. RS82 - Update on Use of Kinetic Test Data for Water Quality Predictions – DRAFT 02. Memorandum to Kim Lapakko, MDNR. February 7, 2009.

SRK Consulting. 2011. Update on Tailings Humidity Cell Test Data – NorthMet Project – Draft. Memorandum to Peter Hinck, Barr Engineering. December 23, 2011.

SRK Consulting (Canada) Inc.

Stephen Day, PGeo
Corporate Consultant (Geochemistry)

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The opinions expressed in this report have been based on the information available to SRK at the time of preparation. SRK has exercised all due care in reviewing information supplied by others for use on this project. Whilst SRK has compared key supplied data with expected values, the accuracy of the results and conclusions from the review are entirely reliant on the accuracy and completeness of the supplied data. SRK does not accept responsibility for any errors or omissions in the supplied information, except to the extent that SRK was hired to verify the data.

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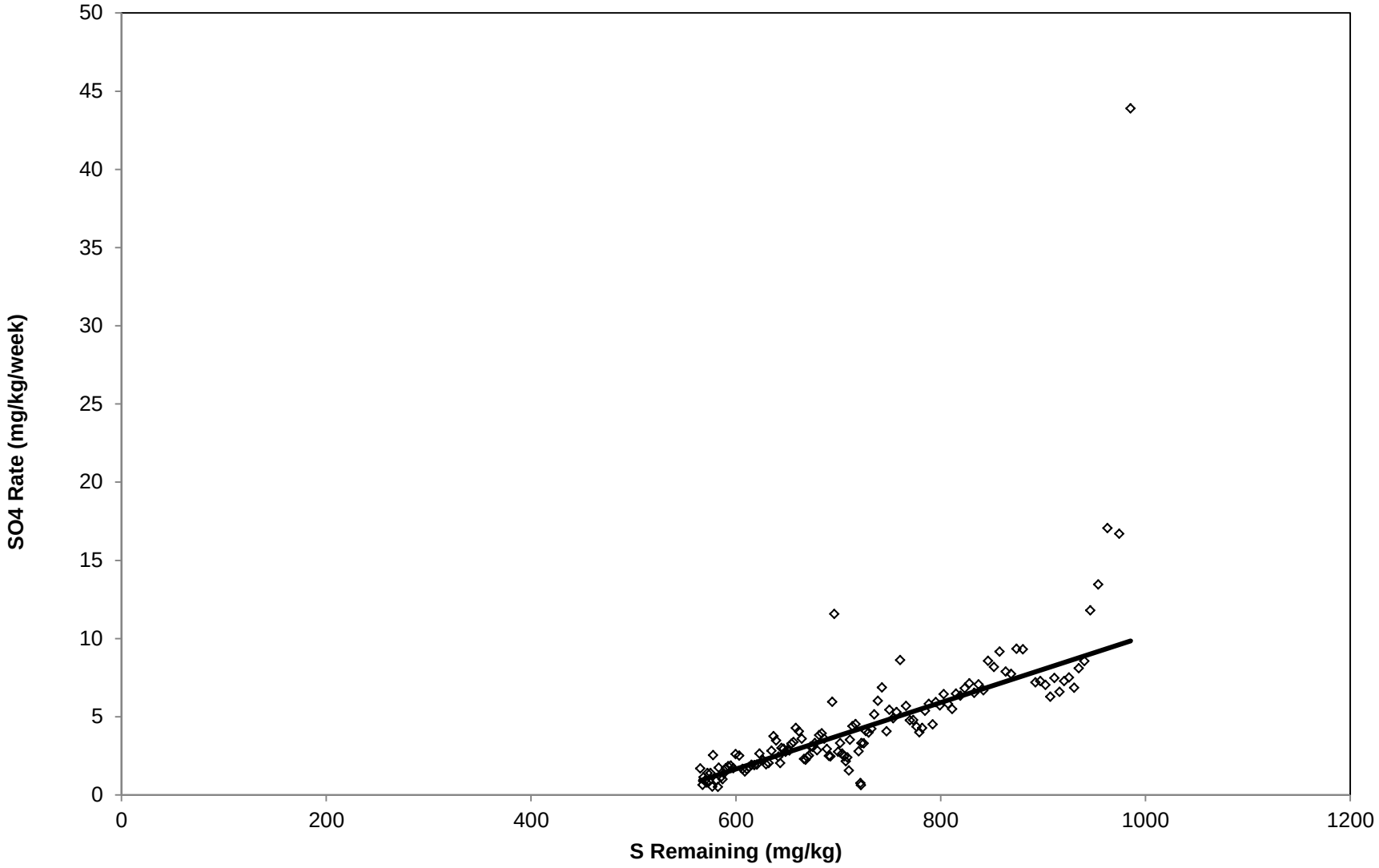
Attachment A: Graphs

P1S



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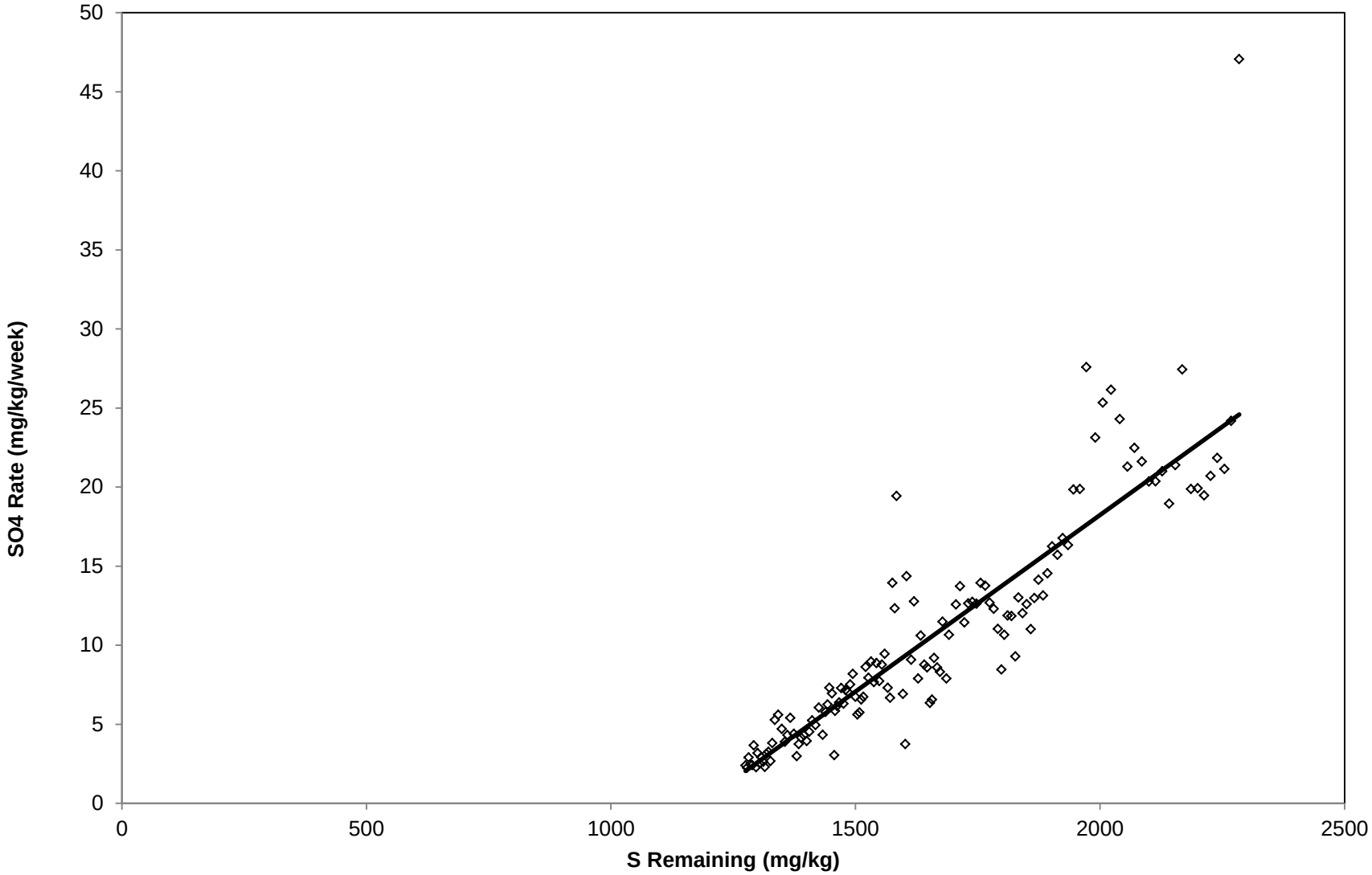

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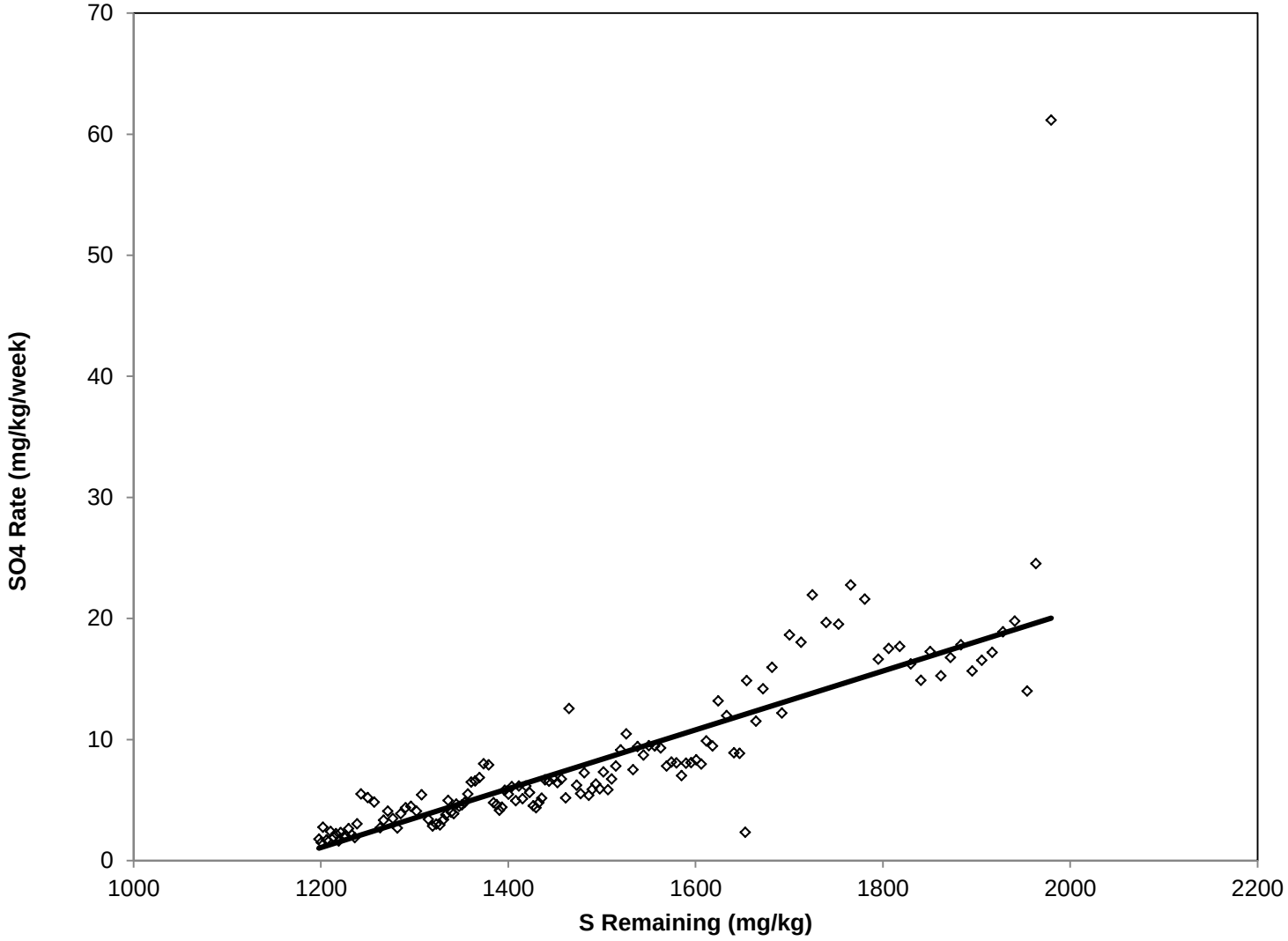

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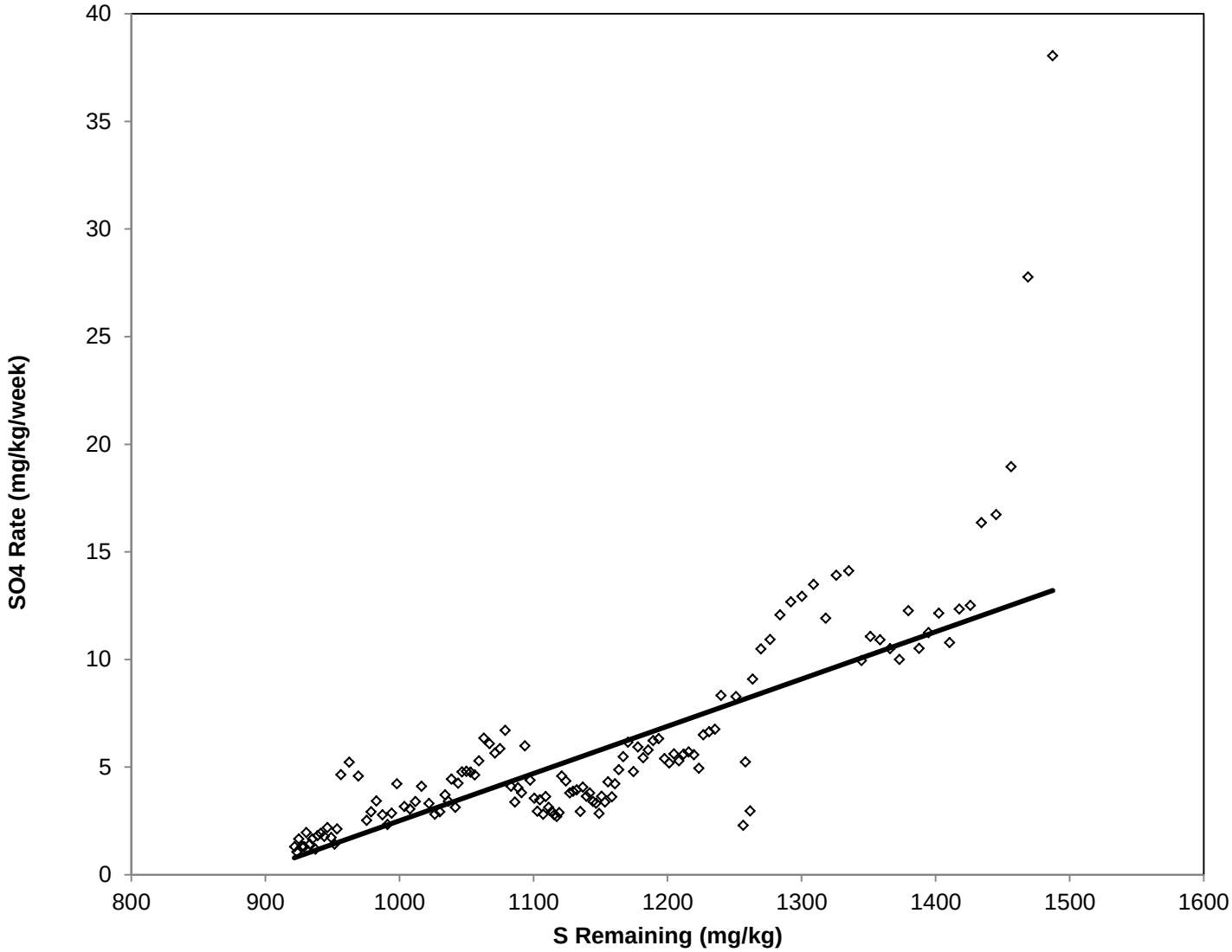

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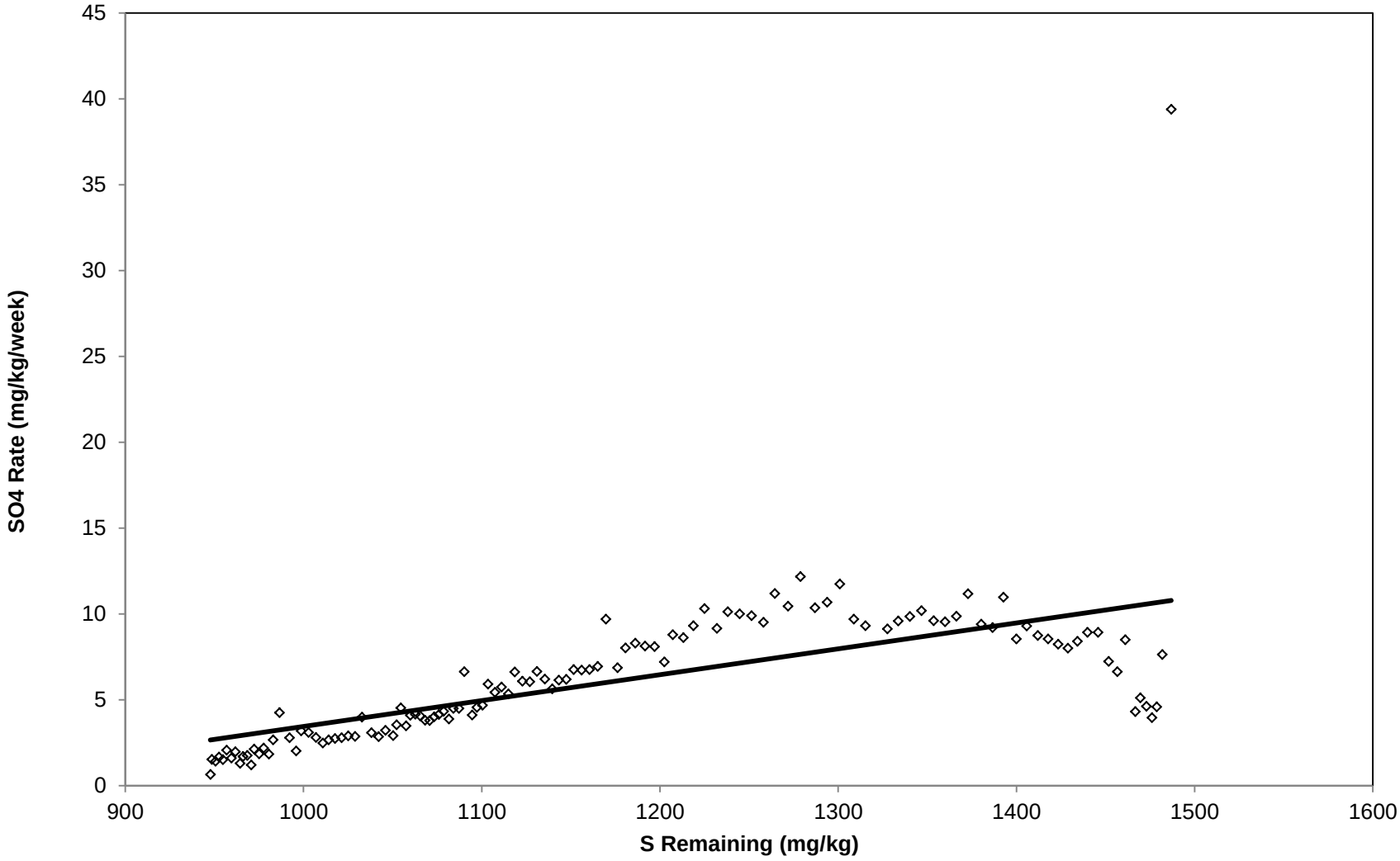

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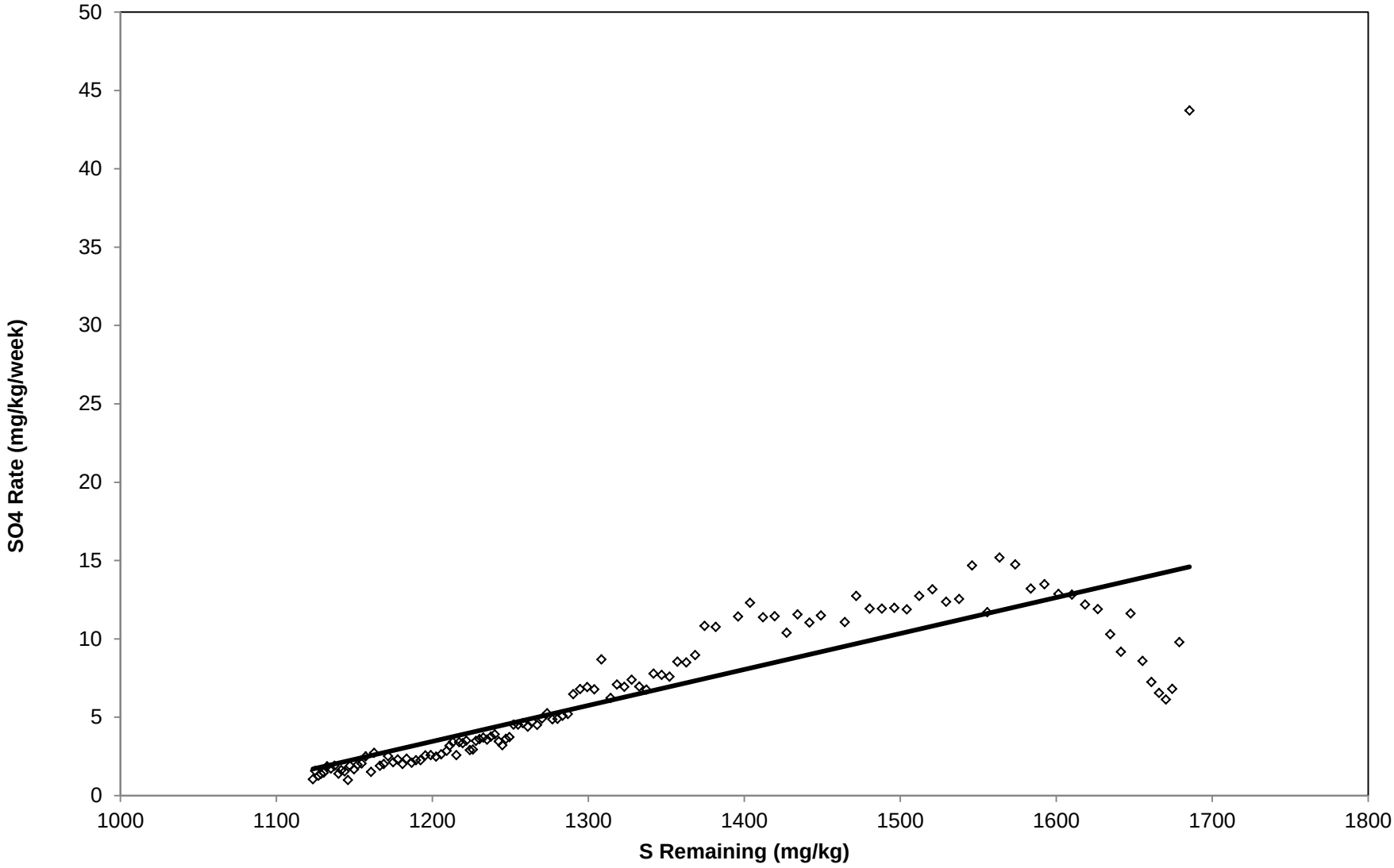

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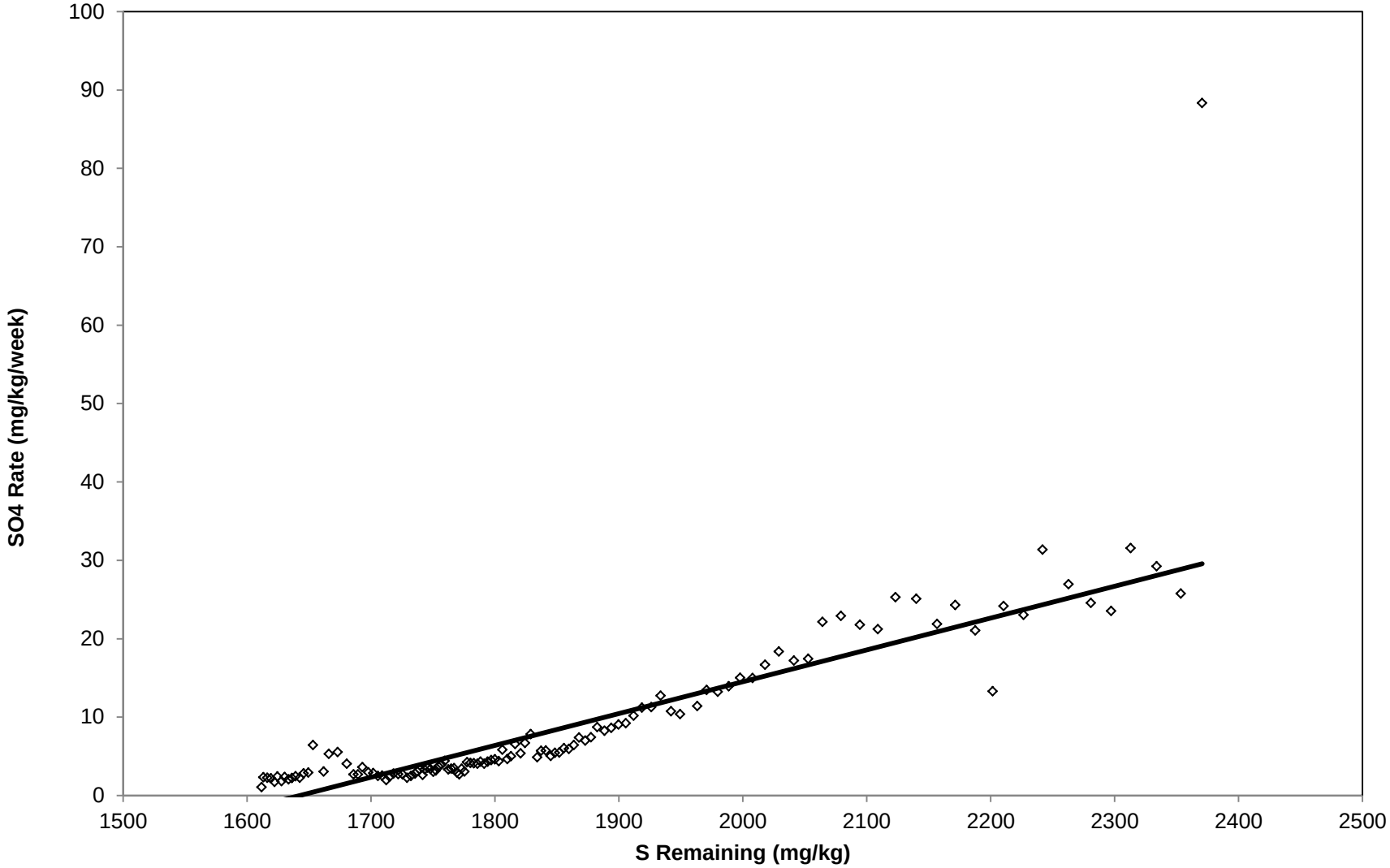

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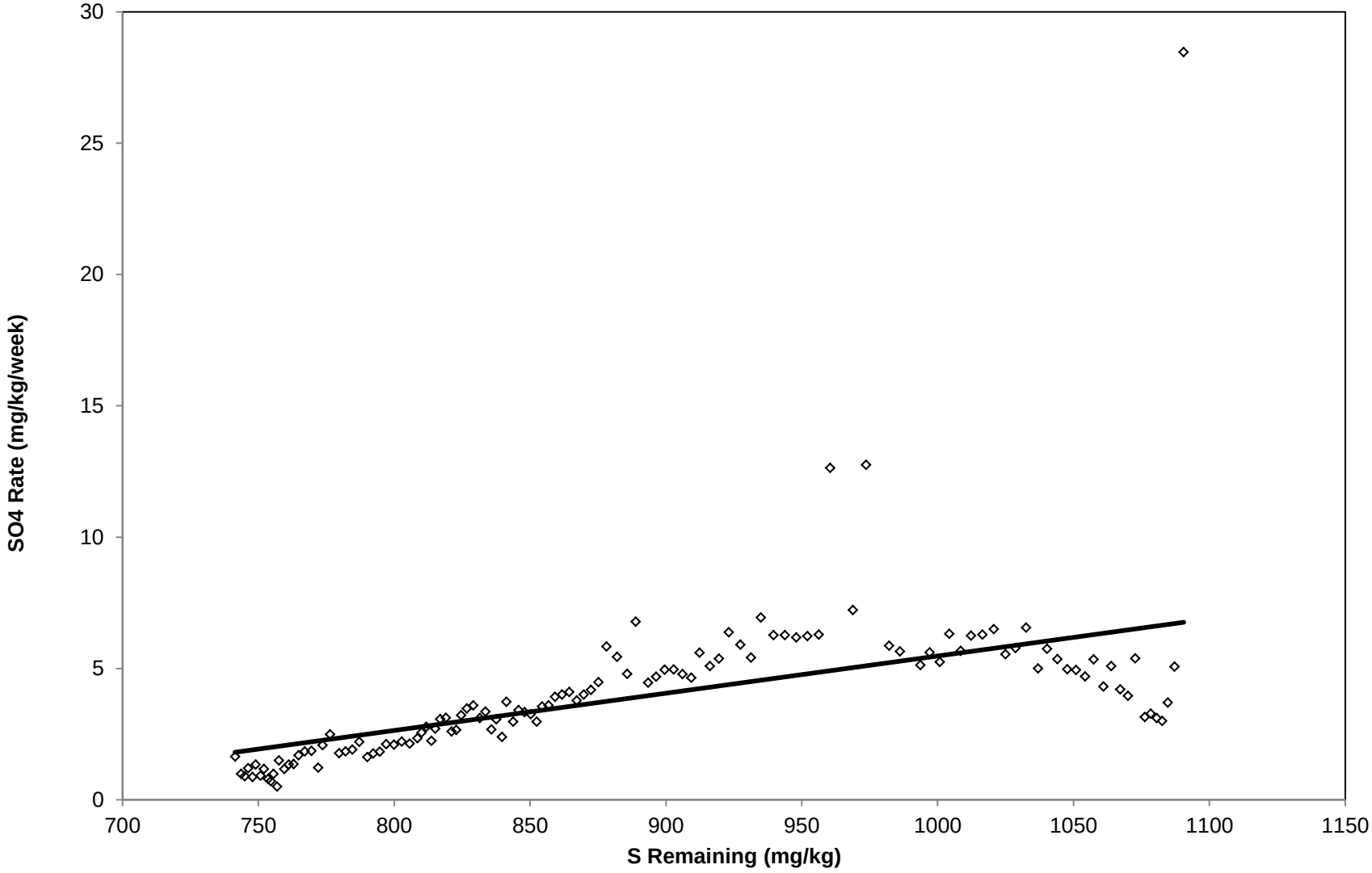

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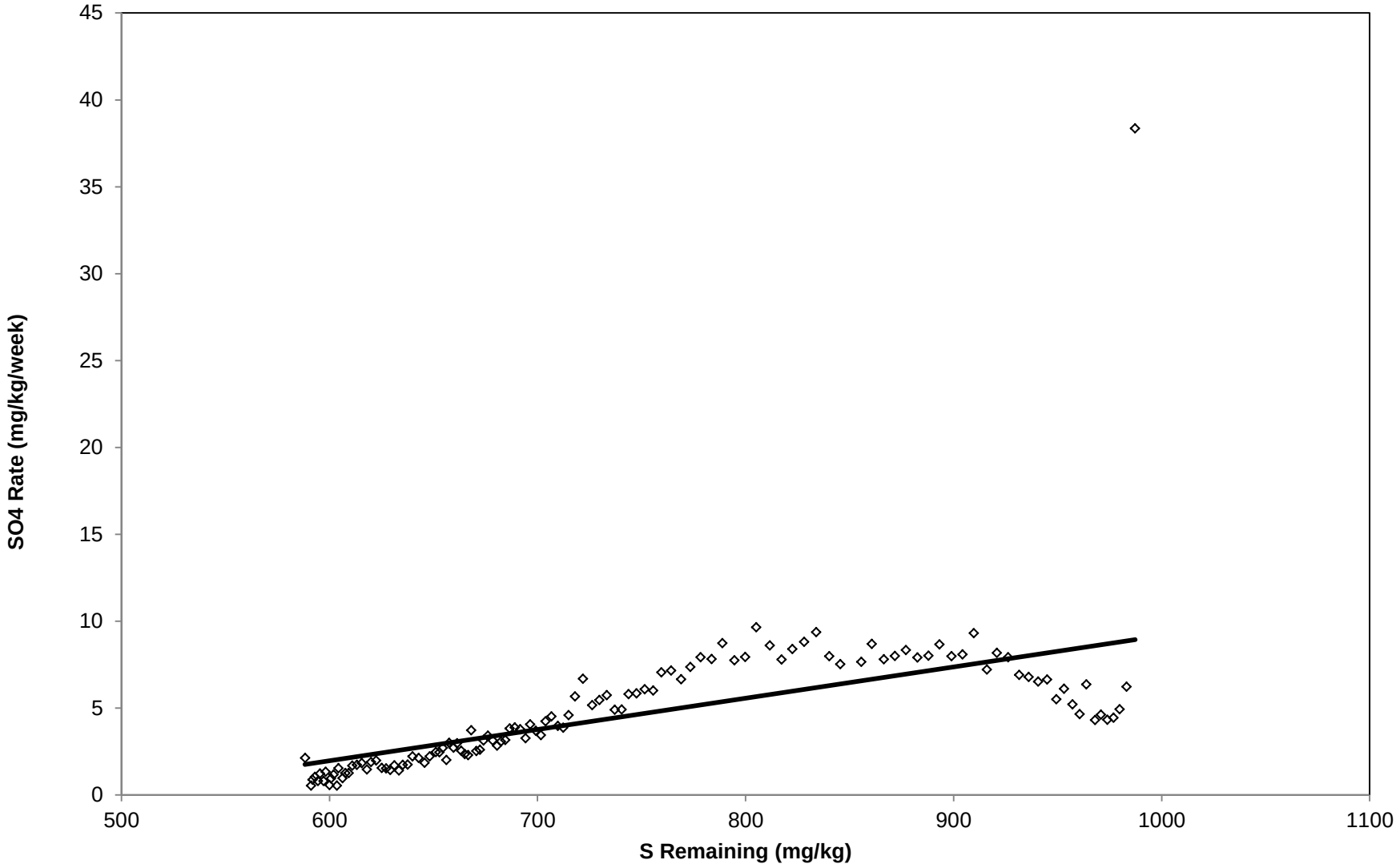

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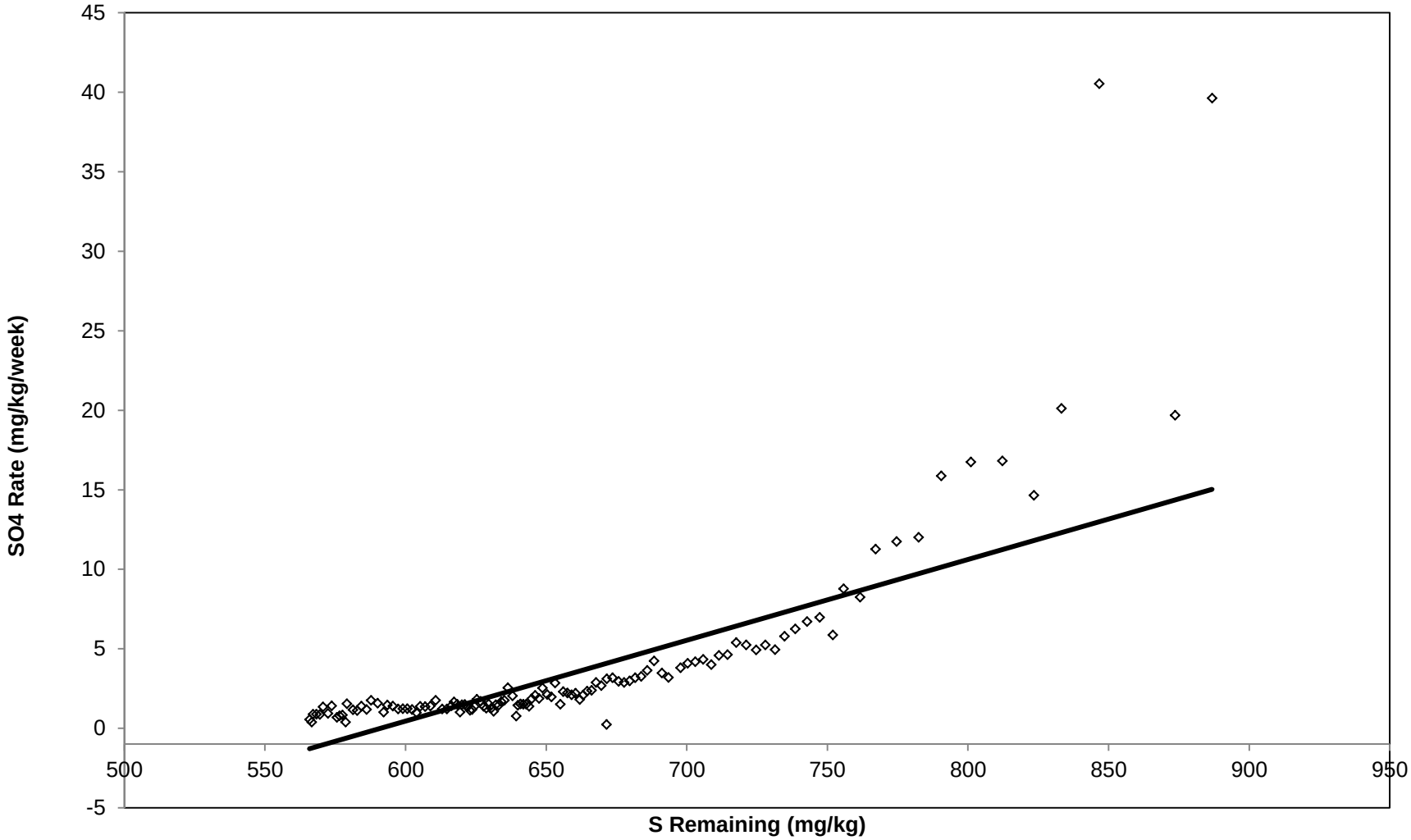

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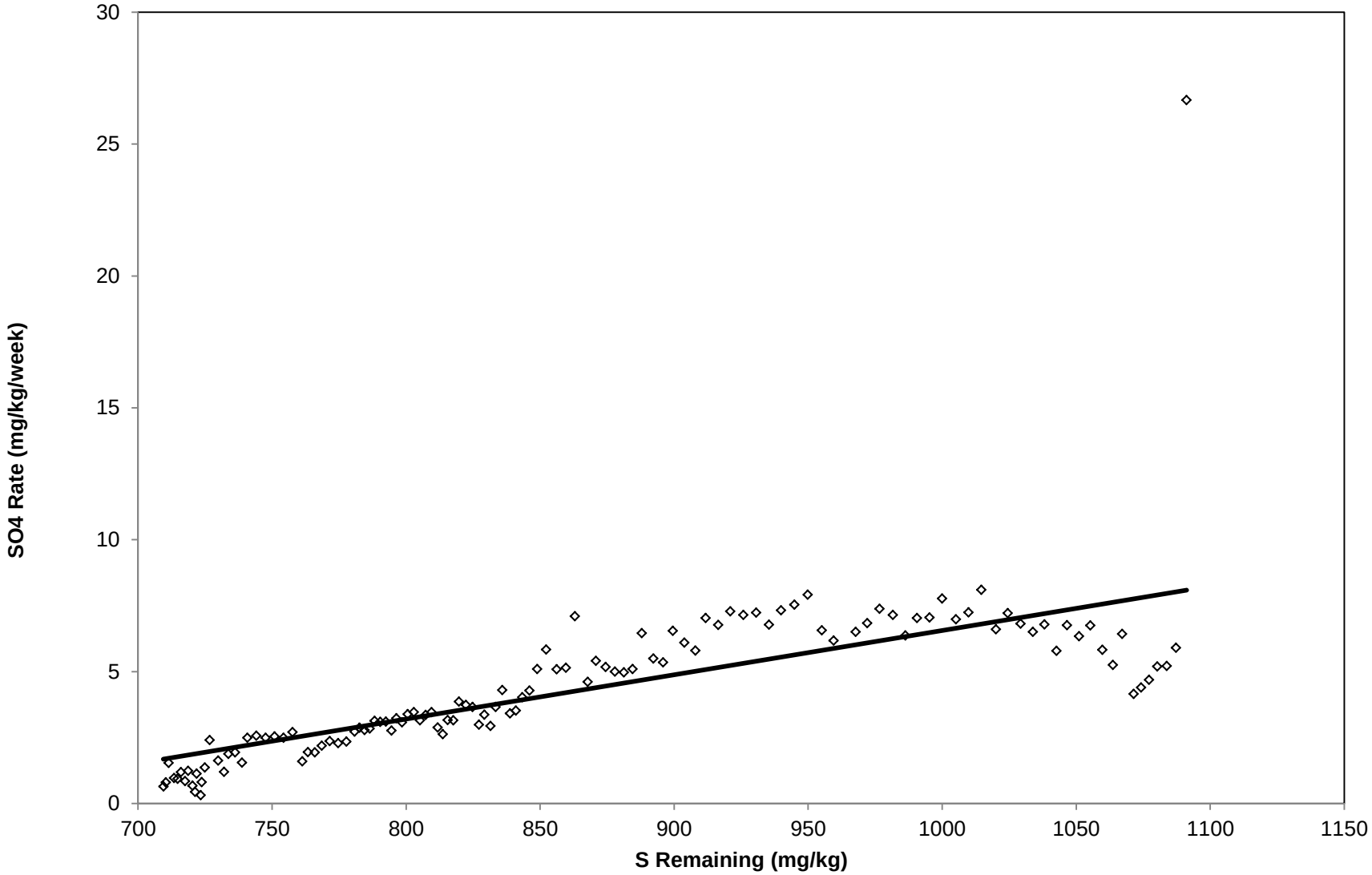

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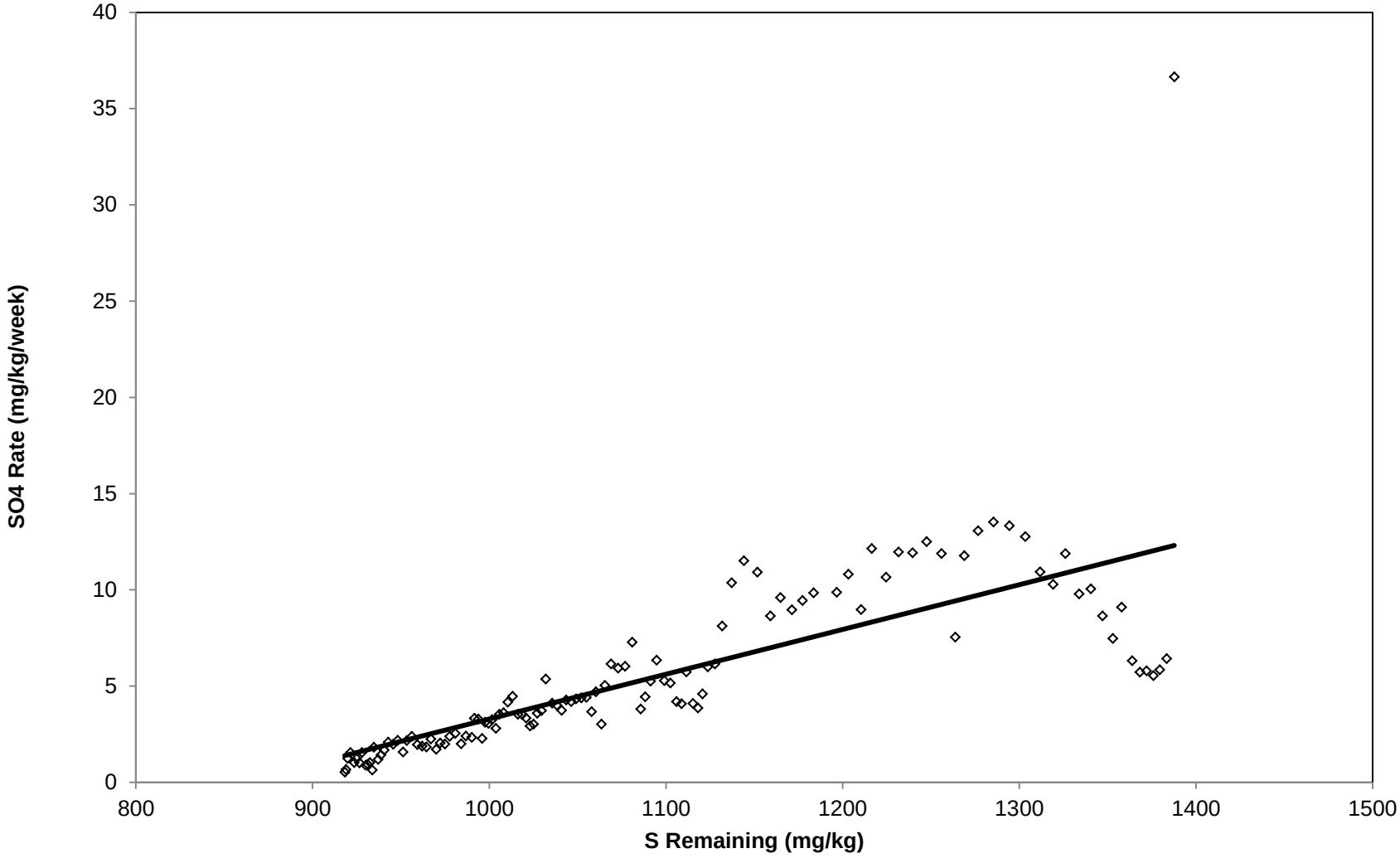

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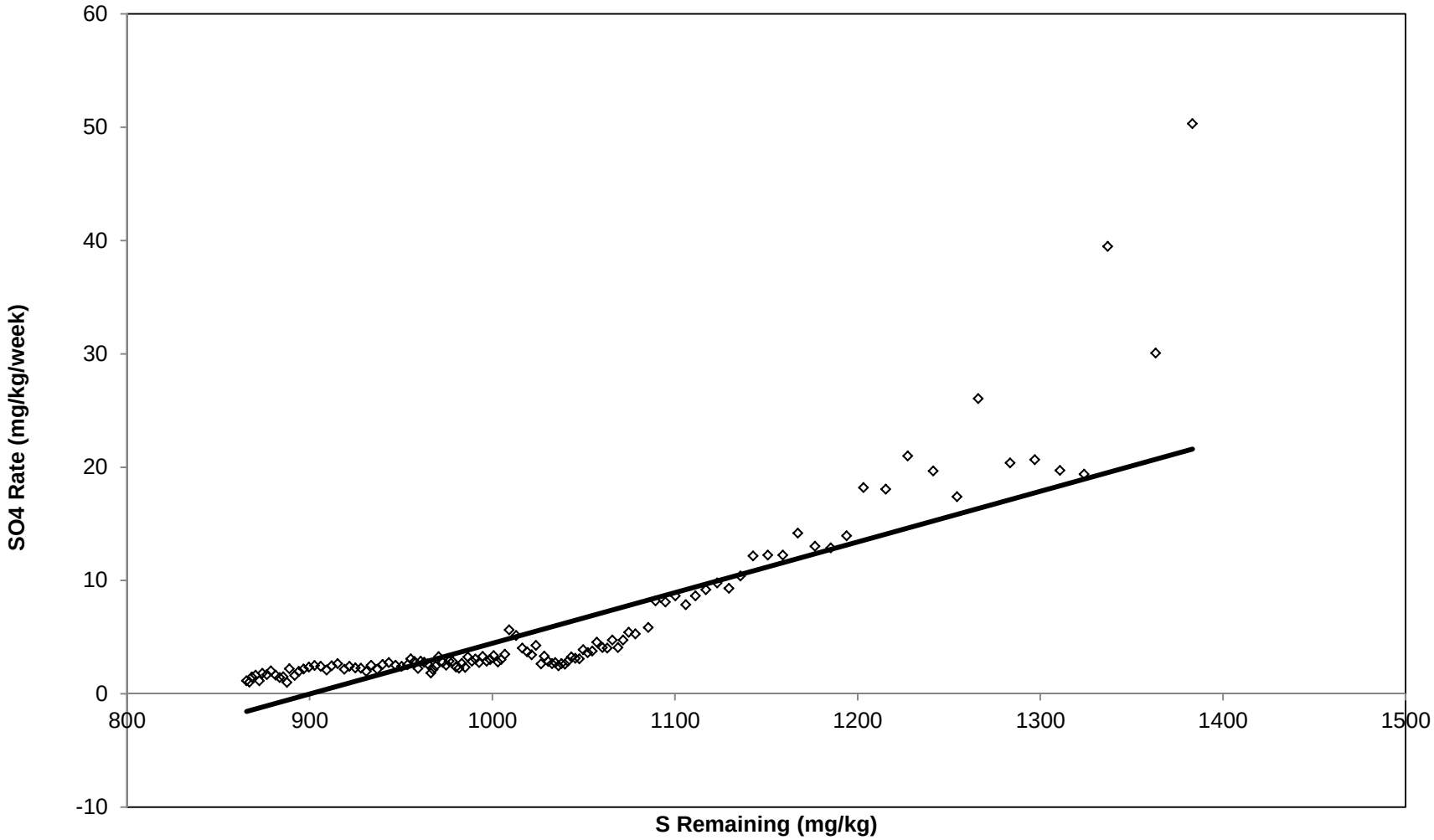

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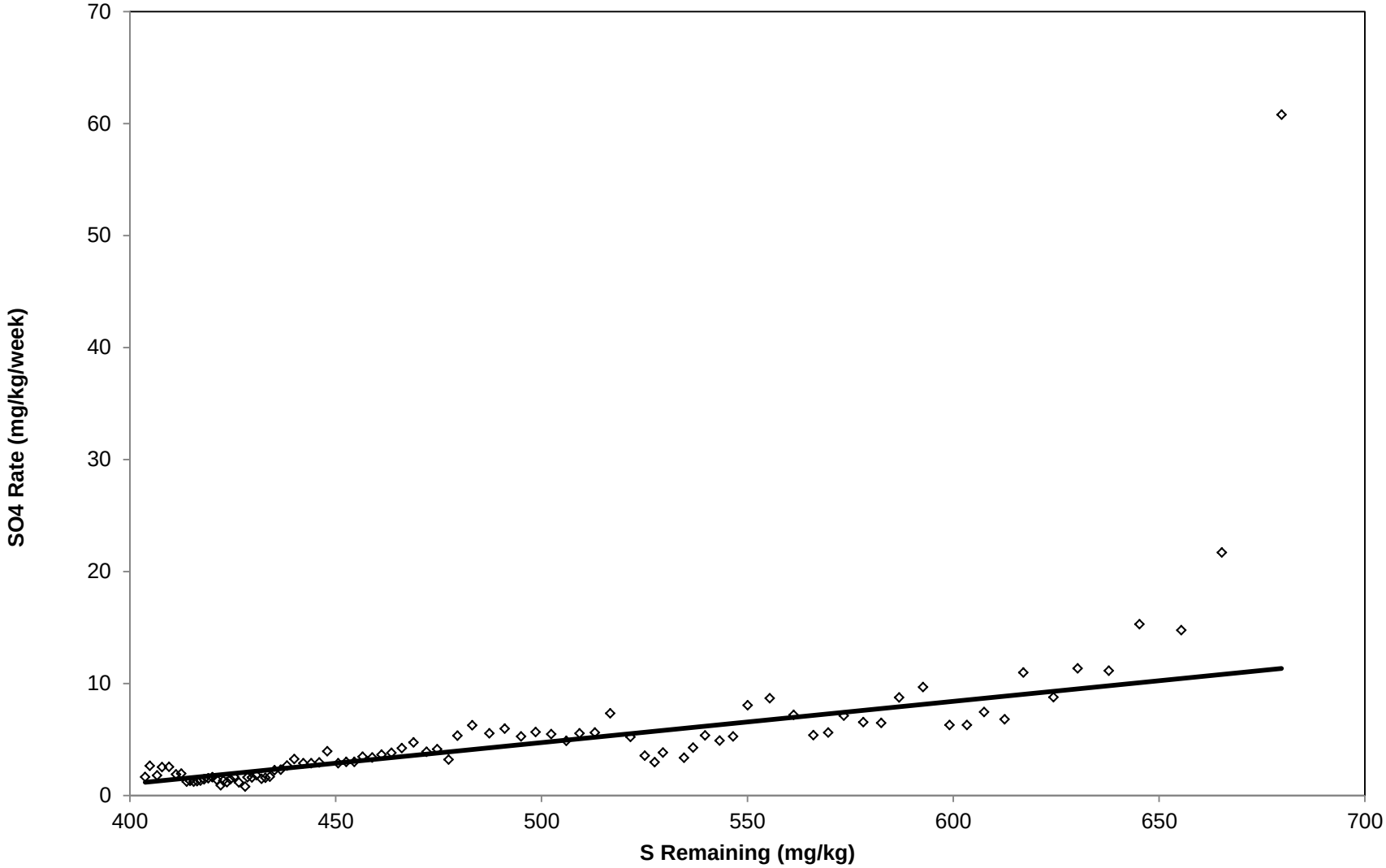

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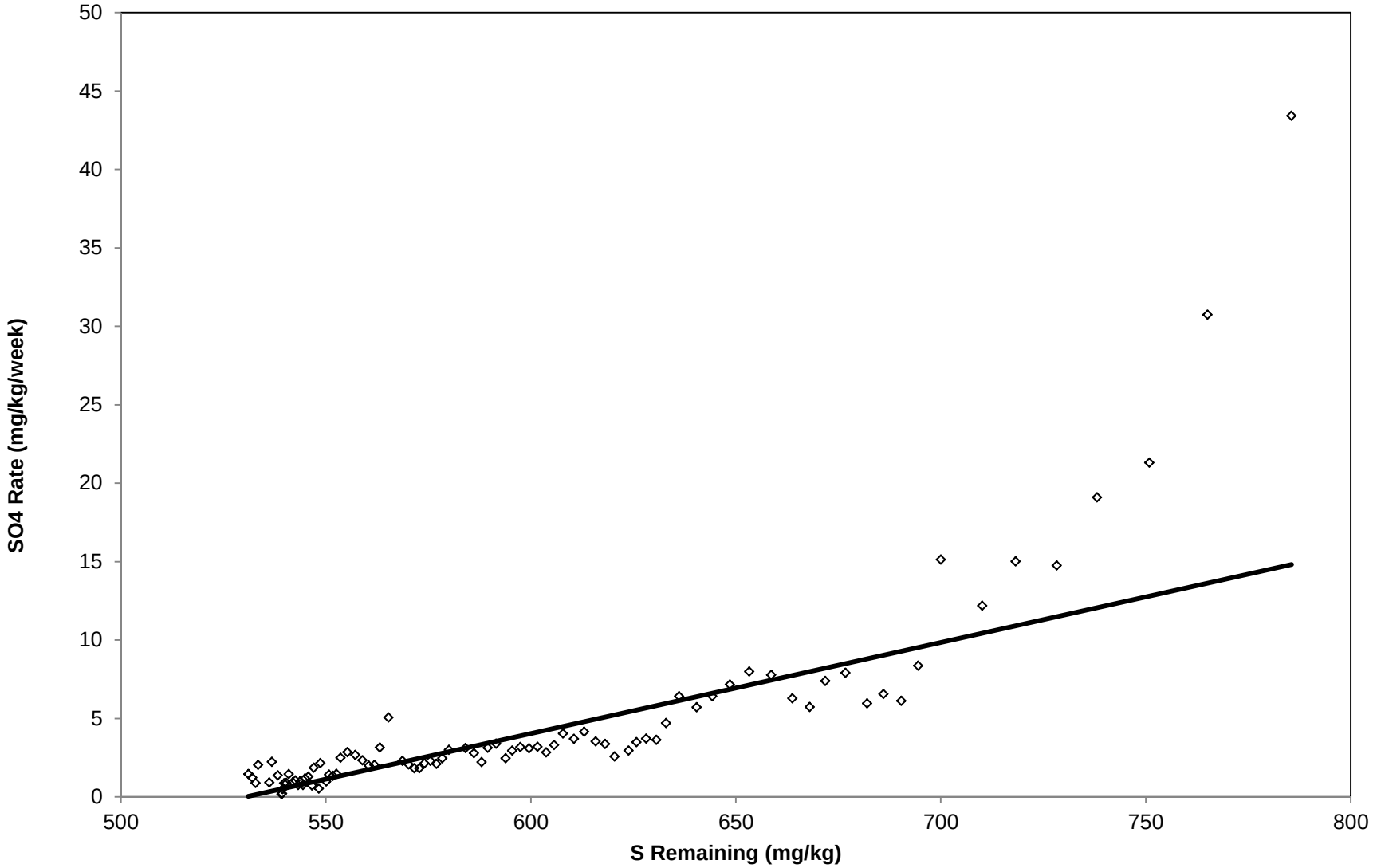

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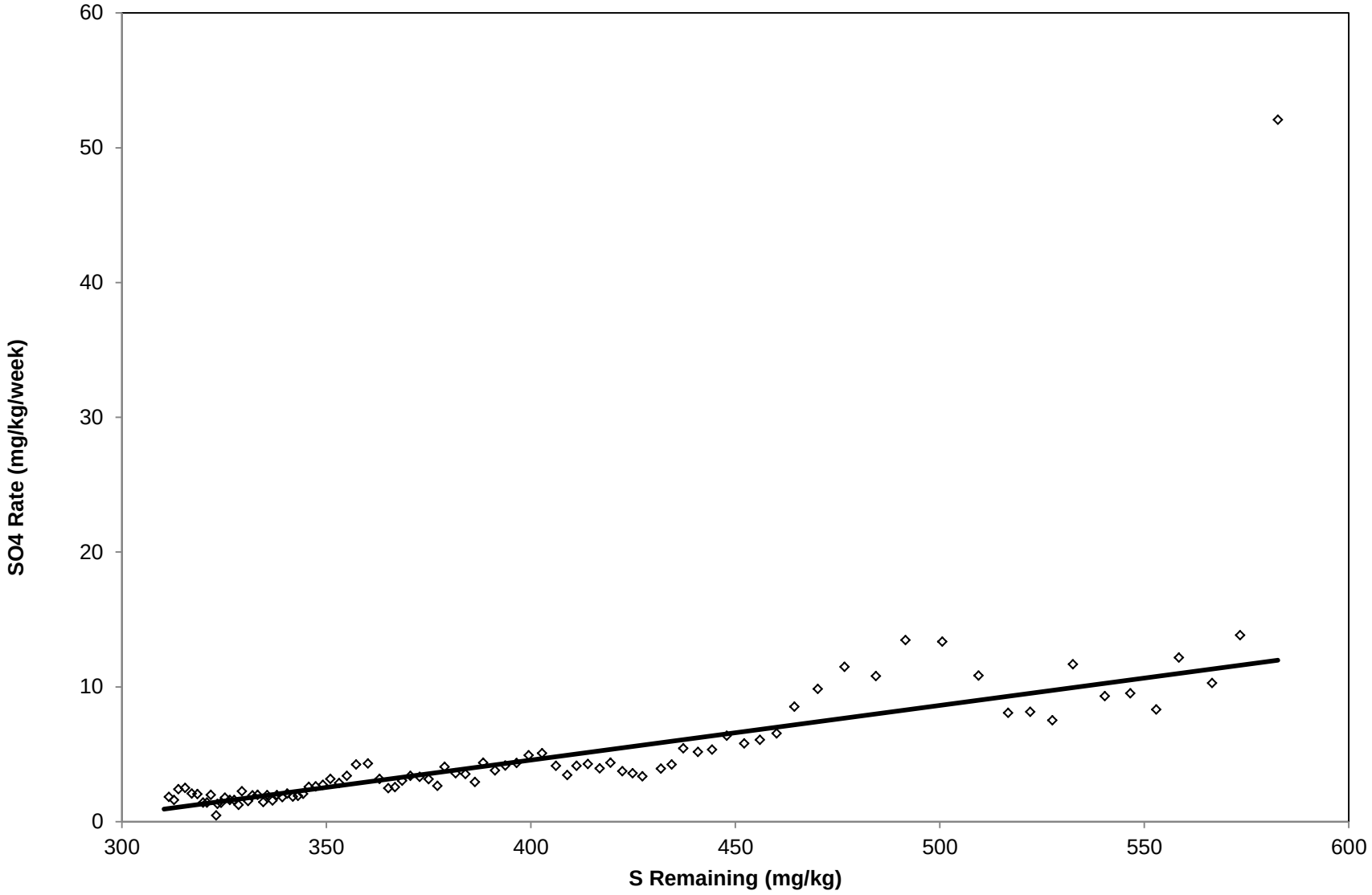

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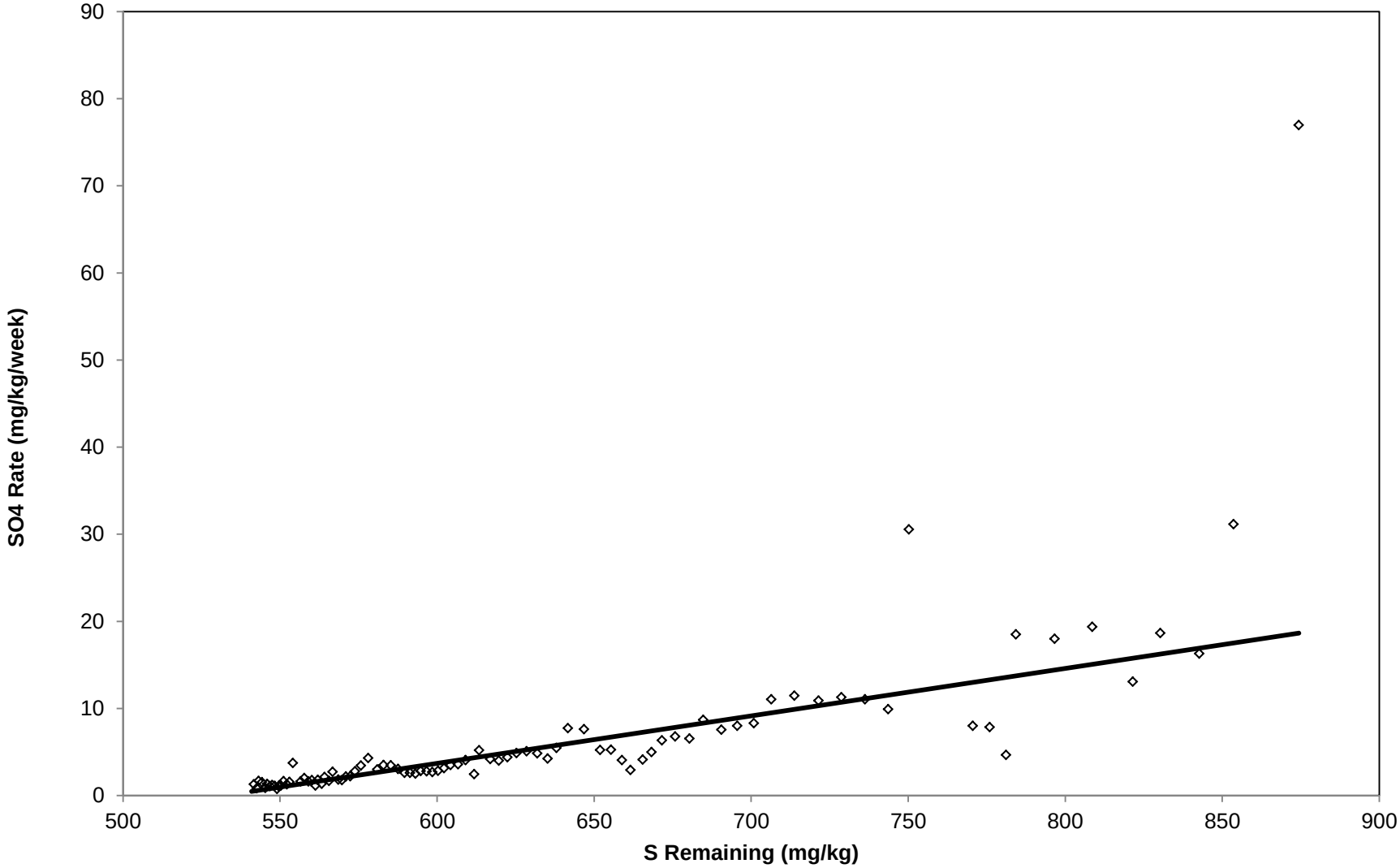

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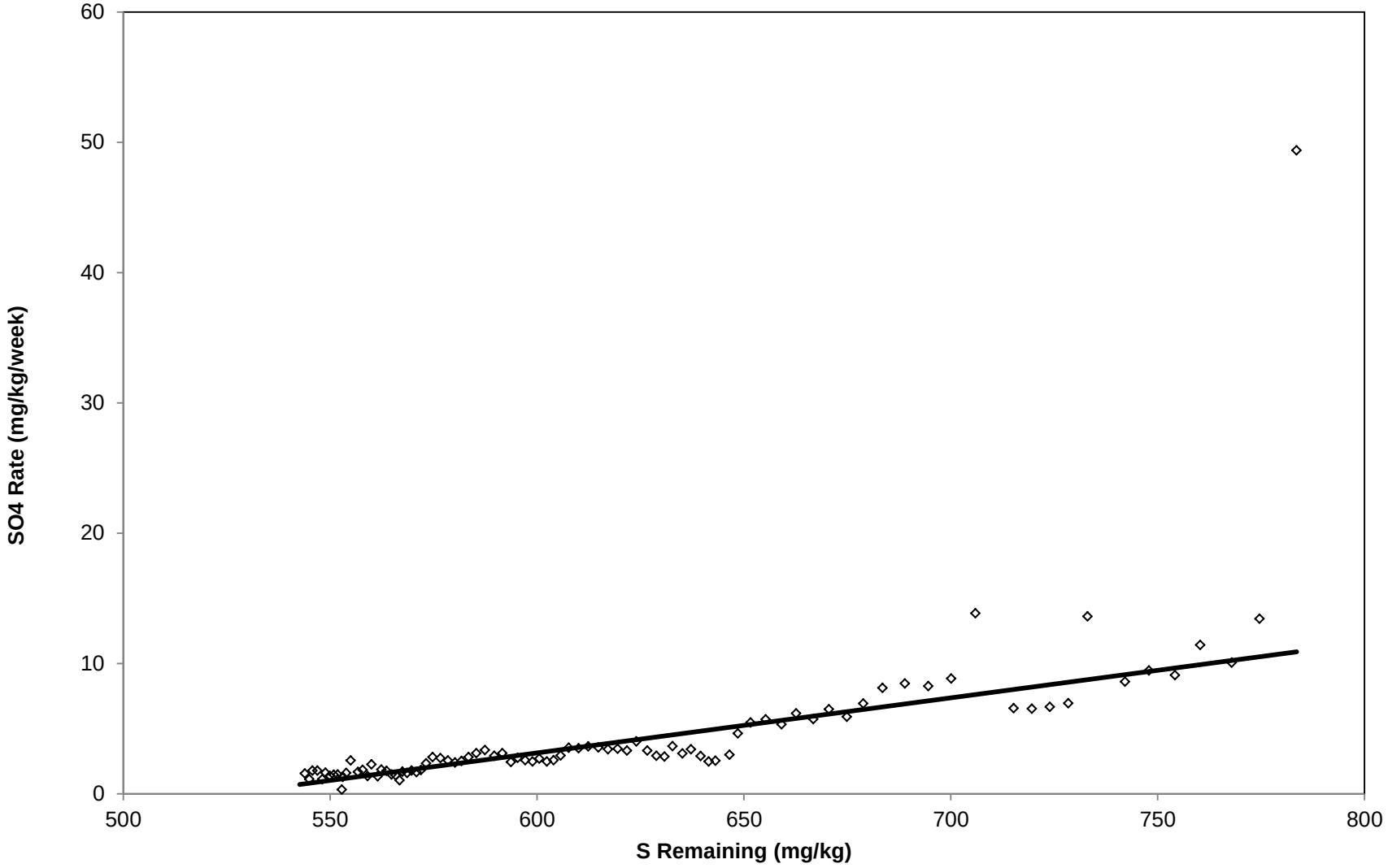

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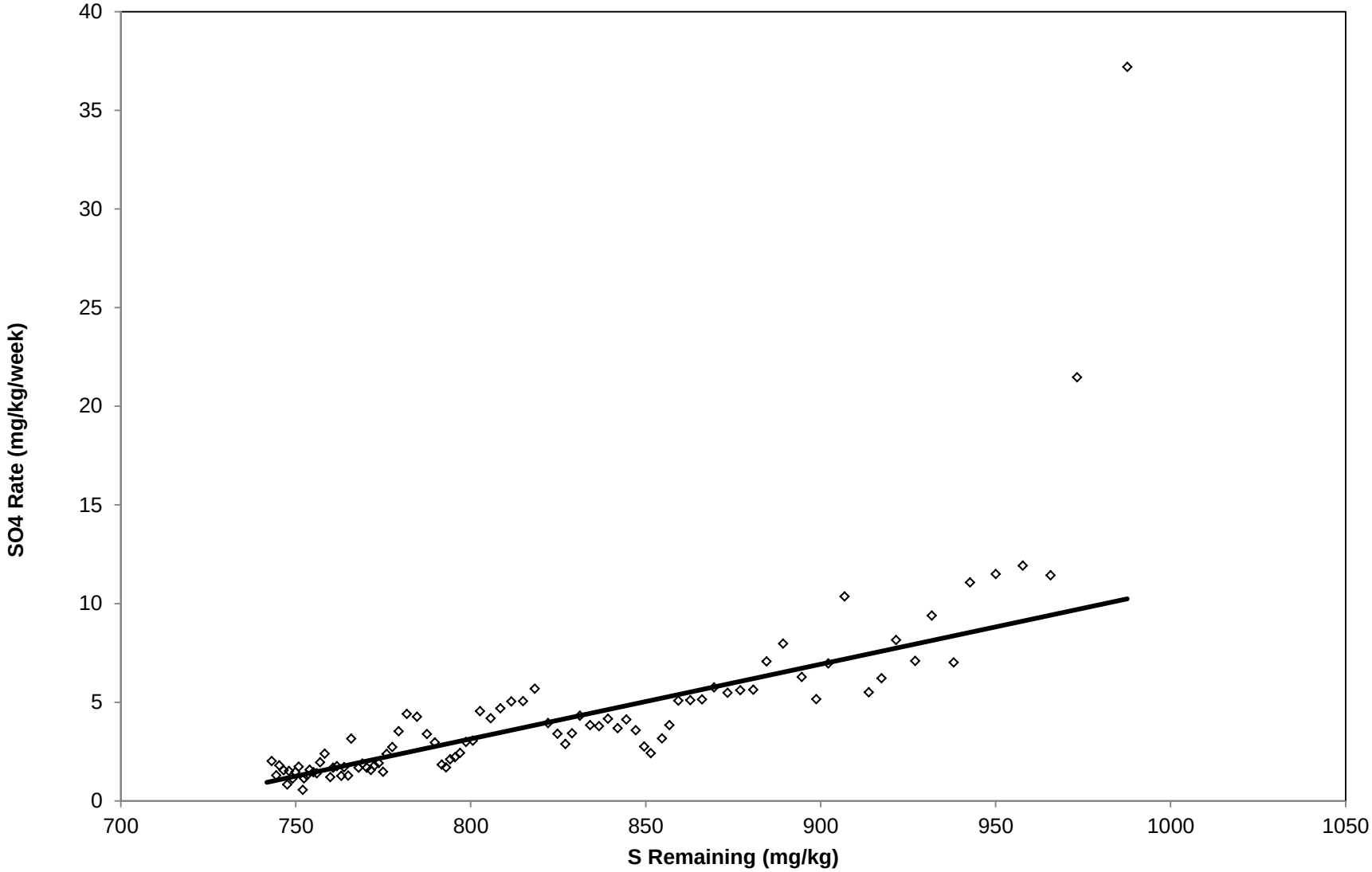

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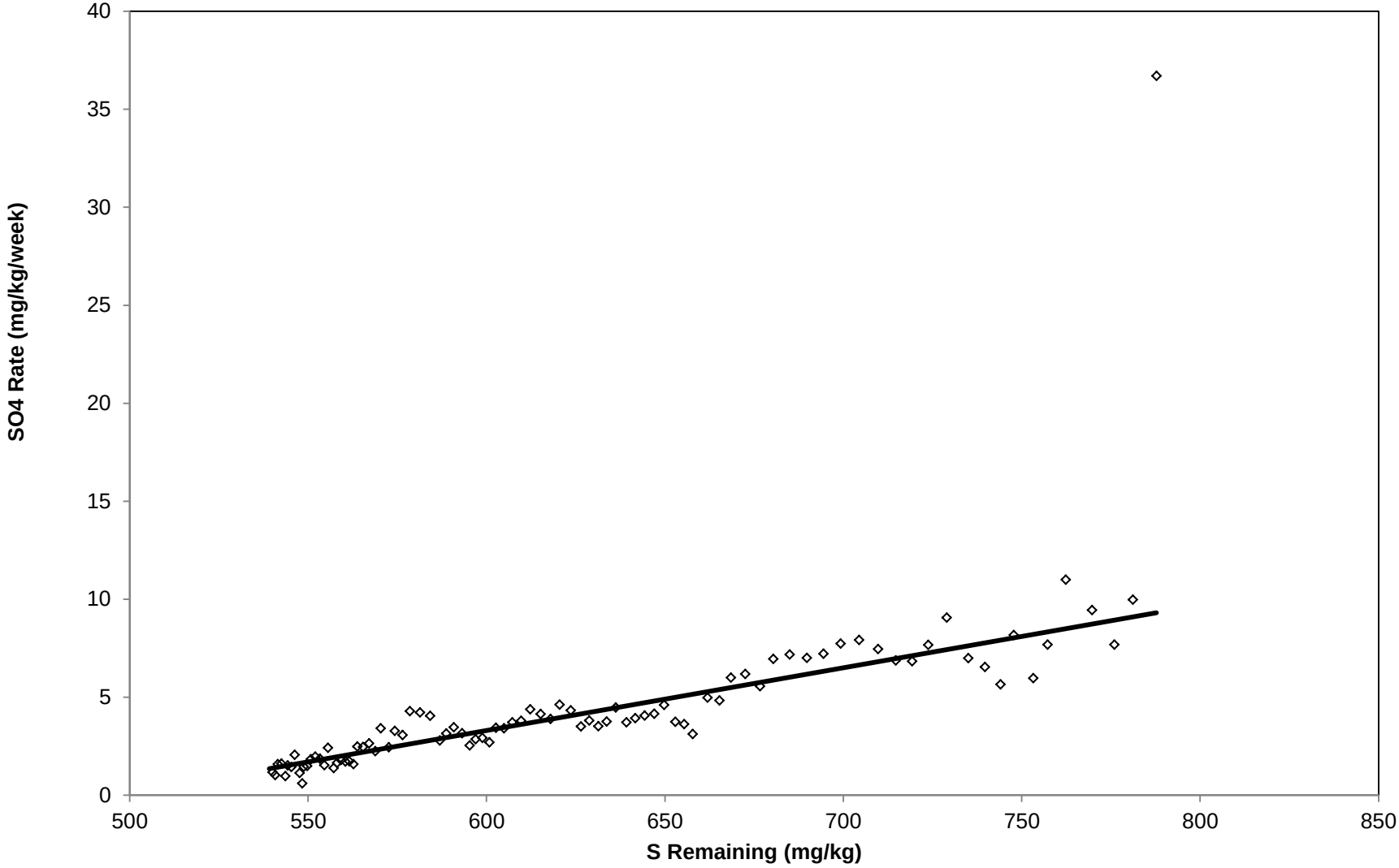

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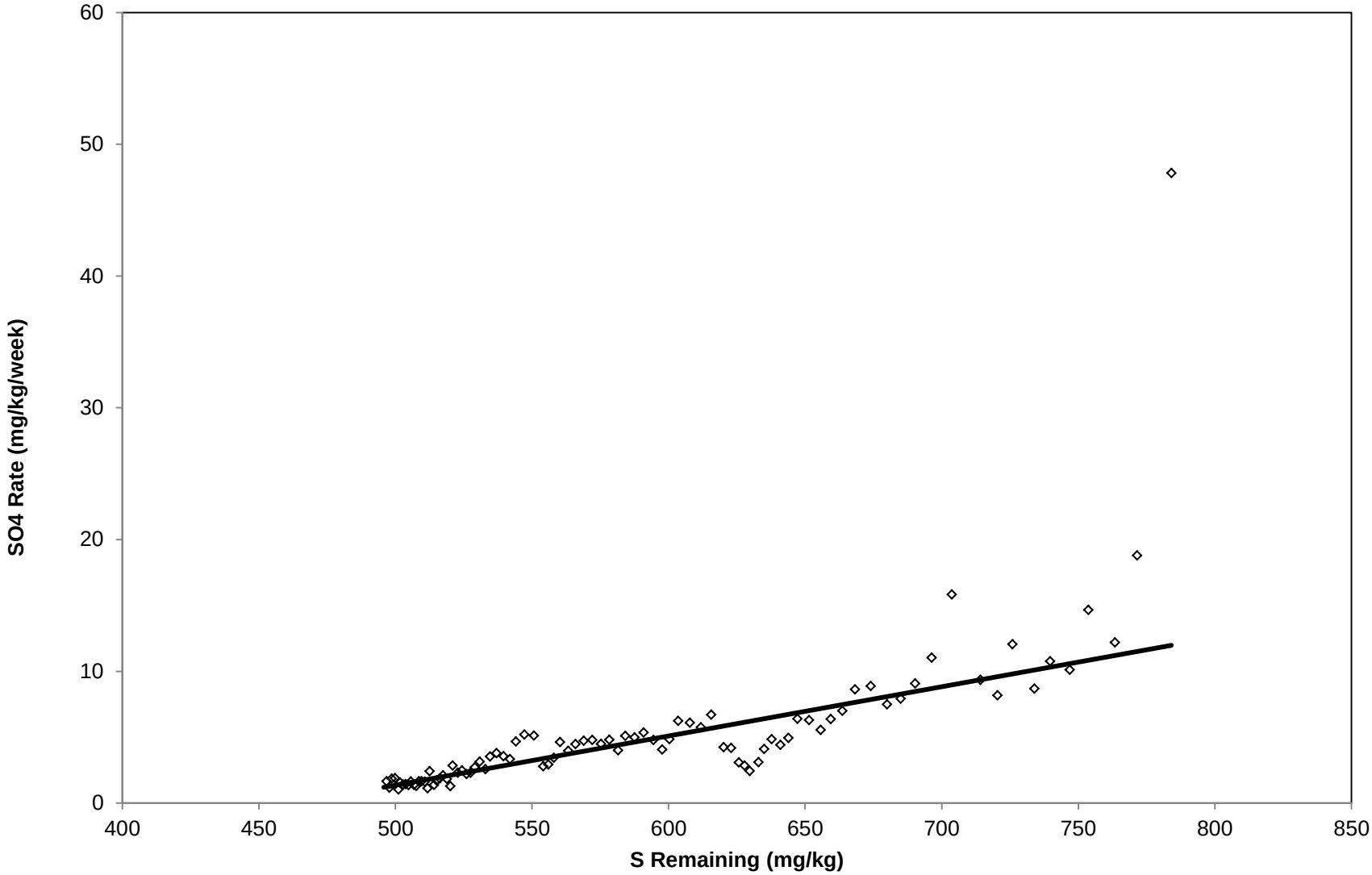

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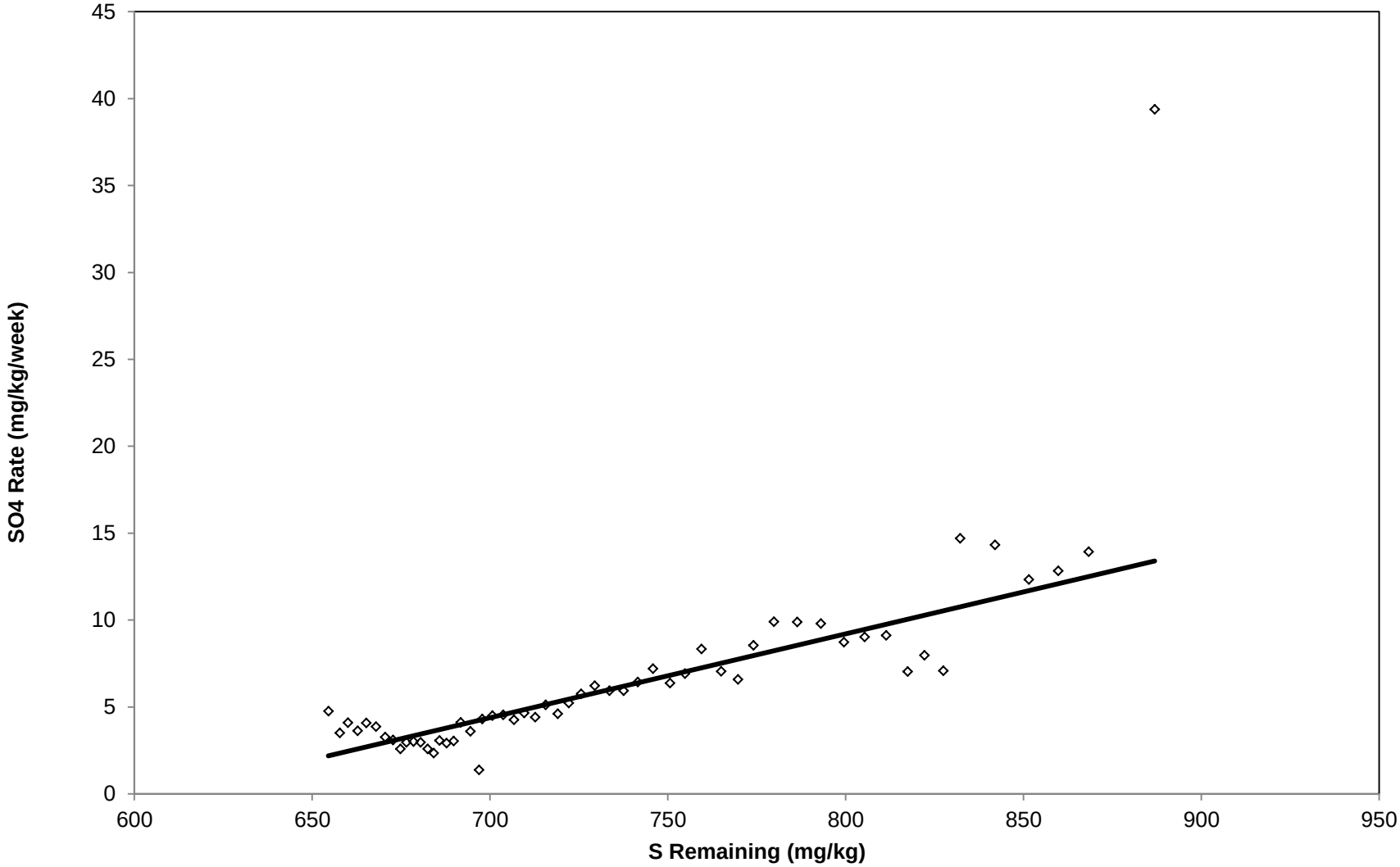

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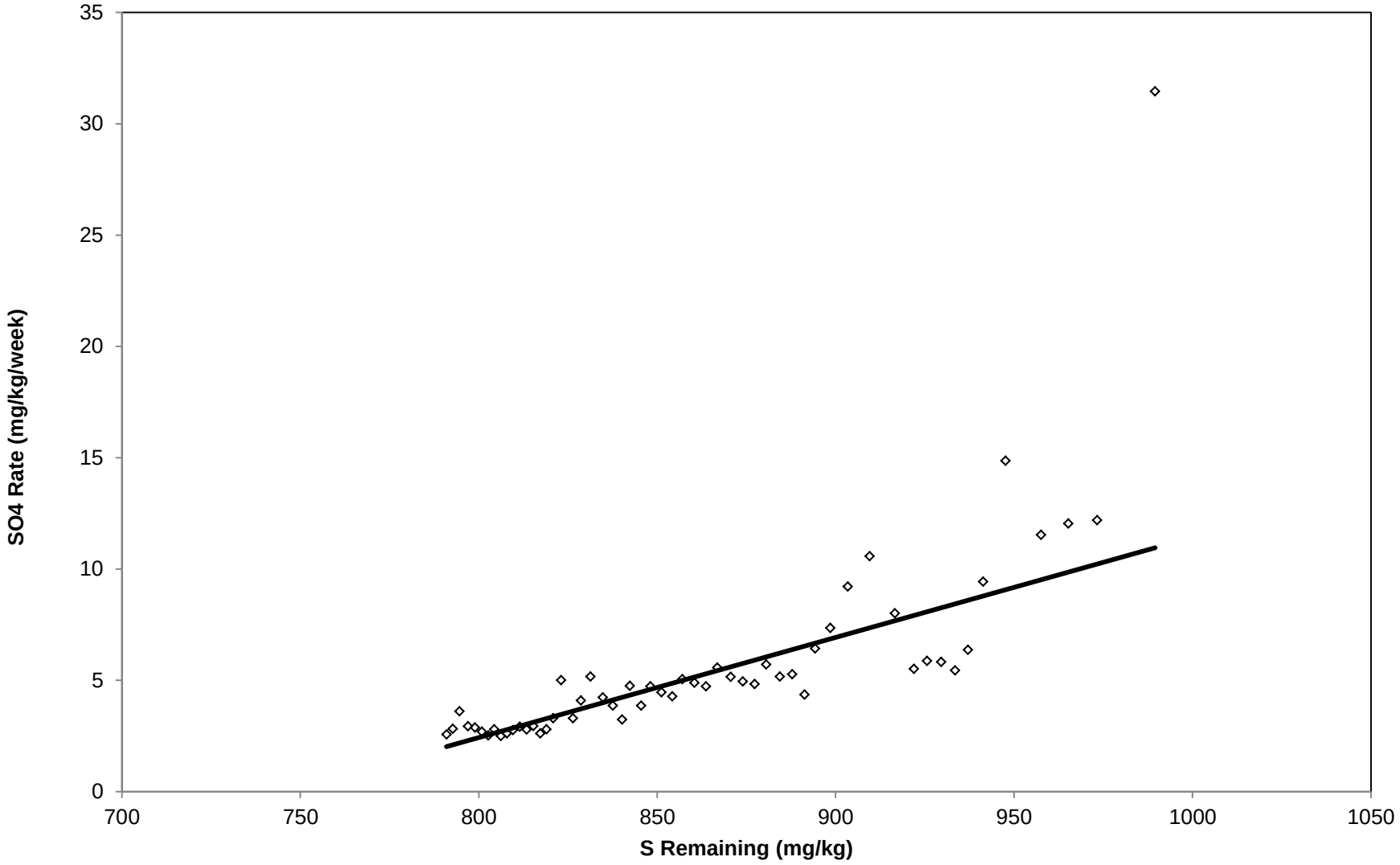

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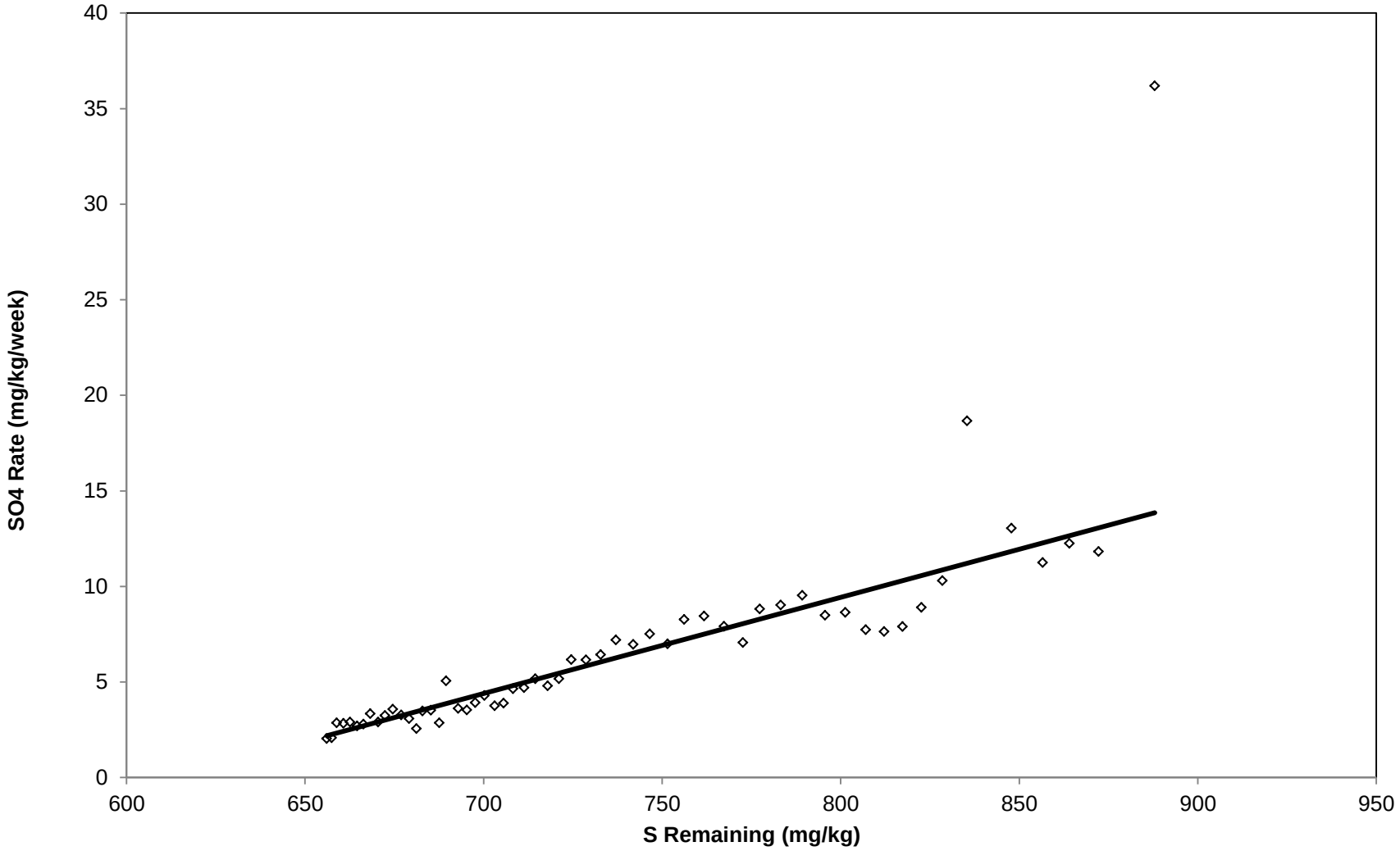

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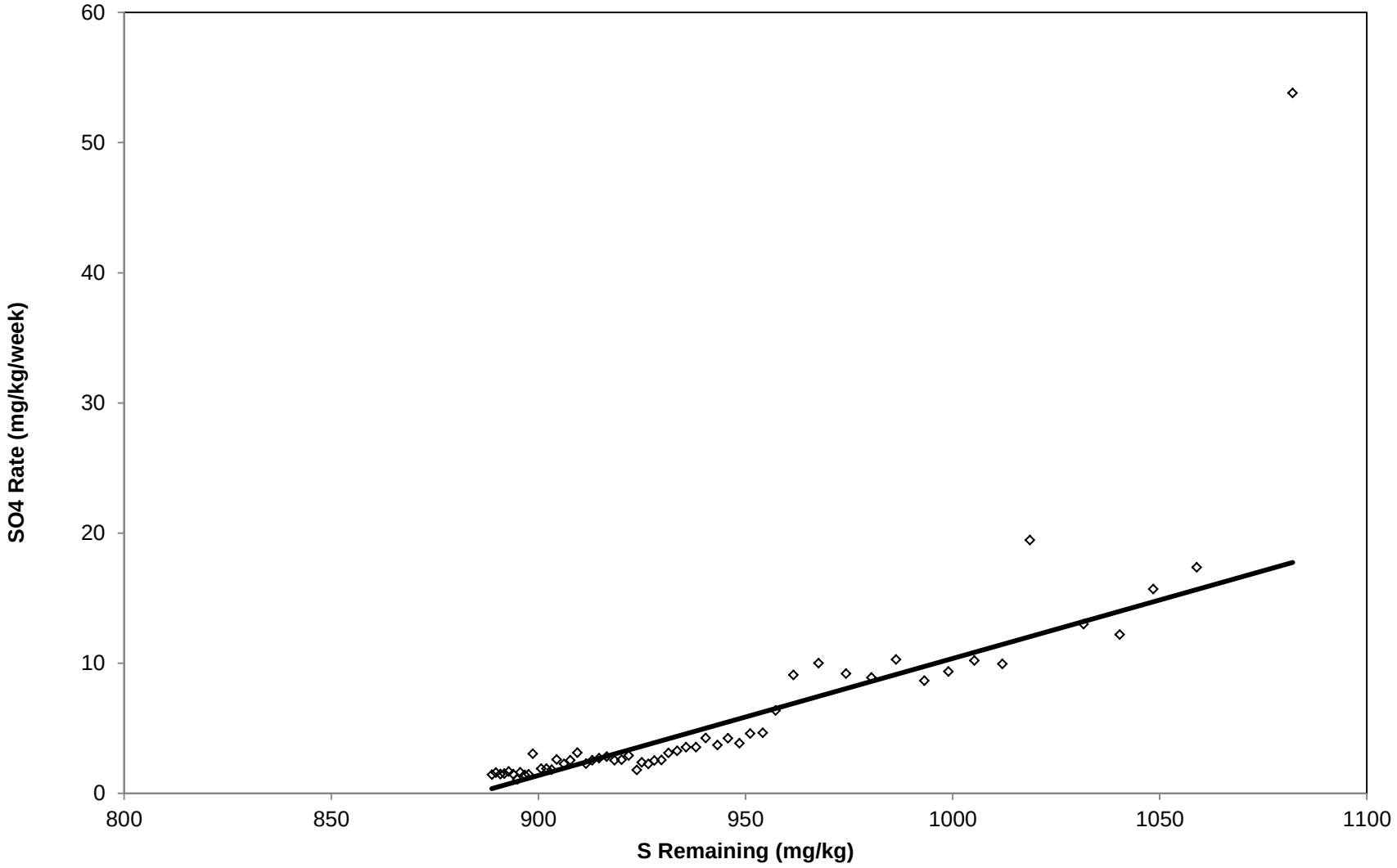

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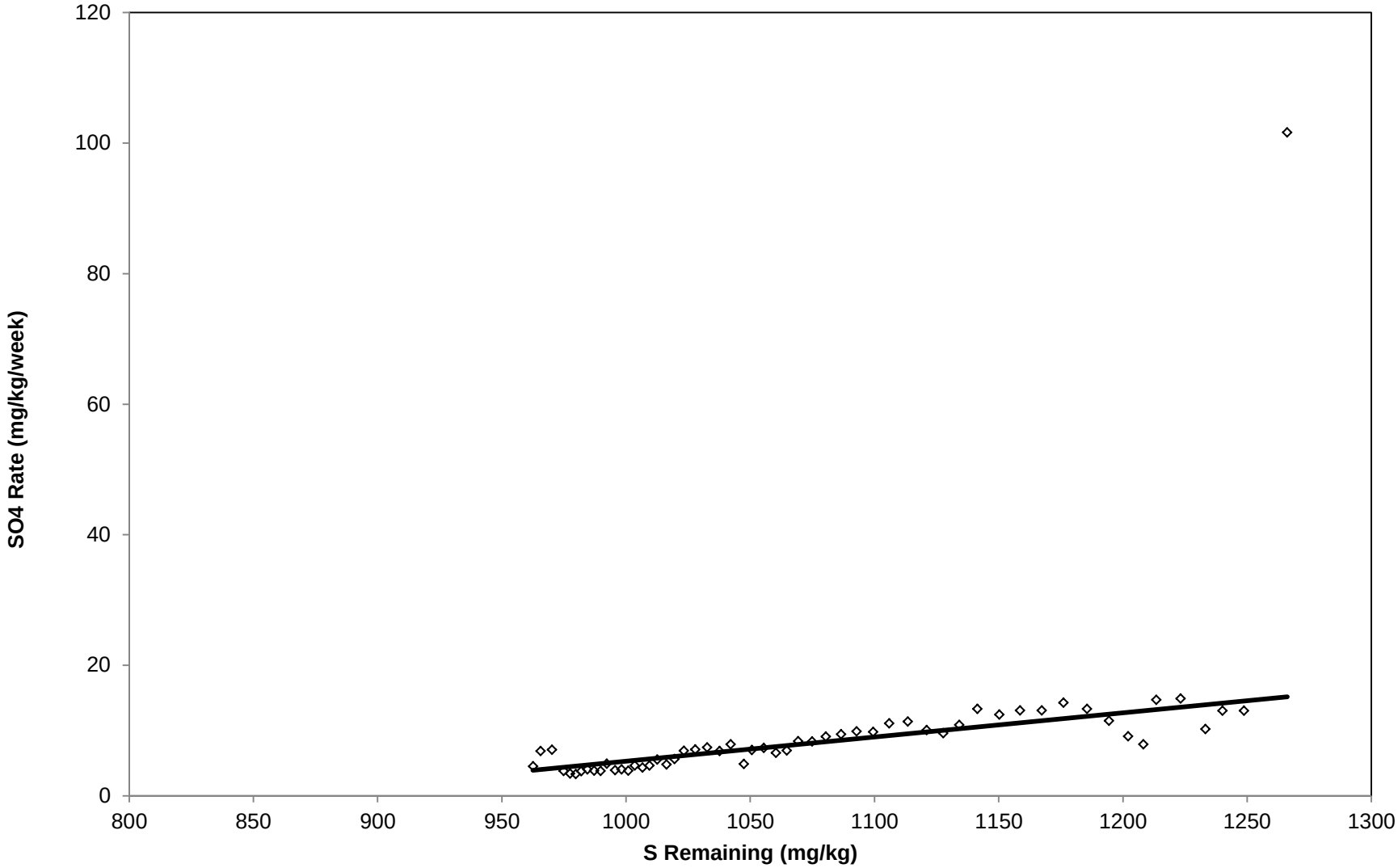

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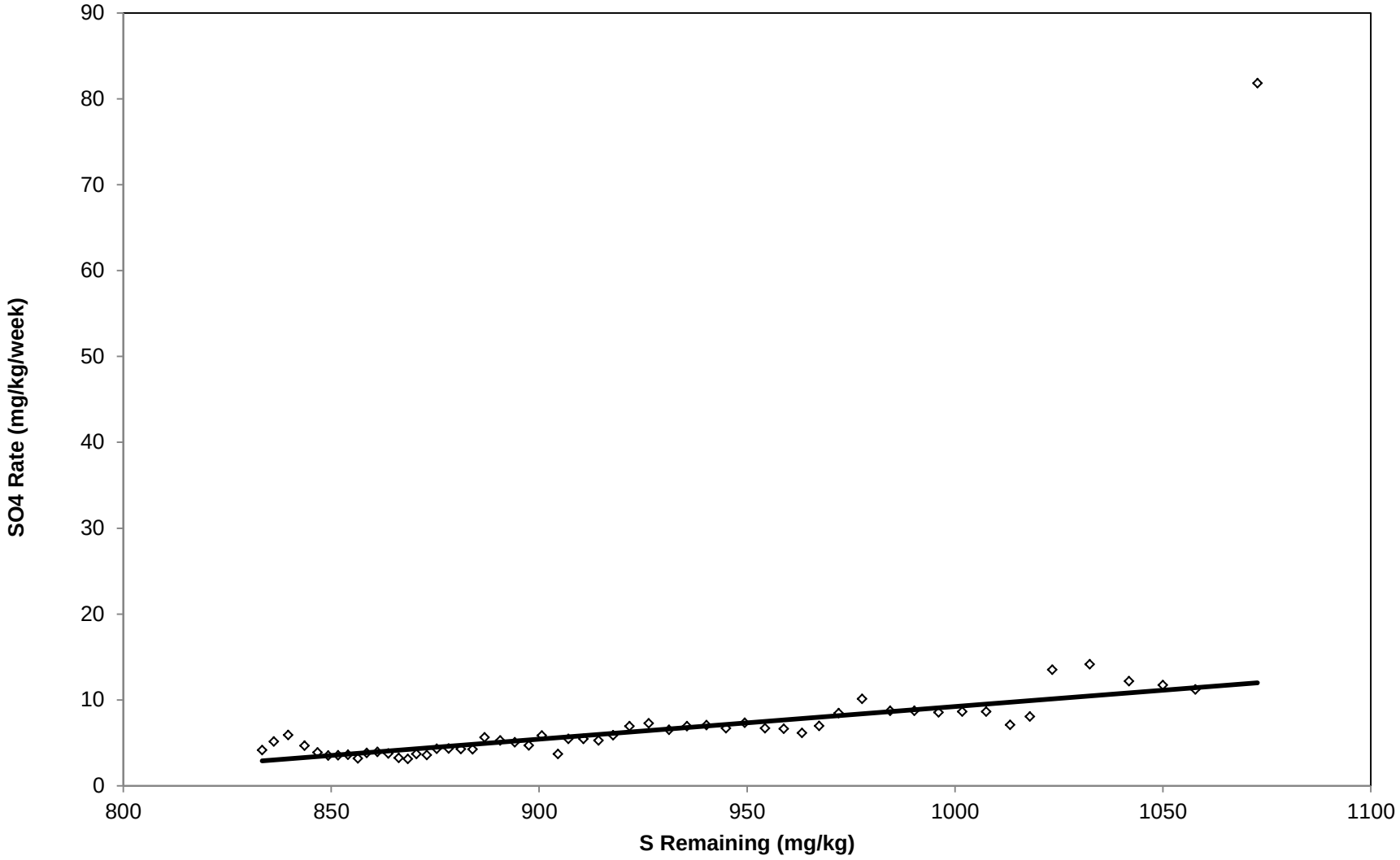

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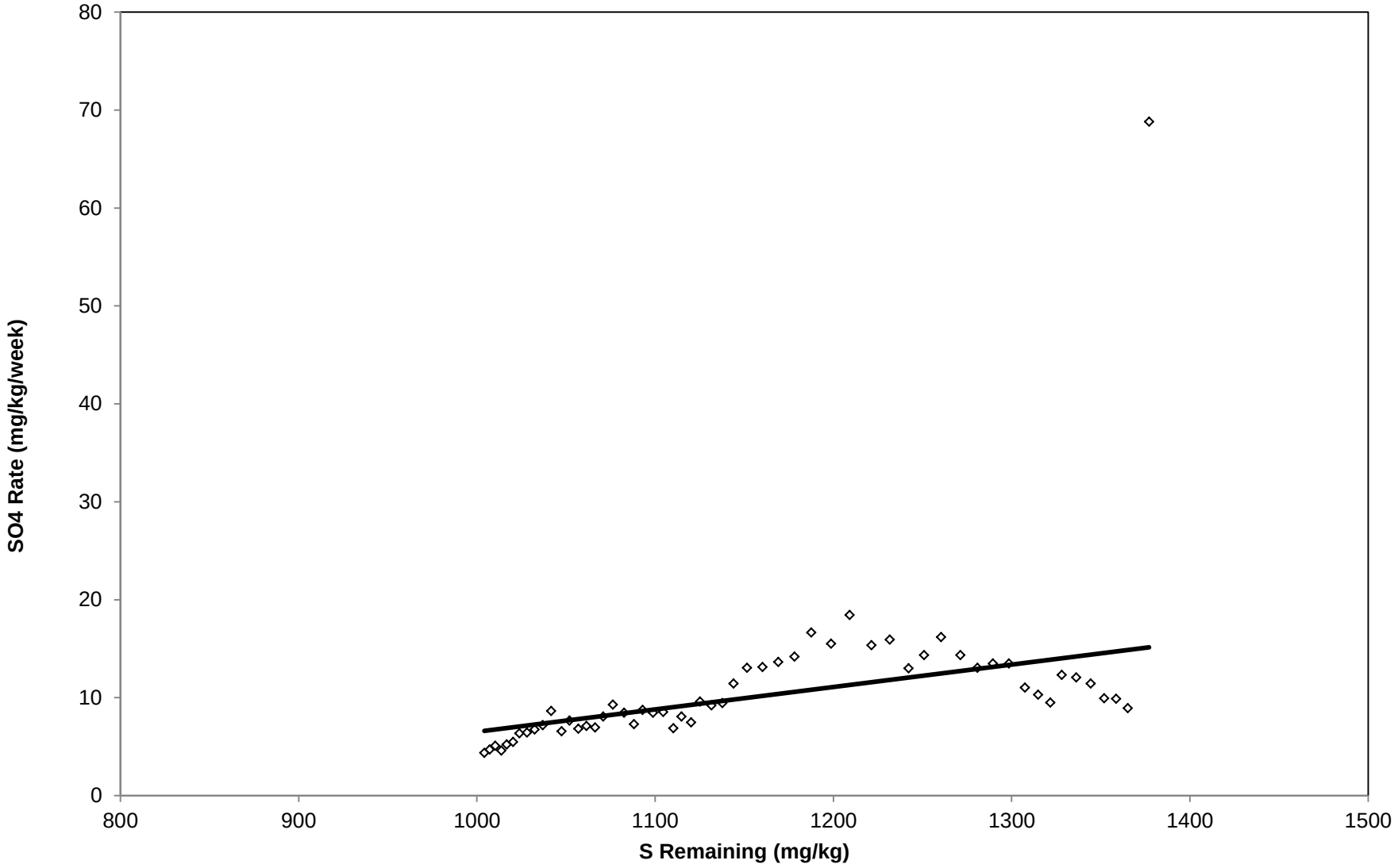

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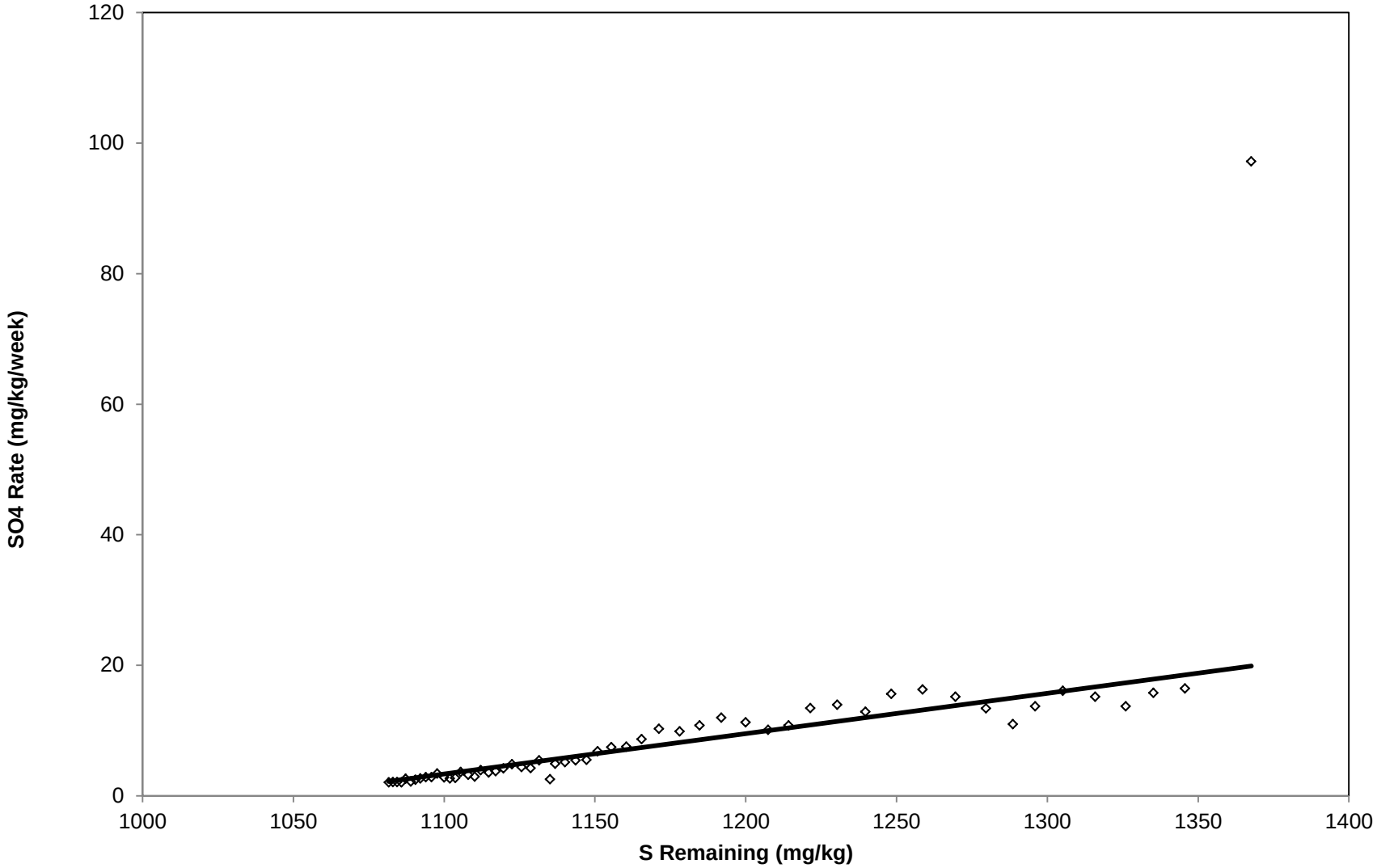

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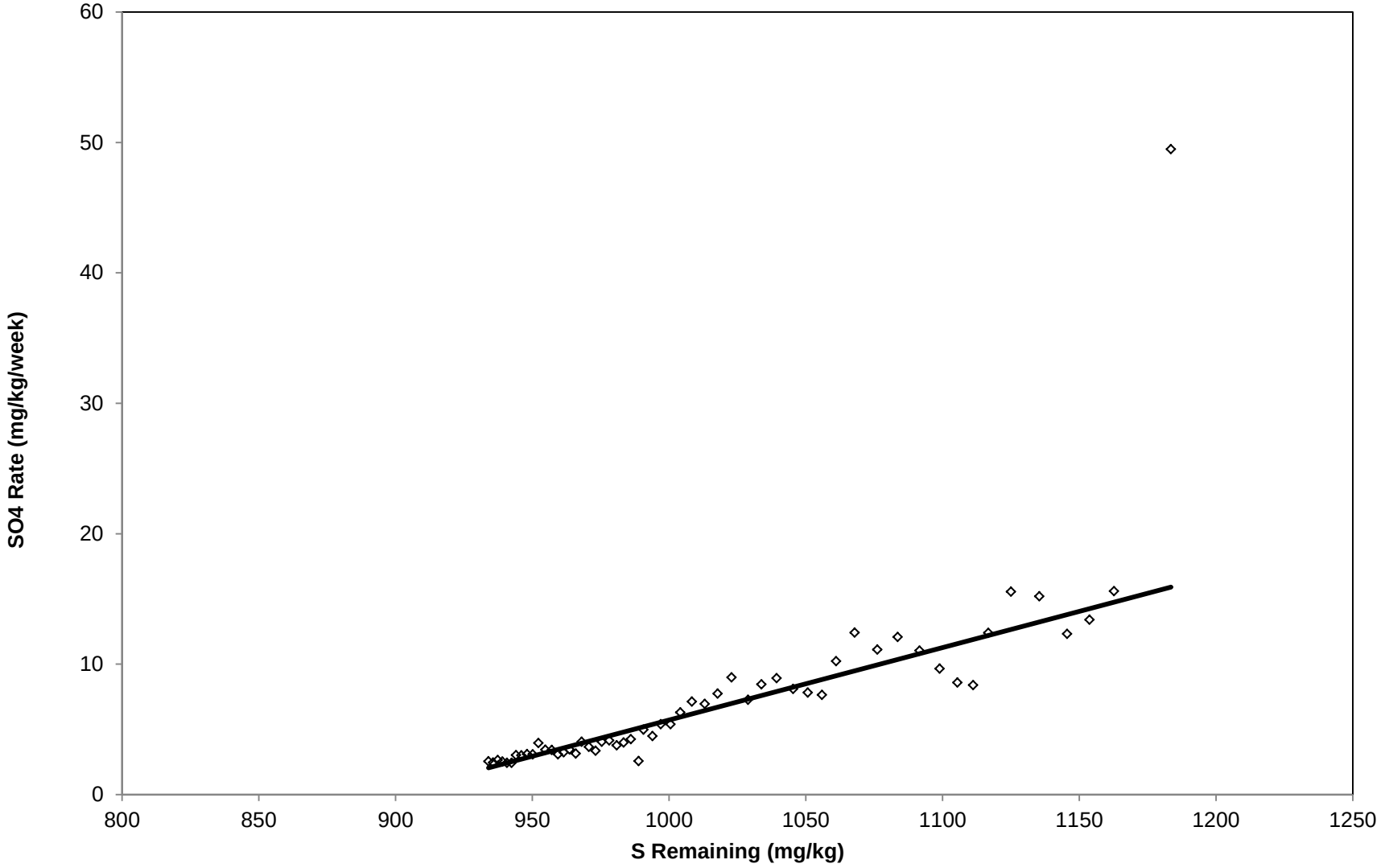

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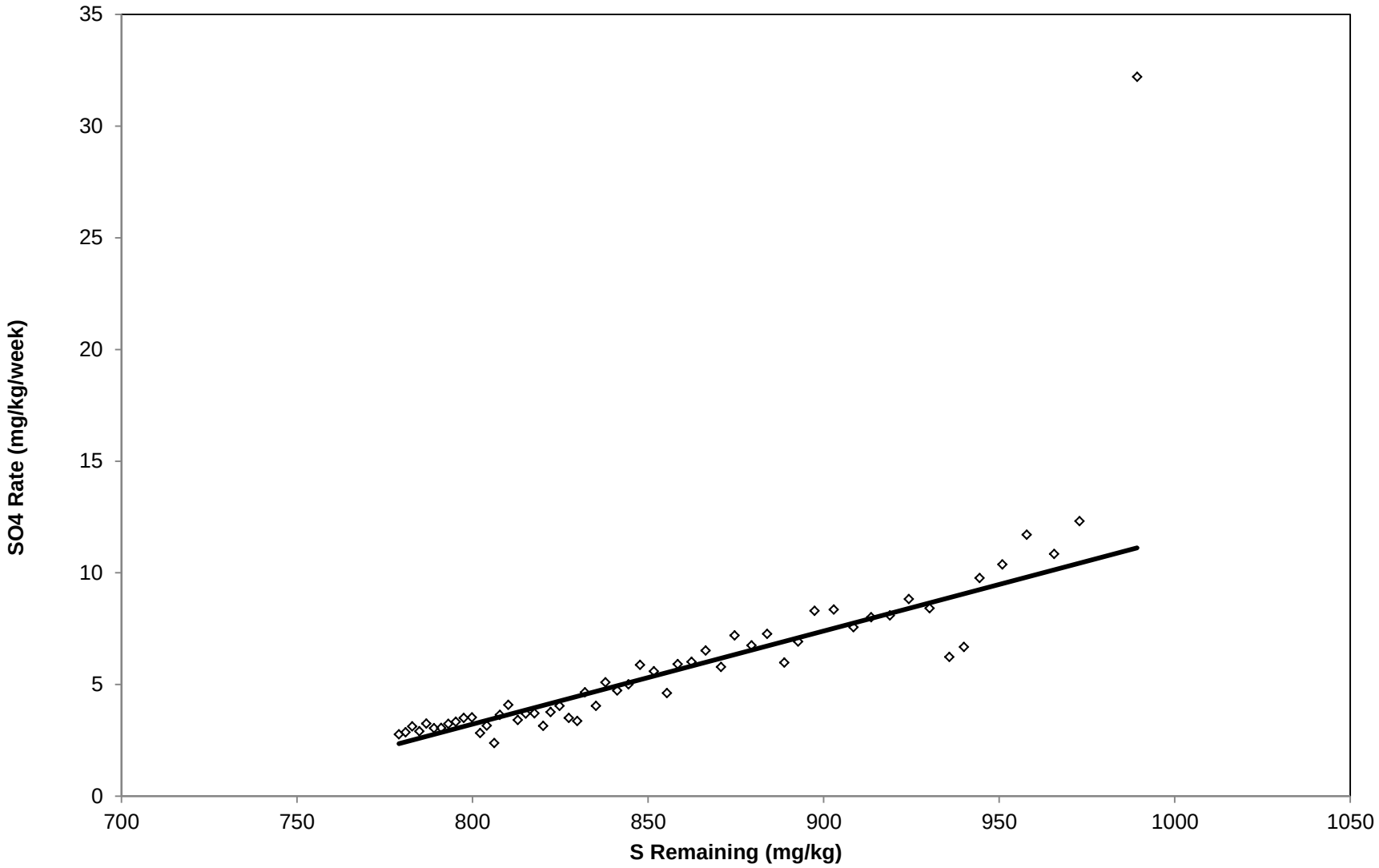

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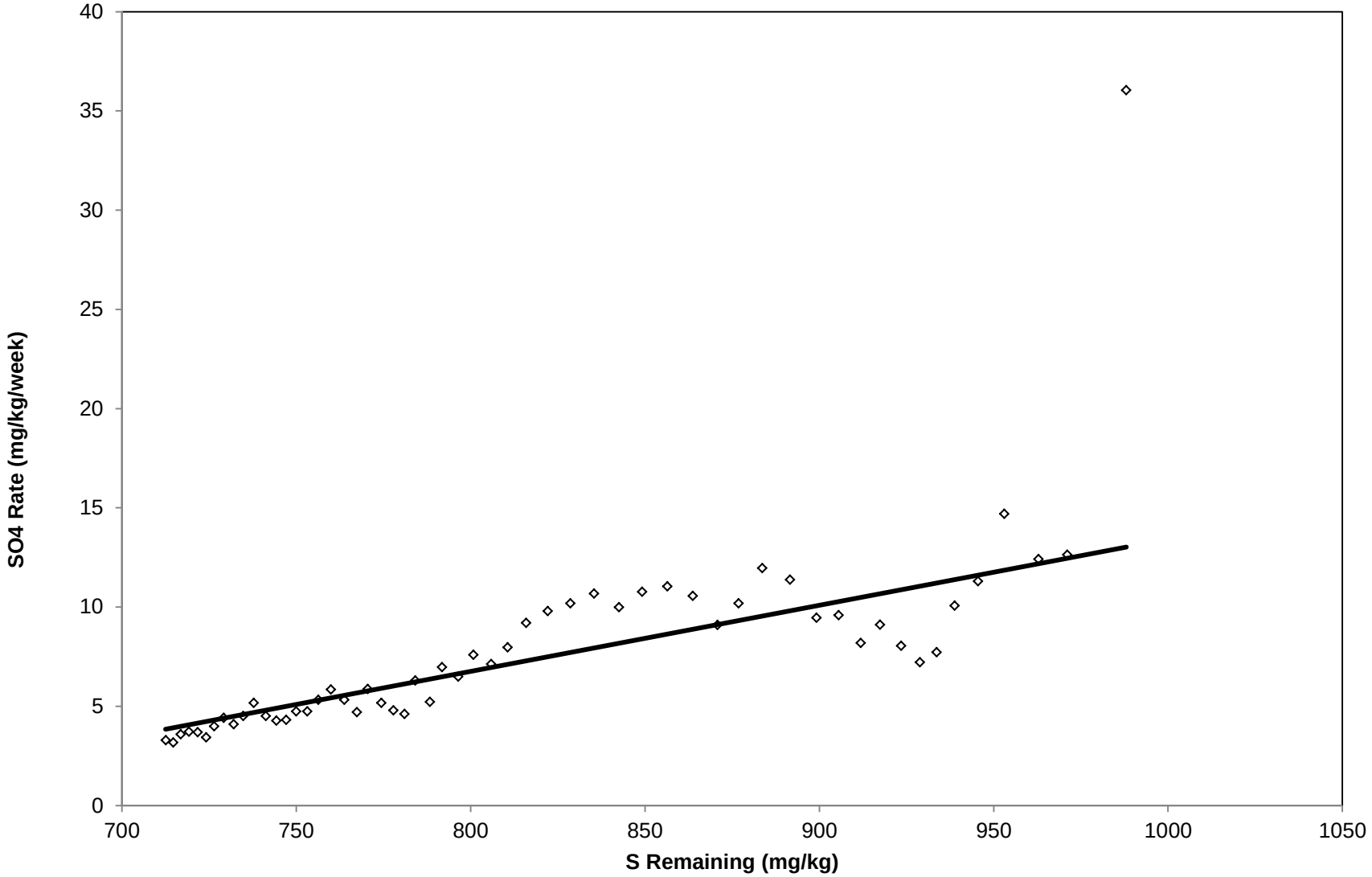

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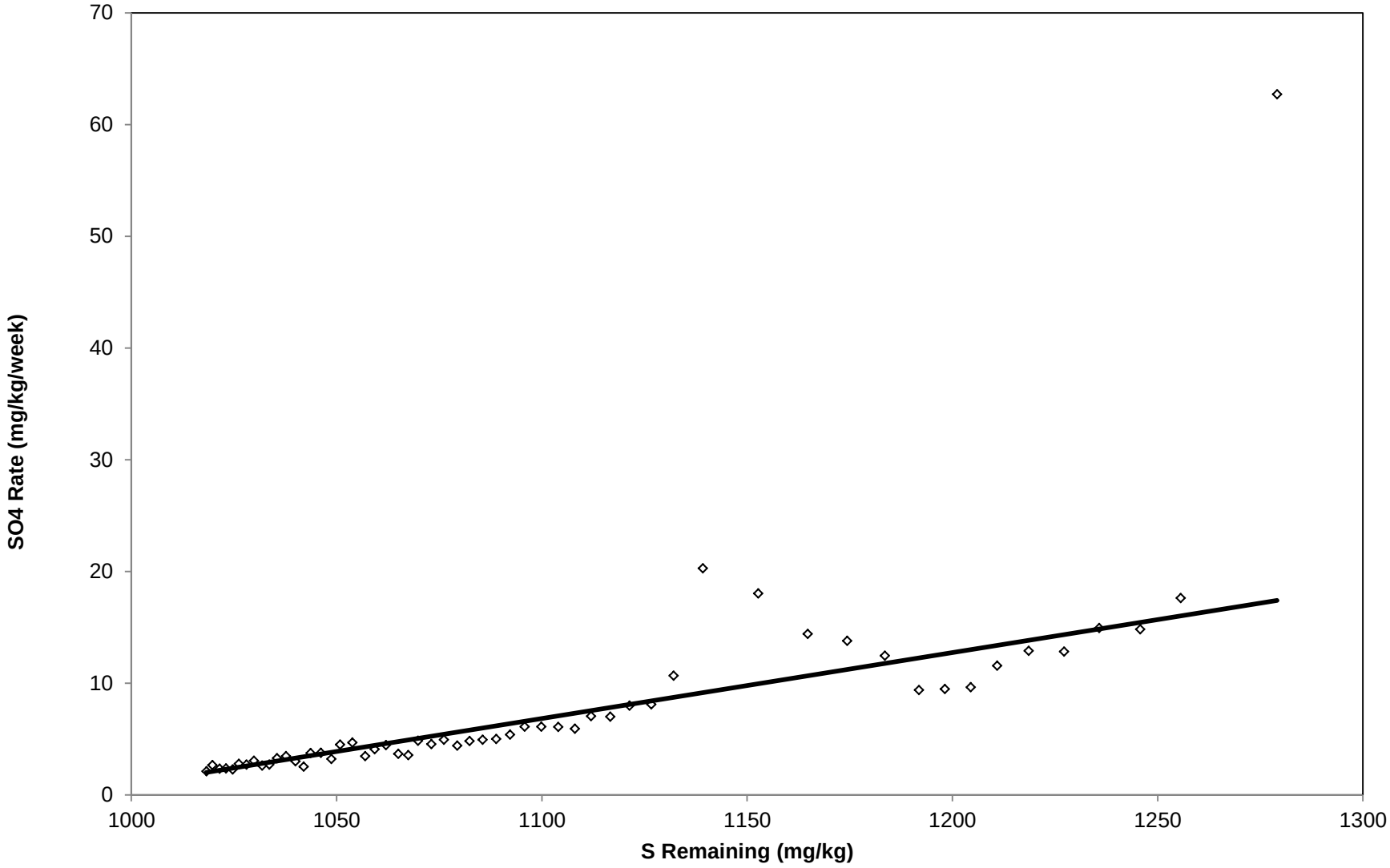

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Memo

To:	Jim Scott	**Date:**	January 21, 2015[1]
Company:	PolyMet Mining	**From:**	Stephen Day
Copy to:		**Project #:**	1UP005.001
Subject:	Update on Kinetic Test Data, NorthMet Project – DRAFT		

1 Introduction

Kinetic tests (mainly humidity cells) were initiated for the NorthMet Project in 2004 on samples of rock, simulated tailings and hydrometallurgical process residues according to overall program designs prepared following discussion between PolyMet Mining Inc. (PolyMet), Minnesota Department of Natural Resources (MDNR) and SRK Consulting (Canada) Inc. (SRK). Results from these tests have been used at various junctures of the project to develop waste management plans and evaluate project environmental effects. In 2009, the program was modified in consultation with MDNR to stop some tests and modify the frequency of analysis of leachates based on trend interpretation (SRK 2009).

The purpose of this memorandum is to provide an update on the progress of the testwork and transmit release rates as it relates to water quality source term predictions in the Environmental Impact Statement (EIS).

2 Data Presentation

Electronic updates to the data graphing and calculations are available on Barr Engineering's NorthMet Project website. The data period covered by the testwork now includes up to 6.5 years because some waste rock humidity cells were initiated in August 2005. The cutoff for interpreting data for this update was the end of January to early February 2012. Due to reduced analytical frequency for most tests (SRK 2009), complete QA review and update of the database occurs quarterly. Data collected subsequent to February 2012 completed QA after this update was prepared and are not included in this review.

3 Waste Rock, Lean Ore and Ore Kinetic Tests

3.1 Status of Program

The original waste rock humidity cell program consisted of 92 tests distributed according to the four initially defined classifications shown in Table 1. Waste rock was subsequently classified into three categories. The current program consists of 43 tests with most ongoing tasks in the mid-sulfur range for waste rock and lean ore (19 tests). Table 2 shows all samples and the testwork duration used to calculate rates for use in water quality predictions.

The primary uses of the data in the waste characterization program are to:

- Understand long term performance of wastes and inform definition of waste management categories based on sulfur content; and
- Provide leaching rates for use in source term geochemical predictions.

[1] This version replaces a memo of the same title dated October 19, 2012 to correct graphs shown in Figure 1.

Table 1. Sample Categories

Initial Classification	Waste Category	Totals	Stopped	Continuing
S≤0.05%	1	27	20	7
S>0.05%	1	10	6	4
	2/3	14	3	11
	4	13	8	5
Lean Ore	1	4	3	1
	2/3	13	5	8
	4	8	4	4
Ore	Not applicable	3	0	3
Totals	All	92	49	43

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Results are described below in the context of these two objectives. Each heading describes results according to the waste categories (1, 2/3 and 4). Charts for indicator parameters are shown to indicate major features (Figures 1 to 4). Overall tendency toward acidic conditions is shown by pH. Sulfate indicates sulfide mineral oxidation rates. Nickel is shown because as observed in this test program and testwork performed previously by DNR it (along with cobalt) tends to respond first to declining pH. Copper is shown because it leaches more rapidly as pH declines further. Arsenic provides an indication of how a heavy oxyanion is affected by changes in pH.

In order to obtain average leaching rates, typical trends have been recognized and average rates calculated for common qualitative leaching features or “conditions” as summarized in Table 3. In these definitions, “stable” means typically neither clearly increasing or decreasing. Due to variability in the testwork results, the definitions are not strict and require some flexibility in interpretation. For example, the transition from Condition 1 to 2 may not occur exactly at pH 7.

Condition 2 is typically accompanied by increasing release of cobalt and nickel which may continue into Condition 3. Lower pH (less than 6) typically occurs during Condition 3 and 4. In some cases, pH recovery is observed during Condition 4 as oxidation rates decrease.

Table 2. List of Rock Humidity Cells

HCT ID	Comment	Original Waste Type	Geological Unit	Rock Type	Category	Sample ID	S %	Initial Date	Data Record Length weeks	Duration Used in Current Modeling[2] weeks
1	9 Dup	RWR	1	Anorthositic	1	99-320C(830-850)	0.09	8/8/2005	337	284
3		RWR	1	Anorthositic	1	00-361C(345-350)	0.05	8/8/2005	337	284
26		NRWR	2	Anorthositic	1	00-366C(185-205)	0.02	8/9/2005	198	198
27		NRWR	2	Anorthositic	1	00-366C(230-240)	0.02	8/9/2005	198	198
28		NRWR	2	Anorthositic	1	99-320C(165-175)	0.03	8/9/2005	198	198
40		NRWR	3	Anorthositic	1	00-334C(30-50)	0.02	8/9/2005	337	284
41	37 Dup	NRWR	3	Anorthositic	1	00-368C(125-145)	0.04	8/10/2005	337	284
42		NRWR	3	Anorthositic	1	00-368C(20-40)	0.04	8/10/2005	198	198
13		NRWR	1	Troctolitic	1	00-340C(595-615)	0.04	8/10/2005	198	198
14		NRWR	1	Troctolitic	1	00-334C(580-600)	0.06	8/15/2005	336	284
15		RWR	1	Troctolitic	1	00-334C(640-660)	0.07	8/8/2005	337	284
16		RWR	1	Troctolitic	1	00-347C(795-815)	0.07	8/8/2005	198	198
29		NRWR	2	Troctolitic	1	99-318C(250-270)	0.04	8/9/2005	198	198
30		NRWR	2	Troctolitic	1	00-373C(95-115)	0.04	8/9/2005	198	198
31		NRWR	2	Troctolitic	1	00-373C(75-95)	0.06	8/9/2005	198	198
32		RWR	2	Troctolitic	1	00-357C(110-130)	0.08	8/9/2005	198	198
33		RWR	2	Troctolitic	1	99-320C(315-330)	0.07	8/9/2005	337	284
43		NRWR	3	Troctolitic	1	00-366C(35-55)	0.02	8/10/2005	198	198
44		NRWR	3	Troctolitic	1	00-334C(110-130)	0.04	8/10/2005	198	198
45		NRWR	3	Troctolitic	1	00-347C(155-175)	0.06	8/10/2005	198	198
46		RWR	3	Troctolitic	1	00-347C(280-300)	0.06	8/10/2005	198	198
49		NRWR	4	Troctolitic	1	00-367C(50-65)	0.03	8/10/2005	198	198
50		NRWR	4	Troctolitic	1	00-367C(260-280)	0.04	8/10/2005	198	198
51	57 Dup	NRWR	4	Troctolitic	1	00-367C(290-310)	0.04	8/10/2005	337	284
52		RWR	4	Troctolitic	1	00-370C(20-30)	0.08	8/10/2005	198	198
56		NRWR	5	Troctolitic	1	26064(44-54)	0.02	8/10/2005	337	284
59		NRWR	5	Troctolitic	1	26064(264+146-269+156)	0.06	8/10/2005	337	284
60		NRWR	6	Troctolitic	1	26056(110-125)	0.04	8/10/2005	198	198
74		NRWR	1	Troctolitic	1	26029(815-825)	0.02	9/8/2005	194	194
78		NRWR	6	Troctolitic	1	26056(135-153)	0.05	9/22/2005	331	278
99		LeanOre	1	Troctolitic	1	00-326C(250-265)	0.08	10/28/2005	187	186
21		RWR	1	Ultramafic	1	00-357C(335-340)	0.08	8/9/2005	198	198
35		NRWR	2	Ultramafic	1	00-368C(460-465)	0.06	8/9/2005	198	198
36		NRWR	2	Ultramafic	1	26055(940-945)	0.06	8/9/2005	198	198
39		NRWR	2	Ultramafic	1	26098+00-337C	0.1	8/9/2005	198	198
72	61 Dup	LeanOre	2	Ultramafic	1	00-361C(240-245)	0.06	8/11/2005	337	284
101		LeanOre	2	Ultramafic	1	26039(310-315)	0.06	10/28/2005	187	186
2		RWR	1	Anorthositic	2/3	00-361C(310-320)	0.18	8/8/2005	337	284
103		LeanOre	1	Anorthositic	2/3	99-320C(400-405)	0.18	10/28/2005	326	273
5		RWR	1	Sed Honfels	2/3	26030(1047-1052)	0.24	8/8/2005	337	284
6		RWR	1	Sed Honfels	2/3	26061(1218-1233)	0.44	8/8/2005	337	284
7		RWR	1	Sed Honfels	2/3	00-340C(990-995)	0.55	8/8/2005	337	284
17		RWR	1	Troctolitic	2/3	00-350C(580-600)	0.19	8/8/2005	337	284
18		RWR	1	Troctolitic	2/3	00-327C(225-245)	0.44	8/8/2005	198	198
34		RWR	2	Troctolitic	2/3	00-369C(335-345)	0.18	8/9/2005	337	284
47		RWR	3	Troctolitic	2/3	00-326C(60-70)	0.14	8/10/2005	337	284
48	38 Dup	RWR	3	Troctolitic	2/3	00-369C(305-325)	0.25	8/10/2005	198	198
53		RWR	4	Troctolitic	2/3	00-369C(20-30)	0.21	8/10/2005	337	284
54		RWR	4	Troctolitic	2/3	00-367C(170-175)	0.51	8/10/2005	337	284
71		LeanOre	2	Troctolitic	2/3	00-340C(380-390)	0.15	8/11/2005	198	198
75		LeanOre	3	Troctolitic	2/3	26049+26030	0.59	8/11/2005	198	198
77		LeanOre	5	Troctolitic	2/3	26056(302-312)	0.23	8/11/2005	337	284
80		LeanOre	6	Troctolitic	2/3	26142(360+345-365+350)	0.18	8/11/2005	337	284
96		LeanOre	2	Troctolitic	2/3	99-318C(325-330)	0.17	10/28/2005	326	273
98		LeanOre	5	Troctolitic	2/3	26056(282-292)	0.32	10/28/2005	187	186
100		LeanOre	1	Troctolitic	2/3	00-340C(910-925)	0.36	10/28/2005	326	273
102		LeanOre	1	Troctolitic	2/3	00-331C(190-210)	0.42	10/28/2005	326	273
105		LeanOre	3	Troctolitic	2/3	00-367C(495-500)	0.28	10/28/2005	326	273
22		RWR	1	Ultramafic	2/3	00-326C(680-685)	0.30	8/9/2005	198	198
23		RWR	1	Ultramafic	2/3	00-357C(535-540)	0.2	8/9/2005	337	284
94		LeanOre	1	Ultramafic	2/3	00-344C(630-635)	0.34	10/28/2005	187	186
95		LeanOre	1	Ultramafic	2/3	00-326C(495-505)	0.16	10/28/2005	187	186

[2] Waste Characterization Data Package Version 9 (July 3 2012).

HCT ID	Comment	Original Waste Type	Geological Unit	Rock Type	Category	Sample ID	S %	Initial Date	Data Record Length weeks	Duration Used in Current Modeling[2] weeks
104		LeanOre	2	Ultramafic	2/3	00-326C(225-235)	0.12	10/28/2005	326	273
4		RWR	1	Anorthositic	4	00-343C(240-250)	0.68	8/8/2005	198	198
65		LeanOre	1	Anorthositic	4	26027(616-626)	1.83	8/11/2005	337	284
93		LeanOre	1	Anorthositic	4	00-331C(255-260)	0.86	10/28/2005	326	273
8		RWR	1	Sed Honfels	4	00-340C(965-974.5)	1.74	8/8/2005	198	198
11		RWR	1	Sed Honfels	4	26043+26027	2.47	8/8/2005	337	284
68		LeanOre	1	Sed Honfels	4	26062+26026	4.46	8/11/2005	337	284
106		LeanOre	1	Sed Honfels	4	26058(704-715)	1.46	10/28/2005	326	273
19		RWR	1	Troctolitic	4	00-371C(435-440)	0.88	8/8/2005	337	284
20	10 Dup	RWR	1	Troctolitic	4	00-340C(765-780)	1.68	8/8/2005	337	284
55		RWR	4	Troctolitic	4	00-367C(395-400)	0.77	8/10/2005	198	198
69		LeanOre	1	Troctolitic	4	00-340C(725-745)	0.91	8/11/2005	198	198
76		LeanOre	4	Troctolitic	4	00-367(400-405)	1.37	8/11/2005	198	198
24		RWR	1	Ultramafic	4	99-318C(725-735)	0.72	8/9/2005	198	198
25		RWR	1	Ultramafic	4	99-317C(460-470)	1.24	8/9/2005	198	198
70		LeanOre	1	Ultramafic	4	00-344C(515-520)	1.2	8/11/2005	198	198
97		LeanOre	1	Ultramafic	4	00-330C(275-280)	0.75	10/28/2005	187	186
62		RWR	20	Virginia	4	00-361C(737-749)	2	8/11/2005	337	284
63	58 Dup	RWR	20	Virginia	4	00-364C(210-229)	3.79	8/11/2005	198	198
64		RWR	20	Virginia	4	00-337C(510-520)	5.68	8/11/2005	198	198
66		Ore			-	P10	0.86	9/8/2005	333	268
67		Ore			-	P20	0.9	9/8/2005	333	268
73		Ore			-	P30	0.86	9/8/2005	333	268

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Table 3. Qualitative Definition of Leaching Conditions

Condition	Trend Features
1	pH>7, stable SO_4
2	pH<7, typically stable SO_4
3	SO_4 sharply increasing and unstable
4	SO_4 peaked and decreasing

3.2 Description of Results by Waste-Rock Category

3.2.1 Category 1

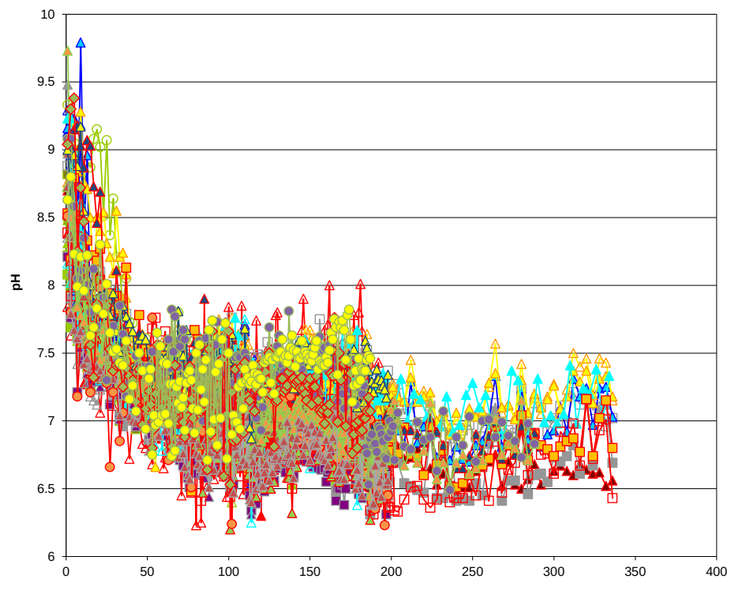
Results for pH, sulfate, nickel, copper and arsenic are shown in Figure 1.

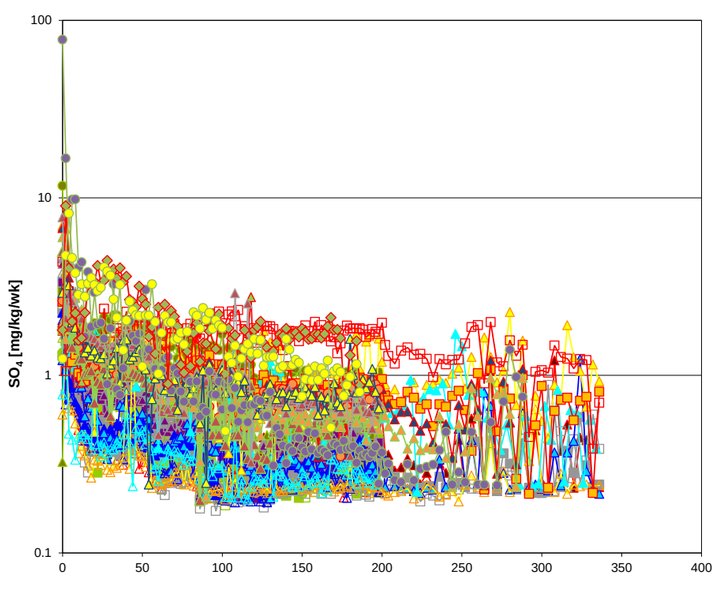
All Category 1 humidity cells have yielded pH above 6 throughout the program. Generally, pHs declined as the tests proceeded. Initial pHs in most cases were above 8 but declined rapidly and have typically fluctuated between 6.5 and 7.5. There is no indication for the 12 continuing tests that pHs are on a declining trend. In a few cases, pHs have recovered slightly. At the same time, the alkalinity trend is stable.

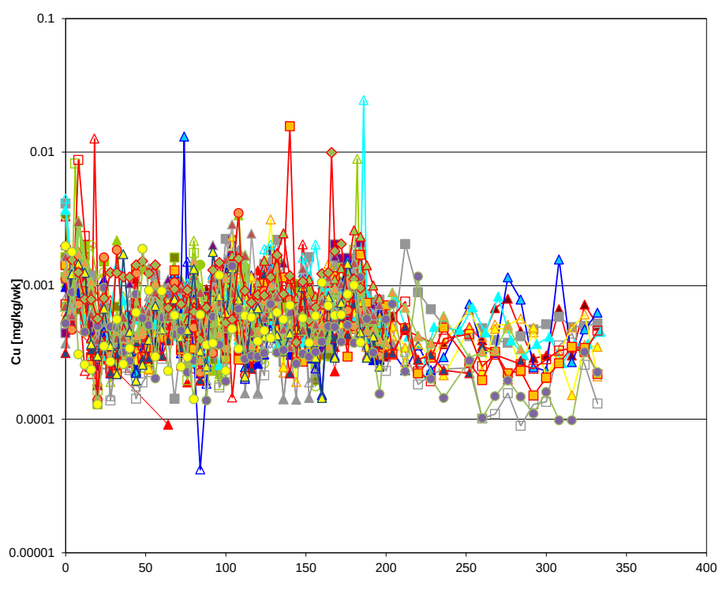
Sulfate leaching rates have been low throughout the program with most continuing tests showing rates below 1 mg/kg/week. At these low rates, variability in trends is apparent but no clear upward trends have been apparent.

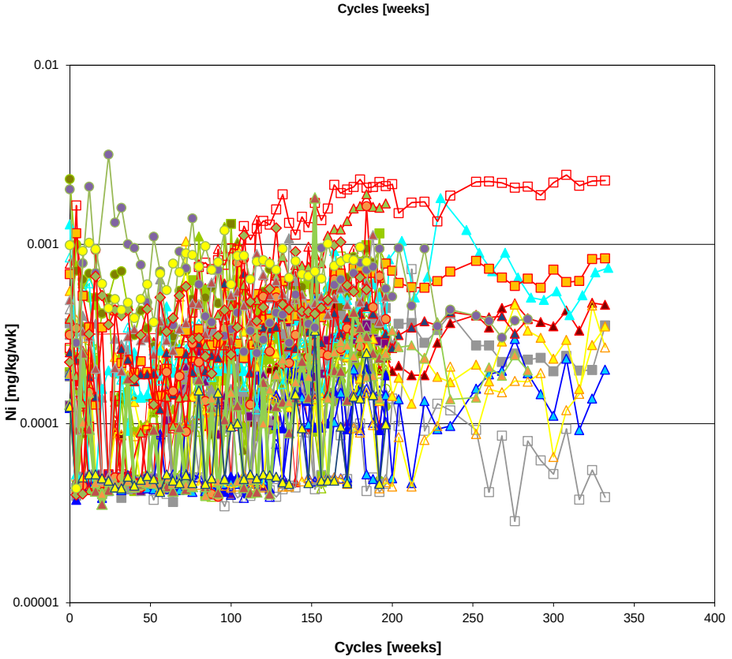
Nickel showed slight increases in leaching rates relative to other tests in the few cells in which pH decreases and peak nickel leaching rates were observed in cell 99-320C(830-850). This sample also showed the highest sulfate and cobalt leaching rates. Copper generally showed stable leaching rates.

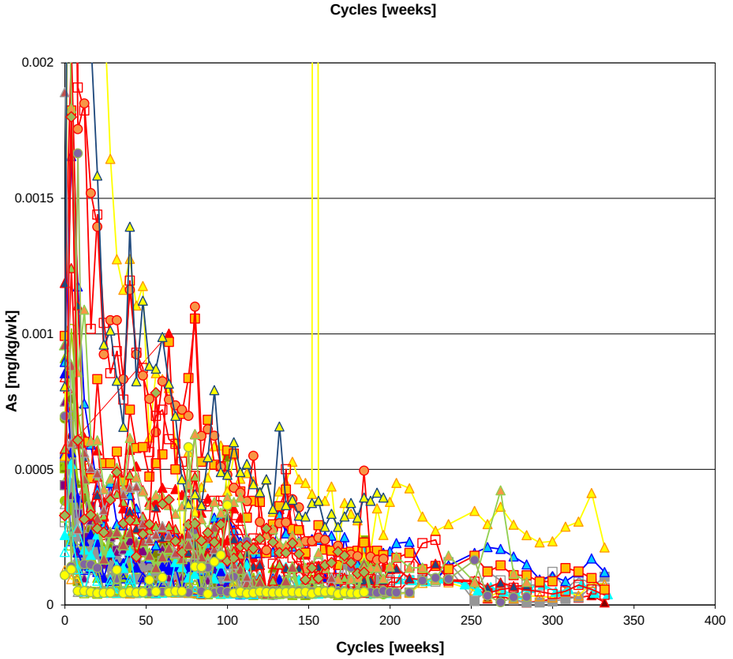
Arsenic showed no increase in leaching rates due to pH changes. Arsenic leaching has either stabilized or decreased steadily.











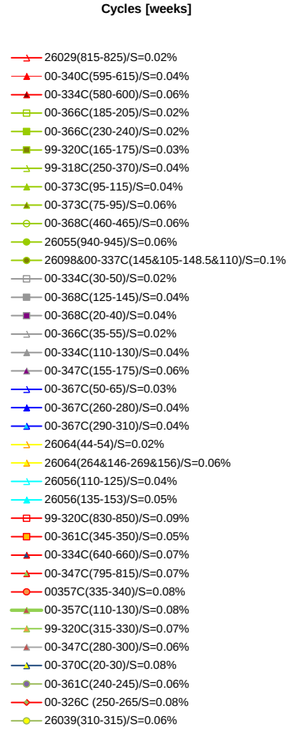


Figure 1. Leaching Trends for Category 1 Samples

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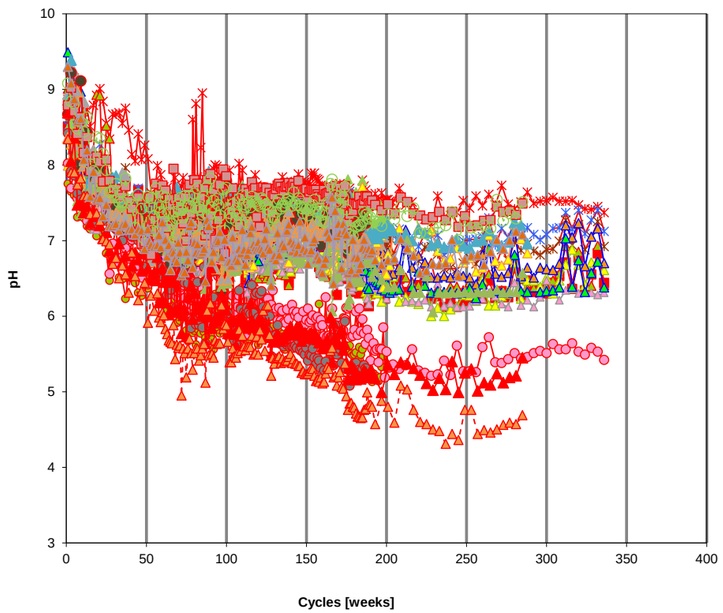

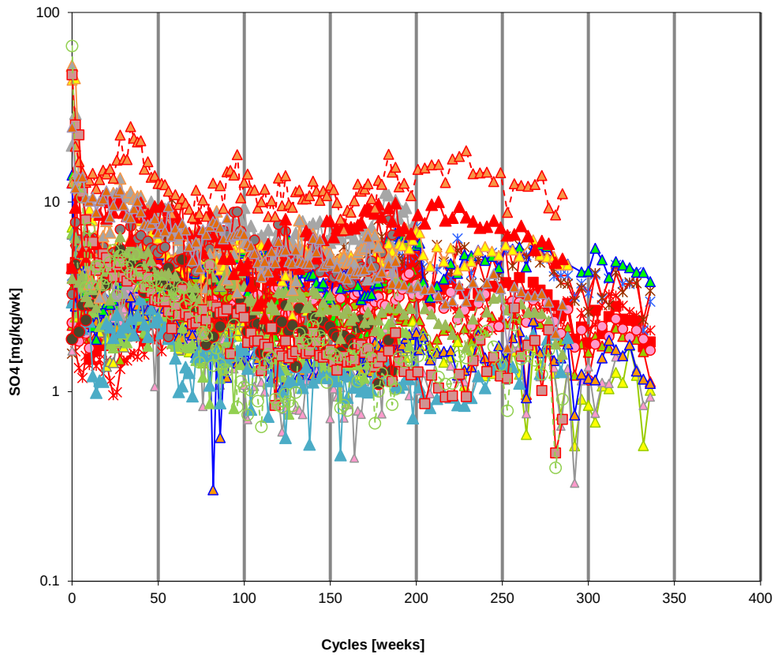

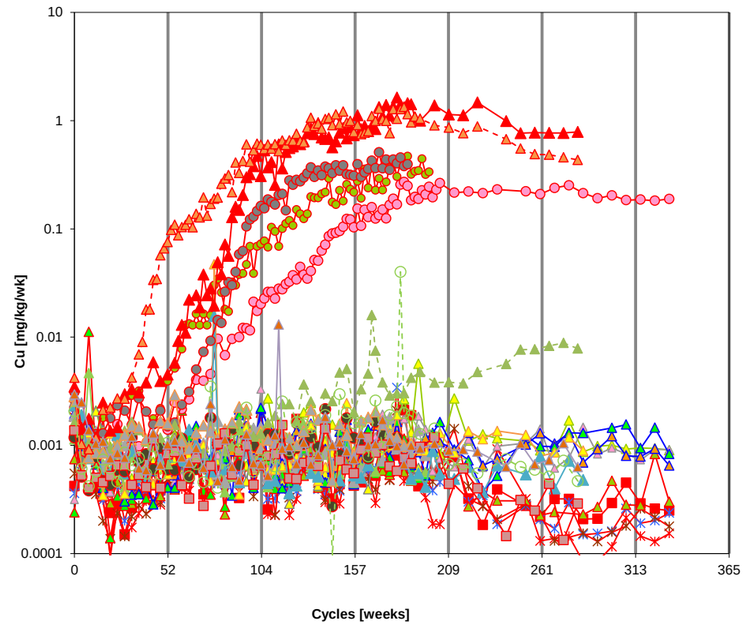

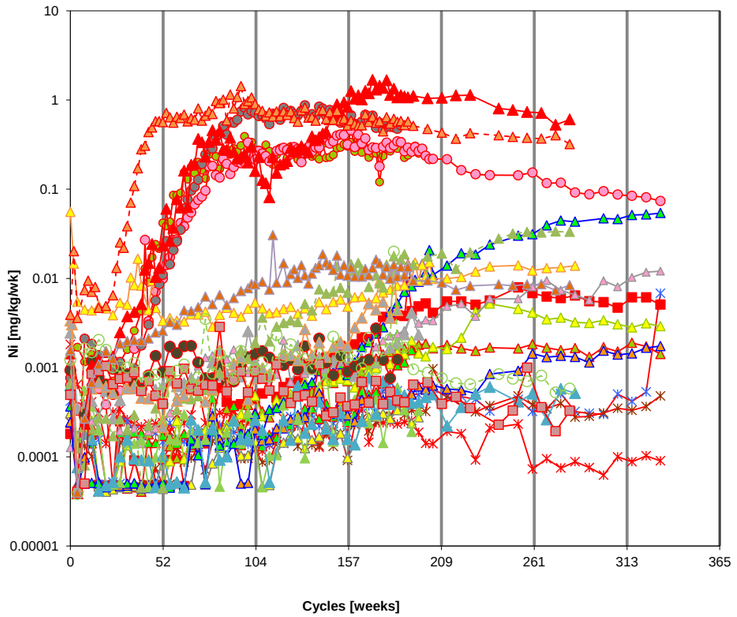

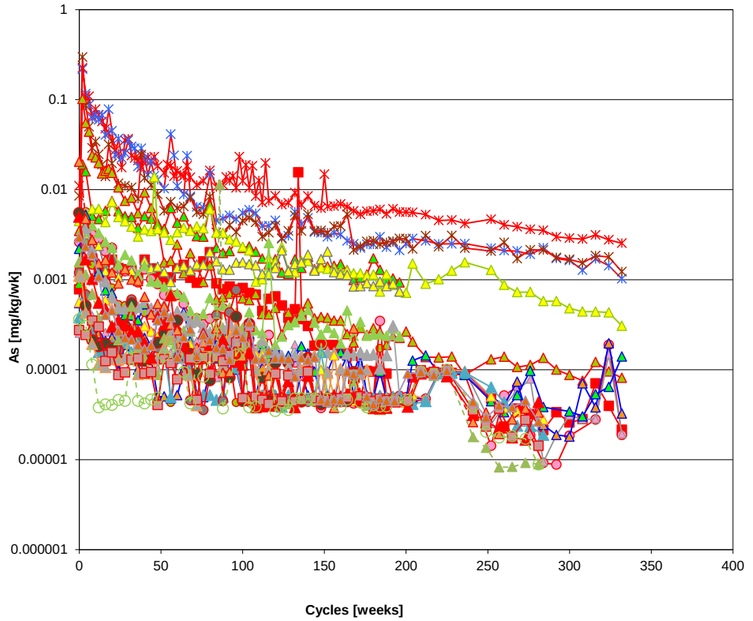

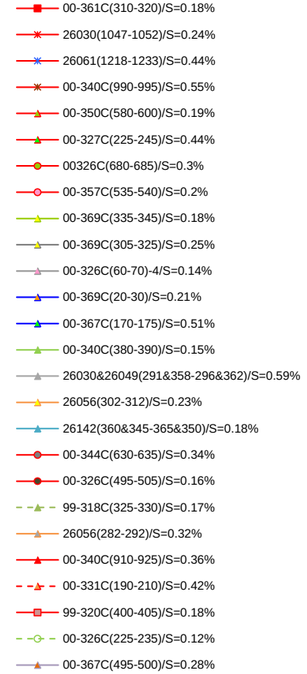

Figure 2. Leaching Trends for Category 2/3 Samples

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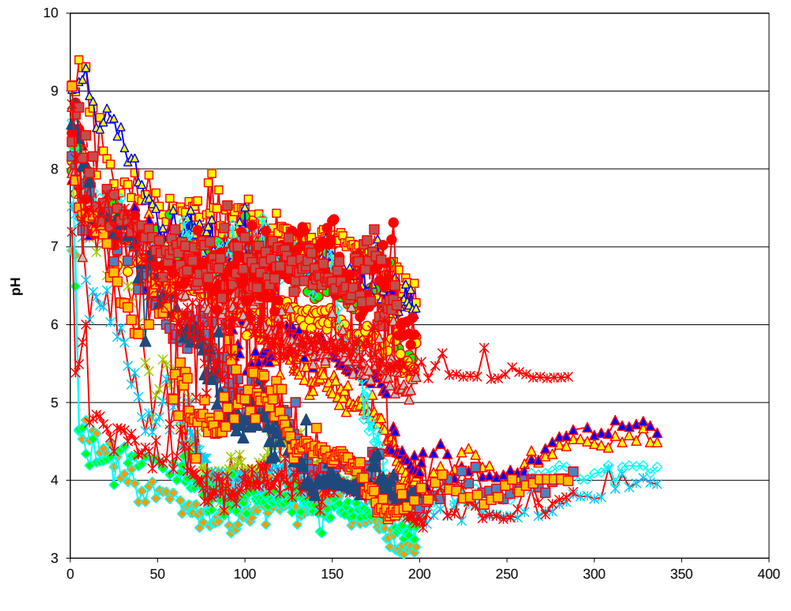

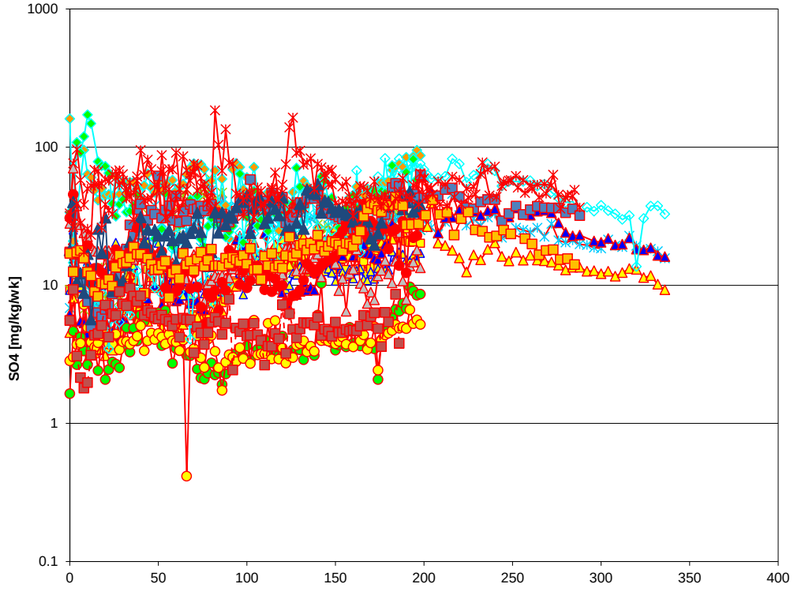

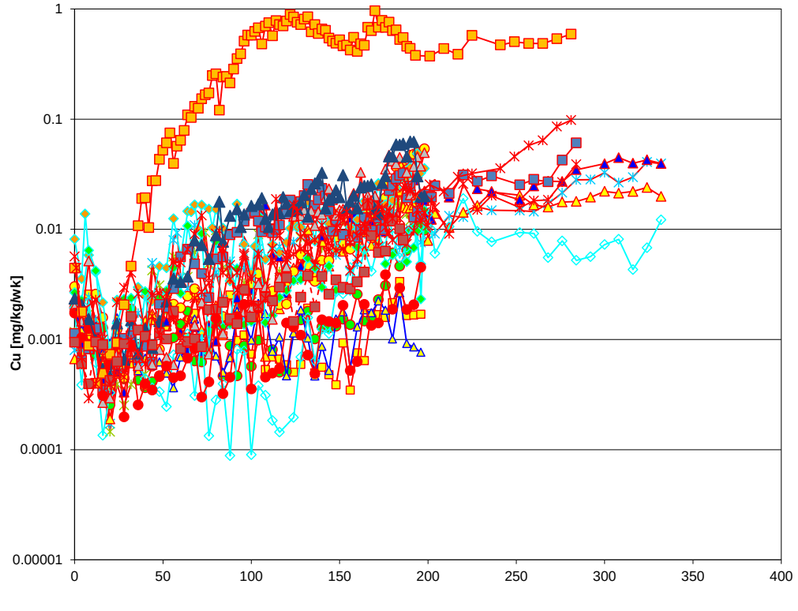

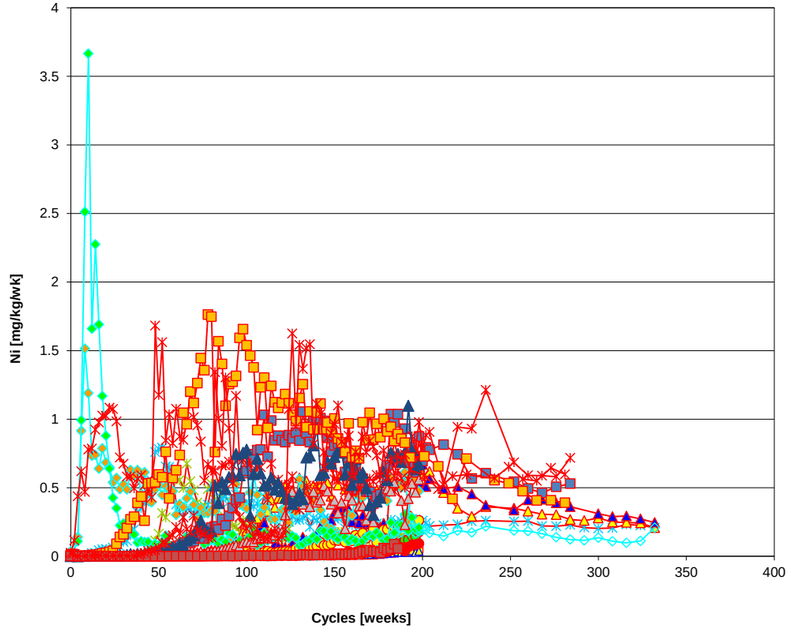

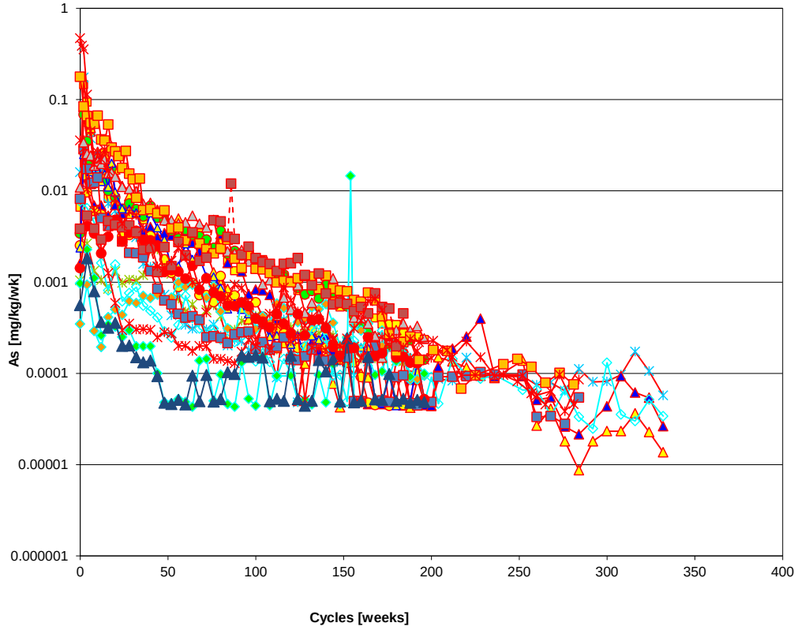

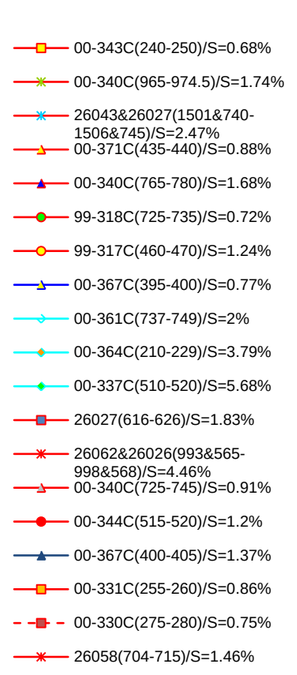

Figure 3. Leaching Trends for Category 4 Samples

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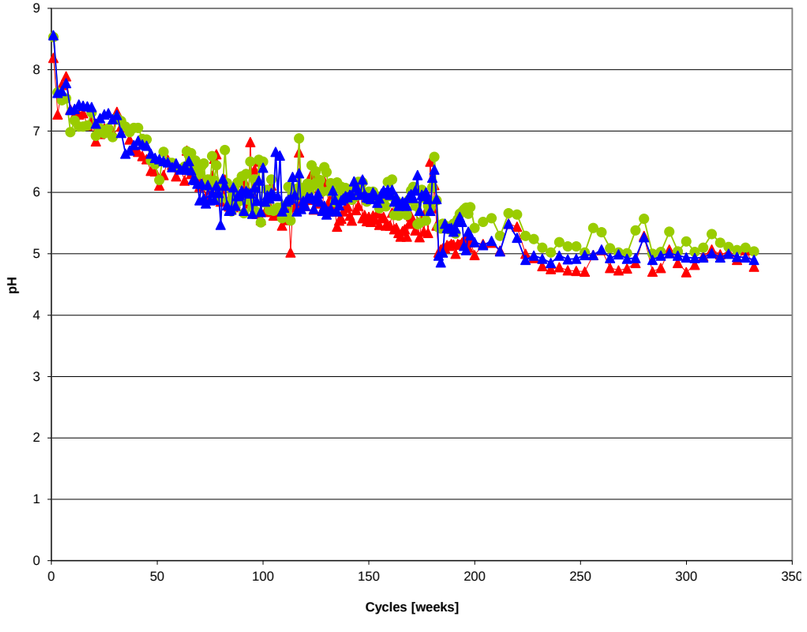

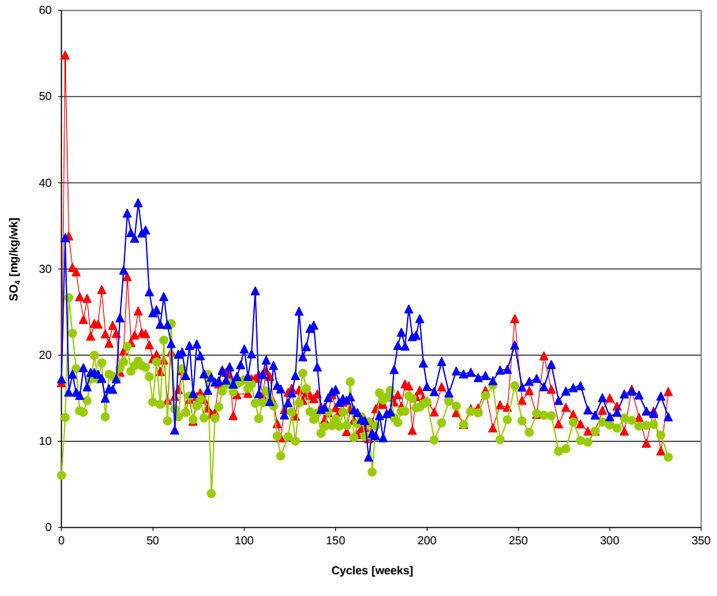

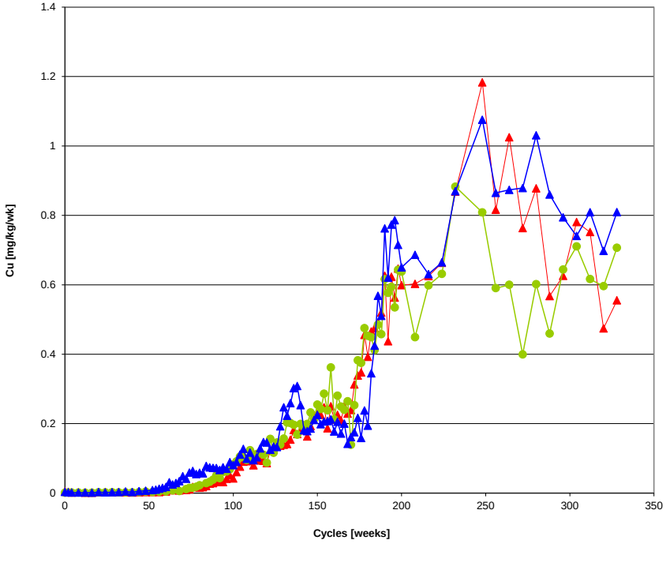

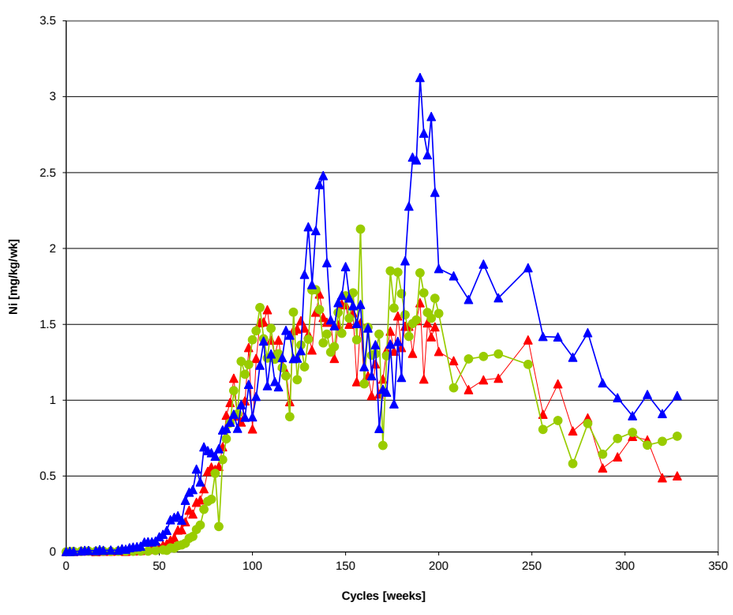

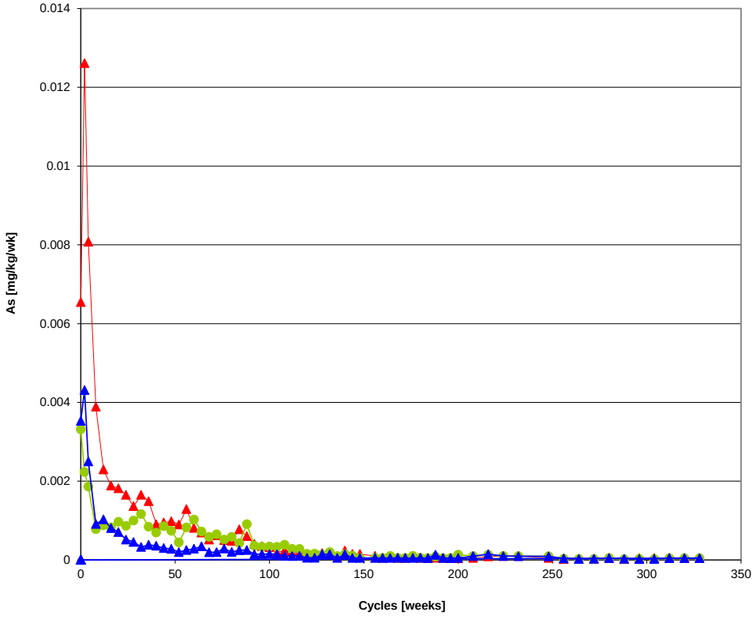

Figure 4. Leaching Trends for Ore Composite Samples

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Testing on Category 1 samples has confirmed to date that alkalinity generated by weathering of silicate minerals is able to result in pH above the pH of de-ionized water. This indicates that the alkalinity is consuming acidity introduced by both the dissolved CO_2, introduced by the deionized water (pH 5 to 6), and acidity produced by oxidation of sulfide minerals (locally pH<4). The steady maintenance of pHs above 6 shows that this is a steady long term mechanism for preventing pH depression below 6 in humidity cells.

3.2.2 Category 2/3

Results for pH, sulfate, nickel, copper and arsenic are shown in Figure 2.

Five samples (including three of the continuing 19 samples) showed pH depression below 6 while all other samples have shown stable pHs above 6 with similar trends to the Category 1 samples. Only one of the six samples has shown pH below 5. The sulfur content of the five samples varied from 0.2% to 0.42%. The sample with 0.2% sulfur showed the highest pH in this set at near 5.5. The sulfur contents of the other four samples were 0.3% and higher.

Sulfate release for Category 2/3 samples varied between 1 and 10 mg/kg/week with the highest rates being observed for samples containing higher sulfur contents. While sulfate release has varied, there was a lack of consistent upward trends.

A number of samples showed upwards trends in nickel and cobalt were related to pH decrease below 7. For the five samples showing distinctive pH depression, nickel release increased steeply during the first year of testing and subsequently showed gentle decline. Other samples showed slower nickel increase later in the program due to slower decline in pH. The same five samples also showed increase in copper leaching though the increase was less rapid than nickel and decline occurred later or not at all. One other sample (99-318C-325-330) showed accelerated copper leaching. Leachate pH was at the low end of those samples not showing substantial pH depression. It is a lean ore sample containing 0.1% copper.

Arsenic leaching trends were similar to Category 1. Rates have trended downwards. Higher rates are associated with samples yielding higher leachate pHs.

3.2.3 Category 4

Results for pH, sulfate, nickel, copper and arsenic are shown in Figure 3. Eight tests are continuing, two of which are Duluth Complex troctolite samples (00-371C(435-440) and 00-340C(765-780)), two are Duluth Complex Anorthositic Troctolites (00-331C (255-260) and 26027(616-626)), three are sedimentary hornfels samples (26058 (704-715), 26043&26027(740&1501-745&1506)) and DDH-26062&26026(993&565-998&568)-68) and one is Virginia Formation (00-361C(737-749)).

All tests have shown pH depression to some degree with pHs below 4. Of the continuing tests, one sedimentary hornfels sample (26058-704-715) showed the least depression with stable pHs near 5.3. The other samples showed leachate pHs near or below 4 and have recently shown pH recovery from minimum values. The two troctolite samples have shown pHs above 4.5 following a steep decline to pH near 4 earlier in the program. The other two Duluth Complex samples, Virginia Formation sample and other sedimentary hornfels samples have shown weaker pH recovery. Stabilization at pHs between 4 and 5 are indicative of buffering by reaction of acidity with alumino-silicate minerals resulting in formation of secondary aluminum minerals.

Some increases in sulfate release have been observed as pH decreased. Peak sulfate release rates have typically been less than a factor of five times lowest levels earlier in the test and peaks are not sharp but erratic. Decline in sulfate release has been observed following the peak. These declines are rapid at first then lessen as shown by straight lines on the log axis plot.

Due to the consistent pH depression below 7, all samples showed accelerated nickel leaching followed by declining leaching after reaching a subdued peak. All continuing tests have shown this decline parallels the sulfate release trend implying that nickel release is derived from sulfide minerals. Cobalt showed similar trends.

Upward trends in copper release were also apparent for all tests but as observed for acidifying Category 2/3 samples, the increase in copper release occurred later than nickel. Also, peaks followed by downward trends in copper release were less apparent for nickel and most ongoing tests have shown increase in copper release rather than a decrease paralleling nickel and sulfate release.

As with all other tests, arsenic release followed a declining trend.

3.2.4 Ore Composites

Results for pH, sulfate, nickel, copper and arsenic are shown in Figure 4.

Leachate pH trends for the three samples are nearly identical and show pH has been stable near 5 for over 2 years. Likewise, sulfate release has been similar for all three tests and has not shown consistent upward or downward trends. As pH decreased, nickel release accelerated, reaching peak rates after about three years then declining. Composite P30 showed a marked separate pH after 4 years that was not apparent in the other tests. A sulfate release peak was observed about the same time though was less pronounced compared to the other two tests.

Copper release also increased as the tests progressed but the upward trend was delayed compared to nickel and reached peak values after more than 4 years of testing. Copper release has subsequently declined.

3.3 Comparison Tests

Five samples tested as four size fractions continue to be tested in parallel with the humidity cell program. These tests are comparing differences in the weathering characteristics of the samples as well as differences in testing configuration (ASTM Humidity Cell and DNR Reactor) Samples are classified as Category 1 (two samples), Category 2/3 (two samples) and Category 4 (1 samples).

Both Category 1 samples have yielded leachate pHs near 7 in all size fractions and flat sulfate release trends. Sulfate release continues to be a function of particle size with lowest release (when detected) coming from the coarser size fractions. Similar findings apply to metal release. Nickel release is greater for the finer size fractions.

Consistent with the conventional humidity cells for Category 2/3 samples, lower pHs were observed but due to the sulfur content near 0.2% for both samples, pHs have remained above 6 and the samples continue to generate alkalinity particularly from the finer size fractions. Sulfate release was variable for both samples and no trends were apparent. Difference between sulfate releases in size fractions was most apparent for one sample (00-361C-310-320) with the fine fractions releasing higher sulfate than the coarse fraction. Both samples showed accelerated nickel leaching from all fractions as pH decreased below 7. Differences between size fractions were strongest for 00-361C-310-320. Temporal trends in copper release were not apparent but copper release rates were greater for the finer size fractions.

Virginia Formation rock classified as Category 4 showed sharp decline in pH to near and below 3.5 in all size fractions. Peak sulfate release was observed around 4 years for the two fine fractions. Peak sulfate release rates for the coarsest fractions have not observed because rates began increasing after six years. These differences in timing were not linked to greater pH depression for the fine fractions, in fact, the finer fractions yielded higher pHs.

These tests have shown that:

- Testing of different size fractions to date has shown that samples generating acidic leachate did so regardless of size fraction tested and protocol used.
- Rates of sulfate and metal release are generally higher for the finer size fractions presumably reflecting differences in surface area.
- For one sample that became strongly acidic, the finer fractions showed more rapid sulfate release earlier than the coarse fractions presumably reflecting faster oxidation and greater liberation of sulfide minerals.

3.4 Duplicate Testwork

Following observation of good reproducibility of trends previously (SRK 2009), all seven standard humidity cell tests being run as duplicates were stopped as part of reduction of the program in consultation with the DNR. Duplicate DNR reactor tests on two samples, and duplicate humidity cells containing size fractions for two samples have continued for a total of four duplicates (eight tests).

The two humidity cell-style duplicates are continuing to show strong reproducibility for the major ions and trace elements of interest (arsenic cobalt, copper, nickel). Weak reproducibility was apparent for barium (one sample).

The DNR-style reactors showed good reproducibility for one sample and poor reproducibility of pH for the other sample. The difference in pH (near 6 compared to 6.5) appeared to be due to differences in oxidation rate (faster for lower pH) which also resulted in greater leaching rates for cobalt and nickel.

3.5 Interpretation

3.5.1 Effect on Sulfur Management Criteria

Figure 5 shows minimum leachate pH as a function of total sulfur content for all humidity cell tests. Lowest pHs are observed for samples containing highest sulfur concentrations (Virginia Formation and Sedimentary Hornfels). The lowest sulfur concentration showing pH depression below 5 (i.e. below the range of pH of deionized water used in the test) was 0.36%. Samples with sulfur content below the Category 1 threshold of 0.12% have shown leachate pH above 6 indicating the threshold is appropriate based on current data.

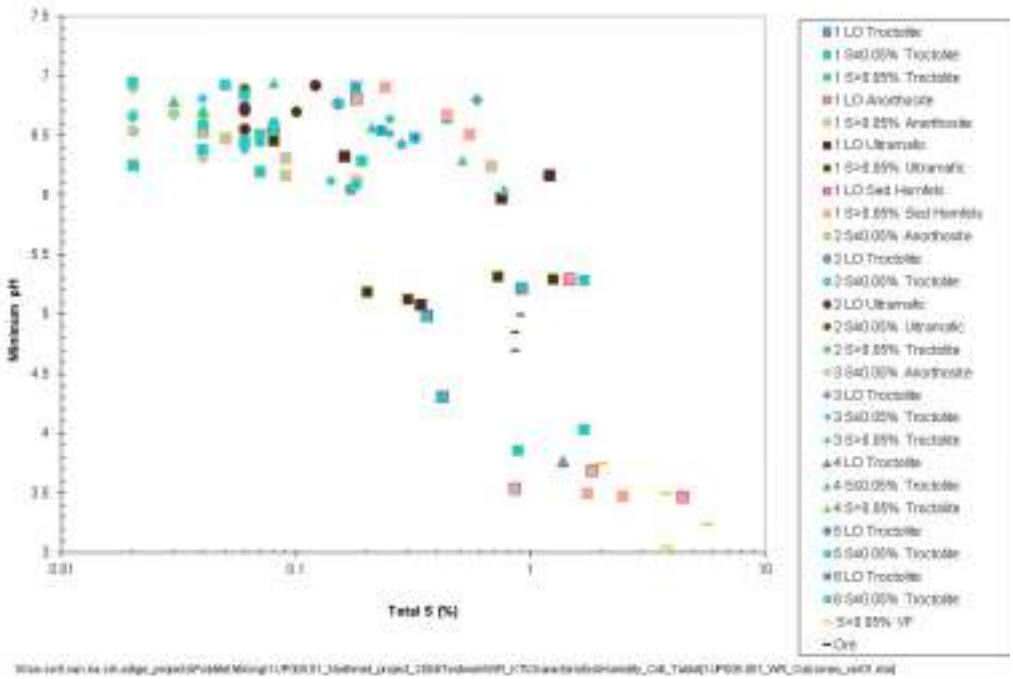

Figure 5. Minimum Leachate pH as a Function of Total Sulfur Content

3.5.2 Effect on Dissolution Rates Used to Estimate Water Quality

The condition definitions shown in Table 3 were used to assess trends for each test.

An example for one test showing all four conditions is provided in Figure 6. Condition 1 lasted for 39 weeks and was marked higher pHs and relatively stable SO_4 following an initial flush in the first few weeks. As pHs declined below 7, sulfate in this case become more variable and leaching of nickel accelerated. Condition 3 began at week 164 and was marked by steep decline in pH to 3.5 and rapidly increasing sulfate release paralleled by nickel. Condition 4 began at week 194 when sulfate release peaked and then began to declining.

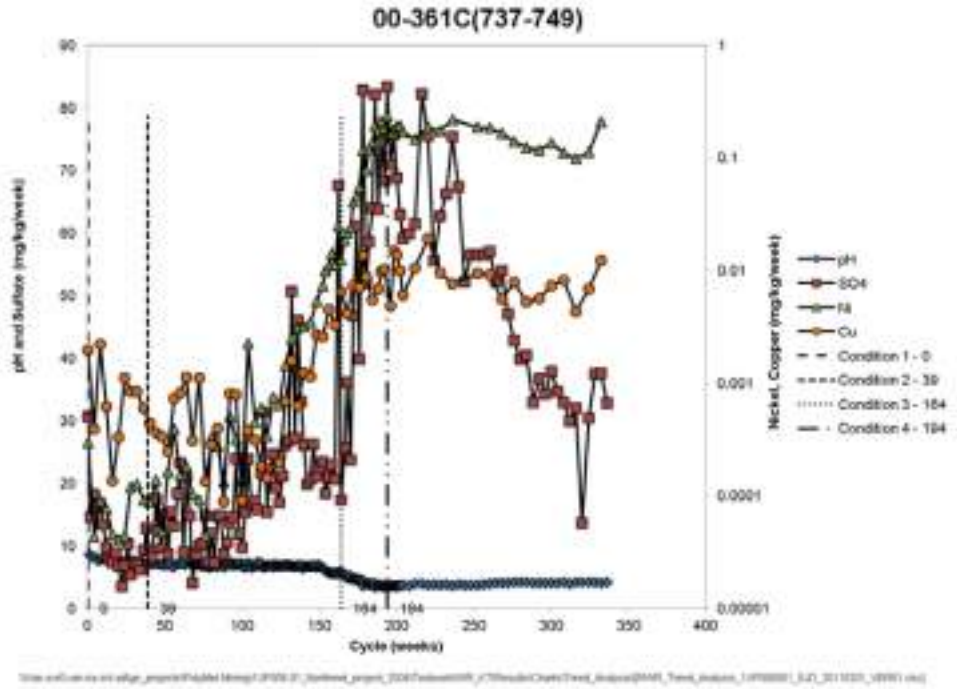

Figure 6. Example of Condition Assignment for a Waste Rock Humidity Cell Test

Resulting rates for each condition have been provided to Barr Engineering and include data collected to late January and early February 2012. These may be used as the basis for various inputs to source term predictions for the water quality model. However, the updated rate table primarily adds a few rates for samples that transitioned to acidic conditions. The majority of samples leaching under non-acidic conditions continue to show stable rates that have not changed significantly. As a result of this finding, there is likely little incremental value in modifying the rates used as inputs to the water quality modeling.

4 Tailings Kinetic Tests

4.1 Status of Program

A review of the tailings program was provided recently (SRK 2011). All tests have continued since findings were reported resulting in about six months of additional data. As this represents less than 15% additional time for most tests (with the exception of tests started in 2009), this memorandum provides a limited review to indicate whether any significant changes have occurred since the previous review.

4.2 Description of Results

4.2.1 NorthMet Project Tailings

Table 3 lists all tailings humidity cells. The test program consists of conventional humidity cells with parallel tests using the MDNR reactor configuration on bulk tailings and tailings size fractions.

Tests started in 2005 and 2006 (Pilot Plant 1) have continued to the same trends reported by SRK (2011) including stable or increasing pH, decreasing sulfate and stable or decreasing metal concentrations.

Tests started in 2008 (Pilot Plants 2 and 3) have reached stable pH with no indication of trends below pH 7 as shown in some samples produced by Pilot Plant 1. Sulfate release has stabilized for all tests. Nickel and cobalt release have not shown the same trends as Pilot Plant 1 samples but have shown stable concentrations in leachates below 0.002 mg/L and 0.0001 mg/L, respectively. PP2 +100 mesh showed an arsenic spike to 0.08 mg/L at week 168 following repair of the cell to address rapid drainage of water during the leach cycle.

Tests started in 2009 (scavenger tailings samples) have shown downwards trends in pH with lowest pHs typically above 7. At the same time, sulfate release has also trended downward and for some tests has stabilized. Nickel and cobalt release trended downward initially for all tests. Upwards trends for both elements were most apparent for samples Oct 1/09 09:00 (-100+200 mesh) which also showed some pHs below 7.

An arsenic spike resembling those for the 2008 samples discussed above was also apparent for two cells. These spikes were correlated with a large number of spikes in other parameters including aluminum, iron and silicon. It appears the cell malfunctioned perhaps allowing solids to be present in the leachates and thereby causing an increase in the dissolved fraction due to the presence of colloidal matter.

Table 4: Tailings Humidity Cells Used as Basis for Update Report

HCT ID	Fraction	HCT Full ID	Total Sulfur (%)	Start Date	Total Duration weeks	Duration Used for Current Modeling[3] weeks
T1	Whole	P1 (CuSO4)	0.1	9/8/2005	333	--
T2	Whole	P1 (no CuSO4)	0.23	9/8/2005	333	--
T3	Whole	P2 (no CuSO4)	0.2	9/8/2005	333	--
T4	Whole	P3 (CuSO4)	0.15	9/8/2005	333	--
T5	100	Parcel 2 P2S +100 mesh	0.15	2/10/2006	311	--
T6	100	Parcel 2 P2S -100 +200 mesh	0.17	2/10/2006	311	--
T7	-200	Parcel 2 P2S -200 mesh	0.24	2/10/2006	311	--
T8	100	Parcel 1-2 PISCS +100 mesh	0.11	2/10/2006	311	271
T9	100	Parcel 1-2 PISCS -100 +200 mesh	0.1	2/10/2006	311	271
T10	-200	Parcel 1-2 PISCS -200 mesh	0.09	2/10/2006	311	271
T11	100	Parcel 3 P3S +100 mesh	0.11	2/10/2006	311	271
T12	100	Parcel 3 P3S -100 +200 mesh	0.14	2/10/2006	311	271
T13	-200	Parcel 3 P3S -200 mesh	0.14	2/10/2006	311	271
T52	Whole	Pilot Plant 2 (0.30 Cu feed) Composite Bulk Tailings (as rec'd)	0.07	7/8/2008	185	--
T53	100	Pilot Plant 2 (0.30 Cu feed) Composite (+100 mesh)	0.08	7/8/2008	185	146
T54	100	Pilot Plant 2 (0.30 Cu feed) Composite (-100 + 200 mesh)	0.06	7/8/2008	185	146
T55	-200	Pilot Plant 2 (0.30 Cu feed) Composite (-200 mesh)	0.09	7/8/2008	185	146
T56	Whole	Pilot Plant 3 (0.25 Cu feed) Composite Bulk Tailings (as rec'd)	0.08	7/8/2008	185	--
T57	100	Pilot Plant 3 (0.25 Cu feed) Composite (+100 mesh)	0.1	7/8/2008	185	146
T58	100	Pilot Plant 3 (0.25 Cu feed) Composite (-100 + 200 mesh)	0.08	7/8/2008	185	146
T59	-200	Pilot Plant 3 (0.25 Cu feed) Composite (-200 mesh)	0.08	7/8/2008	185	146
T60	Whole	SCAV TAILS 9/30/09 1600	0.09	11/24/2009	113	--
T61	100	SCAV TAILS 9/30/09 1600 +100	0.1	11/24/2009	113	74
T62	100	SCAV TAILS 9/30/09 1600 -100+200	0.09	11/24/2009	113	74
T63	-200	SCAV TAILS 9/30/09 1600 -200	0.11	11/24/2009	113	74
T64	Whole	SCAV TAILS 10/1/09 09.00	0.13	11/24/2009	113	--
T65	100	SCAV TAILS 10/1/09 09.00 +100	0.11	11/24/2009	113	74
T66	100	SCAV TAILS 10/1/09 09.00 -100+200	0.14	11/24/2009	113	74
T67	-200	SCAV TAILS 10/1/09 09.00 -200	0.14	11/24/2009	113	74
T68	Whole	SCAV TAILS 10/1/09 17.00	0.12	11/24/2009	113	--
T69	100	SCAV TAILS 10/1/09 17.00 +100	0.1	11/24/2009	113	74
T70	100	SCAV TAILS 10/1/09 17.00 -100+200	0.1	11/24/2009	113	74
T71	-200	SCAV TAILS 10/1/09 17.00 -200	0.13	11/24/2009	113	74

[3] Waste Characterization Data Package Version 9 (July 3 2012)

4.2.2 LTVSMC Tailings

Four samples of LTVSMC tailings are being tested in humidity cells. No significant changes in leachate chemistry have been observed since the previous update (SRK 2011). Leachate pHs have continued to vary in a narrow band between 7.4 and 8 with variable but non-trending alkalinity and sulfate.

Other parameters have continued to show no upward trend or downward trends. Isolated data spikes (for example, arsenic) are observed but do not represent significant trends.

4.3 Interpretation

Ongoing tailings testwork has shown no indication of development of pHs below that of the deionized water used to leach the samples (Figure 7) indicating that weathering of silicates minerals in the samples is continuing to generate alkalinity to offset acidity from sulfide oxidation and the deionized water even in the tests that have been operating for 7 years. The longer term tests yielded pH minima in their trends which are responsible for the lower pHs shown in Figure 7. Upward trends in pH are observed in most cases so that recent pHs are higher than the lowest values.

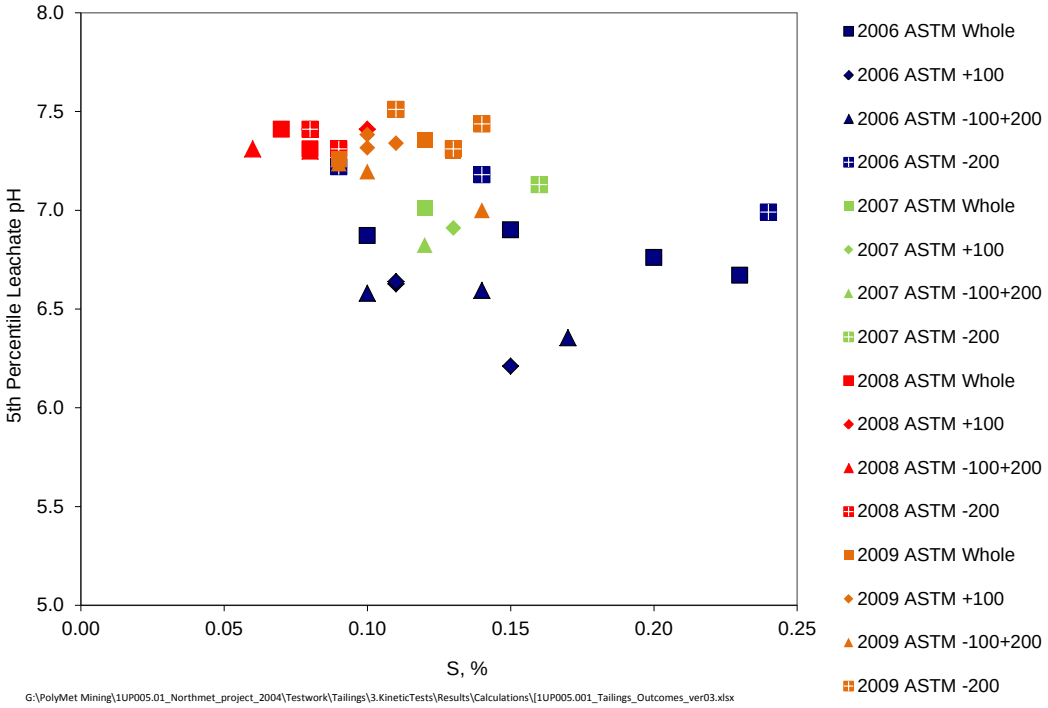

Figure 7. 5th Percentile pH as a Function of Total Sulfur Content for Tailings Humidity Cells

Correlations between initial total sulfur content and average release rates are apparent as have been observed previously (Figure 8). However, this figure suggests different relationships for the 2006 and 2007 datasets compared to the 2008 and 2009 datasets apparently implying that the earlier samples are less reactive. However, the differences are probably caused by significant depletion of sulfide as the tests have progressed with a correlated decrease in release rates, reducing the calculated average release rates. For example, between 30% and 78% of the initial sulfur has been depleted from the earlier tests compared to less than 30% for the more recent tests. For the oldest tests, remaining sulfur content varies from 0.04% to 0.12%. The general downward trend in sulfate release has resulted in progressively lower average sulfate release. The downward trend in sulfate release (and therefore acid generation rate) also provides a reasonable explanation for upward trends in pH.

The implication of results from the older tests is that the potential for long term pH depression below 5 is shown to be very low because silicate weathering was able to offset acid generation when sulfur content was higher than the current depleted levels.

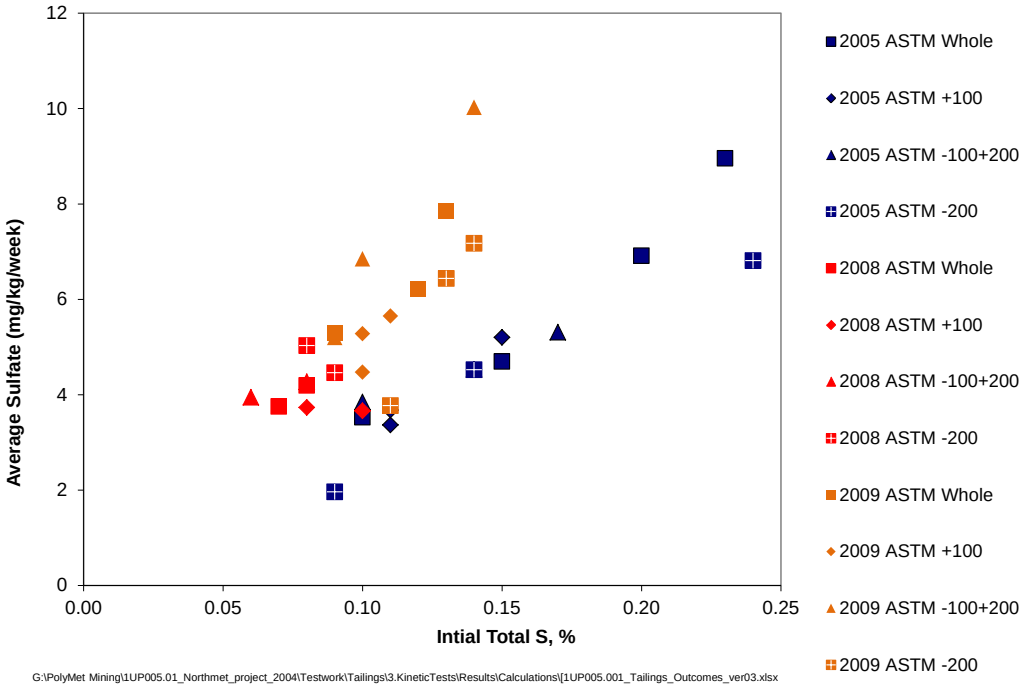

Figure 8. Average Sulfate Release as a Function of Initial Total Sulfur Content

The long term downward trend in sulfate release rates (Figure 9) indicates that average sulfate release rates for individual tests as used as input into the tailings water quality models are a function of time and will decrease as the test duration increases. To address this finding and provide sulfate release rates that are consistent with the assumed zero order reaction rate law for sulfide oxidation in the water quality model, relationships between sulfate release and sulfur depletion were examined to estimate sulfide oxidation rates at the initial sulfur content. The concept is to obtain the initial rates at t=0 by regressing depleted sulfur content against sulfate release rates to obtain equations of the form:

$$\frac{dM_{SO_4}}{dt} = a_1 M_S + a_0$$

Graphs of sulfate release rate as a function of sulfur remaining showed three different relationships. The most common relationship (15 tests) was for the slope of the equation to be steeper at the start than in the longer term. Other trends were (1) initially lower rates followed by a peak then long term decay (about 6 tests); and (2) consistent slope for the duration of the test (12 tests).

The first type of relationship is commonly observed in humidity cell results and is interpreted as initial flushing of stored sulfate oxidation products followed by sulfide oxidation. The initial part of the trend therefore does not represent sulfide oxidation and should not be included in the regression equations to obtain the oxidation rate at t=0. The preferred approach is to use the data for each test on its own merit to evaluate the break point when leaching of stored oxidation products is complete and the long term decay trend is well-established. LAM MDNR disagreed with this approach and required that the regression equations be developed excluding the first 5 weeks of data based on the following commentary in the ASTM humidity cell method D5744-96, (Section 11.2):

> "In the testing of mining wastes, cation and anion loadings are commonly high in the Week 0 leachate due to the dissolution of pre-existing soluble oxidation salts present in the sample prior to sample collection. The average number of weekly accelerated weathering cycles required to flush these pre-existing salts ranges from 3 to 5 weeks. Oxidation products observed during these 3 to 5 weeks are principally from pre-existing salts, while those

products observed after this period are considered to be solely a function of the accelerated weathering procedure."

SRK does not agree with the use of a fixed initial flushing period because it is unlikely that this flushing process can be generalized. In reality, the period required to fully flush stored oxidation products depends on the types, solubility and quantity of the oxidation products, and the physical and mineralogical characteristics of the samples. Tailings samples also contain entrained process water which is expected to be soaked into the finer particles leading to a protracted initial flushing trend. Nonetheless, to advance the process of acceptance, the LAM MDNR requested method was adopted to calculate oxidation rates at t=0.

Graphs for all tests showing regression relationships using the MDNR method are provided in Attachment A

Figure 9 shows initial sulfate release (determined from the regression equation using the initial sulfide content of the sample) as a function of initial total S content of the samples for comparison with Figure 8. Initial sulfate release calculated using the MDNR and SRK methods is shown. The difference in rates between tailings samples generated by different pilot plants implied by Figure 8 is reduced by both methods but the DNR method appears to result in lower rates for the 2005 samples and a weaker correlation with sulfur content than the SRK method which eliminates differences between the different datasets. The similarity of rates implies that the tailings generated at different times have similar reactivity. Correlation between initial sulfur content and initial sulfate release is apparent when considering the entire data range though weaker at lower initial sulfur contents.

For the purpose of water quality modeling, the distribution of sulfate release rates will be based on the initial sulfate rates (Figure 9) calculated using the MDNR method as agreed This method will result in the use of sulfide oxidation rates in the model that are nearly always higher than would actually be expected. Application of a single initial rate will result in predicted sulfate and trace element concentrations that are likely to be much higher than would really occur as sulfide content depletes over time. The approach is therefore conservative for predicting water quality.

For the purpose of modeling other parameters the following recommendations are made for rates:

- For parameters based on a solid ratio to sulfur content, rates should be calculated based on the ratios previously specified.
- For parameters based on average rate to average sulfate rate ratios (Ca, K, Mg, Na, Se), rates should be calculated based on the average rate ratio as previously specified. For the major ions, downward trends are apparent that parallel decrease in sulfate indicating that ratios of average rates are appropriate.

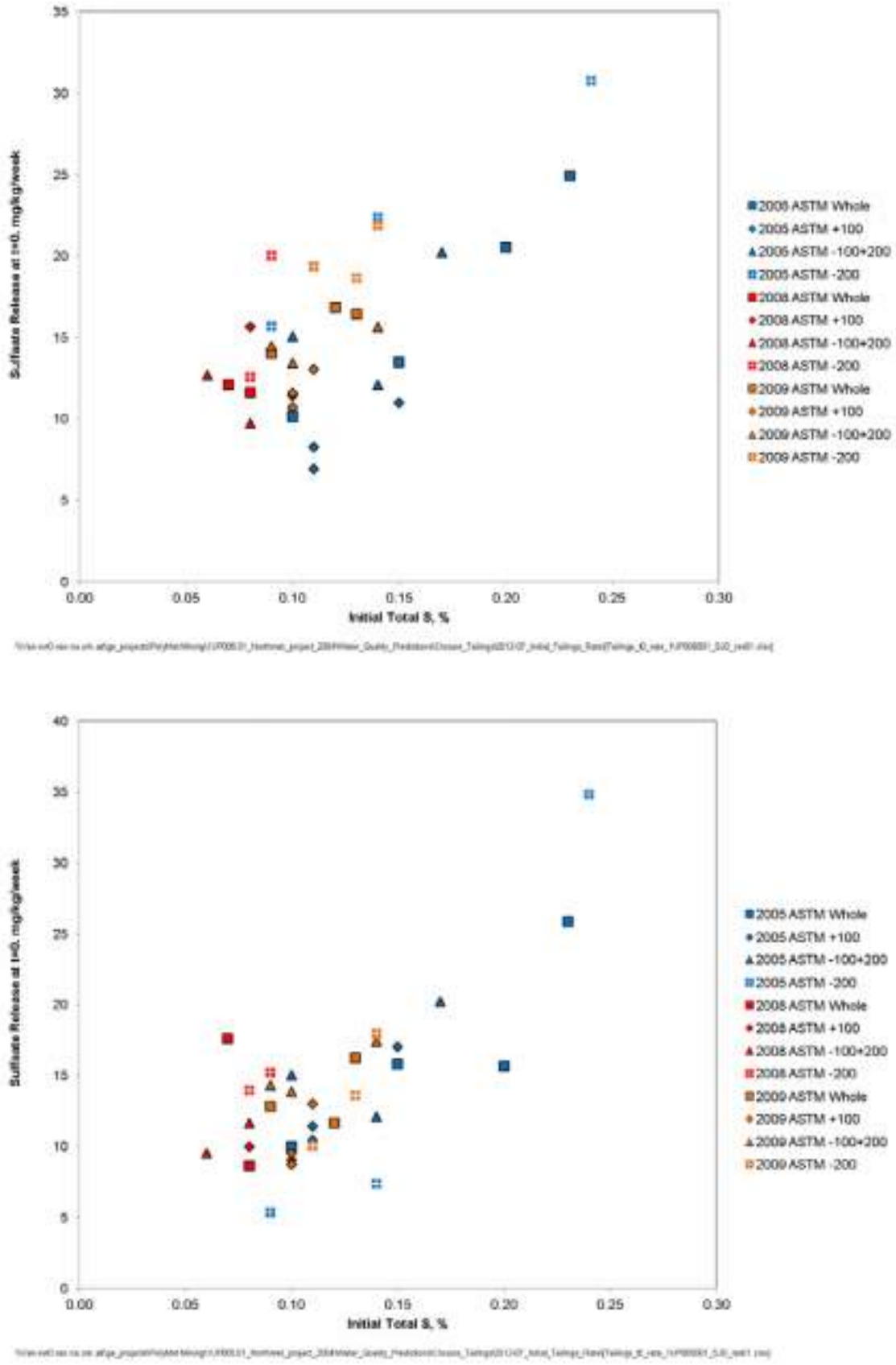

Figure 9. Calculated Initial Sulfate Release as a Function of Initial Total Sulfur Content Using MDNR (top) and SRK (bottom) methods.

5 Hydrometallurgical Residue Kinetic Tests

As agreed with MNDR, hydrometallurgical kinetic tests were stopped shortly after reported by SRK (2009).

6 Conclusions

Review of data generated by up to nearly eight years of testing indicates that:

- Waste rock classification into three categories using sulfur content to address potential for pH acidification appears to be robust. In particular, samples classified as Category 1 have not shown pH depression below that of the deionized water used to leach the samples.
- Sulfide oxidation and metal leaching trends for waste rock samples show well defined phases as pH depression occurs. These trends have been used to calculate release rates for use in development of source terms. Ongoing testwork has resulted in some additional rate information for later phases.
- Tailings humidity cells have not shown pH depression below that of the deionized water indicating that weathering of silicates provides alkalinity to offset acidity from sulfide oxidation.
- Long term trends in tailings humidity cells show declining oxidation rates which are presumably linked to depletion of the sulfide minerals and indicates that sulfide oxidation is following a non-zero order reaction rate law. However, to be consistent with the simplified modeling approach used in GoldSim®, which assumes a zero order reaction rate law, trends in sulfide oxidation rates have been used to back-calculate reaction rates at the initial sulfur content for use in modeling.
- Other than addressing this recommendation, updating the rates used as inputs to tailings water quality modeling is not recommended.

7 References

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SRK Consulting. 2011. Update on Tailings Humidity Cell Test Data – NorthMet Project – Draft. Memorandum to Peter Hinck, Barr Engineering. December 23, 2011.

SRK Consulting (Canada) Inc.

Stephen Day, PGeo
Corporate Consultant (Geochemistry)

Disclaimer

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The opinions expressed in this report have been based on the information available to SRK at the time of preparation. SRK has exercised all due care in reviewing information supplied by others for use on this project. Whilst SRK has compared key supplied data with expected values, the accuracy of the results and conclusions from the review are entirely reliant on the accuracy and completeness of the supplied data. SRK does not accept responsibility for any errors or omissions in the supplied information, except to the extent that SRK was hired to verify the data.

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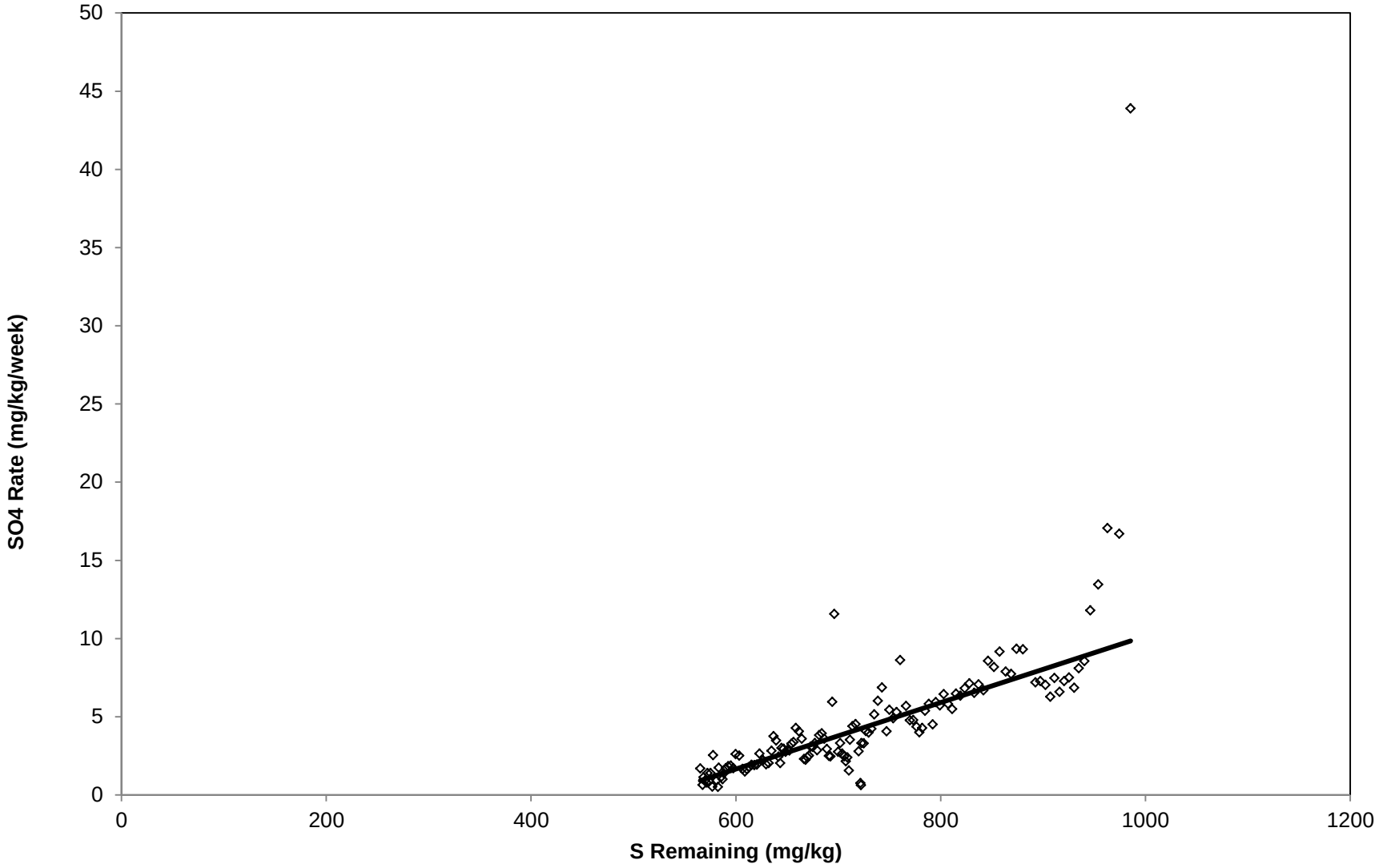
Attachment A: Graphs

P1S



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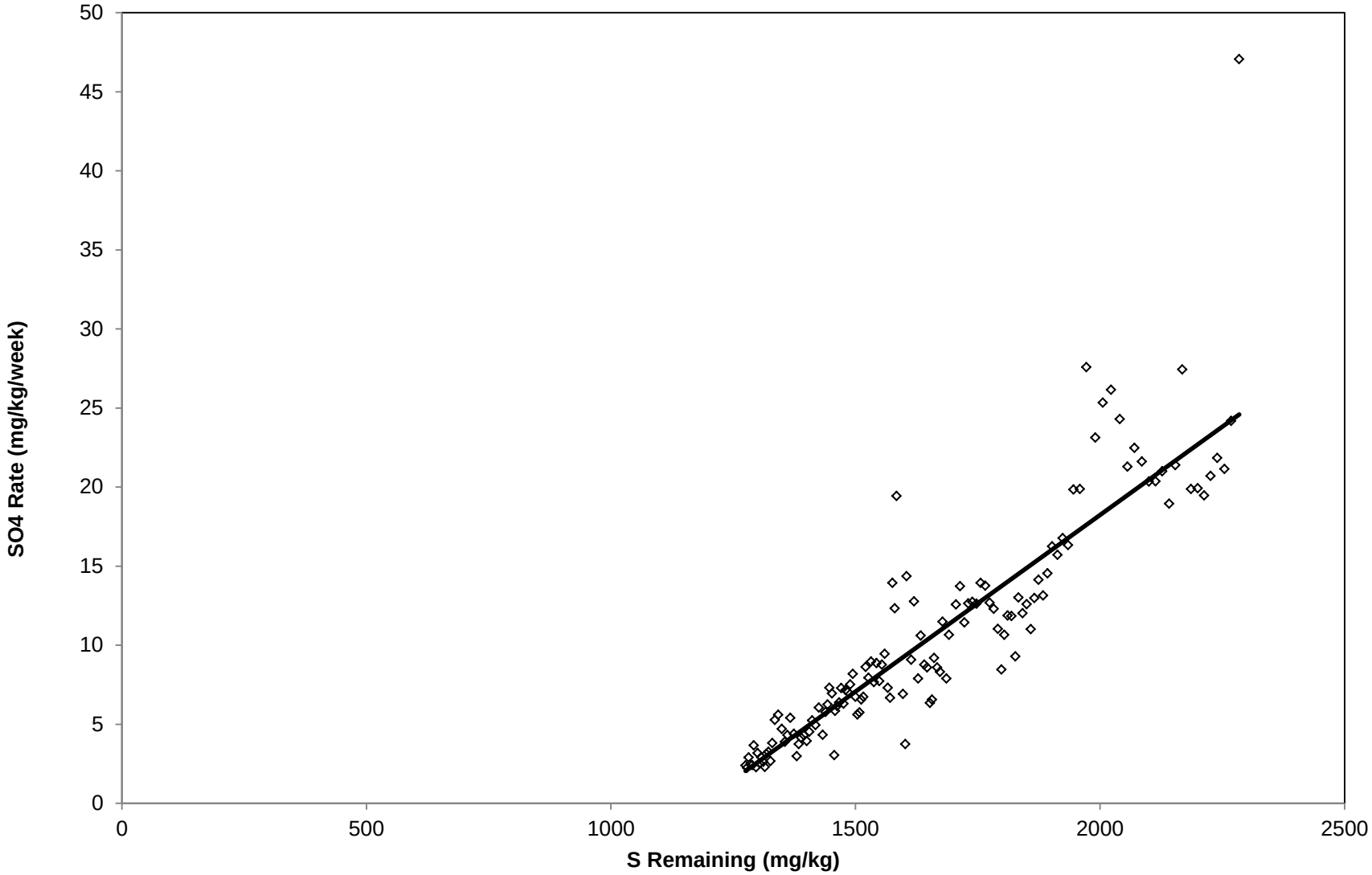

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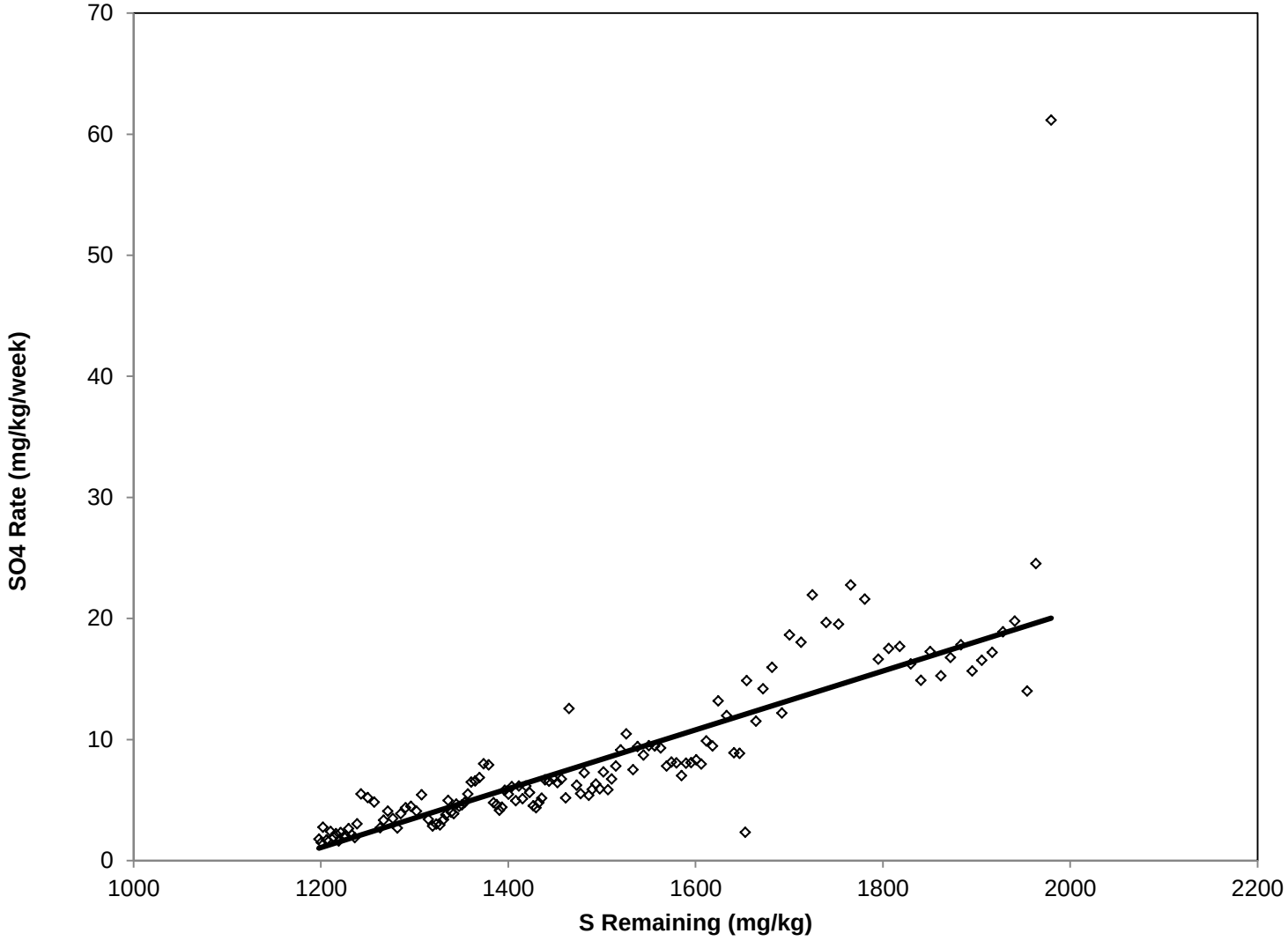

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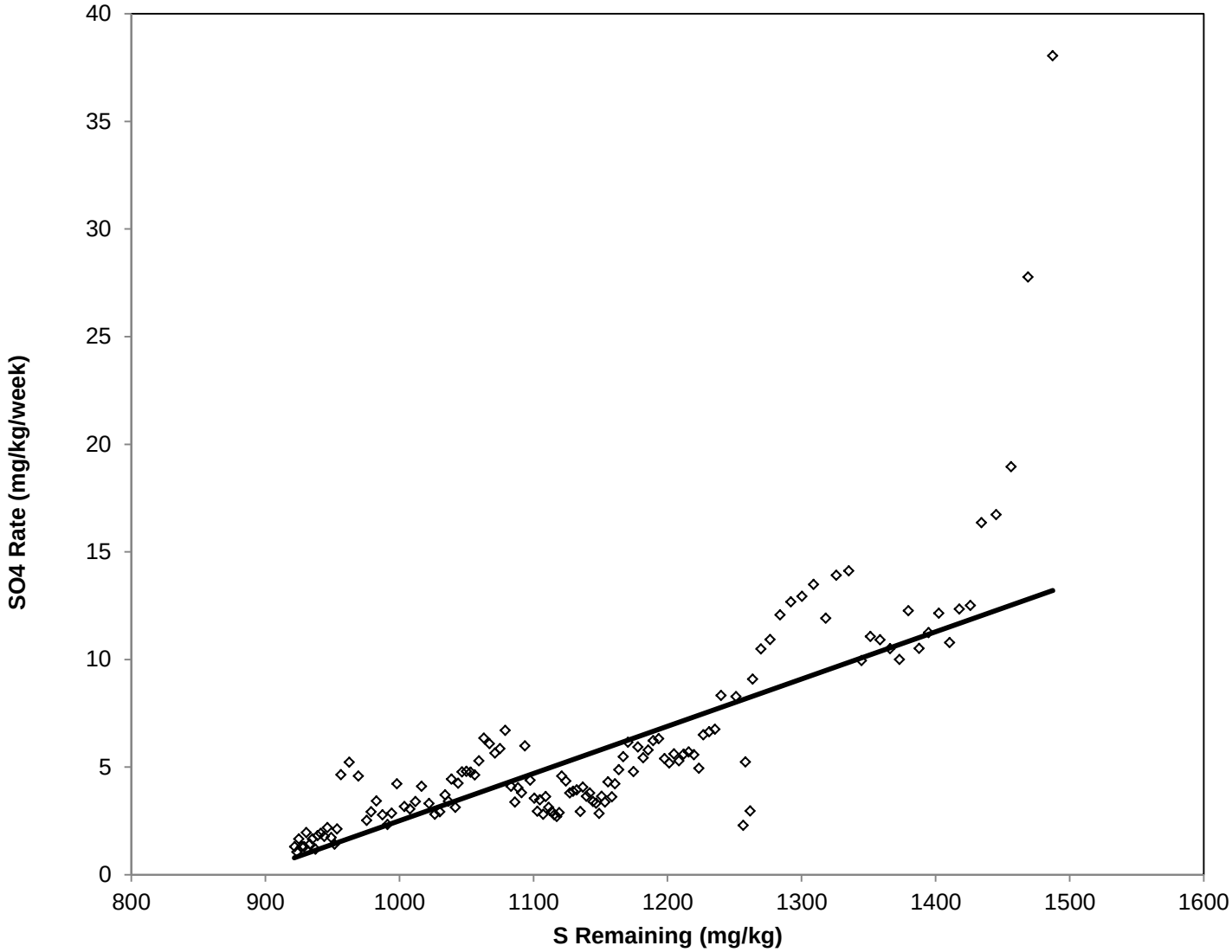

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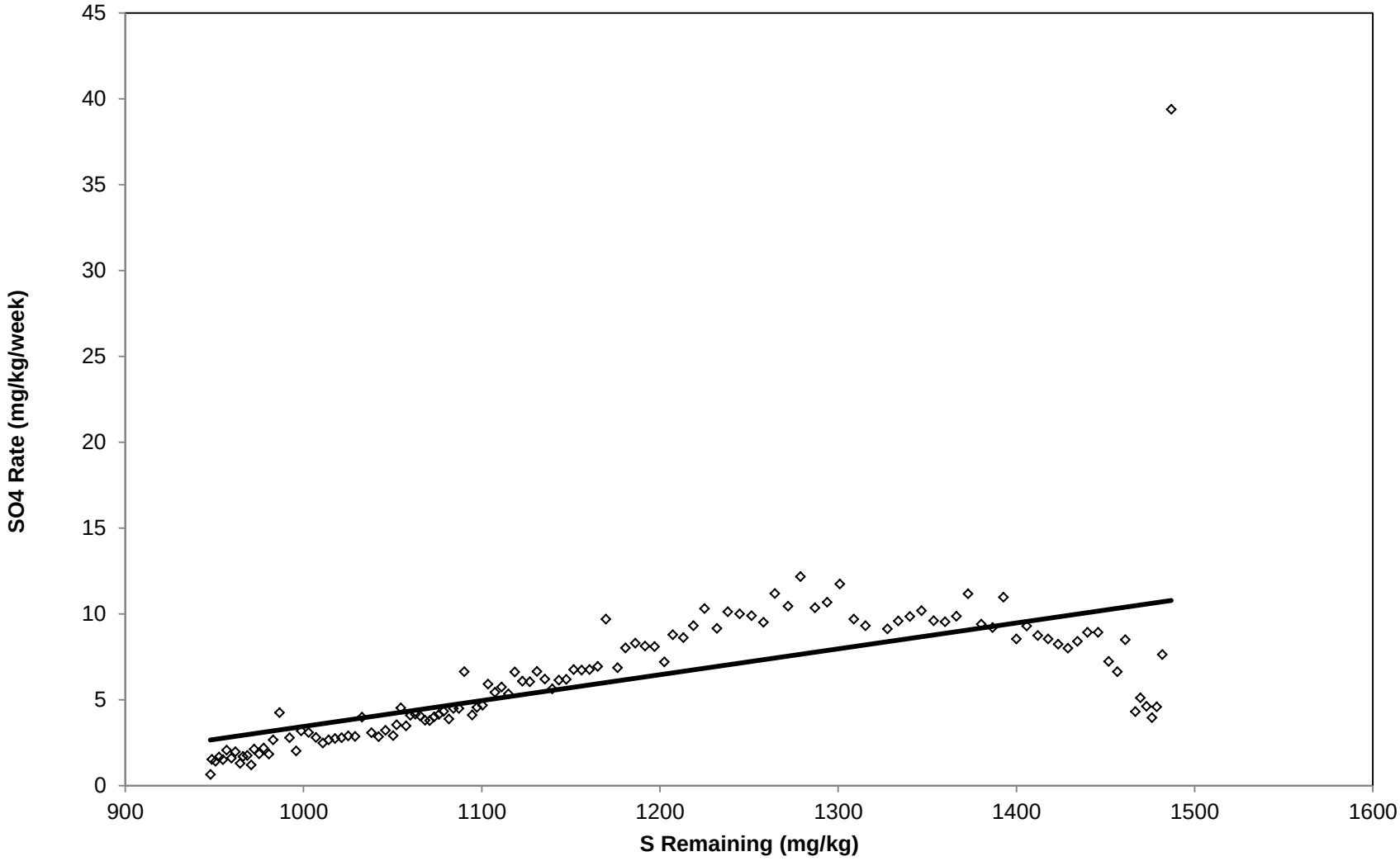

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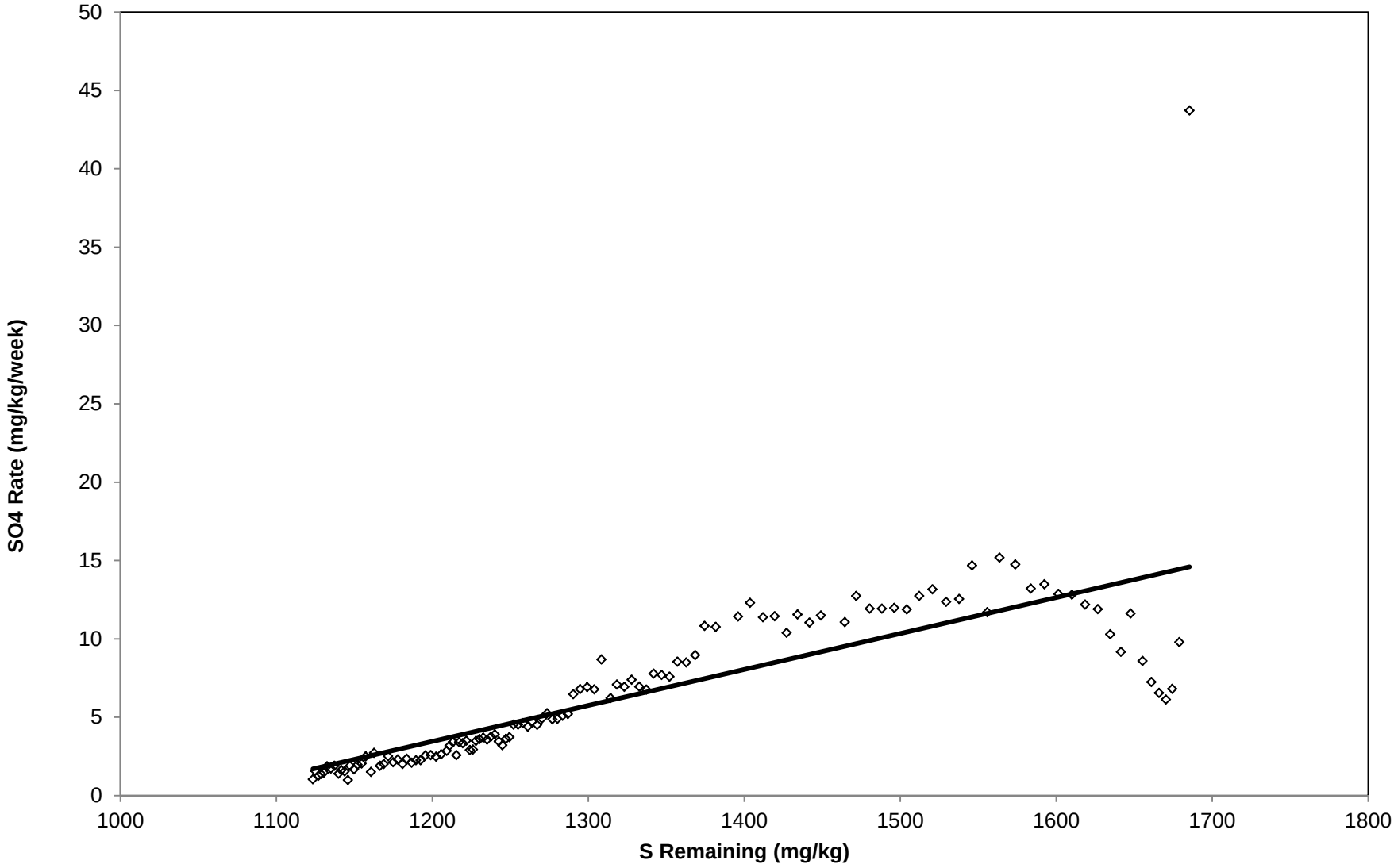

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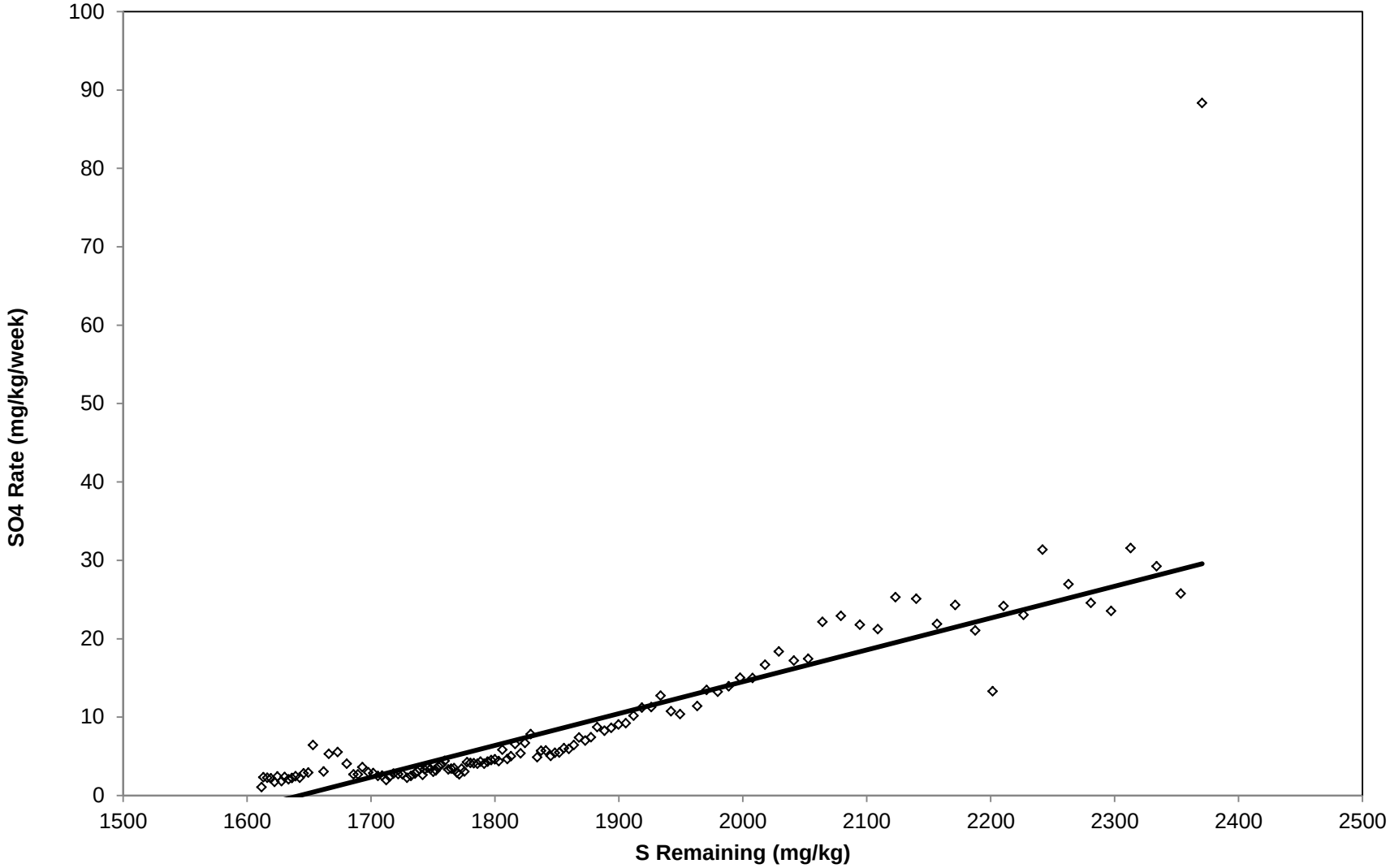

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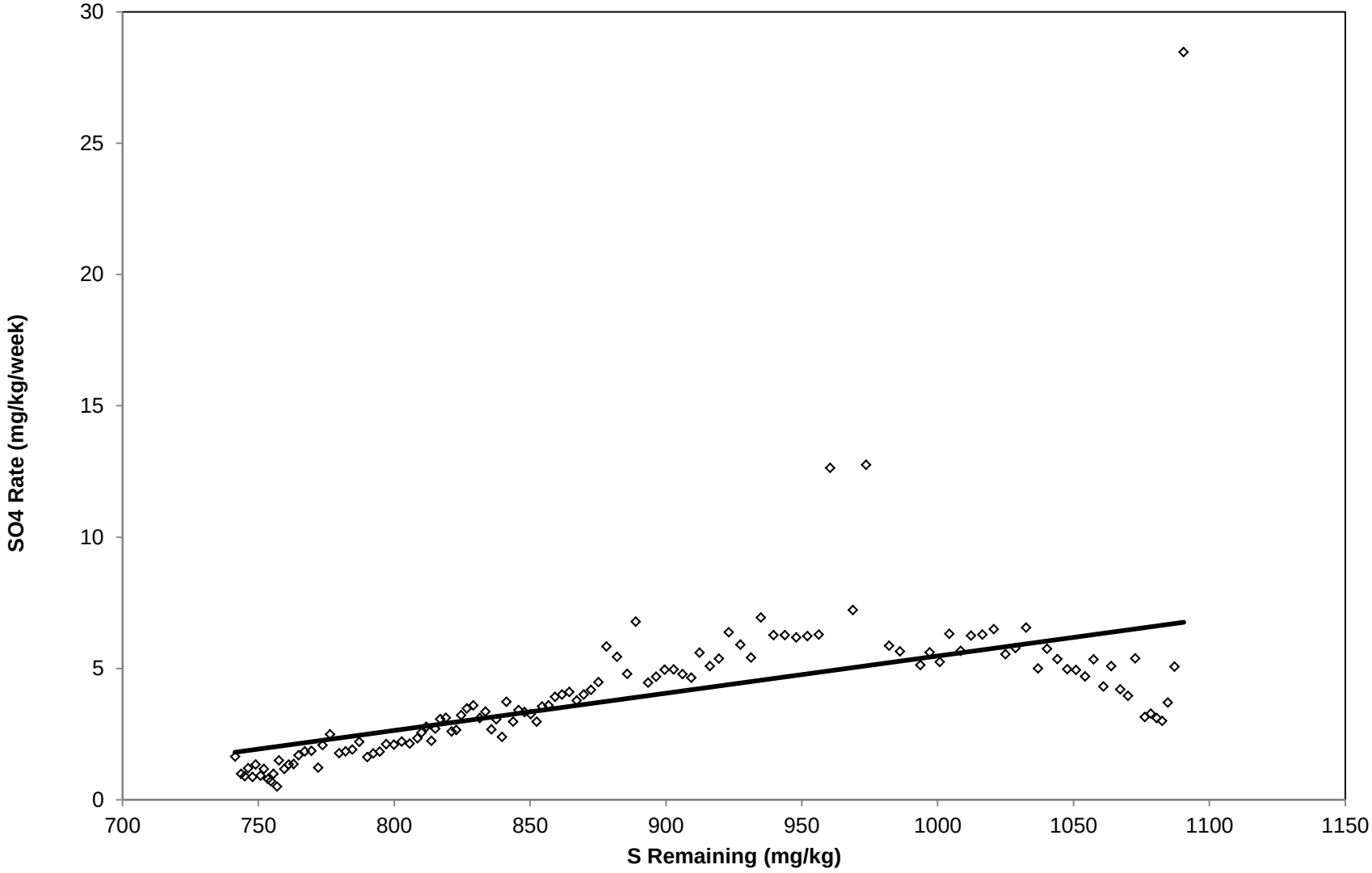

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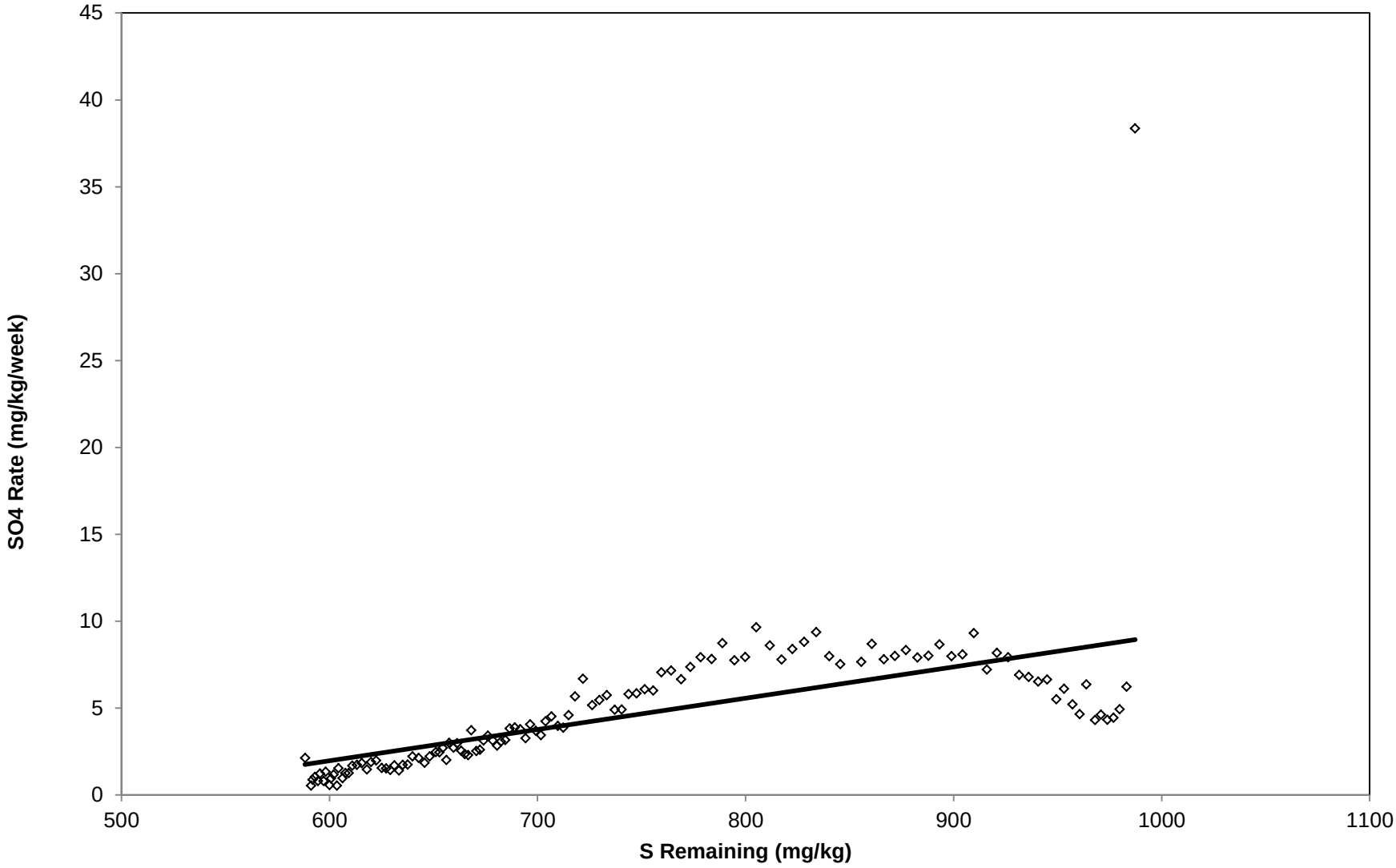

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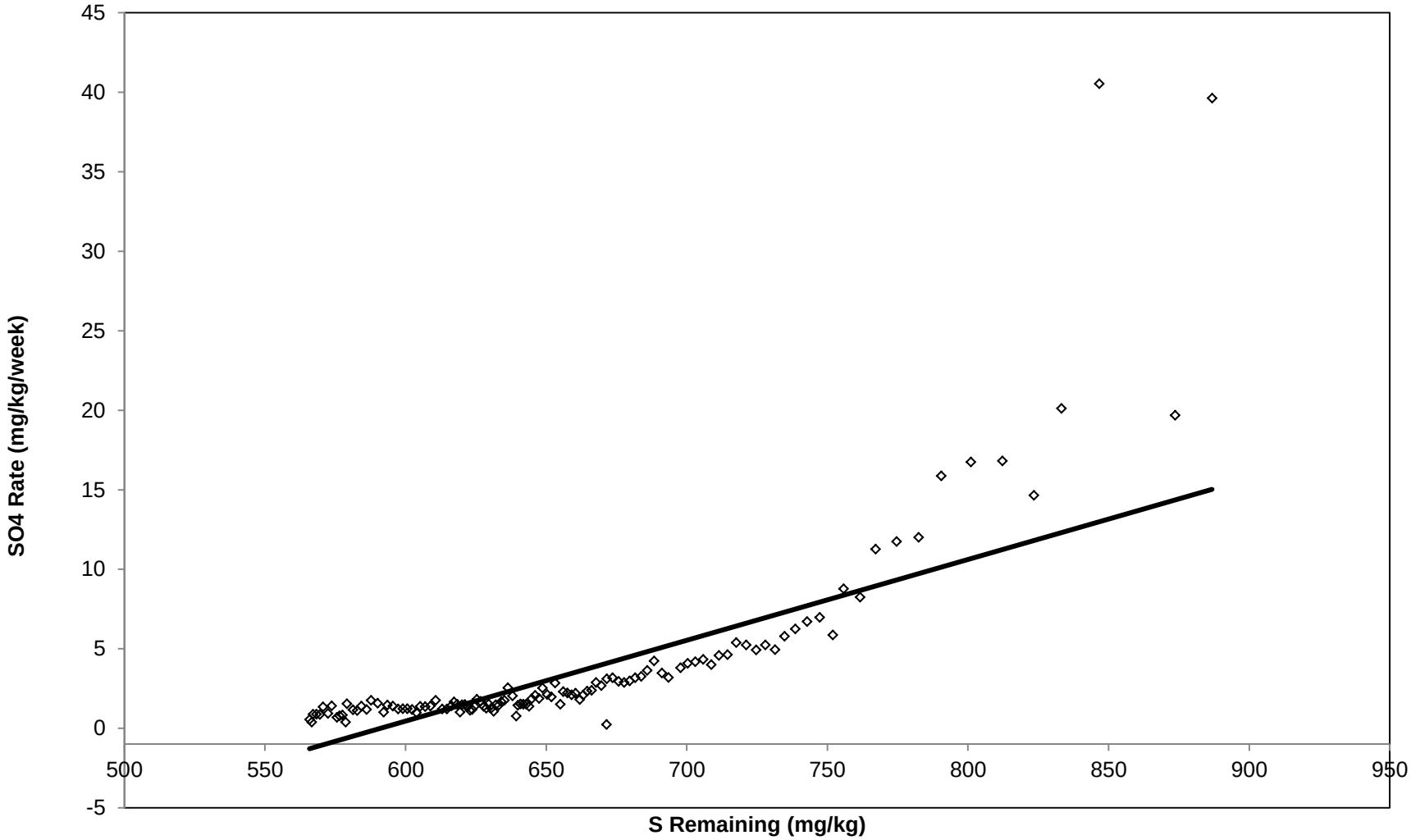

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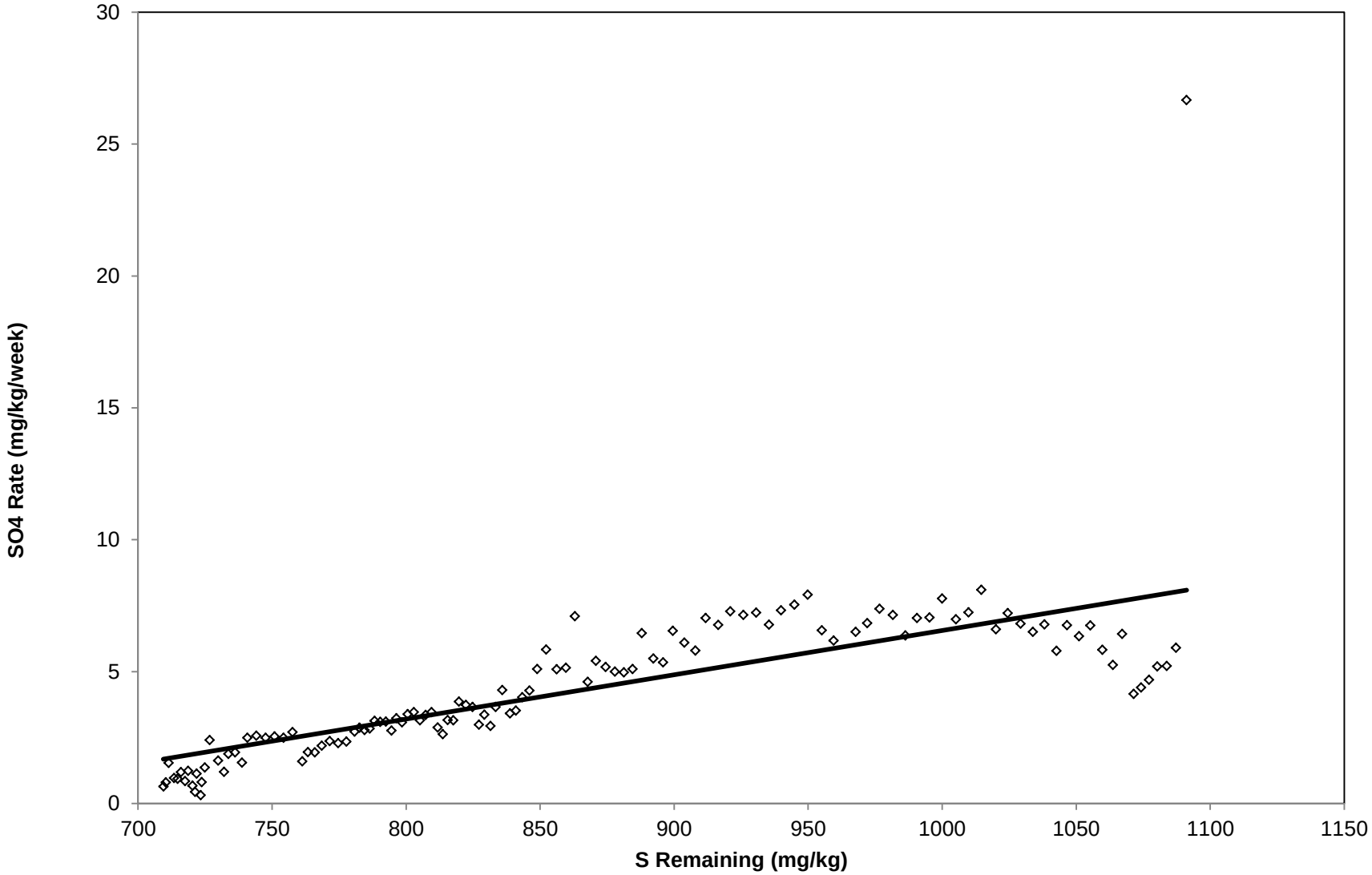

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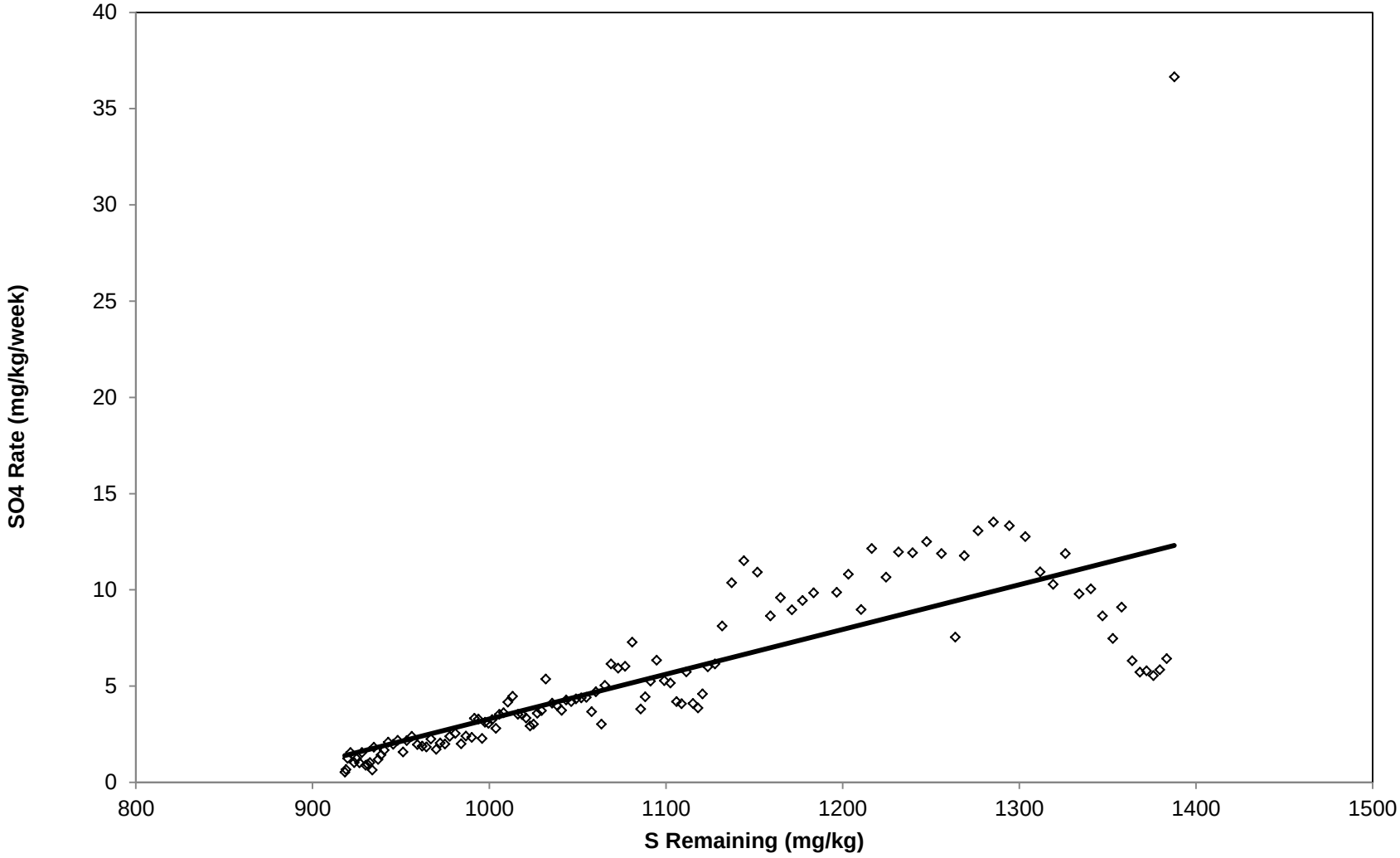

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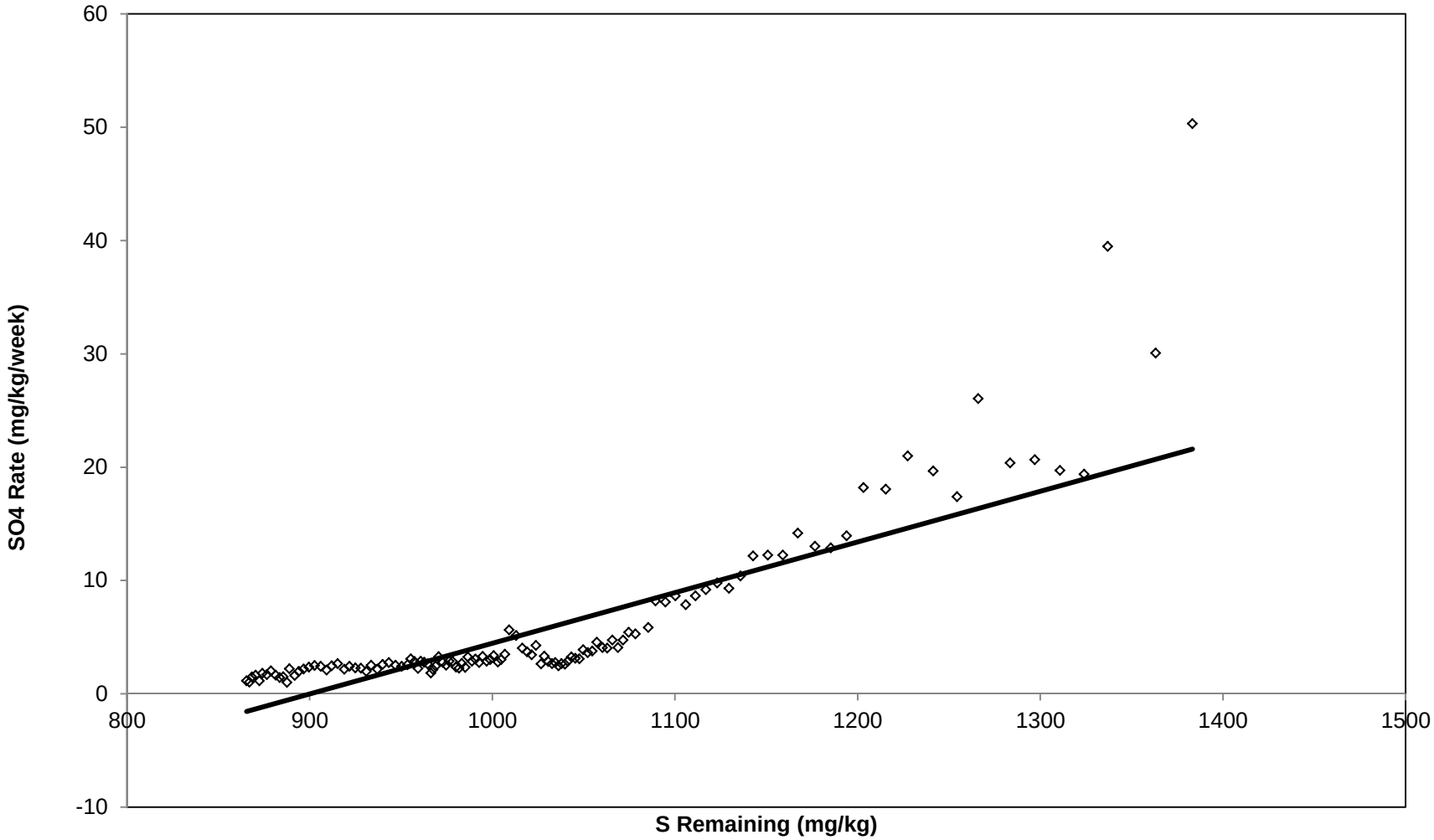

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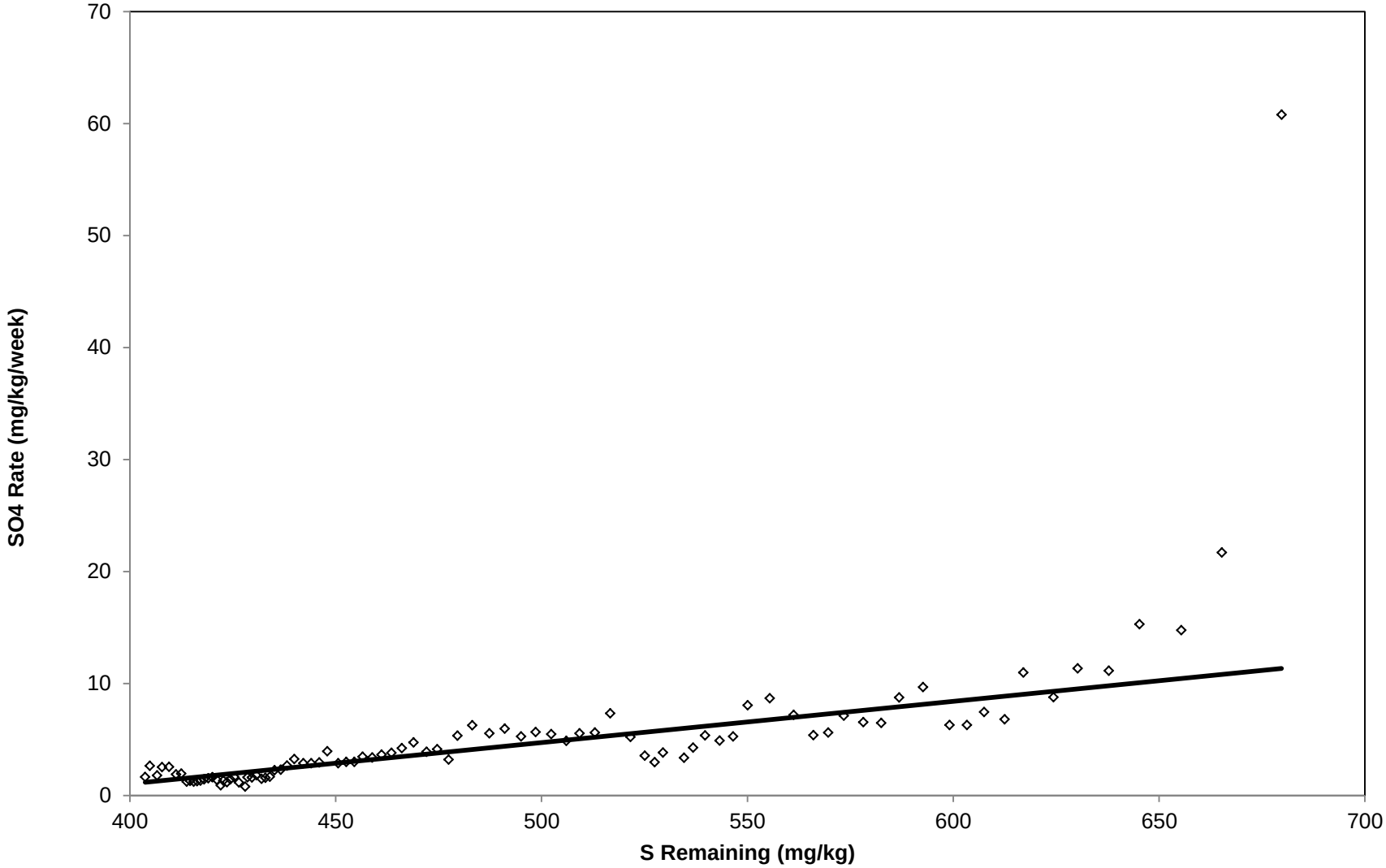

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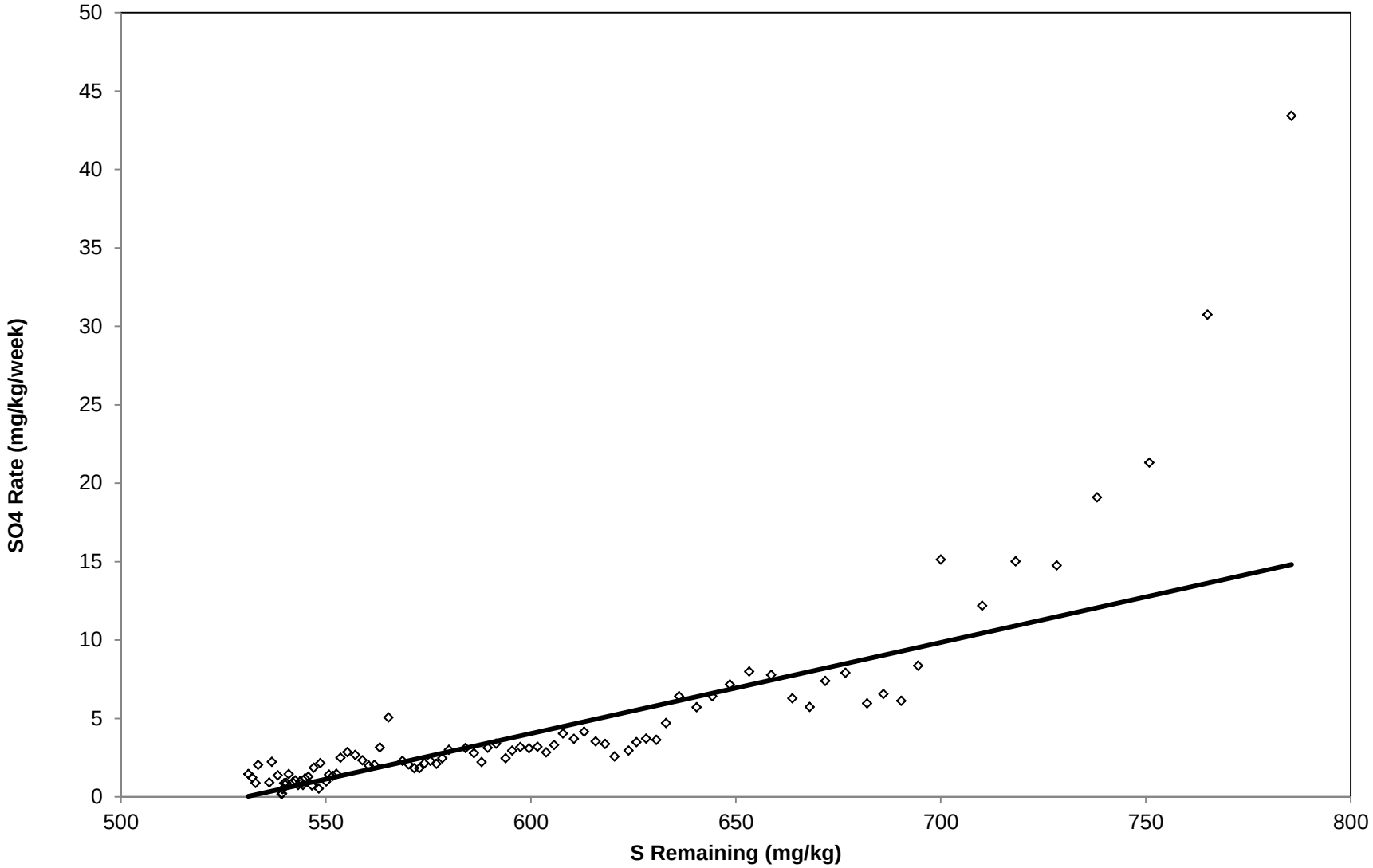

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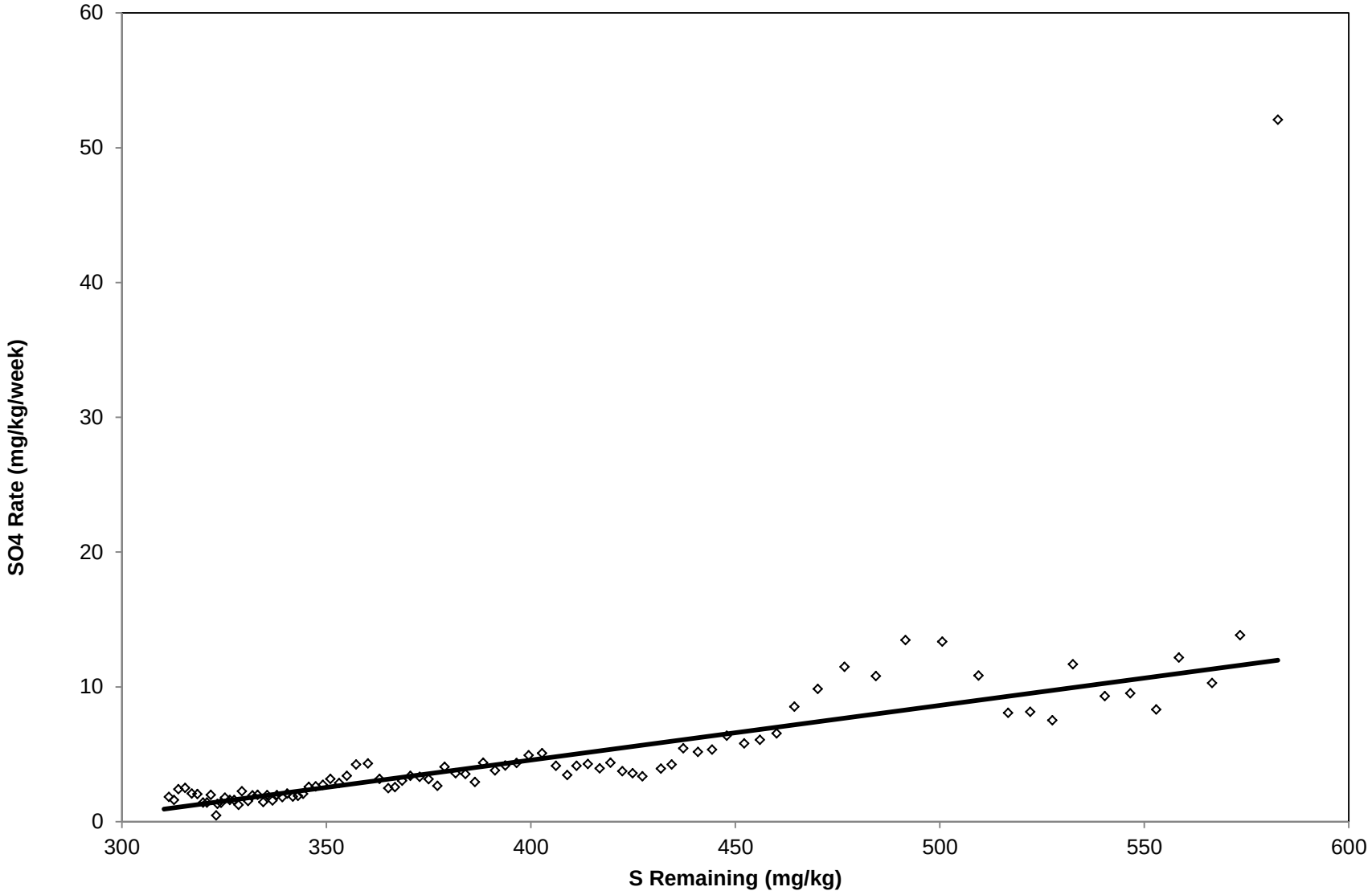

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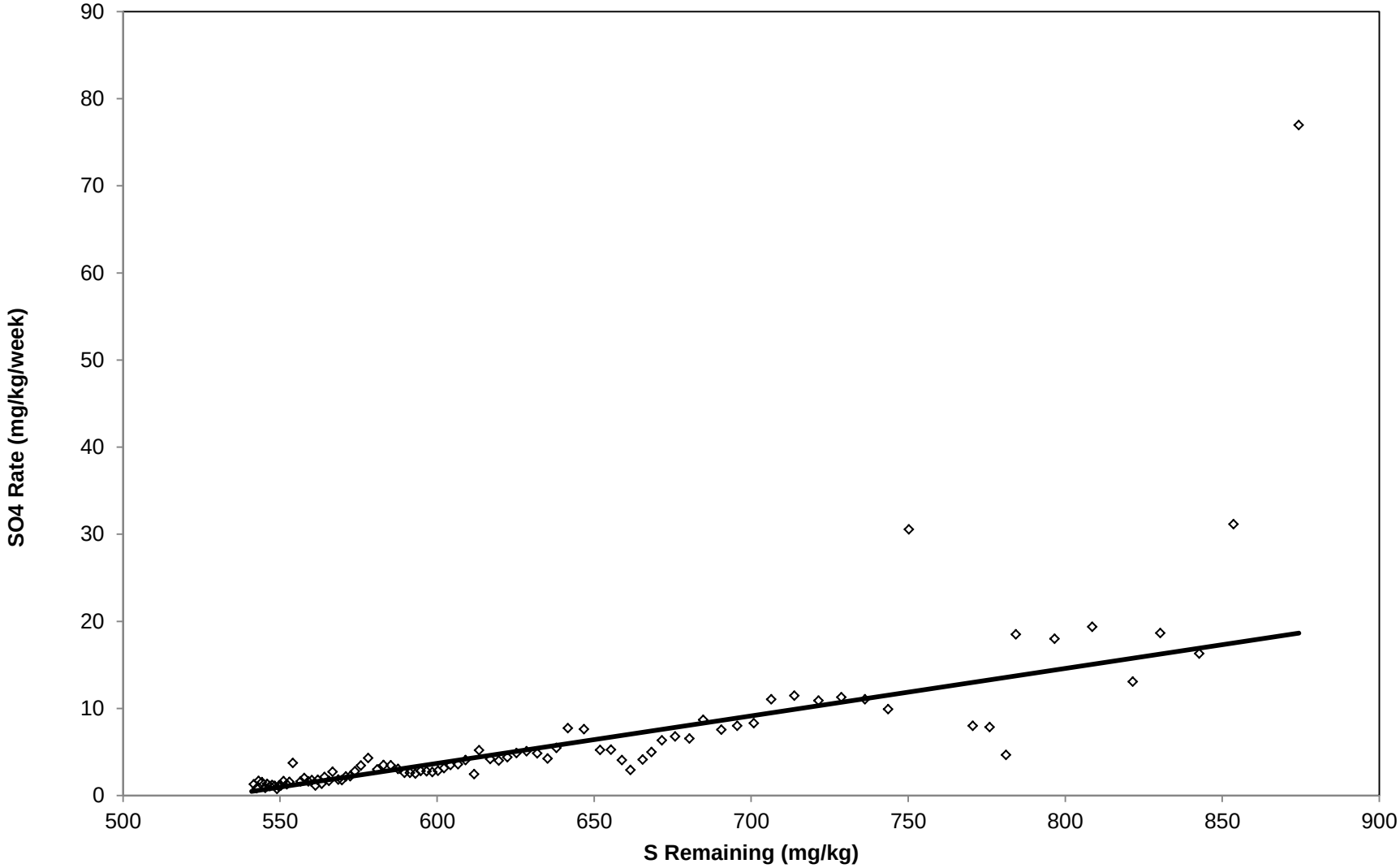

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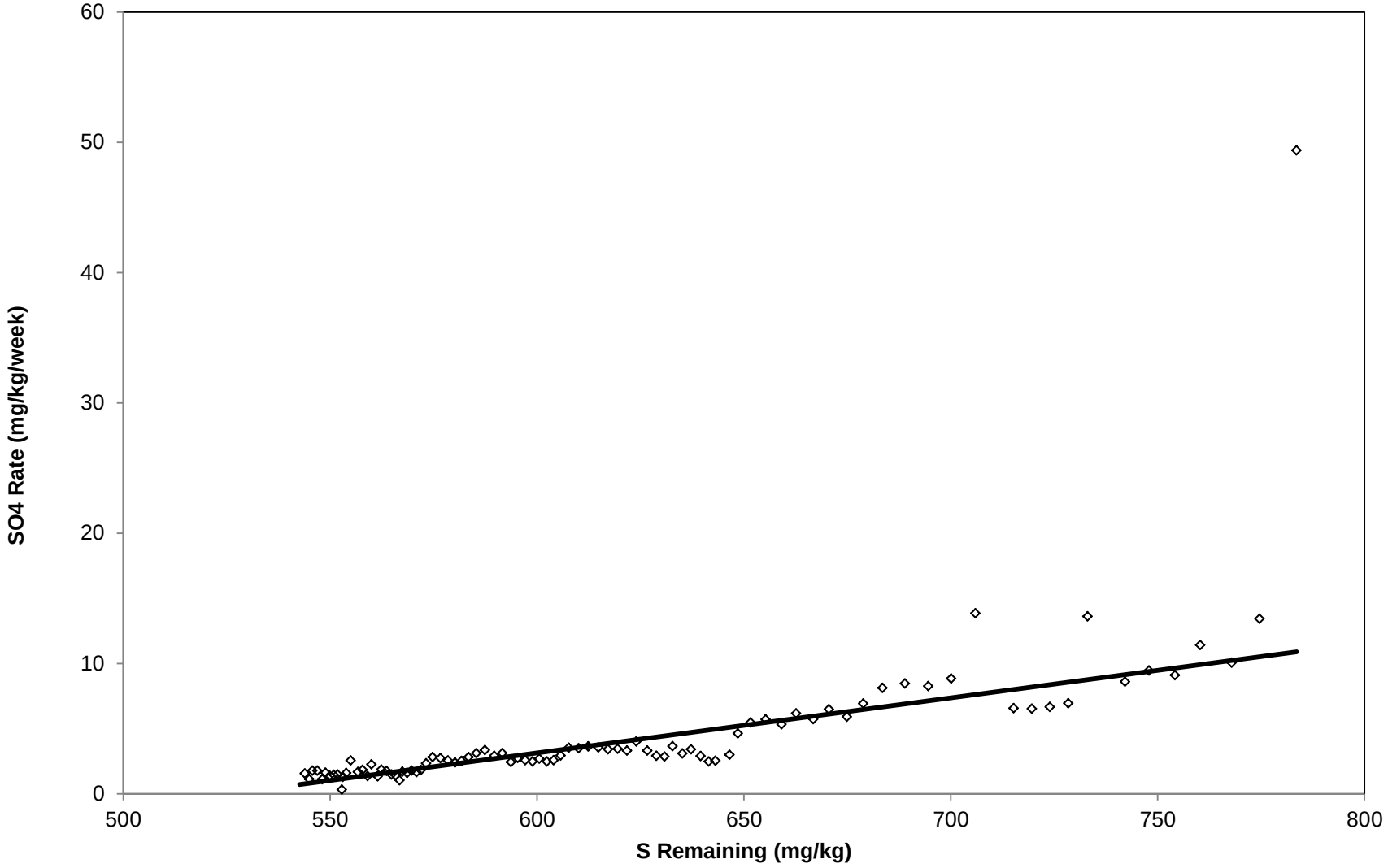

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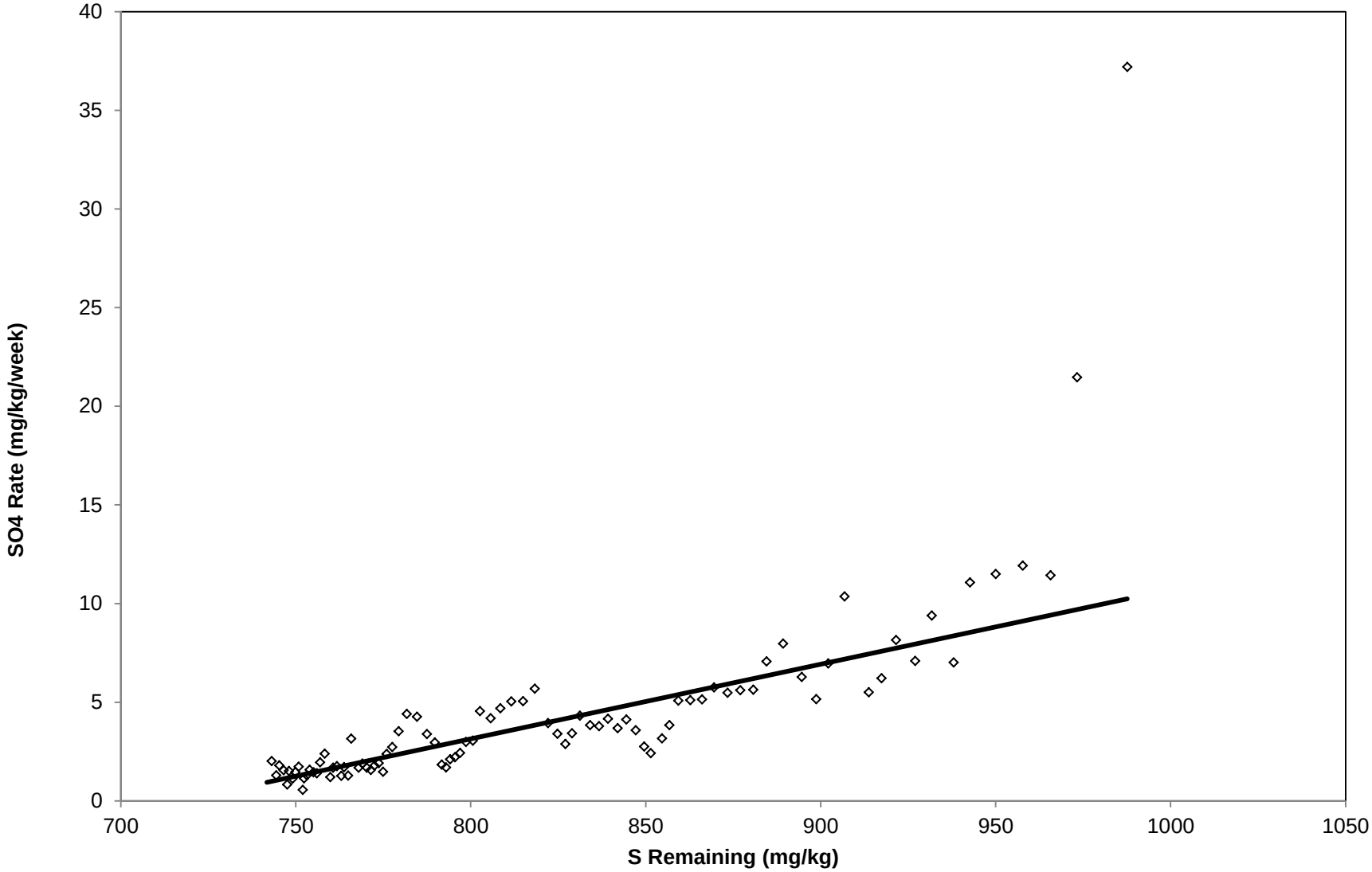

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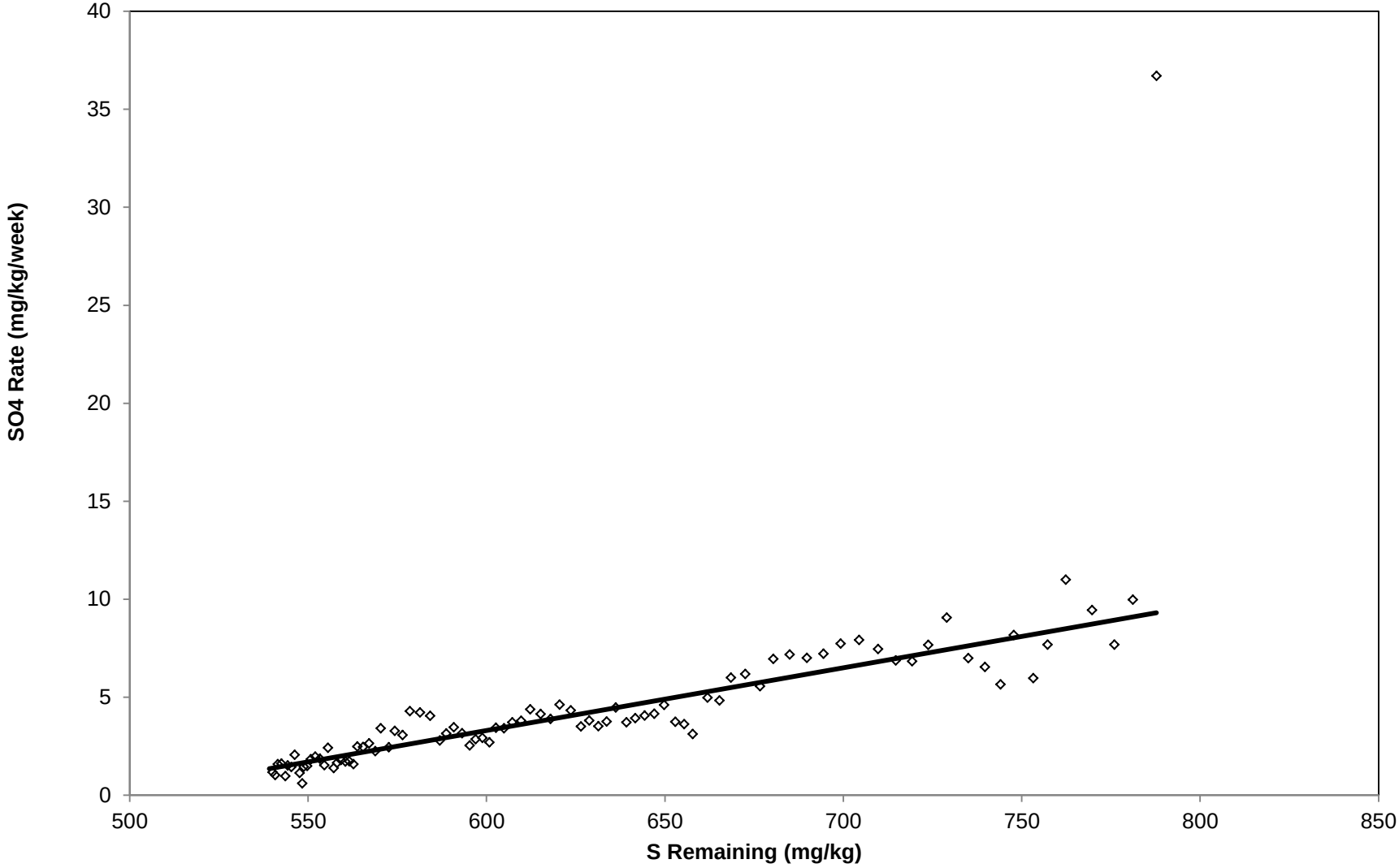

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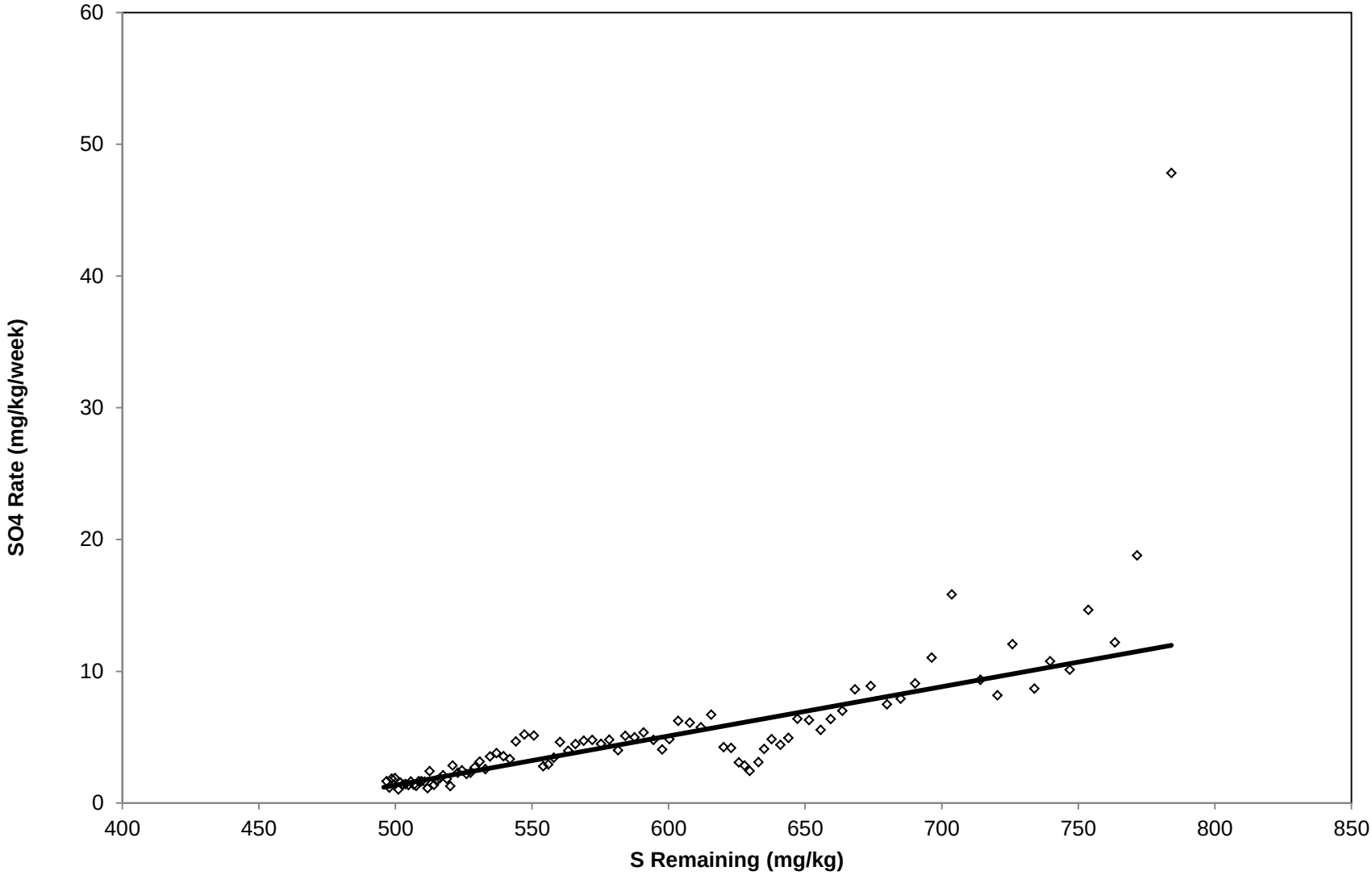

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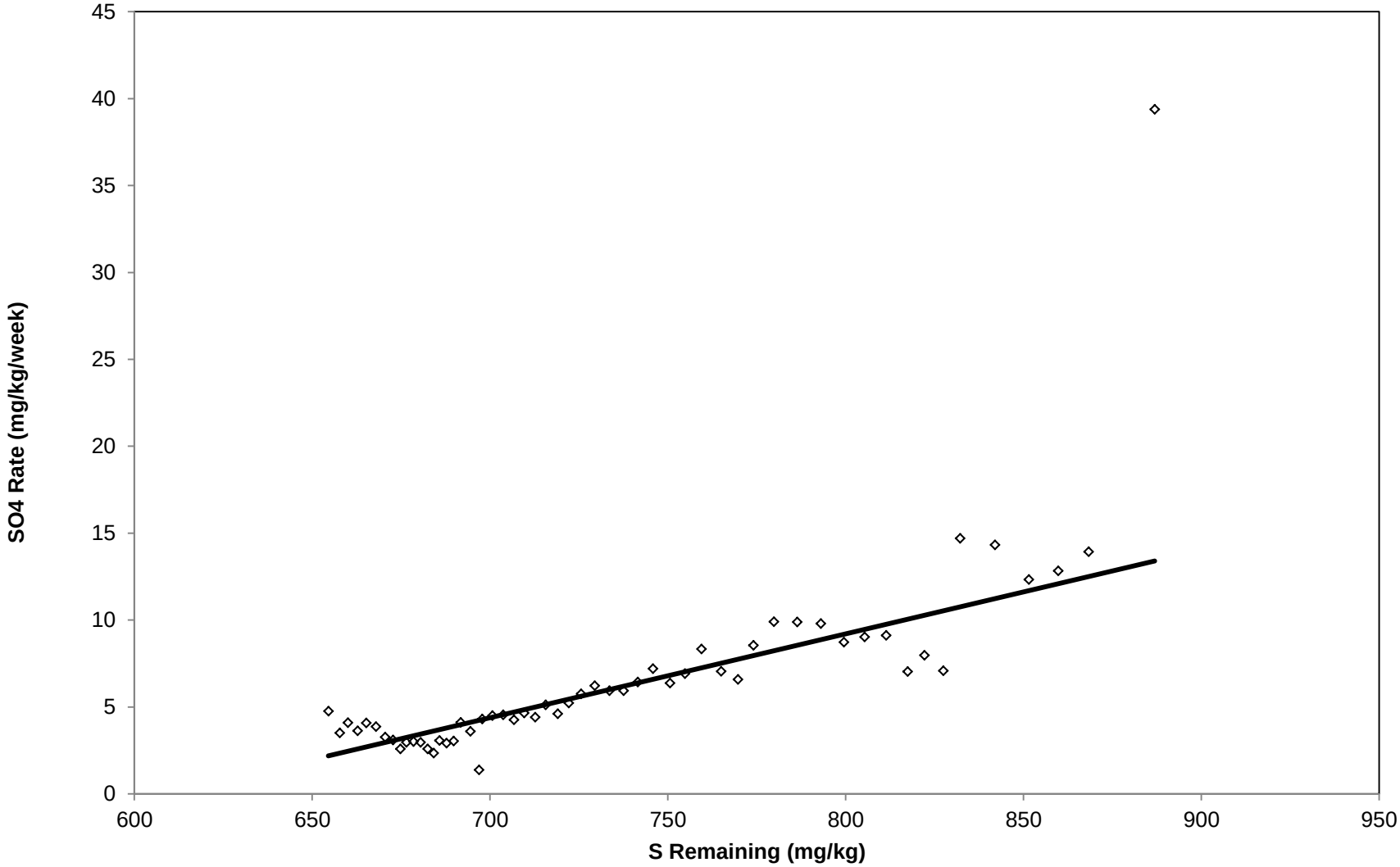

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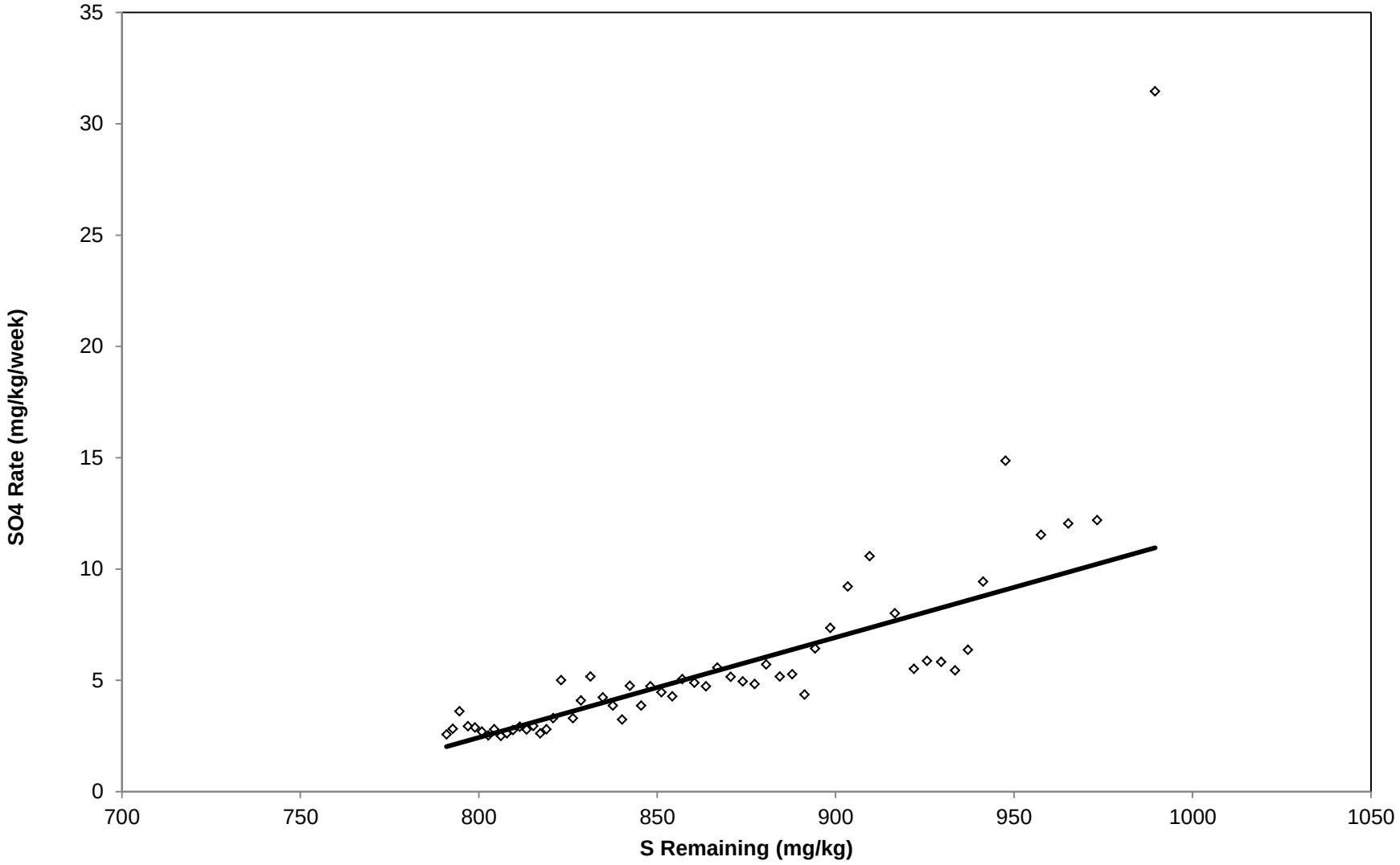

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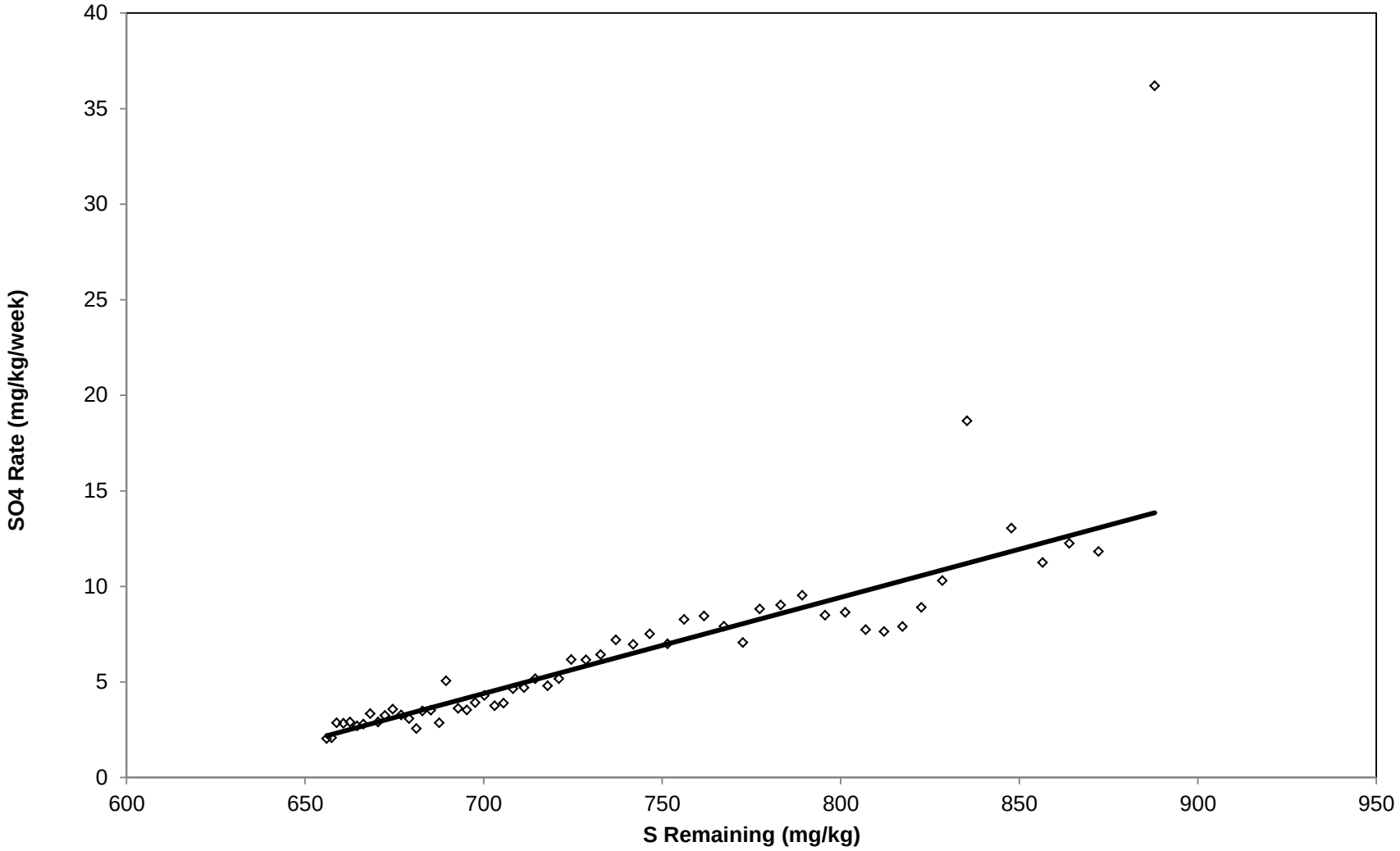

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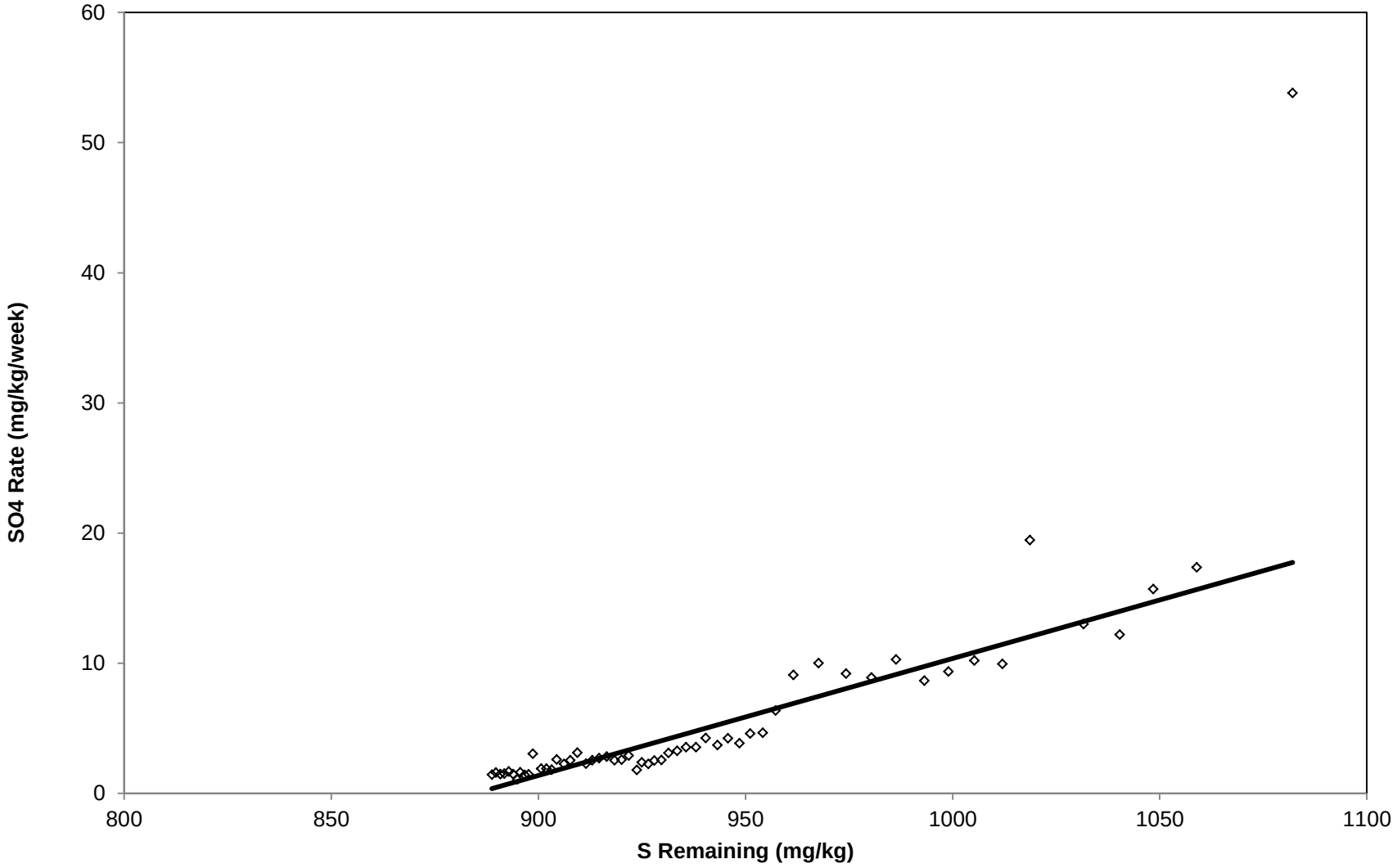

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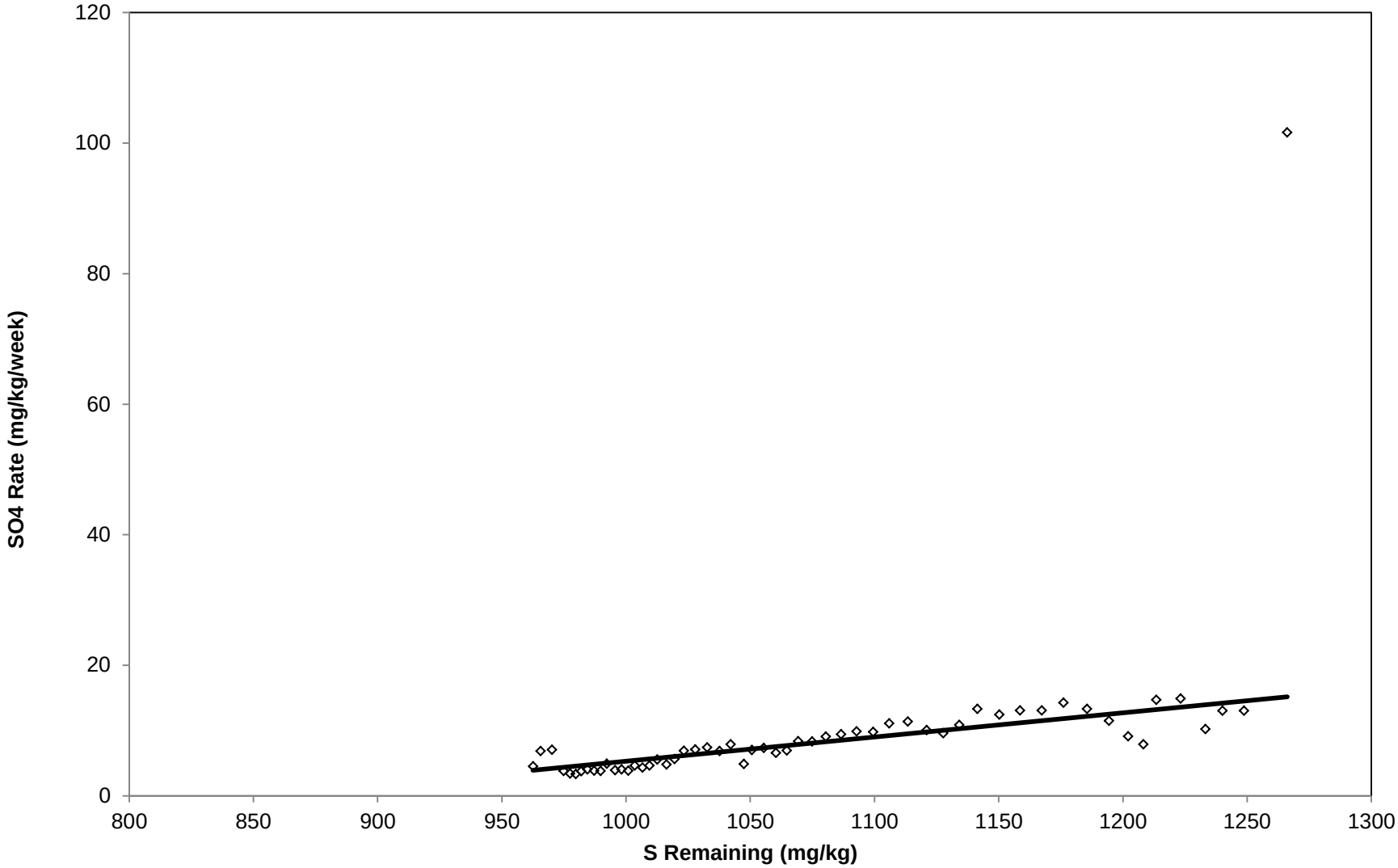

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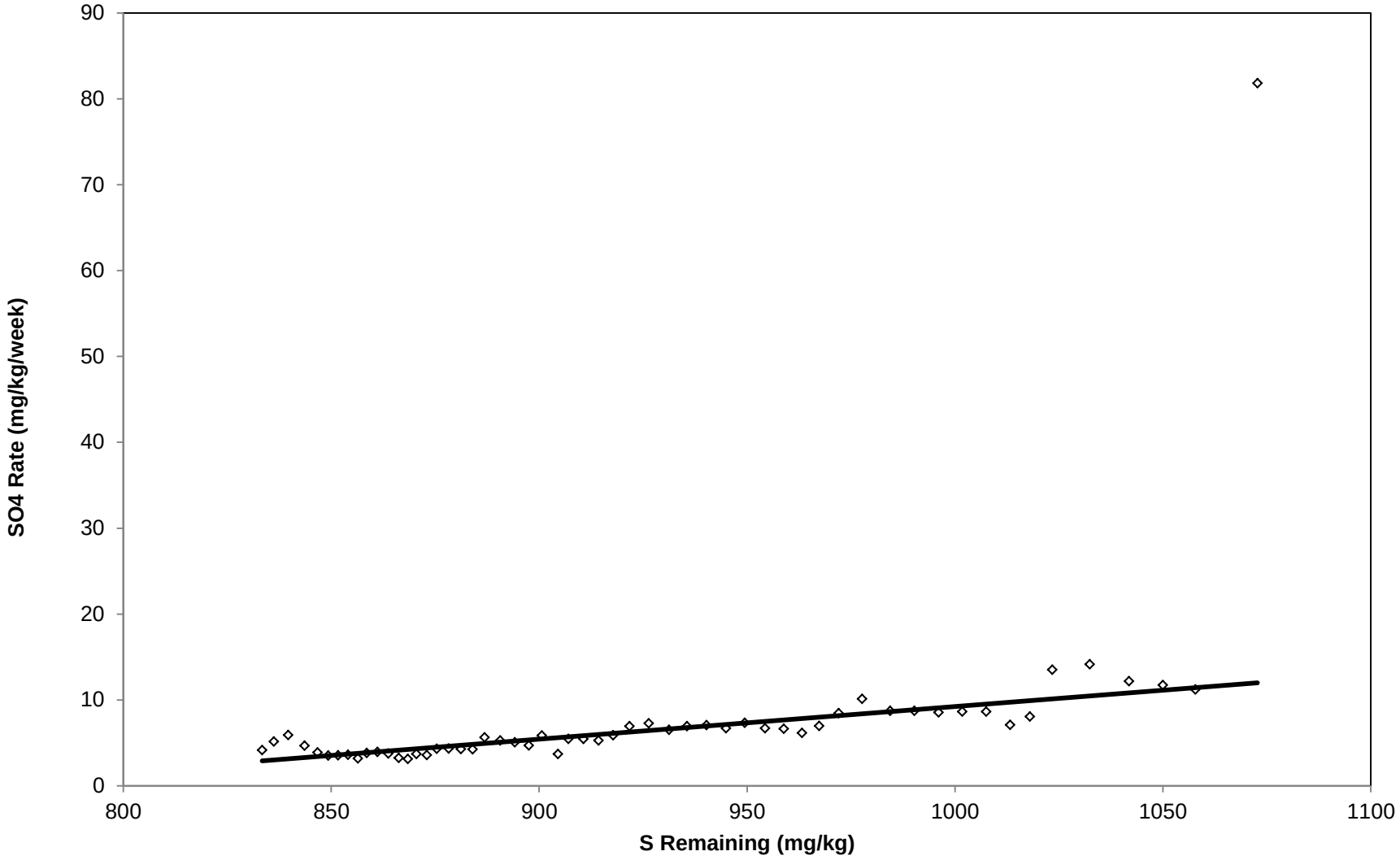

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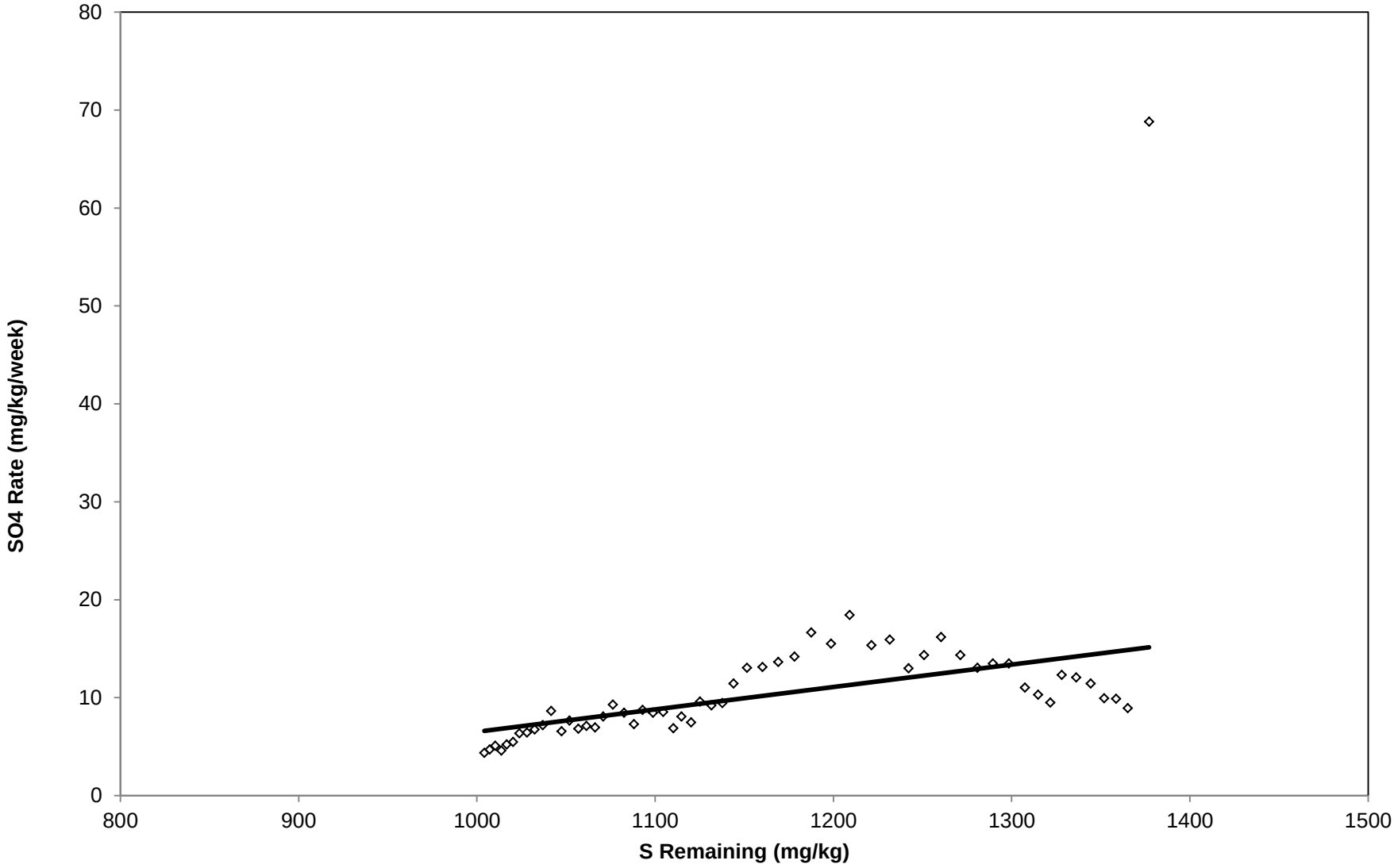

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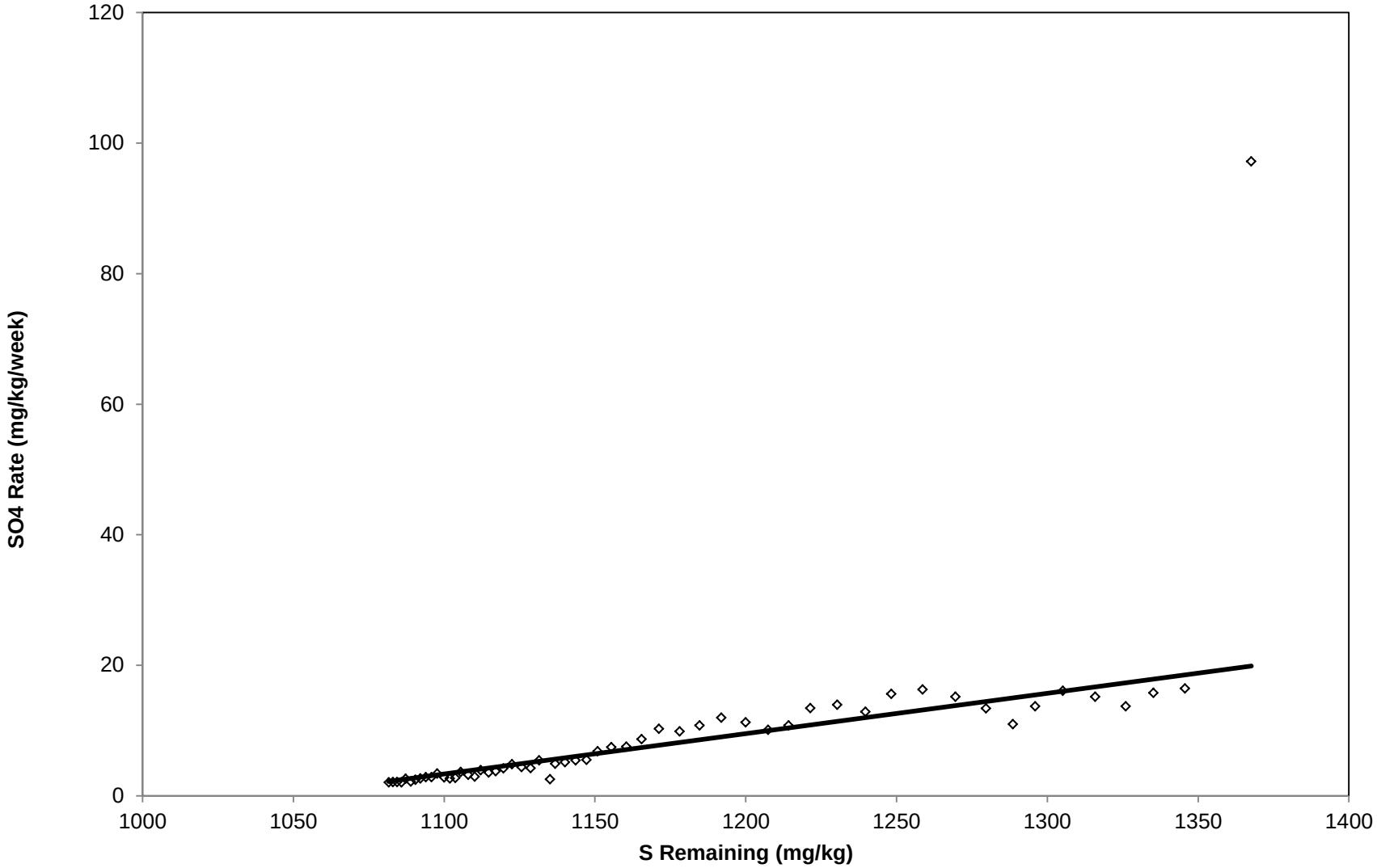

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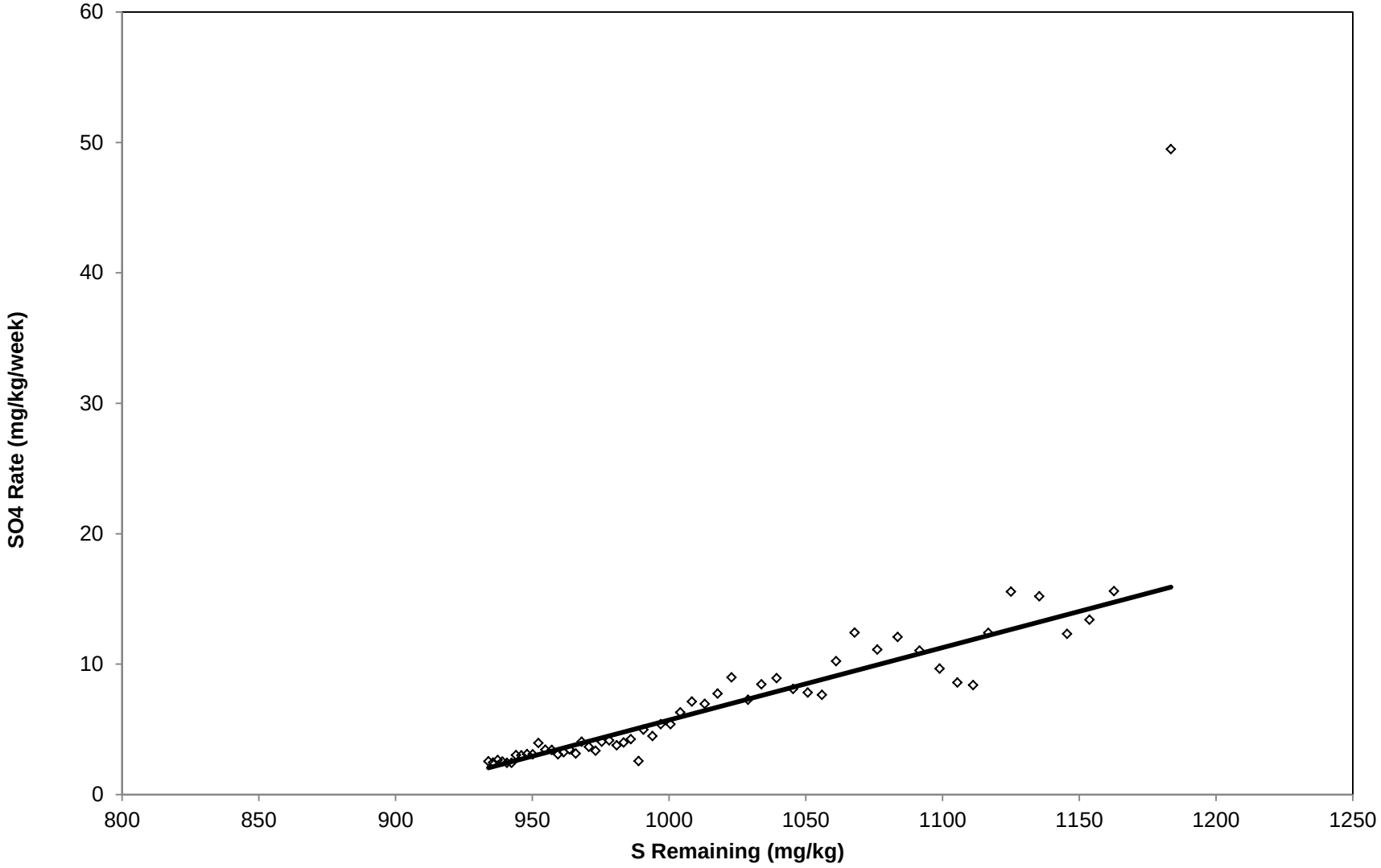

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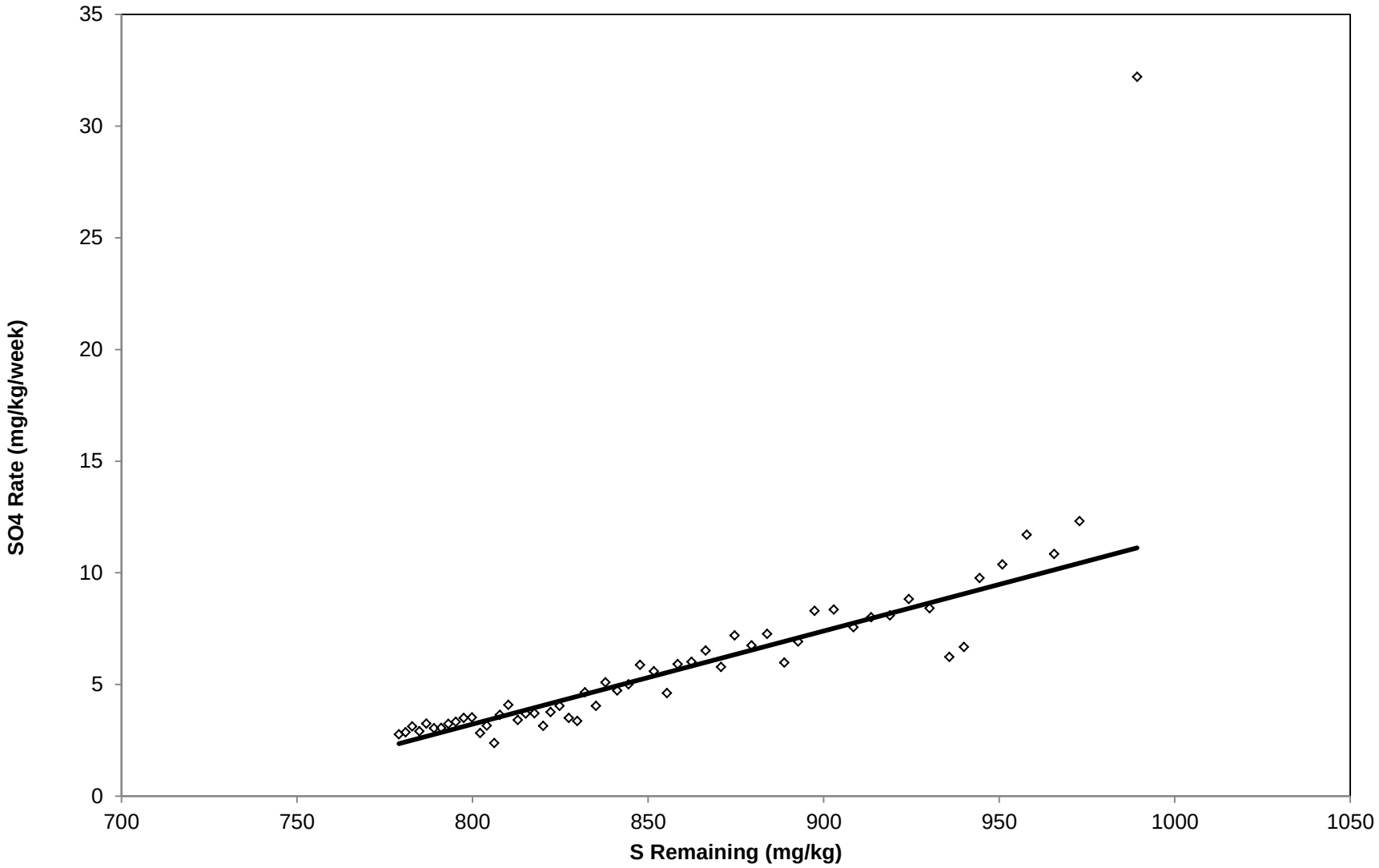

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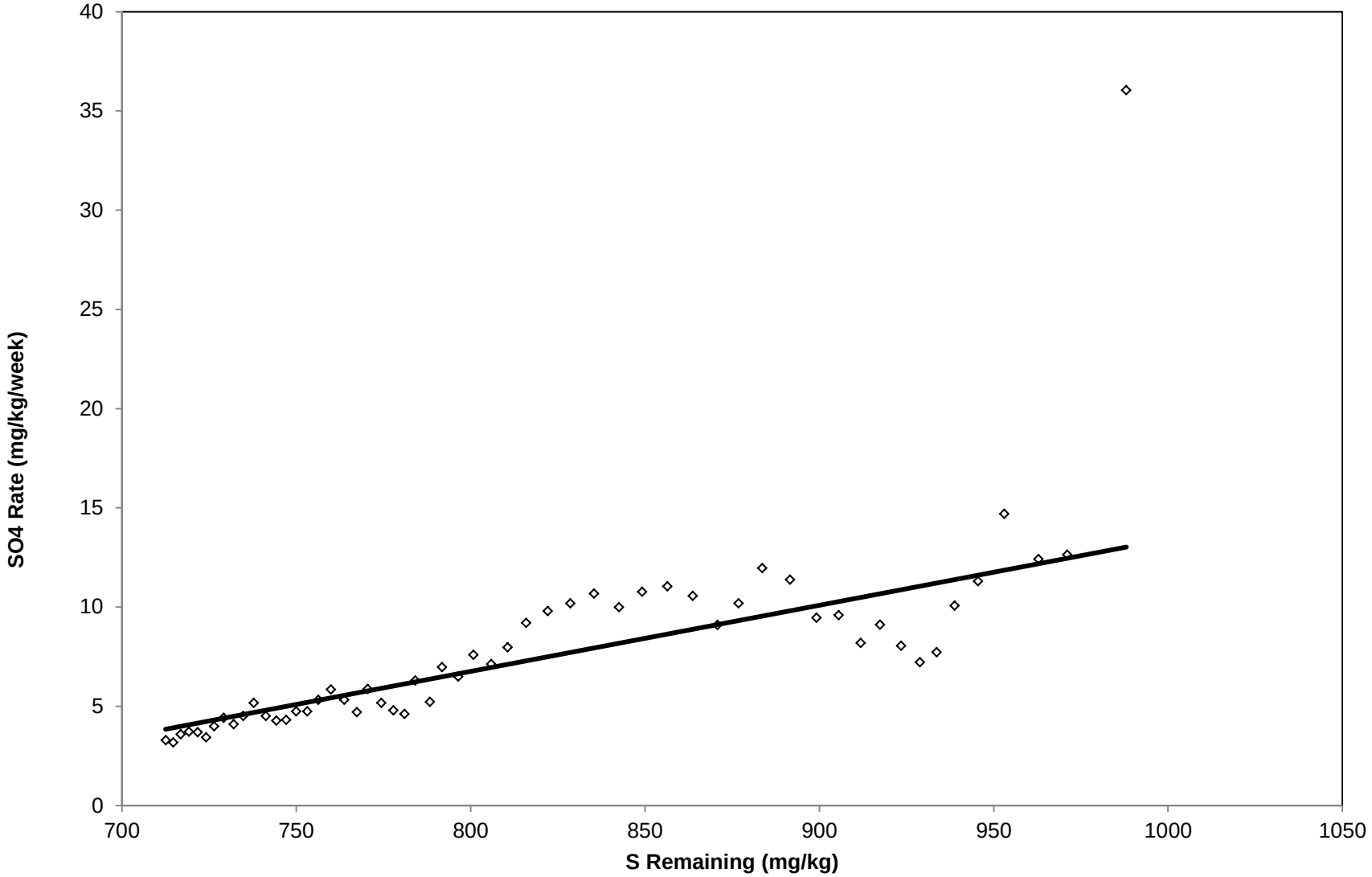

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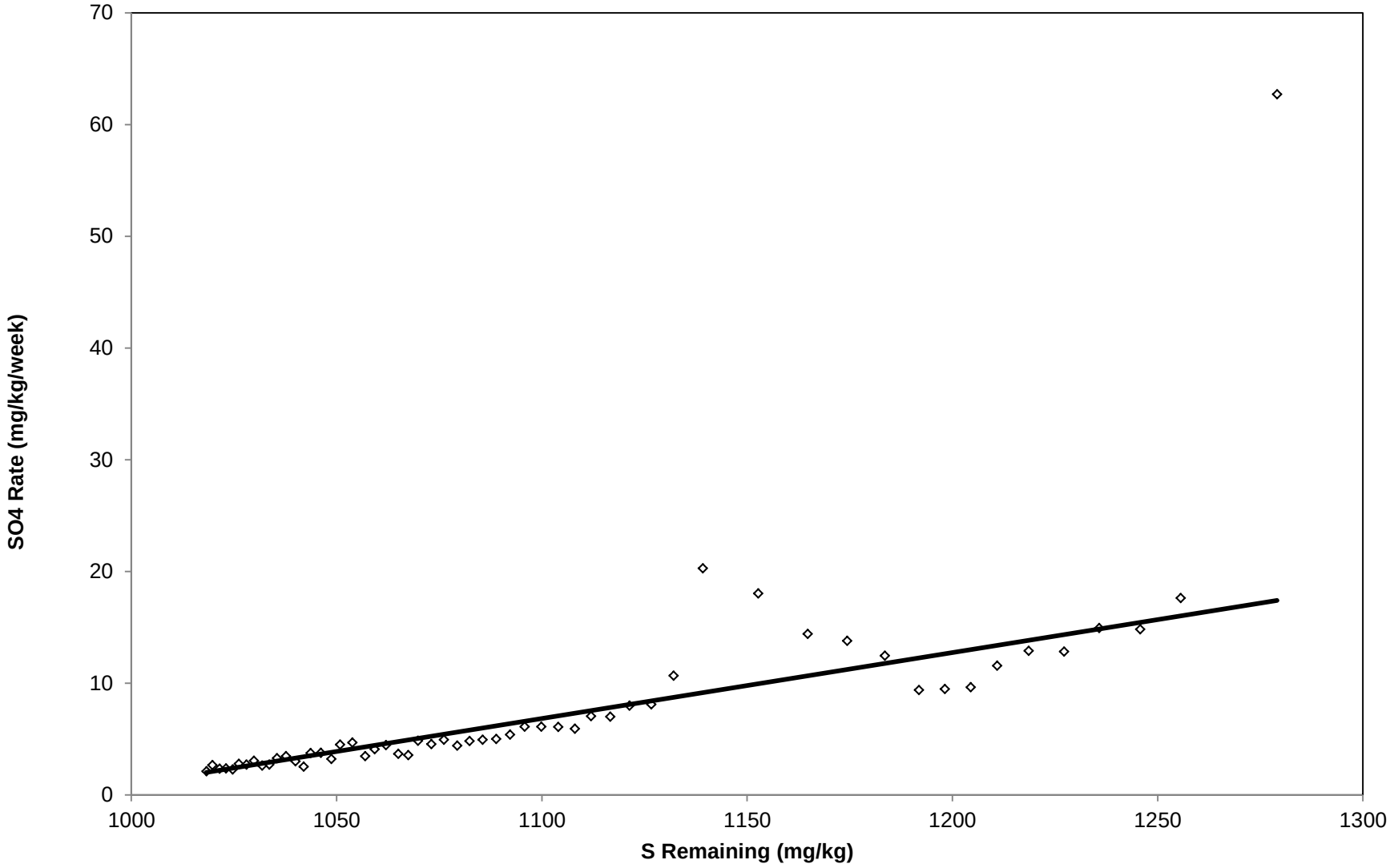

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Attachment 2-20

SRK Consulting. NorthMet Project Kinetic Test Program Status and Recommendations. Memorandum to Jim Scott, PolyMet. March 17, 2014.

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Memo

To:	Jim Scott Jennifer Saran Andrew Ware	**Client:**	PolyMet Mining Corp.
From:	Stephen Day	**Project No:**	1UP005.001
Cc:	Laura Donkervoort Rosemarie Cocuaco	**Date:**	March 17, 2014
Subject:	NorthMet Project Kinetic Test Program Status and Recommendations – DRAFT		

1 Background

Kinetic tests (mainly humidity cells) were initiated for the NorthMet Project beginning in 2004 on samples of rock, Flotation Tailings and Hydrometallurgical Residues produced in a pilot plant according to overall program designs prepared following discussion between PolyMet Mining Inc. (PolyMet), Minnesota Department of Natural Resources – Lands and Minerals (MDNR-LAM) and SRK Consulting (Canada) Inc. (SRK). Results from these tests have been used at various junctures of the project to develop waste management plans and evaluate project environmental effects. Additional tests on LTVSMC tailings planned for dam construction have been started to evaluate that aspect of the project. In 2009, the program was modified in consultation with MDNR-LAM to stop some tests and modify the frequency of analysis of leachates based on trend interpretation (SRK 2009).

The purpose of this memorandum is to provide an update on the progress of the testwork and provide recommendations for modification to the program.

2 Approach

Some components of the program have now yielded over eight years of data on dissolution trends and are showing strong consistency of results. Furthermore, due to slow weathering processes documented by this program and similar programs performed under the direction of MDNR-LAM, few significant changes have been observed in the testing program compared to those reported previously in the last status report (SRK 2012).

For this reason, the approach taken for the current review was to compile comparative observations to determine where modifications can be made to the program. Experience at other mining projects shows there is significant value in continuing some representative tests to verify conclusions drawn from the current dataset as the mine moves into production.

The following sections provide observations on trends observed for each of the major programs (rock, flotation tailings, LTVSMC tailings) and resulting recommendations (summarized in Table 1).

With agreement from MNDR-LAM, testing of residues from the hydromet process produced in 2005 was stopped in 2009 and is not reported in this memorandum. Testing of one sample of hydrometallurgical residue produced from a 2009 pilot plant was started in April 2010 and is continuing. It should be noted that the 2009 hydrometallurgical pilot plant was run with limited concentrate, so it was necessary to run part of the pilot plant using batch processing rather than continuous processing. The batch approach was appropriate to answer metallurgical questions, but produced residue that is not expected to be representative of the NorthMet hydrometallurgical residue during operations. Consequently, plans for environmental testing of the 2009 pilot plant residue (other than the autoclave leach residue, which was produced by a continuous process) were discontinued.

3 Analysis Recommendations

Leachate analysis for continuing tests is being performed at a frequency agreed with MDNR-LAM (SRK 2009a). As the testwork continues, it is clear that trends evolve over periods of years rather than weeks or months, and that trend definition can be achieved with testing on an infrequent basis. Table 2 provides analysis schedule recommendations for all recommended ongoing tests.

SRK (2009b) provided a protocol for adjustment of analytical frequency based on pH trends. Due to the tendency for trends to develop slowly, the proposed schedule shown in Table 2 will be able to detect changes with sufficient resolution that increased analytical frequency is not needed. SRK recommends that the frequency be fixed as proposed in Table 2.

4 Rock Characterization Program

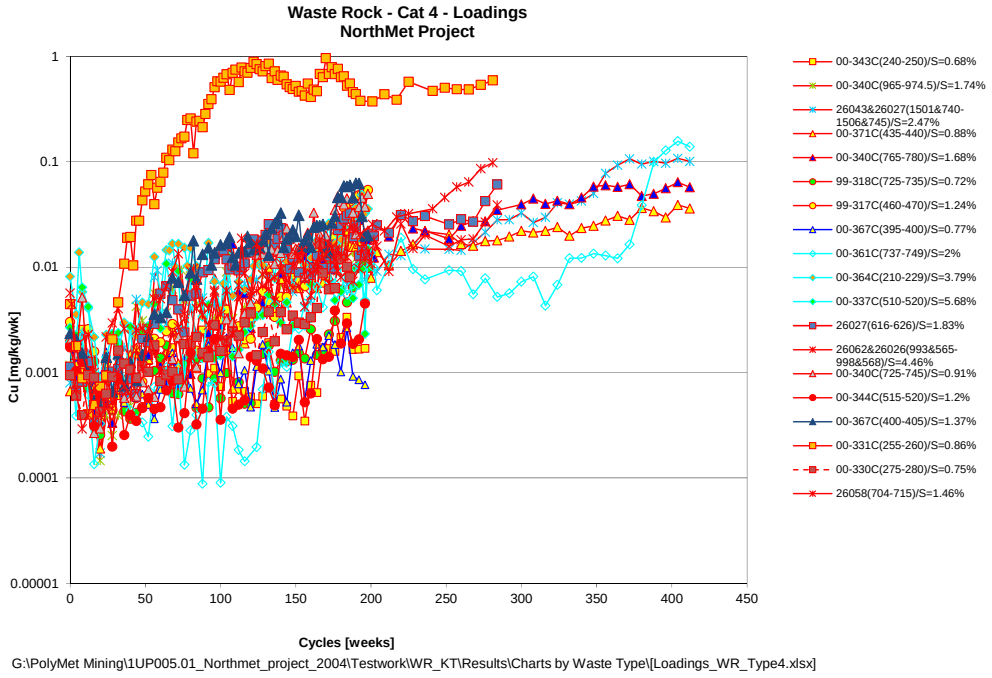
4.1 Observations

Currently, 24 humidity cell tests and 15 “size fraction” tests are operating with the breakdown by waste category and type as indicated in Table 1. Ongoing testwork shows that pH of leachates are consistent with the waste categories:

- No changes in Category 1 tests have been observed since the previous report (SRK 2012). pH variations are apparent but remain within historical ranges.
- Some Category 2/3 tests generated acid as described in previous reports but a number of tests are continuing to yield leachate pHs that are above the pH of the drainage water used to leach the samples and therefore imply a delay to onset of acidic leachate exceeding eight years.
- All continuing Category 4 tests have generated acid as previously reported. The testwork has demonstrated that most Category 4 rock can be expected to generate acid and that the

timeframe to onset is several years in most cases. The main observation for the continuing tests is the upward trend in copper release (Figure 1).

- All three ore composite samples are showing the same trend with pH being comparable to Category 2/3 rock.
- The parallel tests on size fractions and using the DNR Reactor test configuration has shown that differences in rates can be explained by differences in reactive surface area and water application ratio but the tests do not fundamentally change how rock weathering is interpreted.



G:\PolyMet Mining\1UP005.01_Northmet_project_2004\Testwork\WR_KT\Results\Charts by Waste Type\[Loadings_WR_Type4.xlsx]

Figure 1. Copper Release for Category 4 Samples

Table 1. Status of Testwork, Findings and Recommendations

Program	Sub-program	Type of Test	Original Total Tests	Tests Operating	Duration (weeks)	Status	Conclusions	Recommendation and Rationale	Tests to Continue
Rock	Category 1	HCT	37	7	420	Stable, no change since last report.	Tests are confirming pH near neutral and varying within historical range.	Continue three tests to monitor range of S content. Demonstrate that Cat 1 does not generate acid.	3
Rock	Category 2/3	HCT	26	10	420	Eight tests are stable pH above 6 with no change from last report. One test (UM, S=0.2% is showing lower pH with variations within historical range. Metal release peaked several years ago.	Tests have shown the majority of samples in Cat 2/3 do not show pH depression below 5 for at least eight years. The majority of tests are maintaining pH above 6 indicating that acid generation under site conditions would not occur for at least eight years.	Continue three tests with lower S contents to evaluate use of a higher criterion for segregation of Cat 1 from 2/3. Continue one test at highest S content to evaluate delay to onset of acidic conditions.	4
Rock	Category 4	HCT	22	4	420	Two tests with S>=2% have pH<4. Two tests with S<1.7 have 4<pH<5 and declining but pH is within historical range. SO_4 is declining for these tests. Cu leaching rate is increasing for all four tests.	Continuing tests show acidic pH but no indication of further declines after 4 years.	Continue two tests to evaluate copper leaching. One test from each current pH group.	2
Rock	Ore	HCT	3	3	420	Results are very similar for all tests. pH stable at 5.	Average ore type materials show consistent behavior without generating severe acidity after eight years. Variability of oxidation rates for ore grade sulphur content (Cat 4) is shown by Cat 4 HCTs.	All three samples so show similar behavior. Continue 1 test to evaluate long term leaching of ore type materials.	1
Rock	All categories	Parallel HCT, DNR reactors	15	5 samples (3 additional tests per sample)	420	Tests are demonstrating stable leachate chemistry. Results remain consistent with lower rates for coarser size fractions but also influenced by pH of deionized water resulting in higher metal solubility for coarser fractions.	Tests have shown reactivity differences for size fractions.	Terminate all tests.	0
FlotationTailings	Pilot Plant 1	HCT	13	13 (4 bulk, and 9 size fractions)	390 to 420	The majority of tests are yielding pHs above 7 showing pH recovery from as low as pH 6 earlier in test. Sulfide consumption is 50% in some cases and sulfate release continues to show declining trends. Samples with higher S content (from not using $CuSO_4$ during processing) initially showed higher metal release but all are showing steady decline. Metal release from lower S tailings is approaching historical lows.	Test work has shown that significant pH depression is not expected. The consumption of a significant fraction of the sulfide while continuing to generate alkalinity supports the conclusion that alkalinity generated by silicate weathering will permanently offset acid generated by sulfide oxidation.	Due to the consistency of trend after eight years of testing, a limited number of tests should continue to confirm the conclusion that acid generation will not occur. The P3 samples are proposed as they represent typical performance. The bulk, fines (-200 mesh) and coarse (+200 mesh) fractions have been selected.	3
FlotationTailings	Pilot Plant 1	DNR reactors	13	13 (4 bulk, and 9 size fractions)	390 to 420	The trend is similar to the HCTs but with pH shifted down to mostly below 7. Otherwise trends are the same as HCTs. pH is tending upwards from historical lows.	The main factor resulting in differences is the small sample volume and higher liquid to solid ratio.	Due to the similarity with HCTs on the same material and explainable differences, all tests should be discontinued.	0
Flotation Tailings	Pilot Plant 2 and 3	HCT	8	8 (2 bulk, 6 size fractions	270	These tests showed decreasing pH initially but pHs have not decreased below what appears to be a strong buffered pH of about 7.3 which has been maintained for four years. Long term decline in sulfate release is observed. Alkalinity has been leached at a higher level than the PP1 tests which may explain the higher and sustained pH. As a result, metal leaching has been observed at low levels.	The testwork has shown that tailings generated by Pilot Plants 2 and 3 have been able to sustain higher pHs and lower metal leaching. These tailings had lower S content than PP1 and the results are therefore consistent with PP1 - lower S results in less acid generation and higher alkalinity.	Since these tailings demonstrate the performance with lower S content, it is recommended that one set of tests (bulk, +200 and -200) be continued in parallel with PP1 to conform that higher pH can be sustained. PP2 is proposed as it contains the highest sulphur content.	3
Flotation Tailings	Pilot Plant 2 and 3	DNR reactors	8	8 (2 bulk, 6 size fractions	270	The trend is similar to the HCTs but with lower pH.. A dip in pH to below 7 was apparent earlier in the program which was not apparent in the HCTs. The distinctive "buffered" chemistry is apparent but at lower pH.	The main factor resulting in differences is the small sample volume and higher liquid to solid ratio.	Due to the similarity with HCTs on the same material and explainable differences, all tests should be discontinued.	0
Flotation Tailings	Scavenger Tailings	HCT	12	12 (3 bulk, 9 size fractions	200	The tests showed the same general trend as PP1 with pH declining then increasing, though the lowest pH was no lower than 6.8. One test showed an unusual "sawtooth" pH trend resulting in pHs to 6.5 which is being evaluated with the laboratory. The pH depression was accompanied by detectable increase in metal leaching but two orders of magnitude below that of PP1 samples.	Results from these tests were consistent with those of other programs and demonstrate that pH depression and metal leaching are linked to S content. None of the tailings samples tested in any program have generated leachate at a pH below that of the deionized water.	Due to the consistency with other long term testwork, it is recommended that testing of the Oct 1 9 am group be continued as it contains the highest sulphur content.	3
Flotation Tailings	Scavenger Tailings	DNR reactors	12	12 (3 bulk, 9 size fractions	200	The trends are similar to the HCTs but with a lower pH.	The main factor resulting in differences is the small sample volume and higher liquid to solid ratio.	Due to the similarity with HCTs on the same material and explainable differences, all tests should be discontinued.	0
LTVSMC Tailings	Borrow materials	HCT	5	5	160	All tests are showing stable leachate with pH above 7.3, stable alkalinity and sulfate release below 1 mg/kg/week. Very low, barely detectable metal leaching is occurring.	The results are consistent with the low sulfide content and excess of carbonate minerals. A trend toward lower pH is not expected.	Discontinue all tests.	0
Flotation Tailings on LTVSMC Tailings		Column	Two LTVSMC tailings controls, four layered tests	Two LTVSMC tailings controls, four layered tests	390	These tests were designed to evaluate geochemical effects from placing Flotation tailings on LTVSMC tailings. They are yielding stable trends that are consistent with the mineralogy of the samples and the parallel HCTs. Resultsare similar between tests despite testing different variables. The LTVSMC tailings are removing As and V leached from Flotation tailings. A weak Ni removal effect is apparent.	The testwork shows that placing Flotation tailings on LTVSMC tailings does not seem to contribute to leaching of LTVSMC tailings but LTVSMC tailings may be beneficial in removing some components leached from Flotation Tailings. The removal process is probably occurring via adsorption.	Due to the potential value of LTVSMC tailings to remove metals from Flotation leachate by adsorption, continuation of some testwork is recommended to evaluate "break through" effects. Recommended testing includes - Maintain one control due to similar performance of both (coarse selected). Monitor bottom port only. - Maintain the two P1 experiments because they are leaching highest As and are likely to breakthrough sooner. Monitor top and bottom ports only. - Perform testwork on decommissioned columns to look for precipitates.	One control, two column sets
Hydrometallurgical Residue		HCT, SFE, DNR Reactor	3	3	165	Leachate chemistry reflects dissolution of soluble components. pH has stabilized at near 5 as dissolution of the residue continues to occur.	Due to the high liquid to solid ratio, the testwork represents a very long duration under natural site conditions.	Discontinue both tests.	0

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Table 2. Analytical Schedule

Parameter Group	Original Schedule	Current Schedule	Recommended
Leachate recovery	Weekly	Weekly	Weekly
pH, conductivity	Weekly	4-weekly	Quarterly
Alkalinity, sulfate	Bi-weekly	4-weekly	Quarterly
ORP, acidity	Weekly, bi-weekly	4-weekly	None
Total inorganic carbon	Bi-weekly	None	None
Fluoride, chloride	Bi-weekly	None	None
ICP ES Elements	4-weekly (0, 4, 8, 12, etc.)	None	None
ICP MS Elements	4- weekly (2, 6, 10, etc.)	8-weekly (2, 10, 18, etc)	Twice annually

4.2 Recommendations

Ten samples are been recommended for continuation (Table 3) with the following objectives:

- Validate that Category 1 rock does not generate acidic leachate. Three samples were selected to cover the range of sulfur concentration.
- Continue three tests with lower S content in Category 2/3 as this may provide data to determine if the segregation criterion is appropriate.
- Continue one test at higher S content in Category 2/3 that currently has not generated acid to evaluate delay to onset of acidic conditions.
- Continue two Category 4 tests (one from each current pH group) to evaluate the copper leaching trend.
- Continue testing on one ore sample to monitor long term performance of typical ore type materials.

Implementation of these recommendations will result in 14 of the current tests being discontinued. For the residues from the tests, optical mineralogical characterization is recommended to evaluate oxidation features.

Table 3. Recommended Rock Humidity Cells for Continuation

Sample	Rock Type	Waste Category	S %	Reasoning
DDH-00-334C(580-600)-14	Troctolitic	1	0.06	Higher S
DDH-00-367C(290-310)-51	Troctolitic	1	0.04	Mid-Range S
DDH-26064(44-54)-56	Troctolitic	1	0.02	Lowest S
DDH-00-350C(580-600)-17	Troctolitic	3	0.19	S<0.2, evaluate segregation criterion
DDH-00-369C(335-345)-34	Troctolitic	3	0.18	S<0.2, evaluate segregation criterion
DDH-00-367C(170-175)-54	Troctolitic	3	0.51	High S, test onset, lower pH in this group
DDH-00-357C(535-540)-23	Ultramafic	3	0.2	S=0.2, lowest pH leachate
DDH-00-371C(435-440)-19	Troctolitic	4	0.88	Acidic but pH>4
DDH-00-361C(737-749)-62	Virginia	4	2	Acidic and pH<4
P2-0-67		4	0.9	Typical of average ore

Source: G:\PolyMet Mining\1UP005.01_Northmet_project_2004\Reports\2013-11_KT_Update\[TestStatusSummary_1UP005001_SJD_REV00.xlsx]

5 Northmet Project Flotation Tailings Characterization Program

5.1 Observations

The current tailings test program consists of 66 tests in HCT and DNR reactor configurations on Flotation Tailings samples obtained from four different pilot plant runs. The longer tests have run for more than eight years and nearly 50% of sulfur has been depleted in some cases. Consistent patterns have emerged from all these programs:

- No tests have generated pH lower than the deionized water used in the tests and detectable alkalinity is still being measured.
- Sulfide oxidation rates declined as sulfur was depleted indicating that the oxidation rate is not zero order. This was evaluated previously by SRK (2012).
  - Leachate pH declined initially, reached a low point after a few years than recovered. This pH depression is interpreted as representing slightly accelerated acid generation rates. This implies that in the long term there is a low likelihood of acidic conditions developing.
- Differences in sulfur content explain differences in oxidation rates and the degree to which pH is depressed. Lower pHs and higher oxidation rates are measured for samples containing higher sulfur content.
- Lower pHs result in accelerated metal (Ni, Co) leaching though only for the Pilot Plant 1 samples.
- The DNR reactors showed similar pH trends but at lower pHs. This is interpreted as reflecting the higher liquid-to-solid ratio used in the test compared to HCTs resulting in a stronger influence from the deionized water pH.

5.2 Recommendations

The following changes to the program are recommended:

- One set of tests from each of the major programs (PP1, PP2/PP3 and Scavenger tailings) should be continued to validate long term performance. For each set of tests, the bulk, +100 mesh and -200 mesh test would be continued and the -100+200 mesh sample stopped. The sample sets having the highest sulfur content in the bulk tailings and produced in a pilot run with copper sulfate activator added would be selected for continuation.
- All DNR Reactor tests should be discontinued because they are not providing additional information beyond that of the humidity cells.

The resulting number of continuing tests will be nine. For discontinued tests, the following analyses are recommended:

- Total sulfur analysis to provide mass balance confirmation with the original sample and leached sulfate.
- Optical mineralogy to evaluate dissolution features.

6 LTVSMC Tailings Borrow Sources Characterization Program

6.1 Observations

Five samples of weathered LTVSMC tailings are being tested in humidity cells. These samples have shown stable leachate with slightly basic pH consistent with low initial sulfur content and elevated carbonate content. No significant changes in leachate pH are anticipated.

6.2 Recommendation

It is recommended that these tests be terminated because leachate chemistry is stable and no long term changes are expected.

7 NorthMet Flotation Tailings on LTVSMC Tailings Column Tests

7.1 Observations

These tests were designed to evaluate geochemical effects from placing Flotation Tailings on LTVSMC tailings. The tests are constructed as sequential columns with oxygen control at intermediate sampling points. The tests have been operating for nearly eight years.

The following are major observations from this test program:

- Two control tests containing only LTVSMC tailings (coarse and fine fractions) yielded basic pH leachate containing high alkalinity concentrations which increased along the flow path. Some differences were observed between the two tests but leachate characteristics were generally similar and consistent with pH. Long term flushing trends were apparent for a number of parameters including sulfate, arsenic and molybdenum.
- All four column sequences testing the influence of Flotation Tailings on LTVSMC tailings showed similar results.
- Leachate from the Flotation Tailings had slightly lower pH when compared to LTVSMC tailings and showed declining sulfate release indicating depletion of sulfide minerals. Overall leachate chemistry was similar to humidity cells yielding leachate pH above 7. Metal concentrations were very low and accelerated leaching comparable to lower pH humidity cells was not apparent.
- Leachate from Flotation Tailings did not appear to influence leaching of the LTVSMC tailings but the LTVSMC tailings appeared to remove arsenic, vanadium and to a lesser extent nickel from Flotation Tailings leachates (examples shown in Figure 2). This is suspected to be a result of adsorption to LTVSMC tailings.

7.2 Recommendations

Because the LTVSMC tailings may be important in limiting metal leaching from Flotation Tailings, the following continuation of tests is recommended to evaluate breakthrough effects:

- Continue one set of controls due to similar performance of both coarse and fine LTVSMC tailings. The coarse fraction has been selected. Monitor bottom port only.
- Continue the two P1 sequence tests because they are leaching highest arsenic and are likely to breakthrough sooner. Monitor top and bottom ports only.

For decommissioned tests, testing of the column residues is recommended to determine where adsorption occurred in the column.

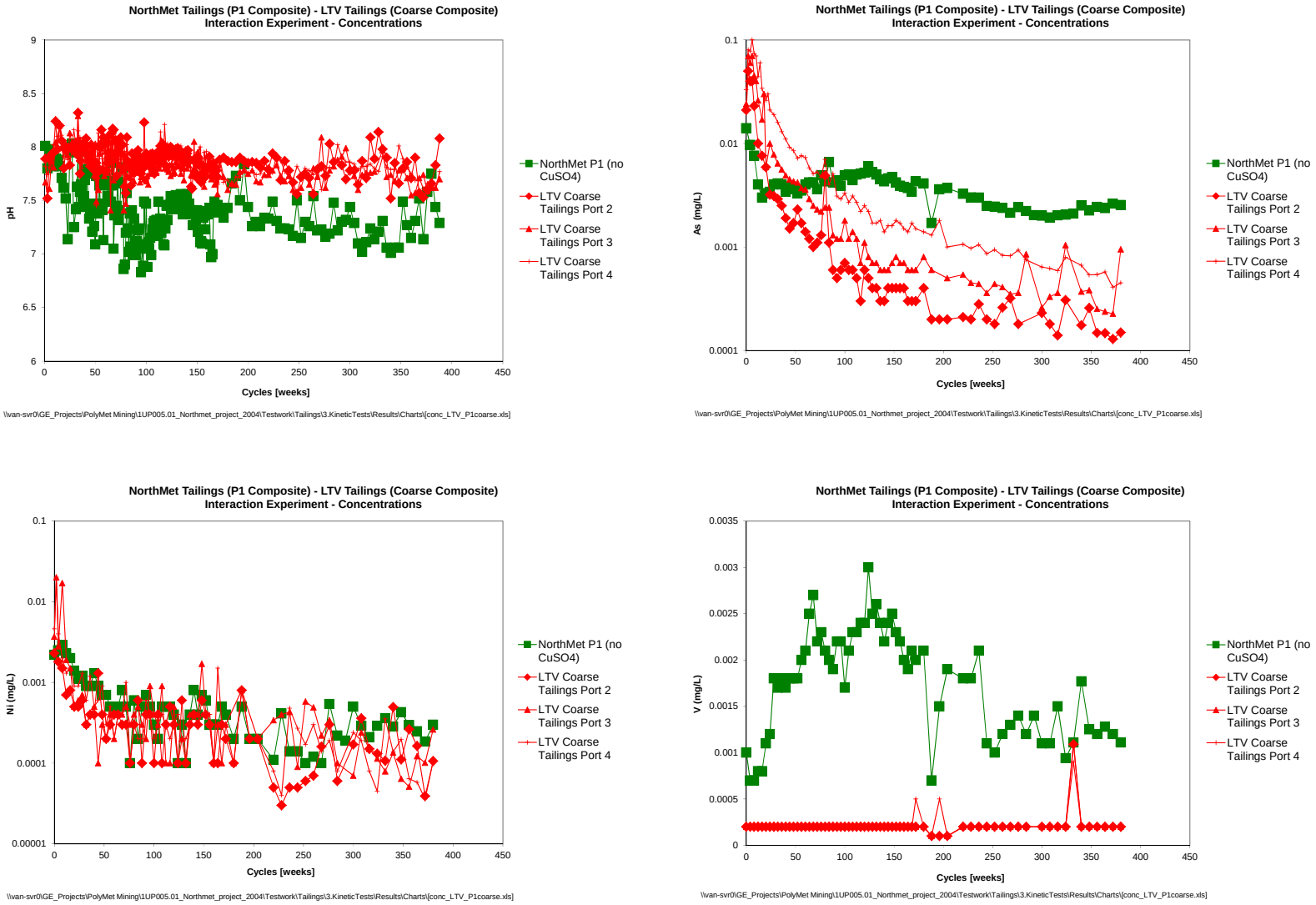

Figure 2. Examples of Leaching Trends for Flotation and LTVSMC Tailings Columns

8 Hydometallurgial Residue

8.1 Observations

One sample is being tested in humidity cell, shake flask and DNR Reactor configurations. All tests show trends consistent with flushing of soluble products. pH has stabilized at a consistent level.

8.2 Recommendation

Discontinuation of all three tests is recommended because the amount of flushing that has occurred far exceeds expected flushing under field storage conditions, and leachate chemistry is either stable or showing downward trends.

9 Conclusions

Long term (exceeding eight years in some cases) kinetic testwork for the NorthMet Project has yielded consistent results that support the interpretations made earlier in the program used to develop waste management criteria and water chemistry predictions.

As a result, SRK recommends that continuation of a limited number of tests is appropriate to validate predicted long term pH stability. In addition, based on the data obtained thus far, changes in leachate chemistry are expected to be detected with very low frequency analysis. It is recommended indicator parameters (pH, conductivity, sulfate and alkalinity) be measured quarterly and metals measured twice annually.

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The opinions expressed in this report have been based on the information available to SRK at the time of preparation. SRK has exercised all due care in reviewing information supplied by others for use on this project. Whilst SRK has compared key supplied data with expected values, the accuracy of the results and conclusions from the review are entirely reliant on the accuracy and completeness of the supplied data. SRK does not accept responsibility for any errors or omissions in the supplied information, except to the extent that SRK was hired to verify the data.

10 References

SRK Consulting. 2009a. RS82 - Update on Use of Kinetic Test Data for Water Quality Predictions – DRAFT 02. Memorandum to Kim Lapakko, MDNR. February 7, 2009.

SRK Consulting. 2009b. pH Criteria for Modification to Kinetic Test Program - NorthMet Project. Memorandum to Jim Scott. April 28, 2009.

SRK Consulting. 2012. Update on Kinetic Test Data, NorthMet Project – DRAFT. Memorandum prepared for PolyMet Mining. SRK Project 1UP005.001. October 19, 2012.

Attachment 2-21

SRK Consulting. Results of Analysis from Overburden Drilling Program – DRAFT, NorthMet Project. Memorandum to Stuart Arkley, MDNR. October 16, 2008.

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Memo

To:	Stuart Arkley, MDNR	**Date:**	October 16, 2008
cc:	Jim Scott, PolyMet	**From:**	Stephen Day
	Miguel Wong, Barr Engineering		Madeleine Corriveau
Subject:	Results of Analysis from Overburden Drilling Program NorthMet Project – Doc Id GC05 – DRAFT	**Project #:**	1UP005.001

1 Introduction

This memorandum provides chemical results from the analytical program for overburden samples collected from the NorthMet Project in the area of the proposed mine site. The memorandum provides conclusions on the main factors controlling differences in leaching characteristics of the overburden and estimates of contact water chemistry.

The Overburden Geochemical Characterization Plan was provided to the MDNR on February 22, 2008[1] and the analytical plan was provided to the MDNR on March 18, 2008[2]. Both plans were reviewed and accepted by the MDNR with the understanding that subsequent characterization might be required prior to and during excavation of overburden material.

2 Summary of Field Program

A total of 225 feet of drilling was completed for geochemical characterization of overburden. The following summary of the field observations was provided in the analytical plan.

As expected, drift in the area has complex lithology. The majority of intervals (75%) were characterized as dominantly sandy till with varying quantities of gravels and silts. A few intervals were dominated by gravels (21 feet). Dominantly silt intercepts were unusual (two intervals totaling 5 feet). The total intersection of peaty materials was 25 feet.

The main feature of the overburden profile was the presence of oxidized (brown) and unoxidized (olive and grey) tills corresponding roughly to the presence of the water table. Of the thirteen mechanically-drilled holes, only two were in a fully unsaturated profile (holes 10 and 14) while the others were either completely saturated (four holes) or were unsaturated near surface and saturated below the saturated elevations (seven holes). Measurements of oxidation reduction potential (ORP) in field rinse tests showed a strong negative correlation with depth. Near surface samples had typical ORPs of 100 to 300 mV, whereas deeper samples had ORPs below 100 mV and as low as -200 mV. Loggers recorded the presence of what appeared to be secondary iron sulfides in the chemically-

[1] SRK Consulting, PolyMet Mining and Barr Engineering. 2008. Overburden Geochemical Characterization Plan in Support of EIS – DRAFT NorthMet Project. February 22, 2008.
[2] SRK Consulting, PolyMet Mining and Barr Engineering. 2008. Analysis of Samples from Overburden Drilling Program, NorthMet Project – DRAFT. March 18, 2008.

reduced overburden. Visual observations were supported by the smelled evolution of hydrogen sulfide gas when 10% hydrochloric acid was applied.

Surface tills appeared to be weakly acidic (rinse pHs less than 6.5) as shown by the correlation of rinse pH with depth. Deeper tills had rinse pHs greater than 6. The presence of acidic conditions generally did not correlate with conductivity) indicating that the variation of rinse pH was not significantly related to the presence of acidic salts as would be produced by oxidation of sulfide minerals. In fact, conductivities for samples showing rinse pHs less than 5.5 (the typical pH of deionized water) were mostly low. The exceptions were two samples with conductivity above 100 µS/cm and pHs below 5.6. These measurements did not correspond to the presence of mineralized rock.

As described below, some samples of the deeper tills became acidic prior to laboratory testing.

Other than brown coatings related to weathering of iron-bearing components of the overburden, chemical precipitates were uncommon. White cement and lenses were observed in drill hole 10, but they did not react with dilute hydrochloric acid.

The overburden rarely reacted with hydrochloric acid which indicated low concentrations of carbonate minerals.

3 Sample Selection and Analysis

A discussion of sample selection, as well as a sample analysis list, was provided in the analytical plan. The analytical plan was completed for the size fractions finer than 2 mm (-2 mm+74 µm and -74 µm), and the meteoric water mobility procedure on splits of the whole samples. The pebble counts are in progress (+4 mm size fraction) and once completed the pebbles will be re-combined with the -4+2 mm fraction for acid-base accounting and metal analysis on the +2 mm fraction.

The laboratory reported that 36 of the 37 samples submitted for analysis had sufficient pebbles (more than 180) for counting.

4 Results

4.1 Data Assessment Approach

The Geochemical Characterization Plan was designed to address various factors that could influence the geochemical characteristics of overburden, which included:

- Geological sources distant from the project area where the glacial ice originated.
- Underlying bedrock geology.
- Distance from mineralized bedrock (mainly Unit 1 of the Duluth Complex and the Virginia Formation).
- Glacial and periglacial deposition environment.
- Groundwater level during deposition.
- Current position of the water table (degree of saturation).

Field observations during drilling indicated that oxidation-reduction conditions in the overburden varied significantly and potentially exerted an important control on leaching potential of the overburden. This factor was also considered in the analysis.

The following sections provide descriptive statistics and discussion with respect to these factors.

The complete dataset is attached as Appendix A. Metal analysis was performed using two acid digestion methods (aqua regia and four acid) both followed by determination of concentrations using ICP. The difference between results for the two analytical methods was insignificant though in the case of nickel slightly higher concentrations were reported for the stronger four acid digestion (Figure 1) probably reflecting the more complete digestion of silicate minerals by this digestion. Therefore, subsequent review of the data considered results from the four-acid digestion.

The summaries below focus on the main parameters believed to be indicative of the influence of mineralized bedrock on the overburden (sulfur, copper and nickel).

4.2 Entire Dataset

4.2.1 Effect of Particle Size

Graphs comparing sulfur, copper and nickel concentrations in -74 μm and -2 mm+74 μm fractions are presented in Figure 2.

For sulfur, concentrations in the finer fraction were greater than in the coarse fraction for concentrations above 0.05% with the exception of two samples which showed nearly equivalent sulfur concentrations in the two fractions. Concentrations in the two fractions were not well correlated.

For copper, concentrations in the two fractions were correlated but were also mostly greater in the fine fraction. Nickel showed similar results except that at higher concentrations, the correlation was absent (Figure 2). Nickel concentrations were relatively stable below 200 ppm in the coarse fraction but increased to near 500 ppm for two samples. These samples also contained the highest sulfur concentrations in mineral overburden. The highest sulfur concentration in a peat (0.61%) did not contain elevated nickel concentrations.

The significant difference in sulfur concentrations for the size fractions may be linked to the observation of secondary sulfide minerals in the overburden rather than differences in the particle size distribution of primary sulfur in the overburden. There is no particular reason to expect sulfur to be concentrated in the fine fraction, and it would be expected that any fine-grained sulfur produced during deposition of the till would be oxidized following deposition. The base level sulfur concentrations in the coarser fraction (below 0.05%) are typical of background sulfur concentrations in rocks whereas the fine fraction concentrations indicated enrichment of sulfur.

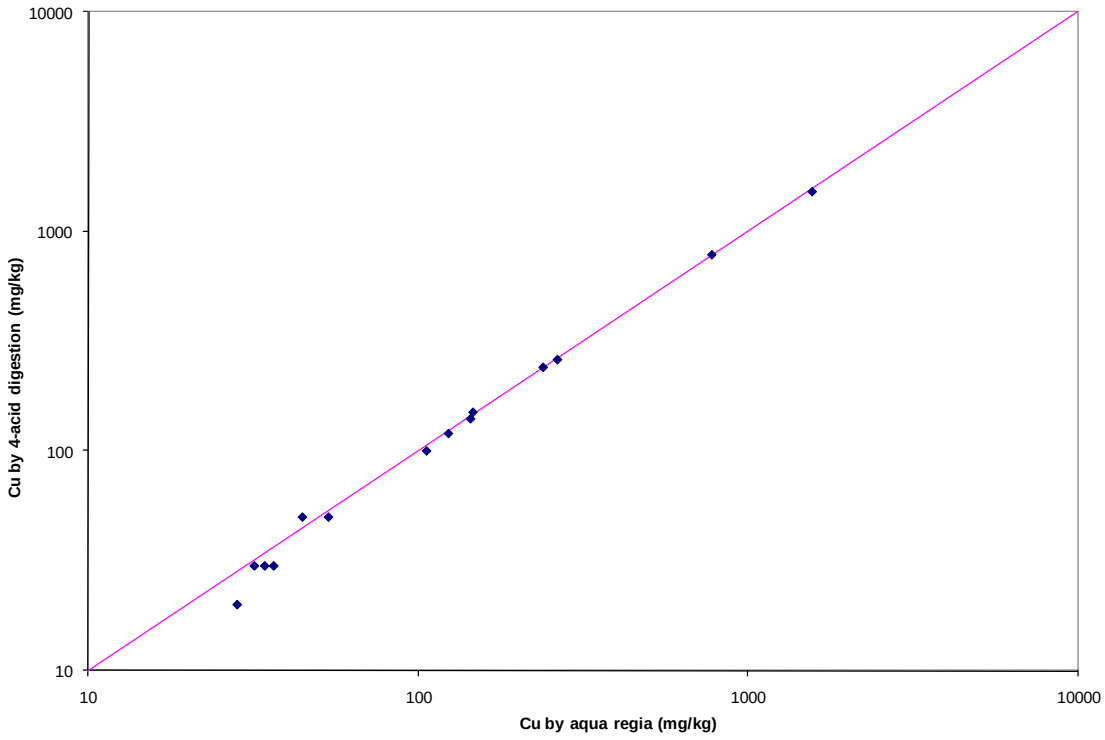
4.2.2 Overall Dataset Distribution

Distributions for the selected parameters are presented in Table 1. Data for many parameters were strongly positively skewed hence the median provides an indication of central or typical values while the 95th percentile and maximum values indicate extreme values.

Measurements of oxidation reduction potential (ORP) had 5th and 95th percentile values of -90 and 290, respectively. As was discussed in the analytical plan, ORP showed a strong negative correlation with depth. Near surface samples had typical ORPs of 100 to 300 mV, whereas deeper samples had ORPs below 100 mV and as low as -200 mV.

The statistics indicate that sulfate, which would likely be formed by oxidation of sulfur occurred at very low concentrations. Sulfur is mainly expected to occur as sulfide with the exception of peat for which sulfur may also occur in organic form.

Neutralization potentials were low and carbonate was rarely detected (Appendix A) indicating that rapidly acid consuming minerals were present at low concentrations.



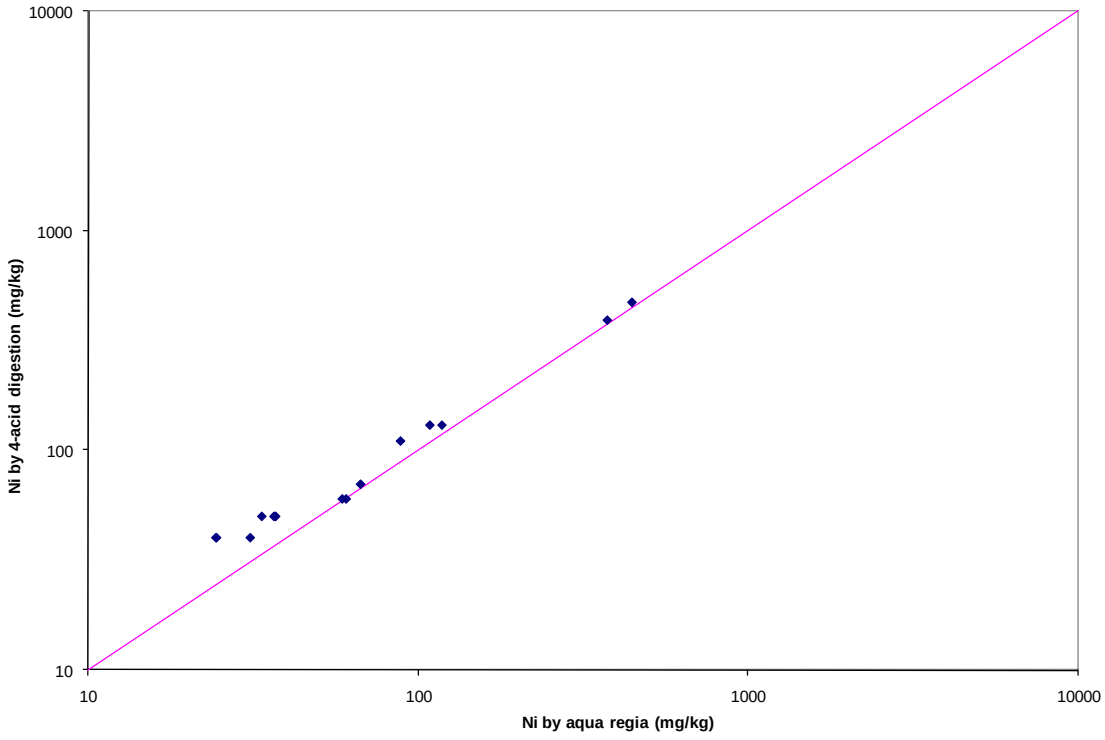
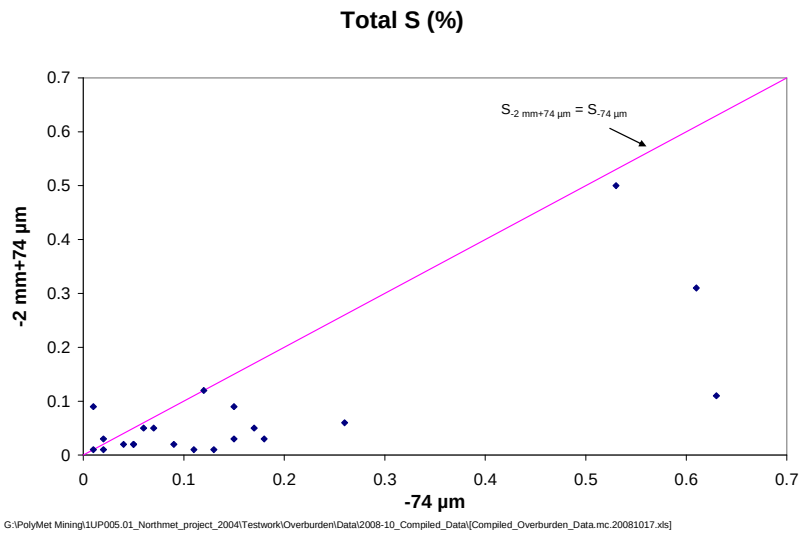
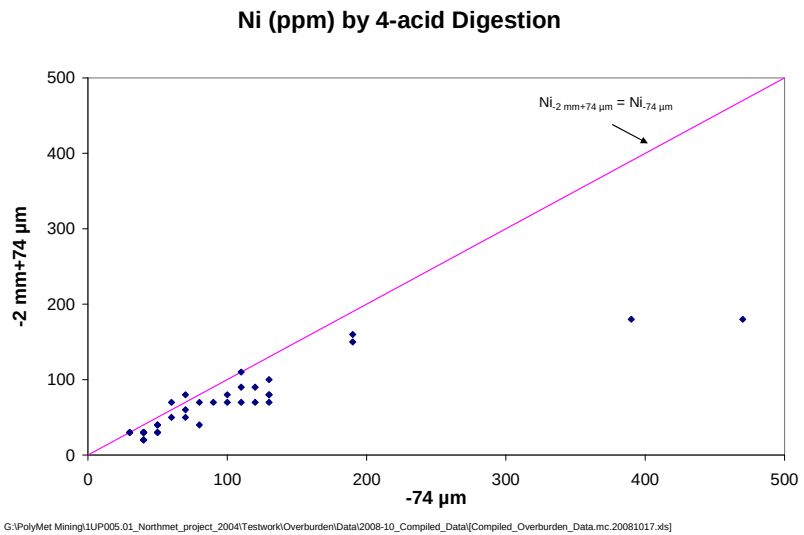
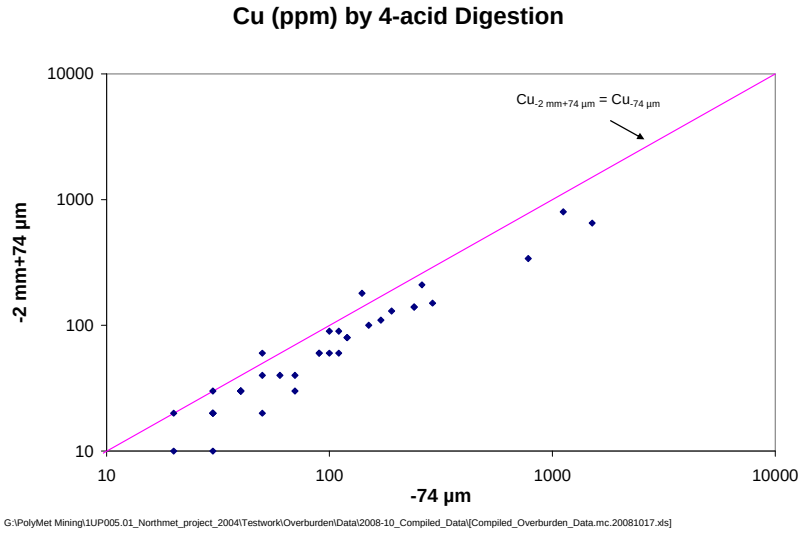


Figure 1. Comparison of Copper and Nickel Concentrations by Different Analytical Methods (-74 µm fraction)







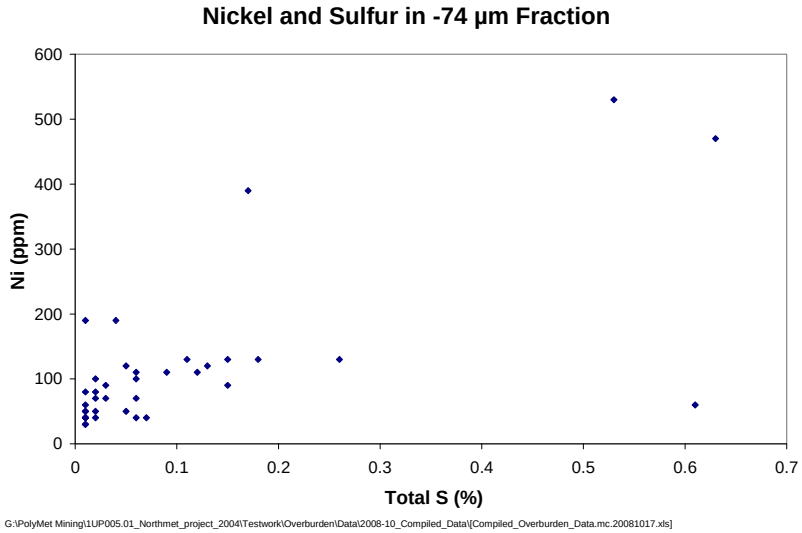


Figure 2. Comparison of Sulfur, Copper and Nickel Concentrations in Two Particle Size Fractions

Table 1. Statistics for Selected Overburden Characteristics

Parameter	Unit	Fraction	n	Minimum	P5	P50	P95	Maximum
pH[1]		-5 mm	44	4.8	5.2	6.4	6.5	8.7
ORP[1]	mV	-5 mm	43	-209	-90	98	95	287
Total Sulfur[2]	% S	-74 µm	37	0.01	0.01	0.05	0.10	0.55
Total Sulfur[2]	% S	-2 mm+74 µm	37	<0.01	<0.01	0.01	0.05	0.16
Sulfate[3]	% S	-74 µm	36	<0.01	<0.01	0.01	0.02	0.04
Sulfate[3]	% S	-2 mm+74 µm	37	<0.01	<0.01	<0.01	0.01	0.03
Cu[4]	ppm	-74 µm	37	20	28	90	180	848
Cu[4]	ppm	-2 mm+74 µm	37	10	18	60	109	402
Ni[4]	ppm	-74 µm	37	30	38	80	113	406
Ni[4]	ppm	-2 mm+74 µm	37	20	20	70	76	180
pH[5]		Whole	14	3.4	3.6	7.1	6.5	7.8
Sulfate[5]	mg/L	Whole	14	1.74	2.8	32	69	203
Cu[5]	mg/L	Whole	14	0.003	0.004	0.009	0.06	0.28
Ni[5]	mg/L	Whole	14	0.0008	0.001	0.01	0.25	1.3

General Method: [1] Field rinse test
[2] Leco Furnace
[3] HCl soluble, Sobek *et al.* 1978
[4] Four acid digestion
[5] MWMP leachable

Median values for sulfur, copper and nickel were comparable to global crustal values with slight enrichment for copper whereas the extreme values are consistent with proximity to sulfide mineralized rock.

MWMP leachate pH for 14 samples was strongly skewed with a median of 7.1, and the 5th and 95th percentile pH values were 3.6 and 7.8, respectively. Two samples had acidic pHs below 4, and two had pHs between 5 and 6. The two samples with lowest pHs were not acidic when tested in the field but became acidic between sample collection and analysis. As described subsequently, this appears to be due to oxidation of secondary sulfide minerals formed under saturated and chemically reducing conditions in the overburden.

In the majority of samples, the dominant anion in the leachate was sulfate, followed by alkalinity and chloride. Major cation concentrations were variable with all four major ions (calcium, magnesium, sodium and potassium) contributing to the ion balance (Figure 3).

Trace metal concentrations in the leachates were correlated to pH with the highest concentrations at lowest pHs. The correlation was apparent for Al, As, Ba, Cd, Co, Cr, Cu, Fe, Pb, Ni and Zn. Copper and nickel are shown in Figure 4.

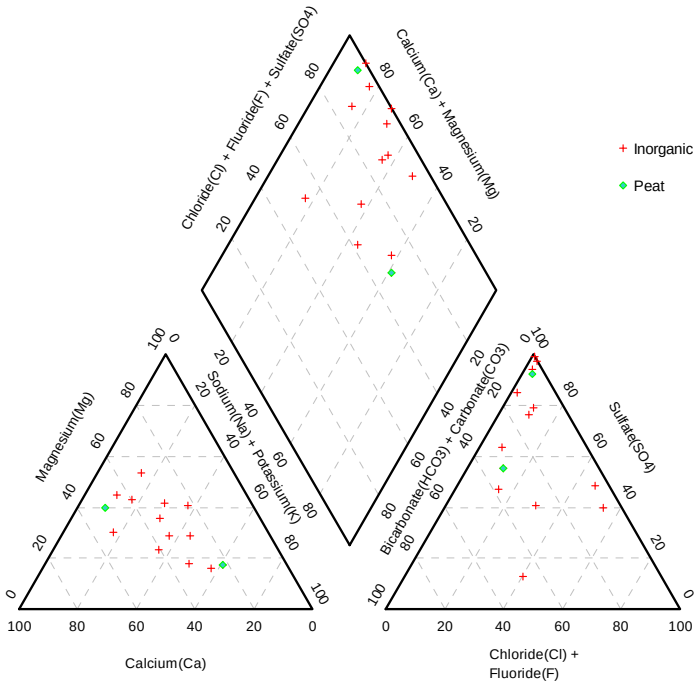


Figure 3. Piper Diagram for MWMP Leachates

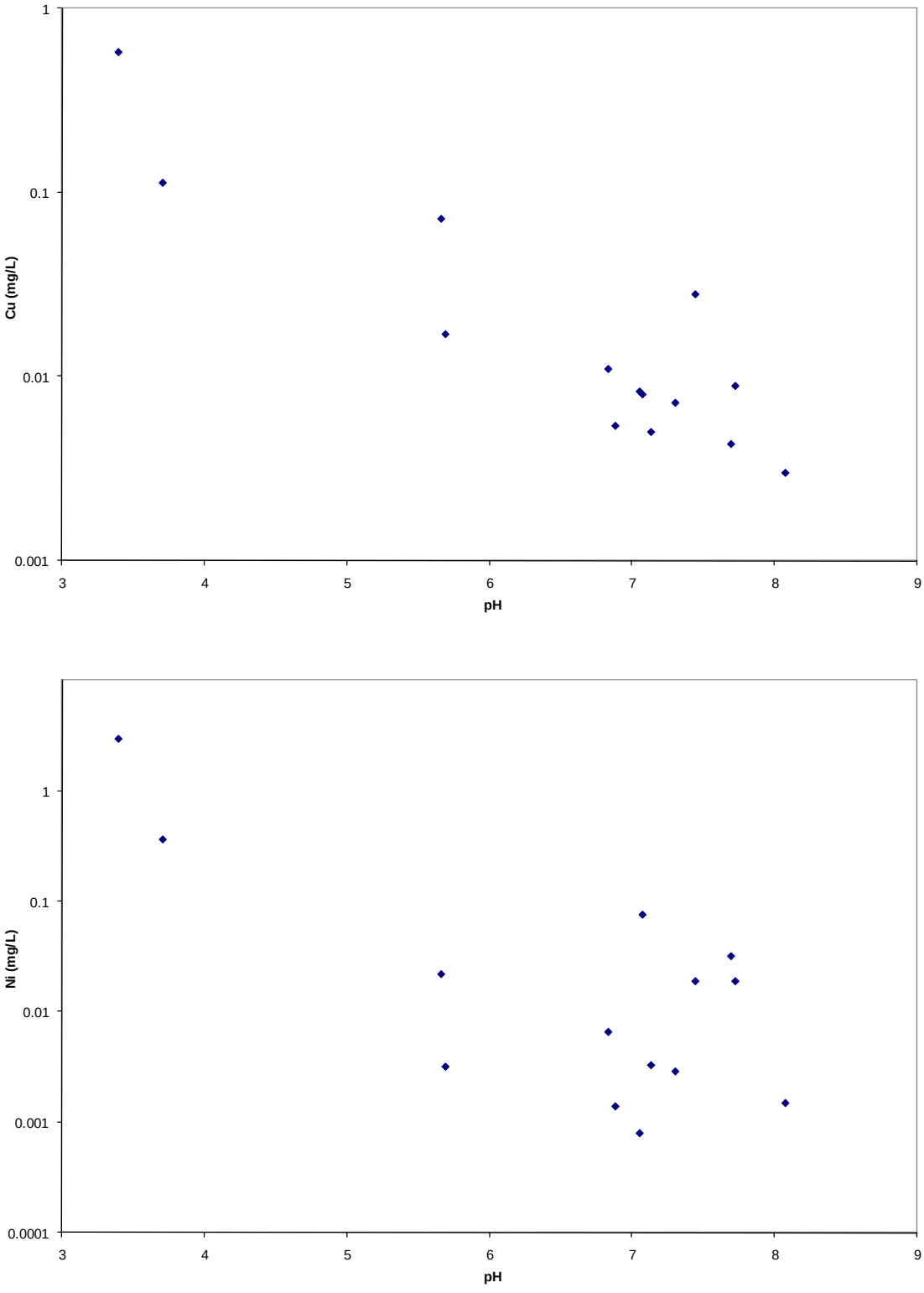


Figure 4. Nickel and Copper Compared to pH for MWMP Leachates

4.3 Geological Sources Up-Ice of the Project Area

Borehole RS-14B was underlain by Virginia Formation in the assumed direction up-ice of the Duluth Complex. The hole encountered bedrock at 5 feet.

The soil sample (peat) collected at RS-14B had elevated sulfur content (0.15%), a typical copper concentration (87 ppm) and a slightly low nickel concentration (59 ppm) when compared to the whole dataset.

Compared to the whole dataset, the upper till sample collected at this location had a low rinse pH (5.41), and slightly higher than typical sulfur, copper and nickel concentrations (0.09%, 190 ppm and 110 ppm, respectively).

4.4 Underlying Bedrock Geology

Table 2 presents comparative statistics for mineral overburden samples overlying Virginia Formation, Unit 1 Troctolite, and "Other" (Units above 1) Troctolite bedrock (5, 12, and 20 samples, respectively). Figure 4 presents plots of metal content versus a ratio of depth to depth to bedrock. The plots indicate that depth is of little significance to metal concentrations.

Median sulfur concentrations were greatest in samples overlying Virginia Formation bedrock (0.13%), followed by Unit 1 Troctolite (0.06%) and Other Troctolite (0.02%). Extreme values, however, appeared to be unrelated to bedrock type.

Median and extreme copper and nickel concentrations were greatest in samples overlying Unit 1 Troctolite.

Leachate from the R14B sample overlying Virginia Formation bedrock had one of the lowest pH values (3.7) and much higher than typical sulfate, copper, and nickel concentrations (188, 0.11, and 0.36 mg/L, respectively).

MWMP leachate pHs for six samples overlying Unit 1 Troctolite bedrock were near neutral (7. 1 to 7.7) in all but one sample which had a pH of 5.7 (Table 3). The median and 95th percentile sulfate concentrations were 32 and 149 mg/L, respectively.

Leachate pH for seven samples overlying Other Troctolite bedrock varied from 3.4 to 8.1 (with a median value of 6.9) resulting in a wide range of leachable sulfate and metal concentrations.

Overall, the data appeared to indicate bedrock geology exerts a subtle control on the sulfur content of the overburden (i.e. the comparatively elevated median concentrations for samples overlying the Virginia Formation and Unit 1) but that higher sulfur concentrations are not linked to bedrock.

4.5 Distance from Mineralized Bedrock

Figure 5 presents the spatial distribution of weighted average sulfur, copper and nickel concentrations in the main stratigraphic horizons for each hole to evaluate down-ice movement of potentially metal-rich bedrock from the mineralized Virginia Formation and Unit 1 of the Duluth Complex. In general, it is expected that these effects might be different for the stratigraphic units.

As noted in the Section 4.4, expected differences in bedrock (range of sulfur concentrations) is apparent in the overburden. This implies that ice movement has not eliminated the effect of bedrock but it is apparent from the range of sulfur concentrations in the overburden that other factors have resulted in variation in sulfur concentrations which may in part be attributed to movement of sulfur-bearing bedrock down ice from their sources.

Table 2. Summarized Statistics for Various Data Groupings (-74 µm fraction)

	Total Sulfur (%)						Copper (ppm)						Nickel (ppm)					
	n	min	P5	P50	P95	max	n	min	P5	P50	P95	max	n	min	P5	P50	P95	max
Virginia Formation	5	0.02	0.03	0.13	0.24	0.26	5	40	54	120	230	240	5	80	82	110	128	130
Unit 1 Troctolite	12	0.01	0.01	0.06	0.33	0.53	12	30	30	110	933	1120	12	40	40	105	453	530
Other Troctolite	20	0.01	0.01	0.02	0.61	0.63	21	20	20	60	290	1510	21	30	30	60	190	470
Peat	3	0.01	0.02	0.15	0.56	0.61	3	20	35	170	251	260	3	40	42	60	123	130
Soil	7	0.02	0.02	0.06	0.14	0.15	7	30	33	110	213	240	7	50	56	90	127	130
Outwash	1			0.63			1			1510			1			470		
Upper Till	17	0.01	0.01	0.01	0.11	0.18	18	20	29	50	205	290	18	30	30	50	139	190
Lower Till	7	0.04	0.04	0.07	0.23	0.17	7	40	55	110	618	780	7	40	58	120	330	390
Saturated Mineral	16	0.01	0.01	0.06	0.56	0.63	17	30	30	60	1198	1510	17	30	38	110	482	530
Unsaturated Mineral	11	0.01	0.01	0.02	0.11	0.13	11	20	25	70	155	190	11	30	35	70	115	120

Table 3. Summarized Statistics for MWMP Tests

	Leachate pH						Sulfate (mg/L)						Copper (mg/L)						Nickel (mg/L)					
	n	min	P5	P50	P95	max	n	min	P5	P50	P95	max	n	min	P5	P50	P95	max	n	min	P5	P50	P95	max
Virginia Formation	1			3.7			1			188			1			0.11			1			0.36		
Unit 1 Troctolite	6	5.7	6.0	7.3	7.7	7.7	6	1.7	3.4	32	149	166	6	0.0043	0.0052	0.0086	0.061	0.072	6	0.00080	0.0054	0.021	0.065	0.076
Other Troctolite	7	3.4	4.1	6.9	7.8	8.1	7	3.4	3.8	21	189	230	7	0.0030	0.0036	0.0072	0.41	0.58	7	0.0014	0.0014	0.0032	2.1	3.0
Peat	2	6.8	6.9	7.5	8.0	8.1	2	68	70	81	92	93	2	0.0030	0.0034	0.0070	0.011	0.011	2	0.0015	0.0018	0.0041	0.0063	0.0066
Soil	2	5.7	5.7	6.3	6.8	6.9	2	3.4	4.4	14	23	24	2	0.0054	0.0087	0.039	0.069	0.072	2	0.0014	0.0024	0.012	0.021	0.022
Outwash	1			3.4			1			230			1			0.58			1			3.0		
Upper Till	6	5.7	6.0	7.2	7.7	7.7	6	1.7	2.5	13	35	40	6	0.0050	0.0056	0.0086	0.025	0.028	6	0.00080	0.0013	0.0033	0.019	0.019
Lower Till	3	3.7	4.0	7.1	7.6	7.7	3	97	104	166	186	188	3	0.0043	0.0047	0.0080	0.10	0.11	3	0.032	0.0364	0.076	0.34	0.36
Saturated Inorganics	6	3.4	4.0	7.3	7.7	7.7	6	4.8	5.7	69	214	230	6	0.0043	0.0052	0.013	0.44	0.58	6	0.0032	0.0072	0.026	2.2	3.0
Unsaturated Inorganics	3	6.9	6.9	7.1	7.1	7.1	3	1.7	1.9	3.4	15	17	3	0.0050	0.0050	0.0054	0.0080	0.0083	3	0.00080	0.00086	0.0014	0.0031	0.0033

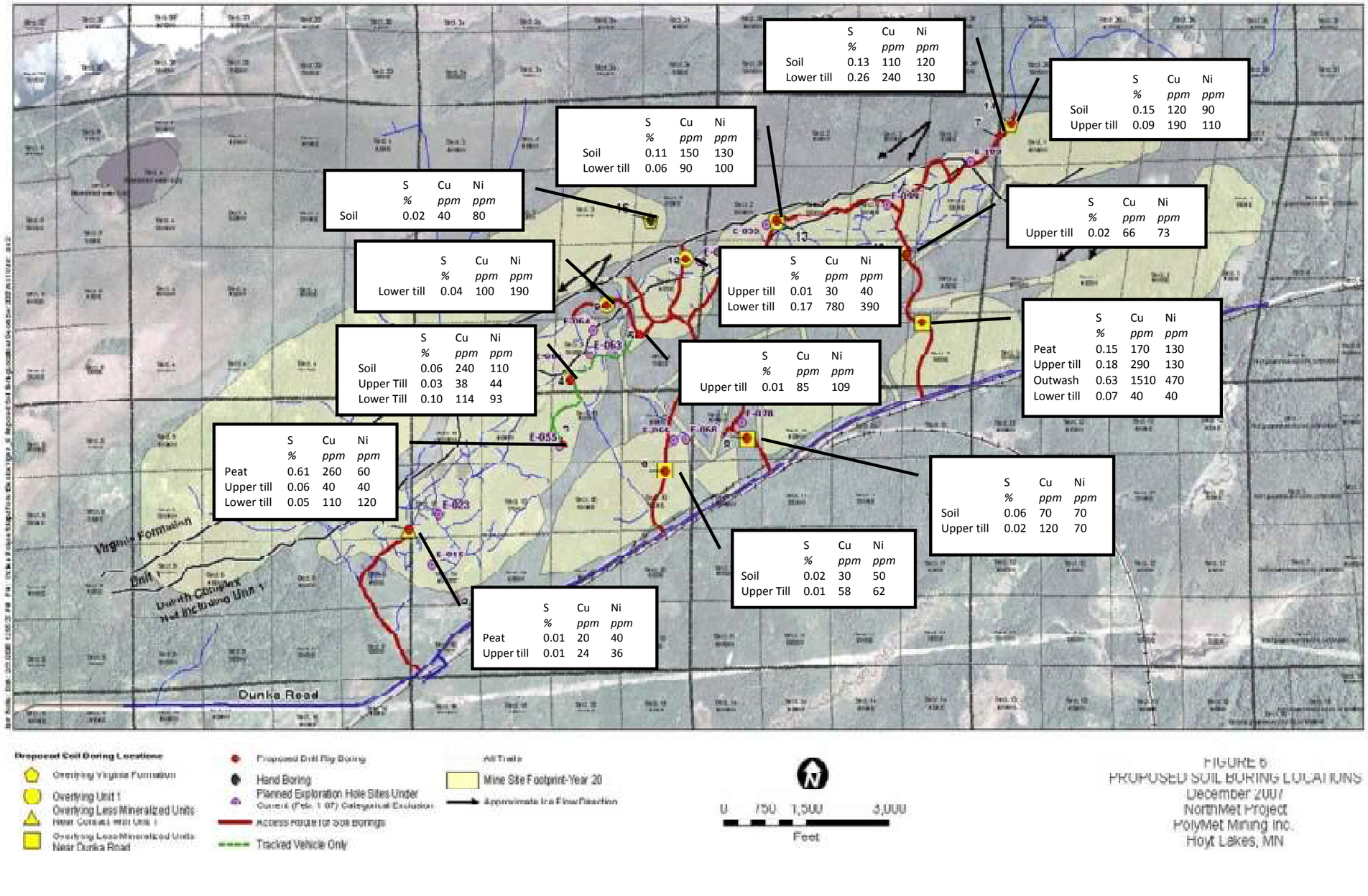


Figure 5. Weighted average sulfur, copper, and nickel concentrations (-74 µm fraction)

4.6 Glacial and Periglacial Deposition environment

Different depositional environments are indicated by stratigraphic units in the overburden. Barr Engineering has interpreted two main glacial units (Upper and Lower Till). Outwash was intersected in one hole. Peat deposits are common in the project area and were specifically characterized. Thin mineral soils are also present although in some cases textural these were described as peat. In the comparative statistics in Table 2, samples indicated as "soil" reflect both peat and mineral soils.

The three samples classified specifically as "peat" had the highest median total sulfur and nickel concentrations. MWMP leachates for the 2 peat samples tested had circum-neutral pH. One sample (from borehole RS-01B) had a typical sulfate concentration, and low copper and nickel concentrations. The second sample (from borehole RS-03) had slightly higher than typical sulfate concentrations and typical copper and nickel concentrations.

Comparatively, Lower Till samples had higher median sulfur, copper and nickel concentrations than the Upper Till unit. MWMP leachate from one of the Lower Till samples (borehole RS-07) had one of the lowest pH values (3.7) and much higher than typical sulfate, copper, and nickel concentrations (188, 0.11, and 0.36 mg/L, respectively) (Table 3). Leachate from the other two Lower Till samples had circum-neutral pH and higher than typical sulfate, low to typical copper, and high nickel concentrations. MWMP leachates from six upper till samples had low to typical sulfate, copper, and nickel concentrations.

The one intersection of "outwash" had the highest sulfur, copper and nickel concentrations in the database and leached elevated sulphate and low pH in the MWMP (Table 3).

4.7 Groundwater Level during Deposition

The groundwater level during deposition of the glacial tills could not be evaluated and was not considered further.

4.8 Degree of Saturation

Table 2 presents comparative statistics of total sulfur, copper and nickel concentrations for saturated and unsaturated mineral overburden. Peat is by definition saturated and is considered in Section 4.6.

When comparing degree of saturation in mineral overburden, sulfur concentrations were greater in the saturated overburden (5^{th}, 50^{th}, and 95^{th} percentile concentrations of 0.01, 0.06, and 0.56%, respectively). Unsaturated mineral overburden had 5^{th}, 50^{th}, and 95^{th} percentile concentrations of 0.01, 0.02, and 0.11% respectively.

Differences in metal concentrations were less apparent than for sulfur. Median copper concentrations were similar regardless of saturation though much higher copper concentrations were apparent at the extremes for saturated overburden. Similarly median nickel concentrations were only slightly greater in the saturated overburden though the difference was much greater at the extreme.

4.9 Effect of Oxidation-Reduction Conditions

As discussed in Section 2, wide variations in oxidation-reduction conditions were observed in the overburden as indicated by ORP determinations in field rinse tests. ORP in general decreased with increasing depth implying a correlation with degree of saturation. In addition, indications of secondary sulfide minerals were recorded throughout the survey area in six drill holes, though the implied presence of secondary sulfide minerals did not always correlate with low ORP or necessarily elevated sulfur concentrations.

As noted previously, sulfur in general appeared to be more enriched in the fine fraction. The latter would not generally be expected if the sulfides originated from bedrock materials (or at least sulfur concentrations would be the same in the fine and coarse fractions).

In general, sulfur concentrations showed a trend to increasing concentrations as ORP decreased (Figure 6) with one or two extreme values in boreholes RS11 and RS13. The sample from borehole RS13 indicated the presence of mineralized rock under saturated and reducing conditions (ORP-69 mV) and therefore the elevated sulfur concentrations in the fines may originate from bedrock rather than secondary sulfide precipitation although the presence of sulfur in the bedrock could contribute to conditions that produce secondary sulfides.

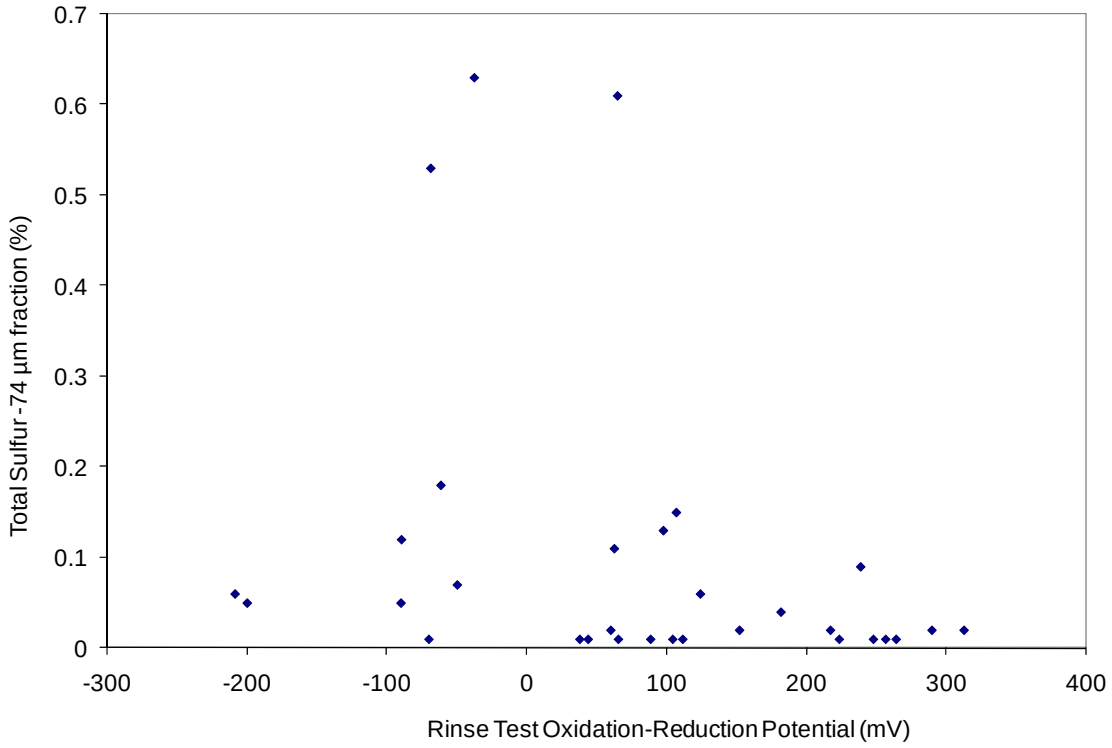


Figure 6. Total Sulfur Compared to Rinse ORP

The two lowest pHs indicated by MWMP tests were on samples that had indications of secondary sulfides (boreholes RS-07 and RS-11) (Table 3). These samples also yielded the highest leachable sulfate (188 and 230 mg/L) and metal concentrations. Other samples yielded non-acidic pHs and intermediate sulfate concentrations compared to samples that did not have secondary sulfides.

4.10 Conclusions

The data appear to indicate a vertical rather than a strong lateral variation in metal and sulfur concentrations in mineral overburden. The vertical effects appear as relatively higher sulfur and metal concentrations in the lower till rather than upper till units, and higher sulfur concentrations in chemically-reduced overburden. The two findings are not independent, however, because the lower till occurs deeper in the overburden profile where the overburden is saturated and chemically reducing conditions are observed.

Bedrock also exerts some control on sulfur and metal concentrations as indicated by relatively elevated though comparatively low median concentrations in overburden overlying the Virginia Formation and Unit 1 of the Duluth Complex, which is consistent with the sulfur content of the bedrock. However, elevated sulfur concentrations occur in the overburden overlying the other units of the Duluth Complex due to the vertical zoning effects.

5 Interpretation

5.1 Possible Explanation for Vertical Zoning

Sporadic elevated sulfur concentrations and the potential for leaching of acidity and metals from the overburden appears to be related to the development of vertical zoning caused by saturated and chemically-reducing conditions. Precipitation of secondary sulfides appears to have occurred in the overburden. The conditions necessary for this to occur include the presence of organic matter to act as a reductant and a source of sulfur to provide sulfide. The presence of peat in the project area indicates that dissolved organic matter may be entering groundwater. Sulfur most likely originates from reduction of sulfate to sulfide with the sulfate originating from the mineralized bedrock.

5.2 Management of Overburden

The data indicate the need to consider selectively managing the deeper saturated overburden to address oxidation of secondary sulfides when exposed to atmospheric conditions during excavation. The current program indicates that secondary sulfides may occur throughout the project area and therefore that currently all deeper overburden should be managed to address the potential for leaching. Management of deep overburden separately would also address exposure of overburden containing mineralized rock overlying Virginia Formation and Unit 1 Troctolite.

In contrast, near surface unsaturated till contains relatively low sulfur and metal concentrations and leached low concentrations of metals.

Two management units are therefore recommended:

- Peat and unsaturated mineral overburden.
- Saturated mineral overburden.

5.3 Water Chemistry

Contact water chemistry estimates have been specifically requested for the mineral overburden units in the excavation areas and stockpiles.

The analysis of MWMP leachates provides an indicator of chemistry when stormwater comes into contact with exposed surfaces for example in excavations. In these situations, contact time is limited and the contact ratio (water to solid) is relatively high. Table 4 summarizes concentrations at the median, 95th percentile and maximum concentrations[3]. Generally, the 95th percentile values are recommended for exposed excavation faces to conservatively represent storm water since these are likely to represent the longest contact time with most soluble materials.

Due to the lack of mineralization in the unsaturated overburden, contact water chemistry is expected to be dilute and similar to existing runoff in the project area. Qualitatively, the MWMP tests confirmed that these materials produce dilute contact waters containing low concentrations of major and trace elements.

Components of the saturated mineral overburden are expected to contain elevated sulfur concentrations and may produce acidic leachate when exposed. The current testwork showed that only samples containing more than 0.2% sulfur produced acidic leachate between recovery by the drill program and testing in the laboratory; however, it is possible that with further exposure, other samples containing lower sulfur concentrations could produce acidity. Also, the extent of elevated sulfur concentrations in the saturated overburden is not known. The acidic pHs represented by the

[3] Prior to complete interpretation of the data, the data were sorted using slightly different groups and provided to Barr Engineering to support RS74 (Draft 02). This version is attached as Appendix B for reference.

95th and maximum values should therefore be applied to the component of water that comes into contact with these materials during excavation and exposure to oxidation.

Application of MWMP results to estimation of seepage chemistry from the overburden stockpile needs to consider the disposal conditions and opportunity for water to contact the materials. Based on the planned construction of the overburden stockpile by compaction to limit infiltration and air entry, the use of 95th percentile contact chemistry for the saturated overburden is not warranted. It is recommended that the median values for the saturated overburden be used as an estimate.

Table 4. Summary of MWMP Leachate Results By Material Type

Material	pH	Alk mg/L	F mg/L	Cl mg/L	SO_4 mg/L	Al mg/L	Sb mg/L	As mg/L	Ba mg/L	Be mg/L	B mg/L	Cd mg/L	Ca mg/L	Cr mg/L	Co mg/L	Cu mg/L	Fe mg/L	Pb mg/L	Mg mg/L	Mn mg/L	Mo mg/L	Ni mg/L	Se mg/L	Ag mg/L	Na mg/L	Te mg/L	Tl mg/L	V mg/L	Zn mg/L
PEAT																													
P50	7.5	46	0.59	5.9	81	0.086	0.0006	0.0037	0.023	<0.0002	0.21	<0.00004	19	0.0004	0.0003	0.007	0.07	0.00009	9.3	0.13	0.019	0.0041	0.00075	0.00068	26	<0.0002	0.000045	0.0034	0.0015
P95	6.9	79	1	8.8	92	0.13	0.00069	0.0043	0.034	<0.0002	0.23	<0.00004	23	0.00094	0.00066	0.011	0.12	0.00022	11	0.19	0.028	0.0063	0.00089	0.0013	45	<0.0002	0.0001	0.0042	0.0038
max	6.8	83	1.1	9.2	93	0.13	0.0007	0.0044	0.035	<0.0002	0.23	<0.00004	23	0.001	0.0007	0.011	0.12	0.00023	11	0.19	0.029	0.0066	0.0009	0.0014	47	<0.0002	0.00011	0.0043	0.004
SATURATED MINERAL OVERBURDEN																													
P50	7.3	13	0.33	2	69	0.14	0.0004	0.0023	0.011	<0.0002	0.027	0.000015	13	0.00005	0.0015	0.013	0.11	<0.00005	11	0.18	0.029	0.026	0.002	<0.00005	6.5	<0.0002	<0.00002	0.0011	0.003
P95	4	36	0.56	3.8	210	0.63	0.0012	0.0028	0.026	0.00055	0.087	0.005	26	0.0012	0.23	0.44	5.5	0.0011	18	1.1	0.034	2.2	0.0034	<0.00005	13	<0.0002	0.000025	0.0022	0.86
Max	3.4	38	0.6	4	230	0.74	0.0012	0.0028	0.028	0.0008	0.098	0.0066	27	0.0013	0.31	0.58	7.3	0.0014	18	1.3	0.034	3	0.0037	<0.00005	13	<0.0002	0.00004	0.0024	1.2
UNSATURATED MINERAL OVERBURDEN																													
P50	7.1	5	0.18	1.9	3.4	0.091	<0.0001	0.0005	0.0035	<0.0002	0.013	0.00005	3.9	<0.0002	0.0006	0.0054	0.05	<0.00005	2	0.051	0.0039	0.0014	<0.0002	<0.00005	3.7	<0.0002	<0.00002	0.0005	0.002
P95	6.9	12	0.45	3.4	15	0.3	0.00098	0.0029	0.013	<0.0002	0.028	0.00015	5.7	0.00097	0.0015	0.008	0.059	<0.00005	2.1	0.1	0.013	0.0031	0.00052	<0.00005	4.2	<0.0002	0.000025	0.00059	0.0056
max	6.9	13	0.48	3.6	17	0.32	0.0011	0.0032	0.014	<0.0002	0.03	0.00016	5.9	0.0011	0.0016	0.0083	0.06	<0.00005	2.1	0.11	0.014	0.0033	0.0006	<0.00005	4.3	<0.0002	0.00003	0.0006	0.006

Appendix A
Overburden Characterization Data

Fraction Analyzed			Whole Sample	Whole Sample	Whole Sample	Whole Sample	Whole Sample	Whole Sample	-5 mm	-5 mm	-5 mm	Whole Sample	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm
Parameter	**Depth**	**Depth**	**Stratigraphic Unit**	**Saturated**	**Oxidized**	**Texture**	**Sulfides**	**Other**	**pH**	**Specific Conductivity**	**ORP**	**Moisture Content**	**Total C**	**CO_2**	**Total Sulphur**	**Sulphate**	**MPA**
Units	From (ft)	To (ft)								(µS/cm)	mV	%	% C	% CO2	% S	% S	kgCaCO3/t ore
Detection Limit													0.01	0.2	0.01	0.01	0.3
General Method			Field Observation	Field Observation	Field Observation	Field Observation	Field Observation	Field Observation	Rinse Test	Rinse Test	Rinse Test		ABA	ABA	ABA	ABA	ABA
Analytical Method Code													C-IR07	C-GAS05	S-IR08	S-GRA06a	OA-VOL08
RS-01B	0	1	Peat			Peat			7.05	24	248.1	39		<0.2	0.01	<0.01	2.8
RS-01B	1	5	Upper till	N	Y	Sand			5.86	10	256.9	4.7		<0.2	0.01	0.01	0.3
RS-01B	14	15	Upper till	Y	Y	Sand			6.37	16	223.7	5.7		<0.2	0.01	0.01	<0.3
RS-01B	18	20	Upper till	Y	N	Sand			7.28	34	65.6	7.8		<0.2	0.01	<0.01	0.3
RS-01B	20	25	Lower till, Bedrock at 20.5ft						8.79	66	-40	9.1					
RS-03	5	10	Peat			Peat			5.17	116	65	65		NSS*	0.61	NSS*	NSS*
RS-03	15	20	Upper till	Y	N	Sand			7.4	50	-208.7	4.3		<0.2	0.06	0.01	1.9
RS-03	20	22	Lower till, Bedrock at 22ft	Y	N	Gravel			9.42	63	-200	5.8		0.3	0.05	0.02	1.6
RS-04	1	5	Soil	Y	Y	Silt			5.77	22	124.3	12		<0.2	0.06	0.01	1.9
RS-04	10	15	Upper till	Y	Y	Sand			6.33	25	104.5	3.8		<0.2	0.01	0.02	0.3
RS-04	15	20	Upper till (15-18ft), Lower till (18-25ft)	Y	N	Sand	Secondary		6.74	25	-90	7.4		0.2	0.05	<0.01	1.6
RS-04	20	25	Lower till, Bedrock at 25ft	Y	N	Sand	Secondary		7.83	17	-89.6	4.7		0.3	0.12	<0.01	3.8
RS-05A	5	10	Upper till	mixed	mixed	mixed			6.55	22	88.7	6.5		<0.2	0.01	0.04	0.3
RS-05A	10	13	Upper till, Bedrock at 13ft	Y	N	Gravel			8.9	88	-70	2.5		<0.2	0.01	0.01	0.3
RS-06A	0.5	2	Soil	N	Y	Sand			4.84	5	313	6.8		0.2	0.02	0.01	0.6
RS-06A	2	4	Upper till	N	Y	Sand			4.99	11	279	8.1					
RS-06A	5	7.5	Upper till	N	Y	Sand			5.82	12	264.4	9.5		<0.2	0.01	<0.01	0.3
RS-06A	7.5	10	Upper till	N	Y	Sand			6.32	17	251	4.3					
RS-06A	15	19	Upper till	Y	N	Sand	Secondary		6.75	18	38	9.5		<0.2	0.01	<0.01	0.3
RS-06A	19	21	Upper till, Bedrock at 21ft	Y	N	Sand	Secondary		7.86	20	18	9.1					
RS-07	1	2	Soil	N	Y	Sand			5.61	45	97.8	15		<0.2	0.13	<0.01	4.1
RS-07	2	3	Soil	N	Y	Sand						13					
RS-07	3	5	Upper till	N	Y	Sand			6.1	52	27	9.4					
RS-07	5	6	Upper till	N	Y	Sand			6.1	52	27	4.2					
RS-07	6	10	Lower till						6.4	17	60	5.8					
RS-07R	10	12	Lower till (10-11ft), Bedrock (11-12ft)									5.0					
RS-07R	13.5	14.5	Bedrock	Y	N	Gravel			7.48	82		7.0					
RS-07/RS-07R	6	14.5	Lower till											<0.2	0.26	0.01	8.1
RS-08A	0	1	Soil	N	N	Sand						15		<0.2	0.06	<0.01	1.9
RS-08A	1	5	Upper till	N	N	Sand			5.18	19	287.6	6.7					
RS-08A	5	11	Upper till	N	N	Sand	Secondary		5.78	22	217.4	6.4		<0.2	0.02	0.05	0.6
RS-09	7	8	Lower till, Bedrock at 8ft	Y	N	Silt	Secondary		5.88	2	182	11		<0.2	0.04	0.03	1.3
RS-10	1	2	Upper till	N	Y	Sand						12		<0.2	0.03	0.02	0.9
RS-10	2	3	Upper till	N	Y	Sand			6.07	30	193	9.8					
RS-10	3	5.5	Upper till	N	Y	Sand			5.73	12	241.6	9.5					
RS-10	5.5	7.5	Upper till					Cement	7.08	20	60.2	6.2		<0.2	0.02	0.01	0.6
RS-10	7.5	10	Upper till	N	N	Gravel			6.81	30	152.3	4.4		<0.2	0.02	<0.01	0.6
RS-10	10	14	Upper till	N	N	Gravel		Cement	6.5	26	145.3	28					
RS-11	0	9.5	Peat			Peat			5.89	40	107.1	33		<0.2	0.15	<0.01	4.7
RS-11	11.5	17	Upper till	Y	N	Sand	Secondary		6.47	47	-61.4	9.3		<0.2	0.18	<0.01	5.6
RS-11	17	25	Outwash	Y	N	Gravel	Secondary		6.56	30	-37.5	6.3		<0.2	0.63	0.02	19.7
RS-11	28	31	Lower till	Y	N	Sand			6.5	70	-49.7	5.9		0.4	0.07	<0.01	2.2
RS-11	31	33	Lower till, Bedrock at 33ft	Y	N	Sand						4.7					
RS-12	7	9	Upper till	N	Y	Sand		Mineral'd Rock	7.17	33	111.7	6.9		0.4	0.01	0.01	0.3
RS-12	16	18	Upper till	Y	Y	Sand			7.14	14	44	10		<0.2	0.01	0.01	0.3
RS-12	20	22	Lower till, Bedrock at 22ft	Y	N	Sand		Mineral'd Rock				5.3		<0.2	0.17	0.02	5.3
RS-13	0	1.5	Soil			peat			6.15	42	62.7	29		<0.2	0.11	0.02	3.4
RS-13	1.5	2.5	Lower till			Sand						15		<0.2	0.06	<0.01	1.9
RS-13	2.5	6	Lower till			Sand			6.07	27	106.6	5.2					
RS-13	8	10	Bedrock	Y	N	Gravel		Mineral'd Rock	7.2	46	-68.7	4.1		<0.2	0.53	0.05	16.6
RS-14B	0	1.5	Soil			Peat						26		<0.2	0.15	<0.01	4.7
RS-14B	1.5	3	Upper till	N	Y	Sand			5.41	19	239	16		<0.2	0.09	<0.01	2.8
RS-14B	3	5	Upper till, Bedrock at 5ft	N	Y	Sand						6.3					
RS-16B	0	2	Soil	N	Y	Sand			5.29	8	290	22		<0.2	0.02	<0.01	0.6

Fraction Analyzed			-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm
Parameter	**Depth**	**Depth**	**NP**	**Fizz Rating**	**Ag**	**Al**	**As**	**Au**	**B**	**Ba**	**Be**	**Bi**	**Ca**	**Cd**	**Ce**	**Co**	**Cr**	**Cs**	**Cu**
Units	From (ft)	To (ft)	kgCaCO3/t ore	Unity	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm
Detection Limit			1	1	0.01	0.01	0.1	0.2	10	10	0.05	0.01	0.01	0.01	0.02	0.1	1	0.05	0.2
General Method			ABA	ABA	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia
Analytical Method Code			OA-VOL08	OA-VOL08	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41
RS-01B	0	1	5	1	4.73	1.29	6.7	<0.2	<10	140	0.43	0.21	0.43	0.54	23.5	8	36	1.21	28.3
RS-01B	1	5	2	1	1.05	1.19	6	<0.2	<10	40	0.4	0.07	0.39	0.09	50.7	7.6	38	1.06	25.2
RS-01B	14	15	5	1	0.63	0.9	5.5	<0.2	<10	40	0.34	0.07	0.55	0.09	40.8	7	36	1.1	31.2
RS-01B	18	20	13	1	9.98	1.24	4.5	<0.2	<10	40	0.36	0.08	0.83	0.08	40.5	9.6	47	1	38.8
RS-01B	20	25																	
RS-03	5	10	NSS*	NSS*	0.54	0.82	5.3	<0.2	<10	60	0.41	0.1	1.19	0.45	30.2	5	46	0.58	264
RS-03	15	20	15	1	0.08	0.74	4.7	<0.2	<10	40	0.24	0.25	0.77	0.11	35.8	6.6	32	0.69	37.3
RS-03	20	22	20	1	5.76	1.79	2.2	<0.2	<10	50	0.23	0.27	1.23	0.13	33.5	22.6	41	1.14	120
RS-04	1	5	6	1	0.55	2.45	7.9	<0.2	<10	80	0.45	0.2	0.53	0.12	50.6	17	62	1.98	240
RS-04	10	15	7	1	1.22	0.8	2.6	<0.2	<10	40	0.25	0.06	0.59	0.08	47.4	8.2	37	0.71	36.5
RS-04	15	20	12	1	1.92	0.76	1.5	<0.2	<10	20	0.19	0.06	0.66	0.07	41.6	7.9	33	0.84	53.5
RS-04	20	25	13	1	1.87	1.01	3	<0.2	<10	30	0.28	0.07	0.75	0.12	36.5	14.9	37	1.17	144
RS-05A	5	10	17	1	0.08	1.64	5.8	<0.2	<10	70	0.43	0.14	0.5	0.12	54.4	14.5	57	1.87	106
RS-05A	10	13	6	1	0.03	4.47	1.9	<0.2	<10	50	0.23	0.05	2.43	0.1	17.95	38.5	80	0.76	59.4
RS-06A	0.5	2	0	1	0.17	2.16	4	<0.2	<10	90	0.48	0.09	0.33	0.1	52.4	12.3	46	1.6	31.9
RS-06A	2	4																	
RS-06A	5	7.5	6	1	0.54	1.46	4.6	<0.2	<10	60	0.41	0.08	0.53	0.05	50.3	9.6	50	1.16	42.1
RS-06A	7.5	10																	
RS-06A	15	19	6	1	0.25	1.05	6.8	<0.2	<10	50	0.32	0.07	0.58	0.1	50.4	11.7	43	1.08	44.6
RS-06A	19	21																	
RS-07	1	2	-2	1	0.04	3.04	9.6	<0.2	<10	110	0.81	0.16	0.31	0.11	40.9	14.7	101	3.75	115
RS-07	2	3																	
RS-07	3	5																	
RS-07	5	6																	
RS-07	6	10																	
RS-07R	10	12																	
RS-07R	13.5	14.5																	
RS-07/RS-07R	6	14.5	12	1	0.25	2.5	15.3	<0.2	<10	110	0.85	0.27	0.48	0.52	56.7	25.2	99	4.13	239
RS-08A	0	1	4	1	0.16	2.9	8.5	<0.2	<10	70	0.67	0.1	0.35	0.09	45.3	15	59	2.08	70.8
RS-08A	1	5																	
RS-08A	5	11	7	1	0.58	1.53	11.7	<0.2	<10	60	0.43	0.09	0.58	0.19	49.6	19.2	53	2.24	123.5
RS-09	7	8	23	1	0.07	4.83	5	<0.2	<10	80	0.54	0.08	2.41	0.17	35.1	34.2	89	2.41	107.5
RS-10	1	2	2	1	0.07	2.67	4	<0.2	<10	110	0.52	0.13	0.31	0.07	42.1	14	56	2.26	39.1
RS-10	2	3																	
RS-10	3	5.5																	
RS-10	5.5	7.5	9	1	0.05	1.88	4.8	<0.2	<10	90	0.49	0.12	0.55	0.06	57.6	12.7	68	3.52	53.4
RS-10	7.5	10	11	1	0.05	2.57	9.7	<0.2	<10	100	0.59	0.11	0.65	0.14	47.1	20.5	85	4.27	88.3
RS-10	10	14																	
RS-11	0	9.5	3	1	1.65	2.48	4.5	<0.2	<10	90	0.68	0.16	0.75	0.4	40.8	18.2	110	2.74	166.5
RS-11	11.5	17	14	1	0.24	2.62	10.7	<0.2	<10	100	0.89	0.25	0.68	0.31	54.9	21.5	107	3.62	307
RS-11	17	25	4	1	27.4	3.09	20.9	<0.2	<10	100	0.77	0.46	0.87	0.6	41.9	41.2	102	4.44	1560
RS-11	28	31	2	1	1.09	1.01	5	<0.2	<10	50	0.31	0.07	0.82	0.13	46.4	9.1	46	1.38	46.1
RS-11	31	33																	
RS-12	7	9	17	1	0.01	1.04	5.3	<0.2	<10	50	0.36	0.07	0.67	0.08	54	8.5	35	1.15	34.3
RS-12	16	18	11	1	0.01	0.85	5.6	<0.2	<10	40	0.36	0.09	0.47	0.13	53.1	7.6	33	1.02	32.2
RS-12	20	22	5	1	0.25	1.97	10	<0.2	<10	50	0.33	0.19	1.04	0.18	43.5	43.9	47	1.18	775
RS-13	0	1.5	19	1	0.11	2.47	4.4	<0.2	<10	60	0.54	0.08	0.61	0.18	35.3	21.9	63	1.22	146.5
RS-13	1.5	2.5	6	1	0.1	2.67	7.2	<0.2	<10	120	0.55	0.06	0.39	0.08	58.9	19.9	90	4.21	88.1
RS-13	2.5	6																	
RS-13	8	10	6	1	0.49	2.69	20.7	<0.2	<10	90	0.54	0.25	1.02	0.39	46.3	67.2	72	2.4	1155
RS-14B	0	1.5	1	1	0.17	2.37	5.4	<0.2	<10	100	0.44	0.14	0.28	0.2	23	12	51	1.63	86.5
RS-14B	1.5	3	3	1	0.22	3	6.4	<0.2	<10	80	0.71	0.11	0.33	0.13	45.6	19.2	63	2.29	180.5
RS-14B	3	5																	
RS-16B	0	2	0	1	0.1	3.71	7	<0.2	<10	90	0.75	0.09	0.24	0.15	42.5	16.7	67	1.41	29.4

Fraction Analyzed			-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm
Parameter	**Depth**	**Depth**	**Fe**	**Ga**	**Ge**	**Hf**	**Hg**	**In**	**K**	**La**	**Li**	**Mg**	**Mn**	**Mo**	**Na**	**Nb**	**Ni**	**P**	**Pb**
Units	From (ft)	To (ft)	%	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	%	ppm	ppm	%	ppm	ppm	ppm	ppm
Detection Limit			0.01	0.05	0.05	0.02	0.01	0.005	0.01	0.2	0.1	0.01	5	0.05	0.01	0.05	0.2	10	0.2
General Method			Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia
Analytical Method Code			ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41
RS-01B	0	1	2.9	6.01	<0.05	<0.02	0.08	0.024	0.09	9.6	6.7	0.21	1550	1.51	0.02	1.02	24.4	1120	21.8
RS-01B	1	5	3.33	4.02	0.09	0.12	0.02	0.013	0.07	20.1	6	0.31	574	0.69	0.04	0.93	22.6	870	5.4
RS-01B	14	15	3.3	3.73	0.08	0.3	0.01	0.013	0.08	19.9	6.6	0.37	469	0.57	0.06	0.25	20.9	900	5.9
RS-01B	18	20	3.02	4.45	<0.05	0.32	0.01	0.013	0.11	19.1	7	0.6	465	0.67	0.16	0.21	32	810	4
RS-01B	20	25																	
RS-03	5	10	2.13	2.53	<0.05	0.07	0.12	0.013	0.04	17.3	4.3	0.3	179	12.1	0.03	0.91	60.5	580	6.5
RS-03	15	20	2.18	3.06	0.08	0.3	0.01	0.011	0.1	17	7.8	0.43	211	0.62	0.05	0.41	28	880	4.1
RS-03	20	22	3.65	5.19	<0.05	0.25	0.01	0.012	0.15	15.3	10	1.54	392	1.57	0.25	0.38	104	770	3.5
RS-04	1	5	4.12	8.02	<0.05	0.13	0.02	0.029	0.11	23.6	15.3	0.66	386	1.44	0.04	1.61	104	860	6.7
RS-04	10	15	2.43	3.4	<0.05	0.25	0.01	0.01	0.08	21	8.1	0.36	255	0.9	0.05	0.56	31	1010	4.4
RS-04	15	20	1.94	3.15	<0.05	0.33	0.01	0.012	0.08	18.4	7.5	0.42	183	0.64	0.06	0.41	37	960	3.8
RS-04	20	25	2.7	3.7	<0.05	0.31	0.01	0.012	0.1	16.7	9.5	0.72	233	0.97	0.08	0.29	88.4	880	4
RS-05A	5	10	3.29	5.88	<0.05	0.28	0.02	0.021	0.17	25.4	13.1	0.56	339	1.14	0.04	0.5	58.9	880	5.2
RS-05A	10	13	4.13	9.24	<0.05	0.08	0.01	0.011	0.13	9	9.3	2.64	501	2.88	0.62	0.18	170	370	2.6
RS-06A	0.5	2	2.95	6.81	<0.05	0.11	0.03	0.023	0.12	18.8	17.6	0.45	303	0.79	0.02	1.93	36.6	1240	6.1
RS-06A	2	4																	
RS-06A	5	7.5	3.01	5.26	<0.05	0.27	0.01	0.018	0.1	25.1	11.2	0.5	253	0.78	0.05	0.49	31.9	880	4.4
RS-06A	7.5	10																	
RS-06A	15	19	2.86	4.25	<0.05	0.34	0.01	0.014	0.13	23.3	9.9	0.46	332	1.1	0.05	0.38	33.6	1000	4.5
RS-06A	19	21																	
RS-07	1	2	5.23	12.4	<0.05	0.16	0.02	0.038	0.27	18.4	55.3	0.92	212	2.88	0.02	4.38	109.5	940	5.7
RS-07	2	3																	
RS-07	3	5																	
RS-07	5	6																	
RS-07	6	10																	
RS-07R	10	12																	
RS-07R	13.5	14.5																	
RS-07/RS-07R	6	14.5	6.42	9.13	0.15	0.32	0.03	0.045	0.51	25	32.7	1.01	289	4.77	0.06	1.62	118	1130	6.7
RS-08A	0	1	3.87	8.35	<0.05	0.14	0.06	0.028	0.12	19.9	20.4	0.53	232	1.34	0.03	2.74	62.6	620	5.8
RS-08A	1	5																	
RS-08A	5	11	3.84	5.52	0.09	0.36	0.03	0.022	0.23	22.5	14.4	0.68	385	1.25	0.06	0.34	66.9	920	4.2
RS-09	7	8	5.33	10.7	0.08	0.22	0.02	0.024	0.31	15.5	17.1	2.36	509	7.75	0.59	0.4	178.5	500	3.7
RS-10	1	2	3.12	8.92	<0.05	0.1	0.04	0.025	0.17	17.5	25	0.49	302	1.56	0.03	2.38	57.2	580	6.6
RS-10	2	3																	
RS-10	3	5.5																	
RS-10	5.5	7.5	3.52	7.67	0.09	0.49	0.01	0.021	0.46	27.7	29.2	0.92	369	1.16	0.03	0.46	38.3	1320	4.2
RS-10	7.5	10	5.78	8.57	0.18	0.48	0.01	0.031	0.54	22.4	22.3	1.12	423	1.44	0.09	0.28	85.1	820	7.6
RS-10	10	14																	
RS-11	0	9.5	3.01	8.34	0.1	0.22	0.03	0.035	0.29	19	26.4	0.93	227	1.45	0.09	2.77	102	740	6.4
RS-11	11.5	17	4.64	9.4	0.13	0.4	0.02	0.056	0.35	25.4	29	1.12	241	3.64	0.09	2.27	115.5	830	8.8
RS-11	17	25	9.75	9.8	0.31	0.47	0.03	0.071	0.52	20.3	36.3	1.83	358	5.44	0.11	1.11	444	920	9.3
RS-11	28	31	3.71	4.24	0.06	0.38	0.01	0.016	0.23	20.7	14.4	0.61	285	1.1	0.06	0.42	30.3	1010	3.8
RS-11	31	33																	
RS-12	7	9	2.68	4	0.05	0.34	0.01	0.013	0.11	26.7	10	0.45	332	0.55	0.03	0.38	24.5	1110	4.3
RS-12	16	18	2.58	3.21	0.06	0.29	0.01	0.014	0.09	22.2	8.3	0.28	290	0.84	0.03	0.39	24	1060	4.1
RS-12	20	22	4.51	5.52	0.09	0.26	0.02	0.028	0.16	19.7	9.9	1.43	447	2.62	0.2	0.35	374	1000	4.6
RS-13	0	1.5	3.74	6.85	<0.05	0.08	0.05	0.026	0.09	13.9	16	0.59	688	2.02	0.07	2.02	108.5	790	6.6
RS-13	1.5	2.5	4.23	8.59	0.05	0.29	0.02	0.029	0.56	23.7	23	0.98	413	1.1	0.04	2.25	84.4	820	4.2
RS-13	2.5	6																	
RS-13	8	10	5.68	7.77	0.1	0.28	0.02	0.033	0.3	20.5	19	0.78	412	5.04	0.18	0.82	517	880	5.7
RS-14B	0	1.5	3.56	6.6	<0.05	0.03	0.08	0.025	0.11	9.2	15.5	0.41	400	1.23	0.02	1.7	59.1	610	13.8
RS-14B	1.5	3	3.88	8.65	<0.05	0.09	0.06	0.029	0.2	17.7	24.2	0.62	327	1.65	0.04	2.42	94.6	670	5.6
RS-14B	3	5																	
RS-16B	0	2	4.02	8.8	<0.05	0.11	0.05	0.031	0.06	12.2	14.4	0.57	277	1.06	0.03	2.04	62.2	920	6.7

Fraction Analyzed			-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm
Parameter	**Depth**	**Depth**	**Rb**	**Re**	**S**	**Sb**	**Sc**	**Se**	**Sn**	**Sr**	**Ta**	**Te**	**Th**	**Ti**	**Tl**	**U**	**V**	**W**	**Y**
Units	From (ft)	To (ft)	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm
Detection Limit			0.1	0.001	0.01	0.05	0.1	0.2	0.2	0.2	0.01	0.01	0.2	0.005	0.02	0.05	1	0.05	0.05
General Method			Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia
Analytical Method Code			ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41
RS-01B	0	1	14.7	<0.001	0.05	0.19	1.9	0.6	1.7	29.3	<0.01	0.03	0.7	0.061	0.09	0.44	56	1.68	2.5
RS-01B	1	5	8.5	<0.001	0.02	0.05	3.6	0.4	0.6	26.1	<0.01	0.01	5	0.099	0.06	0.74	51	0.35	6.2
RS-01B	14	15	8.6	<0.001	0.02	0.08	4.2	0.3	0.6	31	<0.01	0.01	4.6	0.104	0.06	0.68	50	0.32	7.64
RS-01B	18	20	8.8	<0.001	0.03	0.08	3.7	0.3	1.4	42	<0.01	0.01	4.5	0.106	0.07	0.67	46	1.66	7.43
RS-01B	20	25																	
RS-03	5	10	4.1	0.011	0.46	0.35	3.3	2.8	1.2	48.5	0.01	0.03	6.6	0.035	0.06	3.47	31	3.81	8.45
RS-03	15	20	7.7	0.001	0.07	0.14	2.6	0.4	0.4	30.2	<0.01	0.02	3.6	0.096	0.06	0.95	44	0.3	6.96
RS-03	20	22	9.7	<0.001	0.05	0.17	3.7	1.6	1.1	54.9	0.01	0.02	3.7	0.12	0.08	1.05	43	2.82	6.7
RS-04	1	5	16.5	<0.001	0.01	0.24	7	0.6	0.8	29	<0.01	0.03	5	0.137	0.16	1.56	80	0.3	10.6
RS-04	10	15	7.2	<0.001	0.01	0.11	3	0.5	0.6	30	<0.01	0.01	5.2	0.107	0.06	0.9	46	0.62	7.82
RS-04	15	20	6.3	<0.001	0.05	0.07	2.7	0.3	0.6	30.1	<0.01	0.01	4.9	0.106	0.05	1.25	43	0.91	7.48
RS-04	20	25	7.9	0.002	0.12	0.12	3.6	0.4	0.6	30.5	<0.01	0.02	4.3	0.103	0.08	1.26	46	0.57	7.18
RS-05A	5	10	21.8	<0.001	<0.01	0.24	6.4	0.4	0.8	29.1	<0.01	0.03	6	0.141	0.18	1.43	64	1.31	10.7
RS-05A	10	13	7	0.001	<0.01	0.08	3.7	0.3	0.5	114	<0.01	0.01	1.8	0.067	0.06	0.48	25	0.79	3.79
RS-06A	0.5	2	16.7	<0.001	0.01	0.13	4	0.7	0.7	22.3	<0.01	0.02	4.8	0.121	0.13	0.97	65	0.34	5.87
RS-06A	2	4																	
RS-06A	5	7.5	11.4	<0.001	<0.01	0.14	5.4	0.5	0.7	35.3	<0.01	0.02	5.2	0.129	0.11	1.12	58	1.41	10.15
RS-06A	7.5	10																	
RS-06A	15	19	13	<0.001	<0.01	0.14	4.1	0.3	0.6	31.7	<0.01	0.02	5.5	0.119	0.1	1.01	51	0.68	8.58
RS-06A	19	21																	
RS-07	1	2	36.3	0.001	0.04	0.25	9.1	1	1	18.3	<0.01	0.03	5.3	0.221	0.2	1.76	97	0.71	7.18
RS-07	2	3																	
RS-07	3	5																	
RS-07	5	6																	
RS-07	6	10																	
RS-07R	10	12																	
RS-07R	13.5	14.5																	
RS-07/RS-07R	6	14.5	44.4	0.002	0.27	0.72	10.8	1.3	0.9	26	0.01	0.05	6.5	0.197	0.4	3.5	104	6.67	14.6
RS-08A	0	1	19.2	<0.001	0.02	0.21	5.6	1.1	0.8	25.3	<0.01	0.04	4.4	0.143	0.16	1.22	69	0.39	7.86
RS-08A	1	5																	
RS-08A	5	11	24.8	<0.001	0.01	0.31	5.4	0.4	0.7	32.7	0.01	0.02	5.4	0.14	0.19	1.37	58	11.95	8.99
RS-09	7	8	21.1	0.001	0.03	0.27	6.4	0.4	0.8	113.5	0.01	0.03	4	0.129	0.19	1.62	50	1.58	7.26
RS-10	1	2	23.2	<0.001	0.01	0.15	4.8	0.7	1	24.3	<0.01	0.02	5.4	0.131	0.15	1.14	72	3.37	5.67
RS-10	2	3																	
RS-10	3	5.5																	
RS-10	5.5	7.5	38.3	<0.001	0.01	0.29	7.5	0.5	0.8	30.3	<0.01	0.03	6.6	0.189	0.29	1.82	72	0.71	9.44
RS-10	7.5	10	46.5	<0.001	0.01	0.42	9.1	0.4	0.8	36.2	<0.01	0.03	5.7	0.192	0.33	1.92	74	1.96	10
RS-10	10	14																	
RS-11	0	9.5	27.9	0.001	0.07	0.59	8.4	1.1	1	30.9	<0.01	0.04	4.7	0.191	0.26	2.66	102	1.52	9.94
RS-11	11.5	17	37.1	0.002	0.19	1.01	10.7	1.5	1	29.9	0.01	0.05	6	0.223	0.32	4.54	130	0.7	13.35
RS-11	17	25	43.7	0.006	0.65	1.68	10.4	3.3	3.5	36.9	<0.01	0.14	5.5	0.205	0.4	4.46	95	4.87	13.45
RS-11	28	31	16.3	<0.001	0.09	0.18	3.9	0.4	0.7	31.8	<0.01	0.02	5.2	0.126	0.13	1.29	53	0.68	8
RS-11	31	33																	
RS-12	7	9	11.3	<0.001	<0.01	0.12	4.5	0.4	0.5	30	0.01	0.01	5.7	0.109	0.11	1.07	51	0.19	10.6
RS-12	16	18	9.8	<0.001	<0.01	0.18	3.5	0.4	0.4	26.8	0.01	0.02	5.9	0.098	0.11	1.38	48	0.31	9.98
RS-12	20	22	14.2	0.002	0.19	0.23	4.3	0.9	0.7	49.8	0.01	0.04	4.7	0.12	0.1	1.08	49	1.32	9.84
RS-13	0	1.5	10.7	0.001	0.03	0.18	4.4	0.9	0.7	30.8	0.01	0.03	3.2	0.129	0.11	0.96	89	1.53	6.62
RS-13	1.5	2.5	44.4	0.001	0.01	0.16	10.3	0.7	0.9	21.2	0.01	0.03	5.9	0.217	0.33	1.38	97	0.48	9.45
RS-13	2.5	6																	
RS-13	8	10	25.5	0.005	0.58	0.74	6.2	1.9	1	51.1	<0.01	0.07	4.6	0.156	0.28	1.35	69	0.98	9.14
RS-14B	0	1.5	16.1	<0.001	0.04	0.26	3	0.7	0.8	15.8	<0.01	0.03	1.6	0.096	0.12	0.68	71	0.58	3.26
RS-14B	1.5	3	20.5	<0.001	0.02	0.22	5.9	1	0.8	19.7	0.01	0.03	4.4	0.131	0.21	1.54	73	1.2	7.62
RS-14B	3	5																	
RS-16B	0	2	12.7	<0.001	0.01	0.15	4.4	0.8	0.8	15.9	0.01	0.03	4.2	0.138	0.1	0.91	114	0.18	4.77

Fraction Analyzed			-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm
Parameter	**Depth**	**Depth**	**Zn**	**Zr**	**Ag**	**Al**	**As**	**Ba**	**Be**	**Bi**	**Ca**	**Cd**	**Co**	**Cr**	**Cu**	**Fe**	**Ga**	**K**	**La**	**Mg**	**Mn**	**Mo**	**Na**
Units	From (ft)	To (ft)	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	ppm	%	ppm	%	ppm	ppm	%
Detection Limit			2	0.5	1	0.05	50	50	10	20	0.05	10	10	10	10	0.05	50	0.1	50	0.05	10	10	0.05
General Method			Aqua Regia	Aqua Regia	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid
Analytical Method Code			ME-MS41	ME-MS41	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a
RS-01B	0	1	168	<0.5	3	3.61	<50	520	<10	<20	1.55	<10	10	100	20	4.06	<50	0.7	<50	0.6	1690	<10	1.59
RS-01B	1	5	24	5.4	1	4.15	<50	510	<10	<20	1.8	<10	10	80	20	4.39	<50	1.2	<50	0.67	840	<10	2.29
RS-01B	14	15	28	10.5	<1	3.87	<50	500	<10	<20	2.04	<10	10	80	30	4.42	<50	1.1	<50	0.7	750	<10	2.39
RS-01B	18	20	33	10.7	9	2.99	<50	470	<10	<20	2.38	<10	10	100	30	4.24	<50	1.1	<50	0.85	780	<10	2.42
RS-01B	20	25																					
RS-03	5	10	24	2.9	1	2.26	<50	270	<10	<20	1.85	<10	<10	70	260	2.77	<50	0.7	<50	0.52	310	10	0.92
RS-03	15	20	30	9.5	<1	3.83	<50	520	<10	<20	2.68	<10	10	80	40	3.72	<50	1.2	<50	0.9	570	<10	2.47
RS-03	20	22	48	8	5	4.17	<50	440	<10	<20	3.21	<10	20	90	110	4.86	<50	1	<50	1.76	710	<10	2.28
RS-04	1	5	63	5.2	<1	4.89	<50	480	<10	<20	1.76	<10	20	100	240	5.17	<50	0.9	<50	0.93	640	<10	1.87
RS-04	10	15	29	9.3	<1	3.07	<50	520	<10	<20	2.16	<10	10	80	30	3.4	<50	1.3	<50	0.72	520	<10	2.48
RS-04	15	20	26	10.1	<1	3.15	<50	500	<10	<20	2.29	<10	10	80	50	3.06	<50	1.5	<50	0.82	460	<10	2.54
RS-04	20	25	34	9.9	2	3.97	<50	490	<10	<20	2.52	<10	20	90	140	4.05	<50	1.3	<50	1.14	560	<10	2.43
RS-05A	5	10	52	10.8	1	4.63	<50	550	<10	<20	2.02	<10	20	100	100	4.48	<50	1	<50	0.91	640	<10	2.3
RS-05A	10	13	53	2.8	2	6.88	<50	300	<10	<20	5.56	<10	40	270	60	5.26	<50	0.5	<50	2.57	720	<10	2.11
RS-06A	0.5	2	48	4.6	<1	4.28	<50	620	<10	<20	1.63	<10	10	100	30	4.06	<50	0.9	<50	0.84	590	<10	2.04
RS-06A	2	4																					
RS-06A	5	7.5	37	10.7	<1	3.55	<50	520	<10	<20	1.96	<10	10	90	70	4.04	<50	1.2	<50	0.79	550	<10	2.3
RS-06A	7.5	10																					
RS-06A	15	19	38	11.4	<1	3.33	<50	520	<10	<20	2.07	<10	10	80	50	3.94	<50	1.1	<50	0.82	640	<10	2.4
RS-06A	19	21																					
RS-07	1	2	142	6.4	<1	3.91	<50	450	<10	<20	1.29	<10	20	150	110	6.41	<50	0.8	<50	1.3	540	<10	1.45
RS-07	2	3																					
RS-07	3	5																					
RS-07	5	6																					
RS-07	6	10																					
RS-07R	10	12																					
RS-07R	13.5	14.5																					
RS-07/RS-07R	6	14.5	131	12.3	2	4.91	<50	520	<10	<20	1.76	<10	20	130	240	8.26	<50	1.3	<50	1.42	790	<10	1.7
RS-08A	0	1	52	5.5	<1	4.57	<50	510	<10	<20	1.38	<10	10	100	70	4.94	<50	1.1	<50	0.87	550	<10	1.81
RS-08A	1	5																					
RS-08A	5	11	61	13.4	<1	3.9	<50	560	<10	<20	2	<10	20	90	120	5.28	<50	1.3	<50	1.05	810	<10	2.24
RS-09	7	8	74	8.3	<1	5.39	<50	370	<10	<20	3.84	<10	30	140	100	6.15	<50	0.9	<50	2.22	760	<10	1.97
RS-10	1	2	54	4	<1	4.01	<50	510	<10	<20	1.15	<10	20	100	40	3.92	<50	0.9	<50	0.76	520	<10	1.57
RS-10	2	3																					
RS-10	3	5.5																					
RS-10	5.5	7.5	70	18.2	2	4.72	<50	800	<10	<20	1.91	<10	10	80	50	4.46	<50	1.4	<50	1.18	650	<10	2.45
RS-10	7.5	10	78	17.6	<1	4.61	<50	490	<10	<20	1.96	<10	20	110	90	6.81	<50	1.2	<50	1.31	800	<10	1.98
RS-10	10	14																					
RS-11	0	9.5	120	7.7	3	3.69	<50	360	<10	<20	2.33	<10	20	210	170	5	<50	0.9	<50	1.43	660	<10	1.48
RS-11	11.5	17	159	13.8	1	3.93	<50	410	<10	<20	2.2	<10	20	160	290	6.17	<50	0.9	<50	1.48	640	<10	1.69
RS-11	17	25	210	15.8	24	5.6	<50	390	<10	<20	2.35	<10	40	140	1510	12.25	<50	1.1	<50	2.32	930	<10	1.36
RS-11	28	31	40	11.5	1	3.8	<50	600	<10	<20	2.36	<10	10	80	40	5.18	<50	1.4	<50	1.05	740	<10	2.4
RS-11	31	33																					
RS-12	7	9	32	11.1	<1	3.72	<50	560	<10	<20	2.15	<10	10	70	30	3.73	<50	1.3	<50	0.8	640	<10	2.42
RS-12	16	18	38	9.5	<1	3.26	<50	550	<10	<20	1.97	<10	10	80	30	3.65	<50	1.4	<50	0.69	600	<10	2.47
RS-12	20	22	53	8.8	<1	4.62	<50	430	<10	<20	3	<10	40	100	780	5.89	<50	1	<50	1.76	780	<10	2.14
RS-13	0	1.5	61	2.8	1	4.43	<50	410	<10	<20	2.29	<10	20	160	150	5.63	<50	0.7	<50	1.13	1110	<10	1.71
RS-13	1.5	2.5	71	12.3	<1	4.57	<50	430	<10	<20	1.81	<10	20	130	90	5.3	<50	0.9	<50	1.2	700	<10	2.11
RS-13	2.5	6																					
RS-13	8	10	58	11.2	<1	5.5	<50	500	<10	<20	2.39	<10	70	140	1120	8.21	<50	1.2	<50	1.57	990	<10	1.95
RS-14B	0	1.5	69	1	<1	3.36	<50	410	<10	<20	1.15	<10	20	120	120	4.43	<50	1.1	<50	0.73	650	<10	1.23
RS-14B	1.5	3	64	3.2	<1	4.11	<50	460	<10	<20	1.52	<10	20	120	190	4.96	<50	1.2	<50	0.95	640	<10	1.66
RS-14B	3	5																					
RS-16B	0	2	51	4.2	<1	4.62	<50	410	<10	<20	1.59	<10	20	160	40	5.64	<50	1	<50	1.02	630	<10	1.42

Fraction Analyzed			-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm
Parameter	**Depth**	**Depth**	**Ni**	**P**	**Pb**	**S**	**Sb**	**Sc**	**Sr**	**Th**	**Ti**	**Tl**	**U**	**V**	**W**	**Zn**	**CO_2**	**Total Sulphur**	**Sulphate**	**MPA**	**NP**
Units	From (ft)	To (ft)	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	% CO2	% S	% S	kgCaCO3/t ore	kgCaCO3/t ore
Detection Limit			10	50	20	0.1	50	10	10	50	0.05	50	50	10	50	20	0.2	0.01	0.01	0.3	1
General Method			4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	ABA	ABA	ABA	ABA	ABA
Analytical Method Code			ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	C-GAS05	S-IR08	S-GRA06a	OA-VOL08	OA-VOL08
RS-01B	0	1	40	1140	<20	<0.1	<50	<10	260	<50	0.44	<50	<50	90	<50	110	<0.2	0.09	<0.01	1.6	3
RS-01B	1	5	30	810	<20	<0.1	<50	10	360	<50	0.34	<50	<50	80	<50	40	<0.2	<0.01	<0.01	<0.3	1
RS-01B	14	15	30	860	<20	<0.1	<50	10	360	<50	0.34	<50	<50	80	<50	40	<0.2	<0.01	<0.01	<0.3	3
RS-01B	18	20	50	760	<20	<0.1	<50	<10	340	<50	0.32	<50	<50	80	<50	50	<0.2	0.01	<0.01	0.3	9
RS-01B	20	25																			
RS-03	5	10	60	590	<20	0.5	<50	<10	180	<50	0.2	<50	<50	50	<50	30	<0.2	0.31	<0.01	9.7	19
RS-03	15	20	40	860	<20	0.1	<50	10	370	<50	0.39	<50	<50	90	<50	40	0.2	0.05	<0.01	1.6	9
RS-03	20	22	120	730	<20	<0.1	<50	<10	330	<50	0.39	<50	<50	80	<50	60	<0.2	0.02	<0.01	0.6	26
RS-04	1	5	110	880	<20	<0.1	<50	10	300	<50	0.41	<50	<50	110	<50	70	<0.2	<0.01	<0.01	<0.3	6
RS-04	10	15	40	930	<20	<0.1	<50	<10	360	<50	0.35	<50	<50	80	<50	40	<0.2	<0.01	<0.01	<0.3	4
RS-04	15	20	50	890	<20	<0.1	<50	<10	350	<50	0.36	<50	<50	80	<50	40	<0.2	0.02	<0.01	0.6	7
RS-04	20	25	110	860	<20	0.1	<50	10	360	<50	0.37	<50	<50	90	<50	50	<0.2	0.12	<0.01	3.8	19
RS-05A	5	10	60	860	<20	<0.1	<50	10	360	<50	0.39	<50	<50	100	<50	60	<0.2	<0.01	<0.01	<0.3	25
RS-05A	10	13	190	360	<20	<0.1	<50	<10	320	<50	0.25	<50	<50	60	<50	60	<0.2	<0.01	<0.01	<0.3	5
RS-06A	0.5	2	50	1230	<20	<0.1	<50	10	340	<50	0.44	<50	<50	100	<50	60	<0.2	<0.01	<0.01	<0.3	3
RS-06A	2	4																			
RS-06A	5	7.5	80	810	30	<0.1	<50	10	340	<50	0.38	<50	<50	90	<50	60	<0.2	<0.01	<0.01	<0.3	6
RS-06A	7.5	10																			
RS-06A	15	19	50	920	<20	<0.1	<50	10	340	<50	0.37	<50	<50	80	<50	50	<0.2	<0.01	<0.01	<0.3	6
RS-06A	19	21																			
RS-07	1	2	120	1010	<20	<0.1	<50	10	200	<50	0.46	<50	<50	130	<50	160	<0.2	0.01	<0.01	0.3	2
RS-07	2	3																			
RS-07	3	5																			
RS-07	5	6																			
RS-07	6	10																			
RS-07R	10	12																			
RS-07R	13.5	14.5																			
RS-07/RS-07R	6	14.5	130	1160	<20	0.2	<50	10	240	<50	0.42	<50	<50	130	<50	150	<0.2	0.06	<0.01	1.9	4
RS-08A	0	1	70	610	<20	<0.1	<50	10	290	<50	0.39	<50	<50	90	<50	60	<0.2	<0.01	<0.01	<0.3	4
RS-08A	1	5																			
RS-08A	5	11	70	890	<20	<0.1	<50	10	350	<50	0.38	<50	<50	90	<50	80	<0.2	<0.01	<0.01	<0.3	6
RS-09	7	8	190	500	<20	<0.1	<50	10	300	<50	0.3	<50	<50	80	<50	80	<0.2	0.02	<0.01	0.6	19
RS-10	1	2	70	590	<20	<0.1	<50	10	210	<50	0.44	<50	<50	100	<50	70	<0.2	<0.01	<0.01	<0.3	2
RS-10	2	3																			
RS-10	3	5.5																			
RS-10	5.5	7.5	40	1230	<20	<0.1	<50	10	450	<50	0.33	<50	<50	90	<50	80	<0.2	<0.01	<0.01	<0.3	6
RS-10	7.5	10	100	780	<20	<0.1	<50	10	290	<50	0.3	<50	<50	90	<50	80	<0.2	0.01	<0.01	0.3	12
RS-10	10	14																			
RS-11	0	9.5	130	790	<20	0.1	<50	10	180	<50	0.66	<50	<50	150	<50	150	<0.2	0.03	<0.01	0.9	7
RS-11	11.5	17	130	840	<20	0.2	<50	10	210	<50	0.65	<50	<50	170	<50	180	<0.2	0.03	<0.01	0.9	5
RS-11	17	25	470	950	<20	0.6	<50	10	200	<50	0.4	<50	<50	130	<50	240	<0.2	0.11	<0.01	3.4	5
RS-11	28	31	40	920	<20	0.1	<50	10	390	<50	0.33	<50	<50	90	<50	60	0.2	0.05	<0.01	1.6	11
RS-11	31	33																			
RS-12	7	9	40	1030	<20	<0.1	<50	10	380	<50	0.36	<50	<50	80	<50	50	<0.2	<0.01	<0.01	<0.3	5
RS-12	16	18	40	970	<20	<0.1	<50	10	350	<50	0.36	<50	<50	80	<50	50	<0.2	<0.01	<0.01	<0.3	3
RS-12	20	22	390	950	<20	0.2	<50	10	330	<50	0.46	<50	<50	90	<50	70	<0.2	0.05	0.04	1.6	20
RS-13	0	1.5	130	870	<20	<0.1	<50	10	250	<50	0.74	<50	<50	140	<50	80	<0.2	0.01	0.02	0.3	9
RS-13	1.5	2.5	100	850	<20	<0.1	<50	10	300	<50	0.51	<50	<50	120	<50	80	<0.2	<0.01	0.02	<0.3	5
RS-13	2.5	6																			
RS-13	8	10	530	860	<20	0.5	<50	10	330	<50	0.51	<50	<50	110	<50	90	<0.2	0.5	0.07	15.6	10
RS-14B	0	1.5	90	670	40	<0.1	<50	10	180	<50	0.51	<50	<50	110	<50	110	<0.2	0.09	0.01	2.8	3
RS-14B	1.5	3	110	710	<20	<0.1	<50	10	240	<50	0.47	<50	<50	110	<50	90	<0.2	0.02	0.02	0.6	4
RS-14B	3	5																			
RS-16B	0	2	80	950	<20	<0.1	<50	10	190	<50	0.93	<50	<50	180	<50	80	<0.2	0.03	0.03	0.9	3

Fraction Analyzed			-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm
Parameter	**Depth**	**Depth**	**Fizz Rating**	**Ag**	**Al**	**As**	**Au**	**B**	**Ba**	**Be**	**Bi**	**Ca**	**Cd**	**Ce**	**Co**	**Cr**	**Cs**	**Cu**
Units	From (ft)	To (ft)	Unity	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm
Detection Limit			1	0.01	0.01	0.1	0.2	10	10	0.05	0.01	0.01	0.01	0.02	0.1	1	0.05	0.2
General Method			ABA															
Analytical Method Code			OA-VOL08	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41
RS-01B	0	1	1	2.61	0.94	5.3	<0.2	<10	160	0.28	0.2	0.67	0.48	17.6	7.8	66	0.95	13
RS-01B	1	5	1	0.81	1	5.7	<0.2	<10	40	0.34	0.04	0.36	0.07	35.6	7.3	115	0.9	20
RS-01B	14	15	1	0.64	0.78	3.6	<0.2	<10	40	0.29	0.03	0.44	0.08	31.1	7.4	117	0.83	25.8
RS-01B	18	20	1	7.85	0.97	2.8	<0.2	<10	40	0.26	0.03	0.59	0.06	30.3	8.5	105	0.72	26.2
RS-01B	20	25																
RS-03	5	10	1	0.36	0.78	3.4	<0.2	<10	70	0.31	0.04	1.19	0.45	22.1	4.8	246	0.53	217
RS-03	15	20	1	0.09	0.77	4.2	<0.2	<10	40	0.27	0.03	0.62	0.1	30.7	6.3	118	0.68	36.1
RS-03	20	22	1	3.39	1.55	2.2	<0.2	<10	50	0.27	0.04	0.95	0.1	30.2	16.6	102	1.07	91.5
RS-04	1	5	1	0.14	1.74	5.7	<0.2	<10	60	0.41	0.11	0.48	0.1	34.8	14.3	124	1.34	146.5
RS-04	10	15	1	0.47	0.64	1.4	<0.2	<10	30	0.25	0.02	0.41	0.04	30.8	5.3	109	0.53	21.5
RS-04	15	20	1	0.36	0.66	1.2	<0.2	<10	30	0.19	0.03	0.46	0.05	30.8	5.9	110	0.62	63.6
RS-04	20	25	1	0.69	1.08	2.3	<0.2	<10	30	0.26	0.06	0.57	0.13	27.8	14.3	104	1.06	187.5
RS-05A	5	10	1	0.07	1.1	2.8	<0.2	<10	50	0.38	0.06	0.46	0.09	34	10.1	123	0.95	60
RS-05A	10	13	1	0.05	3.58	1.3	<0.2	<10	40	0.19	<0.01	2.03	0.08	15.7	29.8	133	0.57	36.1
RS-06A	0.5	2	1	0.11	1.61	2.7	<0.2	<10	70	0.4	0.06	0.32	0.07	38.1	9.8	95	1.39	20.9
RS-06A	2	4																
RS-06A	5	7.5	1	0.14	1.13	3.2	<0.2	<10	50	0.32	0.06	0.46	0.06	39	9.4	105	1.01	29.3
RS-06A	7.5	10																
RS-06A	15	19	1	0.11	0.76	3.7	<0.2	<10	40	0.29	0.03	0.44	0.08	33.5	7.3	107	0.73	23.7
RS-06A	19	21																
RS-07	1	2	1	0.01	1.63	5.8	<0.2	<10	60	0.46	0.07	0.28	0.07	29.1	8.3	120	2.07	59.2
RS-07	2	3																
RS-07	3	5																
RS-07	5	6																
RS-07	6	10																
RS-07R	10	12																
RS-07R	13.5	14.5																
RS-07/RS-07R	6	14.5	1	0.11	1.82	6.9	<0.2	<10	70	0.66	0.17	0.46	0.27	34.6	16.1	138	2.49	140.5
RS-08A	0	1	1	0.1	1.87	5.6	<0.2	<10	50	0.54	0.08	0.41	0.07	31.7	12.3	98	1.39	46.1
RS-08A	1	5																
RS-08A	5	11	1	0.38	1.17	8.5	<0.2	<10	50	0.36	0.06	0.49	0.15	35.2	14.3	115	1.49	74.5
RS-09	7	8	1	0.08	4.06	4.2	<0.2	<10	70	0.41	0.06	2.04	0.15	29.4	27.5	140	2.03	85.7
RS-10	1	2	1	0.05	2	3.6	<0.2	<10	80	0.45	0.09	0.37	0.05	32.7	12.9	111	1.79	29.4
RS-10	2	3																
RS-10	3	5.5																
RS-10	5.5	7.5	1	0.05	1.57	3.6	<0.2	<10	80	0.47	0.09	0.39	0.05	42.2	10.1	131	2.59	36.6
RS-10	7.5	10	1	0.06	2.13	7.1	<0.2	<10	80	0.47	0.09	0.67	0.12	34.6	16.5	125	2.75	63.7
RS-10	10	14																
RS-11	0	9.5	1	0.79	1.87	3.2	<0.2	<10	70	0.48	0.09	0.64	0.28	32.7	13.3	143	1.96	115
RS-11	11.5	17	1	0.12	1.97	4.7	<0.2	<10	70	0.53	0.1	0.66	0.21	33.9	12.7	142	2.17	155
RS-11	17	25	1	6.33	2.11	6	<0.2	<10	70	0.53	0.17	0.72	0.32	30.3	19.4	137	2.41	680
RS-11	28	31	2	0.36	0.9	3.5	<0.2	<10	50	0.35	0.05	0.57	0.09	31.5	7.1	113	1.15	34.3
RS-11	31	33																
RS-12	7	9	1	0.03	0.71	3	<0.2	<10	40	0.29	0.06	0.4	0.05	29.4	5.4	120	0.67	17
RS-12	16	18	1	0.03	0.63	2.6	<0.2	<10	30	0.26	0.06	0.37	0.08	30.3	5	105	0.63	17.2
RS-12	20	22	1	0.13	1.24	4.2	<0.2	<10	40	0.26	0.07	0.66	0.1	27.4	19.5	113	0.68	342
RS-13	0	1.5	1	0.08	1.83	3.5	<0.2	<10	50	0.44	0.05	0.6	0.15	30.1	17.3	99	0.88	97.2
RS-13	1.5	2.5	1	0.07	1.75	3.8	<0.2	<10	60	0.46	0.03	0.5	0.07	36.2	14.4	113	1.43	62
RS-13	2.5	6																
RS-13	8	10	1	0.3	1.88	9.8	<0.2	<10	60	0.36	0.15	0.78	0.27	34.4	48.4	120	1.43	817
RS-14B	0	1.5	1	0.21	1.83	4.9	<0.2	<10	80	0.38	0.13	0.3	0.24	24.4	13.5	90	1.45	73.8
RS-14B	1.5	3	1	0.14	2.26	4.9	<0.2	<10	60	0.48	0.09	0.34	0.12	31.4	16.4	88	1.78	116.5
RS-14B	3	5																
RS-16B	0	2	1	0.07	2.24	4.6	<0.2	<10	60	0.45	0.07	0.3	0.14	28.2	15.2	84	0.99	22.9

Fraction Analyzed			-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm
Parameter	**Depth**	**Depth**	**Fe**	**Ga**	**Ge**	**Hf**	**Hg**	**In**	**K**	**La**	**Li**	**Mg**	**Mn**	**Mo**	**Na**	**Nb**	**Ni**	**P**
Units	From (ft)	To (ft)	%	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	%	ppm	ppm	%	ppm	ppm	ppm
Detection Limit			0.01	0.05	0.05	0.02	0.01	0.005	0.01	0.2	0.1	0.01	5	0.05	0.01	0.05	0.2	10
General Method																		
Analytical Method Code			ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41
RS-01B	0	1	2.53	3.98	0.05	<0.02	0.08	0.015	0.08	7.6	5.7	0.22	1755	1.7	0.02	0.97	17.6	1010
RS-01B	1	5	2.99	3.49	0.13	0.17	0.02	0.011	0.08	16.6	7.5	0.33	581	2.02	0.04	0.63	18.9	560
RS-01B	14	15	2.72	3.05	0.13	0.27	0.01	0.01	0.09	15.6	7.6	0.34	523	2.01	0.05	0.54	20.6	550
RS-01B	18	20	2.54	3.32	0.12	0.25	0.01	0.01	0.11	14.6	7.5	0.5	411	1.91	0.12	0.21	26.5	510
RS-01B	20	25																
RS-03	5	10	2.91	2.19	0.11	0.05	0.1	0.01	0.04	12.3	3.8	0.25	286	42.9	0.02	0.94	70.2	490
RS-03	15	20	2.09	2.96	0.08	0.26	0.01	0.009	0.11	15.6	9.1	0.38	209	2.47	0.07	0.4	23.6	590
RS-03	20	22	2.98	4.55	0.1	0.24	0.01	0.012	0.16	14.9	10.6	1.15	322	2.68	0.22	0.28	79.6	490
RS-04	1	5	3.5	6.02	0.12	0.08	0.02	0.02	0.09	18.3	13.2	0.51	359	8.09	0.06	1.48	67.4	550
RS-04	10	15	1.62	2.75	0.05	0.23	<0.01	0.007	0.09	14.9	8.2	0.3	175	2.28	0.06	0.37	19.8	460
RS-04	15	20	1.5	2.71	0.06	0.26	0.01	0.008	0.09	15.8	8.1	0.34	158	2.02	0.07	0.32	27.6	490
RS-04	20	25	2.77	3.68	0.1	0.23	0.01	0.013	0.13	13.7	11.7	0.74	258	2.26	0.09	0.2	98	460
RS-05A	5	10	2.64	4.07	0.1	0.25	0.01	0.014	0.11	17.6	10.4	0.46	274	2.49	0.07	0.34	35.7	480
RS-05A	10	13	3.71	7.11	0.11	0.06	0.01	0.008	0.11	8.2	8.1	2.35	440	3.95	0.54	0.18	137.5	260
RS-06A	0.5	2	2.59	5.74	0.07	0.1	0.02	0.017	0.14	16.7	19.4	0.45	270	1.89	0.04	1.74	27.6	930
RS-06A	2	4																
RS-06A	5	7.5	2.62	4.26	0.11	0.27	0.01	0.013	0.12	20.3	10.8	0.46	285	2.09	0.06	0.42	26.1	620
RS-06A	7.5	10																
RS-06A	15	19	2.16	3.23	0.09	0.28	0.01	0.01	0.11	17.2	9.1	0.36	238	2.29	0.06	0.34	21.6	560
RS-06A	19	21																
RS-07	1	2	4.32	6.24	0.05	0.11	0.01	0.019	0.23	13.1	30.1	0.58	180	2.98	0.03	2.48	58.2	600
RS-07	2	3																
RS-07	3	5																
RS-07	5	6																
RS-07	6	10																
RS-07R	10	12																
RS-07R	13.5	14.5																
RS-07/RS-07R	6	14.5	5.42	6.32	0.22	0.25	0.01	0.026	0.37	17.5	28.1	0.76	246	3.71	0.07	1.01	72.3	670
RS-08A	0	1	3.66	5.76	0.12	0.1	0.03	0.018	0.11	15.8	15.8	0.52	257	2.02	0.05	1.86	47.8	620
RS-08A	1	5																
RS-08A	5	11	3.41	4.24	0.14	0.31	0.02	0.015	0.18	17.6	12.6	0.63	346	2.85	0.08	0.27	50.7	570
RS-09	7	8	4.88	8.86	0.17	0.21	0.01	0.019	0.29	13.9	15.3	2.04	470	11.65	0.52	0.34	151	440
RS-10	1	2	3.13	6.92	0.1	0.14	0.02	0.02	0.19	14.2	22	0.54	394	2.55	0.06	2.07	45	470
RS-10	2	3																
RS-10	3	5.5																
RS-10	5.5	7.5	2.87	6.1	0.12	0.39	0.01	0.016	0.37	21.5	23.5	0.69	326	2.39	0.05	0.41	31.1	660
RS-10	7.5	10	5.16	6.86	0.23	0.38	0.02	0.021	0.43	17.8	19.9	0.97	411	2.67	0.11	0.27	65.1	630
RS-10	10	14																
RS-11	0	9.5	2.45	6.28	0.19	0.17	0.02	0.025	0.25	16.2	21.1	0.68	234	2.69	0.09	2.29	68	510
RS-11	11.5	17	3.18	6.64	0.15	0.32	<0.01	0.029	0.29	17.6	22.4	0.73	207	3.16	0.12	1.4	64.3	570
RS-11	17	25	4.73	6.61	0.19	0.26	0.01	0.028	0.38	15.2	27.3	1.11	260	3.31	0.13	0.69	169.5	520
RS-11	28	31	2.74	3.6	0.11	0.27	<0.01	0.012	0.21	16.3	13.7	0.49	274	2.42	0.06	0.31	24.1	550
RS-11	31	33																
RS-12	7	9	2.03	2.93	0.08	0.26	0.01	0.01	0.1	16.2	8.4	0.32	239	2.17	0.05	0.35	15.1	510
RS-12	16	18	1.94	2.66	0.08	0.25	0.01	0.008	0.09	14.7	7.8	0.26	203	2.12	0.05	0.36	15.2	480
RS-12	20	22	3.02	3.76	0.11	0.2	0.01	0.014	0.11	13.5	8.6	0.89	297	3.05	0.13	0.27	160	470
RS-13	0	1.5	3.26	5.43	0.11	0.06	0.03	0.02	0.08	13	12.8	0.62	645	2.58	0.09	1.95	83.8	510
RS-13	1.5	2.5	3.21	5.65	0.1	0.22	0.01	0.017	0.19	16.8	14.9	0.71	344	2.18	0.09	1.4	66.7	500
RS-13	2.5	6																
RS-13	8	10	4.57	5.66	0.15	0.27	0.02	0.021	0.19	17.1	14.3	0.63	339	4.59	0.16	0.6	382	560
RS-14B	0	1.5	3.55	6.73	0.09	0.05	0.07	0.023	0.11	11.1	12.6	0.47	450	2.37	0.04	1.5	54.3	530
RS-14B	1.5	3	3.99	7.48	0.11	0.11	0.03	0.024	0.19	14	17.5	0.65	435	2.38	0.06	1.46	68.4	420
RS-14B	3	5																
RS-16B	0	2	3.46	7.39	0.08	0.13	0.03	0.023	0.06	11.6	10.4	0.67	361	1.72	0.06	1.43	54.9	600

Fraction Analyzed			-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm
Parameter	**Depth**	**Depth**	**Pb**	**Rb**	**Re**	**S**	**Sb**	**Sc**	**Se**	**Sn**	**Sr**	**Ta**	**Te**	**Th**	**Ti**	**Tl**	**U**	**V**
Units	From (ft)	To (ft)	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm
Detection Limit			0.2	0.1	0.001	0.01	0.05	0.1	0.2	0.2	0.2	0.01	0.01	0.2	0.005	0.02	0.05	1
General Method																		
Analytical Method Code			ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41
RS-01B	0	1	16.4	12.2	<0.001	0.06	0.17	1.3	0.6	1	44.6	<0.01	0.03	0.5	0.052	0.09	0.33	40
RS-01B	1	5	3.5	8.2	<0.001	<0.01	0.07	3.2	0.4	0.5	27.8	<0.01	<0.01	3.6	0.1	0.07	0.62	43
RS-01B	14	15	3.8	7.4	<0.001	<0.01	0.1	3.1	0.3	0.6	26.9	<0.01	<0.01	3.5	0.095	0.06	0.58	40
RS-01B	18	20	2.7	7.3	<0.001	0.01	0.08	2.7	0.3	1.1	34.5	<0.01	<0.01	3.5	0.089	0.06	0.53	35
RS-01B	20	25																
RS-03	5	10	3.4	3.7	0.014	0.53	0.34	2.9	2.9	1.9	47.6	0.02	0.01	1.5	0.028	0.07	1.14	25
RS-03	15	20	2.8	7.6	<0.001	0.09	0.1	2.6	0.3	0.4	30.3	<0.01	<0.01	4.1	0.096	0.06	0.87	36
RS-03	20	22	2.7	9.7	<0.001	0.05	0.12	3.2	0.9	0.8	48.4	<0.01	<0.01	3.2	0.116	0.08	0.81	38
RS-04	1	5	4.7	11	0.001	0.01	0.16	4.8	0.5	0.9	28.4	<0.01	<0.01	3.3	0.119	0.11	0.99	65
RS-04	10	15	2.7	6.2	<0.001	<0.01	0.05	2.2	0.4	0.4	27.1	<0.01	<0.01	3.7	0.085	0.04	0.59	30
RS-04	15	20	2.6	5.8	<0.001	0.03	0.05	2.2	0.3	0.4	27.6	<0.01	<0.01	4	0.085	0.04	0.86	30
RS-04	20	25	2.9	8.8	0.002	0.18	0.11	3.4	0.5	0.5	29.6	<0.01	<0.01	3.3	0.097	0.08	0.82	41
RS-05A	5	10	3.1	10.6	<0.001	<0.01	0.13	3.7	0.3	0.5	29	<0.01	<0.01	3.7	0.116	0.08	0.83	46
RS-05A	10	13	1.8	5.2	<0.001	<0.01	0.06	2.9	0.2	0.3	89.9	<0.01	<0.01	1.5	0.062	0.04	0.35	21
RS-06A	0.5	2	4.6	15.6	<0.001	0.01	0.09	3	0.5	0.6	23.1	<0.01	0.01	3.8	0.123	0.1	0.66	57
RS-06A	2	4																
RS-06A	5	7.5	3.4	10.9	<0.001	<0.01	0.12	4.1	0.4	0.6	32.1	<0.01	0.01	4	0.127	0.09	0.85	53
RS-06A	7.5	10																
RS-06A	15	19	2.8	8.9	<0.001	0.01	0.08	2.8	0.2	0.4	28.6	<0.01	<0.01	4.1	0.098	0.06	0.67	38
RS-06A	19	21																
RS-07	1	2	3.4	19.9	0.001	0.02	0.19	4.9	0.5	0.6	19	<0.01	0.02	3.6	0.133	0.13	1.07	61
RS-07	2	3																
RS-07	3	5																
RS-07	5	6																
RS-07	6	10																
RS-07R	10	12																
RS-07R	13.5	14.5																
RS-07/RS-07R	6	14.5	3.4	28.9	0.001	0.11	0.35	7.7	0.7	0.7	26.6	0.01	0.05	4.1	0.164	0.24	1.72	74
RS-08A	0	1	4.2	13	<0.001	0.01	0.16	3.8	0.5	0.6	25.6	<0.01	0.01	3.2	0.122	0.11	0.81	65
RS-08A	1	5																
RS-08A	5	11	2.9	16.4	<0.001	0.01	0.22	4.1	0.4	0.6	30.4	<0.01	0.02	4.3	0.112	0.12	0.98	45
RS-09	7	8	3	18.8	0.001	0.03	0.25	5.9	0.5	0.8	89.7	<0.01	0.01	3.4	0.125	0.15	1.23	48
RS-10	1	2	4.8	19.5	<0.001	0.01	0.13	4.3	0.6	0.8	25.9	<0.01	0.01	4	0.144	0.12	0.88	69
RS-10	2	3																
RS-10	3	5.5																
RS-10	5.5	7.5	3.1	30	<0.001	0.01	0.23	6.2	0.3	0.6	30.7	<0.01	0.01	4.6	0.165	0.2	1.29	58
RS-10	7.5	10	5	32.7	<0.001	0.02	0.27	7.2	0.3	0.7	36.3	<0.01	0.02	4.4	0.153	0.21	1.41	62
RS-10	10	14																
RS-11	0	9.5	4.2	21.6	0.001	0.06	0.37	6.5	0.8	1	29.7	<0.01	0.02	4	0.153	0.19	1.5	73
RS-11	11.5	17	4	24.9	0.001	0.08	0.41	6.8	0.6	0.7	33.4	<0.01	0.02	3.9	0.176	0.21	2.04	77
RS-11	17	25	3.5	28.3	0.002	0.2	0.47	7	1	1.4	36.2	<0.01	0.03	3.8	0.148	0.24	1.85	60
RS-11	28	31	2.5	14	0.001	0.1	0.12	3.4	0.4	0.5	28.9	<0.01	0.01	3.6	0.099	0.11	0.93	39
RS-11	31	33																
RS-12	7	9	2.6	7.8	<0.001	<0.01	0.07	2.8	0.2	0.4	26.3	<0.01	<0.01	3.1	0.09	0.06	0.58	36
RS-12	16	18	2.6	6.9	<0.001	<0.01	0.09	2.5	<0.2	0.4	25.6	<0.01	<0.01	3.3	0.083	0.07	0.77	34
RS-12	20	22	2.7	8.2	0.001	0.08	0.1	2.7	0.4	0.4	36.5	<0.01	<0.01	2.8	0.095	0.06	0.62	34
RS-13	0	1.5	5	7.7	0.001	0.02	0.15	3.2	0.6	0.7	33.3	0.01	<0.01	2.3	0.123	0.08	0.65	71
RS-13	1.5	2.5	3.3	15.1	<0.001	<0.01	0.11	4.5	0.5	0.6	27.6	<0.01	<0.01	3.7	0.152	0.12	0.77	67
RS-13	2.5	6																
RS-13	8	10	3.9	14.9	0.004	0.63	0.4	4.1	1.2	0.6	42.4	<0.01	0.03	3.4	0.133	0.16	0.85	58
RS-14B	0	1.5	13.2	15.3	<0.001	0.03	0.27	3	0.7	3.1	21.1	0.01	0.05	1.8	0.091	0.1	0.64	66
RS-14B	1.5	3	4.5	18.1	<0.001	0.02	0.18	5	0.7	1.1	21.1	0.01	0.04	3.4	0.12	0.16	0.99	65
RS-14B	3	5																
RS-16B	0	2	6.2	9.6	<0.001	0.01	0.14	3.2	0.5	1.4	20.5	0.01	0.03	3	0.126	0.07	0.61	89

Fraction Analyzed			-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm
Parameter	**Depth**	**Depth**	**W**	**Y**	**Zn**	**Zr**	**Ag**	**Al**	**As**	**Ba**	**Be**	**Bi**	**Ca**	**Cd**	**Co**	**Cr**	**Cu**	**Fe**
Units	From (ft)	To (ft)	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%
Detection Limit			0.05	0.05	2	0.5	1	0.05	50	50	10	20	0.05	10	10	10	10	0.05
General Method																		
Analytical Method Code			ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a
RS-01B	0	1	1.49	2.22	79	0.9	2	2.8	<50	500	<10	<20	1.54	<10	10	120	10	3.32
RS-01B	1	5	0.33	5.18	23	7.4	2	5.17	<50	590	<10	<20	1.65	<10	10	160	20	3.98
RS-01B	14	15	0.27	5.75	25	8.8	<1	3.87	<50	580	<10	<20	1.71	<10	10	170	30	3.68
RS-01B	18	20	1.24	5.28	26	8.1	6	4.67	<50	580	<10	<20	2.06	<10	10	160	20	3.59
RS-01B	20	25																
RS-03	5	10	1.69	7.38	18	3.1	<1	1.85	<50	210	<10	<20	1.49	<10	<10	220	210	2.68
RS-03	15	20	0.26	5.84	25	8.2	<1	4.36	<50	610	<10	<20	2.07	<10	10	160	30	3.08
RS-03	20	22	2.19	5.48	38	7.8	3	4.45	<50	530	<10	<20	2.65	<10	20	150	90	4.14
RS-04	1	5	0.28	7.31	41	4.4	<1	4.94	<50	580	<10	<20	1.74	<10	20	150	140	4.28
RS-04	10	15	0.38	4.89	19	7.4	<1	4.65	<50	650	<10	<20	1.76	<10	10	140	20	2.39
RS-04	15	20	0.34	5.12	20	7.6	<1	4.32	<50	630	<10	<20	1.82	<10	10	140	60	2.31
RS-04	20	25	0.57	5.2	32	8	<1	3.92	<50	560	<10	<20	1.91	<10	20	150	180	3.7
RS-05A	5	10	1.14	6.8	31	9.1	<1	5.05	<50	640	<10	<20	1.87	<10	10	180	60	3.81
RS-05A	10	13	0.54	2.97	42	2.7	<1	4.99	<50	320	<10	<20	4.88	<10	30	300	40	4.33
RS-06A	0.5	2	0.32	4.9	40	4.9	<1	5.3	<50	730	<10	<20	1.72	<10	20	150	20	3.71
RS-06A	2	4																
RS-06A	5	7.5	1.6	7.58	31	9.6	<1	4.86	<50	640	<10	<20	1.92	<10	10	150	30	3.81
RS-06A	7.5	10																
RS-06A	15	19	0.57	5.74	25	8.8	<1	5.03	<50	650	<10	<20	1.89	<10	10	150	20	3.27
RS-06A	19	21																
RS-07	1	2	0.59	4.95	64	4.4	<1	4.31	<50	670	<10	<20	1.49	<10	10	190	60	6.14
RS-07	2	3																
RS-07	3	5																
RS-07	5	6																
RS-07	6	10																
RS-07R	10	12																
RS-07R	13.5	14.5																
RS-07/RS-07R	6	14.5	3.21	8.71	82	9.4	<1	4.58	<50	630	<10	<20	1.72	<10	20	170	140	7.41
RS-08A	0	1	0.34	6.32	38	5.1	<1	3.99	<50	610	<10	<20	1.57	<10	10	150	40	4.78
RS-08A	1	5																
RS-08A	5	11	12.45	6.48	42	10.9	1	4.06	<50	630	<10	<20	1.76	<10	20	160	80	4.99
RS-09	7	8	1.42	6.03	63	8.1	1	5.28	<50	430	<10	<20	3.47	<10	30	180	90	5.77
RS-10	1	2	2.7	5.38	45	6.9	<1	3.52	<50	580	<10	<20	1.55	<10	20	170	30	4.08
RS-10	2	3																
RS-10	3	5.5																
RS-10	5.5	7.5	0.91	7.28	55	14.1	<1	5.39	<50	910	<10	<20	1.64	<10	10	150	40	3.77
RS-10	7.5	10	3.79	7.33	58	13.4	<1	4.66	<50	590	<10	<20	2	<10	20	170	60	6.69
RS-10	10	14																
RS-11	0	9.5	0.87	7.09	80	6.7	4	3.77	<50	500	<10	<20	2.13	<10	20	210	110	4.63
RS-11	11.5	17	0.41	8.38	92	10.9	<1	4.5	<50	550	<10	<20	2.41	<10	20	210	150	5.58
RS-11	17	25	2.92	7.3	99	8.8	5	5.62	<50	640	<10	<20	2.14	<10	20	180	650	6.6
RS-11	28	31	1	5.42	31	8.5	<1	4.54	<50	750	<10	<20	1.85	<10	10	140	30	4.33
RS-11	31	33																
RS-12	7	9	0.24	5.84	21	7.6	<1	4.36	<50	690	<10	<20	1.64	<10	10	150	20	2.84
RS-12	16	18	0.24	5.77	23	7.7	<1	5.02	<50	690	<10	<20	1.71	<10	10	130	10	2.84
RS-12	20	22	0.54	5.31	32	6.6	2	5.34	<50	630	<10	<20	2.25	<10	20	160	340	4.33
RS-13	0	1.5	0.73	6.29	44	3.3	3	4.49	<50	480	<10	<20	2.51	<10	30	180	100	5.05
RS-13	1.5	2.5	0.27	6.83	42	9.1	<1	4.49	<50	490	<10	<20	2.21	<10	20	180	60	4.81
RS-13	2.5	6																
RS-13	8	10	0.52	6.57	39	10.2	<1	5.32	<50	550	<10	<20	2.28	<10	50	200	800	7.2
RS-14B	0	1.5	0.57	4.13	58	1.3	2	4.2	<50	490	<10	<20	1.51	<10	20	150	80	5.07
RS-14B	1.5	3	1.5	5.9	53	3.4	1	4.88	<50	550	<10	<20	1.76	<10	20	150	130	5.57
RS-14B	3	5																
RS-16B	0	2	0.21	4.73	44	4.1	<1	4.86	<50	520	<10	<20	1.96	<10	20	170	30	5.15

Fraction Analyzed			-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm
Parameter	**Depth**	**Depth**	**Ga**	**K**	**La**	**Mg**	**Mn**	**Mo**	**Na**	**Ni**	**P**	**Pb**	**S**	**Sb**	**Sc**	**Sr**	**Th**	**Ti**
Units	From (ft)	To (ft)	ppm	%	ppm	%	ppm	ppm	%	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%
Detection Limit			50	0.1	50	0.05	10	10	0.05	10	50	20	0.1	50	10	10	50	0.05
General Method																		
Analytical Method Code			ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a
RS-01B	0	1	<50	0.9	<50	0.48	1860	<10	1.58	20	920	20	0.1	<50	<10	270	<50	0.28
RS-01B	1	5	<50	1.6	<50	0.67	890	<10	2.62	30	520	<20	<0.1	<50	10	450	<50	0.26
RS-01B	14	15	<50	1.4	<50	0.63	820	<10	2.73	30	510	<20	<0.1	<50	<10	420	<50	0.26
RS-01B	18	20	<50	1.7	<50	0.8	740	<10	2.8	40	490	<20	<0.1	<50	10	440	<50	0.25
RS-01B	20	25																
RS-03	5	10	<50	0.5	<50	0.35	330	30	0.65	70	480	<20	0.5	<50	<10	140	<50	0.11
RS-03	15	20	<50	1.7	<50	0.72	490	<10	2.81	30	530	20	0.1	<50	10	460	<50	0.26
RS-03	20	22	<50	1.4	<50	1.42	640	<10	2.65	90	510	<20	0.1	<50	10	400	<50	0.31
RS-04	1	5	<50	1	<50	0.81	620	<10	2.45	70	580	<20	<0.1	<50	10	420	<50	0.33
RS-04	10	15	<50	1.8	<50	0.57	370	<10	3.08	30	460	<20	<0.1	<50	<10	510	<50	0.22
RS-04	15	20	<50	1.8	<50	0.64	360	<10	3	30	460	<20	<0.1	<50	<10	470	<50	0.23
RS-04	20	25	<50	1.6	<50	1.01	540	<10	2.73	110	450	<20	0.1	<50	10	410	<50	0.32
RS-05A	5	10	<50	1.7	<50	0.8	610	<10	2.77	50	460	<20	<0.1	<50	10	470	<50	0.3
RS-05A	10	13	<50	0.7	<50	1.94	620	<10	2.42	150	240	<20	<0.1	<50	<10	350	<50	0.21
RS-06A	0.5	2	<50	1.4	<50	0.89	560	<10	2.64	40	940	<20	<0.1	<50	10	530	<50	0.36
RS-06A	2	4																
RS-06A	5	7.5	<50	1.7	<50	0.81	600	<10	2.78	40	580	<20	<0.1	<50	10	490	<50	0.34
RS-06A	7.5	10																
RS-06A	15	19	<50	1.8	<50	0.73	560	<10	2.91	30	540	20	<0.1	<50	10	500	<50	0.26
RS-06A	19	21																
RS-07	1	2	<50	1.7	<50	1.04	750	<10	2.25	70	620	<20	<0.1	<50	10	400	<50	0.31
RS-07	2	3																
RS-07	3	5																
RS-07	5	6																
RS-07	6	10																
RS-07R	10	12																
RS-07R	13.5	14.5																
RS-07/RS-07R	6	14.5	<50	1.8	<50	1.24	920	<10	2.23	80	650	<20	0.1	<50	10	370	<50	0.34
RS-08A	0	1	<50	1.4	<50	0.81	690	<10	2.34	50	550	<20	<0.1	<50	<10	410	<50	0.34
RS-08A	1	5																
RS-08A	5	11	<50	1.7	<50	0.94	930	<10	2.59	80	530	<20	<0.1	<50	10	440	<50	0.28
RS-09	7	8	<50	1.2	<50	1.93	760	<10	2.26	160	410	<20	<0.1	<50	10	350	<50	0.27
RS-10	1	2	<50	1.2	<50	0.77	670	<10	2.38	60	460	20	<0.1	<50	<10	360	<50	0.39
RS-10	2	3																
RS-10	3	5.5																
RS-10	5.5	7.5	<50	1.8	<50	0.88	600	<10	3.15	30	610	20	<0.1	<50	10	660	<50	0.25
RS-10	7.5	10	<50	1.6	<50	1.3	990	<10	2.35	70	600	<20	<0.1	<50	10	370	<50	0.25
RS-10	10	14																
RS-11	0	9.5	<50	1.1	<50	1.17	780	<10	1.89	70	490	<20	0.1	<50	10	290	<50	0.52
RS-11	11.5	17	<50	1.5	<50	1.26	850	<10	2.25	80	570	<20	0.1	<50	10	340	<50	0.53
RS-11	17	25	<50	1.9	<50	1.52	790	<10	2.26	180	520	<20	0.2	<50	10	390	<50	0.33
RS-11	28	31	<50	1.6	<50	0.84	820	<10	2.85	30	520	<20	0.1	<50	10	500	<50	0.22
RS-11	31	33																
RS-12	7	9	<50	1.8	<50	0.58	520	<10	3	20	440	<20	<0.1	<50	<10	530	<50	0.22
RS-12	16	18	<50	1.9	<50	0.58	490	<10	3	20	430	20	<0.1	<50	<10	520	<50	0.23
RS-12	20	22	<50	1.6	<50	1.22	650	<10	2.83	180	470	<20	0.1	<50	10	490	<50	0.3
RS-13	0	1.5	<50	0.9	<50	1.12	1050	<10	2.06	100	580	<20	<0.1	<50	10	310	<50	0.53
RS-13	1.5	2.5	<50	0.9	<50	1.09	760	<10	2.39	80	520	20	<0.1	<50	10	340	<50	0.47
RS-13	2.5	6																
RS-13	8	10	<50	1.3	<50	1.42	1050	<10	2.39	380	530	<20	0.5	<50	10	400	<50	0.48
RS-14B	0	1.5	<50	0.8	<50	0.9	790	<10	1.64	70	540	20	<0.1	<50	10	260	<50	0.47
RS-14B	1.5	3	<50	0.8	<50	1.04	840	<10	2.09	90	500	<20	<0.1	<50	10	310	<50	0.45
RS-14B	3	5																
RS-16B	0	2	<50	0.8	<50	1.14	720	<10	2.07	70	610	<20	<0.1	<50	10	300	<50	0.69

Fraction Analyzed			-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole
Parameter	**Depth**	**Depth**	**Tl**	**U**	**V**	**W**	**Zn**	**Initial Sample Wt**	**Volume Influent**	**Volume Effluent**	**Sample Wt After Extraction**	**Moisture Content**	**pH**	**Redox**	**Conductivity**	**Acidity (to pH 4.5)**	**Total Acidity (to pH 8.3)**	**Alkalinity**	**Fluoride**
Units	From (ft)	To (ft)	ppm	ppm	ppm	ppm	ppm	kg	mL	mL	kg	%		mV	uS/cm	mg CaCO3/L	mg CaCO3/L	mg CaCO3/L	mg/L
Detection Limit			50	50	10	50	20												
General Method								MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP
Analytical Method Code			ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a						meter	meter	meter	titration	titration	titration	
RS-01B	0	1	<50	<50	70	<50	90	1	670	569	1.104	12.8	8.07	399.9	330.5	#N/A	2.38	82.69	1.1
RS-01B	1	5	<50	<50	70	<50	30												
RS-01B	14	15	<50	<50	70	<50	40												
RS-01B	18	20	<50	<50	60	<50	40												
RS-01B	20	25																	
RS-03	5	10	<50	<50	40	<50	30	1	1650	544	2.105	62.4	6.83	392.57	249.17	#N/A	6.56	8.54	0.08
RS-03	15	20	<50	<50	60	<50	40												
RS-03	20	22	<50	<50	70	<50	50												
RS-04	1	5	<50	<50	90	<50	60												
RS-04	10	15	<50	<50	50	<50	30	1	655	551	1.104	11.6	7.44	361.32	54.29	#N/A	3.17	14.38	0.38
RS-04	15	20	<50	<50	60	<50	30	1	690	531	1.157	12.4	7.72	380.85	164.77	#N/A	2.85	37.68	0.6
RS-04	20	25	<50	<50	70	<50	40	1	625	523	1.103	7.4	7.69	385.25	253.14	#N/A	2.82	31	0.2
RS-05A	5	10	<50	<50	70	<50	50	1	655	510	1.146	13.8	7.3	379.39	70	#N/A	3.02	7.68	0.15
RS-05A	10	13	<50	<50	60	<50	50												
RS-06A	0.5	2	<50	<50	90	<50	60	1	715	518	1.198	20.8	6.88	390.62	30.48	#N/A	4.6	5.04	< 0.05
RS-06A	2	4																	
RS-06A	5	7.5	<50	<50	80	<50	50												
RS-06A	7.5	10																	
RS-06A	15	19	<50	<50	70	<50	40	1	1115	1030	1.141	11.3	5.69	473.63	45.23	#N/A	6.98	1.56	0.42
RS-06A	19	21																	
RS-07	1	2	<50	<50	90	<50	80												
RS-07	2	3																	
RS-07	3	5																	
RS-07	5	6																	
RS-07	6	10																	
RS-07R	10	12																	
RS-07R	13.5	14.5																	
RS-07/RS-07R	6	14.5	<50	<50	100	<50	100	1	1138	1028	1.155	13.9	3.71	490.23	355.82	#N/A	12.25	#N/A	< 0.05
RS-08A	0	1	<50	<50	90	<50	50												
RS-08A	1	5																	
RS-08A	5	11	<50	<50	70	<50	60	1	1250	1044	1.123	9.06	7.13	341.3	55.25	#N/A	3.98	4.62	0.48
RS-09	7	8	<50	<50	80	<50	70												
RS-10	1	2	<50	<50	100	<50	60												
RS-10	2	3																	
RS-10	3	5.5																	
RS-10	5.5	7.5	<50	<50	70	<50	70												
RS-10	7.5	10	<50	<50	80	<50	70												
RS-10	10	14																	
RS-11	0	9.5	<50	<50	110	<50	100												
RS-11	11.5	17	<50	<50	120	<50	110												
RS-11	17	25	<50	<50	90	<50	110	1	1174	1045	1.136	9.24	3.4	512.2	464.64	23.55	59.67	#N/A	< 0.05
RS-11	28	31	<50	<50	60	<50	40												
RS-11	31	33																	
RS-12	7	9	<50	<50	60	<50	30	1	1144	1013	1.125	8.87	7.05	303.22	50.77	#N/A	3.74	13.06	0.18
RS-12	16	18	<50	<50	60	<50	30												
RS-12	20	22	<50	<50	70	<50	50	1	1132	1027	1.097	7.93	7.07	353.02	350.97	#N/A	4.05	11.21	0.28
RS-13	0	1.5	<50	<50	120	<50	70	1	1525	1050	1.874	30.86	5.66	399.9	161.99	#N/A	19.72	4.94	0.14
RS-13	1.5	2.5	<50	<50	110	<50	70												
RS-13	2.5	6																	
RS-13	8	10	<50	<50	100	<50	70												
RS-14B	0	1.5	<50	<50	100	<50	80												
RS-14B	1.5	3	<50	<50	100	<50	70												
RS-14B	3	5																	
RS-16B	0	2	<50	<50	140	<50	70												

Fraction Analyzed			Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole
Parameter	**Depth**	**Depth**	**Chloride**	**Sulphate**	**Hardness $CaCO_3$**	**Al**	**Sb**	**As**	**Ba**	**Be**	**Bi**	**B**	**Cd**	**Ca**	**Cr**	**Co**	**Cu**	**Fe**	**Pb**	**Li**	**Mg**	**Mn**	**Hg**
Units	From (ft)	To (ft)	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
Detection Limit																							
General Method			MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP
Analytical Method Code				Turbidity		ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS
RS-01B	0	1	9.16	68.3	71.5	0.042	0.0005	0.0029	0.01	< 0.0002	< 0.0002	0.23	< 0.00004	15.8	< 0.0002	< 0.0001	0.003	0.02	< 0.00005	0.0099	7.76	0.0592	0.02
RS-01B	1	5																					
RS-01B	14	15																					
RS-01B	18	20																					
RS-01B	20	25																					
RS-03	5	10	2.73	93.4	102	0.13	0.0007	0.0044	0.035	< 0.0002	< 0.0002	0.18	< 0.00004	22.9	0.001	0.0007	0.011	0.12	0.00023	0.0043	10.9	0.192	< 0.02
RS-03	15	20																					
RS-03	20	22																					
RS-04	1	5																					
RS-04	10	15	1.28	8.53	18.6	0.19	0.0004	0.0014	0.0066	< 0.0002	< 0.0002	0.037	< 0.00004	3.62	0.0013	0.0012	0.028	0.31	< 0.00005	0.0016	2.32	0.236	< 0.02
RS-04	15	20	2.33	39.9	57	0.092	0.0012	0.0022	0.014	< 0.0002	< 0.0002	0.098	< 0.00004	7.97	< 0.0002	0.0006	0.0089	0.07	< 0.00005	0.0058	9.01	0.0379	< 0.02
RS-04	20	25	1.37	97.4	120	0.024	0.0011	0.0023	0.018	< 0.0002	< 0.0002	0.055	< 0.00004	17.7	< 0.0002	0.0017	0.0043	< 0.01	< 0.00005	0.0082	18.3	0.0596	< 0.02
RS-05A	5	10	1.91	21.2	23	0.067	0.0003	0.0004	0.0051	< 0.0002	< 0.0002	0.036	< 0.00004	5.86	< 0.0002	0.0005	0.0072	0.04	< 0.00005	0.0017	2.04	0.0453	0.02
RS-05A	10	13																					
RS-06A	0.5	2	1.9	3.41	7.2	0.32	< 0.0001	0.0004	0.014	< 0.0002	< 0.0002	0.03	0.00005	1.77	0.0011	0.0006	0.0054	0.06	< 0.00005	0.0006	0.66	0.0511	0.02
RS-06A	2	4																					
RS-06A	5	7.5																					
RS-06A	7.5	10																					
RS-06A	15	19	4.01	4.77	10.8	0.29	0.0003	0.0027	0.0083	< 0.0002	< 0.0002	0.012	0.00011	2.79	0.0003	0.0013	0.017	0.15	0.00028	0.0008	0.92	0.116	< 0.02
RS-06A	19	21																					
RS-07	1	2																					
RS-07	2	3																					
RS-07	3	5																					
RS-07	5	6																					
RS-07	6	10																					
RS-07R	10	12																					
RS-07R	13.5	14.5																					
RS-07/RS-07R	6	14.5	4.03	188	108	0.14	< 0.0001	0.0033	0.046	< 0.0002	< 0.0002	0.022	0.0025	17.8	0.0006	0.124	0.113	0.95	0.00058	0.02	15.3	1.09	< 0.02
RS-08A	0	1																					
RS-08A	1	5																					
RS-08A	5	11	0.74	16.5	17.7	0.091	0.0011	0.0032	0.003	< 0.0002	< 0.0002	0.013	0.00016	3.85	< 0.0002	0.0016	0.005	0.05	< 0.00005	0.0014	1.95	0.105	< 0.02
RS-09	7	8																					
RS-10	1	2																					
RS-10	2	3																					
RS-10	3	5.5																					
RS-10	5.5	7.5																					
RS-10	7.5	10																					
RS-10	10	14																					
RS-11	0	9.5																					
RS-11	11.5	17																					
RS-11	17	25	1.71	230	105	0.74	< 0.0001	0.0028	0.028	0.0008	< 0.0002	0.016	0.0066	21	0.0008	0.305	0.579	7.25	0.0014	0.028	12.9	1.31	< 0.02
RS-11	28	31																					
RS-11	31	33																					
RS-12	7	9	3.62	1.74	23.3	0.068	< 0.0001	0.0005	0.0035	< 0.0002	< 0.0002	< 0.001	0.00005	5.92	< 0.0002	< 0.0001	0.0083	0.02	< 0.00005	< 0.0002	2.07	0.0075	< 0.02
RS-12	16	18																					
RS-12	20	22	3.33	166	141	0.01	0.0004	0.0011	0.0084	< 0.0002	< 0.0002	< 0.001	0.00007	27	< 0.0002	0.0051	0.008	< 0.01	< 0.00005	0.0003	17.8	0.305	< 0.02
RS-13	0	1.5	16.6	23.6	49.6	1.49	< 0.0001	0.0016	0.028	< 0.0002	< 0.0002	0.025	0.00019	9.64	0.003	0.0028	0.072	0.77	0.0028	0.0004	6.18	0.402	< 0.02
RS-13	1.5	2.5																					
RS-13	2.5	6																					
RS-13	8	10																					
RS-14B	0	1.5																					
RS-14B	1.5	3																					
RS-14B	3	5																					
RS-16B	0	2																					

Appendix B
Initial Summary of MWMP Data Provided to Barr Engineering

	Solids	Leaches	Stat	Whole MWMP pH	Whole MWMP Redox	Whole MWMP Conductivity	Whole MWMP Total Acidity (to pH 8.3)	Whole MWMP Alkalinity	Whole MWMP Fluoride	Whole MWMP Chloride	Whole MWMP Sulphate	Whole MWMP Hardness CaCO3	Whole MWMP Al	Whole MWMP Sb	Whole MWMP As	Whole MWMP Ba	Whole MWMP Be	Whole MWMP B	Whole MWMP Cd	Whole MWMP Ca	Whole MWMP Cr	Whole MWMP Co	Whole MWMP Cu	Whole MWMP Fe	Whole MWMP Pb	Whole MWMP Mg	Whole MWMP Mn	Whole MWMP Mo	Whole MWMP Ni	Whole MWMP P	Whole MWMP K	Whole MWMP Se	Whole MWMP Ag	Whole MWMP Na	Whole MWMP Tl	Whole MWMP Ti	Whole MWMP V	Whole MWMP Zn
					mV	uS/cm	mg CaCO3/L	mg CaCO3/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
Peat	6	2	P5	6.7715	392.9365	166.349	7.218	5.12	0.083	3.4235	27.09	52.22	0.198	-0.00006	0.00174	0.02835	-0.0002	0.03275	-0.0000285	10.303	0.0011	0.000805	0.01405	0.1525	0.0003585	6.416	0.2025	0.00152	0.00737	0.052	1.734	0.00064	-0.00005	4.3095	0.000072	0.001875	0.002115	0.00425
			Median	6.245	396.235	205.58	13.14	6.74	0.11	9.665	58.5	75.8	0.81	0.0003	0.003	0.0315	-0.0002	0.1025	0.000075	16.27	0.002	0.00175	0.0415	0.445	0.001515	8.54	0.297	0.0053	0.0143	0.07	4.11	0.001	-0.00005	5.025	0.00009	0.00255	0.00315	0.0065
			P95	5.7185	399.5335	244.811	19.062	8.36	0.137	15.9065	89.91	99.38	1.422	0.00066	0.00426	0.03465	-0.0002	0.17225	0.0001785	22.237	0.0029	0.002695	0.06895	0.7375	0.0026715	10.664	0.3915	0.00908	0.02123	0.088	6.486	0.00136	-0.00005	5.7405	0.000108	0.003225	0.004185	0.00875
Unsaturated Mineral	13	4	P5	7.283	345.109	32.957	3.116	4.662	0.1665	0.856	4.719	8.25	0.0694	-0.00006	0.0004	0.00321	-0.0002	0.0147	-0.000031	1.978	-0.0002	0.00051	0.00504	0.041	-0.00005	0.789	0.04588	0.000195	0.00155	-0.03	0.744	-0.0002	-0.00005	3.747	-0.00002	0.00113	0.00042	0.0021
			Median	7.13	379.39	55.25	3.98	5.04	0.315	1.9	16.5	17.7	0.091	0.0003	0.0004	0.0051	-0.0002	0.03	0.00005	3.85	-0.0002	0.0006	0.0054	0.05	-0.00005	1.95	0.0511	0.0024	0.0029	-0.03	0.78	-0.0002	-0.00005	4.26	-0.00002	0.0023	0.0006	0.003
			P95	6.905	389.497	68.525	4.538	7.416	0.4635	1.909	20.73	22.47	0.2971	0.00102	0.00292	0.01311	-0.0002	0.0354	0.000149	5.659	0.00097	0.0015	0.00702	0.059	-0.00005	2.031	0.09961	0.01257	0.00326	-0.03	1.23	0.00052	-0.00005	5.331	0.000025	0.00257	0.0006	0.0057
Saturated Mineral	19	6	P5	7.965	367.179	47.948	2.512	#N/A	0.236	1.307	5.898	13.14	0.0294	-0.0001	0.00164	0.00711	-0.0002	0.0132	-0.00004	3.039	-0.0002	0.00011	0.00339	-0.001	-0.00005	1.34	0.04429	0.000332	0.00201	-0.03	1.257	0.00055	-0.00005	3.691	-0.00002	0.00075	-0.0002	-0.0004
			Median	7.44	399.9	253.14	3.17	#N/A	0.42	2.33	68.3	71.5	0.14	0.0004	0.0027	0.014	-0.0002	0.037	-0.00004	15.8	0.0003	0.0013	0.017	0.15	-0.00005	9.01	0.116	0.0274	0.019	-0.03	4.45	0.0019	-0.00005	7.19	-0.00002	0.0016	0.0014	0.003
			P95	3.493	505.609	431.994	45.444	#N/A	1	7.621	217.4	116.4	0.605	0.00117	0.00318	0.0406	0.0005	0.1904	0.00537	20.04	0.00115	0.2507	0.4392	5.36	0.001154	17.4	1.244	0.03191	2.1812	0.04	9.827	0.00377	0.000965	37.07	0.000082	0.00614	0.00247	0.862
OB with Mineralized Rock	3	2	P5	7.069	305.71	65.78	3.7555	11.3025	0.185	3.3445	9.953	29.185	0.0129	-0.000075	0.00053	0.003745	-0.0002	-0.001	0.000051	6.974	-0.0002	0.00016	0.008015	-0.0085	-0.00005	2.8565	0.022375	0.00539	0.00456	-0.03	0.7285	-9E-05	-0.00005	2.334	-0.00002	-9.5E-05	-0.000165	0.00205
			Median	7.06	328.12	200.87	3.895	12.135	0.23	3.475	83.87	82.15	0.039	0.00015	0.0008	0.00595	-0.0002	-0.001	0.00006	16.46	-0.0002	0.0025	0.00815	0.005	-0.00005	9.935	0.15625	0.0188	0.0384	-0.03	1.345	0.0009	-0.00005	6.96	-0.00002	0.00085	0.00015	0.0025
			P95	7.051	350.53	335.96	4.0345	12.9675	0.275	3.6055	157.787	135.115	0.0651	0.000375	0.00107	0.008155	-0.0002	-0.001	0.000069	25.946	-0.0002	0.00484	0.008285	0.0185	-0.00005	17.0135	0.290125	0.03221	0.07224	-0.03	1.9615	0.00189	-0.00005	11.586	-0.00002	0.001795	0.000465	0.00295
All	41	14	P5	7.8425	327.972	40.0675	2.666	#N/A	0.11	1.091	2.8255	9.54	0.0191	-0.0001	0.0004	0.003325	-0.0002	-0.001	-0.00004	2.433	-0.0002	-0.0001	0.003845	-0.01	-0.00005	0.829	0.02726	0.000171	0.00119	-0.03	0.712	-0.0002	-0.00005	2.925	-0.00002	0.00032	-0.0002	0.0003
			Median	7.06	387.935	163.38	4.015	#N/A	0.28	2.53	31.75	53.3	0.111	0.00035	0.00225	0.012	-0.0002	0.0275	0.00005	8.805	0.00005	0.00125	0.0086	0.065	-0.00005	6.97	0.1105	0.008	0.0128	-0.03	1.85	0.001	-0.00005	5.62	-0.00002	0.00185	0.00065	0.003
			P95	3.6015	497.9195	393.907	33.7025	#N/A	0.85	11.764	202.7	127.35	1.0025	0.001135	0.003685	0.03885	0.00015	0.1975	0.003935	24.335	0.001895	0.18735	0.2761	3.155	0.00189	17.975	1.167	0.03318	1.2726	0.077	8.4475	0.003735	0.0004575	25.135	0.0001035	0.00502	0.00313	0.526
			Max	3.4	512.2	464.64	59.67	#N/A	1.1	16.6	230	141	1.49	0.0012	0.0044	0.046	0.0008	0.23	0.0066	27	0.003	0.305	0.579	7.25	0.0028	18.3	1.31	0.0337	2.96	0.09	11.6	0.0038	0.0014	47.3	0.00011	0.0071	0.0043	1.15

Attachment 2-22

SRK Consulting. Overburden Pebble Chemical Analysis – DRAFT. Memorandum to Jim Scott, PolyMet. June 25, 2009.

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Memo

To:	Jim Scott, PolyMet	**Date:**	June 25, 2009
cc:	Miguel Wong, Barr Engineering	**From:**	Stephen Day Richard Patelke, PolyMet
Subject:	Overburden Pebble Chemical Analysis DRAFT	**Project #:**	1UP005.001

1 Background

On October 26, 2008, SRK reported results of analysis of the fine fractions (less than 2 mm) of overburden samples collected from the NorthMet Project in January, 2008[1]. Analysis of the coarse fractions was delayed so that the lithology of pebbles in the overburden could be enumerated. Both enumeration of lithologies and sulfur analysis of the pebbles have now been completed. Results are provided in Attachment 1 and described below. Samples for analysis of pebbles were selected by MDNR in consultation with Richard Patelke, PolyMet. The February 23, 2009 memorandum prepared by PolyMet[2] provides details of the pebble counting procedure.

2 Results and Discussion

2.1 Pebble Counts

As reported in February, 2009, pebble counts were completed on thirty-three intervals from sixteen drill holes. The counting procedure recognized seven lithologic types with numerous subsets. The seven major types were: granite, greenstone, banded iron-formation (Biwabik Iron Formation, or BIF locally), Virginia Formation, Duluth Complex, unidentified , and other. It was decided by MDNR to focus sulfur assays on granite, iron-formation, Virginia Formation, and Duluth Complex materials as they made up the bulk of all intervals.

In the counting process a certain amount of fine-grained, black, non-magnetic rock with no recognizable igneous texture was sorted out. As noted in the February 2009 memo, this material could most likely be either slaty iron-formation or Virginia Formation. It was decided in consultation with MDNR and SRK to classify this material as Virginia Formation, therefore the pebble counts may overstate the amount of Virginia Formation and understate the amount of iron-formation.

Also, it should be recognized that samples from rotosonic drilling are not “pebbles” in the sense one might separate pebbles from a sample of glacial drift, but include rock chips created by drilling through cobbles and boulders and hence are not indicative of the actual particle size distribution in the drift. Samples with a large proportion of one rock type or another probably reflect drilling through larger pieces of rock.

[1] SRK Consulting 2008. Results of Analysis from Overburden Drilling Program NorthMet Project – Doc Id GC05 – DRAFT. SRK Project 1UP005.001. October 26, 2006.
[2] PolyMet. 2009. Pebble Count Results from Rotosonic Soil Borings at Northmet Mine Site In January 2008.February 23, 2009.

Table 1 summarizes the range in weight fractions of each four major rock types. For example, it shows that on average, the 33 samples contained 32% granite pebbles, 14% BIF, 40% Virginia Formation and 14% Duluth Complex (total 100%). The other statistics provide an indication of the range in content. On average, Virginia Formation was the most abundant followed by granite and about equal on average banded iron formation and Duluth Complex. Duluth Complex showed the widest spread.

Table 1. Range of Weight Fractions of Pebbles of Each Rock Type

Statistic	Granite	BIF	Virginia Formation	Duluth Complex
Min	2%	2%	4%	0%
P5	7%	3%	7%	0%
Mean	32%	14%	40%	14%
P95	62%	32%	71%	72%
Max	88%	43%	76%	89%

2.2 Sulfur Analyses

Sulfur concentrations are summarized in Table 2. The lowest average sulfur concentrations were found in granite which averaged 0.04%. The highest reported sulfur concentration in granite was 0.39%. This occurred in a small number of granite pebbles from 8 to 10 feet depth at RS13. This sample contained 87% Duluth Complex pebbles. RS13 is located above mineralized Duluth Complex Unit 1. Mineralized Duluth Complex rock was observed at this location and Duluth Complex pebbles contained 2.8% sulfur. It is possible that the apparently elevated sulfur concentration in granite may be due to some mis-classification of Duluth Complex rock which influenced the granite sulfur content, however, trace pyrite may often be found in granitic rock

Duluth Complex showed the greatest range in sulfur content but the median sulfur content was 0.1%. Highest sulfur contents were observed above Unit 1or near the contact of Unit 1 with Unit 2. Outside the elevated sulfur concentrations associated with Unit 1, concentrations were below 0.3% with the exception of 0.4% at RS03 (15 to 20 feet depth). The range of sulfur concentrations in Duluth Complex samples tended to increase as bedrock was approached (Figure 1) perhaps indicating the effect of local mineralization in the Duluth Complex, and homogenization of sulfur variations in shallower samples.

Table 2. Distribution of Sulfur Concentrations in Pebbles of Each Rock Type

Statistic	Units	Granite	BIF	Virginia Formation	Duluth Complex
n		18	17	19	16
P5	%, S	0.01	0.01	0.04	0.01
Median	%, S	0.01	0.02	0.14	0.11
P95	%, S	0.14	1.0	0.43	1.7
Max	%, S	0.39	1.1	0.77	2.8

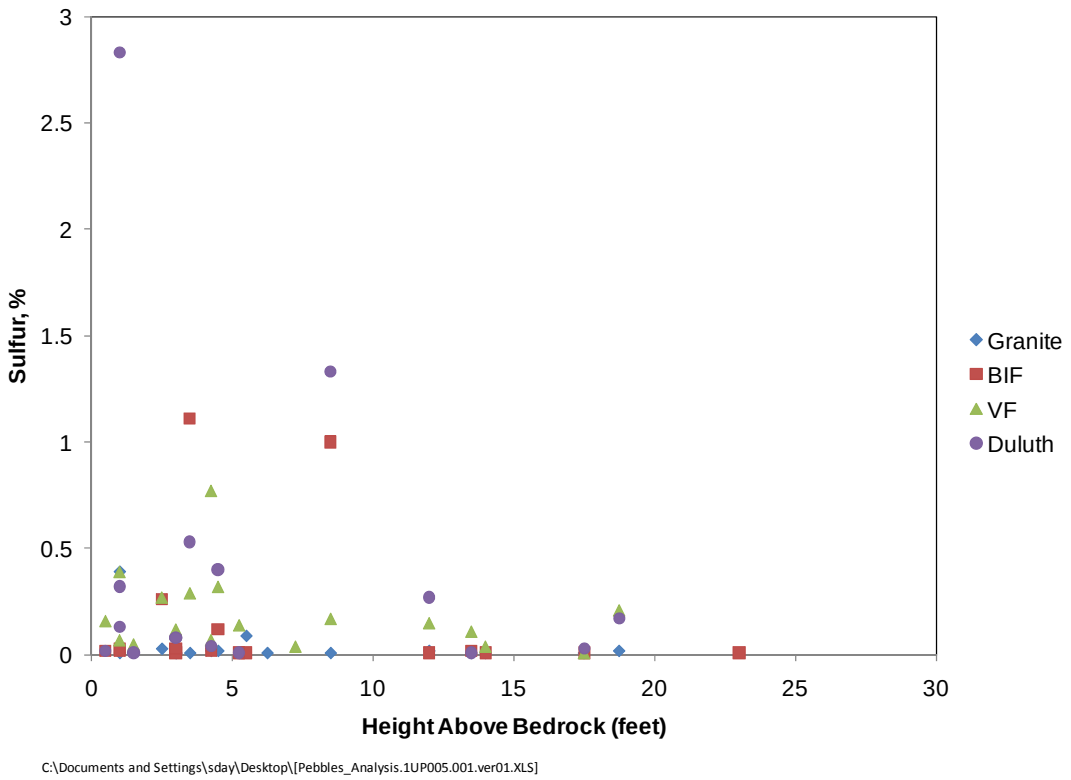


Figure 1. Sulfur Concentrations in Pebbles as a Function of Depth

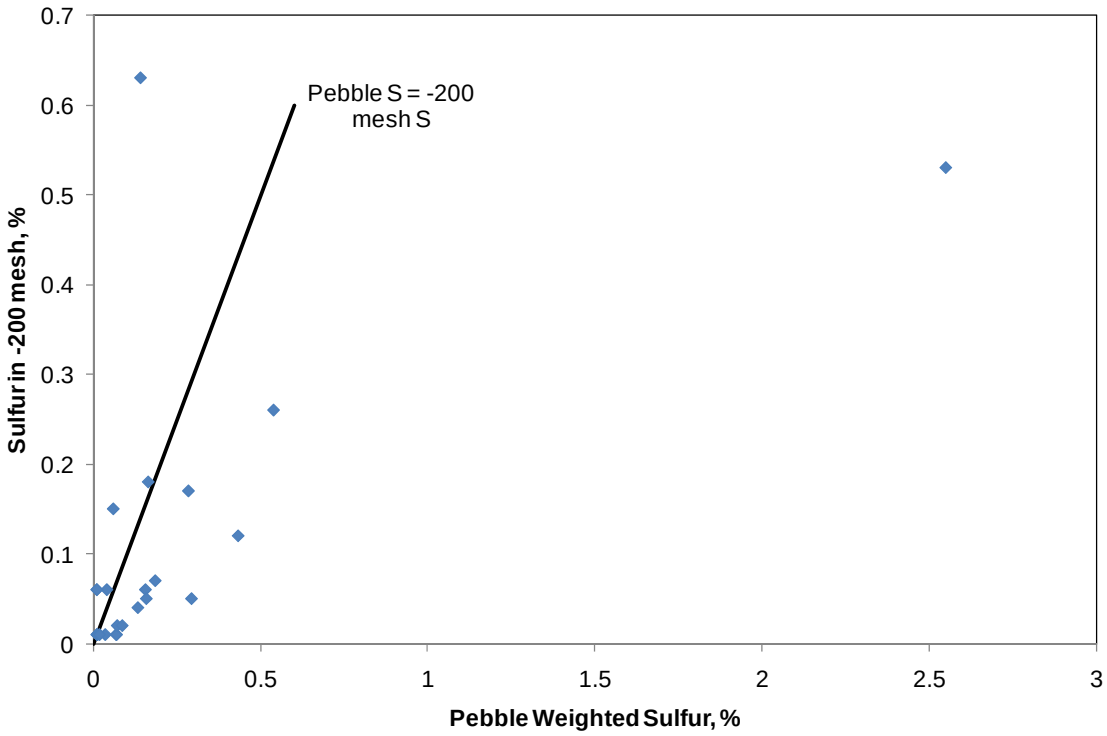
Virginia Formation pebbles showed the highest median sulfur concentration (0.14%) with sulfur concentrations up to 0.77%. This is consistent with elevated sulfur concentrations in the Virginia Formation bedrock though not the highly elevated levels associated with pyrrhotite-rich zones. Again the range of sulfur concentrations increased with proximity to bedrock (Figure 1).

Banded iron formation pebbles had a very low median sulfur concentration of 0.02% but a range extending to 1.1%. The sulfur distribution was distinctly bimodal with three of the 17 samples containing more than 0.2% sulfur. Two samples containing 1% or more were both from within 10 feet bedrock at RS-04. At this location near the contact between Duluth Complex Unit1 and higher, Duluth Complex pebbles contained comparable sulfur concentrations to BIF at this location and Virginia Formation pebbles were only moderately above the median.

The agreed upon classification of fine-grained, non-magnetic, non-igneous textured, black rock as Virginia Formation may have increased the Virginia Formation count relative to the iron formation count.

Sulfur concentrations in pebbles were compared to sulfur concentrations in the finer size fractions by calculating weighted pebble sulfur concentrations from the mass of each pebble lithology. The calculation was performed only for samples in which at least 90% of the pebbles were analyzed. The selection of pebble samples for analysis did not result in all pebbles in all samples being analyzed.

Sulfur concentrations in the -200 mesh fraction were correlated with weighted pebble sulfur concentrations with a tendency for sulfur concentrations in the -200 mesh fraction to be lower than the pebbles (Figure 2). In contrast the -2 mm+200 mesh samples were not correlated with pebble sulfur concentrations and were lower than the pebble sulfur content.



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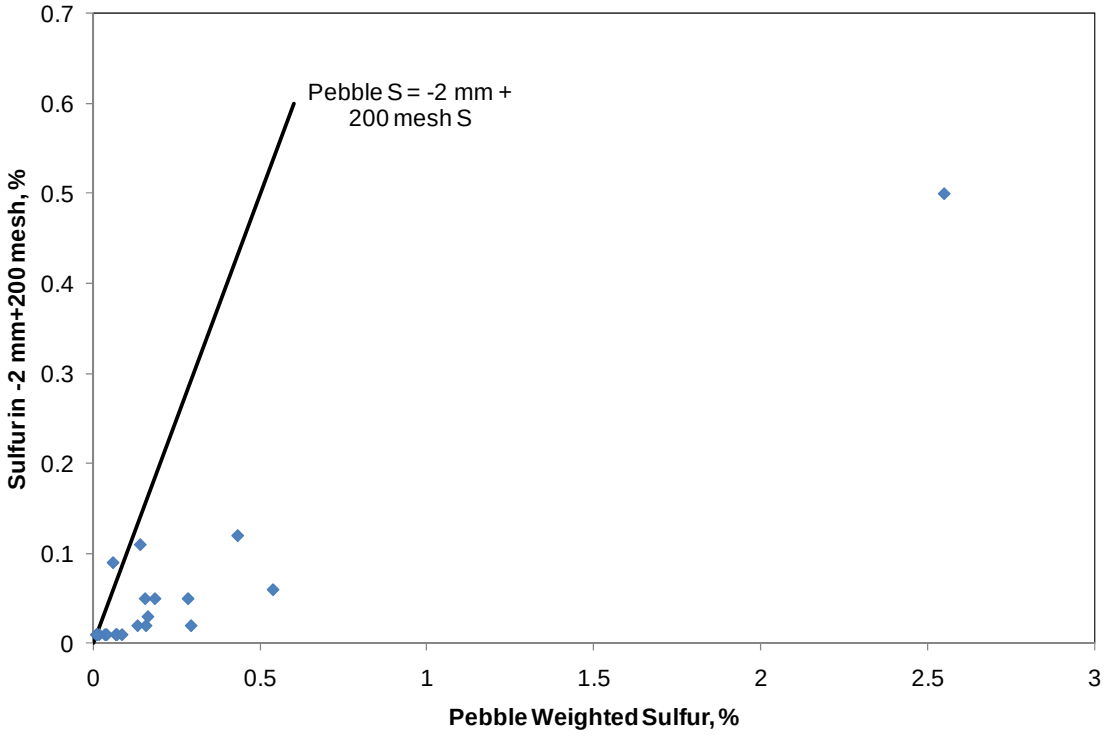

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Figure 2. Comparison of Sulfur Concentration in Pebbles with Finer Size Fractions

SRK (2008) presented the hypothesis that elevated sulfur concentrations in finest size fraction were due to the presence of iron sulfide formed under saturated reducing conditions. The reductant was thought to be provided by organics leached from peat deposits. Sulfate to generate sulfide would be provided by oxidation of sulfide-mineralized rock in the overburden profile. Overall, it is expected that fine-grained secondary iron sulfide would be precipitated on the surface of soil particles. In this model, secondary sulfide would occur at greatest concentrations in the finest fractions where the surface area to mass ratio for soil particles is greatest. Secondary sulfides would also precipitate on pebbles but their overall contribution to overall sulfur content of the pebble would be expected to be negligible compared to primary sulfide in the rock itself.

The results obtained from the pebble analysis confirmed the observation of mineralized rock during drilling and appears to indicate that sulfide mineralization is correlated with the presence of secondary sulfides in the fine fraction. The observations appear to be consistent with the expected conditions in the overburden:

- The finest fraction (-200 mesh) is depleted of primary sulfide by natural weathering processes. Sulfur content of this fraction reflects residual primary sulfide and localized formation of secondary sulfides.
- The intermediate fraction (-2 mm + 200 mesh) is also depleted of primary sulfides but the lower surface area (relative to mass) results less effect on overall sulfide content from secondary sulfides.
- The pebble fraction retains the original sulfur content of the rock because internal sulfide particles are protected from oxidation.

3 Conclusions

Analysis of pebbles showed:

- Lithologies present were consistent with four regional and local rock types: granite, banded iron formation, Virginia Formation, and Duluth Complex.
- Elevated 95th percentile sulfur concentrations in pebbles reflected proximity mainly to Unit 1 in the Duluth Complex.
- Elevated median sulfur concentrations were found in Virginia Formation.
- Proximity to bedrock increased the variability of primary sulfur concentrations in pebbles due to the presence of variable sulfide concentrations in the Duluth Complex.
- Sulfur concentrations in the pebbles higher in the overburden profile appeared to reflect homogenization of the effects of bedrock in overburden by mixing during glacial erosion and deposition. Sulfur concentrations were lower and less variable.
- Primary sulfur concentrations in pebbles appeared to be linked to secondary sulfide in the fine fractions. This is most apparent at depth under saturated conditions.

Overall the results of pebble analyses support the overburden management approach. Segregation of deeper saturated overburden should result in separate management of overburden potentially containing elevated secondary sulfide content and any pebbles containing primary mineralization. Unsaturated shallower overburden is not expected to contain secondary sulfides. Also pebbles higher in the overburden profile are expected to reflect average bedrock content with less influence from locally heterogeneous mineralization in the bedrock.

Attachment 1 – Analytical Results

			Rock Type Weights				Total Sulfur Concentrations			
Hole ID	**From**	**To**	**Granite**	**BIF**	**Virginia Formation**	**DC**	**Granite**	**BIF**	**Virginia Formation**	**DC**
	feet	feet	g	g	g	g	%	%	%	%
RS-01B	1	5	228	329	182	19	0.01	0.01	0.01	0.03
RS-01B	14	15	390	140	93	10	-	-	-	-
RS-03	15	20	311	141	273	9	0.02	0.12	0.32	0.4
RS-03	20	22	271	460	459.5	271	0.01	0.03	0.39	0.13
RS-04	1	5	301	259	227	0	-	0.01	-	-
RS-04	10	15	492	32	35	0	0.01	-	-	-
RS-04	15	20	189	36	41	32	0.01	1.00	0.17	1.33
RS-04	20	25	153	99	199	269	0.01	1.11	0.29	0.53
RS-05A	5	10	331	124	123	2	0.09	0.01	-	-
RS-05A	10	13	73	86	103	2208	-	-	0.05	0.01
RS-06A	0.5	4	619	41	441	88	-	-	-	-
RS-06A	5	10	494	208	276	22	-	0.02	0.11	0.01
RS-06A	15	21	835	192	243	118	0.01	0.01	0.12	0.08
RS-07	1	6	541	650	2832	281	-	-	-	-
RS-07R	6	14.5	768	520	3157	123	0.02	0.02	0.77	0.04
RS-08A	2.5	5	368	645	849	447	-	-	0.04	-
RS-08A	5	11	88	120	261	303	0.01	0.03	0.1	0.08
RS-09	7	8	193	19	607	127	-	0.02	0.16	0.02
RS-10	1	5.5	856	52	685	0	-	-	-	-
RS-10	5.5	7.5	167	31	562	0	-	-	-	-
RS-10	7.5	14	611	578	3165	1051	0.01	0.01	0.14	0.01
RS-10B	0	2	7	2	10	0	-	-	-	-
RS-11	0	9.5	127	67	664	10	-	-	-	-
RS-11	11.5	17	459	109	1413	110	0.02	-	0.21	0.17
RS-11	17	25	96	75	641	119	0.02	0.01	0.15	0.27
RS-11	28	33	755	145	1242	56	0.03	0.26	0.27	-
RS-12	7	9	270	140	144	2	0.01	0.01	0.04	-
RS-12	16	18	209	55	154	2	-	-	-	-
RS-12	20	22	201	100	159	737	-	0.02	-	0.32
RS-13	1.5	6	167	65	260	5	0.01	-	-	-
RS-13	8	10	41	53	142	1579	0.39	-	0.07	2.83
RS-14B	0	1.5	43	16	116	0	0.03	-	0.07	-
RS-14B	1.5	5	694	395	1212	90	-	-	-	-

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Notes:

"-" Not analyzed.

Attachment 2-23

SRK Consulting. Results of Analysis from Overburden Drilling Program – Update for March 2010 Test Pit Program – DRAFT, NorthMet Project. Memorandum to Stuart Arkley, MDNR. August 13, 2010.

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Memo

To:	Stuart Arkley, MDNR	**Date:**	August 13, 2010
cc:	Jim Scott, PolyMet Miguel Wong, Barr Engineering	**From:**	Stephen Day Christie Kearney, Barr Engineering
Subject:	Results of Analysis from Overburden Drilling Program – Update for March 2010 Test Pit Program NorthMet Project – DRAFT	**Project #:**	1UP005.001

1 Introduction

This memorandum provides chemical results from two analytical programs for overburden samples collected from the NorthMet Project in the area of the proposed mine site. The memorandum provides conclusions on the main factors controlling differences in leaching characteristics of the overburden and estimates of contact water chemistry.

The original Overburden Geochemical Characterization Plan was provided to the MDNR on February 22, 2008[1] and the accompanying analytical plan was provided to the MDNR on March 18, 2008[2]. This program involved drilling to characterize the full overburden profile from surface to bedrock. Both plans were reviewed and accepted by the MDNR with the understanding that subsequent characterization might be required prior to and during excavation of overburden material. Results of this program were provided in two memoranda dated October 16, 2008 (fine particle solids and leachate chemical analysis)[3] and June 25, 2009 (overburden pebble counts and analysis)[4].

An opportunity to collect additional overburden samples arose as a result of a USFS requirement to dig sumps to contain drilling fluids from bedrock drilling occurring as part of further resource delineation by PolyMet. As a result, a sump spoil sampling program was designed and implemented by Barr Engineering in cooperation with MDNR to provide additional data on the geochemical characteristics of unsaturated overburden. The details of the program were developed through several emails between Barr and MDNR.

Results of this second program have been compared to the findings from the original program rather than combining results into a single dataset because the sampling methods and investigation scopes were different. The sampling locations were pre-determined by the bedrock drilling locations rather than being designed to characterize particular features of the overburden.

[1] SRK Consulting, PolyMet Mining and Barr Engineering. 2008. Overburden Geochemical Characterization Plan in Support of EIS – DRAFT NorthMet Project. February 22, 2008.
[2] SRK Consulting, PolyMet Mining and Barr Engineering. 2008. Analysis of Samples from Overburden Drilling Program, NorthMet Project – DRAFT. March 18, 2008.
[3] SRK Consulting. 2008. Results of Analysis from Overburden Drilling Program. Doc IDGC05. October 16, 2008.
[4] SRK Consulting. 2009. Overburden Pebble Chemical Analysis – DRAFT. June 25, 2009.

2 January 2008 Drilling Program

2.1 Summary of Field Programs

A total of 225 feet of drilling was completed for geochemical characterization of overburden. The following summary of the field observations was provided in the analytical plan.

As expected, drift in the area has complex lithology. The majority of intervals (75%) were characterized as dominantly sandy till with varying quantities of gravels and silts. A few intervals were dominated by gravels (21 feet). Dominantly silt intercepts were unusual (two intervals totaling 5 feet). The total intersection of peaty materials was 25 feet.

The main feature of the overburden profile was the presence of oxidized (brown) and unoxidized (olive and grey) tills corresponding roughly to the presence of the water table. Of the thirteen mechanically-drilled holes, only two were in a fully unsaturated profile (holes 10 and 14) while the others were either completely saturated (four holes) or were unsaturated near surface and saturated below the saturated elevations (seven holes). Measurements of oxidation reduction potential (ORP) in field rinse tests showed a strong negative correlation with depth. Near surface samples had typical ORPs of 100 to 300 mV, whereas deeper samples had ORPs below 100 mV and as low as -200 mV. Loggers recorded the presence of what appeared to be secondary iron sulfides in the chemically-reduced overburden. Visual observations were supported by the smelled evolution of hydrogen sulfide gas when 10% hydrochloric acid was applied.

Surface tills appeared to be weakly acidic (rinse pHs less than 6.5) as shown by the correlation of rinse pH with depth. Deeper tills had rinse pHs greater than 6. The presence of acidic conditions generally did not correlate with conductivity) indicating that the variation of rinse pH was not significantly related to the presence of acidic salts as would be produced by oxidation of sulfide minerals. In fact, conductivities for samples showing rinse pHs less than 5.5 (the typical pH of deionized water) were mostly low. The exceptions were two samples with conductivity above 100 µS/cm and pHs below 5.6. These measurements did not correspond to the presence of mineralized rock.

As described below, some samples of the deeper tills became acidic prior to laboratory testing.

Other than brown coatings related to weathering of iron-bearing components of the overburden, chemical precipitates were uncommon. White cement and lenses were observed in drill hole 10, but they did not react with dilute hydrochloric acid.

The overburden rarely reacted with hydrochloric acid which indicated low concentrations of carbonate minerals.

2.2 Sample Selection and Analysis

A discussion of sample selection, as well as a sample analysis list, was provided in the analytical plan. The analytical plan was completed for the size fractions finer than 2 mm (-2 mm+74 µm and -74 µm), and the meteoric water mobility procedure on splits of the whole samples. The pebble counts are in progress (+4 mm size fraction) and once completed the pebbles will be re-combined with the -4+2 mm fraction for acid-base accounting and metal analysis on the +2 mm fraction.

The laboratory reported that 36 of the 37 samples submitted for analysis had sufficient pebbles (more than 180) for counting.

2.3 Results

2.3.1 Data Assessment Approach

The Geochemical Characterization Plan was designed to address various factors that could influence the geochemical characteristics of overburden, which included:

- Geological sources distant from the project area where the glacial ice originated.
- Underlying bedrock geology.
- Distance from mineralized bedrock (mainly Unit 1 of the Duluth Complex and the Virginia Formation).
- Glacial and periglacial deposition environment.
- Groundwater level during deposition.
- Current position of the water table (degree of saturation).

Field observations during drilling indicated that oxidation-reduction conditions in the overburden varied significantly and potentially exerted an important control on leaching potential of the overburden. This factor was also considered in the analysis.

The following sections provide descriptive statistics and discussion with respect to these factors.

The complete dataset is attached as Appendix A. Metal analysis was performed using two acid digestion methods (aqua regia and four acid) both followed by determination of concentrations using ICP. The difference between results for the two analytical methods was insignificant though in the case of nickel slightly higher concentrations were reported for the stronger four acid digestion (Figure 1) probably reflecting the more complete digestion of silicate minerals by this digestion. Therefore, subsequent review of the data considered results from the four-acid digestion.

The summaries below focus on the main parameters believed to be indicative of the influence of mineralized bedrock on the overburden (sulfur, copper and nickel).

2.3.2 Entire Dataset

Effect of Particle Size

Graphs comparing sulfur, copper and nickel concentrations in -74 μm and -2 mm+74 μm fractions are presented in Figure 2.

For sulfur, concentrations in the finer fraction were greater than in the coarse fraction for concentrations above 0.05% with the exception of two samples which showed nearly equivalent sulfur concentrations in the two fractions. Concentrations in the two fractions were not well correlated.

For copper, concentrations in the two fractions were correlated but were also mostly greater in the fine fraction. Nickel showed similar results except that at higher concentrations, the correlation was absent (Figure 2). Nickel concentrations were relatively stable below 200 ppm in the coarse fraction but increased to near 500 ppm for two samples. These samples also contained the highest sulfur concentrations in mineral overburden. The highest sulfur concentration in a peat (0.61%) did not contain elevated nickel concentrations.

The significant difference in sulfur concentrations for the size fractions may be linked to the observation of secondary sulfide minerals in the overburden rather than differences in the particle size distribution of primary sulfur in the overburden. There is no particular reason to expect sulfur to be concentrated in the fine fraction, and it would be expected that any fine-grained sulfur produced

during deposition of the till would be oxidized following deposition. The base level sulfur concentrations in the coarser fraction (below 0.05%) are typical of background sulfur concentrations in rocks whereas the fine fraction concentrations indicated enrichment of sulfur.

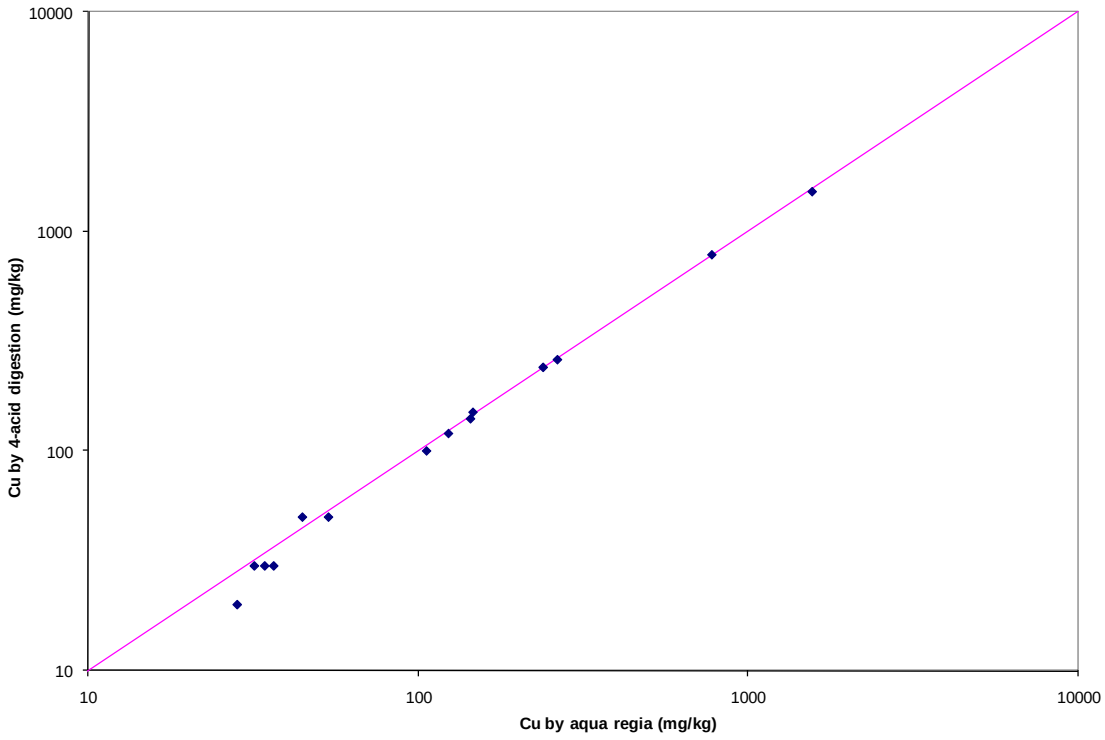
Overall Dataset Distribution

Distributions for the selected parameters are presented in Table 1. Data for many parameters were strongly positively skewed hence the median provides an indication of central or typical values while the 95th percentile and maximum values indicate extreme values.

Measurements of oxidation reduction potential (ORP) had 5th and 95th percentile values of -90 and 290, respectively. As was discussed in the analytical plan, ORP showed a strong negative correlation with depth. Near surface samples had typical ORPs of 100 to 300 mV, whereas deeper samples had ORPs below 100 mV and as low as -200 mV.

The statistics indicate that sulfate, which would likely be formed by oxidation of sulfur occurred at very low concentrations. Sulfur is mainly expected to occur as sulfide with the exception of peat for which sulfur may also occur in organic form.

Neutralization potentials were low and carbonate was rarely detected (Appendix A) indicating that rapidly acid consuming minerals were present at low concentrations.



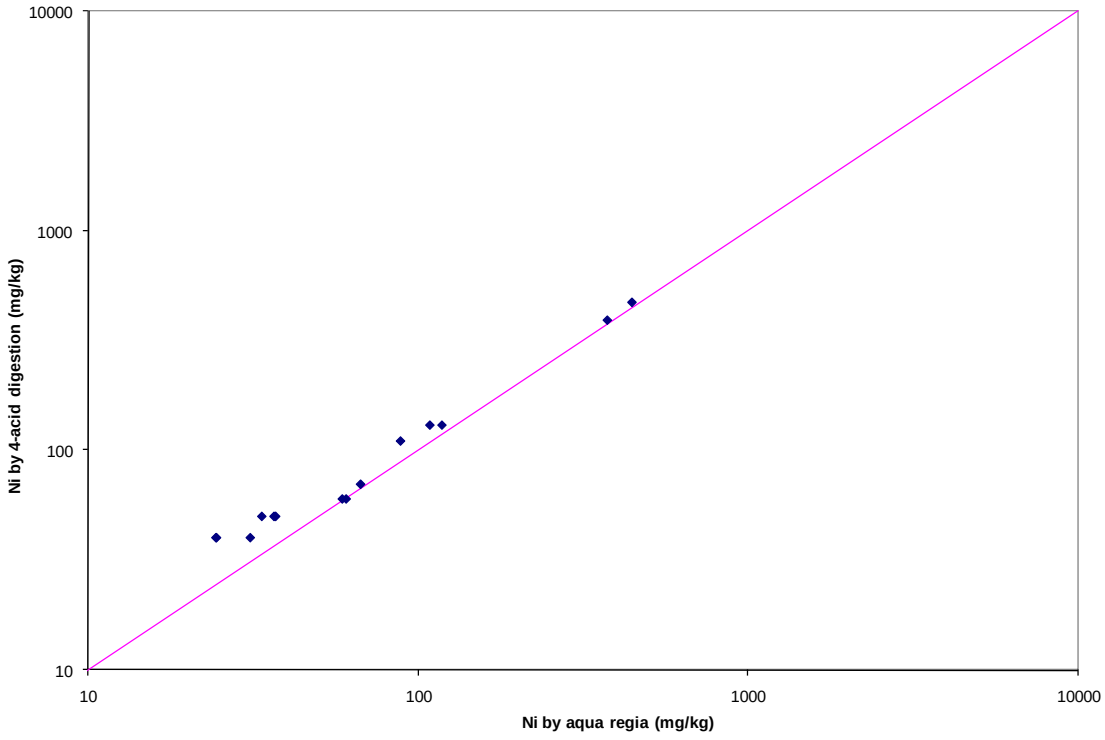
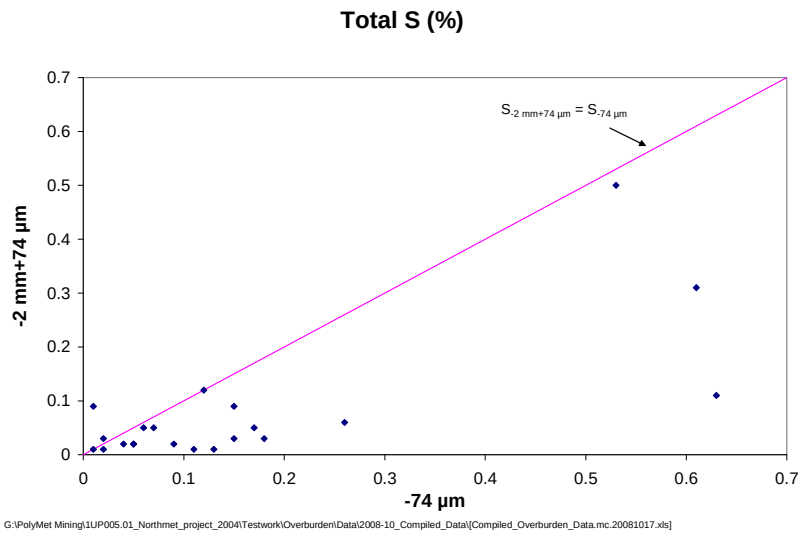
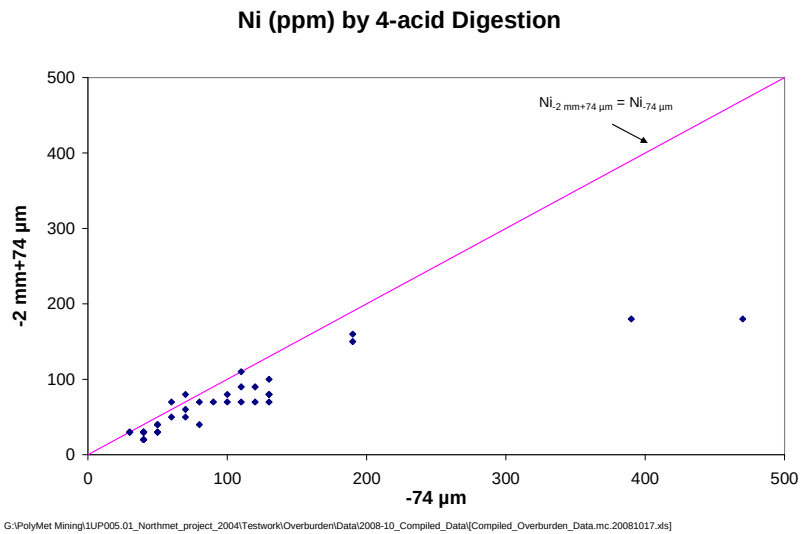
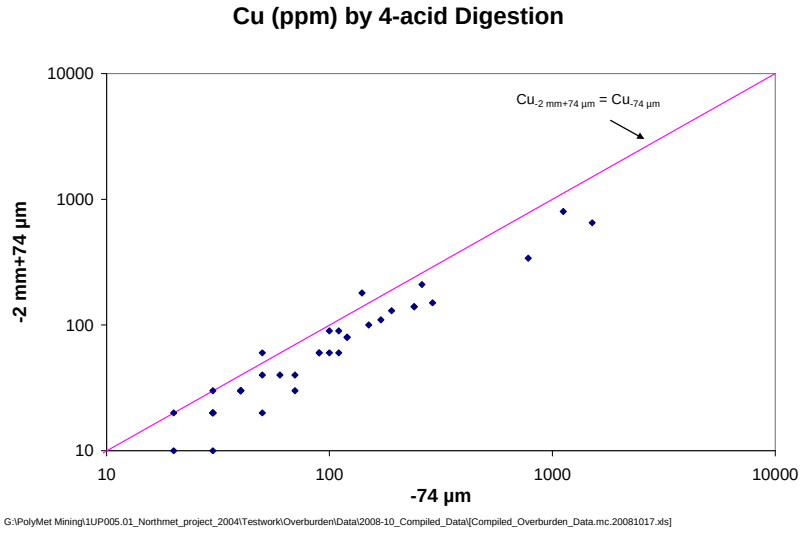


Figure 1: Comparison of Copper and Nickel Concentrations by Different Analytical Methods (-74 µm fraction)







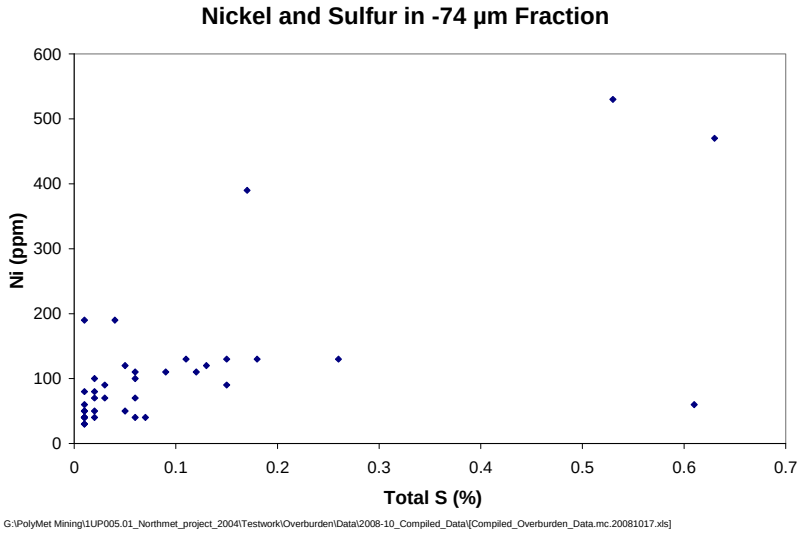


Figure 2: Comparison of Sulfur, Copper and Nickel Concentrations in Two Particle Size Fractions

Table 1: Statistics for Selected Overburden Characteristics

Parameter	Unit	Fraction	n	Minimum	P5	P50	P95	Maximum
pH[1]		-5 mm	44	4.8	5.2	6.4	6.5	8.7
ORP[1]	mV	-5 mm	43	-209	-90	98	95	287
Total Sulfur[2]	% S	-74 µm	37	0.01	0.01	0.05	0.10	0.55
Total Sulfur[2]	% S	-2 mm+74 µm	37	<0.01	<0.01	0.01	0.05	0.16
Sulfate[3]	% S	-74 µm	36	<0.01	<0.01	0.01	0.02	0.04
Sulfate[3]	% S	-2 mm+74 µm	37	<0.01	<0.01	<0.01	0.01	0.03
Cu[4]	ppm	-74 µm	37	20	28	90	180	848
Cu[4]	ppm	-2 mm+74 µm	37	10	18	60	109	402
Ni[4]	ppm	-74 µm	37	30	38	80	113	406
Ni[4]	ppm	-2 mm+74 µm	37	20	20	70	76	180
pH[5]		Whole	14	3.4	3.6	7.1	6.5	7.8
Sulfate[5]	mg/L	Whole	14	1.74	2.8	32	69	203
Cu[5]	mg/L	Whole	14	0.003	0.004	0.009	0.06	0.28
Ni[5]	mg/L	Whole	14	0.0008	0.001	0.01	0.25	1.3

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General Method: [1] Field rinse test
[2] Leco Furnace
[3] HCl soluble, Sobek *et al.* 1978
[4] Four acid digestion
[5] MWMP leachable

Median values for sulfur, copper and nickel were comparable to global crustal values with slight enrichment for copper whereas the extreme values are consistent with proximity to sulfide mineralized rock.

MWMP leachate pH for 14 samples was strongly skewed with a median of 7.1, and the 5th and 95th percentile pH values were 3.6 and 7.8, respectively. Two samples had acidic pHs below 4, and two had pHs between 5 and 6. The two samples with lowest pHs were not acidic when tested in the field but became acidic between sample collection and analysis. As described subsequently, this appears to be due to oxidation of secondary sulfide minerals formed under saturated and chemically reducing conditions in the overburden.

In the majority of samples, the dominant anion in the leachate was sulfate, followed by alkalinity and chloride. Major cation concentrations were variable with all four major ions (calcium, magnesium, sodium and potassium) contributing to the ion balance (Figure 3).

Trace metal concentrations in the leachates were correlated to pH with the highest concentrations at lowest pHs. The correlation was apparent for Al, As, Ba, Cd, Co, Cr, Cu, Fe, Pb, Ni and Zn. Copper and nickel are shown in Figure 4.

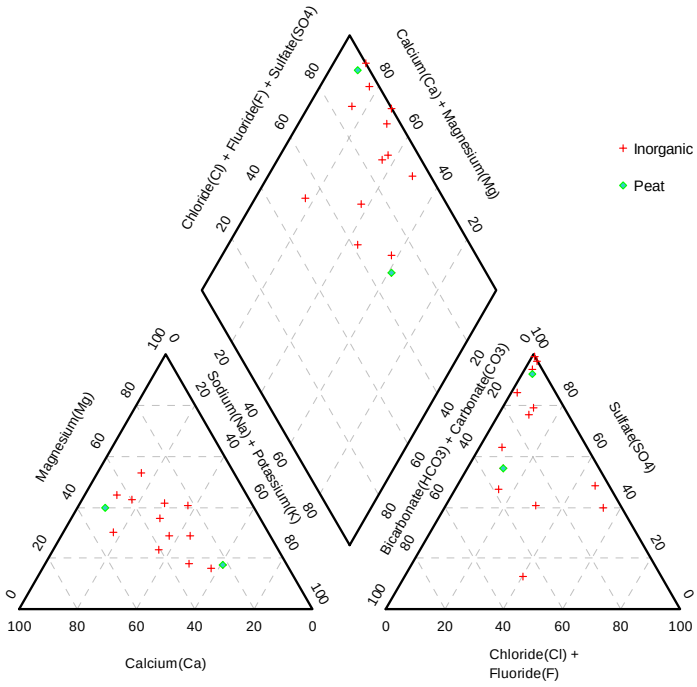

Figure 3: Piper Diagram for MWMP Leachates

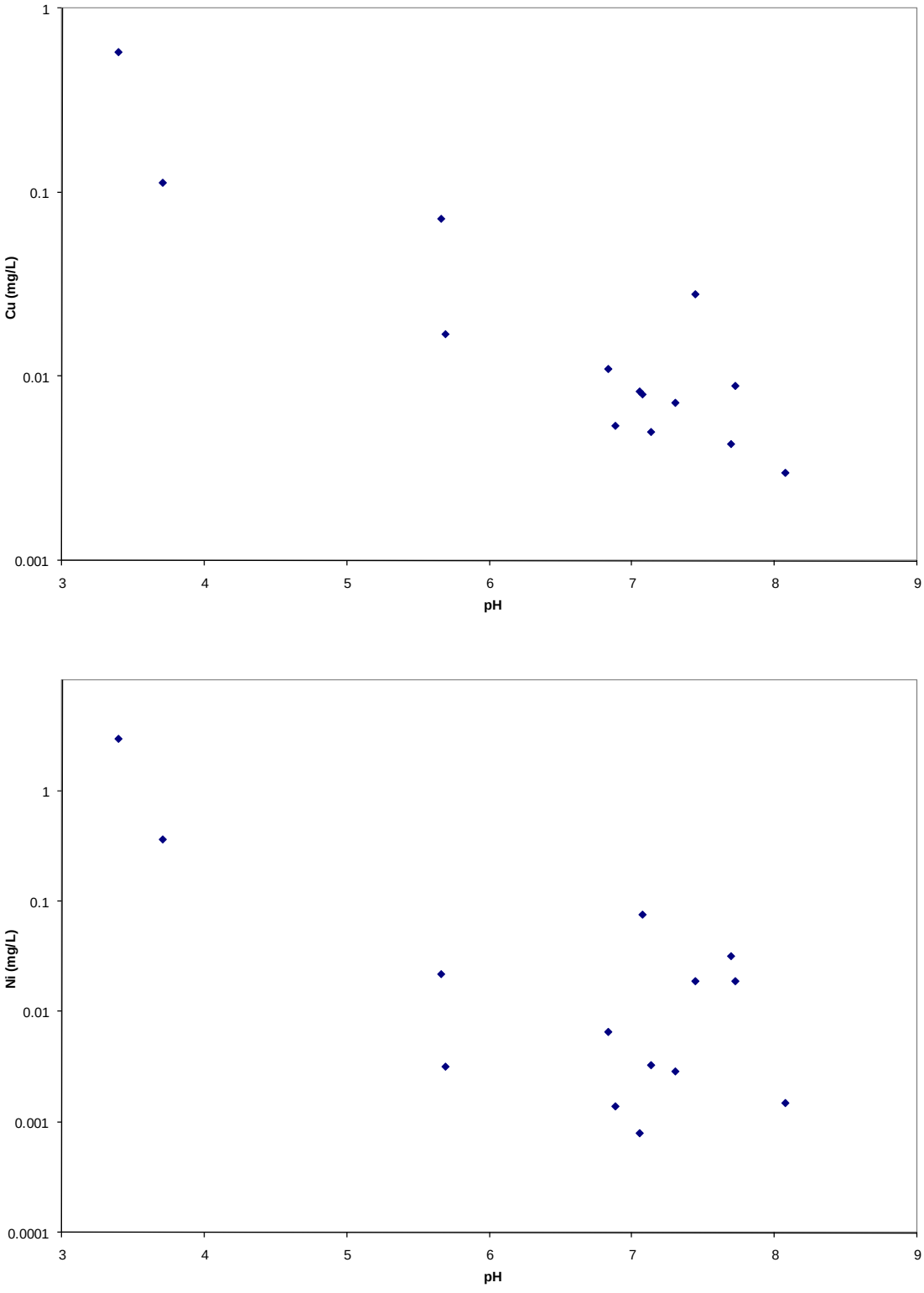

Figure 4: Nickel and Copper Compared to pH for MWMP Leachates

2.3.3 Geological Sources Up-Ice of the Project Area

Borehole RS-14B was underlain by Virginia Formation in the assumed direction up-ice of the Duluth Complex. The hole encountered bedrock at 5 feet.

The soil sample (peat) collected at RS-14B had elevated sulfur content (0.15%), a typical copper concentration (87 ppm) and a slightly low nickel concentration (59 ppm) when compared to the whole dataset.

Compared to the whole dataset, the upper till sample collected at this location had a low rinse pH (5.41), and slightly higher than typical sulfur, copper and nickel concentrations (0.09%, 190 ppm and 110 ppm, respectively).

2.3.4 Underlying Bedrock Geology

Table 2 presents comparative statistics for mineral overburden samples overlying Virginia Formation, Unit 1 Troctolite, and "Other" (Units above 1) Troctolite bedrock (5, 12, and 20 samples, respectively). Figure 4 presents plots of metal content versus a ratio of depth to depth to bedrock. The plots indicate that depth is of little significance to metal concentrations.

Median sulfur concentrations were greatest in samples overlying Virginia Formation bedrock (0.13%), followed by Unit 1 Troctolite (0.06%) and Other Troctolite (0.02%). Extreme values, however, appeared to be unrelated to bedrock type.

Median and extreme copper and nickel concentrations were greatest in samples overlying Unit 1 Troctolite.

Leachate from the R14B sample overlying Virginia Formation bedrock had one of the lowest pH values (3.7) and much higher than typical sulfate, copper, and nickel concentrations (188, 0.11, and 0.36 mg/L, respectively).

MWMP leachate pHs for six samples overlying Unit 1 Troctolite bedrock were near neutral (7. 1 to 7.7) in all but one sample which had a pH of 5.7 (Table 3). The median and 95th percentile sulfate concentrations were 32 and 149 mg/L, respectively.

Leachate pH for seven samples overlying Other Troctolite bedrock varied from 3.4 to 8.1 (with a median value of 6.9) resulting in a wide range of leachable sulfate and metal concentrations.

Overall, the data appeared to indicate bedrock geology exerts a subtle control on the sulfur content of the overburden (i.e. the comparatively elevated median concentrations for samples overlying the Virginia Formation and Unit 1) but that higher sulfur concentrations are not linked to bedrock.

2.3.5 Distance from Mineralized Bedrock

Figure 5 presents the spatial distribution of weighted average sulfur, copper and nickel concentrations in the main stratigraphic horizons for each hole to evaluate down-ice movement of potentially metal-rich bedrock from the mineralized Virginia Formation and Unit 1 of the Duluth Complex. In general, it is expected that these effects might be different for the stratigraphic units.

As noted in the Section 4.4, expected differences in bedrock (range of sulfur concentrations) is apparent in the overburden. This implies that ice movement has not eliminated the effect of bedrock but it is apparent from the range of sulfur concentrations in the overburden that other factors have resulted in variation in sulfur concentrations which may in part be attributed to movement of sulfur-bearing bedrock down ice from their sources.

Table 2: Summarized Statistics for Various Data Groupings (-74 µm fraction)

	Total Sulfur (%)						Copper (ppm)						Nickel (ppm)					
	n	min	P5	P50	P95	max	n	min	P5	P50	P95	max	n	min	P5	P50	P95	max
Virginia Formation	5	0.02	0.03	0.13	0.24	0.26	5	40	54	120	230	240	5	80	82	110	128	130
Unit 1 Troctolite	12	0.01	0.01	0.06	0.33	0.53	12	30	30	110	933	1120	12	40	40	105	453	530
Other Troctolite	20	0.01	0.01	0.02	0.61	0.63	21	20	20	60	290	1510	21	30	30	60	190	470
Peat	3	0.01	0.02	0.15	0.56	0.61	3	20	35	170	251	260	3	40	42	60	123	130
Soil	7	0.02	0.02	0.06	0.14	0.15	7	30	33	110	213	240	7	50	56	90	127	130
Outwash	1			0.63			1			1510			1			470		
Upper Till	17	0.01	0.01	0.01	0.11	0.18	18	20	29	50	205	290	18	30	30	50	139	190
Lower Till	7	0.04	0.04	0.07	0.23	0.17	7	40	55	110	618	780	7	40	58	120	330	390
Saturated Mineral	16	0.01	0.01	0.06	0.56	0.63	17	30	30	60	1198	1510	17	30	38	110	482	530
Unsaturated Mineral	11	0.01	0.01	0.02	0.11	0.13	11	20	25	70	155	190	11	30	35	70	115	120

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Table 3: Summarized Statistics for MWMP Tests

	Leachate pH						Sulfate (mg/L)						Copper (mg/L)						Nickel (mg/L)					
	n	min	P5	P50	P95	max	n	min	P5	P50	P95	max	n	min	P5	P50	P95	max	n	min	P5	P50	P95	Max
Virginia Formation	1			3.7			1			188			1			0.11			1			0.36		
Unit 1 Troctolite	6	5.7	6.0	7.3	7.7	7.7	6	1.7	3.4	32	149	166	6	0.0043	0.0052	0.0086	0.061	0.072	6	0.00080	0.0054	0.021	0.065	0.076
Other Troctolite	7	3.4	4.1	6.9	7.8	8.1	7	3.4	3.8	21	189	230	7	0.0030	0.0036	0.0072	0.41	0.58	7	0.0014	0.0014	0.0032	2.1	3.0
Peat	2	6.8	6.9	7.5	8.0	8.1	2	68	70	81	92	93	2	0.0030	0.0034	0.0070	0.011	0.011	2	0.0015	0.0018	0.0041	0.0063	0.0066
Soil	2	5.7	5.7	6.3	6.8	6.9	2	3.4	4.4	14	23	24	2	0.0054	0.0087	0.039	0.069	0.072	2	0.0014	0.0024	0.012	0.021	0.022
Outwash	1			3.4			1			230			1			0.58			1			3.0		
Upper Till	6	5.7	6.0	7.2	7.7	7.7	6	1.7	2.5	13	35	40	6	0.0050	0.0056	0.0086	0.025	0.028	6	0.00080	0.0013	0.0033	0.019	0.019
Lower Till	3	3.7	4.0	7.1	7.6	7.7	3	97	104	166	186	188	3	0.0043	0.0047	0.0080	0.10	0.11	3	0.032	0.0364	0.076	0.34	0.36
Saturated Inorganics	6	3.4	4.0	7.3	7.7	7.7	6	4.8	5.7	69	214	230	6	0.0043	0.0052	0.013	0.44	0.58	6	0.0032	0.0072	0.026	2.2	3.0
Unsaturated Inorganics	3	6.9	6.9	7.1	7.1	7.1	3	1.7	1.9	3.4	15	17	3	0.0050	0.0050	0.0054	0.0080	0.0083	3	0.00080	0.00086	0.0014	0.0031	0.0033

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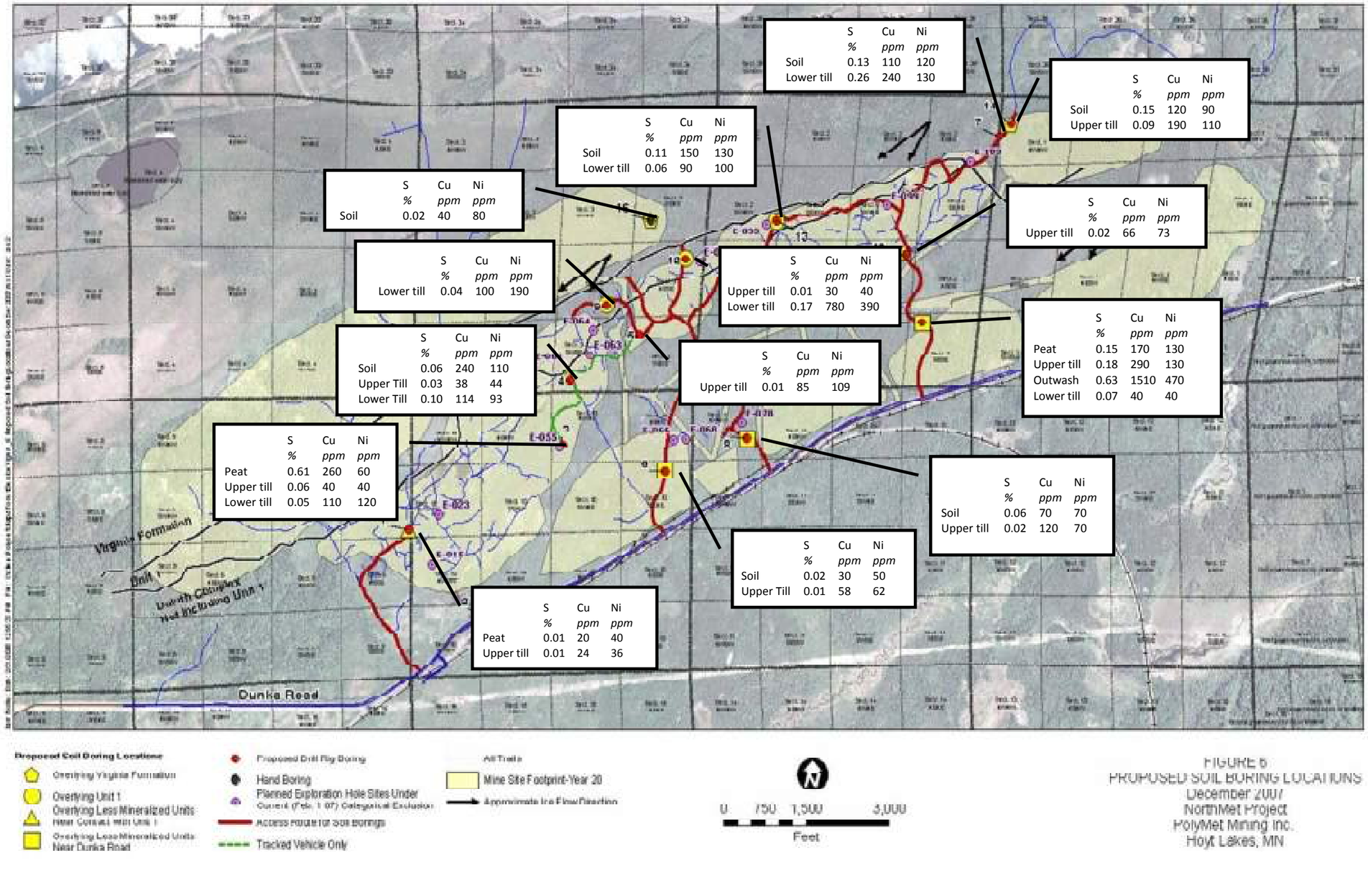

Figure 5: Weighted Average Sulfur, Copper, and Nickel Concentrations (-74 µm fraction)

2.3.6 Glacial and Periglacial Deposition environment

Different depositional environments are indicated by stratigraphic units in the overburden. Barr Engineering has interpreted two main glacial units (Upper and Lower Till). Outwash was intersected in one hole. Peat deposits are common in the project area and were specifically characterized. Thin mineral soils are also present although in some cases these were described as peat. In the comparative statistics in Table 2, samples indicated as "soil" reflect both peat and mineral soils.

The three samples classified specifically as "peat" had the highest median total sulfur and nickel concentrations. MWMP leachates for the 2 peat samples tested had circum-neutral pH. One sample (from borehole RS-01B) had a typical sulfate concentration, and low copper and nickel concentrations. The second sample (from borehole RS-03) had slightly higher than typical sulfate concentrations and typical copper and nickel concentrations.

Comparatively, Lower Till samples had higher median sulfur, copper and nickel concentrations than the Upper Till unit. MWMP leachate from one of the Lower Till samples (borehole RS-07) had one of the lowest pH values (3.7) and much higher than typical sulfate, copper, and nickel concentrations (188, 0.11, and 0.36 mg/L, respectively) (Table 3). Leachate from the other two Lower Till samples had circum-neutral pH and higher than typical sulfate, low to typical copper, and high nickel concentrations. MWMP leachates from six upper till samples had low to typical sulfate, copper, and nickel concentrations.

The one intersection of "outwash" had the highest sulfur, copper and nickel concentrations in the database and leached elevated sulphate and low pH in the MWMP (Table 3).

2.3.7 Groundwater Level during Deposition

The groundwater level during deposition of the glacial tills could not be evaluated and was not considered further.

2.3.8 Degree of Saturation

Table 2 presents comparative statistics of total sulfur, copper and nickel concentrations for saturated and unsaturated mineral overburden. Peat is by definition saturated and is considered in Section 4.6.

When comparing degree of saturation in mineral overburden, sulfur concentrations were greater in the saturated overburden (5^{th}, 50^{th}, and 95^{th} percentile concentrations of 0.01, 0.06, and 0.56%, respectively). Unsaturated mineral overburden had 5^{th}, 50^{th}, and 95^{th} percentile concentrations of 0.01, 0.02, and 0.11% respectively.

Differences in metal concentrations were less apparent than for sulfur. Median copper concentrations were similar regardless of saturation though much higher copper concentrations were apparent at the extremes for saturated overburden. Similarly median nickel concentrations were only slightly greater in the saturated overburden though the difference was much greater at the extreme.

2.3.9 Effect of Oxidation-Reduction Conditions

As discussed in Section 2, wide variations in oxidation-reduction conditions were observed in the overburden as indicated by ORP determinations in field rinse tests. ORP in general decreased with increasing depth implying a correlation with degree of saturation. In addition, indications of secondary sulfide minerals were recorded throughout the survey area in six drill holes, though the implied presence of secondary sulfide minerals did not always correlate with low ORP or necessarily elevated sulfur concentrations.

As noted previously, sulfur in general appeared to be more enriched in the fine fraction. The latter would not generally be expected if the sulfides originated from bedrock materials (or at least sulfur concentrations would be the same in the fine and coarse fractions).

In general, sulfur concentrations showed a trend to increasing concentrations as ORP decreased (Figure 6) with one or two extreme values in boreholes RS11 and RS13. The sample from borehole RS13 indicated the presence of mineralized rock under saturated and reducing conditions (ORP-69 mV) and therefore the elevated sulfur concentrations in the fines may originate from bedrock rather than secondary sulfide precipitation although the presence of sulfur in the bedrock could contribute to conditions that produce secondary sulfides.

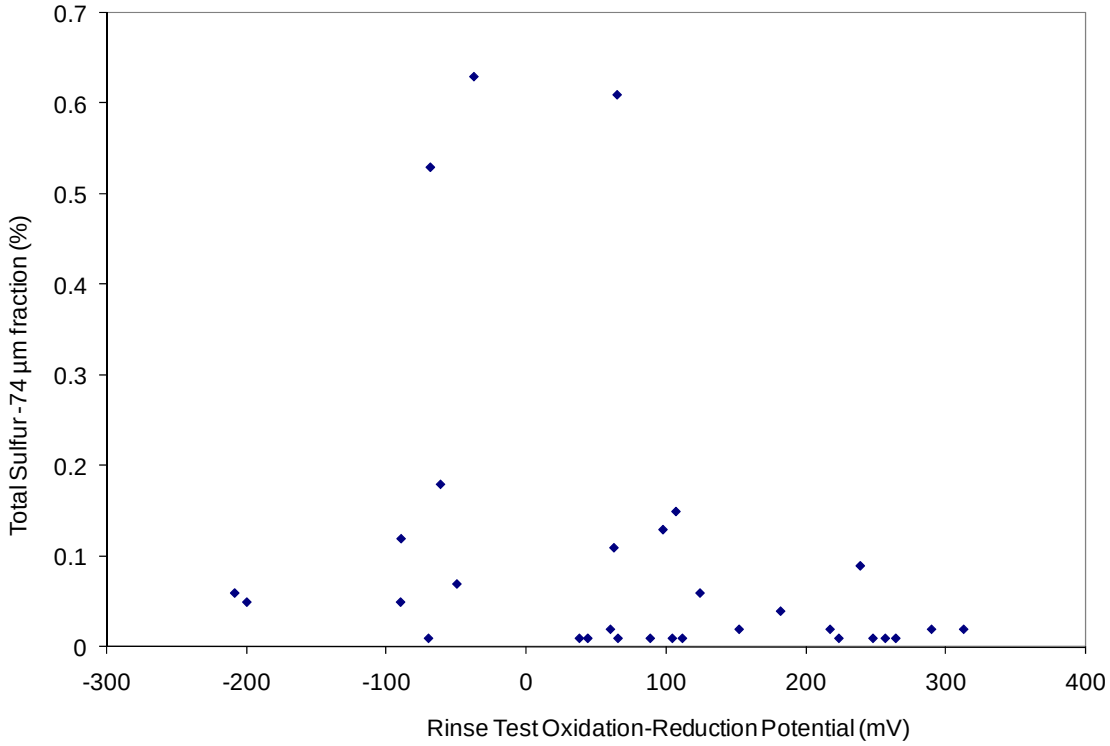

Figure 6: Total Sulfur Compared to Rinse ORP

The two lowest pHs indicated by MWMP tests were on samples that had indications of secondary sulfides (boreholes RS-07 and RS-11) (Table 3). These samples also yielded the highest leachable sulfate (188 and 230 mg/L) and metal concentrations. Other samples yielded non-acidic pHs and intermediate sulfate concentrations compared to samples that did not have secondary sulfides.

2.3.10 Conclusions

The data appear to indicate a vertical rather than a strong lateral variation in metal and sulfur concentrations in mineral overburden. The vertical effects appear as relatively higher sulfur and metal concentrations in the lower till rather than upper till units, and higher sulfur concentrations in chemically-reduced overburden. The two findings are not independent, however, because the lower till occurs deeper in the overburden profile where the overburden is saturated and chemically reducing conditions are observed.

Bedrock also exerts some control on sulfur and metal concentrations as indicated by relatively elevated though comparatively low median concentrations in overburden overlying the Virginia Formation and Unit 1 of the Duluth Complex, which is consistent with the sulfur content of the bedrock. However, elevated sulfur concentrations occur in the overburden overlying the other units of the Duluth Complex due to the vertical zoning effects.

2.4 Interpretation of Overburden Drilling Program Results

2.4.1 Possible Explanation for Vertical Zoning

Sporadic elevated sulfur concentrations and the potential for leaching of acidity and metals from the overburden appears to be related to the development of vertical zoning caused by saturated and chemically-reducing conditions. Precipitation of secondary sulfides appears to have occurred in the overburden. The conditions necessary for this to occur include the presence of organic matter to act as a reductant and a source of sulfur to provide sulfide. The presence of peat in the project area indicates that dissolved organic matter may be entering groundwater. Sulfur most likely originates from reduction of sulfate to sulfide with the sulfate originating from the mineralized bedrock.

2.4.2 Management of Overburden

The data indicate the need to consider selectively managing the deeper saturated overburden to address oxidation of secondary sulfides when exposed to atmospheric conditions during excavation. The current program indicates that secondary sulfides may occur throughout the project area and therefore that currently all deeper overburden should be managed to address the potential for leaching. Management of deep overburden separately would also address exposure of overburden containing mineralized rock overlying Virginia Formation and Unit 1 Troctolite.

In contrast, near surface unsaturated till contains relatively low sulfur and metal concentrations and leached low concentrations of metals.

Two management units are therefore recommended:

- Peat and unsaturated mineral overburden.
- Saturated mineral overburden.

2.4.3 Water Chemistry

Contact water chemistry estimates have been specifically requested for the mineral overburden units in the excavation areas and stockpiles.

The analysis of MWMP leachates provides an indicator of chemistry when stormwater comes into contact with exposed surfaces for example in excavations. In these situations, contact time is limited and the contact ratio (water to solid) is relatively high. Table 4 summarizes concentrations at the median, 95th percentile and maximum concentrations[5]. Generally, the 95th percentile values are recommended for exposed excavation faces to conservatively represent storm water since these are likely to represent the longest contact time with most soluble materials.

Due to the lack of mineralization in the unsaturated overburden, contact water chemistry is expected to be dilute and similar to existing runoff in the project area. Qualitatively, the MWMP tests confirmed that these materials produce dilute contact waters containing low concentrations of major and trace elements.

[5] Prior to complete interpretation of the data, the data were sorted using slightly different groups and provided to Barr Engineering to support RS74 (Draft 02). This version is attached as Appendix B for reference.

Components of the saturated mineral overburden are expected to contain elevated sulfur concentrations and may produce acidic leachate when exposed. The current testwork showed that only samples containing more than 0.2% sulfur produced acidic leachate between recovery by the drill program and testing in the laboratory; however, it is possible that with further exposure, other samples containing lower sulfur concentrations could produce acidity. Also, the extent of elevated sulfur concentrations in the saturated overburden is not known. The acidic pHs represented by the 95th and maximum values should therefore be applied to the component of water that comes into contact with these materials during excavation and exposure to oxidation.

Application of MWMP results to estimation of seepage chemistry for an overburden stockpile needs to consider the disposal conditions and opportunity for water to contact the materials. Based on the current material balance, the only overburden stockpiles planned will be unsaturated overburden and peat stockpiles in the Overburden Storage and Laydown Area. Saturated overburden will mainly be placed within the temporary waste rock stockpiles for ultimate disposal in the backfilled East Pit. Therefore, the use of the 95th percentile contact chemistry would be warranted for the determination of the groundwater impact when comingled with the waste rock in the temporary stockpile without compaction to reduce water infiltration. However, if a saturated overburden stockpile is required in the future, the use of 95th percentile contact chemistry for the saturated overburden is not warranted if infiltration can be minimized using standard compaction techniques. If saturated overburden is stockpiled separately in the future, it is recommended that the median values for the saturated overburden be used as an estimate for water quality.

Table 4: Summary of MWMP Leachate Results By Material Type

Material	pH	Alk mg/L	F mg/L	Cl mg/L	SO_4 mg/L	Al mg/L	Sb mg/L	As mg/L	Ba mg/L	Be mg/L	B mg/L	Cd mg/L	Ca mg/L	Cr mg/L	Co mg/L	Cu mg/L	Fe mg/L	Pb mg/L	Mg mg/L	Mn mg/L	Mo mg/L	Ni mg/L	Se mg/L	Ag mg/L	Na mg/L	Te mg/L	Tl mg/L	V mg/L	Zn mg/L
PEAT																													
P50	7.5	46	0.59	5.9	81	0.086	0.0006	0.0037	0.023	<0.0002	0.21	<0.00004	19	0.0004	0.0003	0.007	0.07	0.00009	9.3	0.13	0.019	0.0041	0.00075	0.00068	26	<0.0002	0.000045	0.0034	0.0015
P95	6.9	79	1	8.8	92	0.13	0.00069	0.0043	0.034	<0.0002	0.23	<0.00004	23	0.00094	0.00066	0.011	0.12	0.00022	11	0.19	0.028	0.0063	0.00089	0.0013	45	<0.0002	0.0001	0.0042	0.0038
Max	6.8	83	1.1	9.2	93	0.13	0.0007	0.0044	0.035	<0.0002	0.23	<0.00004	23	0.001	0.0007	0.011	0.12	0.00023	11	0.19	0.029	0.0066	0.0009	0.0014	47	<0.0002	0.00011	0.0043	0.004
SATURATED MINERAL OVERBURDEN																													
P50	7.3	13	0.33	2	69	0.14	0.0004	0.0023	0.011	<0.0002	0.027	0.000015	13	0.00005	0.0015	0.013	0.11	<0.00005	11	0.18	0.029	0.026	0.002	<0.00005	6.5	<0.0002	<0.00002	0.0011	0.003
P95	4	36	0.56	3.8	210	0.63	0.0012	0.0028	0.026	0.00055	0.087	0.005	26	0.0012	0.23	0.44	5.5	0.0011	18	1.1	0.034	2.2	0.0034	<0.00005	13	<0.0002	0.000025	0.0022	0.86
Max	3.4	38	0.6	4	230	0.74	0.0012	0.0028	0.028	0.0008	0.098	0.0066	27	0.0013	0.31	0.58	7.3	0.0014	18	1.3	0.034	3	0.0037	<0.00005	13	<0.0002	0.00004	0.0024	1.2
UNSATURATED MINERAL OVERBURDEN																													
P50	7.1	5	0.18	1.9	3.4	0.091	<0.0001	0.0005	0.0035	<0.0002	0.013	0.00005	3.9	<0.0002	0.0006	0.0054	0.05	<0.00005	2	0.051	0.0039	0.0014	<0.0002	<0.00005	3.7	<0.0002	<0.00002	0.0005	0.002
P95	6.9	12	0.45	3.4	15	0.3	0.00098	0.0029	0.013	<0.0002	0.028	0.00015	5.7	0.00097	0.0015	0.008	0.059	<0.00005	2.1	0.1	0.013	0.0031	0.00052	<0.00005	4.2	<0.0002	0.000025	0.00059	0.0056
Max	6.9	13	0.48	3.6	17	0.32	0.0011	0.0032	0.014	<0.0002	0.03	0.00016	5.9	0.0011	0.0016	0.0083	0.06	<0.00005	2.1	0.11	0.014	0.0033	0.0006	<0.00005	4.3	<0.0002	0.00003	0.0006	0.006

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3 2010 Sump Spoil Sampling and Analysis Program

3.1 Field Program

Sump excavation occurred as bedrock drilling progressed in the project area in January 2010. Each sump is roughly 8' by 8' and 5' deep. The intent of this sampling effort was to collect additional samples of unsaturated overburden to further characterize the material for use in construction applications. The following sampling procedures were followed by Barr Engineering, as described in an email to MDNR and ERM dated February 26, 2010:

1. For sumps that had already been excavated and filled with drill cuttings but not yet closed (J020, J018, and the "central sump" located near MW-05-08), break up the frozen spoil pile with a spud bar to collect the sample (as described in step 5). Describe any remaining exposed sidewall, and describe the soil sampled, once a split of the samples has been thawed.
2. For sumps not already opened, excavate the sump by segregating bulk soils of different compositions or colors into individual spoil piles. Do not segregate individual layers unless a significant thickness (minimum of 2') is observed, due to sample volume requirements and equipment inefficiencies at smaller scales. Document the depths each spoil pile came from (i.e., 0-3, 3-6, etc), and describe (color; texture [ASTM D2488]; moisture; mottling, if present; reaction with dilute HCl; magnetic properties) the sump profile.
3. Photograph sump profiles and spoil piles.
4. Record location using GPS.
5. Collect samples from the spoil piles at each sump, collecting about 4-5 gallons of soil per sample to provide sufficient samples for analysis.
6. Split the sample to keep about a half gallon of soil thawed for rinse testing and freeze the remaining sample for further analysis.

Sump locations are shown in Figure 7. A total of 13 samples were collected from nine locations as listed in Table 5.

Table 5: Summary of Samples Collected

Location	Depths Sampled
J007	0-3, 3-5
J008	0-3.5, 3.5-4.6
J012	0-3, 3-6
J013	Composite
J019	0-2
J024	0-2
J029	0-5
J037	0-3, 3-6
J107	Composite

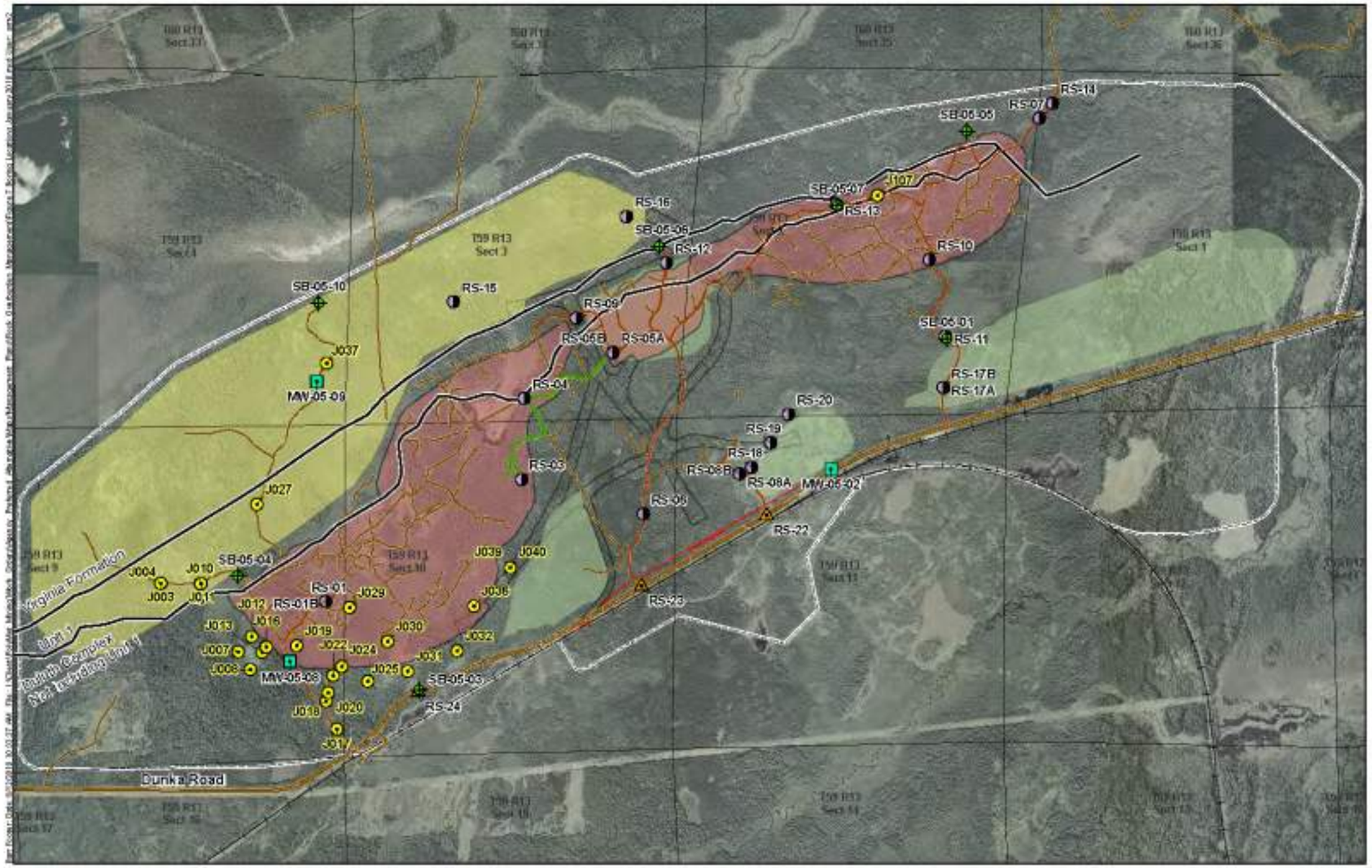

Figure 7: Location of 2010 Sumps

3.2 Analytical Program

The analytical plan was provided to the MDNR by email on March 3, 2010.

3.2.1 Rinse Analysis

Rinse analyses were performed by Barr personnel using a procedure provided by SRK. The procedure involves leaching a sub-sample screened to -5 mm at a liquid to solid ratio of 1:1 (roughly by weight). The resulting leachate is tested for pH and conductivity.

3.2.2 Laboratory Program

At the laboratory, the following analyses were performed:

- A split of the whole sample was used to measure the particle size distribution using ASTM method D422.
- A split of the whole sample was tested using the meteoric water mobility procedure (MWMP, NDEP 1996[6]).
- The -2 mm fraction was tested for acid-base account (Sobek et al 1978[7]), moisture content, element content by aqua regia digestion and element content by 1N nitric acid digestion.

As described in the March 3, 2010 email, the 1N nitric acid digestion was proposed to evaluate reasonably labile methods as recommended by the Regional Technical Committee of the USDA (W 124, The Optimum Utilization of Sewage Sludge on Agricultural Land) (Pierce 1980[8]).

3.3 Results

Laboratory reports are provided in Appendix C.

3.3.1 Field Observations and Rinse Tests

Field observations are attached in Appendix C. All soil samples subsequently tested were described as “Dry”, “Dry to Moist” or “Moist” but none were saturated. Yellowish and brown soil colors were mostly indicative of oxidizing conditions. Three samples were olive brown.

Rinse conductivities were very uniform and low (average 16 µS/cm, range 7 to 26 µS/cm). Rinse pHs averaged 6.7 with a range of 5.8 to 9.0. The low conductivities indicate that the deionized water was weakly modified by contact with the soils and as a result some rinse pHs were only marginally above the typical pH of the deionized water.

3.3.2 Laboratory Analysis

Acid Base Accounting

Acid-base accounting provide an indication of potential for acidic conditions to develop by determining acid potential from the reactive sulfur component, and acid neutralization potential. Results are provided in Table 6.

[6] Nevada Division of Environmental Protection. 1996. Meteoric Water Mobility Procedure (MWMP) Standardized Column Test Procedure. *NDEP publication MWMP.ltr*. May 3 1996. 6p

[7] Sobek A A, Schuller W A, Freeman J R, and Smith R M., 1978, Field and laboratory methods applicable to overburden and minesoils. USEPA Report No. 600/2-78-054, 203 pp.

[8] Pierce, F.J. 1980. The content and distribution of Cd, Cr, Cu, Ni, Pb, and Zn in 16 selected Minnesota soil series. M.S. thesis, University of Minnesota, St. Paul, MN, USA. 140 p.

Paste pH, which is similar to rinse pH except the sample is crushed rather than sieved for analysis, showed similar pHs as the rinse test. Average pH was 6.4 and the range was 5.4 to 7.1.

The total sulfur content of all samples was very low (average 0.02%, maximum 0.03%). These concentrations were very close to the detection limit of 0.01%. Sulfur likely occurs as trace levels of sulfate, sulfide and sulfur associated with carbonaceous matter.

The "fizz test" showed no visible fizz with 10% hydrochloric acid and indicated low carbonate content, which was confirmed by carbonate analysis. Only one sample contained carbonate above the detection level of 0.2% CO_2. Titrated total NP tended to be higher than the carbonate content due to acid buffering by non-carbonate minerals.

These results indicated very low potential for acidic conditions to develop. These soils have very low reactivity from the standpoint of acid generation potential.

3.3.3 Element Content

Element content determined following an aqua regia digestion indicated very uniform concentrations (Table 7) with the exception of one sample (J107). Coefficients of variation were below about 30% for the dataset excluding J107. This sample showed higher concentrations of some elements commonly associated with sulfide minerals, particularly copper (87 mg/kg) and nickel (80 mg/kg). Nickel and magnesium concentrations were strongly correlated in the dataset (Figure 8) suggesting a variable influence from oxidized mafic rocks which could include basaltic rocks and gabbro from the Duluth Complex.

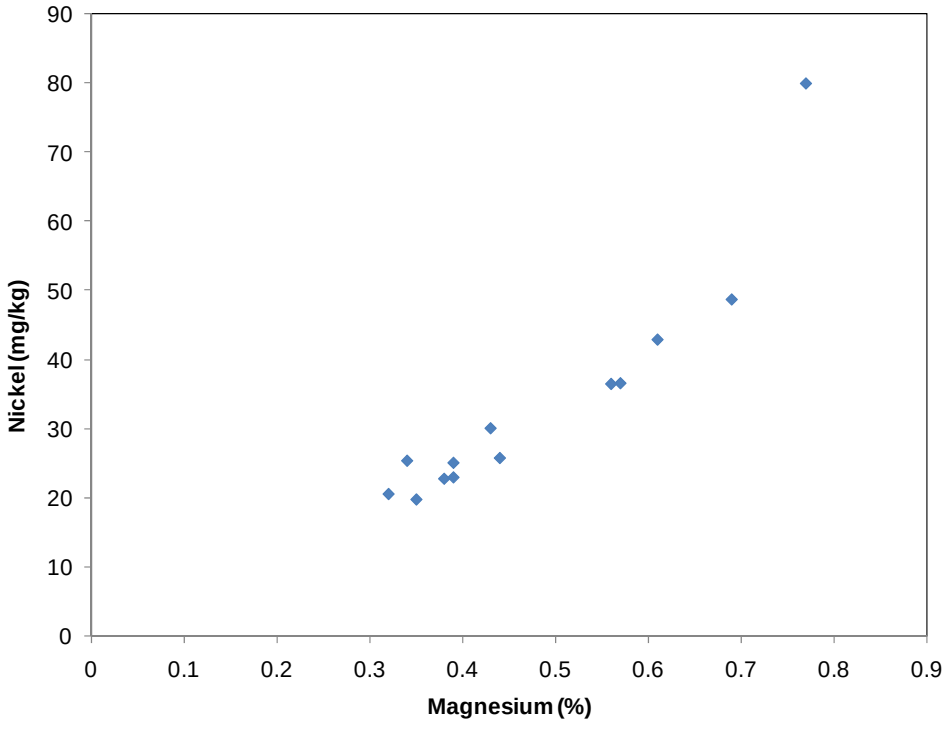

Figure 8: Correlation of Magnesium and Nickel Concentrations for 2010 Overburden Samples

Because the 1N nitric acid digestion required development by the laboratory, all tests were performed in duplicate to evaluate reproducibility. The laboratory analyzed the resulting leachates using two methods (ICP-ES and ICP-MS) to quantify concentrations for all parameters. Duplicate results were assessed with respect to a relative percent difference of 20% for results greater than 10 times the limit of detection. For the main parameters of interest, RPDs were outside this range for Cu (1 sample), Co (no samples) and nickel (5 samples). However, the highest RPD for these parameters was 33% and most were between 20 and 30%. Since this is a weak extraction, reproducibility is considered to be reasonable.

Results were reported by the laboratory as concentration in leachates. For comparison with concentrations yielded by the aqua regia digestion, the leachate concentrations were converted to mg/kg by multiplying by 5 (for results reported in mg/L) and 0.005 (for results reported in µg/L). Like the results of the higher strength aqua regia digestion, concentrations were uniform (Table 8). For cobalt, copper and nickel, coefficients of variation were below 50%.

The difference in extraction by aqua regia and 1N nitric acid were compared as percentages (Table 8). In all cases, nitric acid extracted less than aqua regia and differences in extraction could be attributed to mineral occurrence. Silicate minerals were the least soluble in nitric acid (though aqua reqia also incompletely dissolves silicates). Carbonates, secondary oxides and sulfides can all be expected to dissolve in both nitric and aqua regia to some degree. Major elements associated only with silicates (Na and K) showed the lowest extractions (4 and 6%, respectively). Major elements possibly associated with carbonates (Ca and Mg) showed higher extractions (26 and 16%, respectively). Aluminum and iron, which are likely associated with secondary oxides as well as silicates were more extractable than the alkali metals at 15% and 11%, respectively. Extractions of copper and nickel were higher than iron (19%, 13%) implying a greater association with oxide components compared to iron. The more extractable trace elements were cadmium (27%), cobalt (27%), lead (36%) and manganese (36%).

Aqua regia and nitric acid extractable concentrations were not correlated. Notably, sample J107, which had elevated aqua regia digestible metals concentrations, showed relatively low nitric acid extractable concentrations along with sample J007 (0-3).

Results from these tests were compared with results for the same procedure described by Pierce (1980) (Table 8) for Udept class soils. Concentrations determined for the NorthMet area soils were of the same order but consistently lower than those reported by Pierce (1980). A direct comparison of these datasets for soils from different areas analyzed by different laboratories should not be made but the results suggest that soils in the NorthMet area are not unusual in terms of weak acid extractability.

MWMP leachate pHs were typically between 6 and 7 with the exception of 5.9 for J107 (Table 9). Leachate chemistry was dominated by sulfate, bicarbonate, calcium, sodium and magnesium but concentrations were very low. Median sulfate concentrations were 4.3 mg/L and the maximum 10 mg/L (for J107). Metal concentrations were very low. Median copper and nickel concentrations were 0.005 and 0.0007 mg/L respectively.

3.4 Comparison with Results of Overburden Drilling Program

Table 10 compares sulfur, copper and nickel concentrations in unsaturated overburden collected during the drilling program with those collected from sump spoils. Due to differences in the analytical programs, the comparison is between the calculated concentrations in -2 mm from analysis of the -2+0.074 mm and -0.074 mm fraction of drilling samples and the analyzed -2 mm fraction of the sump spoils. The range of sulfur concentration was very similar in both cases. Copper and nickel concentrations were slightly elevated for the drill hole samples. This may reflect a greater

influence from Duluth Complex rock at depth though the differences are small. The data support the conclusion that bulk characteristics of unsaturated overburden have been adequately characterized by these programs.

Table 11 provides a similar comparison for leachable components of unsaturated overburden. The distribution of concentrations is very similar for the original small dataset of three samples and the 13 sump spoils samples. These results also indicate that the database of 16 samples has demonstrated the low leaching potential of unsaturated overburden throughout the site.

4 Conclusions

Two overburden characterization programs have shown:

- Two overburden managements units (saturated; and unsaturated and peat) will classify excavated materials based on reactivity and leachability;
- Saturated overburden has higher reactivity apparently due to the localized presence of secondary sulfide minerals; and
- Unsaturated overburden consistently has low reactivity based on sulfur and metal content, and shows weak leachability when tested using MWMP.

Table 6: Acid-Base Accounting Results for Sump Spoil Samples

Sample ID	Paste pH Std. Units	CO_2 % CO_2	Total S % S	Sulphate % S	AP kg $CaCO_3$/t	Sobek NP kg $CaCO_3$/t	Net NP kg $CaCO_3$/t	NP/AP Ratio	Fizz Test Visual
LOD	0.01	0.2	0.01	0.01	#N/A	0.1	#N/A	#N/A	#N/A
Method Code	Sobek	C-GAS05	S-IR08	S-GRA06a	Calc.	Sobek NP	Calc.	Calc.	Sobek
J007 (0-3)	5.44	<0.2	0.02	<0.01	0.6	5.1	4.5	8.2	None
J007 (3-5)	6.83	<0.2	0.01	<0.01	0.3	8.4	8.1	26.9	None
J008 (0-3.5)	6.23	<0.2	<0.01	<0.01	<0.3	8.9	8.9	29.7	None
J008 (3.5-4.6)	6.65	<0.2	0.02	0.01	0.3	8.8	8.5	28.2	None
J012 (0-3)	6.57	<0.2	0.01	0.01	<0.3	7.2	7.2	24.0	None
J012 (3-6)	6.89	<0.2	0.03	<0.01	0.9	7.3	6.4	7.8	None
J013	6.28	0.2	0.02	<0.01	0.6	7.8	7.2	12.5	None
J019 (0-2)	6.02	0.2	0.02	<0.01	0.6	5.2	4.6	8.3	None
J024 (0-2)	6.69	0.2	0.01	0.01	<0.3	7.3	7.3	24.3	None
J029 (0-5)	6.55	1.2	0.02	0.01	0.3	9.4	9.1	30.1	None
J037 (0-3)	6.05	0.2	0.01	0.01	<0.3	5.7	5.7	19.0	None
J037 (3-6)	7.10	0.2	0.01	0.01	<0.3	7.0	7.0	23.3	None
J107[1]	5.45	<0.2	0.02	0.01	0.3	4.3	4.0	13.8	None
Duplicates									
J007 (0-3)	5.50					5.7			None

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Table 7: Metal Concentrations Determined Following Aqua Regia Digestion for Sump Spoil Samples

Sample ID	Co ppm	Cu ppm	Mg %	Ni ppm
LOD	0.1	0.2	0.01	0.2
Method Code	ME-MS41	ME-MS41	ME-MS41	ME-MS41
J007 (0-3)	12	24.5	0.61	42.9
J007 (3-5)	7.8	19.4	0.38	22.8
J008 (0-3.5)	11.2	25.4	0.57	36.6
J008 (3.5-4.6)	11.2	28	0.56	36.5
J012 (0-3)	8	28.2	0.39	23
J012 (3-6)	7.6	28.3	0.35	19.8
J013	8.5	20.1	0.44	25.8
J019 (0-2)	9.2	18.5	0.43	30.1
J024 (0-2)	8.6	20	0.39	25.1
J029 (0-5)	12.9	28.7	0.69	48.7
J037 (0-3)	8.5	23.1	0.34	25.4
J037 (3-6)	6.8	17.3	0.32	20.6
J107	15.5	86.7	0.77	79.9

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Table 8: Metal Concentrations Determined Following 1N Nitric Acid Extraction for Sump Spoil Samples

Sample ID	Cd mg/kg	Cu mg/kg	Pb mg/kg	Zn mg/kg	Al mg/kg	Ca mg/kg	Co mg/kg	Fe mg/kg	K %	Mg mg/kg	Mn mg/kg	Na mg/kg	Ni mg/kg
J007 (0-3)	0.01	0.5	1	2.0	0.06	0.01	2.8	0.05	-0.003	0.01	16	0.004	0.6
J007 (3-5)	0.01	5.5	1	3.3	0.20	0.16	2.5	0.32	0.006	0.09	226	0.003	5.1
J008 (0-3.5)	0.01	6.6	1	2.6	0.27	0.16	3.4	0.40	0.008	0.12	165	0.003	6.5
J008 (3.5-4.6)	0.01	8.0	1	2.9	0.25	0.17	3.3	0.36	0.007	0.12	199	0.003	7.5
J012 (0-3)	0.01	8.3	1	3.0	0.22	0.14	2.2	0.34	0.004	0.09	171	0.003	4.3
J012 (3-6)	0.01	8.6	1	3.5	0.18	0.15	2.1	0.33	0.007	0.09	163	0.003	4.0
J013	0.02	4.4	1	2.3	0.28	0.13	2.3	0.27	0.006	0.07	165	0.003	3.8
J019 (0-2)	0.01	4.5	1	1.4	0.43	0.11	2.0	0.41	0.003	0.05	79	0.004	2.4
J024 (0-2)	0.01	5.0	1	3.1	0.20	0.13	2.5	0.41	0.004	0.10	188	0.003	4.6
J029 (0-5)	0.02	6.7	1	2.8	0.39	0.14	3.2	0.34	0.007	0.11	127	0.004	6.7
J037 (0-3)	0.01	4.7	1	2.7	0.21	0.13	3.0	0.44	0.004	0.09	297	0.003	4.2
J037 (3-6)	0.02	4.5	1	4.0	0.16	0.16	2.2	0.32	0.006	0.10	195	0.003	4.4
J107	0.02	2.2	2	3.1	0.08	0.01	3.3	0.05	-0.003	0.01	14	0.004	1.0
Extraction Compared to Aqua Regia													
J007 (0-3)	26%	2%	36%	5%	3%	2%	23%	1%	-4%	1%	4%	5%	1%
J007 (3-5)	29%	28%	36%	14%	18%	35%	32%	12%	8%	24%	43%	5%	22%
J008 (0-3.5)	29%	26%	40%	9%	16%	34%	30%	13%	10%	20%	42%	4%	18%
J008 (3.5-4.6)	23%	29%	38%	9%	16%	31%	30%	11%	7%	21%	43%	3%	20%
J012 (0-3)	30%	29%	41%	13%	19%	36%	27%	13%	5%	23%	35%	5%	19%
J012 (3-6)	24%	30%	34%	14%	17%	37%	27%	12%	7%	25%	31%	4%	20%
J013	33%	22%	40%	9%	20%	29%	27%	11%	6%	15%	42%	4%	15%
J019 (0-2)	26%	24%	31%	3%	19%	16%	21%	16%	4%	11%	32%	3%	8%
J024 (0-2)	23%	25%	33%	13%	18%	34%	29%	14%	5%	25%	38%	4%	18%
J029 (0-5)	24%	23%	33%	9%	22%	21%	25%	11%	9%	16%	37%	3%	14%
J037 (0-3)	27%	20%	36%	13%	18%	32%	35%	18%	6%	26%	59%	5%	17%
J037 (3-6)	29%	26%	29%	18%	18%	36%	33%	13%	7%	31%	44%	5%	21%
J107	27%	3%	36%	7%	4%	2%	21%	1%	-4%	1%	4%	7%	1%
Udept Soils (n=15, Pierce 1980)													
Average	0.07	11	2.3	12.6	-	-	-	-	-	-	-	-	8.9

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Table 9: MWMP Results for Sump Spoil Samples

Sample ID	pH	F	SO_4	Al	Sb	As	Ba	Be	B	Cd	Ca	Cr	Co	Cu	Fe	Pb	Mg	Mn	Hg	Mo	Ni	K	Se	Ag	Na	Tl	V	Zn
		mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	ug/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
J007 (0-3)	6.52	0.04	5	0.0815	0.00005	0.00036	0.00129	0.00002	-0.05	0.000007	1.57	0.0003	0.000295	0.00513	0.015	0.000211	0.54	0.0133	0.02	-0.00005	0.00041	0.14	0.00019	-0.000005	1.98	0.000011	0.0006	0.0016
J007 (3-5)	6.75	0.07	1.6	0.141	0.00005	0.00037	0.00212	0.00002	-0.05	-0.000005	1.35	0.0004	0.000191	0.00468	0.133	0.000125	0.49	0.00452	0.02	0.00007	0.00067	0.12	0.00013	-0.000005	1.66	0.000003	0.0009	0.0009
J008 (0-3.5)	6.58	0.03	1.1	0.257	0.00005	0.00052	0.0024	0.00004	-0.05	0.000005	2.58	0.0008	0.000242	0.0069	0.143	0.000051	0.57	0.00656	0.02	0.00012	0.00093	0.53	0.0003	0.000008	1.25	0.000005	0.0008	0.0007
J008 (3.5-4.6)	6.62	0.03	0.8	0.201	0.00006	0.00057	0.00184	0.00005	-0.05	-0.000005	2.35	0.0008	0.000212	0.00759	0.127	0.000044	0.56	0.00597	0.02	0.00009	0.00113	0.26	0.00024	0.000008	1.65	0.000003	0.0009	0.0006
J012 (0-3)	6.74	0.05	4.8	0.049	0.00004	0.00026	0.00104	0.00001	-0.05	-0.000005	1.64	0.0004	0.000128	0.00437	0.035	0.000036	0.71	0.00344	0.01	-0.00005	0.00064	0.11	0.00015	-0.000005	2.23	-0.000002	0.0004	0.0008
J012 (3-6)	6.9	0.07	5.3	0.205	0.00004	0.00039	0.00334	0.00002	-0.05	-0.000005	1.82	0.0006	0.00017	0.00367	0.239	0.000114	0.82	0.00722	0.01	0.00006	0.00073	0.22	0.00008	-0.000005	2.01	-0.000002	0.001	0.0011
J013	6.53	0.02	-0.5	0.559	0.00006	0.00053	0.00147	0.00007	-0.05	0.000013	3.01	0.0015	0.000293	0.00622	0.179	0.000043	0.68	0.014	0.02	0.00006	0.00113	0.27	0.0004	0.000014	1.62	0.000004	0.0006	0.0019
J019 (0-2)	6.49	0.03	7.4	0.215	0.00006	0.00039	0.00061	0.00003	-0.05	-0.000005	4.07	0.0011	0.000296	0.00683	0.177	0.000034	1.4	0.0033	0.02	-0.00005	0.00035	-0.05	0.00031	0.000007	1.94	0.000004	0.001	0.0011
J024 (0-2)	6.49	0.06	4.3	0.0297	0.00002	0.00022	0.00079	-0.00001	-0.05	-0.000005	1.82	-0.0001	0.000123	0.00398	0.033	0.000074	0.68	0.0051	-0.01	-0.00005	0.00042	0.09	0.00014	-0.000005	1.52	-0.000002	0.0003	0.0012
J029 (0-5)	6.62	0.02	2.8	0.0778	0.00004	0.00026	0.00117	0.00001	-0.05	0.000012	4.46	0.0002	0.00019	0.00643	0.027	0.000006	0.99	0.00475	0.01	0.00012	0.00037	0.37	0.00019	-0.000005	1.95	0.000002	0.0004	0.0015
J037 (0-3)	6.25	0.05	4.4	0.0973	0.00004	0.00024	0.00142	0.00002	-0.05	-0.000005	1.34	0.0014	0.000129	0.0036	0.081	0.000026	0.65	0.0155	0.01	-0.00005	0.00111	0.09	0.00026	-0.000005	1.65	0.000003	0.0003	0.001
J037 (3-6)	6.42	0.13	2.1	0.139	0.00007	0.00043	0.00212	0.00002	-0.05	-0.000005	1.15	0.0014	0.000127	0.00519	0.152	0.000152	0.67	0.00687	0.01	0.00011	0.00116	0.15	0.00019	-0.000005	1.67	0.000002	0.0012	0.001
J107	5.85	0.01	10	0.0284	0.00002	0.0002	0.00402	-0.00001	-0.05	0.000026	2.13	-0.0001	0.00036	0.00221	0.004	0.000038	0.93	0.0601	-0.01	-0.00005	0.00308	0.84	0.00046	-0.000005	2.14	0.000008	0.0002	0.0018

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Table 10: Comparison of Sulfur, Nickel and Copper Content of Unsaturated Overburden

	Total Sulfur (%)						Copper (ppm)						Nickel (ppm)					
	n	min	P5	P50	P95	max	n	min	P5	P50	P95	max	n	min	P5	P50	P95	max
Drilling Samples	11	0.01	0.01	0.01	0.03	0.03	11	20	20	31	104	126	11	17	18	50	71	72
Sump Spoils	13	0.01	0.01	0.02	0.03	0.03	13	17	18	25	52	89	13	20	20	26	46	80

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Table 11: Comparison of Sulfate, Nickel and Copper Concentrations in MWMP Leachates from Unsaturated Overburden

	Sulfate (mg/L)				Copper (mg/L)				Nickel (mg/L)			
	n	P50	P95	max	n	P50	P95	max	n	P50	P95	max
Drilling Samples	3	3.4	15	17	11	0.005	0.008	0.008	11	0.001	0.003	0.003
Sump Spoils	13	4.3	8.4	10	13	0.005	0.007	0.008	13	0.0007	0.002	0.003

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\\Van-svr0.van.na.srk.ad\ge_projects\PolyMet Mining\1UP005.01_Northmet_project_2004\Testwork\Overburden\2010-07_Report\Interpretations\[ePolymet Static Testing Apr 09 10 (May 28 10)_1UP005001_ver00.xls]

Appendix A
Drill Hole Overburden Geochemical Data

Fraction Analyzed			Whole Sample	Whole Sample	Whole Sample	Whole Sample	Whole Sample	Whole Sample	-5 mm	-5 mm	-5 mm	Whole Sample	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm
Parameter	**Depth**	**Depth**	**Stratigraphic Unit**	**Saturated**	**Oxidized**	**Texture**	**Sulfides**	**Other**	**pH**	**Specific Conductivity**	**ORP**	**Moisture Content**	**Total C**	**CO_2**	**Total Sulphur**	**Sulphate**	**MPA**
Units	From (ft)	To (ft)								(µS/cm)	mV	%	% C	% CO2	% S	% S	kgCaCO3/t ore
Detection Limit													0.01	0.2	0.01	0.01	0.3
General Method			Field Observation	Field Observation	Field Observation	Field Observation	Field Observation	Field Observation	Rinse Test	Rinse Test	Rinse Test		ABA	ABA	ABA	ABA	ABA
Analytical Method Code													C-IR07	C-GAS05	S-IR08	S-GRA06a	OA-VOL08
RS-01B	0	1	Peat			Peat			7.05	24	248.1	39		<0.2	0.01	<0.01	2.8
RS-01B	1	5	Upper till	N	Y	Sand			5.86	10	256.9	4.7		<0.2	0.01	0.01	0.3
RS-01B	14	15	Upper till	Y	Y	Sand			6.37	16	223.7	5.7		<0.2	0.01	0.01	<0.3
RS-01B	18	20	Upper till	Y	N	Sand			7.28	34	65.6	7.8		<0.2	0.01	<0.01	0.3
RS-01B	20	25	Lower till, Bedrock at 20.5ft						8.79	66	-40	9.1					
RS-03	5	10	Peat			Peat			5.17	116	65	65		NSS*	0.61	NSS*	NSS*
RS-03	15	20	Upper till	Y	N	Sand			7.4	50	-208.7	4.3		<0.2	0.06	0.01	1.9
RS-03	20	22	Lower till, Bedrock at 22ft	Y	N	Gravel			9.42	63	-200	5.8		0.3	0.05	0.02	1.6
RS-04	1	5	Soil	Y	Y	Silt			5.77	22	124.3	12		<0.2	0.06	0.01	1.9
RS-04	10	15	Upper till	Y	Y	Sand			6.33	25	104.5	3.8		<0.2	0.01	0.02	0.3
RS-04	15	20	Upper till (15-18ft), Lower till (18-25ft)	Y	N	Sand	Secondary		6.74	25	-90	7.4		0.2	0.05	<0.01	1.6
RS-04	20	25	Lower till, Bedrock at 25ft	Y	N	Sand	Secondary		7.83	17	-89.6	4.7		0.3	0.12	<0.01	3.8
RS-05A	5	10	Upper till	mixed	mixed	mixed			6.55	22	88.7	6.5		<0.2	0.01	0.04	0.3
RS-05A	10	13	Upper till, Bedrock at 13ft	Y	N	Gravel			8.9	88	-70	2.5		<0.2	0.01	0.01	0.3
RS-06A	0.5	2	Soil	N	Y	Sand			4.84	5	313	6.8		0.2	0.02	0.01	0.6
RS-06A	2	4	Upper till	N	Y	Sand			4.99	11	279	8.1					
RS-06A	5	7.5	Upper till	N	Y	Sand			5.82	12	264.4	9.5		<0.2	0.01	<0.01	0.3
RS-06A	7.5	10	Upper till	N	Y	Sand			6.32	17	251	4.3					
RS-06A	15	19	Upper till	Y	N	Sand	Secondary		6.75	18	38	9.5		<0.2	0.01	<0.01	0.3
RS-06A	19	21	Upper till, Bedrock at 21ft	Y	N	Sand	Secondary		7.86	20	18	9.1					
RS-07	1	2	Soil	N	Y	Sand			5.61	45	97.8	15		<0.2	0.13	<0.01	4.1
RS-07	2	3	Soil	N	Y	Sand						13					
RS-07	3	5	Upper till	N	Y	Sand			6.1	52	27	9.4					
RS-07	5	6	Upper till	N	Y	Sand			6.1	52	27	4.2					
RS-07	6	10	Lower till						6.4	17	60	5.8					
RS-07R	10	12	Lower till (10-11ft), Bedrock (11-12ft)									5.0					
RS-07R	13.5	14.5	Bedrock	Y	N	Gravel			7.48	82		7.0					
RS-07/RS-07R	6	14.5	Lower till											<0.2	0.26	0.01	8.1
RS-08A	0	1	Soil	N	N	Sand						15		<0.2	0.06	<0.01	1.9
RS-08A	1	5	Upper till	N	N	Sand			5.18	19	287.6	6.7					
RS-08A	5	11	Upper till	N	N	Sand	Secondary		5.78	22	217.4	6.4		<0.2	0.02	0.05	0.6
RS-09	7	8	Lower till, Bedrock at 8ft	Y	N	Silt	Secondary		5.88	2	182	11		<0.2	0.04	0.03	1.3
RS-10	1	2	Upper till	N	Y	Sand						12		<0.2	0.03	0.02	0.9
RS-10	2	3	Upper till	N	Y	Sand			6.07	30	193	9.8					
RS-10	3	5.5	Upper till	N	Y	Sand			5.73	12	241.6	9.5					
RS-10	5.5	7.5	Upper till					Cement	7.08	20	60.2	6.2		<0.2	0.02	0.01	0.6
RS-10	7.5	10	Upper till	N	N	Gravel			6.81	30	152.3	4.4		<0.2	0.02	<0.01	0.6
RS-10	10	14	Upper till	N	N	Gravel		Cement	6.5	26	145.3	28					
RS-11	0	9.5	Peat			Peat			5.89	40	107.1	33		<0.2	0.15	<0.01	4.7
RS-11	11.5	17	Upper till	Y	N	Sand	Secondary		6.47	47	-61.4	9.3		<0.2	0.18	<0.01	5.6
RS-11	17	25	Outwash	Y	N	Gravel	Secondary		6.56	30	-37.5	6.3		<0.2	0.63	0.02	19.7
RS-11	28	31	Lower till	Y	N	Sand			6.5	70	-49.7	5.9		0.4	0.07	<0.01	2.2
RS-11	31	33	Lower till, Bedrock at 33ft	Y	N	Sand						4.7					
RS-12	7	9	Upper till	N	Y	Sand		Mineral'd Rock	7.17	33	111.7	6.9		0.4	0.01	0.01	0.3
RS-12	16	18	Upper till	Y	Y	Sand			7.14	14	44	10		<0.2	0.01	0.01	0.3
RS-12	20	22	Lower till, Bedrock at 22ft	Y	N	Sand		Mineral'd Rock				5.3		<0.2	0.17	0.02	5.3
RS-13	0	1.5	Soil			peat			6.15	42	62.7	29		<0.2	0.11	0.02	3.4
RS-13	1.5	2.5	Lower till			Sand						15		<0.2	0.06	<0.01	1.9
RS-13	2.5	6	Lower till			Sand			6.07	27	106.6	5.2					
RS-13	8	10	Bedrock	Y	N	Gravel		Mineral'd Rock	7.2	46	-68.7	4.1		<0.2	0.53	0.05	16.6
RS-14B	0	1.5	Soil			Peat						26		<0.2	0.15	<0.01	4.7
RS-14B	1.5	3	Upper till	N	Y	Sand			5.41	19	239	16		<0.2	0.09	<0.01	2.8
RS-14B	3	5	Upper till, Bedrock at 5ft	N	Y	Sand						6.3					
RS-16B	0	2	Soil	N	Y	Sand			5.29	8	290	22		<0.2	0.02	<0.01	0.6

Fraction Analyzed			-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm
Parameter	**Depth**	**Depth**	**NP**	**Fizz Rating**	**Ag**	**Al**	**As**	**Au**	**B**	**Ba**	**Be**	**Bi**	**Ca**	**Cd**	**Ce**	**Co**	**Cr**	**Cs**	**Cu**
Units	From (ft)	To (ft)	kgCaCO3/t ore	Unity	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm
Detection Limit			1	1	0.01	0.01	0.1	0.2	10	10	0.05	0.01	0.01	0.01	0.02	0.1	1	0.05	0.2
General Method			ABA	ABA	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia
Analytical Method Code			OA-VOL08	OA-VOL08	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41
RS-01B	0	1	5	1	4.73	1.29	6.7	<0.2	<10	140	0.43	0.21	0.43	0.54	23.5	8	36	1.21	28.3
RS-01B	1	5	2	1	1.05	1.19	6	<0.2	<10	40	0.4	0.07	0.39	0.09	50.7	7.6	38	1.06	25.2
RS-01B	14	15	5	1	0.63	0.9	5.5	<0.2	<10	40	0.34	0.07	0.55	0.09	40.8	7	36	1.1	31.2
RS-01B	18	20	13	1	9.98	1.24	4.5	<0.2	<10	40	0.36	0.08	0.83	0.08	40.5	9.6	47	1	38.8
RS-01B	20	25																	
RS-03	5	10	NSS*	NSS*	0.54	0.82	5.3	<0.2	<10	60	0.41	0.1	1.19	0.45	30.2	5	46	0.58	264
RS-03	15	20	15	1	0.08	0.74	4.7	<0.2	<10	40	0.24	0.25	0.77	0.11	35.8	6.6	32	0.69	37.3
RS-03	20	22	20	1	5.76	1.79	2.2	<0.2	<10	50	0.23	0.27	1.23	0.13	33.5	22.6	41	1.14	120
RS-04	1	5	6	1	0.55	2.45	7.9	<0.2	<10	80	0.45	0.2	0.53	0.12	50.6	17	62	1.98	240
RS-04	10	15	7	1	1.22	0.8	2.6	<0.2	<10	40	0.25	0.06	0.59	0.08	47.4	8.2	37	0.71	36.5
RS-04	15	20	12	1	1.92	0.76	1.5	<0.2	<10	20	0.19	0.06	0.66	0.07	41.6	7.9	33	0.84	53.5
RS-04	20	25	13	1	1.87	1.01	3	<0.2	<10	30	0.28	0.07	0.75	0.12	36.5	14.9	37	1.17	144
RS-05A	5	10	17	1	0.08	1.64	5.8	<0.2	<10	70	0.43	0.14	0.5	0.12	54.4	14.5	57	1.87	106
RS-05A	10	13	6	1	0.03	4.47	1.9	<0.2	<10	50	0.23	0.05	2.43	0.1	17.95	38.5	80	0.76	59.4
RS-06A	0.5	2	0	1	0.17	2.16	4	<0.2	<10	90	0.48	0.09	0.33	0.1	52.4	12.3	46	1.6	31.9
RS-06A	2	4																	
RS-06A	5	7.5	6	1	0.54	1.46	4.6	<0.2	<10	60	0.41	0.08	0.53	0.05	50.3	9.6	50	1.16	42.1
RS-06A	7.5	10																	
RS-06A	15	19	6	1	0.25	1.05	6.8	<0.2	<10	50	0.32	0.07	0.58	0.1	50.4	11.7	43	1.08	44.6
RS-06A	19	21																	
RS-07	1	2	-2	1	0.04	3.04	9.6	<0.2	<10	110	0.81	0.16	0.31	0.11	40.9	14.7	101	3.75	115
RS-07	2	3																	
RS-07	3	5																	
RS-07	5	6																	
RS-07	6	10																	
RS-07R	10	12																	
RS-07R	13.5	14.5																	
RS-07/RS-07R	6	14.5	12	1	0.25	2.5	15.3	<0.2	<10	110	0.85	0.27	0.48	0.52	56.7	25.2	99	4.13	239
RS-08A	0	1	4	1	0.16	2.9	8.5	<0.2	<10	70	0.67	0.1	0.35	0.09	45.3	15	59	2.08	70.8
RS-08A	1	5																	
RS-08A	5	11	7	1	0.58	1.53	11.7	<0.2	<10	60	0.43	0.09	0.58	0.19	49.6	19.2	53	2.24	123.5
RS-09	7	8	23	1	0.07	4.83	5	<0.2	<10	80	0.54	0.08	2.41	0.17	35.1	34.2	89	2.41	107.5
RS-10	1	2	2	1	0.07	2.67	4	<0.2	<10	110	0.52	0.13	0.31	0.07	42.1	14	56	2.26	39.1
RS-10	2	3																	
RS-10	3	5.5																	
RS-10	5.5	7.5	9	1	0.05	1.88	4.8	<0.2	<10	90	0.49	0.12	0.55	0.06	57.6	12.7	68	3.52	53.4
RS-10	7.5	10	11	1	0.05	2.57	9.7	<0.2	<10	100	0.59	0.11	0.65	0.14	47.1	20.5	85	4.27	88.3
RS-10	10	14																	
RS-11	0	9.5	3	1	1.65	2.48	4.5	<0.2	<10	90	0.68	0.16	0.75	0.4	40.8	18.2	110	2.74	166.5
RS-11	11.5	17	14	1	0.24	2.62	10.7	<0.2	<10	100	0.89	0.25	0.68	0.31	54.9	21.5	107	3.62	307
RS-11	17	25	4	1	27.4	3.09	20.9	<0.2	<10	100	0.77	0.46	0.87	0.6	41.9	41.2	102	4.44	1560
RS-11	28	31	2	1	1.09	1.01	5	<0.2	<10	50	0.31	0.07	0.82	0.13	46.4	9.1	46	1.38	46.1
RS-11	31	33																	
RS-12	7	9	17	1	0.01	1.04	5.3	<0.2	<10	50	0.36	0.07	0.67	0.08	54	8.5	35	1.15	34.3
RS-12	16	18	11	1	0.01	0.85	5.6	<0.2	<10	40	0.36	0.09	0.47	0.13	53.1	7.6	33	1.02	32.2
RS-12	20	22	5	1	0.25	1.97	10	<0.2	<10	50	0.33	0.19	1.04	0.18	43.5	43.9	47	1.18	775
RS-13	0	1.5	19	1	0.11	2.47	4.4	<0.2	<10	60	0.54	0.08	0.61	0.18	35.3	21.9	63	1.22	146.5
RS-13	1.5	2.5	6	1	0.1	2.67	7.2	<0.2	<10	120	0.55	0.06	0.39	0.08	58.9	19.9	90	4.21	88.1
RS-13	2.5	6																	
RS-13	8	10	6	1	0.49	2.69	20.7	<0.2	<10	90	0.54	0.25	1.02	0.39	46.3	67.2	72	2.4	1155
RS-14B	0	1.5	1	1	0.17	2.37	5.4	<0.2	<10	100	0.44	0.14	0.28	0.2	23	12	51	1.63	86.5
RS-14B	1.5	3	3	1	0.22	3	6.4	<0.2	<10	80	0.71	0.11	0.33	0.13	45.6	19.2	63	2.29	180.5
RS-14B	3	5																	
RS-16B	0	2	0	1	0.1	3.71	7	<0.2	<10	90	0.75	0.09	0.24	0.15	42.5	16.7	67	1.41	29.4

Fraction Analyzed			-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm
Parameter	**Depth**	**Depth**	**Fe**	**Ga**	**Ge**	**Hf**	**Hg**	**In**	**K**	**La**	**Li**	**Mg**	**Mn**	**Mo**	**Na**	**Nb**	**Ni**	**P**	**Pb**
Units	From (ft)	To (ft)	%	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	%	ppm	ppm	%	ppm	ppm	ppm	ppm
Detection Limit			0.01	0.05	0.05	0.02	0.01	0.005	0.01	0.2	0.1	0.01	5	0.05	0.01	0.05	0.2	10	0.2
General Method			Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia
Analytical Method Code			ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41
RS-01B	0	1	2.9	6.01	<0.05	<0.02	0.08	0.024	0.09	9.6	6.7	0.21	1550	1.51	0.02	1.02	24.4	1120	21.8
RS-01B	1	5	3.33	4.02	0.09	0.12	0.02	0.013	0.07	20.1	6	0.31	574	0.69	0.04	0.93	22.6	870	5.4
RS-01B	14	15	3.3	3.73	0.08	0.3	0.01	0.013	0.08	19.9	6.6	0.37	469	0.57	0.06	0.25	20.9	900	5.9
RS-01B	18	20	3.02	4.45	<0.05	0.32	0.01	0.013	0.11	19.1	7	0.6	465	0.67	0.16	0.21	32	810	4
RS-01B	20	25																	
RS-03	5	10	2.13	2.53	<0.05	0.07	0.12	0.013	0.04	17.3	4.3	0.3	179	12.1	0.03	0.91	60.5	580	6.5
RS-03	15	20	2.18	3.06	0.08	0.3	0.01	0.011	0.1	17	7.8	0.43	211	0.62	0.05	0.41	28	880	4.1
RS-03	20	22	3.65	5.19	<0.05	0.25	0.01	0.012	0.15	15.3	10	1.54	392	1.57	0.25	0.38	104	770	3.5
RS-04	1	5	4.12	8.02	<0.05	0.13	0.02	0.029	0.11	23.6	15.3	0.66	386	1.44	0.04	1.61	104	860	6.7
RS-04	10	15	2.43	3.4	<0.05	0.25	0.01	0.01	0.08	21	8.1	0.36	255	0.9	0.05	0.56	31	1010	4.4
RS-04	15	20	1.94	3.15	<0.05	0.33	0.01	0.012	0.08	18.4	7.5	0.42	183	0.64	0.06	0.41	37	960	3.8
RS-04	20	25	2.7	3.7	<0.05	0.31	0.01	0.012	0.1	16.7	9.5	0.72	233	0.97	0.08	0.29	88.4	880	4
RS-05A	5	10	3.29	5.88	<0.05	0.28	0.02	0.021	0.17	25.4	13.1	0.56	339	1.14	0.04	0.5	58.9	880	5.2
RS-05A	10	13	4.13	9.24	<0.05	0.08	0.01	0.011	0.13	9	9.3	2.64	501	2.88	0.62	0.18	170	370	2.6
RS-06A	0.5	2	2.95	6.81	<0.05	0.11	0.03	0.023	0.12	18.8	17.6	0.45	303	0.79	0.02	1.93	36.6	1240	6.1
RS-06A	2	4																	
RS-06A	5	7.5	3.01	5.26	<0.05	0.27	0.01	0.018	0.1	25.1	11.2	0.5	253	0.78	0.05	0.49	31.9	880	4.4
RS-06A	7.5	10																	
RS-06A	15	19	2.86	4.25	<0.05	0.34	0.01	0.014	0.13	23.3	9.9	0.46	332	1.1	0.05	0.38	33.6	1000	4.5
RS-06A	19	21																	
RS-07	1	2	5.23	12.4	<0.05	0.16	0.02	0.038	0.27	18.4	55.3	0.92	212	2.88	0.02	4.38	109.5	940	5.7
RS-07	2	3																	
RS-07	3	5																	
RS-07	5	6																	
RS-07	6	10																	
RS-07R	10	12																	
RS-07R	13.5	14.5																	
RS-07/RS-07R	6	14.5	6.42	9.13	0.15	0.32	0.03	0.045	0.51	25	32.7	1.01	289	4.77	0.06	1.62	118	1130	6.7
RS-08A	0	1	3.87	8.35	<0.05	0.14	0.06	0.028	0.12	19.9	20.4	0.53	232	1.34	0.03	2.74	62.6	620	5.8
RS-08A	1	5																	
RS-08A	5	11	3.84	5.52	0.09	0.36	0.03	0.022	0.23	22.5	14.4	0.68	385	1.25	0.06	0.34	66.9	920	4.2
RS-09	7	8	5.33	10.7	0.08	0.22	0.02	0.024	0.31	15.5	17.1	2.36	509	7.75	0.59	0.4	178.5	500	3.7
RS-10	1	2	3.12	8.92	<0.05	0.1	0.04	0.025	0.17	17.5	25	0.49	302	1.56	0.03	2.38	57.2	580	6.6
RS-10	2	3																	
RS-10	3	5.5																	
RS-10	5.5	7.5	3.52	7.67	0.09	0.49	0.01	0.021	0.46	27.7	29.2	0.92	369	1.16	0.03	0.46	38.3	1320	4.2
RS-10	7.5	10	5.78	8.57	0.18	0.48	0.01	0.031	0.54	22.4	22.3	1.12	423	1.44	0.09	0.28	85.1	820	7.6
RS-10	10	14																	
RS-11	0	9.5	3.01	8.34	0.1	0.22	0.03	0.035	0.29	19	26.4	0.93	227	1.45	0.09	2.77	102	740	6.4
RS-11	11.5	17	4.64	9.4	0.13	0.4	0.02	0.056	0.35	25.4	29	1.12	241	3.64	0.09	2.27	115.5	830	8.8
RS-11	17	25	9.75	9.8	0.31	0.47	0.03	0.071	0.52	20.3	36.3	1.83	358	5.44	0.11	1.11	444	920	9.3
RS-11	28	31	3.71	4.24	0.06	0.38	0.01	0.016	0.23	20.7	14.4	0.61	285	1.1	0.06	0.42	30.3	1010	3.8
RS-11	31	33																	
RS-12	7	9	2.68	4	0.05	0.34	0.01	0.013	0.11	26.7	10	0.45	332	0.55	0.03	0.38	24.5	1110	4.3
RS-12	16	18	2.58	3.21	0.06	0.29	0.01	0.014	0.09	22.2	8.3	0.28	290	0.84	0.03	0.39	24	1060	4.1
RS-12	20	22	4.51	5.52	0.09	0.26	0.02	0.028	0.16	19.7	9.9	1.43	447	2.62	0.2	0.35	374	1000	4.6
RS-13	0	1.5	3.74	6.85	<0.05	0.08	0.05	0.026	0.09	13.9	16	0.59	688	2.02	0.07	2.02	108.5	790	6.6
RS-13	1.5	2.5	4.23	8.59	0.05	0.29	0.02	0.029	0.56	23.7	23	0.98	413	1.1	0.04	2.25	84.4	820	4.2
RS-13	2.5	6																	
RS-13	8	10	5.68	7.77	0.1	0.28	0.02	0.033	0.3	20.5	19	0.78	412	5.04	0.18	0.82	517	880	5.7
RS-14B	0	1.5	3.56	6.6	<0.05	0.03	0.08	0.025	0.11	9.2	15.5	0.41	400	1.23	0.02	1.7	59.1	610	13.8
RS-14B	1.5	3	3.88	8.65	<0.05	0.09	0.06	0.029	0.2	17.7	24.2	0.62	327	1.65	0.04	2.42	94.6	670	5.6
RS-14B	3	5																	
RS-16B	0	2	4.02	8.8	<0.05	0.11	0.05	0.031	0.06	12.2	14.4	0.57	277	1.06	0.03	2.04	62.2	920	6.7

Fraction Analyzed			-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm
Parameter	**Depth**	**Depth**	**Rb**	**Re**	**S**	**Sb**	**Sc**	**Se**	**Sn**	**Sr**	**Ta**	**Te**	**Th**	**Ti**	**Tl**	**U**	**V**	**W**	**Y**
Units	From (ft)	To (ft)	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm
Detection Limit			0.1	0.001	0.01	0.05	0.1	0.2	0.2	0.2	0.01	0.01	0.2	0.005	0.02	0.05	1	0.05	0.05
General Method			Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia	Aqua Regia
Analytical Method Code			ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41
RS-01B	0	1	14.7	<0.001	0.05	0.19	1.9	0.6	1.7	29.3	<0.01	0.03	0.7	0.061	0.09	0.44	56	1.68	2.5
RS-01B	1	5	8.5	<0.001	0.02	0.05	3.6	0.4	0.6	26.1	<0.01	0.01	5	0.099	0.06	0.74	51	0.35	6.2
RS-01B	14	15	8.6	<0.001	0.02	0.08	4.2	0.3	0.6	31	<0.01	0.01	4.6	0.104	0.06	0.68	50	0.32	7.64
RS-01B	18	20	8.8	<0.001	0.03	0.08	3.7	0.3	1.4	42	<0.01	0.01	4.5	0.106	0.07	0.67	46	1.66	7.43
RS-01B	20	25																	
RS-03	5	10	4.1	0.011	0.46	0.35	3.3	2.8	1.2	48.5	0.01	0.03	6.6	0.035	0.06	3.47	31	3.81	8.45
RS-03	15	20	7.7	0.001	0.07	0.14	2.6	0.4	0.4	30.2	<0.01	0.02	3.6	0.096	0.06	0.95	44	0.3	6.96
RS-03	20	22	9.7	<0.001	0.05	0.17	3.7	1.6	1.1	54.9	0.01	0.02	3.7	0.12	0.08	1.05	43	2.82	6.7
RS-04	1	5	16.5	<0.001	0.01	0.24	7	0.6	0.8	29	<0.01	0.03	5	0.137	0.16	1.56	80	0.3	10.6
RS-04	10	15	7.2	<0.001	0.01	0.11	3	0.5	0.6	30	<0.01	0.01	5.2	0.107	0.06	0.9	46	0.62	7.82
RS-04	15	20	6.3	<0.001	0.05	0.07	2.7	0.3	0.6	30.1	<0.01	0.01	4.9	0.106	0.05	1.25	43	0.91	7.48
RS-04	20	25	7.9	0.002	0.12	0.12	3.6	0.4	0.6	30.5	<0.01	0.02	4.3	0.103	0.08	1.26	46	0.57	7.18
RS-05A	5	10	21.8	<0.001	<0.01	0.24	6.4	0.4	0.8	29.1	<0.01	0.03	6	0.141	0.18	1.43	64	1.31	10.7
RS-05A	10	13	7	0.001	<0.01	0.08	3.7	0.3	0.5	114	<0.01	0.01	1.8	0.067	0.06	0.48	25	0.79	3.79
RS-06A	0.5	2	16.7	<0.001	0.01	0.13	4	0.7	0.7	22.3	<0.01	0.02	4.8	0.121	0.13	0.97	65	0.34	5.87
RS-06A	2	4																	
RS-06A	5	7.5	11.4	<0.001	<0.01	0.14	5.4	0.5	0.7	35.3	<0.01	0.02	5.2	0.129	0.11	1.12	58	1.41	10.15
RS-06A	7.5	10																	
RS-06A	15	19	13	<0.001	<0.01	0.14	4.1	0.3	0.6	31.7	<0.01	0.02	5.5	0.119	0.1	1.01	51	0.68	8.58
RS-06A	19	21																	
RS-07	1	2	36.3	0.001	0.04	0.25	9.1	1	1	18.3	<0.01	0.03	5.3	0.221	0.2	1.76	97	0.71	7.18
RS-07	2	3																	
RS-07	3	5																	
RS-07	5	6																	
RS-07	6	10																	
RS-07R	10	12																	
RS-07R	13.5	14.5																	
RS-07/RS-07R	6	14.5	44.4	0.002	0.27	0.72	10.8	1.3	0.9	26	0.01	0.05	6.5	0.197	0.4	3.5	104	6.67	14.6
RS-08A	0	1	19.2	<0.001	0.02	0.21	5.6	1.1	0.8	25.3	<0.01	0.04	4.4	0.143	0.16	1.22	69	0.39	7.86
RS-08A	1	5																	
RS-08A	5	11	24.8	<0.001	0.01	0.31	5.4	0.4	0.7	32.7	0.01	0.02	5.4	0.14	0.19	1.37	58	11.95	8.99
RS-09	7	8	21.1	0.001	0.03	0.27	6.4	0.4	0.8	113.5	0.01	0.03	4	0.129	0.19	1.62	50	1.58	7.26
RS-10	1	2	23.2	<0.001	0.01	0.15	4.8	0.7	1	24.3	<0.01	0.02	5.4	0.131	0.15	1.14	72	3.37	5.67
RS-10	2	3																	
RS-10	3	5.5																	
RS-10	5.5	7.5	38.3	<0.001	0.01	0.29	7.5	0.5	0.8	30.3	<0.01	0.03	6.6	0.189	0.29	1.82	72	0.71	9.44
RS-10	7.5	10	46.5	<0.001	0.01	0.42	9.1	0.4	0.8	36.2	<0.01	0.03	5.7	0.192	0.33	1.92	74	1.96	10
RS-10	10	14																	
RS-11	0	9.5	27.9	0.001	0.07	0.59	8.4	1.1	1	30.9	<0.01	0.04	4.7	0.191	0.26	2.66	102	1.52	9.94
RS-11	11.5	17	37.1	0.002	0.19	1.01	10.7	1.5	1	29.9	0.01	0.05	6	0.223	0.32	4.54	130	0.7	13.35
RS-11	17	25	43.7	0.006	0.65	1.68	10.4	3.3	3.5	36.9	<0.01	0.14	5.5	0.205	0.4	4.46	95	4.87	13.45
RS-11	28	31	16.3	<0.001	0.09	0.18	3.9	0.4	0.7	31.8	<0.01	0.02	5.2	0.126	0.13	1.29	53	0.68	8
RS-11	31	33																	
RS-12	7	9	11.3	<0.001	<0.01	0.12	4.5	0.4	0.5	30	0.01	0.01	5.7	0.109	0.11	1.07	51	0.19	10.6
RS-12	16	18	9.8	<0.001	<0.01	0.18	3.5	0.4	0.4	26.8	0.01	0.02	5.9	0.098	0.11	1.38	48	0.31	9.98
RS-12	20	22	14.2	0.002	0.19	0.23	4.3	0.9	0.7	49.8	0.01	0.04	4.7	0.12	0.1	1.08	49	1.32	9.84
RS-13	0	1.5	10.7	0.001	0.03	0.18	4.4	0.9	0.7	30.8	0.01	0.03	3.2	0.129	0.11	0.96	89	1.53	6.62
RS-13	1.5	2.5	44.4	0.001	0.01	0.16	10.3	0.7	0.9	21.2	0.01	0.03	5.9	0.217	0.33	1.38	97	0.48	9.45
RS-13	2.5	6																	
RS-13	8	10	25.5	0.005	0.58	0.74	6.2	1.9	1	51.1	<0.01	0.07	4.6	0.156	0.28	1.35	69	0.98	9.14
RS-14B	0	1.5	16.1	<0.001	0.04	0.26	3	0.7	0.8	15.8	<0.01	0.03	1.6	0.096	0.12	0.68	71	0.58	3.26
RS-14B	1.5	3	20.5	<0.001	0.02	0.22	5.9	1	0.8	19.7	0.01	0.03	4.4	0.131	0.21	1.54	73	1.2	7.62
RS-14B	3	5																	
RS-16B	0	2	12.7	<0.001	0.01	0.15	4.4	0.8	0.8	15.9	0.01	0.03	4.2	0.138	0.1	0.91	114	0.18	4.77

Fraction Analyzed			-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm
Parameter	**Depth**	**Depth**	**Zn**	**Zr**	**Ag**	**Al**	**As**	**Ba**	**Be**	**Bi**	**Ca**	**Cd**	**Co**	**Cr**	**Cu**	**Fe**	**Ga**	**K**	**La**	**Mg**	**Mn**	**Mo**	**Na**
Units	From (ft)	To (ft)	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	ppm	%	ppm	%	ppm	ppm	%
Detection Limit			2	0.5	1	0.05	50	50	10	20	0.05	10	10	10	10	0.05	50	0.1	50	0.05	10	10	0.05
General Method			Aqua Regia	Aqua Regia	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid
Analytical Method Code			ME-MS41	ME-MS41	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a
RS-01B	0	1	168	<0.5	3	3.61	<50	520	<10	<20	1.55	<10	10	100	20	4.06	<50	0.7	<50	0.6	1690	<10	1.59
RS-01B	1	5	24	5.4	1	4.15	<50	510	<10	<20	1.8	<10	10	80	20	4.39	<50	1.2	<50	0.67	840	<10	2.29
RS-01B	14	15	28	10.5	<1	3.87	<50	500	<10	<20	2.04	<10	10	80	30	4.42	<50	1.1	<50	0.7	750	<10	2.39
RS-01B	18	20	33	10.7	9	2.99	<50	470	<10	<20	2.38	<10	10	100	30	4.24	<50	1.1	<50	0.85	780	<10	2.42
RS-01B	20	25																					
RS-03	5	10	24	2.9	1	2.26	<50	270	<10	<20	1.85	<10	<10	70	260	2.77	<50	0.7	<50	0.52	310	10	0.92
RS-03	15	20	30	9.5	<1	3.83	<50	520	<10	<20	2.68	<10	10	80	40	3.72	<50	1.2	<50	0.9	570	<10	2.47
RS-03	20	22	48	8	5	4.17	<50	440	<10	<20	3.21	<10	20	90	110	4.86	<50	1	<50	1.76	710	<10	2.28
RS-04	1	5	63	5.2	<1	4.89	<50	480	<10	<20	1.76	<10	20	100	240	5.17	<50	0.9	<50	0.93	640	<10	1.87
RS-04	10	15	29	9.3	<1	3.07	<50	520	<10	<20	2.16	<10	10	80	30	3.4	<50	1.3	<50	0.72	520	<10	2.48
RS-04	15	20	26	10.1	<1	3.15	<50	500	<10	<20	2.29	<10	10	80	50	3.06	<50	1.5	<50	0.82	460	<10	2.54
RS-04	20	25	34	9.9	2	3.97	<50	490	<10	<20	2.52	<10	20	90	140	4.05	<50	1.3	<50	1.14	560	<10	2.43
RS-05A	5	10	52	10.8	1	4.63	<50	550	<10	<20	2.02	<10	20	100	100	4.48	<50	1	<50	0.91	640	<10	2.3
RS-05A	10	13	53	2.8	2	6.88	<50	300	<10	<20	5.56	<10	40	270	60	5.26	<50	0.5	<50	2.57	720	<10	2.11
RS-06A	0.5	2	48	4.6	<1	4.28	<50	620	<10	<20	1.63	<10	10	100	30	4.06	<50	0.9	<50	0.84	590	<10	2.04
RS-06A	2	4																					
RS-06A	5	7.5	37	10.7	<1	3.55	<50	520	<10	<20	1.96	<10	10	90	70	4.04	<50	1.2	<50	0.79	550	<10	2.3
RS-06A	7.5	10																					
RS-06A	15	19	38	11.4	<1	3.33	<50	520	<10	<20	2.07	<10	10	80	50	3.94	<50	1.1	<50	0.82	640	<10	2.4
RS-06A	19	21																					
RS-07	1	2	142	6.4	<1	3.91	<50	450	<10	<20	1.29	<10	20	150	110	6.41	<50	0.8	<50	1.3	540	<10	1.45
RS-07	2	3																					
RS-07	3	5																					
RS-07	5	6																					
RS-07	6	10																					
RS-07R	10	12																					
RS-07R	13.5	14.5																					
RS-07/RS-07R	6	14.5	131	12.3	2	4.91	<50	520	<10	<20	1.76	<10	20	130	240	8.26	<50	1.3	<50	1.42	790	<10	1.7
RS-08A	0	1	52	5.5	<1	4.57	<50	510	<10	<20	1.38	<10	10	100	70	4.94	<50	1.1	<50	0.87	550	<10	1.81
RS-08A	1	5																					
RS-08A	5	11	61	13.4	<1	3.9	<50	560	<10	<20	2	<10	20	90	120	5.28	<50	1.3	<50	1.05	810	<10	2.24
RS-09	7	8	74	8.3	<1	5.39	<50	370	<10	<20	3.84	<10	30	140	100	6.15	<50	0.9	<50	2.22	760	<10	1.97
RS-10	1	2	54	4	<1	4.01	<50	510	<10	<20	1.15	<10	20	100	40	3.92	<50	0.9	<50	0.76	520	<10	1.57
RS-10	2	3																					
RS-10	3	5.5																					
RS-10	5.5	7.5	70	18.2	2	4.72	<50	800	<10	<20	1.91	<10	10	80	50	4.46	<50	1.4	<50	1.18	650	<10	2.45
RS-10	7.5	10	78	17.6	<1	4.61	<50	490	<10	<20	1.96	<10	20	110	90	6.81	<50	1.2	<50	1.31	800	<10	1.98
RS-10	10	14																					
RS-11	0	9.5	120	7.7	3	3.69	<50	360	<10	<20	2.33	<10	20	210	170	5	<50	0.9	<50	1.43	660	<10	1.48
RS-11	11.5	17	159	13.8	1	3.93	<50	410	<10	<20	2.2	<10	20	160	290	6.17	<50	0.9	<50	1.48	640	<10	1.69
RS-11	17	25	210	15.8	24	5.6	<50	390	<10	<20	2.35	<10	40	140	1510	12.25	<50	1.1	<50	2.32	930	<10	1.36
RS-11	28	31	40	11.5	1	3.8	<50	600	<10	<20	2.36	<10	10	80	40	5.18	<50	1.4	<50	1.05	740	<10	2.4
RS-11	31	33																					
RS-12	7	9	32	11.1	<1	3.72	<50	560	<10	<20	2.15	<10	10	70	30	3.73	<50	1.3	<50	0.8	640	<10	2.42
RS-12	16	18	38	9.5	<1	3.26	<50	550	<10	<20	1.97	<10	10	80	30	3.65	<50	1.4	<50	0.69	600	<10	2.47
RS-12	20	22	53	8.8	<1	4.62	<50	430	<10	<20	3	<10	40	100	780	5.89	<50	1	<50	1.76	780	<10	2.14
RS-13	0	1.5	61	2.8	1	4.43	<50	410	<10	<20	2.29	<10	20	160	150	5.63	<50	0.7	<50	1.13	1110	<10	1.71
RS-13	1.5	2.5	71	12.3	<1	4.57	<50	430	<10	<20	1.81	<10	20	130	90	5.3	<50	0.9	<50	1.2	700	<10	2.11
RS-13	2.5	6																					
RS-13	8	10	58	11.2	<1	5.5	<50	500	<10	<20	2.39	<10	70	140	1120	8.21	<50	1.2	<50	1.57	990	<10	1.95
RS-14B	0	1.5	69	1	<1	3.36	<50	410	<10	<20	1.15	<10	20	120	120	4.43	<50	1.1	<50	0.73	650	<10	1.23
RS-14B	1.5	3	64	3.2	<1	4.11	<50	460	<10	<20	1.52	<10	20	120	190	4.96	<50	1.2	<50	0.95	640	<10	1.66
RS-14B	3	5																					
RS-16B	0	2	51	4.2	<1	4.62	<50	410	<10	<20	1.59	<10	20	160	40	5.64	<50	1	<50	1.02	630	<10	1.42

Fraction Analyzed			-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm
Parameter	**Depth**	**Depth**	**Ni**	**P**	**Pb**	**S**	**Sb**	**Sc**	**Sr**	**Th**	**Ti**	**Tl**	**U**	**V**	**W**	**Zn**	**CO_2**	**Total Sulphur**	**Sulphate**	**MPA**	**NP**
Units	From (ft)	To (ft)	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	% CO2	% S	% S	kgCaCO3/t ore	kgCaCO3/t ore
Detection Limit			10	50	20	0.1	50	10	10	50	0.05	50	50	10	50	20	0.2	0.01	0.01	0.3	1
General Method			4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	4-Acid	ABA	ABA	ABA	ABA	ABA
Analytical Method Code			ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	C-GAS05	S-IR08	S-GRA06a	OA-VOL08	OA-VOL08
RS-01B	0	1	40	1140	<20	<0.1	<50	<10	260	<50	0.44	<50	<50	90	<50	110	<0.2	0.09	<0.01	1.6	3
RS-01B	1	5	30	810	<20	<0.1	<50	10	360	<50	0.34	<50	<50	80	<50	40	<0.2	<0.01	<0.01	<0.3	1
RS-01B	14	15	30	860	<20	<0.1	<50	10	360	<50	0.34	<50	<50	80	<50	40	<0.2	<0.01	<0.01	<0.3	3
RS-01B	18	20	50	760	<20	<0.1	<50	<10	340	<50	0.32	<50	<50	80	<50	50	<0.2	0.01	<0.01	0.3	9
RS-01B	20	25																			
RS-03	5	10	60	590	<20	0.5	<50	<10	180	<50	0.2	<50	<50	50	<50	30	<0.2	0.31	<0.01	9.7	19
RS-03	15	20	40	860	<20	0.1	<50	10	370	<50	0.39	<50	<50	90	<50	40	0.2	0.05	<0.01	1.6	9
RS-03	20	22	120	730	<20	<0.1	<50	<10	330	<50	0.39	<50	<50	80	<50	60	<0.2	0.02	<0.01	0.6	26
RS-04	1	5	110	880	<20	<0.1	<50	10	300	<50	0.41	<50	<50	110	<50	70	<0.2	<0.01	<0.01	<0.3	6
RS-04	10	15	40	930	<20	<0.1	<50	<10	360	<50	0.35	<50	<50	80	<50	40	<0.2	<0.01	<0.01	<0.3	4
RS-04	15	20	50	890	<20	<0.1	<50	<10	350	<50	0.36	<50	<50	80	<50	40	<0.2	0.02	<0.01	0.6	7
RS-04	20	25	110	860	<20	0.1	<50	10	360	<50	0.37	<50	<50	90	<50	50	<0.2	0.12	<0.01	3.8	19
RS-05A	5	10	60	860	<20	<0.1	<50	10	360	<50	0.39	<50	<50	100	<50	60	<0.2	<0.01	<0.01	<0.3	25
RS-05A	10	13	190	360	<20	<0.1	<50	<10	320	<50	0.25	<50	<50	60	<50	60	<0.2	<0.01	<0.01	<0.3	5
RS-06A	0.5	2	50	1230	<20	<0.1	<50	10	340	<50	0.44	<50	<50	100	<50	60	<0.2	<0.01	<0.01	<0.3	3
RS-06A	2	4																			
RS-06A	5	7.5	80	810	30	<0.1	<50	10	340	<50	0.38	<50	<50	90	<50	60	<0.2	<0.01	<0.01	<0.3	6
RS-06A	7.5	10																			
RS-06A	15	19	50	920	<20	<0.1	<50	10	340	<50	0.37	<50	<50	80	<50	50	<0.2	<0.01	<0.01	<0.3	6
RS-06A	19	21																			
RS-07	1	2	120	1010	<20	<0.1	<50	10	200	<50	0.46	<50	<50	130	<50	160	<0.2	0.01	<0.01	0.3	2
RS-07	2	3																			
RS-07	3	5																			
RS-07	5	6																			
RS-07	6	10																			
RS-07R	10	12																			
RS-07R	13.5	14.5																			
RS-07/RS-07R	6	14.5	130	1160	<20	0.2	<50	10	240	<50	0.42	<50	<50	130	<50	150	<0.2	0.06	<0.01	1.9	4
RS-08A	0	1	70	610	<20	<0.1	<50	10	290	<50	0.39	<50	<50	90	<50	60	<0.2	<0.01	<0.01	<0.3	4
RS-08A	1	5																			
RS-08A	5	11	70	890	<20	<0.1	<50	10	350	<50	0.38	<50	<50	90	<50	80	<0.2	<0.01	<0.01	<0.3	6
RS-09	7	8	190	500	<20	<0.1	<50	10	300	<50	0.3	<50	<50	80	<50	80	<0.2	0.02	<0.01	0.6	19
RS-10	1	2	70	590	<20	<0.1	<50	10	210	<50	0.44	<50	<50	100	<50	70	<0.2	<0.01	<0.01	<0.3	2
RS-10	2	3																			
RS-10	3	5.5																			
RS-10	5.5	7.5	40	1230	<20	<0.1	<50	10	450	<50	0.33	<50	<50	90	<50	80	<0.2	<0.01	<0.01	<0.3	6
RS-10	7.5	10	100	780	<20	<0.1	<50	10	290	<50	0.3	<50	<50	90	<50	80	<0.2	0.01	<0.01	0.3	12
RS-10	10	14																			
RS-11	0	9.5	130	790	<20	0.1	<50	10	180	<50	0.66	<50	<50	150	<50	150	<0.2	0.03	<0.01	0.9	7
RS-11	11.5	17	130	840	<20	0.2	<50	10	210	<50	0.65	<50	<50	170	<50	180	<0.2	0.03	<0.01	0.9	5
RS-11	17	25	470	950	<20	0.6	<50	10	200	<50	0.4	<50	<50	130	<50	240	<0.2	0.11	<0.01	3.4	5
RS-11	28	31	40	920	<20	0.1	<50	10	390	<50	0.33	<50	<50	90	<50	60	0.2	0.05	<0.01	1.6	11
RS-11	31	33																			
RS-12	7	9	40	1030	<20	<0.1	<50	10	380	<50	0.36	<50	<50	80	<50	50	<0.2	<0.01	<0.01	<0.3	5
RS-12	16	18	40	970	<20	<0.1	<50	10	350	<50	0.36	<50	<50	80	<50	50	<0.2	<0.01	<0.01	<0.3	3
RS-12	20	22	390	950	<20	0.2	<50	10	330	<50	0.46	<50	<50	90	<50	70	<0.2	0.05	0.04	1.6	20
RS-13	0	1.5	130	870	<20	<0.1	<50	10	250	<50	0.74	<50	<50	140	<50	80	<0.2	0.01	0.02	0.3	9
RS-13	1.5	2.5	100	850	<20	<0.1	<50	10	300	<50	0.51	<50	<50	120	<50	80	<0.2	<0.01	0.02	<0.3	5
RS-13	2.5	6																			
RS-13	8	10	530	860	<20	0.5	<50	10	330	<50	0.51	<50	<50	110	<50	90	<0.2	0.5	0.07	15.6	10
RS-14B	0	1.5	90	670	40	<0.1	<50	10	180	<50	0.51	<50	<50	110	<50	110	<0.2	0.09	0.01	2.8	3
RS-14B	1.5	3	110	710	<20	<0.1	<50	10	240	<50	0.47	<50	<50	110	<50	90	<0.2	0.02	0.02	0.6	4
RS-14B	3	5																			
RS-16B	0	2	80	950	<20	<0.1	<50	10	190	<50	0.93	<50	<50	180	<50	80	<0.2	0.03	0.03	0.9	3

Fraction Analyzed			-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm
Parameter	**Depth**	**Depth**	**Fizz Rating**	**Ag**	**Al**	**As**	**Au**	**B**	**Ba**	**Be**	**Bi**	**Ca**	**Cd**	**Ce**	**Co**	**Cr**	**Cs**	**Cu**
Units	From (ft)	To (ft)	Unity	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm
Detection Limit			1	0.01	0.01	0.1	0.2	10	10	0.05	0.01	0.01	0.01	0.02	0.1	1	0.05	0.2
General Method			ABA															
Analytical Method Code			OA-VOL08	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41
RS-01B	0	1	1	2.61	0.94	5.3	<0.2	<10	160	0.28	0.2	0.67	0.48	17.6	7.8	66	0.95	13
RS-01B	1	5	1	0.81	1	5.7	<0.2	<10	40	0.34	0.04	0.36	0.07	35.6	7.3	115	0.9	20
RS-01B	14	15	1	0.64	0.78	3.6	<0.2	<10	40	0.29	0.03	0.44	0.08	31.1	7.4	117	0.83	25.8
RS-01B	18	20	1	7.85	0.97	2.8	<0.2	<10	40	0.26	0.03	0.59	0.06	30.3	8.5	105	0.72	26.2
RS-01B	20	25																
RS-03	5	10	1	0.36	0.78	3.4	<0.2	<10	70	0.31	0.04	1.19	0.45	22.1	4.8	246	0.53	217
RS-03	15	20	1	0.09	0.77	4.2	<0.2	<10	40	0.27	0.03	0.62	0.1	30.7	6.3	118	0.68	36.1
RS-03	20	22	1	3.39	1.55	2.2	<0.2	<10	50	0.27	0.04	0.95	0.1	30.2	16.6	102	1.07	91.5
RS-04	1	5	1	0.14	1.74	5.7	<0.2	<10	60	0.41	0.11	0.48	0.1	34.8	14.3	124	1.34	146.5
RS-04	10	15	1	0.47	0.64	1.4	<0.2	<10	30	0.25	0.02	0.41	0.04	30.8	5.3	109	0.53	21.5
RS-04	15	20	1	0.36	0.66	1.2	<0.2	<10	30	0.19	0.03	0.46	0.05	30.8	5.9	110	0.62	63.6
RS-04	20	25	1	0.69	1.08	2.3	<0.2	<10	30	0.26	0.06	0.57	0.13	27.8	14.3	104	1.06	187.5
RS-05A	5	10	1	0.07	1.1	2.8	<0.2	<10	50	0.38	0.06	0.46	0.09	34	10.1	123	0.95	60
RS-05A	10	13	1	0.05	3.58	1.3	<0.2	<10	40	0.19	<0.01	2.03	0.08	15.7	29.8	133	0.57	36.1
RS-06A	0.5	2	1	0.11	1.61	2.7	<0.2	<10	70	0.4	0.06	0.32	0.07	38.1	9.8	95	1.39	20.9
RS-06A	2	4																
RS-06A	5	7.5	1	0.14	1.13	3.2	<0.2	<10	50	0.32	0.06	0.46	0.06	39	9.4	105	1.01	29.3
RS-06A	7.5	10																
RS-06A	15	19	1	0.11	0.76	3.7	<0.2	<10	40	0.29	0.03	0.44	0.08	33.5	7.3	107	0.73	23.7
RS-06A	19	21																
RS-07	1	2	1	0.01	1.63	5.8	<0.2	<10	60	0.46	0.07	0.28	0.07	29.1	8.3	120	2.07	59.2
RS-07	2	3																
RS-07	3	5																
RS-07	5	6																
RS-07	6	10																
RS-07R	10	12																
RS-07R	13.5	14.5																
RS-07/RS-07R	6	14.5	1	0.11	1.82	6.9	<0.2	<10	70	0.66	0.17	0.46	0.27	34.6	16.1	138	2.49	140.5
RS-08A	0	1	1	0.1	1.87	5.6	<0.2	<10	50	0.54	0.08	0.41	0.07	31.7	12.3	98	1.39	46.1
RS-08A	1	5																
RS-08A	5	11	1	0.38	1.17	8.5	<0.2	<10	50	0.36	0.06	0.49	0.15	35.2	14.3	115	1.49	74.5
RS-09	7	8	1	0.08	4.06	4.2	<0.2	<10	70	0.41	0.06	2.04	0.15	29.4	27.5	140	2.03	85.7
RS-10	1	2	1	0.05	2	3.6	<0.2	<10	80	0.45	0.09	0.37	0.05	32.7	12.9	111	1.79	29.4
RS-10	2	3																
RS-10	3	5.5																
RS-10	5.5	7.5	1	0.05	1.57	3.6	<0.2	<10	80	0.47	0.09	0.39	0.05	42.2	10.1	131	2.59	36.6
RS-10	7.5	10	1	0.06	2.13	7.1	<0.2	<10	80	0.47	0.09	0.67	0.12	34.6	16.5	125	2.75	63.7
RS-10	10	14																
RS-11	0	9.5	1	0.79	1.87	3.2	<0.2	<10	70	0.48	0.09	0.64	0.28	32.7	13.3	143	1.96	115
RS-11	11.5	17	1	0.12	1.97	4.7	<0.2	<10	70	0.53	0.1	0.66	0.21	33.9	12.7	142	2.17	155
RS-11	17	25	1	6.33	2.11	6	<0.2	<10	70	0.53	0.17	0.72	0.32	30.3	19.4	137	2.41	680
RS-11	28	31	2	0.36	0.9	3.5	<0.2	<10	50	0.35	0.05	0.57	0.09	31.5	7.1	113	1.15	34.3
RS-11	31	33																
RS-12	7	9	1	0.03	0.71	3	<0.2	<10	40	0.29	0.06	0.4	0.05	29.4	5.4	120	0.67	17
RS-12	16	18	1	0.03	0.63	2.6	<0.2	<10	30	0.26	0.06	0.37	0.08	30.3	5	105	0.63	17.2
RS-12	20	22	1	0.13	1.24	4.2	<0.2	<10	40	0.26	0.07	0.66	0.1	27.4	19.5	113	0.68	342
RS-13	0	1.5	1	0.08	1.83	3.5	<0.2	<10	50	0.44	0.05	0.6	0.15	30.1	17.3	99	0.88	97.2
RS-13	1.5	2.5	1	0.07	1.75	3.8	<0.2	<10	60	0.46	0.03	0.5	0.07	36.2	14.4	113	1.43	62
RS-13	2.5	6																
RS-13	8	10	1	0.3	1.88	9.8	<0.2	<10	60	0.36	0.15	0.78	0.27	34.4	48.4	120	1.43	817
RS-14B	0	1.5	1	0.21	1.83	4.9	<0.2	<10	80	0.38	0.13	0.3	0.24	24.4	13.5	90	1.45	73.8
RS-14B	1.5	3	1	0.14	2.26	4.9	<0.2	<10	60	0.48	0.09	0.34	0.12	31.4	16.4	88	1.78	116.5
RS-14B	3	5																
RS-16B	0	2	1	0.07	2.24	4.6	<0.2	<10	60	0.45	0.07	0.3	0.14	28.2	15.2	84	0.99	22.9

Fraction Analyzed			-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm
Parameter	**Depth**	**Depth**	**Fe**	**Ga**	**Ge**	**Hf**	**Hg**	**In**	**K**	**La**	**Li**	**Mg**	**Mn**	**Mo**	**Na**	**Nb**	**Ni**	**P**
Units	From (ft)	To (ft)	%	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	%	ppm	ppm	%	ppm	ppm	ppm
Detection Limit			0.01	0.05	0.05	0.02	0.01	0.005	0.01	0.2	0.1	0.01	5	0.05	0.01	0.05	0.2	10
General Method																		
Analytical Method Code			ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41
RS-01B	0	1	2.53	3.98	0.05	<0.02	0.08	0.015	0.08	7.6	5.7	0.22	1755	1.7	0.02	0.97	17.6	1010
RS-01B	1	5	2.99	3.49	0.13	0.17	0.02	0.011	0.08	16.6	7.5	0.33	581	2.02	0.04	0.63	18.9	560
RS-01B	14	15	2.72	3.05	0.13	0.27	0.01	0.01	0.09	15.6	7.6	0.34	523	2.01	0.05	0.54	20.6	550
RS-01B	18	20	2.54	3.32	0.12	0.25	0.01	0.01	0.11	14.6	7.5	0.5	411	1.91	0.12	0.21	26.5	510
RS-01B	20	25																
RS-03	5	10	2.91	2.19	0.11	0.05	0.1	0.01	0.04	12.3	3.8	0.25	286	42.9	0.02	0.94	70.2	490
RS-03	15	20	2.09	2.96	0.08	0.26	0.01	0.009	0.11	15.6	9.1	0.38	209	2.47	0.07	0.4	23.6	590
RS-03	20	22	2.98	4.55	0.1	0.24	0.01	0.012	0.16	14.9	10.6	1.15	322	2.68	0.22	0.28	79.6	490
RS-04	1	5	3.5	6.02	0.12	0.08	0.02	0.02	0.09	18.3	13.2	0.51	359	8.09	0.06	1.48	67.4	550
RS-04	10	15	1.62	2.75	0.05	0.23	<0.01	0.007	0.09	14.9	8.2	0.3	175	2.28	0.06	0.37	19.8	460
RS-04	15	20	1.5	2.71	0.06	0.26	0.01	0.008	0.09	15.8	8.1	0.34	158	2.02	0.07	0.32	27.6	490
RS-04	20	25	2.77	3.68	0.1	0.23	0.01	0.013	0.13	13.7	11.7	0.74	258	2.26	0.09	0.2	98	460
RS-05A	5	10	2.64	4.07	0.1	0.25	0.01	0.014	0.11	17.6	10.4	0.46	274	2.49	0.07	0.34	35.7	480
RS-05A	10	13	3.71	7.11	0.11	0.06	0.01	0.008	0.11	8.2	8.1	2.35	440	3.95	0.54	0.18	137.5	260
RS-06A	0.5	2	2.59	5.74	0.07	0.1	0.02	0.017	0.14	16.7	19.4	0.45	270	1.89	0.04	1.74	27.6	930
RS-06A	2	4																
RS-06A	5	7.5	2.62	4.26	0.11	0.27	0.01	0.013	0.12	20.3	10.8	0.46	285	2.09	0.06	0.42	26.1	620
RS-06A	7.5	10																
RS-06A	15	19	2.16	3.23	0.09	0.28	0.01	0.01	0.11	17.2	9.1	0.36	238	2.29	0.06	0.34	21.6	560
RS-06A	19	21																
RS-07	1	2	4.32	6.24	0.05	0.11	0.01	0.019	0.23	13.1	30.1	0.58	180	2.98	0.03	2.48	58.2	600
RS-07	2	3																
RS-07	3	5																
RS-07	5	6																
RS-07	6	10																
RS-07R	10	12																
RS-07R	13.5	14.5																
RS-07/RS-07R	6	14.5	5.42	6.32	0.22	0.25	0.01	0.026	0.37	17.5	28.1	0.76	246	3.71	0.07	1.01	72.3	670
RS-08A	0	1	3.66	5.76	0.12	0.1	0.03	0.018	0.11	15.8	15.8	0.52	257	2.02	0.05	1.86	47.8	620
RS-08A	1	5																
RS-08A	5	11	3.41	4.24	0.14	0.31	0.02	0.015	0.18	17.6	12.6	0.63	346	2.85	0.08	0.27	50.7	570
RS-09	7	8	4.88	8.86	0.17	0.21	0.01	0.019	0.29	13.9	15.3	2.04	470	11.65	0.52	0.34	151	440
RS-10	1	2	3.13	6.92	0.1	0.14	0.02	0.02	0.19	14.2	22	0.54	394	2.55	0.06	2.07	45	470
RS-10	2	3																
RS-10	3	5.5																
RS-10	5.5	7.5	2.87	6.1	0.12	0.39	0.01	0.016	0.37	21.5	23.5	0.69	326	2.39	0.05	0.41	31.1	660
RS-10	7.5	10	5.16	6.86	0.23	0.38	0.02	0.021	0.43	17.8	19.9	0.97	411	2.67	0.11	0.27	65.1	630
RS-10	10	14																
RS-11	0	9.5	2.45	6.28	0.19	0.17	0.02	0.025	0.25	16.2	21.1	0.68	234	2.69	0.09	2.29	68	510
RS-11	11.5	17	3.18	6.64	0.15	0.32	<0.01	0.029	0.29	17.6	22.4	0.73	207	3.16	0.12	1.4	64.3	570
RS-11	17	25	4.73	6.61	0.19	0.26	0.01	0.028	0.38	15.2	27.3	1.11	260	3.31	0.13	0.69	169.5	520
RS-11	28	31	2.74	3.6	0.11	0.27	<0.01	0.012	0.21	16.3	13.7	0.49	274	2.42	0.06	0.31	24.1	550
RS-11	31	33																
RS-12	7	9	2.03	2.93	0.08	0.26	0.01	0.01	0.1	16.2	8.4	0.32	239	2.17	0.05	0.35	15.1	510
RS-12	16	18	1.94	2.66	0.08	0.25	0.01	0.008	0.09	14.7	7.8	0.26	203	2.12	0.05	0.36	15.2	480
RS-12	20	22	3.02	3.76	0.11	0.2	0.01	0.014	0.11	13.5	8.6	0.89	297	3.05	0.13	0.27	160	470
RS-13	0	1.5	3.26	5.43	0.11	0.06	0.03	0.02	0.08	13	12.8	0.62	645	2.58	0.09	1.95	83.8	510
RS-13	1.5	2.5	3.21	5.65	0.1	0.22	0.01	0.017	0.19	16.8	14.9	0.71	344	2.18	0.09	1.4	66.7	500
RS-13	2.5	6																
RS-13	8	10	4.57	5.66	0.15	0.27	0.02	0.021	0.19	17.1	14.3	0.63	339	4.59	0.16	0.6	382	560
RS-14B	0	1.5	3.55	6.73	0.09	0.05	0.07	0.023	0.11	11.1	12.6	0.47	450	2.37	0.04	1.5	54.3	530
RS-14B	1.5	3	3.99	7.48	0.11	0.11	0.03	0.024	0.19	14	17.5	0.65	435	2.38	0.06	1.46	68.4	420
RS-14B	3	5																
RS-16B	0	2	3.46	7.39	0.08	0.13	0.03	0.023	0.06	11.6	10.4	0.67	361	1.72	0.06	1.43	54.9	600

Fraction Analyzed			-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm
Parameter	**Depth**	**Depth**	**Pb**	**Rb**	**Re**	**S**	**Sb**	**Sc**	**Se**	**Sn**	**Sr**	**Ta**	**Te**	**Th**	**Ti**	**Tl**	**U**	**V**
Units	From (ft)	To (ft)	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm
Detection Limit			0.2	0.1	0.001	0.01	0.05	0.1	0.2	0.2	0.2	0.01	0.01	0.2	0.005	0.02	0.05	1
General Method																		
Analytical Method Code			ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41
RS-01B	0	1	16.4	12.2	<0.001	0.06	0.17	1.3	0.6	1	44.6	<0.01	0.03	0.5	0.052	0.09	0.33	40
RS-01B	1	5	3.5	8.2	<0.001	<0.01	0.07	3.2	0.4	0.5	27.8	<0.01	<0.01	3.6	0.1	0.07	0.62	43
RS-01B	14	15	3.8	7.4	<0.001	<0.01	0.1	3.1	0.3	0.6	26.9	<0.01	<0.01	3.5	0.095	0.06	0.58	40
RS-01B	18	20	2.7	7.3	<0.001	0.01	0.08	2.7	0.3	1.1	34.5	<0.01	<0.01	3.5	0.089	0.06	0.53	35
RS-01B	20	25																
RS-03	5	10	3.4	3.7	0.014	0.53	0.34	2.9	2.9	1.9	47.6	0.02	0.01	1.5	0.028	0.07	1.14	25
RS-03	15	20	2.8	7.6	<0.001	0.09	0.1	2.6	0.3	0.4	30.3	<0.01	<0.01	4.1	0.096	0.06	0.87	36
RS-03	20	22	2.7	9.7	<0.001	0.05	0.12	3.2	0.9	0.8	48.4	<0.01	<0.01	3.2	0.116	0.08	0.81	38
RS-04	1	5	4.7	11	0.001	0.01	0.16	4.8	0.5	0.9	28.4	<0.01	<0.01	3.3	0.119	0.11	0.99	65
RS-04	10	15	2.7	6.2	<0.001	<0.01	0.05	2.2	0.4	0.4	27.1	<0.01	<0.01	3.7	0.085	0.04	0.59	30
RS-04	15	20	2.6	5.8	<0.001	0.03	0.05	2.2	0.3	0.4	27.6	<0.01	<0.01	4	0.085	0.04	0.86	30
RS-04	20	25	2.9	8.8	0.002	0.18	0.11	3.4	0.5	0.5	29.6	<0.01	<0.01	3.3	0.097	0.08	0.82	41
RS-05A	5	10	3.1	10.6	<0.001	<0.01	0.13	3.7	0.3	0.5	29	<0.01	<0.01	3.7	0.116	0.08	0.83	46
RS-05A	10	13	1.8	5.2	<0.001	<0.01	0.06	2.9	0.2	0.3	89.9	<0.01	<0.01	1.5	0.062	0.04	0.35	21
RS-06A	0.5	2	4.6	15.6	<0.001	0.01	0.09	3	0.5	0.6	23.1	<0.01	0.01	3.8	0.123	0.1	0.66	57
RS-06A	2	4																
RS-06A	5	7.5	3.4	10.9	<0.001	<0.01	0.12	4.1	0.4	0.6	32.1	<0.01	0.01	4	0.127	0.09	0.85	53
RS-06A	7.5	10																
RS-06A	15	19	2.8	8.9	<0.001	0.01	0.08	2.8	0.2	0.4	28.6	<0.01	<0.01	4.1	0.098	0.06	0.67	38
RS-06A	19	21																
RS-07	1	2	3.4	19.9	0.001	0.02	0.19	4.9	0.5	0.6	19	<0.01	0.02	3.6	0.133	0.13	1.07	61
RS-07	2	3																
RS-07	3	5																
RS-07	5	6																
RS-07	6	10																
RS-07R	10	12																
RS-07R	13.5	14.5																
RS-07/RS-07R	6	14.5	3.4	28.9	0.001	0.11	0.35	7.7	0.7	0.7	26.6	0.01	0.05	4.1	0.164	0.24	1.72	74
RS-08A	0	1	4.2	13	<0.001	0.01	0.16	3.8	0.5	0.6	25.6	<0.01	0.01	3.2	0.122	0.11	0.81	65
RS-08A	1	5																
RS-08A	5	11	2.9	16.4	<0.001	0.01	0.22	4.1	0.4	0.6	30.4	<0.01	0.02	4.3	0.112	0.12	0.98	45
RS-09	7	8	3	18.8	0.001	0.03	0.25	5.9	0.5	0.8	89.7	<0.01	0.01	3.4	0.125	0.15	1.23	48
RS-10	1	2	4.8	19.5	<0.001	0.01	0.13	4.3	0.6	0.8	25.9	<0.01	0.01	4	0.144	0.12	0.88	69
RS-10	2	3																
RS-10	3	5.5																
RS-10	5.5	7.5	3.1	30	<0.001	0.01	0.23	6.2	0.3	0.6	30.7	<0.01	0.01	4.6	0.165	0.2	1.29	58
RS-10	7.5	10	5	32.7	<0.001	0.02	0.27	7.2	0.3	0.7	36.3	<0.01	0.02	4.4	0.153	0.21	1.41	62
RS-10	10	14																
RS-11	0	9.5	4.2	21.6	0.001	0.06	0.37	6.5	0.8	1	29.7	<0.01	0.02	4	0.153	0.19	1.5	73
RS-11	11.5	17	4	24.9	0.001	0.08	0.41	6.8	0.6	0.7	33.4	<0.01	0.02	3.9	0.176	0.21	2.04	77
RS-11	17	25	3.5	28.3	0.002	0.2	0.47	7	1	1.4	36.2	<0.01	0.03	3.8	0.148	0.24	1.85	60
RS-11	28	31	2.5	14	0.001	0.1	0.12	3.4	0.4	0.5	28.9	<0.01	0.01	3.6	0.099	0.11	0.93	39
RS-11	31	33																
RS-12	7	9	2.6	7.8	<0.001	<0.01	0.07	2.8	0.2	0.4	26.3	<0.01	<0.01	3.1	0.09	0.06	0.58	36
RS-12	16	18	2.6	6.9	<0.001	<0.01	0.09	2.5	<0.2	0.4	25.6	<0.01	<0.01	3.3	0.083	0.07	0.77	34
RS-12	20	22	2.7	8.2	0.001	0.08	0.1	2.7	0.4	0.4	36.5	<0.01	<0.01	2.8	0.095	0.06	0.62	34
RS-13	0	1.5	5	7.7	0.001	0.02	0.15	3.2	0.6	0.7	33.3	0.01	<0.01	2.3	0.123	0.08	0.65	71
RS-13	1.5	2.5	3.3	15.1	<0.001	<0.01	0.11	4.5	0.5	0.6	27.6	<0.01	<0.01	3.7	0.152	0.12	0.77	67
RS-13	2.5	6																
RS-13	8	10	3.9	14.9	0.004	0.63	0.4	4.1	1.2	0.6	42.4	<0.01	0.03	3.4	0.133	0.16	0.85	58
RS-14B	0	1.5	13.2	15.3	<0.001	0.03	0.27	3	0.7	3.1	21.1	0.01	0.05	1.8	0.091	0.1	0.64	66
RS-14B	1.5	3	4.5	18.1	<0.001	0.02	0.18	5	0.7	1.1	21.1	0.01	0.04	3.4	0.12	0.16	0.99	65
RS-14B	3	5																
RS-16B	0	2	6.2	9.6	<0.001	0.01	0.14	3.2	0.5	1.4	20.5	0.01	0.03	3	0.126	0.07	0.61	89

Fraction Analyzed			-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm
Parameter	**Depth**	**Depth**	**W**	**Y**	**Zn**	**Zr**	**Ag**	**Al**	**As**	**Ba**	**Be**	**Bi**	**Ca**	**Cd**	**Co**	**Cr**	**Cu**	**Fe**
Units	From (ft)	To (ft)	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%
Detection Limit			0.05	0.05	2	0.5	1	0.05	50	50	10	20	0.05	10	10	10	10	0.05
General Method																		
Analytical Method Code			ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a
RS-01B	0	1	1.49	2.22	79	0.9	2	2.8	<50	500	<10	<20	1.54	<10	10	120	10	3.32
RS-01B	1	5	0.33	5.18	23	7.4	2	5.17	<50	590	<10	<20	1.65	<10	10	160	20	3.98
RS-01B	14	15	0.27	5.75	25	8.8	<1	3.87	<50	580	<10	<20	1.71	<10	10	170	30	3.68
RS-01B	18	20	1.24	5.28	26	8.1	6	4.67	<50	580	<10	<20	2.06	<10	10	160	20	3.59
RS-01B	20	25																
RS-03	5	10	1.69	7.38	18	3.1	<1	1.85	<50	210	<10	<20	1.49	<10	<10	220	210	2.68
RS-03	15	20	0.26	5.84	25	8.2	<1	4.36	<50	610	<10	<20	2.07	<10	10	160	30	3.08
RS-03	20	22	2.19	5.48	38	7.8	3	4.45	<50	530	<10	<20	2.65	<10	20	150	90	4.14
RS-04	1	5	0.28	7.31	41	4.4	<1	4.94	<50	580	<10	<20	1.74	<10	20	150	140	4.28
RS-04	10	15	0.38	4.89	19	7.4	<1	4.65	<50	650	<10	<20	1.76	<10	10	140	20	2.39
RS-04	15	20	0.34	5.12	20	7.6	<1	4.32	<50	630	<10	<20	1.82	<10	10	140	60	2.31
RS-04	20	25	0.57	5.2	32	8	<1	3.92	<50	560	<10	<20	1.91	<10	20	150	180	3.7
RS-05A	5	10	1.14	6.8	31	9.1	<1	5.05	<50	640	<10	<20	1.87	<10	10	180	60	3.81
RS-05A	10	13	0.54	2.97	42	2.7	<1	4.99	<50	320	<10	<20	4.88	<10	30	300	40	4.33
RS-06A	0.5	2	0.32	4.9	40	4.9	<1	5.3	<50	730	<10	<20	1.72	<10	20	150	20	3.71
RS-06A	2	4																
RS-06A	5	7.5	1.6	7.58	31	9.6	<1	4.86	<50	640	<10	<20	1.92	<10	10	150	30	3.81
RS-06A	7.5	10																
RS-06A	15	19	0.57	5.74	25	8.8	<1	5.03	<50	650	<10	<20	1.89	<10	10	150	20	3.27
RS-06A	19	21																
RS-07	1	2	0.59	4.95	64	4.4	<1	4.31	<50	670	<10	<20	1.49	<10	10	190	60	6.14
RS-07	2	3																
RS-07	3	5																
RS-07	5	6																
RS-07	6	10																
RS-07R	10	12																
RS-07R	13.5	14.5																
RS-07/RS-07R	6	14.5	3.21	8.71	82	9.4	<1	4.58	<50	630	<10	<20	1.72	<10	20	170	140	7.41
RS-08A	0	1	0.34	6.32	38	5.1	<1	3.99	<50	610	<10	<20	1.57	<10	10	150	40	4.78
RS-08A	1	5																
RS-08A	5	11	12.45	6.48	42	10.9	1	4.06	<50	630	<10	<20	1.76	<10	20	160	80	4.99
RS-09	7	8	1.42	6.03	63	8.1	1	5.28	<50	430	<10	<20	3.47	<10	30	180	90	5.77
RS-10	1	2	2.7	5.38	45	6.9	<1	3.52	<50	580	<10	<20	1.55	<10	20	170	30	4.08
RS-10	2	3																
RS-10	3	5.5																
RS-10	5.5	7.5	0.91	7.28	55	14.1	<1	5.39	<50	910	<10	<20	1.64	<10	10	150	40	3.77
RS-10	7.5	10	3.79	7.33	58	13.4	<1	4.66	<50	590	<10	<20	2	<10	20	170	60	6.69
RS-10	10	14																
RS-11	0	9.5	0.87	7.09	80	6.7	4	3.77	<50	500	<10	<20	2.13	<10	20	210	110	4.63
RS-11	11.5	17	0.41	8.38	92	10.9	<1	4.5	<50	550	<10	<20	2.41	<10	20	210	150	5.58
RS-11	17	25	2.92	7.3	99	8.8	5	5.62	<50	640	<10	<20	2.14	<10	20	180	650	6.6
RS-11	28	31	1	5.42	31	8.5	<1	4.54	<50	750	<10	<20	1.85	<10	10	140	30	4.33
RS-11	31	33																
RS-12	7	9	0.24	5.84	21	7.6	<1	4.36	<50	690	<10	<20	1.64	<10	10	150	20	2.84
RS-12	16	18	0.24	5.77	23	7.7	<1	5.02	<50	690	<10	<20	1.71	<10	10	130	10	2.84
RS-12	20	22	0.54	5.31	32	6.6	2	5.34	<50	630	<10	<20	2.25	<10	20	160	340	4.33
RS-13	0	1.5	0.73	6.29	44	3.3	3	4.49	<50	480	<10	<20	2.51	<10	30	180	100	5.05
RS-13	1.5	2.5	0.27	6.83	42	9.1	<1	4.49	<50	490	<10	<20	2.21	<10	20	180	60	4.81
RS-13	2.5	6																
RS-13	8	10	0.52	6.57	39	10.2	<1	5.32	<50	550	<10	<20	2.28	<10	50	200	800	7.2
RS-14B	0	1.5	0.57	4.13	58	1.3	2	4.2	<50	490	<10	<20	1.51	<10	20	150	80	5.07
RS-14B	1.5	3	1.5	5.9	53	3.4	1	4.88	<50	550	<10	<20	1.76	<10	20	150	130	5.57
RS-14B	3	5																
RS-16B	0	2	0.21	4.73	44	4.1	<1	4.86	<50	520	<10	<20	1.96	<10	20	170	30	5.15

Fraction Analyzed			-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm
Parameter	**Depth**	**Depth**	**Ga**	**K**	**La**	**Mg**	**Mn**	**Mo**	**Na**	**Ni**	**P**	**Pb**	**S**	**Sb**	**Sc**	**Sr**	**Th**	**Ti**
Units	From (ft)	To (ft)	ppm	%	ppm	%	ppm	ppm	%	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%
Detection Limit			50	0.1	50	0.05	10	10	0.05	10	50	20	0.1	50	10	10	50	0.05
General Method																		
Analytical Method Code			ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a
RS-01B	0	1	<50	0.9	<50	0.48	1860	<10	1.58	20	920	20	0.1	<50	<10	270	<50	0.28
RS-01B	1	5	<50	1.6	<50	0.67	890	<10	2.62	30	520	<20	<0.1	<50	10	450	<50	0.26
RS-01B	14	15	<50	1.4	<50	0.63	820	<10	2.73	30	510	<20	<0.1	<50	<10	420	<50	0.26
RS-01B	18	20	<50	1.7	<50	0.8	740	<10	2.8	40	490	<20	<0.1	<50	10	440	<50	0.25
RS-01B	20	25																
RS-03	5	10	<50	0.5	<50	0.35	330	30	0.65	70	480	<20	0.5	<50	<10	140	<50	0.11
RS-03	15	20	<50	1.7	<50	0.72	490	<10	2.81	30	530	20	0.1	<50	10	460	<50	0.26
RS-03	20	22	<50	1.4	<50	1.42	640	<10	2.65	90	510	<20	0.1	<50	10	400	<50	0.31
RS-04	1	5	<50	1	<50	0.81	620	<10	2.45	70	580	<20	<0.1	<50	10	420	<50	0.33
RS-04	10	15	<50	1.8	<50	0.57	370	<10	3.08	30	460	<20	<0.1	<50	<10	510	<50	0.22
RS-04	15	20	<50	1.8	<50	0.64	360	<10	3	30	460	<20	<0.1	<50	<10	470	<50	0.23
RS-04	20	25	<50	1.6	<50	1.01	540	<10	2.73	110	450	<20	0.1	<50	10	410	<50	0.32
RS-05A	5	10	<50	1.7	<50	0.8	610	<10	2.77	50	460	<20	<0.1	<50	10	470	<50	0.3
RS-05A	10	13	<50	0.7	<50	1.94	620	<10	2.42	150	240	<20	<0.1	<50	<10	350	<50	0.21
RS-06A	0.5	2	<50	1.4	<50	0.89	560	<10	2.64	40	940	<20	<0.1	<50	10	530	<50	0.36
RS-06A	2	4																
RS-06A	5	7.5	<50	1.7	<50	0.81	600	<10	2.78	40	580	<20	<0.1	<50	10	490	<50	0.34
RS-06A	7.5	10																
RS-06A	15	19	<50	1.8	<50	0.73	560	<10	2.91	30	540	20	<0.1	<50	10	500	<50	0.26
RS-06A	19	21																
RS-07	1	2	<50	1.7	<50	1.04	750	<10	2.25	70	620	<20	<0.1	<50	10	400	<50	0.31
RS-07	2	3																
RS-07	3	5																
RS-07	5	6																
RS-07	6	10																
RS-07R	10	12																
RS-07R	13.5	14.5																
RS-07/RS-07R	6	14.5	<50	1.8	<50	1.24	920	<10	2.23	80	650	<20	0.1	<50	10	370	<50	0.34
RS-08A	0	1	<50	1.4	<50	0.81	690	<10	2.34	50	550	<20	<0.1	<50	<10	410	<50	0.34
RS-08A	1	5																
RS-08A	5	11	<50	1.7	<50	0.94	930	<10	2.59	80	530	<20	<0.1	<50	10	440	<50	0.28
RS-09	7	8	<50	1.2	<50	1.93	760	<10	2.26	160	410	<20	<0.1	<50	10	350	<50	0.27
RS-10	1	2	<50	1.2	<50	0.77	670	<10	2.38	60	460	20	<0.1	<50	<10	360	<50	0.39
RS-10	2	3																
RS-10	3	5.5																
RS-10	5.5	7.5	<50	1.8	<50	0.88	600	<10	3.15	30	610	20	<0.1	<50	10	660	<50	0.25
RS-10	7.5	10	<50	1.6	<50	1.3	990	<10	2.35	70	600	<20	<0.1	<50	10	370	<50	0.25
RS-10	10	14																
RS-11	0	9.5	<50	1.1	<50	1.17	780	<10	1.89	70	490	<20	0.1	<50	10	290	<50	0.52
RS-11	11.5	17	<50	1.5	<50	1.26	850	<10	2.25	80	570	<20	0.1	<50	10	340	<50	0.53
RS-11	17	25	<50	1.9	<50	1.52	790	<10	2.26	180	520	<20	0.2	<50	10	390	<50	0.33
RS-11	28	31	<50	1.6	<50	0.84	820	<10	2.85	30	520	<20	0.1	<50	10	500	<50	0.22
RS-11	31	33																
RS-12	7	9	<50	1.8	<50	0.58	520	<10	3	20	440	<20	<0.1	<50	<10	530	<50	0.22
RS-12	16	18	<50	1.9	<50	0.58	490	<10	3	20	430	20	<0.1	<50	<10	520	<50	0.23
RS-12	20	22	<50	1.6	<50	1.22	650	<10	2.83	180	470	<20	0.1	<50	10	490	<50	0.3
RS-13	0	1.5	<50	0.9	<50	1.12	1050	<10	2.06	100	580	<20	<0.1	<50	10	310	<50	0.53
RS-13	1.5	2.5	<50	0.9	<50	1.09	760	<10	2.39	80	520	20	<0.1	<50	10	340	<50	0.47
RS-13	2.5	6																
RS-13	8	10	<50	1.3	<50	1.42	1050	<10	2.39	380	530	<20	0.5	<50	10	400	<50	0.48
RS-14B	0	1.5	<50	0.8	<50	0.9	790	<10	1.64	70	540	20	<0.1	<50	10	260	<50	0.47
RS-14B	1.5	3	<50	0.8	<50	1.04	840	<10	2.09	90	500	<20	<0.1	<50	10	310	<50	0.45
RS-14B	3	5																
RS-16B	0	2	<50	0.8	<50	1.14	720	<10	2.07	70	610	<20	<0.1	<50	10	300	<50	0.69

Fraction Analyzed			-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	-2 mm+74 µm	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole
Parameter	**Depth**	**Depth**	**Tl**	**U**	**V**	**W**	**Zn**	**Initial Sample Wt**	**Volume Influent**	**Volume Effluent**	**Sample Wt After Extraction**	**Moisture Content**	**pH**	**Redox**	**Conductivity**	**Acidity (to pH 4.5)**	**Total Acidity (to pH 8.3)**	**Alkalinity**	**Fluoride**
Units	From (ft)	To (ft)	ppm	ppm	ppm	ppm	ppm	kg	mL	mL	kg	%		mV	uS/cm	mg CaCO3/L	mg CaCO3/L	mg CaCO3/L	mg/L
Detection Limit			50	50	10	50	20												
General Method								MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP
Analytical Method Code			ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a	ME-ICP61a						meter	meter	meter	titration	titration	titration	
RS-01B	0	1	<50	<50	70	<50	90	1	670	569	1.104	12.8	8.07	399.9	330.5	#N/A	2.38	82.69	1.1
RS-01B	1	5	<50	<50	70	<50	30												
RS-01B	14	15	<50	<50	70	<50	40												
RS-01B	18	20	<50	<50	60	<50	40												
RS-01B	20	25																	
RS-03	5	10	<50	<50	40	<50	30	1	1650	544	2.105	62.4	6.83	392.57	249.17	#N/A	6.56	8.54	0.08
RS-03	15	20	<50	<50	60	<50	40												
RS-03	20	22	<50	<50	70	<50	50												
RS-04	1	5	<50	<50	90	<50	60												
RS-04	10	15	<50	<50	50	<50	30	1	655	551	1.104	11.6	7.44	361.32	54.29	#N/A	3.17	14.38	0.38
RS-04	15	20	<50	<50	60	<50	30	1	690	531	1.157	12.4	7.72	380.85	164.77	#N/A	2.85	37.68	0.6
RS-04	20	25	<50	<50	70	<50	40	1	625	523	1.103	7.4	7.69	385.25	253.14	#N/A	2.82	31	0.2
RS-05A	5	10	<50	<50	70	<50	50	1	655	510	1.146	13.8	7.3	379.39	70	#N/A	3.02	7.68	0.15
RS-05A	10	13	<50	<50	60	<50	50												
RS-06A	0.5	2	<50	<50	90	<50	60	1	715	518	1.198	20.8	6.88	390.62	30.48	#N/A	4.6	5.04	< 0.05
RS-06A	2	4																	
RS-06A	5	7.5	<50	<50	80	<50	50												
RS-06A	7.5	10																	
RS-06A	15	19	<50	<50	70	<50	40	1	1115	1030	1.141	11.3	5.69	473.63	45.23	#N/A	6.98	1.56	0.42
RS-06A	19	21																	
RS-07	1	2	<50	<50	90	<50	80												
RS-07	2	3																	
RS-07	3	5																	
RS-07	5	6																	
RS-07	6	10																	
RS-07R	10	12																	
RS-07R	13.5	14.5																	
RS-07/RS-07R	6	14.5	<50	<50	100	<50	100	1	1138	1028	1.155	13.9	3.71	490.23	355.82	#N/A	12.25	#N/A	< 0.05
RS-08A	0	1	<50	<50	90	<50	50												
RS-08A	1	5																	
RS-08A	5	11	<50	<50	70	<50	60	1	1250	1044	1.123	9.06	7.13	341.3	55.25	#N/A	3.98	4.62	0.48
RS-09	7	8	<50	<50	80	<50	70												
RS-10	1	2	<50	<50	100	<50	60												
RS-10	2	3																	
RS-10	3	5.5																	
RS-10	5.5	7.5	<50	<50	70	<50	70												
RS-10	7.5	10	<50	<50	80	<50	70												
RS-10	10	14																	
RS-11	0	9.5	<50	<50	110	<50	100												
RS-11	11.5	17	<50	<50	120	<50	110												
RS-11	17	25	<50	<50	90	<50	110	1	1174	1045	1.136	9.24	3.4	512.2	464.64	23.55	59.67	#N/A	< 0.05
RS-11	28	31	<50	<50	60	<50	40												
RS-11	31	33																	
RS-12	7	9	<50	<50	60	<50	30	1	1144	1013	1.125	8.87	7.05	303.22	50.77	#N/A	3.74	13.06	0.18
RS-12	16	18	<50	<50	60	<50	30												
RS-12	20	22	<50	<50	70	<50	50	1	1132	1027	1.097	7.93	7.07	353.02	350.97	#N/A	4.05	11.21	0.28
RS-13	0	1.5	<50	<50	120	<50	70	1	1525	1050	1.874	30.86	5.66	399.9	161.99	#N/A	19.72	4.94	0.14
RS-13	1.5	2.5	<50	<50	110	<50	70												
RS-13	2.5	6																	
RS-13	8	10	<50	<50	100	<50	70												
RS-14B	0	1.5	<50	<50	100	<50	80												
RS-14B	1.5	3	<50	<50	100	<50	70												
RS-14B	3	5																	
RS-16B	0	2	<50	<50	140	<50	70												

Fraction Analyzed			Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole	Whole
Parameter	**Depth**	**Depth**	**Chloride**	**Sulphate**	**Hardness $CaCO_3$**	**Al**	**Sb**	**As**	**Ba**	**Be**	**Bi**	**B**	**Cd**	**Ca**	**Cr**	**Co**	**Cu**	**Fe**	**Pb**	**Li**	**Mg**	**Mn**	**Hg**
Units	From (ft)	To (ft)	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
Detection Limit																							
General Method			MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP	MWMP
Analytical Method Code				Turbidity		ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS
RS-01B	0	1	9.16	68.3	71.5	0.042	0.0005	0.0029	0.01	< 0.0002	< 0.0002	0.23	< 0.00004	15.8	< 0.0002	< 0.0001	0.003	0.02	< 0.00005	0.0099	7.76	0.0592	0.02
RS-01B	1	5																					
RS-01B	14	15																					
RS-01B	18	20																					
RS-01B	20	25																					
RS-03	5	10	2.73	93.4	102	0.13	0.0007	0.0044	0.035	< 0.0002	< 0.0002	0.18	< 0.00004	22.9	0.001	0.0007	0.011	0.12	0.00023	0.0043	10.9	0.192	< 0.02
RS-03	15	20																					
RS-03	20	22																					
RS-04	1	5																					
RS-04	10	15	1.28	8.53	18.6	0.19	0.0004	0.0014	0.0066	< 0.0002	< 0.0002	0.037	< 0.00004	3.62	0.0013	0.0012	0.028	0.31	< 0.00005	0.0016	2.32	0.236	< 0.02
RS-04	15	20	2.33	39.9	57	0.092	0.0012	0.0022	0.014	< 0.0002	< 0.0002	0.098	< 0.00004	7.97	< 0.0002	0.0006	0.0089	0.07	< 0.00005	0.0058	9.01	0.0379	< 0.02
RS-04	20	25	1.37	97.4	120	0.024	0.0011	0.0023	0.018	< 0.0002	< 0.0002	0.055	< 0.00004	17.7	< 0.0002	0.0017	0.0043	< 0.01	< 0.00005	0.0082	18.3	0.0596	< 0.02
RS-05A	5	10	1.91	21.2	23	0.067	0.0003	0.0004	0.0051	< 0.0002	< 0.0002	0.036	< 0.00004	5.86	< 0.0002	0.0005	0.0072	0.04	< 0.00005	0.0017	2.04	0.0453	0.02
RS-05A	10	13																					
RS-06A	0.5	2	1.9	3.41	7.2	0.32	< 0.0001	0.0004	0.014	< 0.0002	< 0.0002	0.03	0.00005	1.77	0.0011	0.0006	0.0054	0.06	< 0.00005	0.0006	0.66	0.0511	0.02
RS-06A	2	4																					
RS-06A	5	7.5																					
RS-06A	7.5	10																					
RS-06A	15	19	4.01	4.77	10.8	0.29	0.0003	0.0027	0.0083	< 0.0002	< 0.0002	0.012	0.00011	2.79	0.0003	0.0013	0.017	0.15	0.00028	0.0008	0.92	0.116	< 0.02
RS-06A	19	21																					
RS-07	1	2																					
RS-07	2	3																					
RS-07	3	5																					
RS-07	5	6																					
RS-07	6	10																					
RS-07R	10	12																					
RS-07R	13.5	14.5																					
RS-07/RS-07R	6	14.5	4.03	188	108	0.14	< 0.0001	0.0033	0.046	< 0.0002	< 0.0002	0.022	0.0025	17.8	0.0006	0.124	0.113	0.95	0.00058	0.02	15.3	1.09	< 0.02
RS-08A	0	1																					
RS-08A	1	5																					
RS-08A	5	11	0.74	16.5	17.7	0.091	0.0011	0.0032	0.003	< 0.0002	< 0.0002	0.013	0.00016	3.85	< 0.0002	0.0016	0.005	0.05	< 0.00005	0.0014	1.95	0.105	< 0.02
RS-09	7	8																					
RS-10	1	2																					
RS-10	2	3																					
RS-10	3	5.5																					
RS-10	5.5	7.5																					
RS-10	7.5	10																					
RS-10	10	14																					
RS-11	0	9.5																					
RS-11	11.5	17																					
RS-11	17	25	1.71	230	105	0.74	< 0.0001	0.0028	0.028	0.0008	< 0.0002	0.016	0.0066	21	0.0008	0.305	0.579	7.25	0.0014	0.028	12.9	1.31	< 0.02
RS-11	28	31																					
RS-11	31	33																					
RS-12	7	9	3.62	1.74	23.3	0.068	< 0.0001	0.0005	0.0035	< 0.0002	< 0.0002	< 0.001	0.00005	5.92	< 0.0002	< 0.0001	0.0083	0.02	< 0.00005	< 0.0002	2.07	0.0075	< 0.02
RS-12	16	18																					
RS-12	20	22	3.33	166	141	0.01	0.0004	0.0011	0.0084	< 0.0002	< 0.0002	< 0.001	0.00007	27	< 0.0002	0.0051	0.008	< 0.01	< 0.00005	0.0003	17.8	0.305	< 0.02
RS-13	0	1.5	16.6	23.6	49.6	1.49	< 0.0001	0.0016	0.028	< 0.0002	< 0.0002	0.025	0.00019	9.64	0.003	0.0028	0.072	0.77	0.0028	0.0004	6.18	0.402	< 0.02
RS-13	1.5	2.5																					
RS-13	2.5	6																					
RS-13	8	10																					
RS-14B	0	1.5																					
RS-14B	1.5	3																					
RS-14B	3	5																					
RS-16B	0	2																					

Appendix B
Initial Summary of MWMP Data Provided to Barr Engineering

	Solids	Leaches	Stat	Whole MWMP pH	Whole MWMP Redox	Whole MWMP Conductivity	Whole MWMP Total Acidity (to pH 8.3)	Whole MWMP Alkalinity	Whole MWMP Fluoride	Whole MWMP Chloride	Whole MWMP Sulphate	Whole MWMP Hardness CaCO3	Whole MWMP Al	Whole MWMP Sb	Whole MWMP As	Whole MWMP Ba	Whole MWMP Be	Whole MWMP B	Whole MWMP Cd	Whole MWMP Ca	Whole MWMP Cr	Whole MWMP Co	Whole MWMP Cu	Whole MWMP Fe	Whole MWMP Pb	Whole MWMP Mg	Whole MWMP Mn	Whole MWMP Mo	Whole MWMP Ni	Whole MWMP P	Whole MWMP K	Whole MWMP Se	Whole MWMP Ag	Whole MWMP Na	Whole MWMP Tl	Whole MWMP Ti	Whole MWMP V	Whole MWMP Zn
					mV	uS/cm	mg CaCO3/L	mg CaCO3/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
Peat	6	2	P5	6.7715	392.9365	166.349	7.218	5.12	0.083	3.4235	27.09	52.22	0.198	-0.00006	0.00174	0.02835	-0.0002	0.03275	-0.0000285	10.303	0.0011	0.000805	0.01405	0.1525	0.0003585	6.416	0.2025	0.00152	0.00737	0.052	1.734	0.00064	-0.00005	4.3095	0.000072	0.001875	0.002115	0.00425
			Median	6.245	396.235	205.58	13.14	6.74	0.11	9.665	58.5	75.8	0.81	0.0003	0.003	0.0315	-0.0002	0.1025	0.000075	16.27	0.002	0.00175	0.0415	0.445	0.001515	8.54	0.297	0.0053	0.0143	0.07	4.11	0.001	-0.00005	5.025	0.00009	0.00255	0.00315	0.0065
			P95	5.7185	399.5335	244.811	19.062	8.36	0.137	15.9065	89.91	99.38	1.422	0.00066	0.00426	0.03465	-0.0002	0.17225	0.0001785	22.237	0.0029	0.002695	0.06895	0.7375	0.0026715	10.664	0.3915	0.00908	0.02123	0.088	6.486	0.00136	-0.00005	5.7405	0.000108	0.003225	0.004185	0.00875
Unsaturated Mineral	13	4	P5	7.283	345.109	32.957	3.116	4.662	0.1665	0.856	4.719	8.25	0.0694	-0.00006	0.0004	0.00321	-0.0002	0.0147	-0.000031	1.978	-0.0002	0.00051	0.00504	0.041	-0.00005	0.789	0.04588	0.000195	0.00155	-0.03	0.744	-0.0002	-0.00005	3.747	-0.00002	0.00113	0.00042	0.0021
			Median	7.13	379.39	55.25	3.98	5.04	0.315	1.9	16.5	17.7	0.091	0.0003	0.0004	0.0051	-0.0002	0.03	0.00005	3.85	-0.0002	0.0006	0.0054	0.05	-0.00005	1.95	0.0511	0.0024	0.0029	-0.03	0.78	-0.0002	-0.00005	4.26	-0.00002	0.0023	0.0006	0.003
			P95	6.905	389.497	68.525	4.538	7.416	0.4635	1.909	20.73	22.47	0.2971	0.00102	0.00292	0.01311	-0.0002	0.0354	0.000149	5.659	0.00097	0.0015	0.00702	0.059	-0.00005	2.031	0.09961	0.01257	0.00326	-0.03	1.23	0.00052	-0.00005	5.331	0.000025	0.00257	0.0006	0.0057
Saturated Mineral	19	6	P5	7.965	367.179	47.948	2.512	#N/A	0.236	1.307	5.898	13.14	0.0294	-0.0001	0.00164	0.00711	-0.0002	0.0132	-0.00004	3.039	-0.0002	0.00011	0.00339	-0.001	-0.00005	1.34	0.04429	0.000332	0.00201	-0.03	1.257	0.00055	-0.00005	3.691	-0.00002	0.00075	-0.0002	-0.0004
			Median	7.44	399.9	253.14	3.17	#N/A	0.42	2.33	68.3	71.5	0.14	0.0004	0.0027	0.014	-0.0002	0.037	-0.00004	15.8	0.0003	0.0013	0.017	0.15	-0.00005	9.01	0.116	0.0274	0.019	-0.03	4.45	0.0019	-0.00005	7.19	-0.00002	0.0016	0.0014	0.003
			P95	3.493	505.609	431.994	45.444	#N/A	1	7.621	217.4	116.4	0.605	0.00117	0.00318	0.0406	0.0005	0.1904	0.00537	20.04	0.00115	0.2507	0.4392	5.36	0.001154	17.4	1.244	0.03191	2.1812	0.04	9.827	0.00377	0.000965	37.07	0.000082	0.00614	0.00247	0.862
OB with Mineralized Rock	3	2	P5	7.069	305.71	65.78	3.7555	11.3025	0.185	3.3445	9.953	29.185	0.0129	-0.000075	0.00053	0.003745	-0.0002	-0.001	0.000051	6.974	-0.0002	0.00016	0.008015	-0.0085	-0.00005	2.8565	0.022375	0.00539	0.00456	-0.03	0.7285	-9E-05	-0.00005	2.334	-0.00002	-9.5E-05	-0.000165	0.00205
			Median	7.06	328.12	200.87	3.895	12.135	0.23	3.475	83.87	82.15	0.039	0.00015	0.0008	0.00595	-0.0002	-0.001	0.00006	16.46	-0.0002	0.0025	0.00815	0.005	-0.00005	9.935	0.15625	0.0188	0.0384	-0.03	1.345	0.0009	-0.00005	6.96	-0.00002	0.00085	0.00015	0.0025
			P95	7.051	350.53	335.96	4.0345	12.9675	0.275	3.6055	157.787	135.115	0.0651	0.000375	0.00107	0.008155	-0.0002	-0.001	0.000069	25.946	-0.0002	0.00484	0.008285	0.0185	-0.00005	17.0135	0.290125	0.03221	0.07224	-0.03	1.9615	0.00189	-0.00005	11.586	-0.00002	0.001795	0.000465	0.00295
All	41	14	P5	7.8425	327.972	40.0675	2.666	#N/A	0.11	1.091	2.8255	9.54	0.0191	-0.0001	0.0004	0.003325	-0.0002	-0.001	-0.00004	2.433	-0.0002	-0.0001	0.003845	-0.01	-0.00005	0.829	0.02726	0.000171	0.00119	-0.03	0.712	-0.0002	-0.00005	2.925	-0.00002	0.00032	-0.0002	0.0003
			Median	7.06	387.935	163.38	4.015	#N/A	0.28	2.53	31.75	53.3	0.111	0.00035	0.00225	0.012	-0.0002	0.0275	0.00005	8.805	0.00005	0.00125	0.0086	0.065	-0.00005	6.97	0.1105	0.008	0.0128	-0.03	1.85	0.001	-0.00005	5.62	-0.00002	0.00185	0.00065	0.003
			P95	3.6015	497.9195	393.907	33.7025	#N/A	0.85	11.764	202.7	127.35	1.0025	0.001135	0.003685	0.03885	0.00015	0.1975	0.003935	24.335	0.001895	0.18735	0.2761	3.155	0.00189	17.975	1.167	0.03318	1.2726	0.077	8.4475	0.003735	0.0004575	25.135	0.0001035	0.00502	0.00313	0.526
			Max	3.4	512.2	464.64	59.67	#N/A	1.1	16.6	230	141	1.49	0.0012	0.0044	0.046	0.0008	0.23	0.0066	27	0.003	0.305	0.579	7.25	0.0028	18.3	1.31	0.0337	2.96	0.09	11.6	0.0038	0.0014	47.3	0.00011	0.0071	0.0043	1.15

Appendix C
Sump Spoil Geochemical Data

CLIENT : SRK Consulting
PROJECT : Polymet Soil Samples
PROJECT # : 0518
TEST : Moisture Content
Date : April 13, 2010

Pre-MWMP Leach Test

Sample ID	Wet Weight (kg)	Dry Weight (kg)	Moisture %
J007 (0-3)	11.45	10.40	9.17
J007 (3-5)	11.45	10.70	6.55
J008 (0-3.5)	11.95	10.95	8.37
J008 (3.5-4.6)	11.60	10.75	7.33
J012 (0-3)	11.50	10.65	7.39
J012 (3-6)	11.50	10.55	8.26
J013	10.40	9.70	6.73
J019 (0-2)	12.95	10.85	16.22
J024 (0-2)	11.75	10.85	7.66
J029 (0-5)	11.55	10.50	9.09
J037 (0-3)	11.50	10.50	8.70
J037 (3-6)	11.30	10.35	8.41
J107	11.35	10.15	10.57

Post-MWMP Leach Test

Sample ID	Wet Weight (kg)	Dry Weight (kg)	Moisture %
J007 (0-3)	6.20	4.95	20.16
J007 (3-5)	5.70	4.95	13.16
J008 (0-3.5)	5.95	4.90	17.65
J008 (3.5-4.6)	5.85	4.95	15.38
J012 (0-3)	5.80	4.95	14.66
J012 (3-6)	5.90	5.05	14.41
J013	5.85	4.95	15.38
J019 (0-2)	6.25	4.95	20.80
J024 (0-2)	5.60	4.95	11.61
J029 (0-5)	5.90	4.95	16.10
J037 (0-3)	5.80	4.95	14.66
J037 (3-6)	5.65	4.95	12.39
J107	6.20	4.95	20.16

CLIENT : **SRK Consulting**
PROJECT : **Polymet Soil Samples**
PROJECT # : **0518**
Test : **Meteoric Water Mobility Procedure**
Date : **April 26 - May 4, 2010**

Leachate Analysis

Sample ID			J007 (0-3)	J007 (3-5)	J008 (0-3.5)	J008 (3.5-4.6)	J012 (0-3)	J012 (3-6)	J013	J019 (0-2)	J024 (0-2)	J029 (0-5)	J037 (0-3)	J037 (3-6)	J107	Blank
Parameter	**Method**	**Units**														
Initial Dry Sample Wt		kg	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	#N/A
Volume Influent		mL	6.25	5.80	6.04	5.89	5.84	5.79	5.90	6.30	5.75	6.00	5.85	5.75	6.30	5.10
Volume Effluent		mL	5.00	5.00	5.00	5.01	5.01	5.00	5.00	5.00	5.05	5.00	5.00	5.00	5.00	5.10
Sample Wet Wt After Extraction		kg	6.20	5.70	5.95	5.85	5.80	5.90	5.85	6.25	5.60	5.90	5.80	5.65	6.20	#N/A
Sample Dry Wt After Extraction		kg	4.95	4.95	4.90	4.95	4.95	5.05	4.95	4.95	4.95	4.95	4.95	4.95	4.95	#N/A
pH	meter		6.52	6.75	6.58	6.62	6.74	6.9	6.53	6.49	6.49	6.62	6.25	6.42	5.85	6.65
Redox	meter	mV	340	338	352	354	346	346	361	264	262	283	297	293	318	223
Conductivity	meter	uS/cm	23	16	22	22	26	26	25	39	24	45	22	20	38	5
Acidity (to pH 4.5)	titration	mg CaCO3/L	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
Total Acidity (to pH 8.3)	titration	mg CaCO3/L	3.1	2.9	3.3	2.9	4.0	3.1	3.9	5.7	4.5	4.7	5.2	5.2	4.6	3.6
Alkalinity	titration	mg CaCO3/L	3.5	5.3	5.2	5.7	5.8	7.2	6.8	5.5	3.3	5.6	2.7	3.8	1.8	2.9
Inorganic Carbon		mg/L	1	1	1	1	1	2	1	1	-1	1	-1	-1	-1	-1
Fluoride		mg/L	0.04	0.07	0.03	0.03	0.05	0.07	0.02	0.03	0.06	0.02	0.05	0.13	0.01	-0.01
Chloride		mg/L	-0.5	-0.5	-0.5	0.8	-0.5	0.6	0.8	-0.5	0.8	5.8	-0.5	-0.5	1.3	-0.5
Sulphate	Turbidity	mg/L	5	1.6	1.1	0.8	4.8	5.3	-0.5	7.4	4.3	2.8	4.4	2.1	10	-0.5
Ion Balance																
Major Anions	calc.	meq/L	0.18	0.14	0.13	0.15	0.22	0.27	0.16	0.26	0.18	0.33	0.15	0.13	0.28	#N/A
Major Cations	calc.	meq/L	0.21	0.21	0.28	0.27	0.24	0.29	0.36	0.44	0.22	0.40	0.19	0.21	0.30	#N/A
Difference	calc.	meq/L	-0.04	-0.06	-0.15	-0.12	-0.02	-0.01	-0.20	-0.17	-0.03	-0.06	-0.05	-0.09	-0.02	#N/A
Diff. (%)	calc.	meq/L	-9.4%	-18.3%	-37.2%	-27.5%	-4.8%	-2.2%	-37.8%	-24.4%	-8.4%	-8.8%	-13.6%	-25.6%	-2.8%	#N/A
Dissolved Metals																
Hardness CaCO3		mg/L	6.2	5.4	8.8	8.2	7	7.9	10.3	15.9	7.3	15.2	6	5.6	9.2	-0.5
Aluminum Al	ICP-MS	mg/L	0.0815	0.141	0.257	0.201	0.049	0.205	0.559	0.215	0.0297	0.0778	0.0973	0.139	0.0284	0.006
Antimony Sb	ICP-MS	mg/L	0.00005	0.00005	0.00005	0.00006	0.00004	0.00004	0.00006	0.00006	0.00002	0.00004	0.00004	0.00007	0.00002	-0.0004
Arsenic As	ICP-MS	mg/L	0.00036	0.00037	0.00052	0.00057	0.00026	0.00039	0.00053	0.00039	0.00022	0.00026	0.00024	0.00043	0.0002	-0.0004
Barium Ba	ICP-MS	mg/L	0.00129	0.00212	0.0024	0.00184	0.00104	0.00334	0.00147	0.00061	0.00079	0.00117	0.00142	0.00212	0.00402	0.0009
Beryllium Be	ICP-MS	mg/L	0.00002	0.00002	0.00004	0.00005	0.00001	0.00002	0.00007	0.00003	-0.00001	0.00001	0.00002	0.00002	-0.00001	-0.0002
Bismuth Bi	ICP-MS	mg/L	-0.000005	-0.000005	-0.000005	-0.000005	-0.000005	-0.000005	-0.000005	-0.000005	0.000019	-0.000005	-0.000005	-0.000005	-0.000005	-0.0001
Boron B	ICP-MS	mg/L	-0.05	-0.05	-0.05	-0.05	-0.05	-0.05	-0.05	-0.05	-0.05	-0.05	-0.05	-0.05	-0.05	-1
Cadmium Cd	ICP-MS	mg/L	0.000007	-0.000005	0.000005	-0.000005	-0.000005	-0.000005	0.000013	-0.000005	-0.000005	0.000012	-0.000005	-0.000005	0.000026	-0.0001
Calcium Ca	ICP-MS	mg/L	1.57	1.35	2.58	2.35	1.64	1.82	3.01	4.07	1.82	4.46	1.34	1.15	2.13	-1
Chromium Cr	ICP-MS	mg/L	0.0003	0.0004	0.0008	0.0008	0.0004	0.0006	0.0015	0.0011	-0.0001	0.0002	0.0014	0.0014	-0.0001	-0.002
Cobalt Co	ICP-MS	mg/L	0.000295	0.000191	0.000242	0.000212	0.000128	0.00017	0.000293	0.000296	0.000123	0.00019	0.000129	0.000127	0.00036	-0.0001
Copper Cu	ICP-MS	mg/L	0.00513	0.00468	0.0069	0.00759	0.00437	0.00367	0.00622	0.00683	0.00398	0.00643	0.0036	0.00519	0.00221	-0.001
Iron Fe	ICP-MS	mg/L	0.015	0.133	0.143	0.127	0.035	0.239	0.179	0.177	0.033	0.027	0.081	0.152	0.004	-0.02
Lead Pb	ICP-MS	mg/L	0.000211	0.000125	0.000051	0.000044	0.000036	0.000114	0.000043	0.000034	0.000074	0.000006	0.000026	0.000152	0.000038	0.0012
Lithium Li	ICP-MS	mg/L	-0.0005	-0.0005	-0.0005	-0.0005	-0.0005	0.0006	-0.0005	-0.0005	-0.0005	-0.0005	-0.0005	0.0006	-0.0005	-0.01
Magnesium Mg	ICP-MS	mg/L	0.54	0.49	0.57	0.56	0.71	0.82	0.68	1.4	0.68	0.99	0.65	0.67	0.93	-1
Manganese Mn	ICP-MS	mg/L	0.0133	0.00452	0.00656	0.00597	0.00344	0.00722	0.014	0.0033	0.0051	0.00475	0.0155	0.00687	0.0601	-0.001
Mercury Hg	ICP-MS	ug/L	0.02	0.02	0.02	0.02	0.01	0.01	0.02	0.02	-0.01	0.01	0.01	0.01	-0.01	-0.2
Molybdenum Mo	ICP-MS	mg/L	-0.00005	0.00007	0.00012	0.00009	-0.00005	0.00006	0.00006	-0.00005	-0.00005	0.00012	-0.00005	0.00011	-0.00005	-0.001
Nickel Ni	ICP-MS	mg/L	0.00041	0.00067	0.00093	0.00113	0.00064	0.00073	0.00113	0.00035	0.00042	0.00037	0.00111	0.00116	0.00308	-0.0004
Potassium K	ICP-MS	mg/L	0.14	0.12	0.53	0.26	0.11	0.22	0.27	-0.05	0.09	0.37	0.09	0.15	0.84	-1
Selenium Se	ICP-MS	mg/L	0.00019	0.00013	0.0003	0.00024	0.00015	0.00008	0.0004	0.00031	0.00014	0.00019	0.00026	0.00019	0.00046	-0.0008
Silicon Si	ICP-MS	mg/L	13.5	11.5	8.26	8.65	11.3	11.4	7.45	12.5	11.2	7.59	12.1	14.5	8.22	-2
Silver Ag	ICP-MS	mg/L	-0.000005	-0.000005	0.000008	0.000008	-0.000005	-0.000005	0.000014	0.000007	-0.000005	-0.000005	-0.000005	-0.000005	-0.000005	-0.0001
Sodium Na	ICP-MS	mg/L	1.98	1.66	1.25	1.65	2.23	2.01	1.62	1.94	1.52	1.95	1.65	1.67	2.14	-1
Strontium Sr	ICP-MS	mg/L	0.0109	0.008	0.0161	0.014	0.0109	0.0116	0.02	0.014	0.0123	0.0237	0.00786	0.00538	0.0186	-0.001
Tellurium Te	ICP-MS	mg/L	-0.00002	-0.00002	-0.00002	-0.00002	-0.00002	-0.00002	-0.00002	-0.00002	-0.00002	-0.00002	-0.00002	-0.00002	-0.00002	-0.0004
Thallium Tl	ICP-MS	mg/L	0.000011	0.000003	0.000005	0.000003	-0.000002	-0.000002	0.000004	0.000004	-0.000002	0.000002	0.000003	0.000002	0.000008	-0.00004
Thorium Th	ICP-MS	mg/L	0.000022	0.000045	0.000199	0.000231	0.000028	0.000045	0.000626	0.000226	0.000013	0.000053	0.000036	0.000076	0.000006	-0.0001
Tin Sn	ICP-MS	mg/L	0.00004	0.0002	0.00009	0.00019	0.00025	0.00035	0.00045	0.00281	0.00272	0.00323	0.00282	0.0032	0.0036	0.0812
Titanium Ti	ICP-MS	mg/L	-0.0005	0.0042	0.0046	0.004	0.001	0.0092	0.0051	0.0026	0.0006	0.0007	0.0018	0.0045	-0.0005	-0.01
Uranium U	ICP-MS	mg/L	0.000054	0.000089	0.000313	0.000339	0.000085	0.000057	0.000348	0.000333	0.000027	0.000094	0.000088	0.000122	0.000012	-0.00004
Vanadium V	ICP-MS	mg/L	0.0006	0.0009	0.0008	0.0009	0.0004	0.001	0.0006	0.001	0.0003	0.0004	0.0003	0.0012	0.0002	-0.004
Zinc Zn	ICP-MS	mg/L	0.0016	0.0009	0.0007	0.0006	0.0008	0.0011	0.0019	0.0011	0.0012	0.0015	0.001	0.001	0.0018	0.006
Zirconium Zr	ICP-MS	mg/L	0.0004	0.0004	0.0026	0.0028	0.0003	0.0003	0.0045	0.0026	-0.0001	0.0006	0.0004	0.0004	-0.0001	-0.002

CLIENT : SRK Consulting
PROJECT : Polymet Soil Samples
PROJECT # : 0518
TEST : Screen Assay & Moisture Content
Date : April 13, 2010

Sample ID	Weight	+2mm		-2mm	
	(g)	(g)	(%)	(g)	(%)
J007 (0-3)	300.0	77.9	26.0%	221.9	74.0%
J007 (3-5)	300.0	106.5	35.5%	193.4	64.5%
J008 (0-3.5)	300.0	98.8	33.0%	201.0	67.0%
J008 (3.5-4.6)	300.0	126.1	42.1%	173.6	57.9%
J012 (0-3)	300.0	109.2	36.4%	190.6	63.6%
J012 (3-6)	300.0	128.5	42.9%	171.3	57.1%
J013	300.0	107.9	36.0%	192	64.0%
J019 (0-2)	300.0	49.9	16.6%	250	83.4%
J024 (0-2)	300.0	160.1	53.4%	139.7	46.6%
J029 (0-5)	300.0	67.4	22.5%	232.4	77.5%
J037 (0-3)	300.0	117.8	39.3%	181.7	60.7%
J037 (3-6)	300.0	147.1	49.0%	152.8	51.0%
J107	300.0	80.3	26.8%	219.4	73.2%

Sample ID	Wet Weight	Dry Weight	Moisture
	(g)	(g)	%
J007 (0-3)	25.0	24.48	2.08
J007 (3-5)	25.0	24.67	1.32
J008 (0-3.5)	25.0	24.51	1.96
J008 (3.5-4.6)	25.0	24.64	1.44
J012 (0-3)	25.0	24.72	1.12
J012 (3-6)	25.0	24.70	1.20
J013	25.0	24.69	1.24
J019 (0-2)	25.0	24.47	2.12
J024 (0-2)	25.0	24.72	1.12
J029 (0-5)	25.0	24.65	1.40
J037 (0-3)	25.0	24.49	2.04
J037 (3-6)	25.0	24.71	1.16
J107	25.0	24.23	3.08

Moisture Content on the -2mm Fraction

CLIENT : SRK Consulting
PROJECT : Polymet Soil Samples
PROJECT # : 0518
TEST : Sobek Acid-Base Accounting
Date : April 22, 2010

Sample ID	Paste pH Std. Units	CO2 % CO2	Equiv. CaCO3 kg CaCO3/t	Total S % S	Sulphate % S	Sulphur Diff. % S	AP kg CaCO3/t	Sobek NP kg CaCO3/t	Net NP kg CaCO3/t	NP/AP Ratio	Fizz Test Visual
LOD	0.01	0.2	#N/A	0.01	0.01	#N/A	#N/A	0.1	#N/A	#N/A	#N/A
Method Code	Sobek	C-GAS05	Calc.	S-IR08	S-GRA06a	Calc.	Calc.	Sobek NP	Calc.	Calc.	Sobek
J007 (0-3)	5.44	<0.2	<4.5	0.02	<0.01	0.02	0.6	5.1	4.5	8.2	None
J007 (3-5)	6.83	<0.2	<4.5	0.01	<0.01	0.01	0.3	8.4	8.1	26.9	None
J008 (0-3.5)	6.23	<0.2	<4.5	<0.01	<0.01	<0.01	<0.3	8.9	8.9	29.7	None
J008 (3.5-4.6)	6.65	<0.2	<4.5	0.02	0.01	0.01	0.3	8.8	8.5	28.2	None
J012 (0-3)	6.57	<0.2	<4.5	0.01	0.01	<0.01	<0.3	7.2	7.2	24.0	None
J012 (3-6)	6.89	<0.2	<4.5	0.03	<0.01	0.03	0.9	7.3	6.4	7.8	None
J013	6.28	0.2	4.5	0.02	<0.01	0.02	0.6	7.8	7.2	12.5	None
J019 (0-2)	6.02	0.2	4.5	0.02	<0.01	0.02	0.6	5.2	4.6	8.3	None
J024 (0-2)	6.69	0.2	4.5	0.01	0.01	<0.01	<0.3	7.3	7.3	24.3	None
J029 (0-5)	6.55	1.2	27.3	0.02	0.01	0.01	0.3	9.4	9.1	30.1	None
J037 (0-3)	6.05	0.2	4.5	0.01	0.01	<0.01	<0.3	5.7	5.7	19.0	None
J037 (3-6)	7.10	0.2	4.5	0.01	0.01	<0.01	<0.3	7.0	7.0	23.3	None
J107	5.45	<0.2	<4.5	0.02	0.01	0.01	0.3	4.3	4.0	13.8	None
Duplicates											
J007 (0-3)	5.50							5.7			None

Note:

Equivalent CaCO3 is calculated from the CO2 originating from carbonate minerals.

Sulphur Difference = Total S - Sulphate S

AP = Acid Potential in tonnes CaCO3 equivalent per 1000 tonnes of material. AP is calculated from the sulphur difference.

Sobek NP = Neutralization Potential in tonnes CaCO3 equivalent per 1000 tonnes of material.

NET NP = Sobek NP - AP

CLIENT : **SRK Consulting**
PROJECT : **Polymet Soil Samples**
PROJECT # : **0518**
TEST : **Metals by Aqua Regia Digestion with ICP-MS Finish**
Date : **April 28, 2010**

Sample ID	Ag	Al	As	Au	B	Ba	Be	Bi	Ca	Cd	Ce	Co	Cr	Cs	Cu	Fe	Ga	Ge	Hf	Hg	In	K	La	Li	Mg	Mn	Mo	Na	Nb	Ni	P	Pb	Rb	Re	S	Sb	Sc	Se	Sn	Sr	Ta	Te	Th	Ti	Tl	U	V	W	Y	Zn	Zr
	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	%	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm
LOD	0.01	0.01	0.1	0.2	10	10	0.05	0.01	0.01	0.01	0.02	0.1	1	0.05	0.2	0.01	0.05	0.05	0.02	0.01	0.005	0.01	0.2	0.1	0.01	5	0.05	0.01	0.05	0.2	10	0.2	0.1	0.001	0.01	0.05	0.1	0.2	0.2	0.2	0.01	0.01	0.2	0.005	0.02	0.05	1	0.05	0.05	2	0.5
Method Code	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41	ME-MS41
J007 (0-3)	0.04	2.01	2.2	<0.2	<10	50	0.4	0.06	0.44	0.05	35	12	64	0.7	24.5	3.25	5.82	0.09	0.17	0.01	0.021	0.06	13.8	8.8	0.61	388	0.53	0.09	1.26	42.9	440	3.9	7.1	<0.001	0.01	0.09	3.1	0.4	0.7	30.4	0.01	0.01	2.5	0.133	0.06	0.49	76	0.11	6.8	38	8
J007 (3-5)	0.03	1.1	2.4	<0.2	<10	40	0.28	0.05	0.45	0.05	34.5	7.8	65	0.71	19.4	2.63	3.8	0.12	0.28	<0.01	0.012	0.08	16.4	7.2	0.38	529	0.35	0.07	0.42	22.8	510	3.2	7.4	<0.001	0.01	0.07	3.1	0.3	0.5	34.5	<0.01	0.01	3	0.106	0.06	0.47	46	0.13	7.01	23	10.6
J008 (0-3.5)	0.02	1.61	2.3	<0.2	<10	50	0.33	0.05	0.46	0.05	35.3	11.2	69	0.57	25.4	2.98	4.92	0.1	0.22	0.01	0.017	0.08	18.3	8.2	0.57	395	0.44	0.08	0.62	36.6	480	3.2	7.8	<0.001	0.01	0.08	3.6	0.4	0.5	36.2	<0.01	<0.01	2.9	0.127	0.06	0.56	73	0.1	7.15	30	9.7
J008 (3.5-4.6)	0.02	1.57	2.6	<0.2	<10	50	0.34	0.06	0.55	0.06	37.8	11.2	76	0.64	28	3.14	5.04	0.13	0.29	<0.01	0.018	0.09	20.2	8.2	0.56	464	0.45	0.1	0.51	36.5	540	3.6	8.1	<0.001	0.01	0.09	4	0.4	0.6	39.7	<0.01	0.01	3.3	0.139	0.06	0.58	74	0.1	8.98	32	11.7
J012 (0-3)	0.02	1.16	2.4	<0.2	<10	40	0.3	0.06	0.38	0.04	36.9	8	73	0.75	28.2	2.68	3.99	0.12	0.22	0.01	0.013	0.08	18.4	7.6	0.39	488	0.38	0.06	0.49	23	500	3.5	7.5	<0.001	0.01	0.07	3.6	0.3	0.4	32	<0.01	0.01	3.4	0.111	0.06	0.55	49	0.12	7.8	24	9.1
J012 (3-6)	0.03	1.06	3.1	<0.2	<10	40	0.29	0.06	0.41	0.06	36.1	7.6	77	0.78	28.3	2.66	3.73	0.13	0.26	<0.01	0.012	0.09	17.1	7.4	0.35	516	0.39	0.07	0.38	19.8	520	3.5	8.2	<0.001	0.01	0.07	3.4	0.3	0.4	32.4	<0.01	0.01	3.2	0.109	0.06	0.52	45	0.12	6.87	24	10
J013	0.02	1.38	2	<0.2	<10	40	0.35	0.05	0.43	0.05	35.9	8.5	72	0.63	20.1	2.48	4.36	0.1	0.16	<0.01	0.015	0.09	15.2	8.6	0.44	390	0.4	0.07	1.05	25.8	470	3.3	8.3	<0.001	0.01	0.07	2.8	0.3	0.5	33.1	<0.01	<0.01	3.1	0.116	0.06	0.47	50	0.11	5.83	26	7.3
J019 (0-2)	0.05	2.23	1.4	<0.2	<10	40	0.44	0.05	0.67	0.05	33.3	9.2	47	0.63	18.5	2.56	6.09	0.1	0.12	0.01	0.021	0.07	21.3	9	0.43	251	0.48	0.15	1.42	30.1	300	3.3	8.1	<0.001	0.01	0.08	3.8	0.4	0.7	42.5	0.01	0.01	2.1	0.114	0.05	0.81	55	0.11	9.82	40	5.4
J024 (0-2)	0.03	1.11	3.2	<0.2	<10	40	0.33	0.06	0.39	0.05	30.6	8.6	82	0.83	20	3.01	4.21	0.12	0.22	0.01	0.015	0.08	14.6	8.6	0.39	499	1.85	0.07	0.45	25.1	440	3.6	8.1	0.001	<0.01	0.09	3.5	0.2	0.5	29.8	<0.01	0.01	3.9	0.106	0.07	0.64	51	0.12	6.98	24	9.1
J029 (0-5)	0.02	1.81	1.9	<0.2	<10	40	0.36	0.05	0.66	0.07	31.6	12.9	59	0.61	28.7	3	5.73	0.1	0.23	0.01	0.02	0.07	14.8	10	0.69	345	0.88	0.13	0.66	48.7	460	3.3	6.9	0.001	<0.01	0.09	3.6	0.2	0.6	40.7	0.01	0.01	3.2	0.113	0.07	0.62	64	0.1	8	32	10.1
J037 (0-3)	0.03	1.2	2.3	<0.2	<10	40	0.33	0.07	0.39	0.03	31.9	8.5	70	0.69	23.1	2.53	4.26	0.1	0.16	0.01	0.014	0.06	15.2	8.1	0.34	505	0.5	0.06	0.77	25.4	440	4	7.1	<0.001	<0.01	0.09	3.5	0.3	0.5	28.1	<0.01	0.01	3.9	0.097	0.06	0.61	46	0.11	7.1	21	7.4
J037 (3-6)	0.03	0.88	2.6	<0.2	<10	40	0.29	0.08	0.43	0.06	31.2	6.8	72	0.66	17.3	2.44	3.39	0.11	0.24	0.01	0.012	0.09	14.2	7.6	0.32	442	0.49	0.07	0.36	20.6	520	3.8	7.1	<0.001	<0.01	0.08	2.9	0.2	0.4	28.4	<0.01	0.01	3.8	0.087	0.06	0.52	38	0.11	6.13	22	9.2
J107	0.07	2.26	3.4	<0.2	<10	50	0.49	0.09	0.35	0.08	33.1	15.5	69	1.08	86.7	3.87	7.25	0.1	0.14	0.02	0.022	0.07	12	15.7	0.77	371	0.87	0.06	1.62	79.9	560	4.9	13	<0.001	<0.01	0.13	3.6	0.4	0.7	23.2	0.01	0.01	3.7	0.131	0.09	0.79	91	0.14	5.19	44	6.3

CLIENT : **SRK Consulting**
PROJECT : **Polymet Soil Samples**
PROJECT # : **0518**
TEST : **Metals by 1N Nitric Acid Digestion with ICP-MS Finish on -2mm fraction**
Date : **April 26/27, 2010**

Sample ID	As ug/L	Cd ug/L	Cr ug/L	Cu ug/L	Hg ug/L	Pb ug/L	Se ug/L	Zn ug/L	Ag ug/L	Al ug/L	B ug/L	Be ug/L	Bi ug/L	Ca ug/L	Ce ug/L	Co ug/L	Cs ug/L	Fe ug/L	Ga ug/L	Hf ug/L	In ug/L	K ug/L	La ug/L	Li ug/L	Mg ug/L	Mn ug/L	Mo ug/L	Na ug/L	Nb ug/L	Ni ug/L	P ug/L	Rb ug/L	Sb ug/L	Sn ug/L	Sr ug/L	Ta ug/L	Te ug/L	Th ug/L	Tl ug/L	U ug/L	V ug/L	W ug/L	Y ug/L	Zr ug/L
LOD	10	0.1	1	0.3	0.2	0.2	10	2	0.06	50	10	0.3	0.1	200	0.02	0.1	0.01	20	0.05	0.05	0.04	60	0.02	10	4	0.1	1	300	0.02	0.5	100	0.1	0.1	0.3	0.1	0.01	10	0.01	0.04	0.01	1	0.2	0.03	0.05
Method Code	ME-MS14a	ME-MS14a	ME-MS14a	ME-MS14a	ME-MS14a	ME-MS14a	ME-MS14a	ME-MS14a	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b	ME-MS14b
J007 (0-3)	20	2.8	549	603	<0.2	292	<10	419	0.27	>10000	40	38.2	12.1	>10000	>1000	566	7.74	>10000	231	10.95	1.3	7880	>1000	140	>10000	>1000	5	8900	16.45	849	>10000	144.5	2.2	18	>1000	1.72	<10	89.2	1.03	43.7	>1000	25.3	692	336
J007 (0-3) DUP	20	2.4	540	451	<0.2	277	<10	389	0.8	>10000	40	38	5	>10000	>1000	561	7.32	>10000	218	11.05	0.8	7630	>1000	150	>10000	>1000	4	>10000	15.6	914	>10000	135.5	2	28.2	>1000	1.62	<10	89.2	0.97	39.5	>1000	3.6	657	333
J007 (3-5)	30	2.9	462	646	<0.2	232	<10	683	0.71	>10000	10	19	4.6	>10000	>1000	483	8.18	>10000	152.5	2.86	1.19	>10000	>1000	300	>10000	>1000	2	6600	2.33	864	>10000	119.5	0.6	2.9	>1000	0.7	<10	153.5	0.81	30.5	880	3.6	618	108.5
J007 (3-5) DUP	30	2.8	465	590	<0.2	224	<10	632	0.35	>10000	10	20.1	2.4	>10000	>1000	518	8.4	>10000	163.5	2.78	0.5	>10000	>1000	290	>10000	>1000	2	7300	2.29	923	>10000	129	0.4	1.7	>1000	0.72	<10	163	0.89	32.1	944	1.2	631	110.5
J008 (0-3.5)	30	3	482	702	0.6	252	10	518	0.57	>10000	10	26	2.4	>10000	>1000	643	2.24	>10000	185	6.61	0.73	>10000	>1000	250	>10000	>1000	2	6600	6.03	>1000	>10000	116.5	1	2.5	>1000	0.67	<10	133	1.52	46	>1000	2.3	679	235
J008 (0-3.5) DUP	30	2.8	499	644	0.8	265	10	526	0.54	>10000	10	26.5	2.1	>10000	>1000	701	2.45	>10000	195.5	7	0.77	>10000	>1000	250	>10000	>1000	3	6100	7.23	>1000	>10000	124.5	0.6	1.3	>1000	0.81	<10	153.5	1.75	48.1	>1000	1	706	269
J008 (3.5-4.6)	40	2.7	389	783	0.2	260	10	569	0.98	>10000	10	22.2	3.2	>10000	>1000	617	2.32	>10000	165.5	4.8	1.27	>10000	>1000	220	>10000	>1000	2	5900	4.08	>1000	>10000	81.2	0.9	2.5	>1000	0.59	<10	143.5	1.2	36.6	905	1.4	791	187.5
J008 (3.5-4.6) DUP	40	2.7	420	748	0.6	282	10	601	0.51	>10000	10	22.2	2.9	>10000	>1000	710	2.78	>10000	181	5.28	0.66	>10000	>1000	240	>10000	>1000	3	6600	5.22	>1000	>10000	96.5	0.7	2.1	>1000	0.81	<10	176.5	1.56	40.6	>1000	1.4	871	219
J012 (0-3)	20	2.1	571	842	<0.2	300	<10	581	0.54	>10000	10	18.9	2	>10000	>1000	406	10.05	>10000	148	3.48	0.51	7560	>1000	240	>10000	>1000	2	5600	5.66	708	>10000	114.5	1.5	2	>1000	1.04	<10	133	1.24	39.4	>1000	1	716	135
J012 (0-3) DUP	30	2.7	604	911	0.4	271	<10	621	0.37	>10000	10	21	2.2	>10000	>1000	472	10.85	>10000	166	3.72	0.55	8610	>1000	270	>10000	>1000	2	5600	6.52	826	>10000	129	0.9	1.4	>1000	1.27	<10	157.5	1.58	40.9	>1000	0.9	773	156
J012 (3-6)	30	3.1	535	926	<0.2	231	<10	707	0.32	>10000	10	18.1	2.2	>10000	>1000	389	7.4	>10000	147.5	2.66	0.68	>10000	>1000	280	>10000	>1000	1	5900	2.63	687	>10000	115.5	1.1	1.7	>1000	0.75	<10	165.5	0.67	35	1000	0.9	595	97.5
J012 (3-6) DUP	30	2.7	530	936	<0.2	246	<10	680	0.25	>10000	10	18.5	2.2	>10000	>1000	432	8.22	>10000	156.5	2.76	0.48	>10000	>1000	300	>10000	>1000	1	6200	2.61	709	>10000	130.5	0.6	1.4	>1000	0.87	<10	172	0.55	35.1	>1000	0.8	614	100.5
J013	30	3	436	529	0.2	243	<10	460	0.83	>10000	10	18.2	2.9	>10000	>1000	419	4.58	>10000	139.5	5.11	0.94	>10000	>1000	180	>10000	>1000	2	5100	11.05	584	>10000	121.5	0.6	1.9	>1000	0.72	<10	90.2	1.48	26.5	764	3.6	410	173
J013 DUP	30	3.5	453	521	<0.2	279	<10	467	0.3	>10000	10	20.8	2.6	>10000	>1000	511	4.33	>10000	160.5	5.87	0.54	>10000	>1000	180	>10000	>1000	3	5700	14.85	785	>10000	127.5	1.1	2.2	>1000	0.77	<10	92.3	1.65	29.3	830	1	456	212
J019 (0-2)	10	2.6	351	468	<0.2	194.5	10	257	0.29	>10000	10	31.3	1.1	>10000	>1000	341	3.69	>10000	179	9.67	0.76	4870	>1000	70	>10000	>1000	3	8600	10.4	379	>10000	129.5	0.7	1	>1000	0.88	<10	77.3	1.34	96.3	>1000	1.3	>1000	276
J019 (0-2) DUP	10	2.6	385	454	<0.2	213	10	301	0.41	>10000	<10	37.9	4.7	>10000	>1000	449	4.04	>10000	208	9.26	0.81	7070	>1000	90	>10000	>1000	3	>10000	12.55	592	>10000	141	1.7	29.3	>1000	1.28	<10	86.9	0.3	105	>1000	4.6	>1000	301
J024 (0-2)	30	2.4	684	548	<0.2	238	<10	633	0.29	>10000	10	22.2	8.3	>10000	>1000	463	9.24	>10000	164	3.28	0.66	8310	>1000	310	>10000	>1000	2	5400	4.82	768	>10000	120	1.1	2.4	>1000	1.16	<10	141.5	1.25	37.3	>1000	20.8	657	124
J024 (0-2) DUP	30	2.2	664	495	0.2	243	<10	606	0.27	>10000	10	22.1	2.5	>10000	>1000	528	9.36	>10000	163	3.05	0.56	8830	>1000	320	>10000	>1000	2	5600	4.81	860	>10000	122.5	0.7	1.5	>1000	1.34	<10	169.5	1.51	39.3	>1000	0.9	674	127
J029 (0-5)	30	3.2	324	718	0.6	209	<10	533	0.82	>10000	10	24.1	1.7	>10000	>1000	593	3.05	>10000	186	4.97	0.76	>10000	>1000	180	>10000	>1000	2	7400	4.89	>1000	>10000	108	0.5	1.8	>1000	0.61	<10	87.6	1.5	34.7	733	1.1	672	191
J029 (0-5) DUP	30	3.5	350	713	0.7	221	<10	569	0.64	>10000	10	25.1	1.8	>10000	>1000	675	3.25	>10000	203	5.8	0.72	>10000	>1000	180	>10000	>1000	2	8500	6.32	>1000	>10000	117.5	0.6	0.9	>1000	0.92	<10	99.2	1.6	37.9	816	1.8	756	222
J037 (0-3)	20	1.7	655	515	<0.2	297	10	559	0.35	>10000	10	20.9	2.1	>10000	>1000	584	7.89	>10000	151.5	4.29	0.66	6880	>1000	240	>10000	>1000	2	5800	6.69	738	>10000	123.5	1	1.9	>1000	1.05	<10	142	2.02	37.9	>1000	1.1	665	153.5
J037 (0-3) DUP	20	1.5	690	489	<0.2	282	10	538	0.36	>10000	10	21	2.4	>10000	>1000	596	8.92	>10000	157.5	4.57	0.56	7590	>1000	250	>10000	>1000	2	5800	7.07	742	>10000	135.5	0.8	1.3	>1000	1.16	<10	147.5	2.25	38.3	>1000	0.7	642	155.5
J037 (3-6)	30	3.4	551	524	<0.2	200	10	783	0.21	>10000	10	15	2.3	>10000	>1000	430	7.88	>10000	141	2.02	0.47	>10000	>1000	310	>10000	>1000	1	6400	2.33	763	>10000	119	0.4	1.8	>1000	0.79	<10	164.5	0.65	29.6	>1000	0.8	520	73.3
J037 (3-6) DUP	40	3.6	564	522	0.2	235	<10	807	0.17	>10000	10	15.4	2.2	>10000	>1000	466	9.08	>10000	148	2.03	0.39	>10000	>1000	310	>10000	>1000	2	6700	2.18	808	>10000	135	0.4	1.8	>1000	0.78	<10	169.5	0.6	29	>1000	0.8	526	73.4
J107	40	4.4	799	>1000	0.2	347	<10	617	0.58	>10000	30	52.4	5	>10000	>1000	656	10.3	>10000	312	12.05	1.05	>10000	783	240	>10000	>1000	4	8700	22	>1000	>10000	321	2.5	19.9	950	1.68	<10	64.3	1.18	51.3	>1000	3.4	429	336
J107 DUP	30	4.3	761	>1000	0.2	357	<10	606	0.54	>10000	40	49	6	>10000	>1000	675	10.9	>10000	324	12.75	0.97	>10000	782	240	>10000	>1000	4	9000	24.4	>1000	>10000	326	3	21	955	1.97	<10	66.8	1.62	53.2	>1000	4	422	354
Blank	<10	0.1	2	84.4	<0.2	3.1	<10	34	0.09	70	<10	<0.3	0.1	200	0.05	0.8	0.02	130	0.07	<0.05	<0.04	<60	0.02	<10	103	1.6	<1	300	<0.02	1.1	100	0.1	0.1	3.7	0.5	0.01	<10	0.01	<0.04	0.01	<1	0.2	<0.03	0.11

Sample ID	Ag mg/L	Al mg/L	As mg/L	Ba mg/L	Be mg/L	Ca mg/L	Cd mg/L	Co mg/L	Cr mg/L	Cu mg/L	Fe mg/L	K mg/L	Mg mg/L	Mn mg/L	Mo mg/L	Na mg/L	Ni mg/L	P mg/L	Pb mg/L	Sb mg/L	Sr mg/L	Ti mg/L	V mg/L	Zn mg/L
LOD	0.01	1	0.05	0.1	0.001	0.5	0.001	0.02	0.02	0.01	1	5	0.05	0.01	0.01	1	0.01	1	0.05	0.05	0.01	1	0.01	0.01
Method Code	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14	ME-ICP14
J007 (0-3)	0.01	123	<0.05	0.5	0.004	17.6	0.001	0.06	0.1	0.11	95	<5	13.25	3.17	<0.01	<1	0.11	5	<0.05	<0.05	0.11	1	0.17	0.07
J007 (0-3) DUP	<0.01	116	<0.05	0.5	0.004	17.1	<0.001	0.06	0.09	0.07	95	<5	14.55	3.14	<0.01	<1	0.12	5	<0.05	<0.05	0.11	1	0.17	0.06
J007 (3-5)	<0.01	388	0.1	3.1	0.02	308	0.004	0.5	0.78	1.13	615	12	178	41.3	<0.01	6	1.01	82	0.27	<0.05	1.69	6	0.95	1.19
J007 (3-5) DUP	<0.01	408	<0.05	3.4	0.02	323	0.005	0.55	0.8	1.07	671	13	190.5	49.2	<0.01	7	1.01	86	0.25	<0.05	1.81	7	1.04	1.16
J008 (0-3.5)	<0.01	514	<0.05	4.4	0.027	310	0.004	0.68	0.88	1.36	734	16	208	31.2	<0.01	7	1.17	70	0.22	<0.05	2.32	7	1.35	0.98
J008 (0-3.5) DUP	<0.01	546	0.07	4.7	0.028	310	0.006	0.74	0.94	1.29	862	17	258	34.8	<0.01	6	1.44	68	0.25	<0.05	2.37	9	1.47	1.02
J008 (3.5-4.6)	<0.01	457	0.09	4.1	0.023	317	0.003	0.66	0.71	1.59	642	12	204	36.7	<0.01	6	1.3	81	0.23	<0.05	1.78	6	1.02	1.08
J008 (3.5-4.6) DUP	<0.01	527	0.09	4.6	0.025	356	0.003	0.78	0.83	1.62	801	14	267	42.9	<0.01	8	1.69	93	0.31	<0.05	2.02	8	1.21	1.24
J012 (0-3)	<0.01	422	0.08	3.5	0.02	269	0.002	0.43	1.05	1.6	653	8	167.5	31.5	<0.01	5	0.81	74	0.3	<0.05	1.49	12	1.24	1.06
J012 (0-3) DUP	<0.01	447	0.09	3.7	0.022	274	0.003	0.5	1.07	1.7	708	9	186.5	36.8	0.01	6	0.91	75	0.26	<0.05	1.53	13	1.32	1.12
J012 (3-6)	<0.01	349	0.15	3.4	0.019	298	0.003	0.41	0.9	1.67	631	12	167	30	0.01	6	0.76	80	0.23	<0.05	1.46	9	1.13	1.23
J012 (3-6) DUP	<0.01	360	<0.05	3.5	0.02	303	0.005	0.44	0.93	1.75	675	14	176	35	0.01	6	0.82	81	0.24	<0.05	1.54	9	1.2	1.23
J013	<0.01	502	<0.05	3.6	0.019	243	0.005	0.45	0.7	0.9	490	12	117	29.7	<0.01	5	0.67	68	0.27	<0.05	1.39	8	0.84	0.72
J013 DUP	<0.01	599	0.15	3.8	0.022	258	0.006	0.54	0.71	0.87	579	11	146.5	36.3	<0.01	6	0.84	75	0.32	<0.05	1.48	9	0.91	0.74
J019 (0-2)	<0.01	805	0.05	3.5	0.032	203	0.007	0.35	0.64	0.91	772	5	80	14.6	<0.01	8	0.4	26	0.2	<0.05	1.79	3	2.09	0.48
J019 (0-2) DUP	<0.01	896	0.08	3.7	0.033	225	0.006	0.42	0.68	0.88	883	6	109.5	17.05	<0.01	8	0.56	33	0.17	<0.05	1.96	4	2.22	0.55
J024 (0-2)	<0.01	402	<0.05	3	0.023	260	0.004	0.47	1.25	1.05	823	8	186	34.8	0.01	5	0.89	62	0.24	<0.05	1.57	14	1.54	1.13
J024 (0-2) DUP	<0.01	416	<0.05	3.2	0.023	268	0.005	0.55	1.27	0.96	833	9	197.5	40.5	<0.01	6	0.94	66	0.26	<0.05	1.64	15	1.57	1.15
J029 (0-5)	<0.01	747	<0.05	4.4	0.025	267	0.005	0.62	0.56	1.31	616	13	194	24.3	<0.01	7	1.18	66	0.21	<0.05	2.06	5	0.79	0.94
J029 (0-5) DUP	<0.01	810	<0.05	4.7	0.026	292	0.005	0.7	0.62	1.36	739	13	246	26.5	<0.01	9	1.51	74	0.19	<0.05	2.18	5	0.87	1.04
J037 (0-3)	<0.01	419	<0.05	4.1	0.022	249	0.002	0.61	1.16	0.95	858	7	172	58.2	<0.01	6	0.82	59	0.27	<0.05	1.33	10	1.76	0.98
J037 (0-3) DUP	<0.01	435	0.05	4.1	0.023	253	0.002	0.66	1.23	0.91	914	8	180	60.5	<0.01	6	0.86	59	0.28	<0.05	1.35	11	1.96	0.96
J037 (3-6)	<0.01	319	0.06	3.4	0.016	309	0.005	0.47	0.92	0.91	619	12	192.5	37.2	<0.01	7	0.86	87	0.17	<0.05	1.27	8	1.18	1.33
J037 (3-6) DUP	<0.01	332	0.08	3.2	0.017	314	0.004	0.49	0.93	0.9	653	13	199.5	40.9	<0.01	7	0.89	89	0.21	<0.05	1.29	8	1.21	1.34
J107	<0.01	167	<0.05	0.6	0.005	14.4	<0.001	0.07	0.13	0.46	90	<5	16.8	2.74	<0.01	<1	0.2	5	<0.05	<0.05	0.09	1	0.18	0.09
J107 DUP	<0.01	172	<0.05	0.6	0.005	14.1	<0.001	0.07	0.13	0.42	98	<5	19.4	2.86	<0.01	<1	0.21	5	<0.05	<0.05	0.1	1	0.18	0.09
Blank																								

Duplicate 3.00 g samples (oven dry basis) were extraced with 15 mL of 1N nitric acid (2N nitric acid for samples containing free carbonates) in 50 mL polypropylene tubes and shaken for 20 h on a reciprocal shaker.

The tubes were then centrifuged at maximum RPM for 10 minutes and an aliquote filtered for metal analysis by the same suite of metals as conducted on the solids (Code ME-MS41). A Blank was setup and run through the analysis process.

Attachment 2-24

PolyMet. NorthMet Project Waste Characterization Data Package. Version 12. February 13, 2015.

NorthMet Project

Waste Characterization Data Package

Version 12

Issue Date: February 13, 2015

This document was prepared for Poly Met Mining Inc. by Barr Engineering Co.

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1.0 Introduction

This document presents the waste characterization data used by the Rock and Overburden, Flotation Tailings and Residue Management Plans, and for the water quality impact modeling presented in the NorthMet Water Modeling Packages (Volumes 1 and 2). In this document, Flotation Tailings are the NorthMet bulk flotation tailings, the Tailings Basin is the existing former LTV Steel Mining Company (LTVSMC) tailings basin, and the Flotation Tailings Basin refers to the Tailings Basin with the NorthMet basin. In addition, the Flotation Tailings Basin is designated FTB.

In cases where a supporting document is referenced, a general description of the supporting document is provided.

1.1 Outline

The outline of this document is:

Section 2 Describes the regulatory basis for the waste characterization data.

Section 3 Describes the data for waste characterization of overburden.

Section 4 Describes the data for waste characterization of waste rock and wall rock.

Section 5 Describes the data for waste characterization of tailings.

Section 6 Describes the data for waste characterization of residue.

Section 7 Describes the geochemical parameters for modeling overburden.

Section 8 Describes the geochemical parameters for modeling waste rock.

Section 9 Describes the geochemical parameters for modeling the pit lake.

Section 10 Describes the geochemical parameters for modeling tailings.

Section 11 Describes the geochemical parameters for modeling residue.

This document is intended to evolve through the environmental review, permitting, operating and closure phases of the NorthMet Project (Project). A Revision History is included at the end of the document and the most recently updated sections are highlighted in gray.

2.0 Regulatory Basis

Minnesota Rules, part 6132.1000 requires characterization of mine wastes from non-ferrous mining projects as part of the Permit to Mine process. The project proposer must meet with the Minnesota Department of Natural Resources (MDNR) staff to outline chemical and mineralogical analyses and laboratory tests to be conducted for mine waste characterization.

Poly Met Mining Inc. (PolyMet) met with the Lands and Minerals Division of the MDNR in 2004 and 2005 to develop waste characterization test plans that included chemical analysis of the mine waste, mineralogical/petrological analysis of the mine waste and laboratory tests describing acid generation and dissolved solids release from the mine waste. The tests were performed on material generated by exploration, preproduction sampling and process testing. The mine waste characterization has been conducted by SRK which has demonstrated proficiency in such analysis and was approved by the MDNR.

Requirements for the management of reactive mine waste are described in Minnesota Rules, part 6132.2200. The rule's objective is to prevent the release of substances that result in adverse impacts on natural resources. Per the rule, a generator of reactive mine waste must either:

- modify the physical or chemical characteristics of the mine waste, or store it in an environment, such that the waste is no longer reactive; or

- during construction to the extent practicable and at closure, permanently prevent substantially all water from moving through or over the mine waste and provide for the collection and disposal of any remaining residual waters that drain from the mine waste, in compliance with federal and state standards.

Overburden will be managed as a reactive mine waste. See Section 3.2 for a description of the mine waste management concept.

Waste Rock will be managed as a reactive mine waste. See Section 4.2 for a description of the mine waste management concept.

Flotation Tailings will be managed as a reactive mine waste. See Section 5.2 for a description of the mine waste management concept.

Hydrometallurgical Residue will be managed as a reactive mine waste. See Section 6.2 for a description of the mine waste management concept.

3.0 Overburden

The purpose of waste characterization for overburden is to determine the geochemical characteristics of the overburden generated by the Project and use those characteristics to develop a general overburden management concept.

3.1 Reports

3.1.1 Initial Report

The initial report (Reference (1)) was submitted in 2008. A summary of the report is presented in the following paragraphs.

The results of initial chemical testing (37 samples) and leaching testing (14 samples) of NorthMet overburden were integrated with knowledge of glacial movement to develop an overall classification of the overburden:

- Saturated Overburden – overburden that has remained below the water table and has not been oxidized and can release metals when exposed to air and oxidized
- Unsaturated Overburden – overburden that has been above the water table and has been oxidized so that metals have already been released
- Peat – organic material that will not release metals associated with sulfide minerals but can release mercury

The leach testing results were used to develop reasonable worst case (95th percentile) contact water chemistry information for constituents from each class of overburden. The reasonable worst case contact water concentrations are presented in Table 3-1.

An updated report (Reference (2)) was submitted in 2010. A summary of the report is presented in the following paragraph.

Additional samples were collected and the results of leach testing were used to reassess the reasonable worst case contact water chemistry for constituents from Unsaturated Overburden. The additional data confirmed the characterization of the Unsaturated Overburden throughout the site, as shown by the comparison of reasonable worst case contact water chemistry in Table 3-2.

Table 3-1 95th Percentile Results from Overburden Leaching Tests

Constituent	Units	Peat	Mineral Overburden	
			Saturated	Unsaturated
pH		6.9	4	6.9
Alkalinity	mg $CaCO_3$/L	79	36	12
Fluoride	mg/L	1	0.56	0.45
Chloride	mg/L	8.8	3.8	3.4
Sulfate	mg/L	92	210	15
Aluminum (Al)	mg/L	0.13	0.63	0.3
Antimony (Sb)	mg/L	0.00069	0.0012	0.00098
Arsenic (As)	mg/L	0.0043	0.0028	0.0029
Barium (Ba)	mg/L	0.034	0.026	0.013
Beryllium (Be)	mg/L	<0.0002	0.00055	<0.0002
Boron (B)	mg/L	0.23	0.087	0.028
Cadmium (Cd)	mg/L	<0.00004	0.005	0.00015
Calcium (Ca)	mg/L	23	26	5.7
Chromium (Cr)	mg/L	0.00094	0.0012	0.00097
Cobalt (Co)	mg/L	0.00066	0.23	0.0015
Copper (Cu)	mg/L	0.011	0.44	0.008
Iron (Fe)	mg/L	0.12	5.5	0.059
Lead (Pb)	mg/L	0.00022	0.0011	<0.00005
Magnesium (Mg)	mg/L	11	18	2.1
Manganese (Mn)	mg/L	0.19	1.1	0.1
Mercury (Hg)	µg/L	0.018	< 0.02	0.016
Molybdenum (Mo)	mg/L	0.028	0.034	0.013
Nickel (Ni)	mg/L	0.0063	2.2	0.0031
Selenium (Se)	mg/L	0.00089	0.0034	0.00052
Silver (Ag)	mg/L	0.0013	<0.00005	<0.00005
Sodium (Na)	mg/L	45	13	4.2
Tellurium (Te)	mg/L	<0.0002	<0.0002	<0.0002
Thallium (Tl)	mg/L	0.0001	0.000025	0.000025
Vanadium (V)	mg/L	0.0042	0.0022	0.00059
Zinc (Zn)	mg/L	0.0038	0.86	0.0056

Table 3-2 Comparison of Results from Unsaturated Overburden Leaching Tests

Constituent	Drilling Samples (2008)	Sump Spoils (2010)
Total Concentration, 95th Percentile		
n	11	13
Sulfur (%)	0.03	0.03
Copper (ppm)	104	52
Nickel (ppm)	71	46
Contact Water Leachate, 95th Percentile		
n	3	13
Sulfate (mg/L)	15	8.4
Copper (mg/L)	0.008	0.007
Nickel (mg/L)	0.003	0.002

3.1.2 2011 Update

To accommodate the change in overall water quality modeling concept from deterministic to probabilistic, the results of leach testing (Table 3-1) were used to develop probability distributions of leachate chemistry for constituents from Unsaturated Overburden and Peat. Uniform distributions were defined from the maximum and minimum concentrations reported in Reference (1). Ranges for the leachate chemistry and the modeling methodology are discussed in Section 7.0.

No probability distributions were developed for the leachate chemistry for Saturated Overburden. As discussed in Section 7.0, Saturated Overburden will be treated as Category 2, 3 or 4 waste rock.

3.2 Overburden Management Concept

The concept for management of overburden is to classify overburden and to manage each classification as appropriate to its potential to release constituents.

- Unsaturated Overburden – Low potential to release constituents at levels that could have significant environmental impact; use as general construction material

- Saturated Overburden – Potential to release constituents at levels that could have significant environmental impact; treat as Category 2, 3 or 4 waste rock (Section 7.3) or use in construction applications where it will be placed below the water table or above a membrane liner

- Peat – Potential to release mercury in drainage water; place in a primary storage location where surface runoff is managed and monitored, material to be used for reclamation and restoration

4.0 Waste Rock

The purpose of waste characterization for waste rock is to determine the geochemical characteristics of the waste rock generated by the Project and use those characteristics to develop a general waste rock management concept.

4.1 Reports

4.1.1 Initial Report

The initial report (Reference (3)) was submitted in 2007. A summary of the report is presented in the following paragraphs.

The results of initial chemical testing and long-term kinetic testing of 89 samples of NorthMet waste rock (82 samples plus 7 duplicates) and 3 samples of NorthMet ore were integrated with very long-term kinetic test work on rock with similar geology to develop an overall classification of the waste rock:

- Category 1 – sulfur content less than or equal to 0.12% - will not generate acid but may release metals
- Category 2 – sulfur content greater than 0.12% and less than or equal to 0.31% - may generate acid and consequently release metals at higher rates than Category 1
- Category 3 – sulfur content greater than 0.31% and less than or equal to 0.60% - will eventually generate acid and consequently release metals at higher rates than Category 2
- Category 4 – sulfur content greater than 0.60% - will rapidly generate acid and consequently release metals at higher rates than Category 3

Note that the above classification is a modification of the classification in Reference (3) in that the use of Cu/S ratio was eliminated and a sulfur content only classification developed.

The kinetic testing results were used to develop reasonable worst case (95th percentile) release rates for constituents from each category of waste rock. The reasonable worst case release rates are presented in Table 4-1.

Table 4-1 2007 95th Percentile Results from Humidity Cell Tests (mg/kg/week)

Constituent	2[(1)]	3 (nonacidic)	4 (nonacidic)	4 (acidic Virginia)	Ore
Total Acidity	1.3	1.4	1.4	24	1.4
Alkalinity	7.9	9.5	15	0.88	5.2
Fluoride	0.027	0.024	0.030	0.034	0.041
Chloride	0.11	0.14	0.11	0.11	0.11
Sulfate	2.3	11	11	50	23
Aluminum (Al)	0.087	0.052	0.063	0.37	0.017
Antimony (Sb)	0.0044	0.0068	0.0044	0.00071	0.0014
Arsenic (As)	0.0088	0.0081	0.0075	0.0052	0.0059
Barium (Ba)	1.3	1.4	1.4	24	1.4
Boron (B)	0.0039	0.0054	0.016	0.021	0.011
Cadmium (Cd)	0.000020	0.000028	0.000021	0.0032	0.000022
Calcium (Ca)	2.2	4.7	3.4	3.5	7.3
Chromium (Cr)	0.00011	0.00011	0.00013	0.00012	0.00010
Cobalt (Co)	0.000053	0.0059	0.000086	0.039	0.0028
Copper (Cu)	0.00085	0.0084	0.00078	0.0048	0.0053
Iron (Fe)	0.015	0.011	0.030	9.5	0.0074
Lead (Pb)	0.000063	0.000069	0.000059	0.0011	0.000076
Magnesium (Mg)	0.44	0.82	0.31	3.9	1.5
Manganese (Mn)	0.00096	0.023	0.0033	0.12	0.022
Molybdenum (Mo)	0.000027	0.000043	0.00014	0.000026	0.000034
Nickel (Ni)	0.00024	0.070	0.00090	0.56	0.057
Selenium (Se)	0.00011	0.00020	0.00042	0.00060	0.00012
Silver (Ag)	0.000025	0.000031	0.000096	0.000029	0.000025
Thallium (Tl)	0.000010	0.000010	0.000010	0.000012	0.000010
Zinc (Zn)	0.0013	0.0040	0.00069	0.60	0.0021

(1) Humidity cell test results for Category 2 waste rock were used to represent the mixed Category 1/2 Waste Rock Stockpile.

4.1.2 2009 Update

A general geochemical update report (Reference (4) Section 3) was submitted in 2009. A summary of the report is presented in the following paragraphs.

The results of ongoing kinetic testing were used to reassess and update the reasonable worst case release rates for constituents from each category of waste rock. In addition, Reasonable Alternative 1 was incorporated into the Mine Plan, removing Category 2 waste rock from the combined Category 1/2 Waste Rock Stockpile and instead including Category 2 waste rock in a combined Category 2/3 Waste Rock Stockpile. These changes are reflected in the updated reasonable worst case release rates presented in Table 4-2.

Following the publication of the 2009 geochemical update report, the waste rock characterization program was modified in consultation with MDNR to stop some tests and modify the frequency of analysis of leachates based on interpretation of the geochemical trends. The tests that were continued are indicated in Large Table 1 as having a total duration greater than 198 weeks (43 samples total were continued).

Table 4-2 Waste Rock and Ore: 2009 95th Percentile Results from Humidity Cell Tests (mg/kg/week)

Constituent	Category 1[1]	Category 3[2] (nonacidic)	Category 4: nonacidic	Category 4: acidic (Virginia)	Ore
Total Acidity	1.4	1.8	1.9	26	2.1
Alkalinity	3.3	3.5	2.8	0.16	1.9
Fluoride	0.025	0.025	0.030	0.033	0.031
Chloride	0.10	0.10	0.14	0.10	0.11
Sulfate	1.3	8.3	17	50	20
Aluminum (Al)	0.063	0.043	0.022	0.51	0.0081
Antimony (Sb)	0.0025	0.0021	0.0021	0.0020	0.0025
Arsenic (As)	0.00035	0.0033	0.0054	0.00054	0.00077
Barium (Ba)	0.0056	0.0086	0.0064	0.0038	0.0063
Beryllium (Be)	0.00011	0.00011	0.00012	0.00056	0.000098
Boron (B)	0.0013	0.0046	0.012	0.015	0.011
Cadmium (Cd)	0.000020	0.000065	0.000043	0.0031	0.000070
Calcium (Ca)	1.8	3.6	5.2	2.2	5.5
Chromium (Cr)	0.00010	0.00010	0.00011	0.00012	0.000098
Cobalt (Co)	0.000053	0.017	0.019	0.039	0.037
Copper (Cu)	0.00088	0.096	0.0080	0.0071	0.059
Iron (Fe)	0.0067	0.012	0.10	9.7	0.0060
Lead (Pb)	0.000031	0.000052	0.00019	0.00060	0.000054
Manganese (Mn)	0.00080	0.027	0.049	0.087	0.086
Mercury (Hg)	0.0000045	0.0000045	0.0000045	0.0000099	0.0000098
Molybdenum (Mo)	0.000041	0.000032	0.000045	0.000024	0.000029
Nickel (Ni)	0.00034	0.21	0.14	0.47	0.62
Selenium (Se)	0.000098	0.00024	0.00033	0.00050	0.00023
Silver (Ag)	0.000024	0.000042	0.00015	0.000026	0.000025
Thallium (Tl)	0.0000098	0.000011	0.000011	0.000012	0.000014
Vanadium (V)	0.00021	0.0016	0.0012	0.000097	0.00011
Zinc (Zn)	0.0012	0.011	0.0073	0.49	0.015

(1) These values were incorrectly labeled as Category 2 in Reference (4).
(2) Includes humidity cell test results for Category 2 waste rock to represent the mixed Category 2/3 Waste Rock Stockpile.

4.1.3 2011 Update

4.1.3.1 2011 Geochemical Update

An updated analysis of the humidity cell data was submitted in 2011 (included as Attachment A). A summary of the report is presented in the following paragraphs. Note that a minor plotting error has been identified in Attachment A (the plot in Attachment 3 of Attachment A labeled total acidity is of total alkalinity) that does not affect the release rates used for modeling the NorthMet waste rock.

In order to accommodate the change in overall water quality modeling concept from deterministic to probabilistic, the results of ongoing kinetic testing were reevaluated and a new modeling approach was developed. The modeling approach is discussed in detail in Sections 8.1, 8.2, and 8.3.

Previous interpretation of the humidity cell test results assumed that the rates of constituent leaching indicated by the humidity cells were not constrained by solubility limitations due to the high liquid-to-solid ratios used in laboratory kinetic tests. Data from the humidity cells demonstrate that this is not the case for NorthMet waste rock: nickel leaching, for example, can increase by orders of magnitude even as oxidation and weathering of the sources of nickel decrease. A new approach was needed to account for the influence of solubility limits on the humidity cell leaching rates.

Chemical constituents, including, major mineral components and trace metals are released from host primary minerals as these minerals weather. The Project data indicated that after constituents are released from primary minerals, the concentrations of some of the chemical constituents in humidity cell tests were limited by solubility relationships with secondary minerals, while others were not. The release rates of constituents that appeared to be subject to solubility limitations were determined by identifying the likely mineral sources for each constituent using Project data and information from literature sources. The release rates of constituents that were potentially controlled by solubility relationships were identified by correlations with constituents in the same primary mineral host that were not controlled by solubility limitations.

This approach to estimating release rates relies on empirical analysis of the humidity cell data and data for metal content in the drill core samples (aqua regia digestion) and individual minerals (microprobe analysis). The rates and relationships determined for the different waste rock categories are therefore specific to this data set. The specific humidity cells used in this analysis, their rock type, sulfur content, and test duration are shown in Large Table 1. The proposed approach for developing release rates for each constituent is outlined in Large Table 2 and described further in Section 8.1.

4.1.3.2 Category 1 Concentration Caps Study

A laboratory study was initiated in 2010 in order to determine concentration caps for the Category 1 waste rock, as documented in Attachment B. A summary of the report is presented in the following paragraphs.

Concentration caps for the Category 1 waste rock were previously developed from the MDNR's AMAX field experiments, which were performed on Duluth Complex rock with much higher sulfur and metals content than is expected in Category 1 waste rock. The 2010 laboratory study was designed to provide direct measurements of concentration caps for use on Category 1 waste rock using a Sequential Meteoric Water Mobility Procedure (SMWMP).

The concentration caps study used oxidized coarse rejects from NorthMet drill cores to construct seven test columns for each of six samples representing the full range of sulfur in the Category 1 Waste Rock Stockpile. The sample material was all from 2005 drilling and was not freshly crushed (Attachment 1 of Attachment B). Water was slowly dripped through a column over the course of about one week and the exit water chemistry analyzed. This test is comparable to the State of Nevada's Meteoric Water Mobility Procedure. The collected leachate (less water removed for analysis) was passed through the subsequent column at the same leachate to sample ratio, and the procedure was repeated for all seven test columns for a given sulfur content (seven columns x six sulfur contents = 42 columns total). This method effectively simulates the performance of a tall column of oxidized waste rock (i.e., a waste rock stockpile) and provides ample contact time in order to achieve equilibrium chemistry.

Based on the results of this study, maximum concentration caps for the Category 1 waste rock have been developed for the majority of the constituents studied. Concentration caps are identified from the SMWMP test when concentrations stabilize in the later sequences of the test. For most constituents, this stabilizing trend occurs after several sequences of increasing concentrations as additional solute is mobilized from contact with newly-rinsed rock. For the purposes of this test, "stability" is defined as a lack of clear upward trends in all of the test samples. The concentration caps developed from this study are considered representative of the geochemical conditions at the pH range observed in the study, from pH 7.9 to pH 8.3.

For many constituents, the reported concentrations are close to the analytical detection limits, which varied throughout the study as leachate dilution was required in the final leaching step (resulting in higher detection limits). At concentrations less than five times the detection limit, analytical precision is lowest, with confidence limits ranging from ± 20% to 100% of the reported value. Because of the generally low concentrations for many trace metals, some erratic behavior is anticipated due to this low precision especially for the final leaching sequence. At concentrations between five and ten times the detection limit, confidence limits range from ± 10% to 15%, and above ten times the detection limit confidence limits are typically ± 10%. If stable concentrations are observed for multiple test sequences, erratic behavior at the final leaching sequence should not be interpreted as a change in the overall determination of concentration caps. In general, the concentration cap for each test sample is

defined for modeling purposes as the highest concentration observed in the final two leaching sequences.

The SMWMP test also included a method blank, in which column construction materials were flushed using the same procedure as the test rock. For some constituents, the trace metal concentration in the blank was at or above the concentration in the rock leachate, reflecting possible contamination by the column materials. Because the water quality of the leachate was only analyzed at the final sequence, however, the effect of trends in trace metal concentrations is unknown.

Table 4-3 presents the maximum observed concentrations for all of the constituents in the analysis. Concentrations for some constituents were not constrained in the study, indicating that concentration caps were not reached. Alternative methods of determining concentration caps for select constituents are recommended in Attachment B. This information is used to develop probability distributions for the PolyMet method for modeling concentration caps, as discussed in Section 8.3. These results are not used in the water quality modeling at the direction of the Co-lead Agencies, as discussed in Section 8.3.1.

Table 4-3 Maximum Concentrations from Category 1 SMWMP Test

Constituent	units	Indicated Maximum Range(1)	Capped	Alternative Method
pH(2)	s.u.	7.9 – 8.3	yes	
Alkalinity	mg $CaCO_3$/L	119 – 300	yes	
Fluoride	mg/L	0.23 – 0.53	yes	fluorite model(4)
Sulfate	mg/L	150 – 910	no	gypsum model(4)
Aluminum (Al)	mg/L	0.026 – 0.073	yes	
Antimony (Sb)	mg/L	0.007 – 0.012	yes	
Arsenic (As)	mg/L	0.008 – 0.053	yes	
Barium (Ba)	mg/L	0.004 – 0.034	no	barite model(4)
Beryllium (Be)	mg/L	0.00001	yes(3)	
Boron (B)	mg/L	0.10 – 0.46	no	field data(4)
Cadmium (Cd)	µg/L	0.008 – 0.051	yes	
Calcium (Ca)	mg/L	12.3 – 122	no	gypsum model(4)
Chromium (Cr)	mg/L	0.002 – 0.003	no	field data(4)
Cobalt (Co)	mg/L	0.05 – 0.31	yes	
Copper (Cu)	mg/L	0.0024 – 0.0065	no	field data(4)
Iron (Fe)	mg/L	0.006 – 0.017	yes	
Lead (Pb)	mg/L	0.00016 – 0.0012	yes(5)	
Manganese (Mn)	mg/L	0.005 – 0.017	yes	
Mercury (Hg)	µg/L	0.01 – 0.03	yes	
Molybdenum (Mo)	mg/L	0.014 – 0.021	yes	
Nickel (Ni)	mg/L	0.003 – 0.021	no	field data(4)
Selenium (Se)	mg/L	0.0025 – 0.0077	no	gypsum sequest.(4)
Silver (Ag)	µg/L	0.006 – 0.035	yes	
Thallium (Tl)	µg/L	0.014 – 0.050	yes	
Vanadium (V)	mg/L	0.0006 – 0.0030	yes	
Zinc (Zn)	mg/L	0.0022 – 0.0089	yes	

(1) Range shown is generally the highest observed from the final two leaching cycles for each of the 6 test samples
(2) pH value shown is the lowest observed after the first cycle
(3) Beryllium was not detected in the SMWMP leachate, the detection limit is shown as the concentration cap
(4) Section 8.3.1 for proposed alternative data sources to define concentration caps
(5) Lead concentrations peaked in cycles 3-4 then stabilized at lower levels in final cycles, peak values shown

4.1.4 2012 and 2014 Humidity Cell Trends Updates

Analyses of the ongoing humidity cell data were submitted in 2012 (Attachment C, including data through February 2012, and 2014 (Attachment D, including data through August 2013). A summary of these documents is presented in the following paragraphs.

- All Category 1 humidity cells have yielded pH above 6 throughout the program, typically fluctuating between 6.5 and 7.5 after an initial decline. For the 12 continuing tests, there is no indication that pHs will decline. Sulfate leaching rates have been low throughout the program with most continuing tests showing rates below 1 mg/kg/week.
- Six of the Category 2/3 humidity cells (including 3 of the 19 continuing tests, all of which contain Category 3 rock) generated acidic leachate (pH below 6), while all other samples have shown stable pHs (pH above 6) with similar trends to the Category 1 samples. The continuing nonacidic samples have now shown no acidic leachate after more than eight years of laboratory testing. Sulfate leaching rates have been between 1 and 10 mg/kg/week with the highest rates observed for samples with higher sulfur contents. A number of samples showed upward trends in nickel and cobalt leaching related to pH decrease below 7, with the trend being most pronounced for the samples generating acidic leachate (pH below 6). Nickel, cobalt and copper leaching have peaked and declined for most samples.
- All Category 4 humidity cells (Duluth Complex and Virginia Formation) have generated acidic leachate, demonstrating that most Category 4 rock can be expected to generate acid and that the timeframe to onset in humidity cell tests is several years in most cases. Some increases in sulfate leaching have been observed as pH decreases. Also, following a peak concentration sulfate leaching has been observed to decline. All samples show accelerated nickel leaching as pH decreases, as well as continuing upward trends in copper leaching.
- All three ore composite samples have shown pH and leaching trends similar to the Category 2/3 waste rock samples that have generated acidic leachate. The tests have shown a stable pH near 5 and nickel and copper leaching to a peak concentration, and then declining.

4.2 Waste Rock Management Concept

The concept for management of waste rock is to classify waste rock and to manage each classification as appropriate to its potential to release constituents.

Waste rock that will not generate acid (Category 1) will be placed in a permanent stockpile with engineered systems to collect water that has contacted the rock and reduce the amount of water that contacts the rock at closure.

Waste rock that will or may generate acid (Category 2,3,4) will be placed on temporary stockpiles with engineered systems to collect water that has contacted the rock and this rock will be relocated to the East Pit when mining is completed in that pit.

4.3 Waste Characterization Program as Applied to Waste Rock Stockpiles

Large Table 3 shows the waste rock that will be placed in stockpiles or the East Pit organized by waste classification and geological unit compared to waste characterization samples for that same waste classification and geological unit.

There are 89 samples in the kinetic testing program. All samples selected for kinetic testing were also characterized using physical, mineralogical and chemical methods. Physical characterization involved determination of specific gravity and particle size distribution. Mineralogical characterization was optical and sub-optical. For the latter, microprobe was used to determine metal content of major sulfides, silicates and oxides. Static chemical analysis included sulfur and metal content determination on whole samples and particle size fractions and carbonate determination on whole samples. The base method for kinetic testing was the ASTM humidity cell (Procedure D 5744 – 96). Leachate analysis was performed to obtain the best possible detection limits using ICP-MS.

The Mine Plan will generate 217 million tons of Category 1 waste rock with an average sulfur content of 0.06%. The waste characterization program has 42 samples of this rock averaging 0.06% sulfur content. There are no samples for the Unit 7 rock, but as discussed Section 4.3.1, the waste characterization samples for Units 4, 5 and 6 represent the geochemical behavior of Unit 7.

The Mine Plan will generate 83 million tons of Category 2/3 waste rock with an average sulfur content of 0.21%. The waste characterization program has 26 samples of this rock averaging 0.29% sulfur content. There are no samples for the Unit 7 rock, but as discussed in Section 4.3.1, the waste characterization samples for Units 4, 5 and 6 represent the geochemical behavior of Unit 7.

The Mine Plan will generate 8.6 million tons of Category 4 waste rock with an average sulfur content of 1.90%. The waste characterization program has 21 samples of this rock averaging 1.90% sulfur content. There are no samples for Unit 3 rock, but the average sulfur content of 0.99% is less than the average sulfur content of the Virginia Formation (2.43%), which comprises most of the rock in the stockpile.

4.3.1 Waste Characterization by Geologic Unit and Rock Type

4.3.1.1 Combination of Units 2 & 3 and 4 & 5

Units 2 & 3 and Units 4 & 5 were combined in the Block Model into Units 3 and 5, respectively. Units 4 and 5 are now seen as one unit with a gradational change in texture from heterogeneous at the base (Unit 4) to more homogenous near the top (Unit 5). The

division between the two has always been arbitrary and the combined thickness of the two is much more consistent across the deposit than the thicknesses of the individual units as defined in drill core logging. The data as to whether Units 2 and 3 are a single package are much more ambiguous, but as with Units 4 & 5, the combined thicknesses are more consistent across the deposit than the thicknesses of the individual units and the contact between 2 & 3 can be somewhat arbitrary. In general, the basal contacts of Units 7, 6, 4, 2 and 1 are sharp, that is that as one goes along the core there is a visible change over a very short distance in texture, mineral proportion, or sulfide content. The basal contacts of Units 5 and 3 are gradational in that these changes occur over a distance of feet to tens of feet

4.3.1.2 Unit 7

About 23 million tons (about 7.4% of total waste rock) of Unit 7 rock will be placed in stockpiles or the East Pit. Because the preliminary 20 year mine pit shell that was available when the Waste Characterization Plan was developed did not show that Unit 7 will be mined, no samples from Unit 7 were included in the waste characterization plan. The waste characterization samples for Units 4, 5 and 6 represent the geochemical behavior of Unit 7 because:

- The basic geology of the Units 4, 5, 6 and 7 (the upper anorthositic rocks of the deposit) is essentially the same. Whole rock analysis (Figure 4-1) shows Units 4, 5 and 7 to chemically be nearly identical and Units 6 and 7 to be similar, with differences in iron and magnesium content between the two reflecting slightly different proportions of olivine and pyroxene in those units. Generalized mineralogy is the same for all units at NorthMet, plagioclase, olivine and pyroxene in varying proportions make up the bulk of the rock (Reference (5) pages 16-19). This is also reflected in silicate, sulfide and oxide mineral chemistry (Reference (5) appendices).

- Sulfur distribution in Units 4, 5, 6 and 7 is very similar. Figure 4-2 shows the sulfur distribution in the Unit 5 (4 & 5), 6 and 7 core (light line with markers) used to select samples for the kinetic test program and in the larger amount of core available now (heavy line). This shows that the additional drill core data have not changed the distribution. The large circular markers show the percent sulfur in the Unit 5 and Unit 6 waste characterization tests, which cover the range of Unit 7 sulfur content. Figure 4-3 through Figure 4-7 show the minimum, average and maximum for sulfur, copper, nickel, cobalt and zinc in the current core data by unit. The Unit 7 values are within the range of the other units. These figures show that there is not more sulfur or metals present in Unit 7 compared to the other units.

- Stratigraphic unit is not a significant driver to the acid producing or metal leaching rates (Reference (6)).

Unit 7 in the area to be mined contains an estimated 10 to 20 percent ultramafic rock while other units have less. Because the data set used to generate Figure 4-1 through Figure 4-7

includes ultramafic and troctolitic/anorthositic rocks, the fact that there is more ultramafic in Unit 7 than the other units is included in the above analysis. Table 4-4 shows that each Category of waste rock in the waste characterization program has ultramafic samples. As stated above, in general, metals do not vary by rock type in the waste rock. There are a few broad exceptions driven by mineralogy, such as higher aluminum in more anorthositic rocks (higher plagioclase content) and higher iron and magnesium in more ultramafic (higher olivine content) rocks.

Table 4-4 Waste Characterization Tests by Rock Type

			Sulfur		
Stockpile	**Rock Type**	**Samples**	**Min**	**Avg**	**Max**
Cat 1	Anorthositic	10	0.02	0.04	0.09
Cat 1	Troctolitic	24	0.02	0.05	0.08
Cat 1	Ultramafic	8	0.06	0.08	0.12
Cat 2/3	Anorthositic	2	0.18	0.18	0.18
Cat 2/3	Sedimentary Hornfels	3	0.24	0.41	0.55
Cat 2/3	Troctolitic	17	0.14	0.29	0.59
Cat 2/3	Ultramafic	4	0.16	0.25	0.34
Cat 4	Anorthositic	3	0.68	1.12	1.83
Cat 4	Sedimentary Hornfels	4	1.46	2.53	4.46
Cat 4	Troctolitic	6	0.77	1.22	1.68
Cat 4	Ultramafic	4	0.72	0.98	1.24
Cat 4	Virginia	4	2.00	3.82	5.68

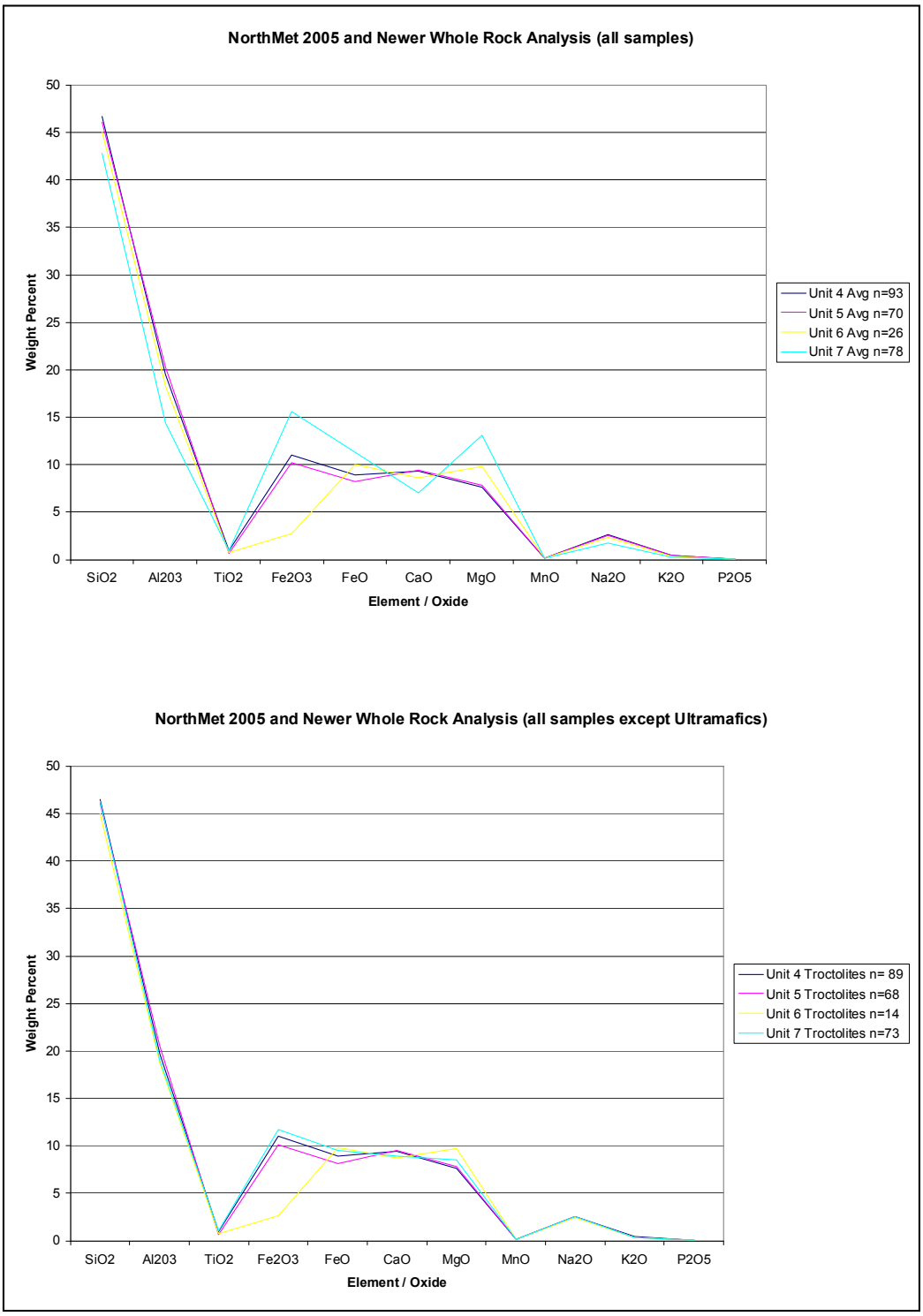

Figure 4-1 Whole Rock Analysis

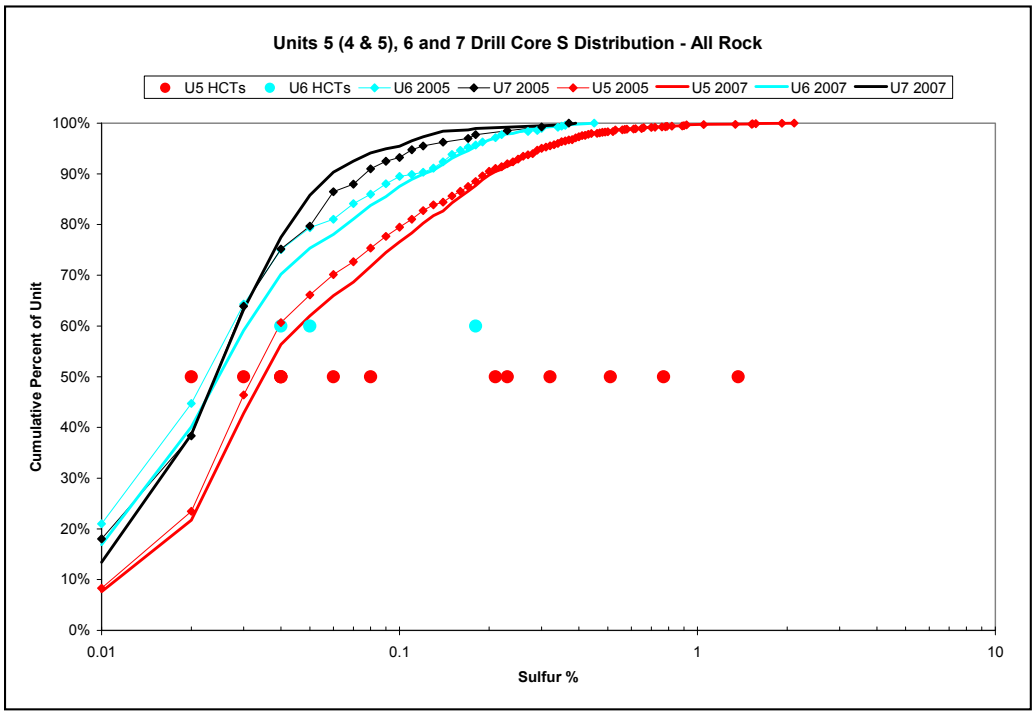

Figure 4-2 Drill Core Sulfur Distribution with Samples

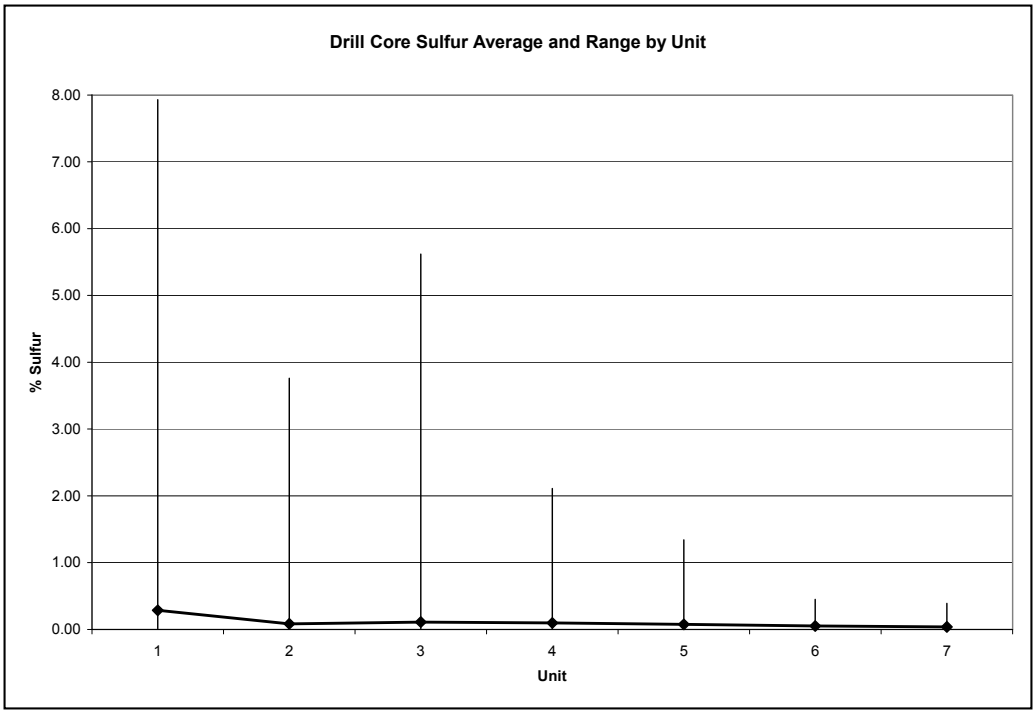

Figure 4-3 Drill Core Sulfur

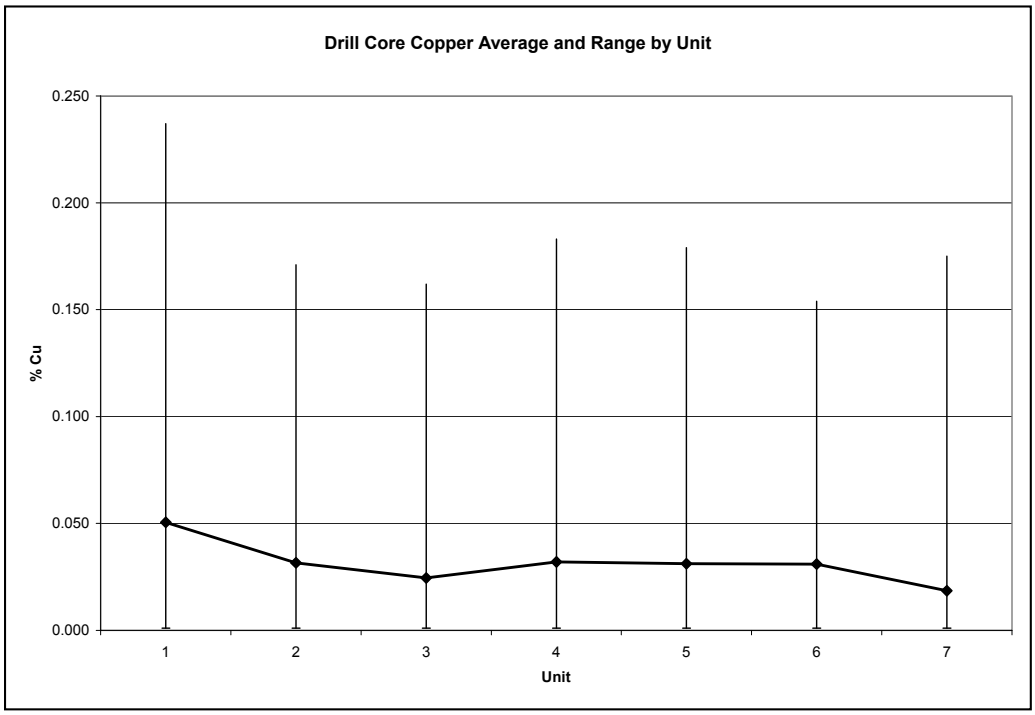

Figure 4-4 Drill Core Copper

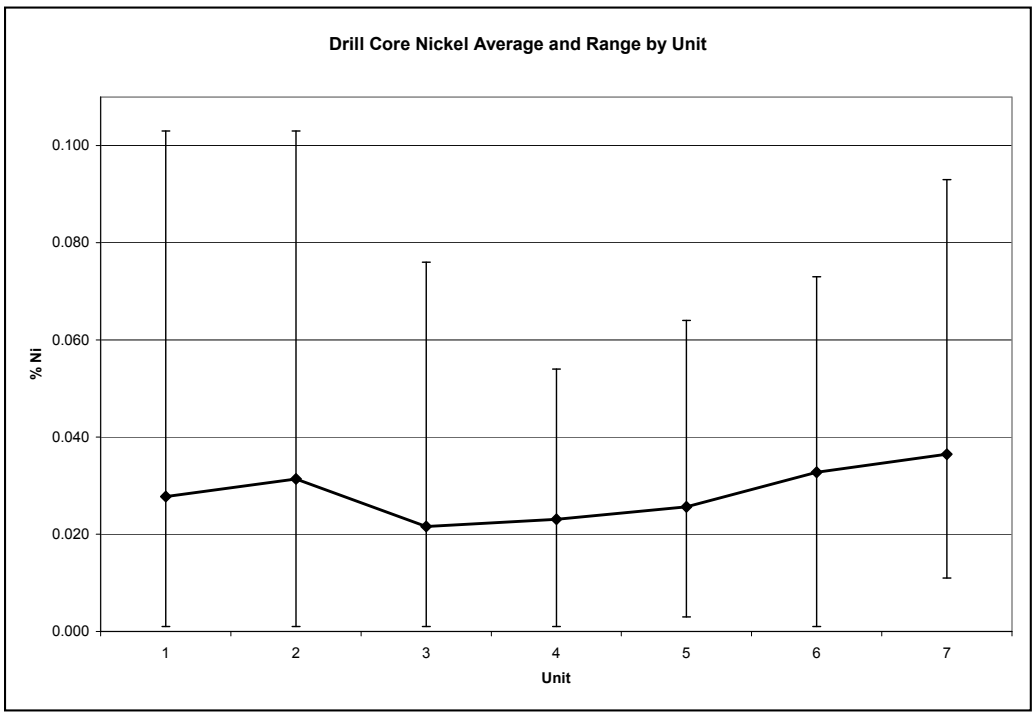

Figure 4-5 Drill Core Nickel

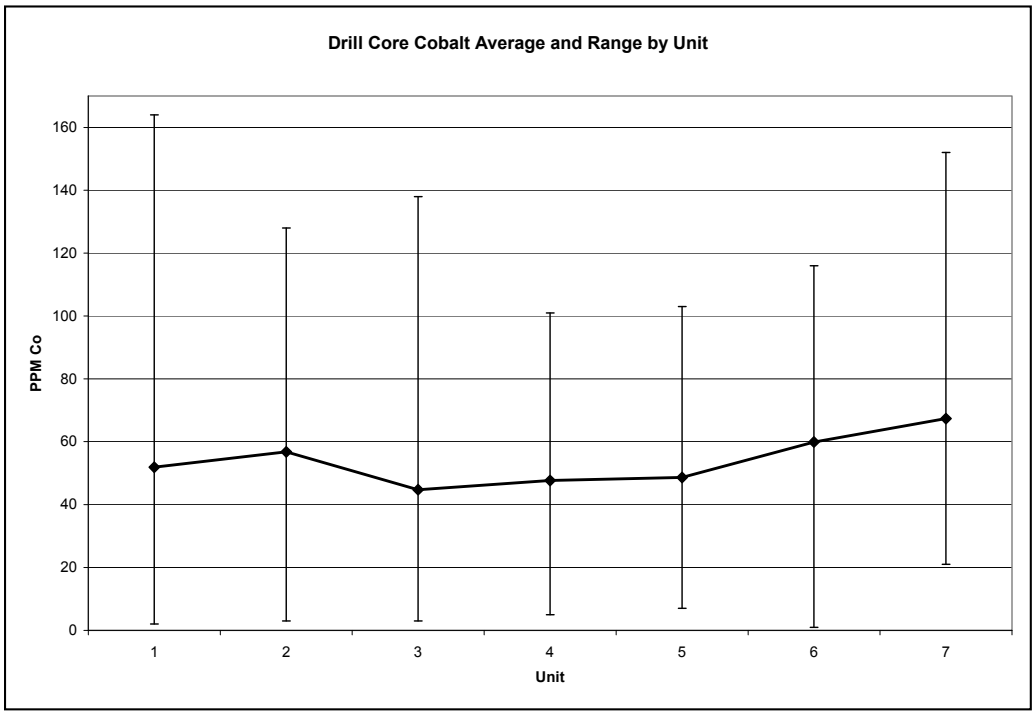

Figure 4-6 Drill Core Cobalt

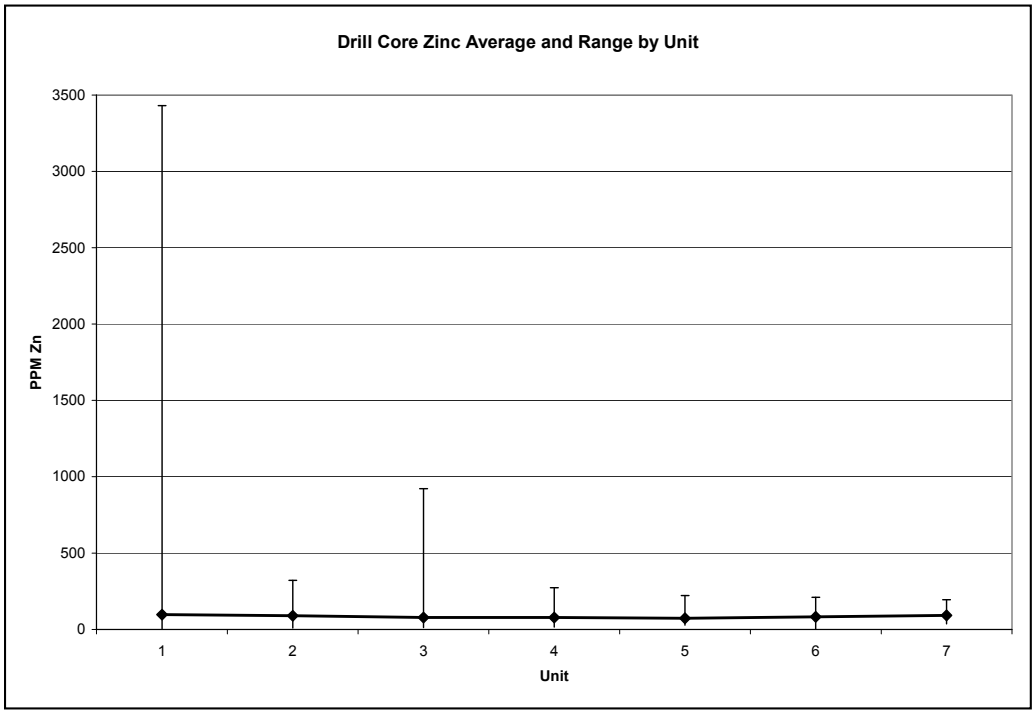

Figure 4-7 Drill Core Zinc

4.3.2 Sulfur Content of Stockpiles and Pit Walls

The Block Model contains the modeled sulfur content of each block of waste rock and ore. This information is used as described in Section 8.1.1.1 to model sulfate release for the Category 1 and Category 2/3 waste rock and ore (stockpiles and pit walls) and is not used for modeling the other waste rock categories. Sulfur distribution data for all waste and wall rock categories are presented here for completeness.

The average sulfur content of the Category 1 waste rock as a whole is 0.064%. The sulfur content of the portion of the Category 1 waste rock to be stored in the Category 1 Waste Rock Stockpile (approximately three quarters of all Category 1 material generated) is 0.063%. The sulfur content of the Category 1 waste rock placed directly in the East Pit is 0.068%. The distribution of sulfur content by mass for the Category 1 waste rock is shown in Figure 4-8.

The average sulfur content of the Category 2/3 waste rock as a whole is 0.21%. The sulfur content of the portion of the Category 2/3 waste rock to be temporarily stored in the Category 2/3 Waste Rock Stockpile (approximately half of all Category 2/3 material generated) is 0.20%. The sulfur content of the Category 2/3 waste rock placed directly in the East Pit is 0.21%. The distribution of sulfur content by mass for the Category 2/3 waste rock is shown in Figure 4-9.

The average sulfur content of the NorthMet ore is 0.608%. The total mass of ore produced under the Mine Plan is 225,280,000 short tons. The ore will be stored temporarily as necessary during mining in the Ore Surge Pile (OSP) before being transported to the Plant Site. The material in the OSP at any time will be a blend of ore from different active mining locations and is assumed to have nearly-constant average sulfur content equal to the average sulfur content in the Block Model.

The average sulfur content of the Duluth Complex Category 4 waste rock as a whole is 0.95%. Nearly 80% of this material will be placed directly in the East Pit and will not be stored in the Category 4 Waste Rock Stockpile. The Virginia Formation Category 4 waste rock, in contrast, will all be temporarily stored in the Category 4 Waste Rock Stockpile. The average sulfur content of the Virginia Formation Category 4 waste rock is 2.43%. The distribution of sulfur content by mass for the Category 4 waste rock (Duluth Complex and Virginia Formation separately) is shown in Figure 4-10. Note that this information is not used in the water quality modeling described in Section 8.1.1.1 but is included here for completeness.

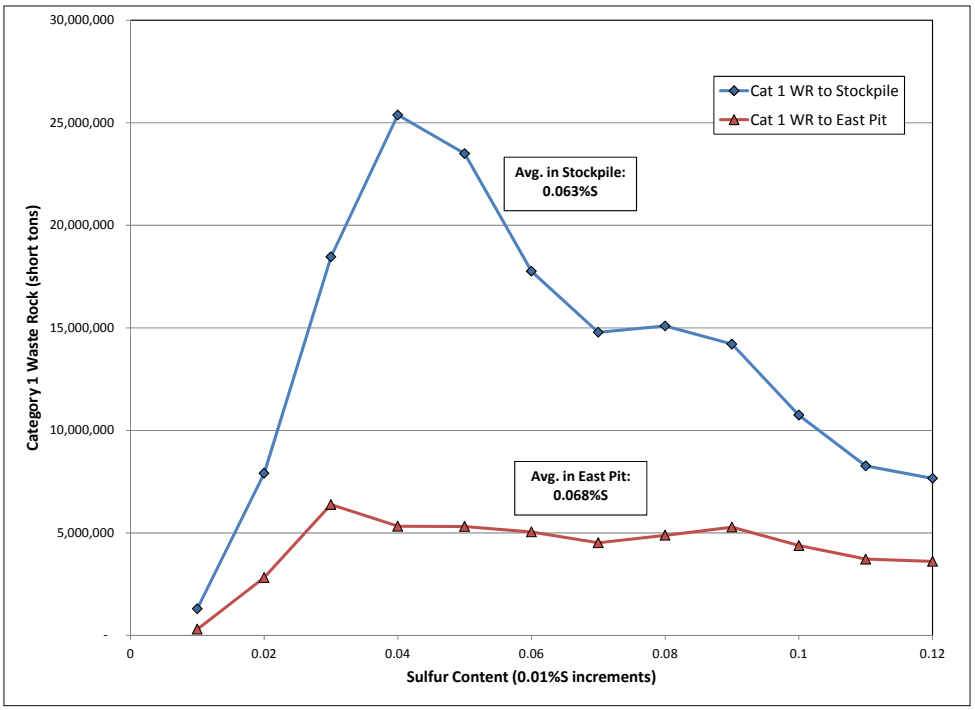

Figure 4-8 Sulfur Content Distribution for Category 1 Waste Rock

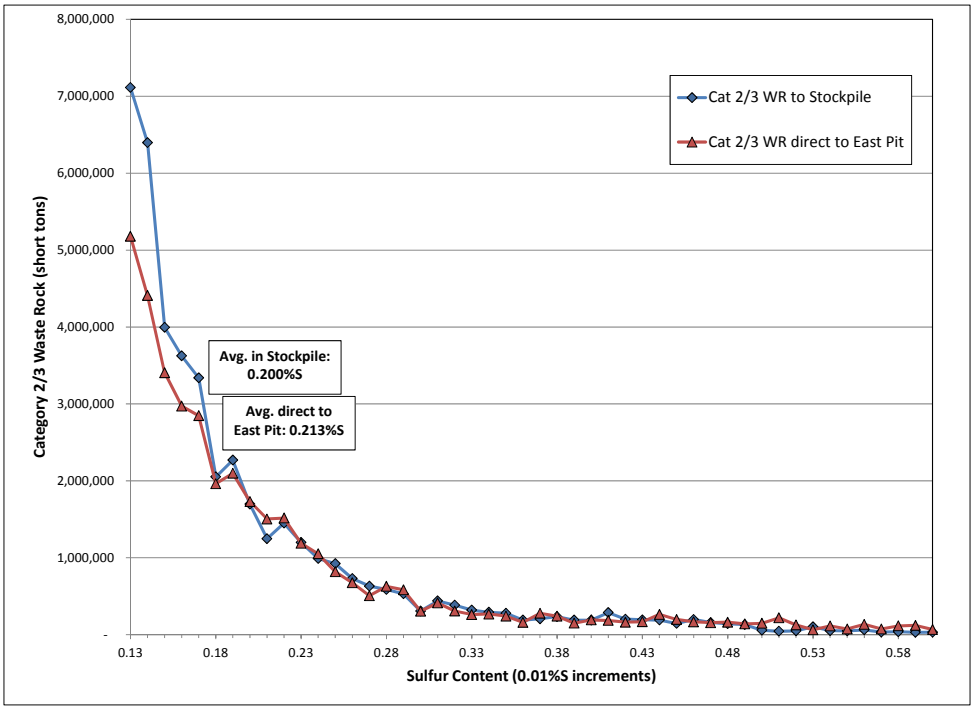

Figure 4-9 Sulfur Content Distribution for Category 2/3 Waste Rock

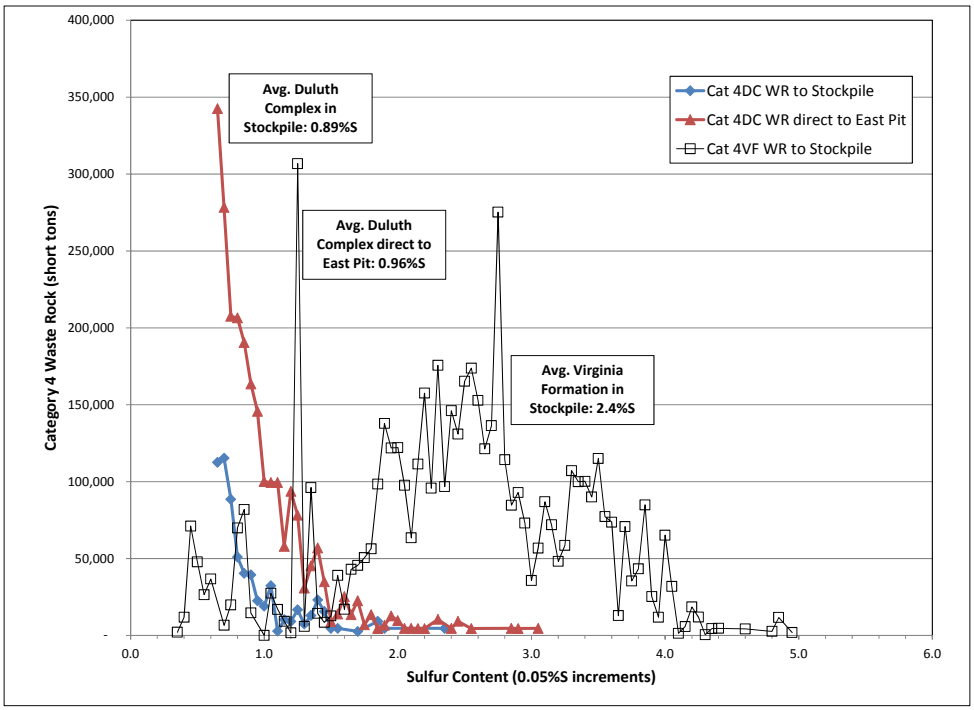

Figure 4-10 Sulfur Content Distribution for Category 4 Waste Rock

Data from the Block Model are also used to model the quantity and sulfur content of each category of wall rock. The relative proportion of each category of wall rock at 20-foot intervals was determined based on the Block Model, as shown in Figure 4-11 for the East Pit and in Figure 4-12 for the West Pit. The plan area (looking down) of each pit was determined from the pit geometry, with the pit walls blasted below the bedrock rim and the 20-foot pit rim set back at the top of bedrock considered separately (Section 9.2). The total plan area above a given elevation of pit wall is shown for both pits in Figure 4-13. These two data sets are used together to determine the exposed area of each category of wall rock at any elevation. The pit rim set back is assumed to have the same relative composition of the various rock categories as the main pit wall.

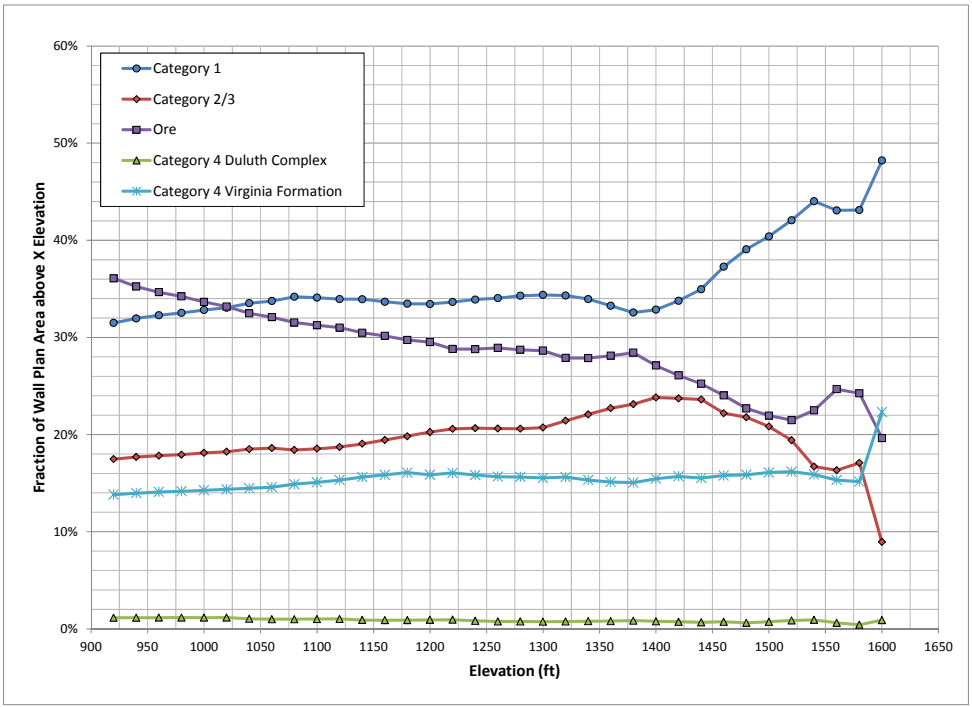

Figure 4-11 Wall Rock Category Fractions for the East Pit

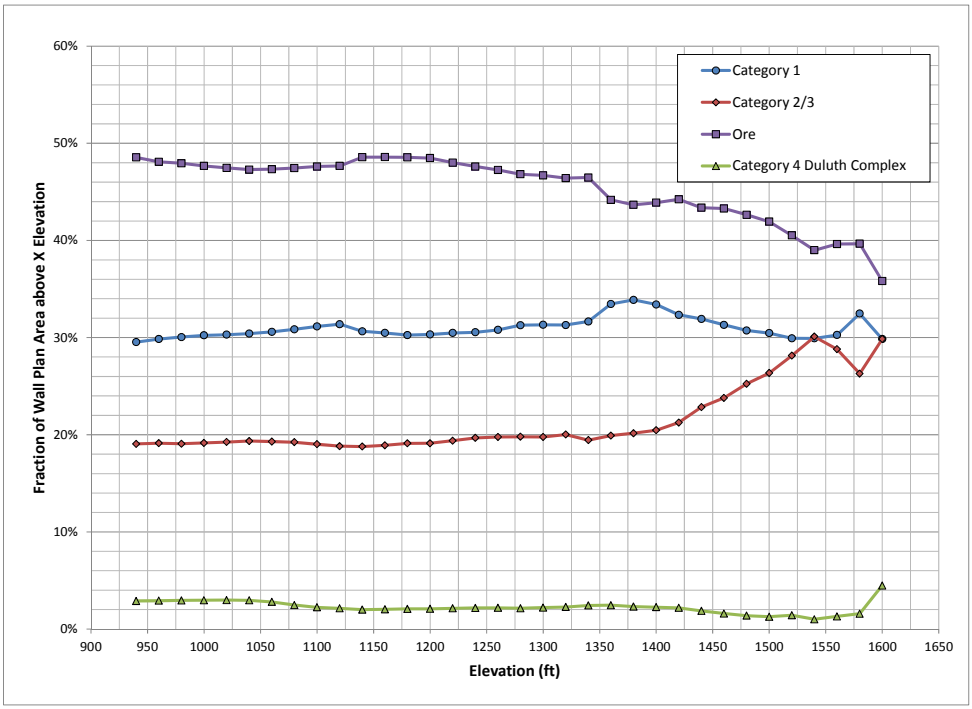

Figure 4-12 Wall Rock Category Fractions for the West Pit

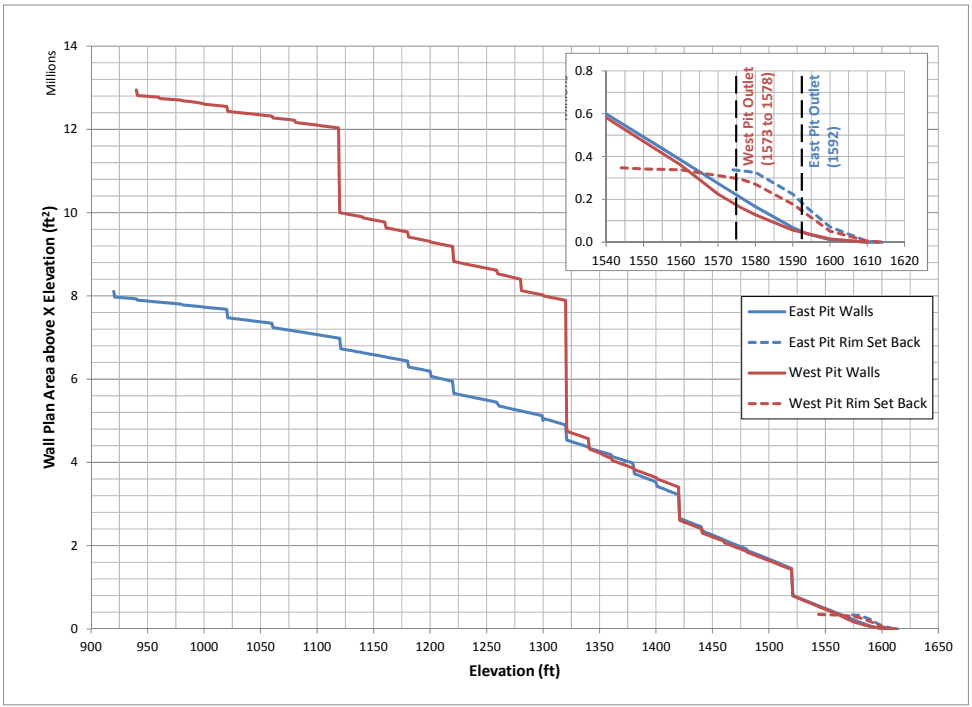

Figure 4-13 Total Pit Wall Rock Areas

Similar data from the Block Model were used to determine the area-weighted average sulfur content for all wall rock above a given elevation (by rock category). Because the pit walls are modeled as having a constant thickness of reactive wall rock (Section 9.2), the area-weighted average sulfur content is analogous to a mass-weighted average. This information is used as described in Section 8.1.1.1 to model sulfate release for the Category 1 and Category 2/3 waste rock and ore and is not used for modeling the other waste rock categories. Sulfur distribution data for all waste and wall rock categories are presented here for completeness.

Figure 4-14 shows the average sulfur content for the East Pit and West Pit for the Category 1 wall rock. Figure 4-15 shows the average sulfur content for the East Pit and West Pit for the Category 2/3 wall rock. Figure 4-16 shows the average sulfur content for the East Pit and West Pit for the Duluth Complex Category 4 wall rock. Figure 4-17 shows the average sulfur content for the East Pit for the Virginia Formation Category 4 wall rock (there is no Virginia Formation wall rock in the West Pit). Figure 4-18 shows the average sulfur content for the East Pit and West Pit for all wall rock classified as ore.

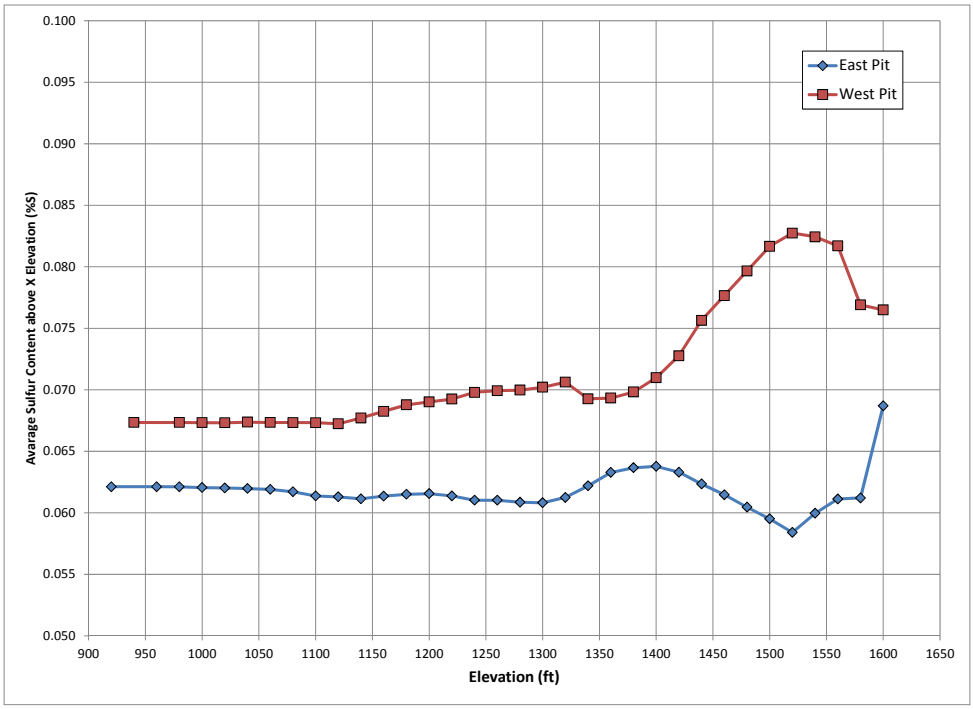

Figure 4-14 Sulfur Content with Depth for Category 1 Wall Rock

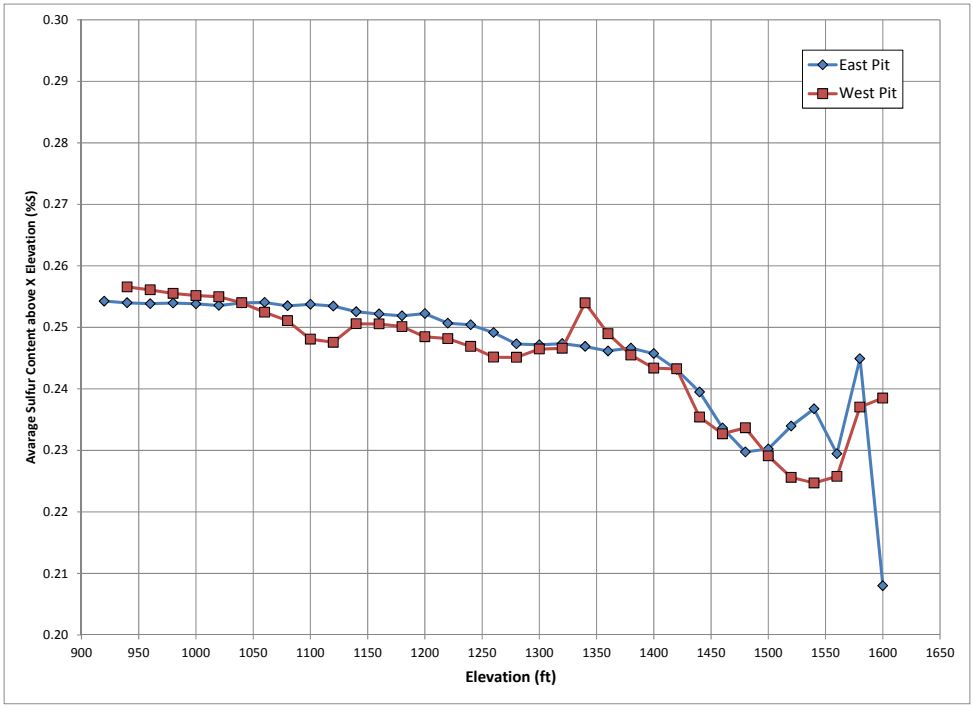

Figure 4-15 Sulfur Content with Depth for Category 2/3 Wall Rock

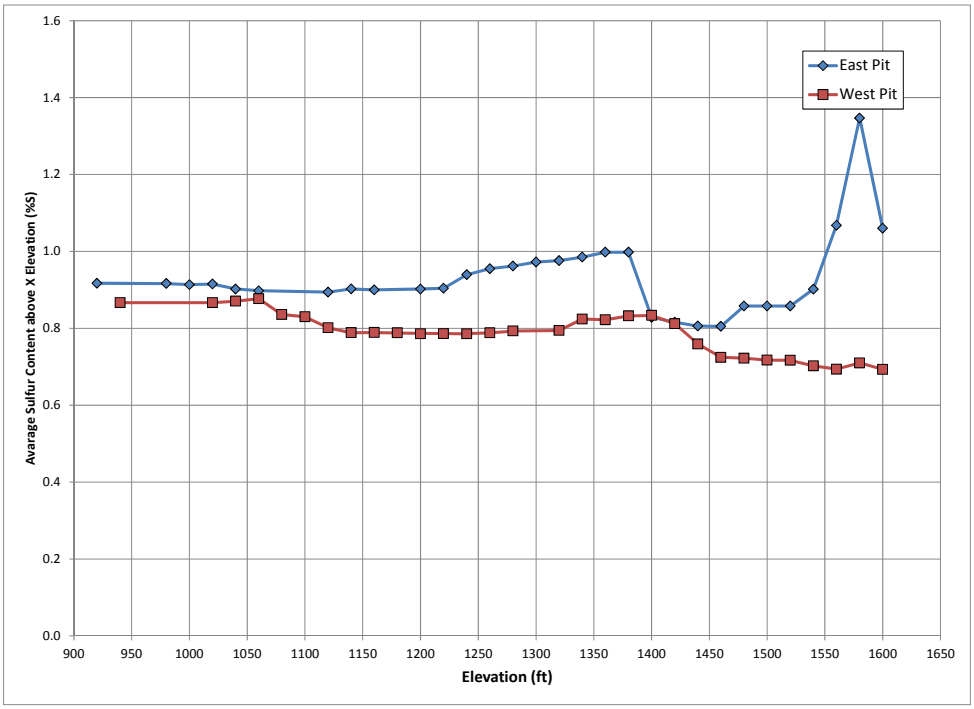

Figure 4-16 Sulfur Content with Depth for Duluth Complex Category 4 Wall Rock

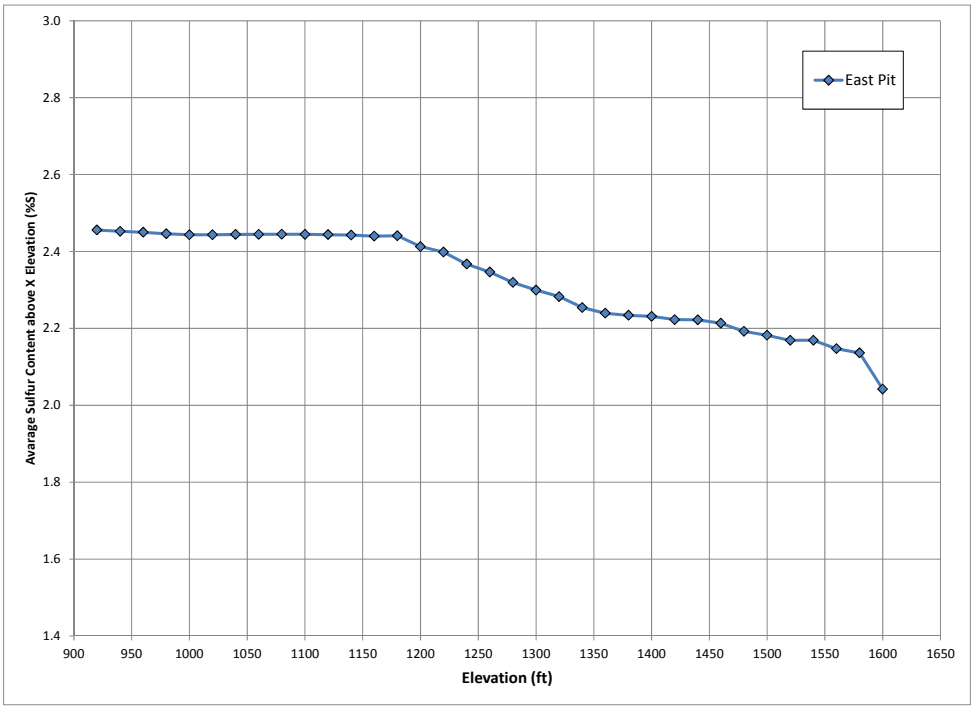

Figure 4-17 Sulfur Content with Depth for Virginia Formation Category 4 Wall Rock

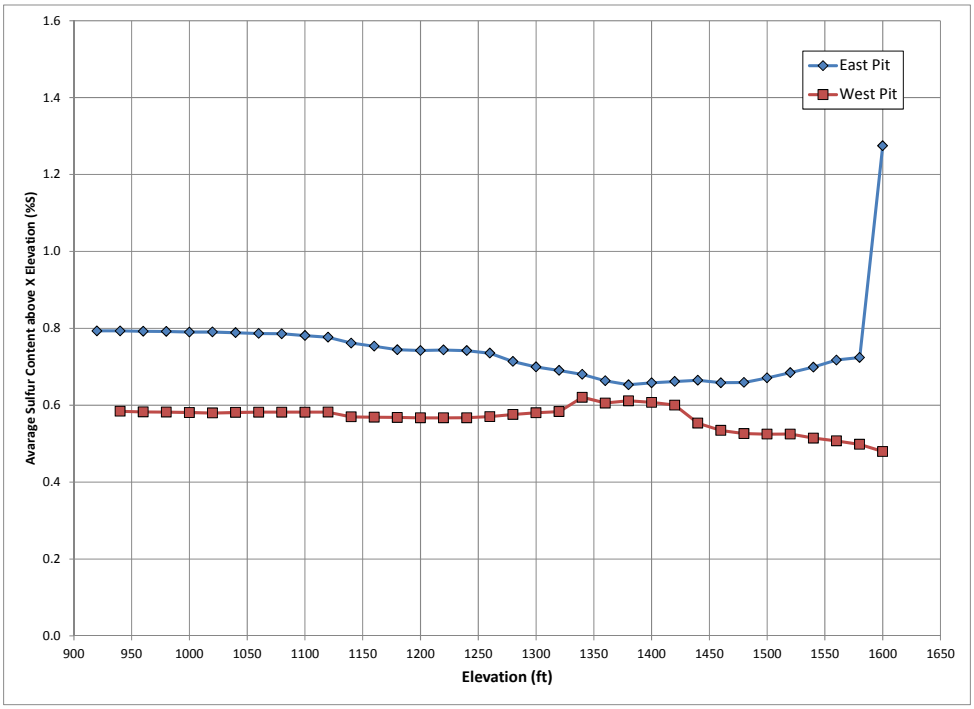

Figure 4-18 Sulfur Content with Depth for Ore Wall Rock

5.0 Flotation Tailings

The purpose of waste characterization for Flotation Tailings is to determine the geochemical characteristics of the Flotation Tailings generated by the Project and use those characteristics to develop a general Flotation Tailings management concept.

5.1 Reports

5.1.1 Initial Report

The initial report (Reference (7)) was submitted in 2007. A summary of the report is presented in the following paragraphs.

The results of initial chemical testing and long-term kinetic testing of 13 samples of Flotation Tailings were integrated with kinetic test work on tailings from processing ore with similar geology to develop an overall understanding of the tailings:

- bulk tailings will not generate acid
- bulk tailings will release metals

Tailings size fractions have different geochemical characteristics (Testing of different size fractions was initiated because PolyMet originally proposed to construct dams from the coarse and mid-size tailings. This is no longer being proposed.)

The kinetic testing results were used to develop average case reaction rate constants for the oxidation of sulfides in coarse and fine Flotation Tailings. The average reaction rate constants are presented in Table 5-1. Average case release of other constituents was determined based on the kinetic testing results in terms of the ratio between molar release of sulfate (from sulfide oxidation) and molar release of other constituents. The average release ratios are presented in Table 5-2.

Table 5-1 2007 Average Reaction Rate Constants from Humidity Cell Tests

Tailings Type	Reaction Rate Constant (1/s)
Coarse	3.38×10^{-8}
Fine	6.50×10^{-8}

Table 5-2 2007 Average Release Ratios from Humidity Cell Tests (mol/mol SO_4)

Constituent	Coarse Tailings	Fine Tailings
Antimony (Sb)	4.0×10^{-5}	4.0×10^{-5}
Arsenic (As)	2.0×10^{-4}	4.2×10^{-4}
Beryllium (Be)	1.4×10^{-4}	1.3×10^{-4}
Boron (B)	6.0×10^{-4}	8.5×10^{-4}
Cadmium (Cd)	2.3×10^{-6}	2.1×10^{-6}
Calcium (Ca)	1.8×10^{-1}	4.1×10^{-2}
Cobalt (Co)	3.9×10^{-5}	1.3×10^{-5}
Copper (Cu)	1.5×10^{-4}	1.4×10^{-4}
Lead (Pb)	1.6×10^{-6}	1.4×10^{-6}
Magnesium (Mg)	3.8×10^{-1}	4.5×10^{-1}
Nickel (Ni)	6.6×10^{-4}	6.5×10^{-5}
Potassium (K)	3.3×10^{-1}	2.8×10^{-1}
Selenium (Se)	1.6×10^{-5}	1.5×10^{-5}
Silver (Ag)	3.0×10^{-6}	2.7×10^{-6}
Sodium (Na)	1.1×10^{-1}	2.3×10^{-1}
Thallium (Tl)	6.3×10^{-7}	5.8×10^{-7}
Zinc (Zn)	1.2×10^{-3}	6.4×10^{-4}

5.1.2 2009 Update

A general geochemical update report (Reference (4) Section 4) was submitted in 2009. A summary of the report is presented in the following paragraphs.

The results of ongoing kinetic testing were used to reassess and update the average release ratios for constituents from the coarse and fine tailings. The updated data indicated no change in reaction rate constants for either the coarse or fine tailings. The Mine Plan was updated to provide for deposition of Flotation Tailings as bulk tailings across the entire basin, rather than as segregated depositional areas. The reaction rate constant for bulk tailings was summarized in a modeling assumptions memorandum (Reference (8)) along with the average release ratios for the bulk tailings. Updated reaction rate constants for coarse, fine and bulk tailings are presented in Table 5-3 and updated average release ratios are presented in Table 5-4.

Table 5-3 2009 Average Reaction Rate Constants from Humidity Cell Tests

Tailings Type	Reaction Rate Constant (1/s)
Coarse(1)	3.38×10^{-8}
Fine(1)	6.50×10^{-8}
Bulk	3.82×10^{-8}

(1) 2009 modeling assumed only bulk tailings. Values for coarse and fine tailings are shown for comparison only.

Table 5-4 2009 Average Release Ratios from Humidity Cell Tests (mol/mol SO_4)

Constituent	Coarse Tailings(1)	Fine Tailings(1)	Bulk Tailings
Aluminum (Al)	1.8×10^{-2}	5.2×10^{-2}	3.5×10^{-2}
Antimony (Sb)	2.5×10^{-5}	7.2×10^{-5}	9.8×10^{-6}
Arsenic (As)	1.5×10^{-4}	6.1×10^{-4}	2.9×10^{-4}
Beryllium (Be)	1.3×10^{-4}	2.1×10^{-4}	1.6×10^{-4}
Boron (B)	1.8×10^{-3}	4.5×10^{-3}	2.0×10^{-3}
Cadmium (Cd)	2.5×10^{-6}	3.7×10^{-6}	2.6×10^{-6}
Calcium (Ca)	1.8×10^{-1}	4.1×10^{-2}	6.7×10^{-1}
Cobalt (Co)	1.3×10^{-4}	1.6×10^{-5}	1.3×10^{-5}
Copper (Cu)	2.1×10^{-4}	4.1×10^{-4}	1.4×10^{-4}
Lead (Pb)	1.4×10^{-6}	2.9×10^{-6}	2.0×10^{-6}
Magnesium (Mg)	3.8×10^{-1}	4.5×10^{-1}	5.6×10^{-1}
Nickel (Ni)	2.6×10^{-3}	1.2×10^{-4}	5.0×10^{-5}
Potassium (K)	3.3×10^{-1}	2.8×10^{-1}	4.5×10^{-1}
Selenium (Se)	1.8×10^{-5}	2.7×10^{-5}	1.8×10^{-5}
Silver (Ag)	2.7×10^{-6}	4.4×10^{-6}	3.4×10^{-6}
Sodium (Na)	1.0×10^{-1}	3.0×10^{-1}	2.2×10^{-1}
Thallium (Tl)	6.3×10^{-7}	9.3×10^{-7}	7.1×10^{-7}
Zinc (Zn)	4.6×10^{-4}	2.4×10^{-4}	2.1×10^{-4}

(1) 2009 modeling assumed only bulk tailings. Values for coarse and fine tailings are shown for comparison only.

5.1.3 2011 Update

5.1.3.1 Depositional Study

For modeling purposes, it is important to know whether the delta that is formed as tailings are deposited in the FTB can be treated as one "bulk" zone of tailings or needs to be further refined into multiple zones of different grain size fractions. The University of Minnesota's St. Anthony Falls Laboratory (SAFL) performed experiments to quantify the potential for segregation in the tailings delta and to determine some of the important hydraulic properties of the deposited tailings. SAFL performed two phases of experiments. Phase I was a flume experiment, designed to use field-scale flow conditions to evaluate the potential for debris flow versus channelized or sheet flow as the delta is formed. Phase II was a 2D lab scale experiment designed to answer the questions about segregation and hydraulic properties. A report was prepared by SAFL and submitted in March, 2011 (included as Attachment E).

The Phase I experiment clearly showed that the behavior of the delta is one of a fluvial system characterized by channelized or sheet flow. In other words, debris or "mud" flow will not occur. The solids were transported throughout the delta as bedload and suspended load. The tendency of the tailings slurry discharge was to channelize and form braids, with bars and bedforms that developed in the active channels. Therefore, the deposition patterns are characteristic of those observed in other tailings basins and deltaic systems in general, where the combined fluvial processes of erosion, transport and deposition at the channel and larger scales (not at the individual grain size scale) determine the configuration and characteristics of the delta.

The Phase II experiment was scaled down from field to laboratory conditions to provide similarity in Froude number (i.e., the ratio of inertial to gravity forces), general sediment-transport regime and the aspect ratio. Extensive field and laboratory research has shown that the aspect ratio (here defined as normal flow depth to radial width of the delta) is a simple but very robust predictor of channel morphology in deltaic systems. As described in the report and due to the method of scaling (aspect ratio), the laboratory delta is expected to grow and maintain its surface by the same mechanisms as the field scale delta. The fines retention and the degree of grain size sorting seen in the experiment should be similar to that in the field scale delta.

Because one of the objectives of the Phase II experiment was to determine the potential for segregation of coarse and fine tailings in the delta, conditions were created to maximize segregation. High flows were used which would tend to transport more material to the pond area; during the falling phase of the Phase II experiment, the pool elevation was slowly decreased to promote delivery of tailings to the shoreline position and minimize deposition except for the coarsest material. Despite the attempt to maximize segregation, one of the major and firm conclusions from the Phase II experiment is that there will be a minimum of 30% (by mass) fines (passing mesh #200; particle sizes smaller than 74 micron) in the delta.

Even under the most extreme plausible transport conditions, it was difficult to generate a deposit with less than 30% fines content.

It is important to note that the individual samples taken from the delta for analysis are just that; individual samples at specific locations. These results are useful for estimating the approximate degree of sorting in the delta and characterizing the tailings at any one location. However, they are not necessarily appropriate (as individual samples) for characterizing the tailings delta as a whole.

Barr used the SAFL report, supplemented with information from operational tailings basins, to reach important conclusions regarding data inputs that are necessary for the probabilistic water quality modeling.

1) Due to naturally developing slopes in the Phase I experiment, the 1% slope used for the field-scale design and therefore the water quality modeling, is reasonable.

2) It can be assumed that the tailings delta (portion of the tailings deposited above the water pond) is one zone. Although downstream fining is evident in the experiments and it has been documented in operational examples, the expected range of variation for the fines fraction (i.e., passing mesh #200) for Flotation Tailings (more than 50% is passing mesh #200) does not justify modeling two or more zones.

3) Of the tailings discharged aerially to form the NorthMet delta, the tailings delta will contain 100% of the coarse fraction (i.e., above mesh #200).

4) In any given year, the Flotation Tailings delta will on average have a fines fraction characterized by P5 = 30%, P50 = 35% and P95 = 40% throughout the entire delta area. A normal distribution with mean and standard deviation of 0.3500 and 0.0304 respectively is used to describe the uncertainty in the percent fines in the delta as it is formed (Figure 5-1).

5) The average porosity of the tailings throughout the NorthMet delta is primarily a function of fluvial mixing rather than grain size distributions and is characterized by P5 = 0.38, P50 = 0.41 and P95 = 0.45. A triangular distribution with lower, mode and upper values of 0.3668, 0.4012 and 0.4685 respectively is used to describe the uncertainty in the NorthMet delta porosity (Figure 5-2).

6) The average porosity of the tailings under the proposed pond is primarily a function of turbidity current and settling and is characterized by P5 = 0.43, P50 = 0.52 and P95 = 0.56. A triangular distribution with lower, mode and upper values of 0.4049, 0.5602 and 0.5696 respectively is used to describe the uncertainty in the tailings porosity under the pond (Figure 5-3).

7) The annual average solid fraction of the slurry discharged from the Beneficiation Plant will be a mixture of coarse tailings and fine tailings. This coarse tailings fraction (by mass) in the mixture will be characterized by P5 = 0.35, P50 = 0.38 and P95 = 0.41. Note that this distribution does not reflect what is stated in the SAFL report but is slightly lower (Attachment E). This is because the distribution parameters represent the coarse fraction from the most recent pilot plant testing (the grain size distribution results from Soil Engineering Testing, Inc., in April of 2011, Job number 7917). A normal distribution with mean and standard deviation of 0.3800 and 0.0182 respectively is used to describe the uncertainty in the percent coarse in the feed material (Figure 5-4).

The SAFL experiment was conducted using tailings from the initial pilot plant run (2005) and the report states that the fraction (by mass) of feed material that is coarse tailings (retained on the #200 mesh) is about 0.41 (ranging from 0.35 to 0.48). The most recent pilot plant testing, which was used to define the input distribution for the feed material, suggests that the mean is around 0.38. The fraction from the newest grain size distributions falls within the range reported in the SAFL report. Therefore, there is not a significant difference in the feed material between the initial pilot plant run and the recent pilot plant runs.

As outlined above in the SAFL report conclusions, the input distributions that define the porosity of the Flotation Tailings beach and the Flotation Tailings under the pond are primarily functions of fluvial mixing and settling rather than the grain size distribution. Therefore, the minor difference in the coarse fraction of feed material between the initial pilot plant run and the recent pilot plant runs is not anticipated to have a significant effect on these input parameters.

The input distribution that defines the fraction of fines retained in the delta could be a function of the make-up of the feed material. Lowering the coarse tailings fraction could cause the slope to be shallower, increasing the delta length and the fraction of fines in the delta. However, the difference in the coarse fraction of feed material between the initial pilot plant run and the recent pilot plant runs is very small and it is anticipated that this difference does not have a significant effect on this input parameter.

The conclusions drawn and listed above help define the other important hydraulic parameters needed to complete the water quality modeling. These other parameters are discussed in Section 10.2.1 of this document.

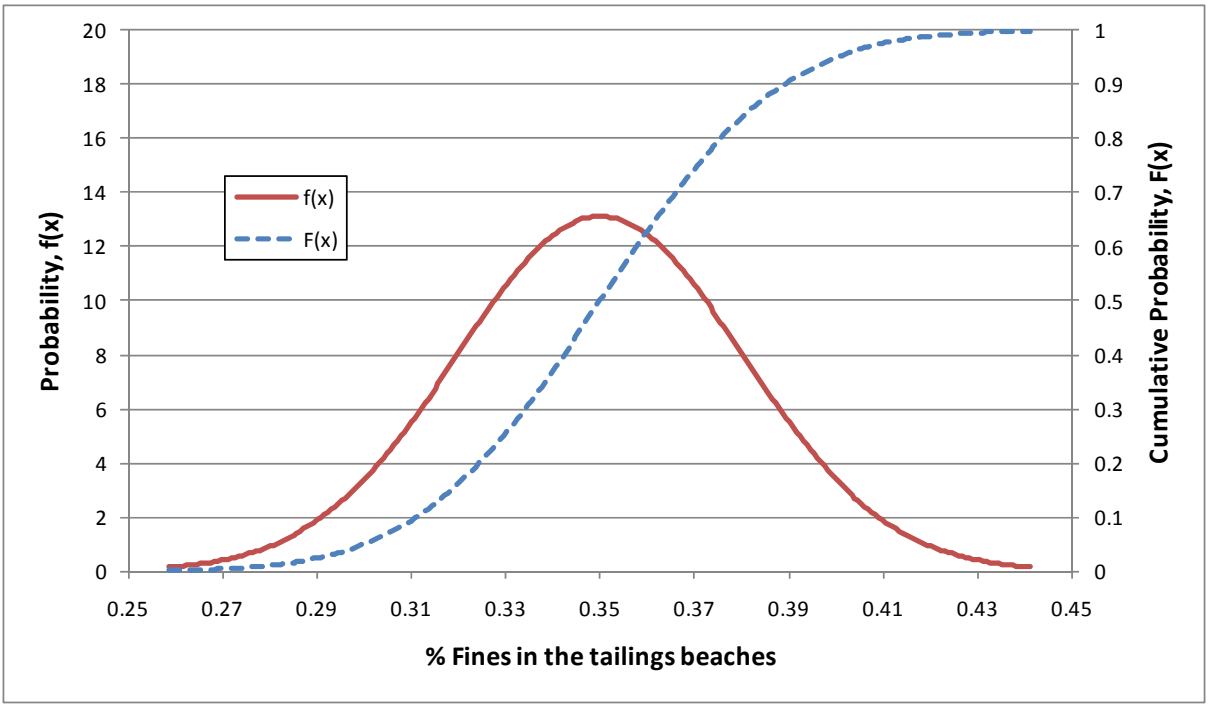

Figure 5-1 Distribution for the Percent Fines in the Tailings Beaches

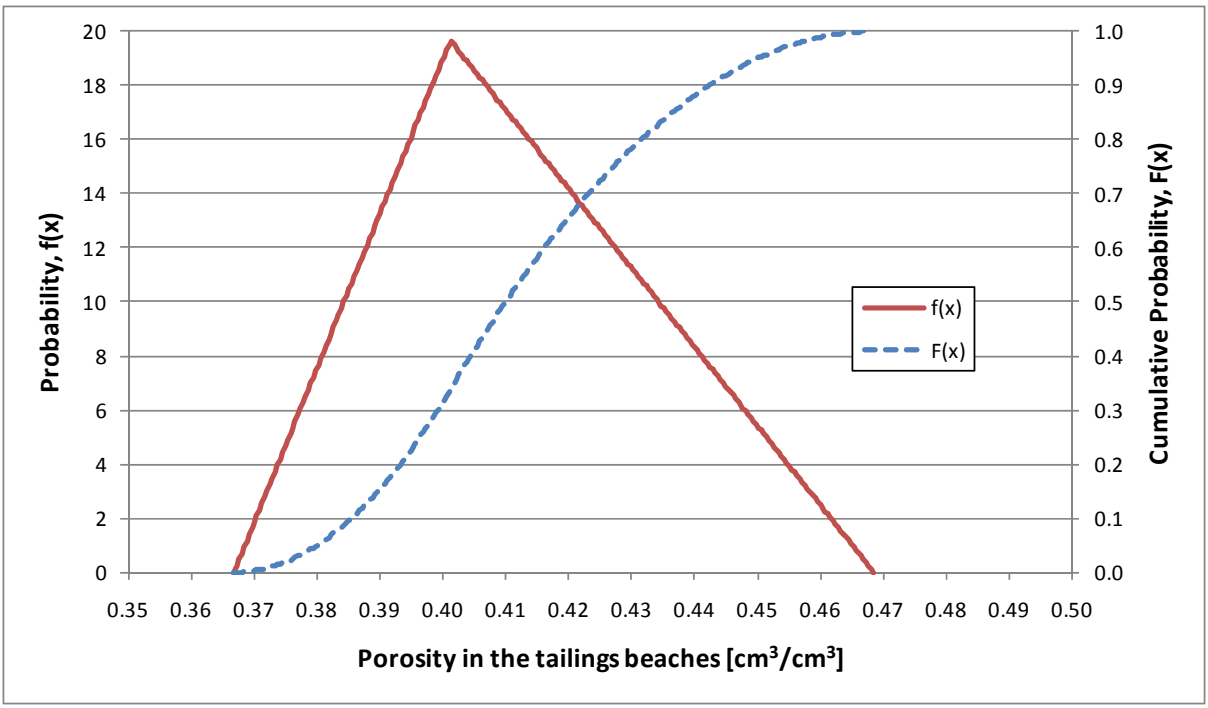

Figure 5-2 Distribution for the Porosity in the Tailings Beaches

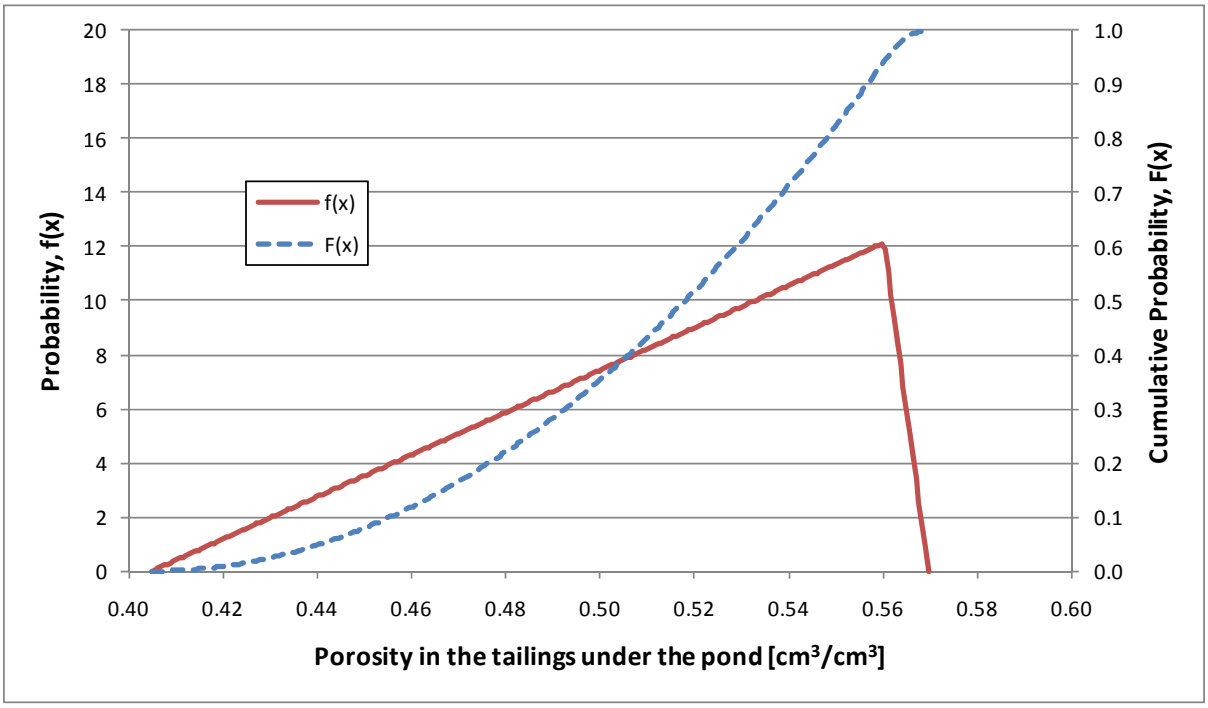

Figure 5-3 Distribution for the Porosity under the Pond

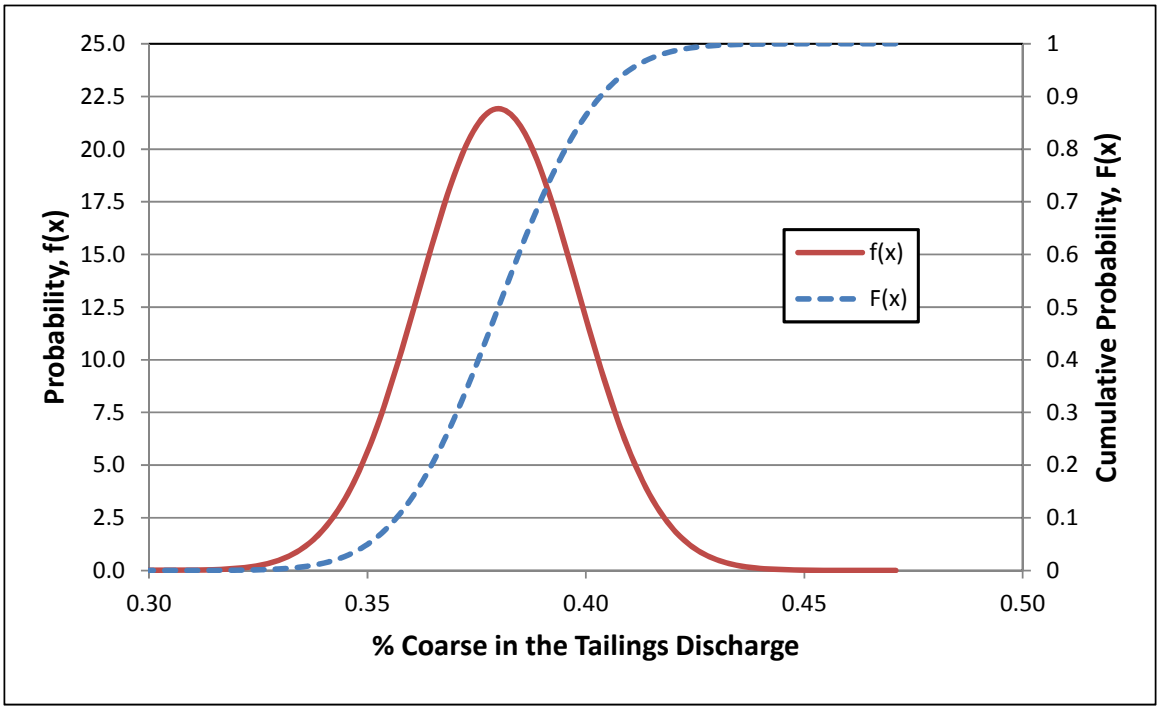

Figure 5-4 Distribution for the Percent Coarse in the Feed Material

5.1.3.2 2011 Geochemical Update

In order to accommodate the change in overall water quality modeling concept from deterministic to probabilistic, the results of ongoing kinetic testing were reevaluated and a new modeling approach was developed. This updated approach for the Flotation Tailings is similar to that developed for the waste rock, as discussed in Section 4.1.3.1 and presented in more detail in Attachment A. The application of this modeling approach is discussed in detail in Sections 10.1 and 10.4.

Previous interpretation of the humidity cell test results assumed that the rates of constituent leaching indicated by the humidity cells were free of solubility limitations due to the high liquid-to-solid ratios used in laboratory kinetic tests. Data from the humidity cells demonstrate that this is not the case for Flotation Tailings: total nickel release (leaching), for example, can increase by orders of magnitude even as oxidation and weathering of the sources of nickel decrease. A new approach was needed to account for the influence of solubility limitations on the humidity cell leaching rates.

Chemical constituents, including, major mineral components and trace metals, are released from host primary minerals as these minerals weather. The Project data indicated that after constituents are released from primary minerals, the concentrations of some of the chemical constituents in humidity cell tests were limited by solubility relationships with secondary minerals, while others were not. The release rates of constituents that appeared to be subject to solubility limitations were determined by identifying the likely mineral sources for each constituent using Project data and information from literature sources. The release rates of constituents that were potentially controlled by solubility relationships were identified by correlations with constituents in the same primary mineral host that were not controlled by solubility limitations.

This approach to estimating release rates relies on empirical analysis of the humidity cell data (for both tailings and waste rock) and data for metal content in the whole tailings samples (aqua regia digestion) and individual minerals (microprobe analysis of the minerals found in NorthMet waste rock and therefore tailings). The rates and relationships determined for the coarse and fine Flotation Tailings and the bulk LTVSMC tailings are therefore specific to this data set. The specific humidity cells used in this analysis, their tailings type, sulfur content and duration are shown in Large Table 4. The proposed approach for developing release rates for each constituent is outlined in Large Table 5 and described further in Section 10.1.

5.1.3.3 2011 Humidity Cell Trends Update

A geochemical update report specific to the Flotation Tailings humidity cells (included as Attachment F) was submitted in 2011. A summary is presented in the following paragraphs.

In the time since kinetic testing of pilot plant samples was initiated (tests summarized in the 2009 update report, Section 5.1.2), PolyMet has made refinements to the Beneficiation Plant flowsheet. The main refinements are an *Added Regrind* capacity and the ability to operate in *Split Cleaner Mode*, which are both included in the current Project Description (Reference (9)). The objective of these refinements is to optimize the flotation process and to allow for operational flexibility to produce a variety of concentrate products. With each process refinement PolyMet has performed pilot plant testing and has submitted the resulting tailings samples for the same geochemical testing. The humidity cells containing tailings produced by pilot plants in 2008 and 2009 are identified by the tailings source of "Pilot Plant 2", "Pilot Plant 3", and "SCAV" in Large Table 4.

The results of ongoing kinetic testing were used to reassess the assumed sulfate oxidation rates and pH conditions used in modeling the Flotation Tailings. In general, the testwork performed at various times and on tailings from various pilot plants shows similar relationships between sulfur content and sulfate release. All tests showed that sulfate release declined over time along with sulfur content, indicating that the decline in sulfate release may be due to reduction in available oxidation rates. This implies the sulfide oxidation reaction is non-zero order and that the use of initial oxidation rates in modeling remains conservative.

Some differences in pH trends have been observed between the initial (2005 & 2006) tests and subsequent (2008 & 2009) tests. The initial dataset showed that pH decreased to a level at which nickel released due to increased solubility (below about pH 7). This trend has not been observed to the same degree in more recent testwork, suggesting that there is more buffering capacity in the 2008 and 2009 tailings samples (stable pH > 7). Use of the first dataset to set the sulfur criterion for tailings management appears to be conservative.

PolyMet also initiated geochemical kinetic testing of composite samples of LTVSMC tailings in 2010, using the same testing methods as for the flotation tailings samples. The results from this testing show that LTVSMC tailings chemistry is dominated by buffering from the dissolution of carbonate minerals, with stable pH values between 7.3 and 8.1. Release rates for the primary constituents of concern have been stable since approximately 25 weeks of testing, providing good estimates of the average release rates for use in water quality modeling.

5.1.4 2012 and 2014 Humidity Cell Trends Updates

Analyses of the ongoing tailings humidity cell data were included in the updates submitted in 2012 (Attachment C, including data through February 2012), and 2014 (Attachment D, including data through August 2013). A summary of these documents is presented in the following paragraphs.

- No tests have generated pH lower than 6. .

- Sulfur leaching rates have declined as sulfur is depleted, and differences in sulfur content explain the observed differences in sulfide oxidation rates and the degree to which pH is depressed.
- Samples with lower pH had accelerated leaching of nickel and cobalt, though only for the Pilot Plant 1 samples.
- Samples of LTVSMC tailings have shown stable leachate chemistry with slightly basic pH.

5.2 Flotation Tailings Management Concept

The concept for management of Flotation Tailings is:

- Process the ore using a bulk sulfide flotation process to minimize the amount of sulfides reporting to the FTB. This includes the use of copper-sulfate in the flotation process to promote additional sulfide flotation.

- Deposit tailings as a bulk tailing to avoid the lower saturation and higher release rates associated with the coarse-sized fraction.

- Maintain a pond in closure to minimize oxidation of Flotation Tailings. In closure, the beaches will cover about 425 acres, and the pond (including wetland area) will cover about 900 acres (Reference (10)).

- Amend the surface of the FTB dams and beaches, as well as the bed of the pond in closure, with bentonite to reduce oxygen penetration and minimize oxidation of Flotation Tailings.

- Install engineered systems at the toe of the FTB dams to collect water that has contacted the tailings and prevent seepage from migrating into the surrounding surficial materials.

6.0 Hydrometallurgical Residue

The purpose of waste characterization for hydrometallurgical residue is to determine the geochemical characteristics of the hydrometallurgical residue generated by the Project and use those characteristics to develop a general hydrometallurgical residue management concept.

6.1 Reports

6.1.1 Initial Report

The initial report (Reference (11)) was submitted in 2007. A summary of the report is presented in the following paragraphs.

Discrete and combined samples of the hydrometallurgical residues from pilot-testing have been tested for bulk solids characteristics, mineralogy and leaching characteristics. Combined samples with and without the gypsum filter residue were tested to allow for possible marketing of the gypsum filter residue.

Four of the residues (leach, solution neutralization, iron/aluminum and raffinate neutralization) were acidic. The leach residue consists dominantly of natrojarosite and hematite along with gypsum and residual plagioclase. The other three residues consist mainly of gypsum (96 to 99.8%). The magnesium residue was not acidic. It contains mostly gypsum (77%) but also 22% brucite (magnesium hydroxide), which is a source of alkalinity. Due to the presence of the magnesium residue, the combined residues were nonacidic and dominated by gypsum. Theoretically, the natrojarosite in the leach residues could consume buffering capacity resulting in acidic conditions of the combined residues in the future.

Humidity cell tests on residues showed an initial rapid flush of acidity and metals as process water was rinsed from the residues. As the tests proceeded, the individual leachates remained acidic but leaching of metals and acidity decreased reflecting dissolution of the residue. Sulfate concentrations remained elevated due to ongoing dissolution of gypsum.

Table 6-1 provides the maximum concentrations observed in the humidity cell testing on the combined residue, typically occurring in the initial test sample. These values were reported in the initial (Reference (11)) and in an expanded list of solutes prepared as an appendix to the water quality modeling document (Reference (12)).

Table 6-1 Maximum Observed Hydrometallurgical Residue Humidity Cell Concentrations

Constituent	Concentration (mg/L)
pH Range	6.6 – 8.5
Chloride	204
Sulfate	7,347
Aluminum (Al)	0.18(1)
Antimony (Sb)	0.004
Arsenic (As)	0.004(2)
Barium (Ba)	0.005
Beryllium (Be)	0.002
Boron (B)	0.14
Cadmium (Cd)	0.0004
Calcium (Ca)	626
Chromium (Cr)	0.05
Cobalt (Co)	0.005
Copper (Cu)	0.015
Iron (Fe)	0.4
Lead (Pb)	0.0005
Manganese (Mn)	0.0023
Magnesium (Mg)	1,040
Molybdenum (Mo)	0.14
Nickel (Ni)	0.098
Selenium (Se)	0.054
Silver (Ag)	0.0005
Sodium (Na)	1,250
Thallium (Tl)	0.0002
Vanadium (V)	0.002
Zinc (Zn)	0.01

(1) Maximum Al concentration from shake flask testing (not humidity cells)
(2) As concentration shown is for initial leachates for which maxima occurred for other constituents

6.1.2 2009 Update

A general geochemical update report (Reference (4) Section 5) was submitted in 2009. A summary of the report is presented in the following paragraphs.

The results of ongoing kinetic testing showed that leachate chemistry, including pH, is relatively stable until gypsum is completely leached, which results in decreasing sulfate leaching and changes to trace element leaching. Because the number of pore water volumes flushed in the ongoing tests far exceeds the pore water displacement that will occur in the lined and covered residue cells, these long-term effects likely do not represent site conditions. The initial maximum leachate concentrations and pH range shown in Table 6-1 will continue to be used as the basis for water quality modeling.

6.1.3 2011 Design Update

The Hydrometallurgical Residue Facility (HRF) liner design has changed since the modeling for the DEIS. Currently, the proposed design is a double liner with leak detection. Therefore, it is expected that essentially no leakage will occur, or it will be detected and corrected. According to the Groundwater Impact Assessment Planning summary memo, it is assumed that the HRF will have negligible leakage and there is no compelling need to model the leakage from this source (Reference (13)). At final closure, the water in the HRF will be decanted and pumped to the FTB. Any water captured by the leakage system will also be pumped to the FTB. However, all of this water pumped to the FTB will be treated first at the WWTP at the Plant Site such that the FTB will not exceed water quality standards at the compliance points (Reference (14)).

Because the HRF can be assumed (for modeling purposes) to have no leakage and the water pumped from the HRF to the FTB during closure will be treated, there is no reason to model the chemical loading from the HRF. Therefore, a geochemical characterization of the hydrometallurgical residue was not necessary for the modeling of the SDEIS. Geochemical characterization of the hydrometallurgical residue was conducted, and the results of these analyses are described in the HRF Residue Management Plan (Reference (14)).

6.1.4 2013 Water Quality Update

As part of the design for the Plant Site WWTP, the expected quality of the drainage from the HRF was compared to the water quality of the other modeled inflows to the WWTP. The humidity cell data from the initial report (Reference (11)) were used to determine the average concentrations observed in the “combined (with $CuSO_4$)” residue test cell. This is the most likely representation of the waste material that will be placed in the HRF during operations. Average values for the constituents of interest were calculated for both data sets (residue and other inflows to the WWTP) after setting non-detect values to half the detection limit. The average values from the humidity cells likely approach the long-term steady state values for most of the constituents of interest. The average concentrations are shown in Table 6-2. This

information is used for the design of the WWTP, not for modeling the chemical loading from the HRF.

Table 6-2 Average Observed Hydrometallurgical Residue Humidity Cell Concentrations

Constituent	Concentration (mg/L)[1]
Alkalinity (as $CaCO_3$)	27
Fluoride	0.2
Chloride	4
Sulfate	1600
Aluminum (Al)	0.012
Arsenic (As)	0.010
Barium (Ba)	0.002
Boron (B)	0.013
Cadmium (Cd)	0.002
Calcium (Ca)	580
Cobalt (Co)	0.005
Copper (Cu)	0.006
Iron (Fe)	0.016
Lead (Pb)	0.002
Manganese (Mn)	0.001
Magnesium (Mg)	32
Nickel (Ni)	0.008
Potassium (K)	0.3
Selenium (Se)	0.011
Sodium (Na)	20
Thallium (Tl)	0.00002
Vanadium (V)	<0.00022
Zinc (Zn)	0.003

(1) Average observed concentration in the combined residue with $CuSO_4$ humidity cell

6.2 Hydrometallurgical Residue Management Concept

The concept for management of hydrometallurgical residue is:

- add lime or limestone to the residue to avoid long-term acid generation (to be confirmed once hydrometallurgical plant operations are underway)
- place the residue in a double-lined facility with leak detection
- treat all of the water pumped and collected during dewatering
- cover the facility in reclamation to minimize water infiltration so that it is negligible

7.0 Geochemical Parameters – Overburden

This section covers geochemical parameters relating to overburden that are used in water quality modeling.

7.1 Leachate Water Quality – Unsaturated Overburden

As described in Section 3.1.1 and Section 3.1.2 two different sampling campaigns and methods of analysis have yielded a consistent understanding of the geochemical behavior of the Unsaturated Overburden at the Mine Site. This material has been above the water table and in oxidizing conditions for millennia and has little potential for ongoing significant release of constituents of concern.

Unlike the modeling of waste rock described in Section 8.0, the quality of the water leaching from stockpiled Unsaturated Overburden is expected to be constant over time and is not a function of the quantity of stockpiled material. The leachate chemistry from a portion of the Overburden Storage and Laydown Area and other areas where Unsaturated Overburden is used as construction material, is represented by the results of the 2008 MWMP testing (Table 3-1). The results of this testing were used to develop probability distributions of leachate chemistry for constituents from Unsaturated Overburden. Uniform distributions were defined from the maximum and minimum concentrations reported in Reference (1). The ranges of the leachate chemistry are presented in Table 7-1.

7.2 Leachate Water Quality – Peat

Similar to the Unsaturated Overburden, the quality of the water leaching from Peat stockpiled in a portion of the Overburden Storage and Laydown Area is modeled using the results of the 2008 MWMP testing (Table 3-1). The results of this testing were used to develop probability distributions of leachate chemistry for constituents from Peat. Uniform distributions were defined from the maximum and minimum concentrations reported in Reference (1). The ranges of the leachate chemistry are presented in Table 7-1.

Although peat has the potential to release mercury in drainage water when stockpiled, mercury was not included in the probabilistic model for several reasons: (1) there is a lack of comprehensive low level mercury data for all mine site wastes and water streams; (2) the primary source of mercury in northeastern Minnesota, including the Mine Site, is aerial deposition; and (3) experience at other mining operations in the region shows that mine pit water is generally below background concentrations with respect to mercury. Drainage from the peat portion of the OSLA will be monitored for mercury during operations and mitigation measures will be taken if necessary (Reference (15)).

Table 7-1 Distributions for Unsaturated Overburden and Peat Leachate

Constituent	Units	Unsaturated OB		Peat	
		Min(1)	Max(2)	Min(1)	Max(2)
Silver (Ag)	mg/L	0.000025	0.00005	0.000025	0.0014
Aluminum (Al)	mg/L	0.068	0.32	0.042	0.13
Alkalinity	mg $CaCO_3$/L	4.6	13.1	8.5	82.7
Arsenic (As)	mg/L	0.0004	0.0032	0.0029	0.0044
Boron (B)	mg/L	0.0005	0.030	0.18	0.23
Barium (Ba)	mg/L	0.003	0.014	0.01	0.035
Beryllium (Be)	mg/L	0.0001	0.0002	0.0001	0.0002
Calcium (Ca)	mg/L	1.8	5.9	15.8	22.9
Cadmium (Cd)	mg/L	0.00005	0.00016	0.00002	0.00004
Chlorine (Cl)	mg/L	0.74	3.6	2.7	9.2
Cobalt (Co)	mg/L	0.00005	0.0016	0.00005	0.0007
Chromium (Cr)	mg/L	0.0001	0.0011	0.0001	0.001
Copper (Cu)	mg/L	0.005	0.0083	0.003	0.011
Fluoride (F)	mg/L	0.025	0.48	0.08	1.1
Iron (Fe)	mg/L	0.02	0.06	0.02	0.12
Potassium (K)	mg/L	0.66	1.3	1.5	5.7
Magnesium (Mg)	mg/L	0.66	2.1	7.8	10.9
Manganese (Mn)	mg/L	0.0075	0.11	0.059	0.19
Sodium (Na)	mg/L	1.8	4.3	4.2	47.3
Nickel (Ni)	mg/L	0.0008	0.0033	0.0015	0.0066
Lead (Pb)	mg/L	0.000025	0.00005	0.000025	0.00023
Antimony (Sb)	mg/L	0.00005	0.0011	0.0005	0.0007
Selenium (Se)	mg/L	0.0001	0.0006	0.0006	0.0009
Sulfate (SO_4)	mg/L	1.7	16.5	68.3	93.4
Thallium (Tl)	mg/L	0.00001	0.00003	0.00001	0.00011
Vanadium (V)	mg/L	0.0004	0.0006	0.0025	0.0043
Zinc (Zn)	mg/L	0.002	0.006	0.0005	0.004

(1) Minimum value from MWMP tests. LOD/2 substituted for non-detects.
(2) Maximum value from MWMP tests. LOD substituted for non-detects.

7.3 Leachate Water Quality – Saturated Overburden

As discussed in Section 3.2, Saturated Overburden from the Mine Site has the potential to release constituents at levels that could have significant environmental impact. Accordingly, the Saturated Overburden material will be treated as Category 2/3 or Category 4 waste rock: placed in the temporary lined waste rock stockpiles and ultimately relocated to subaqueous placement in the East Pit.

Because the Saturated Overburden will be commingled with the Category 2/3 and Category 4 waste rock, it was not modeled as geochemically distinct from the waste rock. Rather, the Saturated Overburden added to the temporary waste rock stockpiles is conservatively assumed to behave identically to the Duluth Complex waste rock. The mass of Saturated Overburden added to each stockpile is considered to be an equal mass of the Duluth Complex waste rock.

8.0 Geochemical Parameters – Waste Rock

This section covers geochemical parameters relating to waste rock that are used in water quality modeling.

8.1 Laboratory Release Rates

An updated methodology for interpreting the results of the NorthMet humidity cell tests is described in Section 4.1.3.1 and Large Table 2, based on the methods proposed in Attachment A. The specific methods used to develop probability distributions for the various constituents and waste rock categories are detailed here. The probability distributions discussed in this section are presented in Large Table 6 through Large Table 10 and shown in Large Figure 1 through Large Figure 22.

In general, the release rates described in this section apply to nonacidic conditions for all Duluth Complex waste rock categories and ore. The effects of acidification are addressed in Section 8.2.5 below. For Virginia Formation Category 4 waste rock, acidic conditions were observed almost immediately in the humidity cell tests and the effects of acidification are represented in the methods described here.

8.1.1.1 Sulfate Release Rates for Category 1, Category 2/3 and Ore

As described in Attachment A and shown in Large Table 2, the release of sulfate from oxidation of sulfide minerals is a primary driver for modeling the release of many other constituents. This section describes the methods for modeling the sulfate release rates for each category of waste rock.

8.1.1.2 Linear Regression Analysis

Laboratory data from NorthMet humidity cells show that there is a strong correlation between sulfur content and average nonacidic sulfate release. Data from all Duluth Complex NorthMet humidity cells are summarized in Figure 8-1 and Figure 8-2. The Category 4 Virginia Formation humidity cells did not experience significant nonacidic conditions and are therefore not presented here. As is evident from the data, the correlation between sulfur content and average sulfate release is strongest for Category 1 and Category 2/3 waste rock. These waste rock categories contain relatively low amounts of sulfur and do not rapidly become acidic (Category 1 waste rock never becomes acidic).

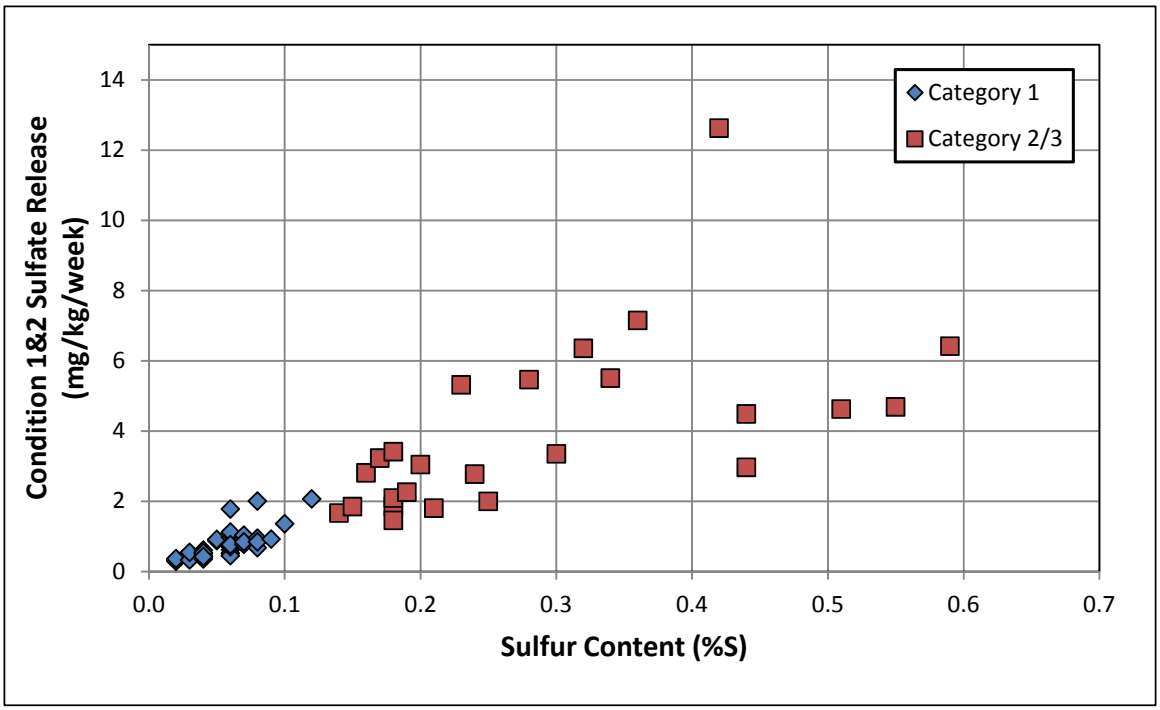

Figure 8-1 Sulfur Content versus Nonacidic Sulfate Release for Category 1 and 2/3

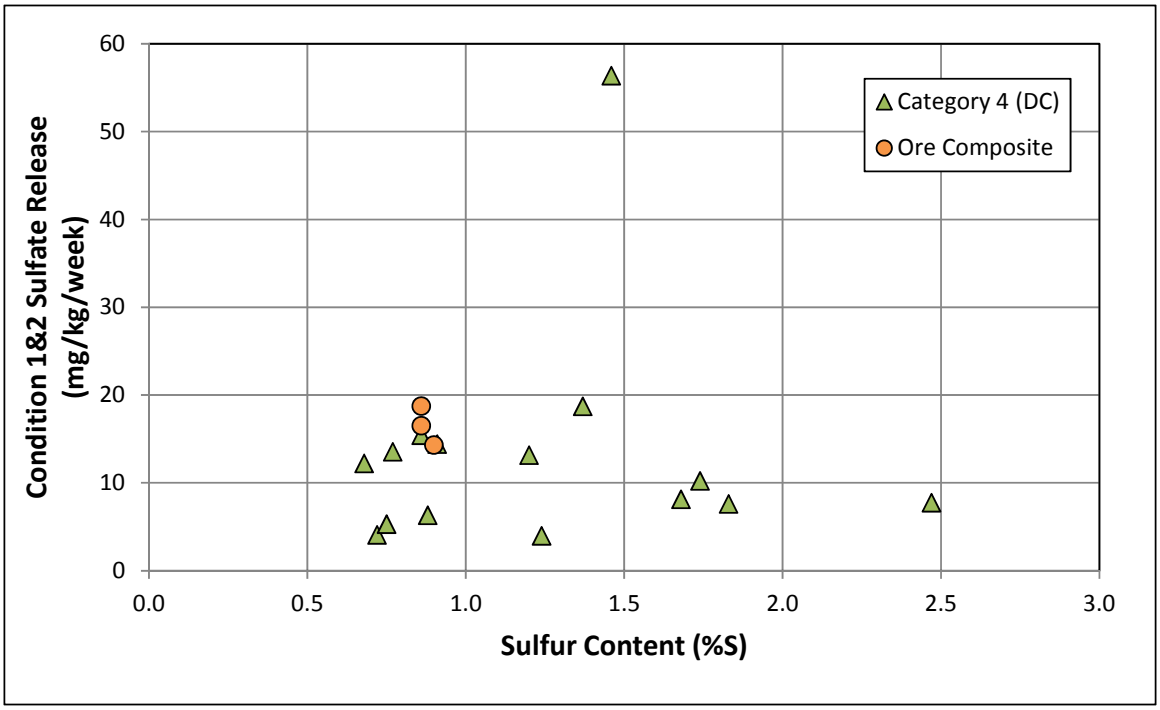

Figure 8-2 Sulfur Content versus Nonacidic Sulfate Release for Duluth Complex (DC) Category 4 and Ore

Sulfate release for the Duluth Complex Category 4 waste rock and for the ore composites does not appear as well-correlated with sulfur content, especially for sulfur contents above about 1.5%. At sulfur contents below 1.5%, however, and excluding two visually-determined outlier values (3.93 mg/kg/week at 1.24%S and 56.3 mg/kg/week at 1.46%S), the Duluth Complex Category 4 and ore data appear to follow the same trend as the lower-sulfur Category 1 and Category 2/3 data (Figure 8-3), with a correlation between sulfur content and average nonacidic sulfate release.

This section describes the development of a mathematical model for estimating nonacidic sulfate release rate from Category 1 and Category 2/3 waste rock and ore as a function of sulfur content. Such a model is needed in order to estimate the oxidation rates from the exposed mine pit walls, which primarily consist of these rock types and have sulfur content that varies with elevation (see Section 4.3.2). This model is especially important for modeling the wall rock classified as ore, which has average sulfur contents around 0.5% in the final West Pit highwall (Figure 4-18), compared to 0.9% sulfur in the ore humidity cell tests. Note that this model will not be used for sulfate release from the Category 4 Duluth Complex or Virginia Formation rock.

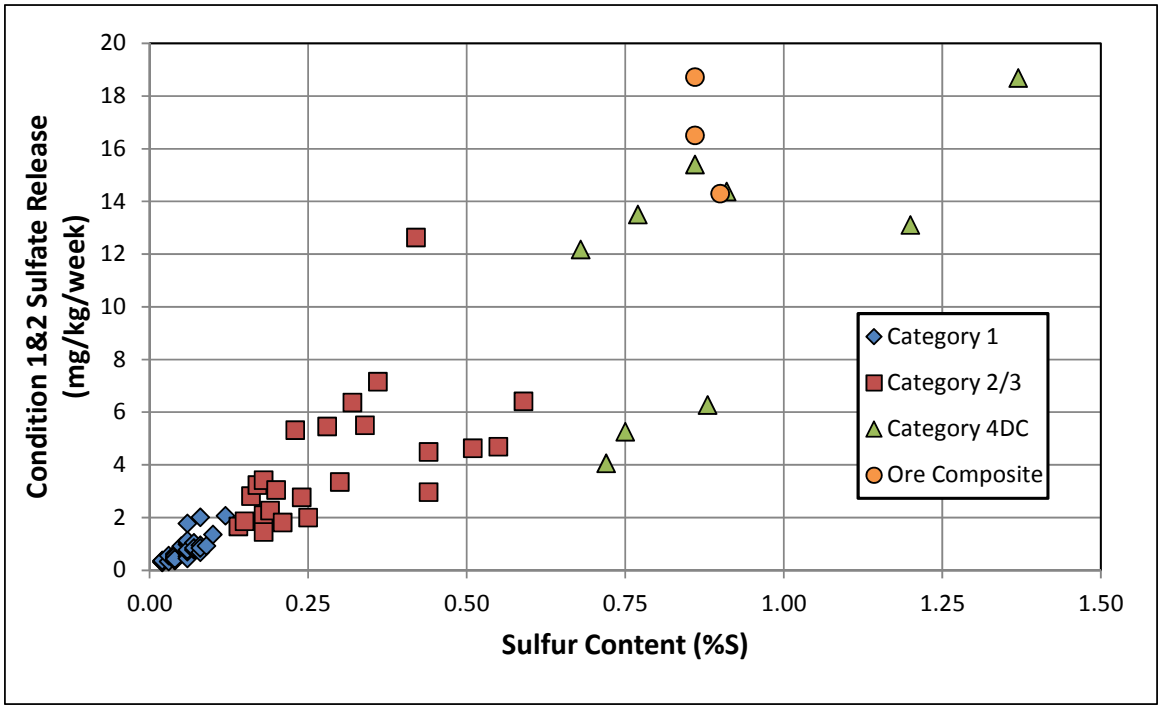

Figure 8-3 Sulfur Content versus Nonacidic Sulfate Release for Duluth Complex (DC) Samples with less than 1.5% Sulfur (Outliers Removed)

As a relatively simple mathematical model, linear regression by the method of ordinary least squares is used to develop a relationship between the sulfur content and nonacidic sulfate release of the form

$$R = aS + b \qquad \textbf{8-1}$$

where R is the sulfate release rate [mg/kg/week], S is the waste rock sulfur content by mass [%S] and a and b are the slope and intercept of the regression line, respectively. The resulting regression parameters and their confidence intervals are shown in Table 8-1 for each category of waste rock considered separately. The Duluth Complex Category 4 samples and the ore composites are considered together in this analysis because they overlap sulfur content ranges.

Rock with zero sulfur content could be reasonably expected to release zero sulfate. Given the initial geochemical premise of zero sulfate release from rock that contains no sulfur, it is hypothesized that the intercept of the general model could be zero. Therefore, the regression analysis was performed both with the general model shown in Equation 8-1 and with a fixed zero intercept (Equation 8-1 with $b = 0$). If the zero-intercept model can be reasonably used, the goodness of fit of this and the general model can be compared to assess whether one model accounts for a greater proportion of the variance in sulfate release rate. Results of the regression analysis for each rock category considered separately, and the combined data set, are presented in Table 8-1.

For all three sample groups (Category 1, Category 2/3, and Duluth Complex Category 4 plus ore) considered individually, the linear regression intercept parameter b is not significantly different from zero at the 5% significance level (p-values well above 0.05). In all three cases, the zero intercept models have a goodness-of-fit that is nearly identical to the general models (as measured by the root mean square error or RMSE, see footnote to Table 8-1) and account for the same proportion of the variance in the sulfate release data set.

Even though the statistical tests cannot prove that the intercept parameter is equivalent to zero, the zero-intercept model is selected going forward because it is consistent with the linear relationship that a zero sulfur content rock will have a zero sulfate release.

Table 8-1 Linear Regression Results for Sulfate Release

Category	Model	Par.	Units	Value	Std. Error (par.)	95% CI	RMSE[1] (model)
Category 1	general	a	$\frac{mg}{kg \times week \times \%S}$	14.45	1.98	10.43 - 18.47	0.28
		b	$\frac{mg}{kg \times week}$	-0.04	0.12	-0.27 - 0.2	
	zero intercept	a	$\boldsymbol{\frac{mg}{kg \times week \times \%S}}$	13.86	0.80	12.24 - 15.48	0.28
Category 2/3	general	a	$\frac{mg}{kg \times week \times \%S}$	10.71	3.12	4.25 - 17.17	1.96
		b	$\frac{mg}{kg \times week}$	0.88	0.99	-1.17 - 2.92	
	zero intercept	a	$\frac{mg}{kg \times week \times \%S}$	13.22	1.29	10.57 - 15.88	2.00
Category 4 (Duluth) (<1.5%S) plus ore	general	a	$\boldsymbol{\frac{mg}{kg \times week \times \%S}}$	12.07	6.91	-3.33 - 27.47	4.17
		b	$\frac{mg}{kg \times week}$	1.86	6.34	-12.26 - 15.98	
	zero intercept	a	$\frac{mg}{kg \times week \times \%S}$	14.05	1.38	11.02 - 17.08	4.19
Combined data set (Duluth, <1.5%S)	general	a	$\boldsymbol{\frac{mg}{kg \times week \times \%S}}$	13.84	0.76	12.32 - 15.36	2.05
		b	$\frac{mg}{kg \times week}$	0.03	0.31	-0.59 - 0.66	
	zero intercept	a	$\frac{mg}{kg \times week \times \%S}$	13.89	0.58	12.74 - 15.04	2.05

(1) The root mean square error (RMSE) is a goodness of fit parameter used for comparing goodness of fit of different models for the same data set. The RMSE is sensitive to the sample size and therefore is not comparable between data sets.

8.1.1.3 Combining Data Sets

The fitted slope parameters for the three sample groups shown in Table 8-1 are comparable and the 95% confidence intervals mostly overlap. Because these rock groups share a similar bulk rock composition and are only categorized by differences in sulfur content, it is reasonable to evaluate whether they can be treated as a single data set for the purpose of evaluating sulfate release. If the rock categories can be combined for this analysis, the larger data set will decrease the uncertainty in the estimate of the sulfate release rate and allow for modeling sulfate release at intermediate sulfur contents (i.e., ore wall rock with sulfur content ranging from 0.5% to 0.8%). The statistical hypothesis test for equivalence of the slope parameters, using the standard error values shown in Table 8-1, yields a test statistic *z* given by Equation 8-2:

$$z = \frac{a_i - a_j}{\sqrt{SE_i^2 + SE_j^2}} \qquad \textbf{8-2}$$

Larger z values indicate a greater difference between the two slope parameters and a lower likelihood that they are equal to one another. For example, a z value of 1.96 corresponds to a p-value of 0.05 (two-tailed normal test for z), indicating that the two slope parameters are significantly different at the 5% significance level. Comparing the data sets two at a time, the calculated z values are:

- Categories 1 and 2/3: 0.42 for zero intercept; 0.31 for general model
- Categories 1 and 4 (Duluth, <1.5%S) plus ore: 0.12 for zero intercept; 0.33 for general model
- Categories 2/3 and 4 (Duluth, <1.5%S) plus ore: 0.44 for zero intercept; 0.18 for general model

Using a two-tailed normal test for *z* and a 5% significance level α, the null hypothesis (that the slope parameters are equal to one another) cannot be rejected for any of these six hypothesis tests (p-values of 0.66 to 0.91 for zero-intercept model and 0.31 to 0.86 for the general model). Data from the Category 1, Category 2/3, and Category 4 (Duluth, <1.5%S) plus ore humidity cells are therefore assumed to come from the same population, and are combined to assess sulfate release as a function of sulfur content.

The combined data was analyzed using the ordinary least squares method and the zero-intercept model. Results for the combined data are shown in Table 8-1. The general model has an intercept value that does not significantly differ from zero and the same RMSE value as the zero-intercept model. Finally, the z-value for a test of differences in the slopes between the general model and the zero-intercept model for the combined data is 0.052, indicating that the two models may not significantly differ in their slopes (p-value of 0.96).

Even though the statistical tests cannot prove there is no difference between the slopes for the different rock categories, the combined zero-intercept model is selected going forward because it provides a consistent method for modeling Duluth Complex rock at sulfur contents across the range of expected conditions in the pit walls.

8.1.1.4 Testing Assumptions of Ordinary Least Squares

Ordinary least squares, like most statistical methods, requires several assumptions about the characteristics of the dataset in order to be fully valid:

1. The residuals (difference between model-predicted release rate and observed release rate) have a mean value of zero;

2. The residuals fit a normal distribution; and
3. The residuals show no significant correlation with the dependent variable (%S) (i.e., they are not heteroscedastic).

These assumptions were tested in the statistical software Minitab 17 to evaluate the validity of the above results for the combined zero-intercept model: First, a *z*-test indicated that the residuals of the zero-intercept model do not significantly differ from zero ($p>0.5$), so the data are consistent with the first assumption. However, further testing showed that the data do not conform to the second and third assumptions. An Anderson-Darling normality test indicated that the residuals are not normally distributed ($p<0.005$; Figure 8-4, left figure). The absolute values (magnitude) of the residuals also showed a significant linear relationship with %S ($p<0.001$, $R^2=52\%$; Figure 8-4, right figure), indicating heteroscedasticity. Overall, the latter two findings indicate that the data do not conform to the assumptions of ordinary least squares, so an alternative method to ordinary least squares should be applied.

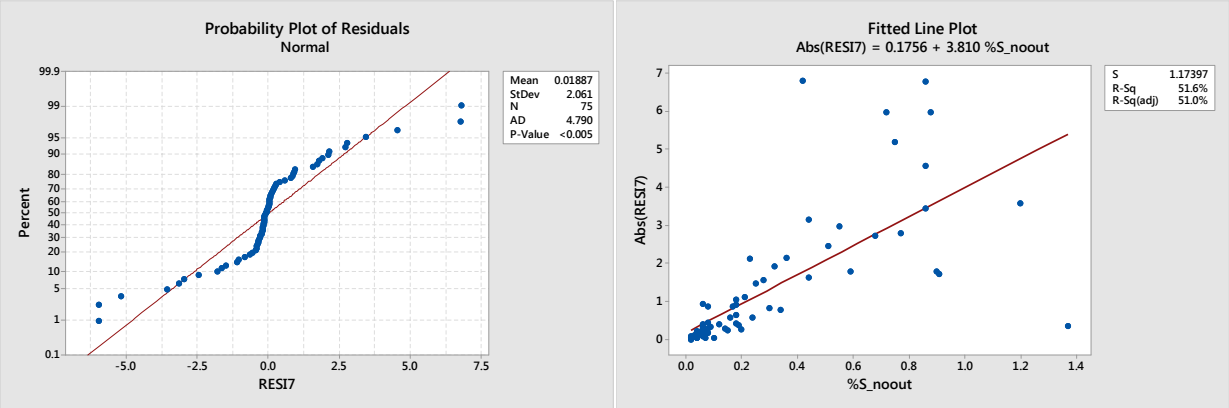

Figure 8-4 Residuals Testing for the Regression Model for Combined Duluth Complex Data

8.1.1.5 Correction for Non-Constant Variance

It is clear from Figure 8-4 that the sulfate release data are heteroscedastic; the dispersion of the data around a best-fit line increases as sulfur content increases, violating the third basic assumption of the ordinary least squares regression model. To correct this problem the data can be analyzed using weighted linear regression (Reference (16)), a method that assumes that the standard deviation of the model residuals e_i (error between the observed data and the best-fit line) increases linearly in proportion to the independent variable (sulfur content):

$$Std(e_i) = \kappa S_i \qquad \textbf{8-3}$$

where κ is the unknown proportionality constant. For the linear regression model with zero intercept, the slope parameter a can be calculated from

$$a = \frac{1}{n}\sum_{i=1}^{n}\left(\frac{R_i}{S_i}\right)^2 \qquad \textbf{8-4}$$

where R_i and S_i are the sample sulfate release rate and sulfur content previously defined. The proportionality constant κ is estimated as shown in Equation 8-5 and the standard error of the slope parameter a is calculated from Equation 8-6.

$$\kappa = \sqrt{\frac{1}{n-1}\sum_{i=1}^{n}\left(\frac{R_i}{S_i} - a\right)^2} \qquad \textbf{8-5}$$

$$\sigma_a = \frac{\kappa}{\sqrt{n}} \qquad \textbf{8-6}$$

Because a is calculated as the sum of n independent random variables, the Central Limit Theorem suggests that a will be well-modeled by a normal distribution regardless of the distribution of the model residuals.

The analysis described above yields a slope parameter a equal to 13.92 [mg/kg/week/%S] with standard error σ_a equal to 0.581 [mg/kg/week/%S]. This single-parameter best-fit line is shown for the NorthMet humidity cell data in Figure 8-5 and the normalized residuals are shown in Figure 8-6. The mean of the residuals is not significantly different from zero ($p>0.9$), and the normalized residuals show no significant relationship with %S ($p>0.3$). Because no particular distribution of the residuals is assumed for weighted least squares to be valid, these tests indicate that the method of weighted least squares was successful in addressing the problem of heteroscedasticity and that the resulting relationship can be effectively applied in probabilistic modeling. Note that the 95% confidence interval for a shown in Figure 8-5 is 12.78 to 15.06 [mg/kg/week/%S], a range that includes all of the zero-intercept slope estimates for the individual rock categories as presented in Table 8-1.

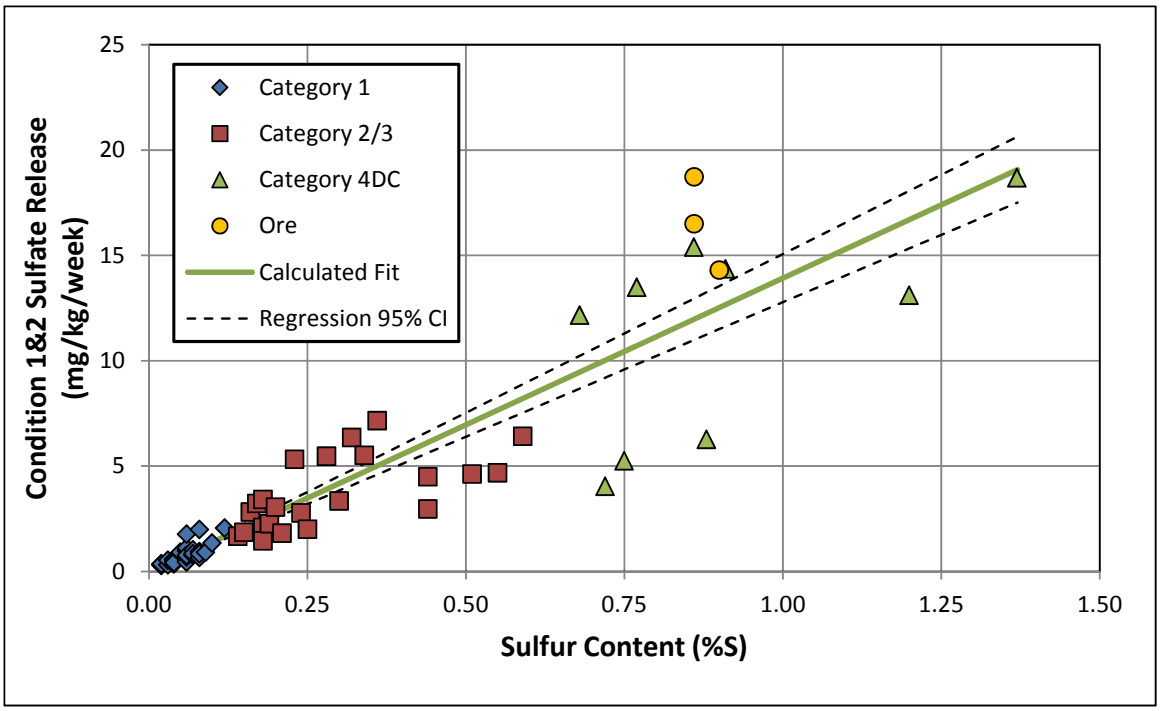

Figure 8-5 Weighted Linear Regression Fit for Combined Duluth Complex Data

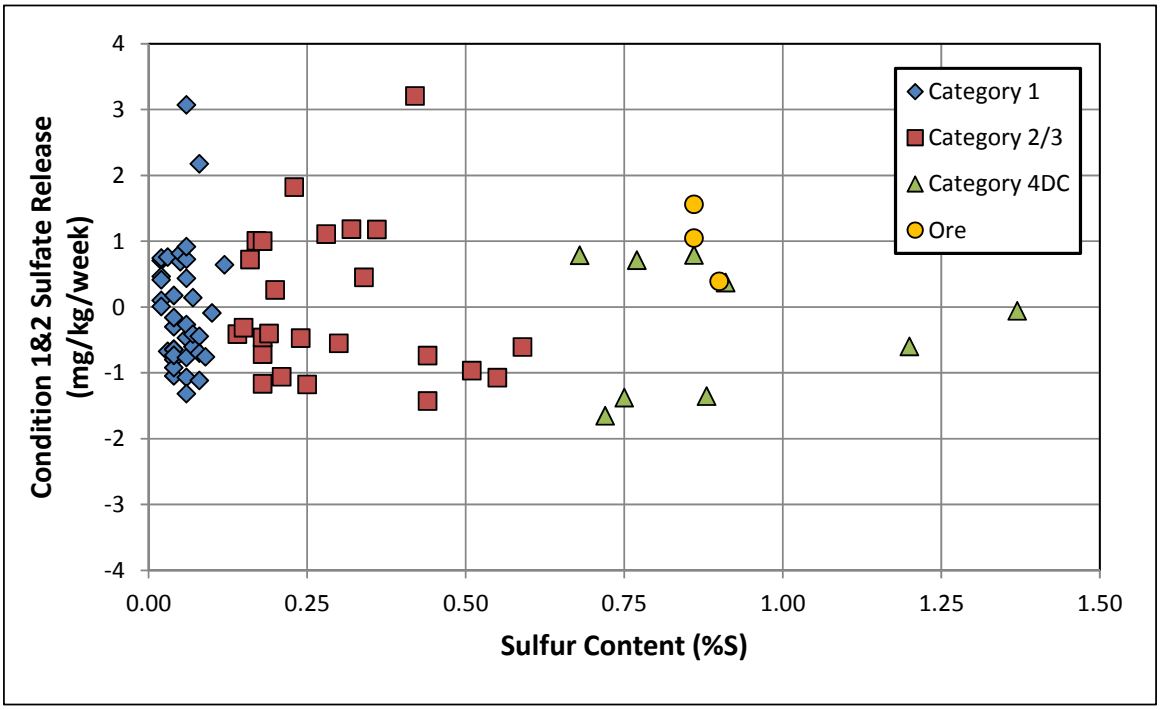

Figure 8-6 Normalized Residuals from Weighted Linear Regression Fit

8.1.1.6 Application to Probabilistic Modeling

The sulfate release rate used in the probabilistic water quality model for each waste rock category can be understood as the average release rate from an entire waste rock stockpile or category of wall rock. Each stockpile will contain millions of tons of waste rock, whereas the humidity cells contain approximately one kilogram of waste rock each. The stockpile sulfate release, therefore, is equivalent to an average release rate over billions of combined humidity cells. The uncertainty in the average sulfate release rate is not the same as the possible variability over the population of humidity cells.

Using the regression analysis described above, the average sulfate release rate from a given rock category can be calculated as a function of the average sulfur content from Equation 8-7. This is simply the zero-intercept equation for linear regression (Equation 8-1)

$$R = aS \qquad \textbf{8-7}$$

The uncertainty in the slope parameter a is represented with the normal distribution developed above and shown in Figure 8-7. The average stockpile sulfur content is equal to the weighted average value from the Block Model presented in Section 4.3.2, considered to be deterministic (known). This method of modeling sulfate release is only applied to Category 1 and Category 2/3 waste rock and ore in the water quality modeling based on the linear model developed for sulfur contents less than 1.5%. The sulfate release rate for all other waste rock categories is modeled as described in Section 8.1.2 for “constituents with release rates determined directly from humidity cells.”

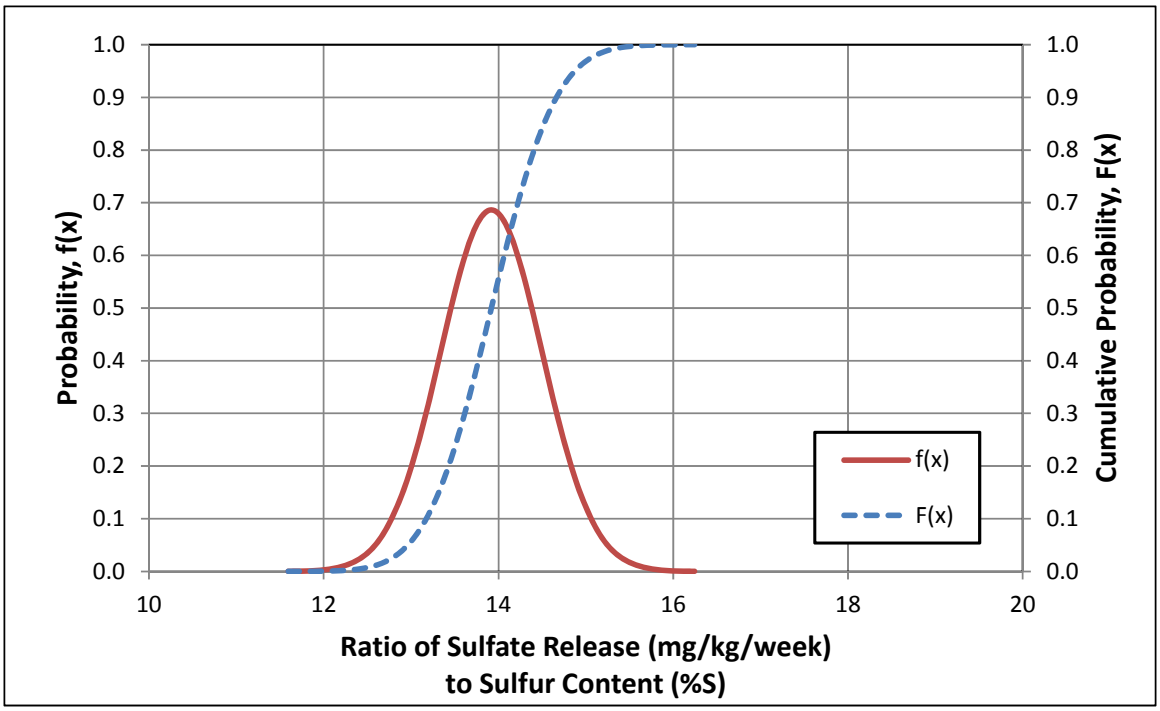

Figure 8-7 Distribution for the Sulfate Release Slope for Category 1, Category 2/3 and Ore

8.1.2 All Other Release Rates

The methods for modeling the uncertainty in laboratory-scale release rates for all constituents other than sulfate are described below.

As discussed in Section 4.1.3.1 and Attachment A, data from the NorthMet humidity cells demonstrate that solubility limitations or concentration capping effects can reduce or increase metal leaching rates from the humidity cells, especially for constituents that are highly sensitive to small changes in leachate pH. The modeling approach described here has been developed in consultation with the Co-lead Agencies to account for the influence of concentration caps on the humidity cell leaching rates.

Chemical constituents, including, major mineral components and trace metals are released from host primary minerals as these minerals weather. The Project data indicated that after constituents are released from primary minerals, the concentrations of some of the chemical constituents in humidity cell tests were limited by solubility relationships with secondary minerals, while others were not. The release rates of constituents that appeared to be subject to solubility limitations were determined by identifying the likely mineral sources for each constituent using Project data and information from literature sources. The release rates of constituents that were potentially controlled by solubility relationships were identified by correlations with constituents in the same primary mineral host that were not controlled by solubility limitations.

This approach to estimating release rates relies on empirical analysis of the humidity cell data and data for metal content in the drill core samples (aqua regia digestion) and individual minerals (microprobe analysis). The rates and relationships determined for the different waste rock categories are therefore specific to this data set. The specific humidity cells used in this analysis, their rock type, sulfur content, and test duration are shown in Large Table 1. The proposed approach for developing release rates for each constituent is outlined in Large Table 2. Distribution parameters and data set date ranges used in the probabilistic modeling are shown in Large Table 6 through Large Table 10 and Large Figure 1 through Large Figure 22.

8.1.2.1 Release Rates from Humidity Cells

For all constituents with release rate methods identified by "XX Rate" in Large Table 2, probability distributions have been developed directly from the humidity cell data. For each cell a temporal average release rate has been determined for nonacidic conditions (described as Condition 1 and Condition 2 in Large Table 1) and acidic conditions, as applicable (described as Condition 3 in Large Table 1, the period of acidic conditions prior to any decrease in sulfate release). Non-detect samples have been taken as equal to the detection limit. As previously noted, the release rates determined in this manner for the Virginia Formation Category 4 waste rock and some constituents for the Category 2, 3 and 4 Duluth Complex waste rock are for acidic conditions.

A probability distribution has been fit to the sample data set of average release rates (one per humidity cell) applicable for the specific geology and waste rock category. The complete range of release rates observed in the humidity cell testing has been assumed to represent the possible average release rate from an entire waste rock stockpile. Except as discussed in Section 8.1.1.1 for sulfate, no attempt has been made to weight the humidity cell data by sulfur content or otherwise bias the determined release rates towards the expected average conditions in the field. This results in conservatively wide ranges for the modeled release rates relative to the likely average field conditions.

For a group of trace metals in the Category 1 waste rock (Ag, As, B, Be, Cr, Pb, Se, Tl, and V), the large number of non-detects in the humidity cell dataset warrants additional consideration. For these constituents, the average release rates for each humidity cell have been determined with a more detailed consideration of the effects of non-detect data for the time period of data indicated in Large Table 6. This calculation made use of the ProUCL software, an EPA-approved software for the analysis of non-detect data.

- For humidity cells where all analyses results in detected values, the leachate concentration was calculated as a simple average.
- For humidity cells with at least 6 total detected values, ProUCL was used to estimate the mean leachate concentration using the nonparametric, Kaplan-Meier Method.

- For humidity cells with less than 6 total detected values, ProUCL could not be used. The average leachate concentration was calculated from all data, substituting zero for the non-detect values.
- For humidity cells with no detected values, the average leachate concentration was assumed to be zero.
- The resulting estimated average leachate concentrations (one for each humidity cell) were converted to release rate units (mg/kg/week) using the sample mass and average leachate volume. For Se, the resulting release rates were divided by the average sulfate release rates in the corresponding humidity cells to calculate the Se/SO_4 ratio.
- A lognormal distribution was fit to the resulting release rate values.

These same methods have been applied to a smaller group of constituents (B, Cr, Tl) for the other Duluth Complex waste rock categories for nonacidic conditions only.

A similar method for considering the effect of non-detects has been used to develop the Sb release rate for Category 1 waste rock, but using smaller MDNR-style reactor experiments due to Sb contamination in the NorthMet humidity cells.

- The time period isolated for this analysis was the first 80 weeks of testing. This is shorter than the time period used for analysis of humidity cell data because of the more-rapid oxidation and release that is observed in the MDNR-style reactors due to the finer material.
- Only reactors with sulfur content less than 0.12% (equivalent to Category 1 waste rock) were used; in addition, only reactors with the smallest size fraction (passing 100 mesh) are used, consistent with MDNR testing methods. There are two reactors meeting these criteria [from samples DDH-00-334C (640-660) and DDH-00-367C (290-310)].
- Only laboratory analyses using the ICP-MS method were used, with a typical detection limit of 0.1 µg/L.
- The resulting data set (20 samples from each reactor, 40 samples total) were combined into a single data set and ProUCL was used to estimate the mean leachate concentration using the nonparametric, Kaplan-Meier Method.
- The resulting mean and standard deviation concentrations were converted to release rate units (mg/kg/week) using the sample mass (75g), average leachate volume (182 mL), and the MDNR-developed release rate conversion between reactors and humidity cells (humidity cell rate = reactor rate x 0.34) (Reference (17)). These parameters were assumed to describe a lognormal distribution for the Sb release rate from Category 1 waste rock.

For ore, the “ore composite” humidity cells have been used to define probability distributions for the material in the Ore Surge Pile (except for sulfate as discussed in Section 8.1.1.1). These humidity cells represent blended samples, similar to the mixed material that will be in the stockpile. For modeling of wall rock as discussed in Section 9.1, Category 2/3 humidity

cell distributions are used for the ore wall rock rather than the "ore composite" humidity cell distributions. This method more accurately represents the range of variability in release rates that is expected across the unmixed ore wall rock.

8.1.2.2 Release Rates from Humidity Cell Release Ratios

For all constituents with release rate methods identified by "XX/XX Rate" in Large Table 2, probability distributions have been developed from the ratio of release rates in the NorthMet humidity cells. For each cell a temporal average release rate ratio (ex. the ratio of average Mn release to average SO_4 release) has been determined for nonacidic conditions (Condition 2 in Large Table 1). A probability distribution has been fit to the population of average release rate ratios (one per humidity cell) applicable for the specific geology and waste rock category. The actual release rate for the constituent is simulated by multiplying the randomly-selected release rate ratio by the randomly-selected release rate of the constituent in the denominator (ex. Mn release rate = Mn/SO_4 release ratio x SO_4 release rate). Release rates under acidic conditions are modeled using the acidity factor to increase the SO_4 release rate (and correspondingly increase the ratioed constituent release rates) as discussed in Section 8.2.5. As discussed above for the release rates themselves, the complete range of release rate ratios observed in the humidity cell testing has been conservatively assumed to represent the possible average release rate ratio from an entire waste rock stockpile or category of pit wall, with no weighting or relationship to sulfur content. Release ratios for ore have been developed from the ore composite and Category 2/3 humidity cells as described above.

8.1.2.3 Release Rates from Solid Ratios

For all constituents with release rate methods identified by "XX/XX" and data sources other than "HCT" in Large Table 2, probability distributions are developed from the solids content data in the NorthMet drill core database. For each constituent, a probability distribution for the solids ratio (ex. ratio of Cu/S in the whole rock analysis) is fit to the population of observed solids ratios applicable for the specific geology and waste rock category.

For constituents with "Aqua Regia" identified as the source data, the entire NorthMet drill core database from 2005 to 2010 has been used to develop distributions, not just the analysis of the humidity cell samples. This database forms the basis for the Block Model and includes approximately 18,800 individual samples. Each sample is identified as either waste rock or ore based on PolyMet's economic criteria as of May 2011 and is classified by rock category, rock type and geologic unit. Only those samples in the drill core database identified as waste rock have been used to develop distributions for the metals content applicable to the waste rock stockpiles (and corresponding portions of the pit walls); only samples identified as ore have been used to develop distributions applicable to the Ore Surge Pile (and ore portions of the pit walls).

The NorthMet drill core database includes samples collected from outside of the proposed pit shells and does not necessarily represent the expected abundance of each rock type or geologic unit in the actual waste rock or ore. For example, 6.3% of the Category 1 drill core samples are from the combined Units 4 and 5, but material from these units makes up 21.3% of the Category 1 waste rock according to the Block Model. To correct for this difference between the drill core data and the Block Model, all analysis of the drill core aqua regia data was performed by weighting each sample according to the quantity of each geologic unit in each waste rock category and the ore in the Block Model. The influence of each geologic unit on the statistics described below is therefore consistent with the Block Model.

Because of the large number of samples available in the NorthMet drill core database, it is reasonable to assume that the weighted mean solids content (and solids ratio) from the database represents the mean solids content expected in the NorthMet waste rock. The probability distributions developed from this dataset are intended to represent the average solids ratio for an entire waste rock stockpile or category of pit wall (weighted by geologic unit as described above). Therefore, the probability distributions developed from the drill core data are distributions for the uncertainty in the average solids ratios, rather than the distribution for the population of solids ratios. The uncertainty in the sample average solids ratio is described as a normal distribution for each constituent according to the Central Limit Theorem.

For constituents with "Microprobe" identified as the source data, the solids ratios identified from analysis of individual mineral grains in the humidity cell tests have been used to develop distributions. As discussed above for the humidity cell release rates, the complete range of solids ratios observed in the microprobe testing (irrespective of sample sulfur content or category) has been conservatively assumed to represent the possible average release rate ratio from an entire waste rock stockpile or category of pit wall, with no weighting or relationship to sulfur content. The distributions developed from the microprobe data are applied identically to all applicable rock categories (i.e., the Fe/S distribution in pyrrhotite is the same for the Duluth Complex Category 1, 2/3, 4, waste rock and ore). The differences in mineral content between the waste rock categories are represented by the differences in the modeled release of the constituent in the denominator (e.g., differences in pyrrhotite content are represented by the modeled SO_4 release rate for each rock category).

The simulated release rate for each constituent is calculated by multiplying the randomly-selected solids ratio by the randomly-selected release rate of the constituent in the denominator (ex. Cu release rate = Cu/S ratio x SO_4 release rate in terms of S). Release rates under acidic conditions are modeled using the acidity factor to increase the SO_4 release rate (and correspondingly increase the ratioed constituent release rates) as discussed in Section 8.2.5.

8.1.2.3.1 Nickel Release

Nickel release rates are calculated in a similar manner to the other constituents defined from solid ratios, with several adjustments to reflect the behavior of nickel in the NorthMet humidity cells and drill core test work. Nickel release in the humidity cells is complicated by storage and release caused by concentration caps and changing pH conditions, resulting in low apparent initial release followed by very high apparent release. The proposed method has been developed in consultation with the Co-lead Agencies as a better estimate of the true underlying nickel release prior to the application of concentration caps.

The nickel to sulfur ratio from the Duluth Complex Category 4 aqua regia (drill core) data is used to conservatively represent the combined influence of all sulfide minerals. This method is preferred over mineral-specific methods previously proposed because it does not require an estimate of the relative content of various minerals (i.e., the relative occurrence of pyrrhotite and pentlandite). The nickel content of the Duluth Complex Category 4 rock is understood to be almost entirely from sulfide minerals, while lower-sulfur rock categories contain more nickel in olivine. The nickel to magnesium ratio from the microprobe data is used to represent the influence of olivine. These ratios are applied identically to all Duluth Complex rock categories, with the only distinction that the ore nickel to sulfur ratio is used for modeling ore rather than the Category 4 ratio.

For nickel (and other constituents with microprobe data for specific minerals used to develop metal ratios), the ratio is multiplied by the total release rate of the constituent in the denominator (i.e., not specific to the minerals identified for the microprobe data). Equation 8-8 below shows an example calculation for nickel release for the Category 1 waste rock, which is a function of: (1) the nickel to sulfur ratio in the Category 4 Duluth Complex aqua regia data; (2) the mass ratio of sulfur to sulfate; (3) the sulfate release rate for Category 1 waste rock, developed from humidity cell data; (4) the nickel to magnesium ratio in the olivine microprobe data; and (5) the magnesium release rate for Category 1 waste rock, developed from humidity cell data. Of these five inputs to the equation, only the mass ratio of sulfur to sulfate is a deterministic value; the remaining inputs are randomly generated using the methods described above.

$$Ni_{Cat1}\left(\frac{mg}{kg \cdot week}\right) = \left[\frac{Ni}{S}\right]_{Cat4} \cdot \left[\frac{S}{SO_4}\right] \cdot SO_4\left(\frac{mg}{kg \cdot week}\right)_{Cat1} + \left[\frac{Ni}{Mg}\right]_{Ol} \cdot Mg\left(\frac{mg}{kg \cdot week}\right)_{Cat1} \quad \textbf{8-8}$$

8.2 Lab to Field Scale-Up

8.2.1 Background

As outlined in the initial waste rock modeling report (Reference (3) Section 8.2), a composite scale-up factor was used in the 2008 deterministic calculations of solute release from stockpiled waste rock. The scale-up factor is applied to the nonacidic laboratory release rates (from the humidity cell tests) in order to calculate release rates from full scale waste rock disposal facilities. The bulk scale-up factor (S_B) is dependent on four sub-factors that represent the effects of differences between laboratory and field conditions with respect to the water contact (k_c), particle size (k_s), temperature (k_t), and pH (k_{pH}).

$$S_B = k_t * k_s * k_c * k_{pH} \qquad \textbf{8-9}$$

This approach is similar to that developed in a published research study for mining wastes in Sweden (Reference (18)). The only difference is the method used to determine the value of the sub-factor to account for differences in pH between laboratory and field conditions (empirical vs. theoretical). Note that the modeled laboratory release rates described in Section 8.1 are for nonacidic conditions; the pH correction factor therefore represents the potential increase in constituent release rates as pH becomes acidic.

The deterministic water quality modeling used different composite scale-up factors for the various types of NorthMet waste rock:

1) Category 1 waste rock: the scale-up factor was assumed to be dependent on all sub-factors except k_{pH}, because Category 1 waste rock in the field is not expected experience pH lower than that observed in the laboratory (Section 8.3.1).
2) Category 2/3/4 waste rock and ore (Duluth Complex):
    a. The nonacidic scale-up factor was assumed to be dependent only on the particle size and water contact sub-factors. This is similar to the Category 1 waste rock but without k_T. It was conservatively assumed that the reactions within the more reactive (potentially acid generating) waste rock stockpiles will heat up the core of the full scale stockpiles so that there will be negligible temperature difference between full scale and lab conditions, even before the stockpiles have become fully acidic.
    b. The acidic scale-up factor was assumed to be dependent on all sub-factors except k_T.
3) Category 4 Virginia Formation waste rock: the scale-up factor was assumed to be only dependent on the particle size and water contact sub-factors, because this rock is assumed to be immediately acidic and the humidity cell data used are for acidic conditions.

These bulk scale factors and their values in the deterministic modeling are summarized in Table 8-2.

The same assumptions used in the deterministic modeling for which sub-factors apply to which rock type and condition (temperature, particle size, etc.) are also used in the probabilistic modeling. The difference between the scale factors developed for deterministic and probabilistic modeling is that the sub-factors are modeled as a range of values with an associated probability function. The method for simulating the value of each sub-factor is discussed in detail in the following sections.

Table 8-2 Deterministic Scale-Up Factor Assumptions

Rock Type	Condition	Sub-Factors	Deterministic Value(1)
Category 1	Nonacidic (all)	k_t, k_s, k_c	0.03
Category 2/3/4/ore (Duluth Complex)	Nonacidic	k_s, k_c	0.10
	Acidic	k_s, k_c, k_{pH}	1.0
Category 4 (Virginia Formation)	Acidic (all)(2)	k_s, k_c	0.10

(1) Value used in 2008 deterministic water quality modeling (Reference (12))
(2) The acidity factor is not used for modeling Virginia Formation rock because the humidity cells are acidic (no scaling for pH needed)

The proposed methodology for probabilistically modeling the components of the scale-up factor can be applied to all of the waste rock on the Mine Site, with appropriate adjustments to reflect the assumed differences in temperature and pH conditions in the different stockpiles (pH is not explicitly modeled except as discussed in Section 8.3.1). Accounting for the four contributing sub-factors individually (rather than as a single "bulk" scale-up factor) allows the probabilistic models to track the fate of oxidized solutes that are not contacted by water but may be available for future leaching. This is especially important for the waste rock in the temporary stockpiles which is backfilled in the East Pit in the later years of mine operations and flooded with water, mobilizing any remaining oxidized solutes. Regardless of the method to be used for scaling laboratory rates to the field scale, an assumed water contact factor is necessary in order to partition mass that is available for dissolution from that which remains at the source until flooding.

The steps in modeling constituent release from waste rock are as follows:

1) Generate random nonacidic laboratory release rates (Section 8.1)
2) Generate random scale-up correction factors for water contact, particle size, temperature, and acidity (Section 8.2)
3) Calculate field release rates by multiplying nonacidic laboratory release rates by the particle size correction and temperature and acidity corrections (if applicable).
4) Partition the released mass between contact (flushed by runoff) and non-contact mass by multiplying by the contact correction.

a. Contact mass is flushed by runoff and used to estimate drainage water quality.
b. Non-contact mass remains with the waste rock and is flushed into water if the rock is submerged (i.e., in East Pit backfilling).

8.2.2 Water Contact Factor

In the deterministic water quality modeling the contact factor was assumed to be equal to 0.5. This means that half of the rock was assumed to be contacted by water as water percolates through the stockpile, compared to 100% of the rock in the laboratory tests. The percolating water was assumed to contact 50% of the waste rock in the stockpile and collects 50% of the generated solutes, subject to concentration caps (Section 8.3). Theoretically, the contact factor can range from 0 to 1. However, the probability of water contacting 0% or 100% of the waste rock is zero. Therefore, a triangular distribution is proposed for the contact factor, ranging from 0.1 to 0.9 with a mean of 0.5. Figure 8-8 shows the assumed water contact factor probability distribution.

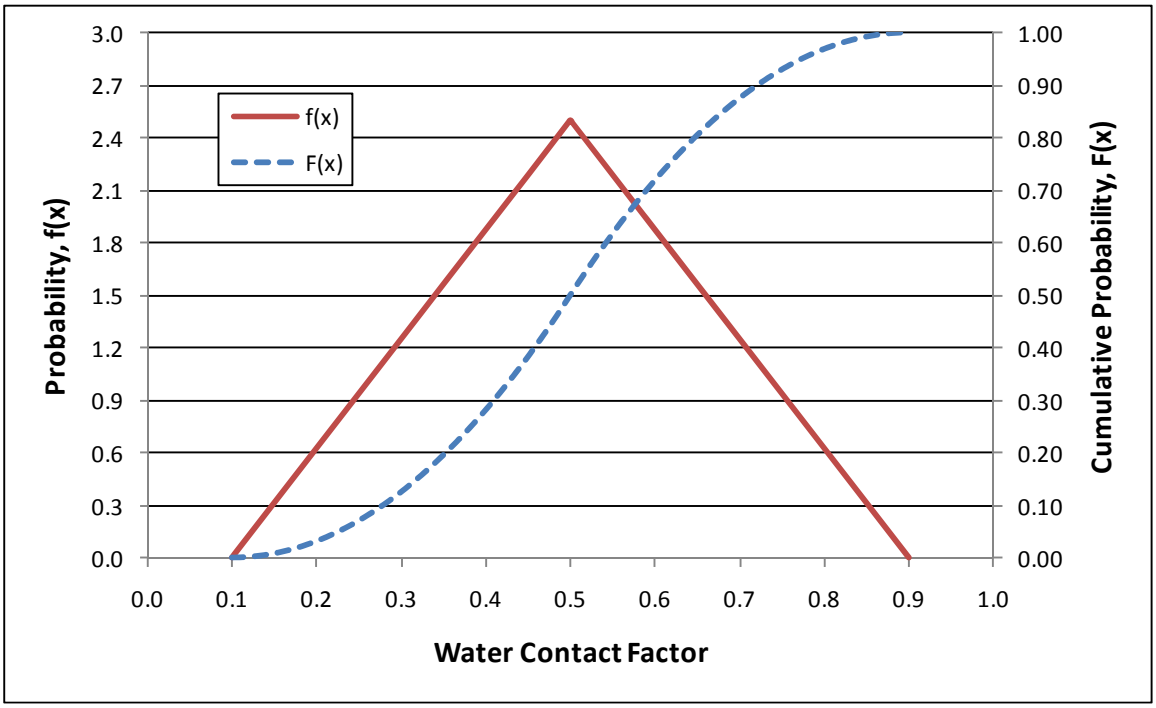

Figure 8-8 Distribution for the Water Contact Factor

8.2.3 Particle Size Factor

In the deterministic water quality modeling the particle size factor was assumed to be equal to 0.2, meaning that one ton of field-scale waste rock has 20% of the surface area of one ton of humidity cell rock. Theoretically, the particle size factor can range from 0 to infinity.

Particle size distribution data are available for Duluth Complex waste rock from the AMAX test piles (Table 24 of Reference (19)). This material was extracted by underground mining, and therefor is expected to contain more fine material than the open pit mining proposed for the Project due to differences in blasting techniques. This leads to a higher surface area to mass ratio, which is conservative from the standpoint of determining a particle size factor. Assuming that the rock breaks into cubes on average, the AMAX samples are calculated to have surface area to mass ratios ranging from 1.0 to 2.4 [m^2/kg], with an average of 1.7 [m^2/kg].

An example calculation of the surface area to mass ratio based on the median AMAX sample is shown in Table 8-3 and outlined in Equations 8-10 to 8-15. From Table 8-3 it is clear that while less than 20% of the sample by mass is finer than gravel (passing the 6.4 mm sieve), the fine material (especially the finest silt- and clay-sized particles) contains virtually all of the particle surface area associated with the sample.

$$\text{Side width}[mm] = \frac{\text{Sieve size}[mm] + \text{Sieve opening}_{\text{larger}}[mm]}{2} \quad \textbf{8-10}$$

$$\text{Surface area}[m^2] = 6 \cdot \text{Side width}^2[m](assume\ cubes) \quad \textbf{8-11}$$

$$\text{Volume}[m^3] = \text{Side width}^3[m](assume\ cubes) \quad \textbf{8-12}$$

$$\text{Mass}[kg] = \text{Volume}[m^3] \cdot 2.93(assume\ spec.\ grav.) \cdot 1000[\tfrac{kg}{m^3}] \quad \textbf{8-13}$$

$$\text{Particles Retained}[\tfrac{1}{100g}] = \frac{\text{Mass}[g]}{\text{Mass Retained}[\tfrac{g}{100g}]} \quad \textbf{8-14}$$

$$\text{Total Retained Surface Area}[\tfrac{m^2}{100g}] = \text{Surface area}[m^2] \cdot \text{Particles Retained}[\tfrac{1}{100g}] \quad \textbf{8-15}$$

Table 8-3 Surface Area to Mass Ratio Calculation Example

	Average Retained Particle Dimensions				Calculations for AMAX 0.06%S Sample			
Sieve Size [mm]	Side Width [mm]	Surface Area [m^2]	Volume [m^3]	Mass [kg]	Mass Passing [g/100g]	Mass Retained [g/100g]	Particles Retained [1/100g]	Total Retained Surface Area [m^2/100g]
305	3.1E+02	5.6E-01	2.8E-02	8.3E+01	100	0	0	0
152	2.3E+02	3.1E-01	1.2E-02	3.5E+01	96.6	3.4	0	0.0000
76.2	1.1E+02	7.8E-02	1.5E-03	4.4E+00	80.2	16.4	0	0.0003
38.1	5.7E+01	2.0E-02	1.9E-04	5.5E-01	51.2	29	0.1	0.0010
19.1	2.9E+01	4.9E-03	2.3E-05	6.9E-02	32.3	18.9	0.3	0.0014
12.7	1.6E+01	1.5E-03	4.0E-06	1.2E-02	20.3	12	1.0	0.0015
6.4	9.6E+00	5.5E-04	8.7E-07	2.6E-03	18.7	1.6	0.6	0.0003
2	4.2E+00	1.1E-04	7.4E-08	2.2E-04	11	7.7	35	0.0038
0.5	1.3E+00	9.4E-06	2.0E-09	5.7E-06	6	5	874	0.0082
0.177	3.4E-01	6.9E-07	3.9E-11	1.1E-07	3.3	2.7	23759	0.0163
0.149	1.6E-01	1.6E-07	4.3E-12	1.3E-08	2.9	0.4	31523	0.0050
0.105	1.3E-01	9.7E-08	2.0E-12	6.0E-09	2.5	0.4	66647	0.0064
0.074	9.0E-02	4.8E-08	7.2E-13	2.1E-09	1.9	0.6	285637	0.0137
0.053	6.4E-02	2.4E-08	2.6E-13	7.5E-10	1.4	0.5	666471	0.0161
finer	2.7E-02	4.2E-09	1.9E-14	5.5E-11	0	1.4	25675730	0.1082
							Cumulative Retained Surface Area [m^2/100g]	0.182
							Cumulative Retained Surface Area [m^2/kg]	**1.82**

According to a memo from Stephen Day, SRK (Reference (20)), a similar calculation for the Project laboratory data indicate that the material in the humidity cells has a surface area to mass ratio generally between 5.7 and 16 [m^2/kg], with an median of 9.4 [m^2/kg]. Using the average value from the AMAX piles and the distribution of humidity cell values, the field-scale waste rock is conservatively estimated to contain between 11% and 30% of the surface area per unit mass of the humidity cell rock (90% confidence interval). The particle size factor is represented by the fitted triangular distribution shown in Figure 8-9.

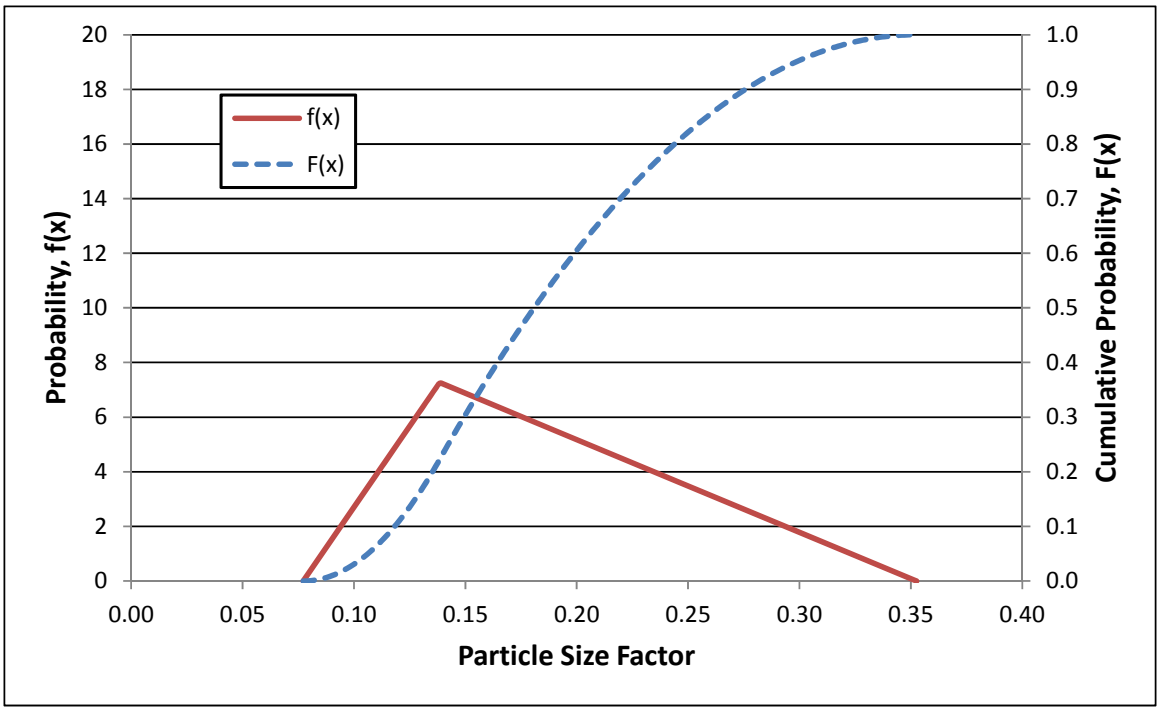

Figure 8-9 Distribution for the Particle Size Factor

8.2.4 Temperature Factor

The expected temperature factor can be calculated based on site temperatures, laboratory temperatures, the activation energy for the waste rock and the Arrhenius equation, shown in Equation 8-16.

$$k = Ae^{-E_a/RT} \qquad \textbf{8-16}$$

In the Arrhenius equation, k is the rate at which a given reaction is taking place. A is a constant multiplier but is not important here because it will drop out in the following analysis. E_a is the activation energy [kJ/mol], R is the universal gas constant of 0.008314 [kJ/mol/K] and T is the temperature [°Kelvin]. The temperature factor is calculated by dividing the reaction rate on site by the reaction rate in the lab.

$$k_T = \frac{k_{Field}}{k_{Lab}} = \frac{e^{-E_a/RT_{Field}}}{e^{-E_a/RT_{Lab}}} = e^{\frac{E_a}{R}\left[\frac{1}{T_{Lab}} - \frac{1}{T_{Field}}\right]} \qquad \textbf{8-17}$$

Because the laboratory temperature (T_{Lab} = 20°C) and universal gas constant R are known, the distribution for the temperature factor is dependent on the distributions for the site temperature T_{Field} and the activation energy E_a. By assuming that the average site temperature in the Category 1 Waste Rock Stockpile is equal to the average temperature in any given year, a normal distribution for T_{Field} can be developed from daily temperature data

from 1981-2000 at locations within 10 miles of the Mine Site (National Weather Service data). Figure 8-10 shows the annual average temperature PDF and CDF.

For the activation energy, the literature-reported range of activation energy for pyrrhotite is between 47 and 63 kJ/mol (Reference (21)). Figure 8-11 shows the PDF and CDF for this range of activation energies.

At the mean site temperature (2.004°C) and the mean activation energy (55 kJ/mol) from the distributions shown in Figure 8-10 and Figure 8-11, the resulting temperature factor k_T equals 0.228. This compares with a temperature factor of 0.3 used in the deterministic modeling, which was calculated using a temperature of 3°C and assumed activation energy of 47 kJ/mol.

A 5000-run Monte Carlo analysis was performed to assess the combined effects of uncertainty in mean site temperature and activation energy, using the input distributions shown in Figure 8-10 and Figure 8-11. The resulting distribution for k_T from the input distributions is shown in Figure 8-12.

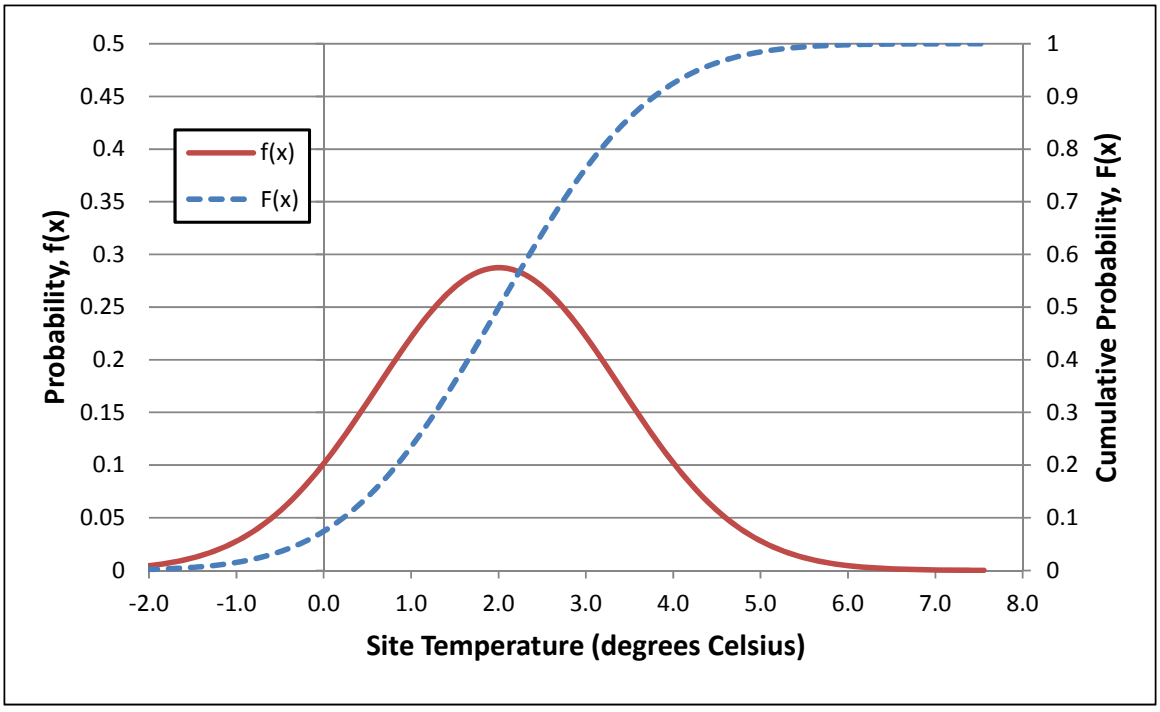

Figure 8-10 Distribution for the Site Temperature

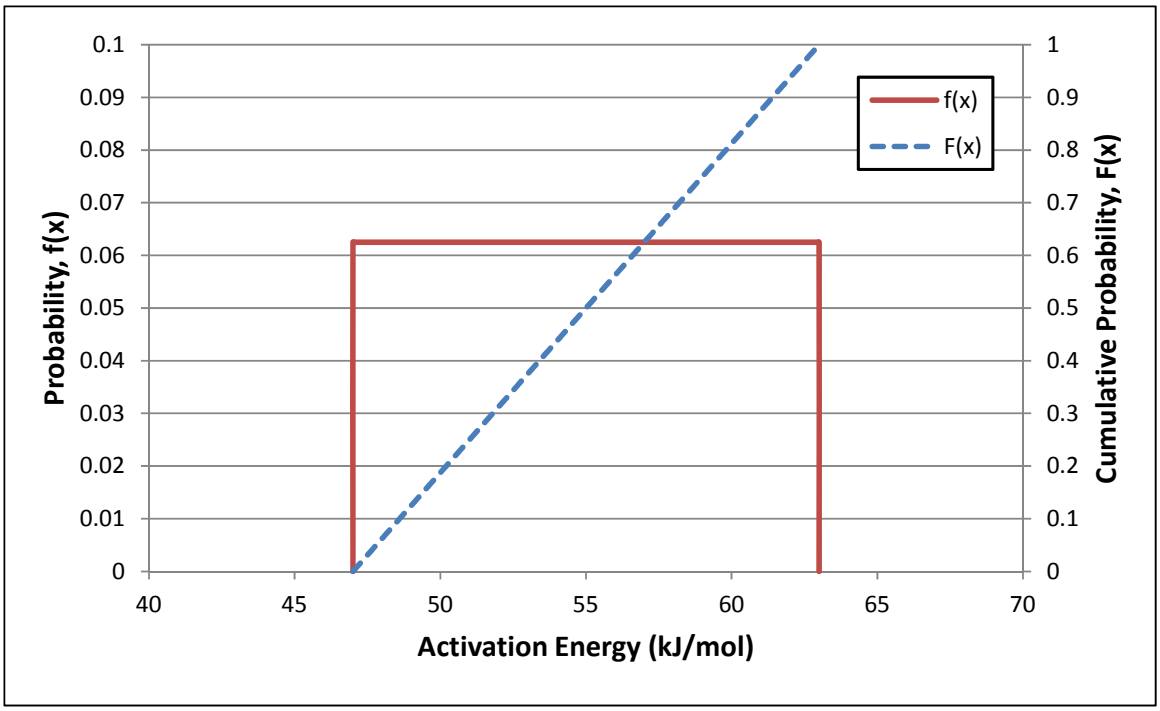

Figure 8-11 Distribution for the Activation Energy

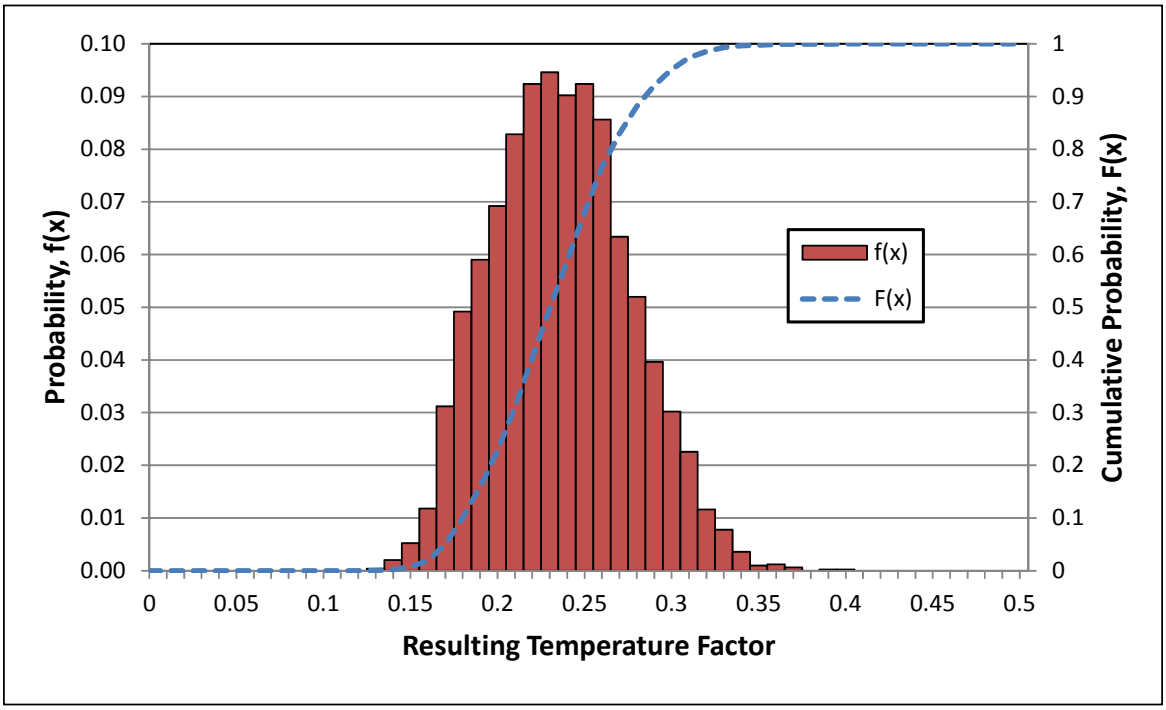

Figure 8-12 Simulated Temperature Factor Distribution

8.2.5 Acidity Factor

As discussed in Section 8.2.1, an additional scaling factor is necessary to represent the change from nonacidic to acidic conditions for Duluth Complex Category 2/3 and 4 waste rock and ore because the majority of the data from the humidity cell tests represent nonacidic conditions. Category 1 waste rock is not expected to experience acidic conditions under any circumstance. For Virginia Formation waste rock, acidic conditions are represented in the humidity cells and are assumed to begin immediately upon exposure of the rock to the atmosphere. No additional correction for acidic conditions is necessary for modeling the behavior of Virginia Formation waste rock.

The effects of the onset of acidic conditions in the humidity cell tests include an increase in the rate of sulfide oxidation (expressed by the acidity factor k_{pH}) and a change in the concentration caps for all constituents. Concentration caps for nonacidic and acidic conditions will be discussed separately in Section 8.3.

The method for calculating nonacidic constituent release rates for Duluth Complex Category 2/3 and 4 waste rock and ore described in Section 8.1 (and outlined in Large Table 2) relies primarily on a determination of the rate of sulfide oxidation as expressed in sulfate release rates. Nonacidic release rates of most other constituents are scaled to sulfate release rates. An increase in sulfate release from the onset of acidic conditions, therefore, will cause increases in the calculated release rates for most other constituents.

As shown in Table 8-2, the acidity factor has only been applied to the Duluth Complex Category 2/3 and 4 waste rock and ore once acidic conditions occur. The time from initial exposure to the atmosphere to the onset of acidic conditions is a function of the relative weathering rates of various minerals and the capacity of the alkalinity released by the silicate minerals in the waste rock to buffer the acidity released by the sulfide minerals. The lag time until acidic conditions become prevalent is indicated by laboratory kinetic testing data.

As discussed in Attachment A, data from the long-term MDNR tests on Duluth Complex rock provide information on the length of time from initiation of the tests until sulfate release increases (indicating increased oxidation rates and acidic conditions) and the magnitude of the increase over nonacidic sulfate release rates. Table 3 of Attachment A for a list of the specific MDNR reactors used in this analysis. These data were used to develop probability distributions for the time to onset of acidic conditions (Figure 8-13) and the ratio of acidic to nonacidic sulfate release (acidity factor k_{pH}) as discussed below. This method allows for the simulation of effective acidic release rates for the majority of constituents.

Because the material in the long-term MDNR reactors is, by design, significantly finer than that in the NorthMet humidity cells, it has been suggested that the sulfate release under acidic conditions in the MDNR reactors may be different than what is expected for the NorthMet humidity cells. The finer-grained material has a much higher ratio of exposed surface area to the total mass of sulfide minerals which could potentially result in more-rapid

consumption of reactive minerals under acidic conditions. This effect could cause a larger increase in sulfate release rates (relative to nonacidic conditions) and a faster depletion and decay in sulfate release rates over time. The application of this analysis to depletion is discussed in Section 9.4.

As discussed in Attachment A, the empirical acidity factor in the MDNR reactors ranges from 4 to more than 32, with a median value of 9.0. For the NorthMet humidity cells that have reached acidic conditions, the acidity factor ranges from 1 (no increase) to more than 5 (Figure 3 of Attachment A), with a median value of 2.3. The range of sulfur contents is comparable between the two data sets. The MDNR reactors used in this analysis are identified in Table 3 of Attachment A, the NorthMet humidity cells are identified in Large Table 1.

By combining the two data sets (17 MDNR reactors and 8 NorthMet humidity cells), a distribution for the acidity factor can be established (Figure 8-14). A beta distribution has been fit to the full data set, with a minimum value of 1.0 (the lowest observed value), a median value of 5.5, and a maximum value of 32.4 (the highest observed value). This distribution reflects the wide uncertainty over the true acidity factor for sulfate release rates, but encompasses the full range of available data. In the probabilistic modeling the value generated from this distribution is applied to all Duluth Complex Category 2/3, 4, and ore waste and wall rock throughout each realization. The acidity factor has also been correlated to parameters for modeling long-term decay as discussed in Section 9.4.

Note that the distributions shown for the time to onset of acidity in Figure 8-13 were developed from laboratory data and reflect laboratory temperatures. The onset of acidic conditions is expected to take longer for waste rock and wall rock at field temperatures. This is reflected in the model by dividing the time to onset of acidity by the simulated temperature factor for Duluth Complex Category 2/3,4 and ore wall rock (Section 9.4). Waste rock in the Category 2/3 and Category 4 Waste Rock Stockpiles and the Ore Surge Pile is assumed to react at laboratory temperatures, so no correction is needed.

For those constituents for which the nonacidic Duluth Complex Category 2/3 and 4 waste rock and ore release rates are based on humidity cell release ratios (ex. calcium) or are not scaled to sulfate (ex. boron), the appropriate data from the humidity cell tests under acidic conditions (i.e., Condition 3 in Large Table 1) were used to develop probability distributions. Simulation of the release of these constituents proceeds as discussed in Section 8.1.

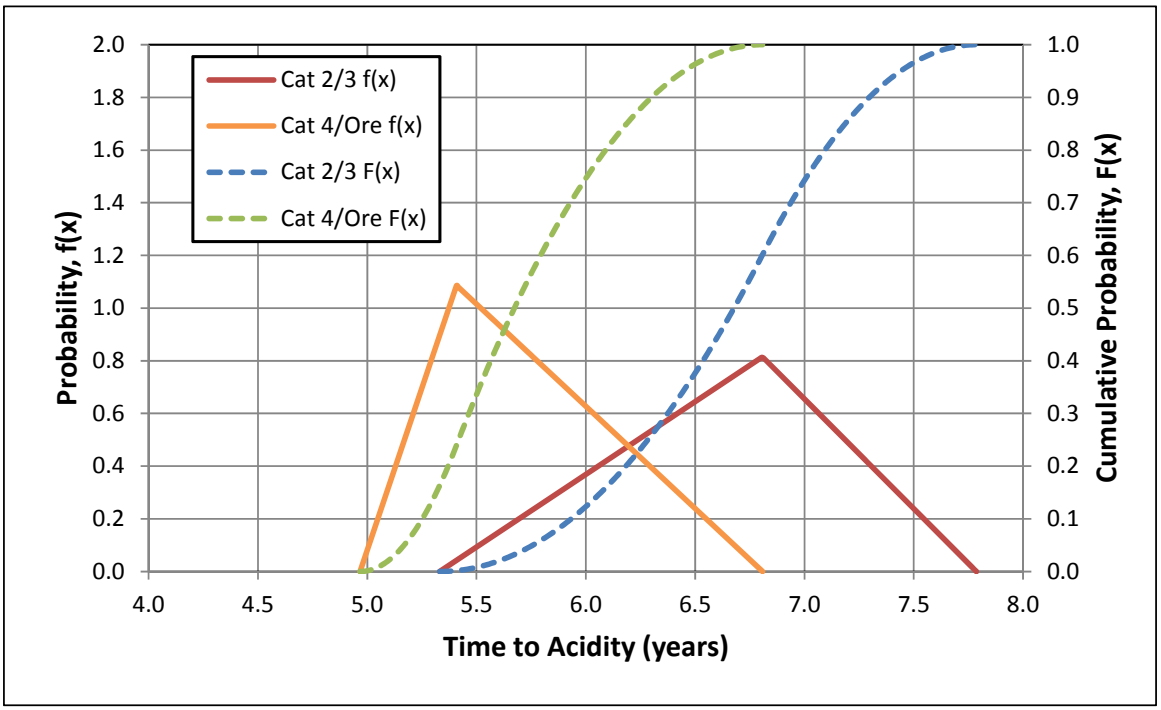

Figure 8-13 Distributions for the Time to Onset of Acidity

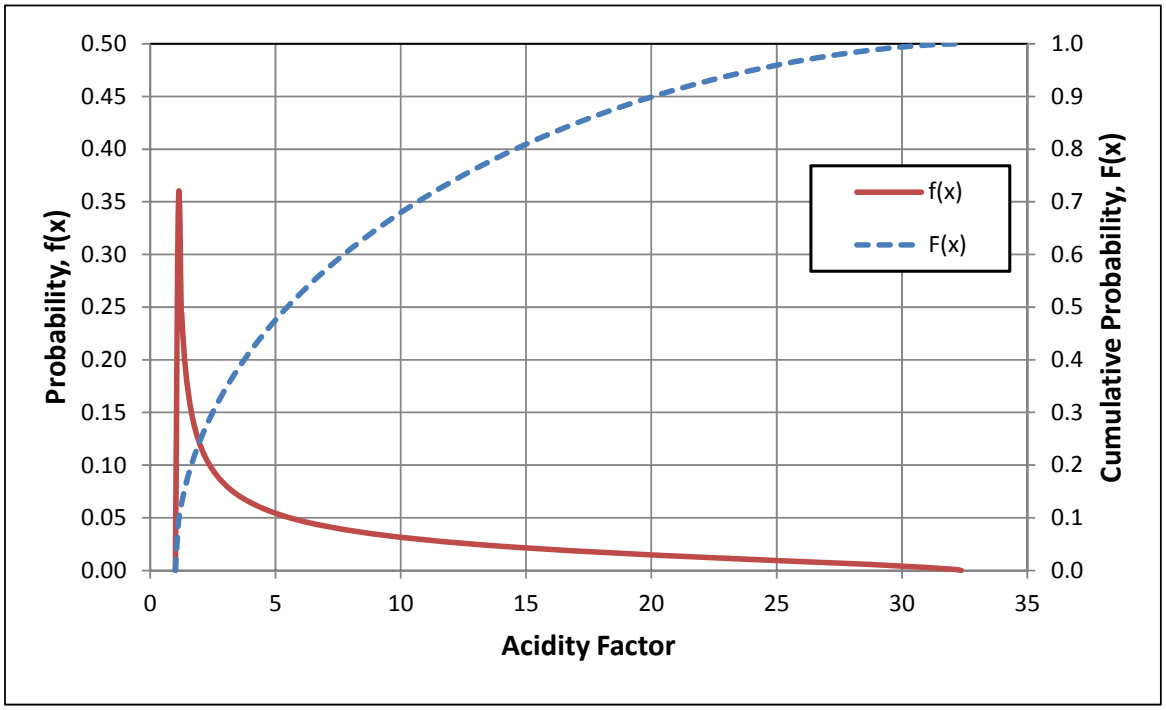

Figure 8-14 Distribution for the Acidity Factor

8.2.6 Composite "Bulk" Scale-Up Factor

Table 8-4 is a summary of the probability distributions presented in the above sections.

Table 8-4 Input Variables for Scale-Up Factor Modeling

Variable	Description	Units	Dist. Type	Mean or Mode	Std. Dev.	Max	Min
k_c	Contact Factor	--	Triang.	0.5	--	0.9	0.1
k_s	Size Factor	--	Triang.	0.138	--	0.077	0.353
T_{Lab}	Lab Temp	°C	Const.	20	--	--	--
T_{Field}	Site Temp	°C	Normal	2.004	1.388	--	--
E_a	Activation Energy	kJ/mol	Uniform	--	--	63	47
R	Gas Constant	kJ/mol/K	Const.	8.314×10^{-3}	--	--	--
k_{pH}	Acidity Factor	--	Beta	8.2	7.48	32.4	1.01

Using the GoldSim simulation software a 5000-run Monte Carlo analysis was performed to calculate a composite "bulk" scale-up factor distribution using the input distributions detailed in Table 8-4. The sub-factors were combined as described in Table 8-2 to develop three separate bulk scale-up distributions: (1) water contact, particle size, and temperature effects; (2) water contact and particle size effects; (3) water contact, particle size, and acidity effects. The resulting distributions are shown in Figure 8-15 and Figure 8-16 and summarized in Table 8-5.

Table 8-5 Results from Monte Carlo Analysis of Bulk Scale-Up Factors

Cumulative Probability	SB_1 $k_c*k_s*k_T$	SB_2 k_c*k_s	SB_3 $k_c*k_s*k_{pH}$
0.05	0.0076	0.035	0.070
0.10	0.0096	0.044	0.093
0.25	0.014	0.062	0.17
0.50	0.020	0.088	0.45
0.75	0.028	0.12	1.1
0.90	0.037	0.16	1.9
0.95	0.043	0.18	2.5
Average	0.022	0.095	0.77

Note that these bulk scale-up factors represent the apparent "bulk" scaling for constituent release in stockpile drainage. Through the application of the contact factor additional constituent mass is modeled as being released and stored with the waste rock until inundation.

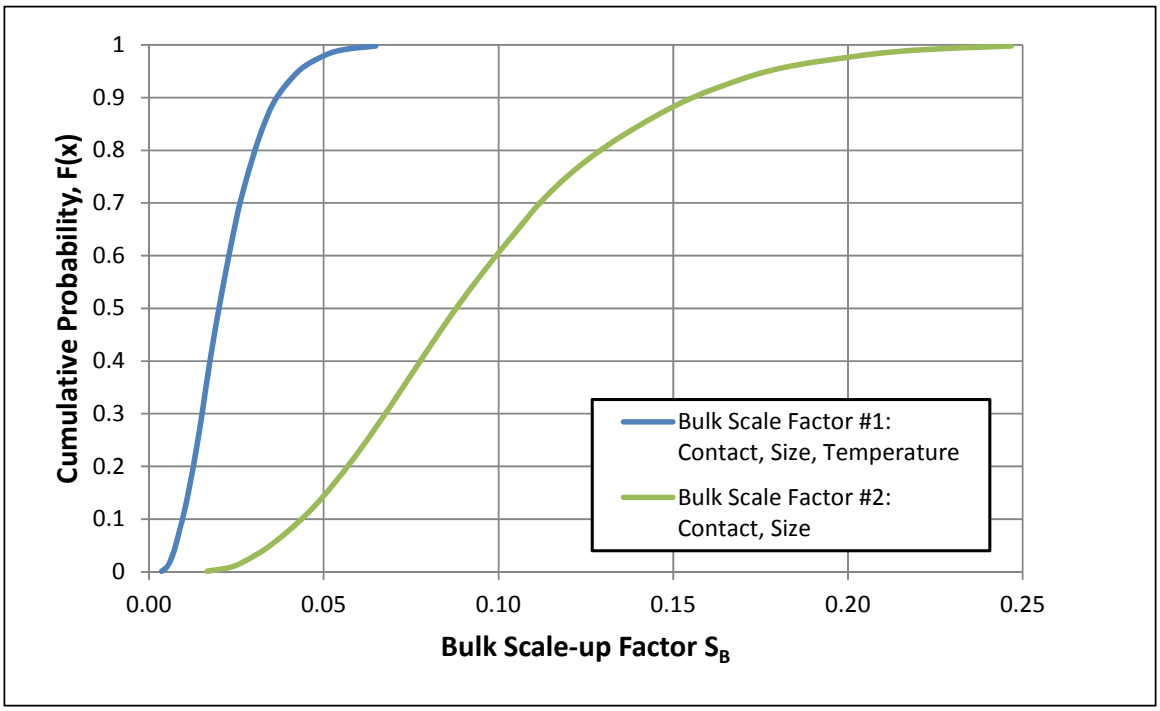

Figure 8-15 Bulk Scale Factor Distributions (nonacidic)

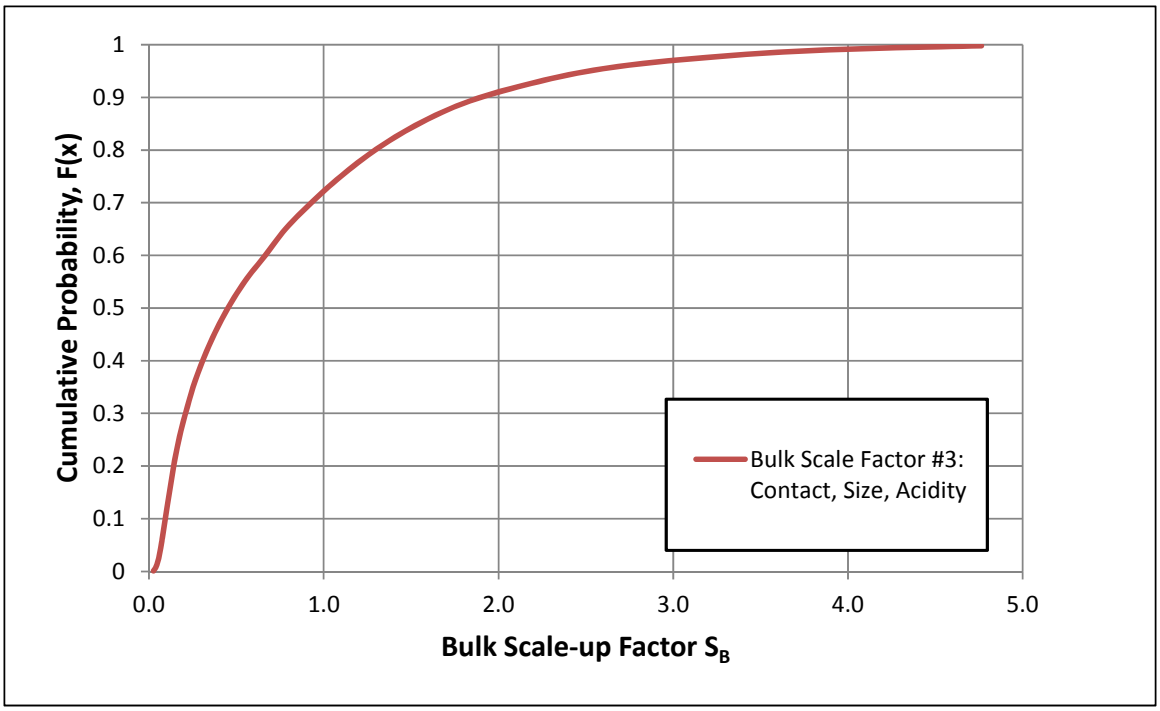

Figure 8-16 Bulk Scale Factor Distribution (acidic)

8.2.7 Scale-Up Model Verification

The proposed scale-up approach was tested for acidic conditions by modeling the AMAX test piles and comparing the modeled and observed sulfate release rates (Reference (22)). The model inputs described in Sections 8.2.2 through 8.2.5 were used as described (including the acidity factor), with several adjustments to reflect the characteristics of the AMAX piles:

- sulfate release rates from humidity cell tests on AMAX rock (Reference (19)) were analyzed with the linear regression method described in Section 8.1.1.2
- the water contact factor distribution was shifted higher to reflect greater water contact in the small test piles
- the particle size factor distribution was developed from documented particle size distribution data for the AMAX field and laboratory samples (References (19) and (23))
- the temperature factor was used (correcting for lower field temperatures) even though the piles were modeled as being acidic to reflect the small heat-retention capacity of the small piles

The lab to field scale-up approach presented here provided good estimates of the median annual observed sulfate release without any calibration. In all cases, the high end of the distribution of modeled sulfate release was well above the highest single-year observed sulfate release. The results of this test model for the four acidic AMAX piles are shown in Table 8-6.

Table 8-6 AMAX Test Model Results for Sulfate Release

Pile	Avg S [%]	Acidic Period	Observed Sulfate Release [mol/yr](1)			Modeled Sulfate Release [mol/yr](2)		
			Min.	Med.	Max.	P5	Med.	P95
FL3	0.64	1984-1989	1100	1200	1700	200	1200	4300
FL4	0.64	1984-1989	740	1300	1500	240	1400	6600
FL5	1.41	1978-1993	1200	2500	3900	380	2200	9100
FL6	0.79	1980-1993	1100	1800	4200	430	2300	6500

(1) Source: Reference (19)
(2) Source: Reference (22)

A similar test of this scale-up approach for nonacidic conditions is not possible, because there is no well-characterized field analog for low-sulfur Duluth Complex waste rock (analogous to Category 1 waste rock). All of the AMAX test piles became fully acidic after less than 13 years and have sulfur content that is at least an order of magnitude higher than the average for the Category 1 waste rock. Even during the initial period of nonacidic seepage it is likely that an unknown fraction these test piles experienced local acidic conditions, resulting in elevated sulfate release relative to "true" nonacidic conditions. The other existing locations of field-scale stockpiled Duluth Complex rock, at the former LTVSMC Dunka Mine, were not designed as research stockpiles and have uncertainties in their composition (including sulfur content) and drainage pathways. In addition, there are no directly-comparable humidity cell tests on the Dunka Mine waste rock that allow the development of a lab to field comparison.

The lab to field scale-up method presented here has been developed from theoretical geochemical principles and tested against data from test piles of acidic Duluth Complex rock. This test, along with published work (Reference (18)) corroborates the conceptual model being applied for all of the Project waste rock stockpiles using the appropriate sub-factors as described in Table 8-2. It is the professional opinion of PolyMet and its consultants that this probabilistic method provides a realistic range of the lab to field scale-up ratio for both acidic and nonacidic conditions. The model results in Table 8-6 show that this method

produces conservatively high estimates of sulfate release with respect to the maximum observed release at field scale under acidic conditions.

8.2.8 MDNR Method for Category 1 Waste Rock Stockpile Scale-Up

MDNR has proposed an alternate method for simulating the lab to field scale-up factor, based on observed sulfate leaching data from stockpiles at the Dunka Mine (Reference (24)). This method is specific to the Category 1 Waste Rock Stockpile and does not replace the composite scale-up factor method described above for the Category 2/3 and Category 4 Waste Rock Stockpiles, the Ore Surge Pile and the wall rock.

The MDNR scale-up method uses laboratory sulfate release rates as a function of sulfur content, developed from MDNR reactor tests on rock from the Dunka Mine blast holes (Reference (23)). The average sulfate release for each MDNR reactor is determined for the first 71 weeks of testing (excluding the initial 5 weeks to remove flushing effects), and duplicate reactors are averaged. The laboratory release rates from the reactor tests are scaled to the equivalent humidity cell release rates by a factor of 0.34 (the finer material in the MDNR reactors releases sulfate about three times faster than the standard ASTM humidity cells). By this method 17 average sulfate release rates as a function of sulfur content [mass SO_4/(mass rock * time * %S] are determined.

The MDNR scale-up method uses field sulfate release rates as a function of sulfur content, developed from seepage water quality data from Dunka Mine stockpiles. The seeps and time periods used in this analysis are shown in Table 8-7, and represent the entire period of flow and concentration data for each seep (Reference (25)). Annual sulfate release is determined from the concentration and flow data for each seep. By this method 42 annual sulfate release rates as a function of sulfur content [mass SO_4/(mass rock * time * %S] are determined.

MDNR has calculated a range of “bulk” scale-up factor values as the ratio between annual field sulfate release and the average laboratory sulfate release, using all possible combinations of field and laboratory release rates. The calculated values are shown as the histogram in Figure 8-17, and a beta distribution fit to the observed values is shown by the lines in Figure 8-17. Because this distribution represents the temporal variability in the field data and spatial variability represented by the laboratory data, a value for this distribution is re-generated for each year of the model.

Table 8-7 MDNR Dunka Stockpile Data used for Scale Factor Analysis

Seep	Avg S [%]	Duration [years]	Time Period
Seep X	0.24	3	1990 - 1992
Seep 1	0.24	7	1986 - 1992
EM8	0.31	13	1979 - 1991
W4	0.35	12	1980 - 1991
W1D	0.97	7	1986 - 1992

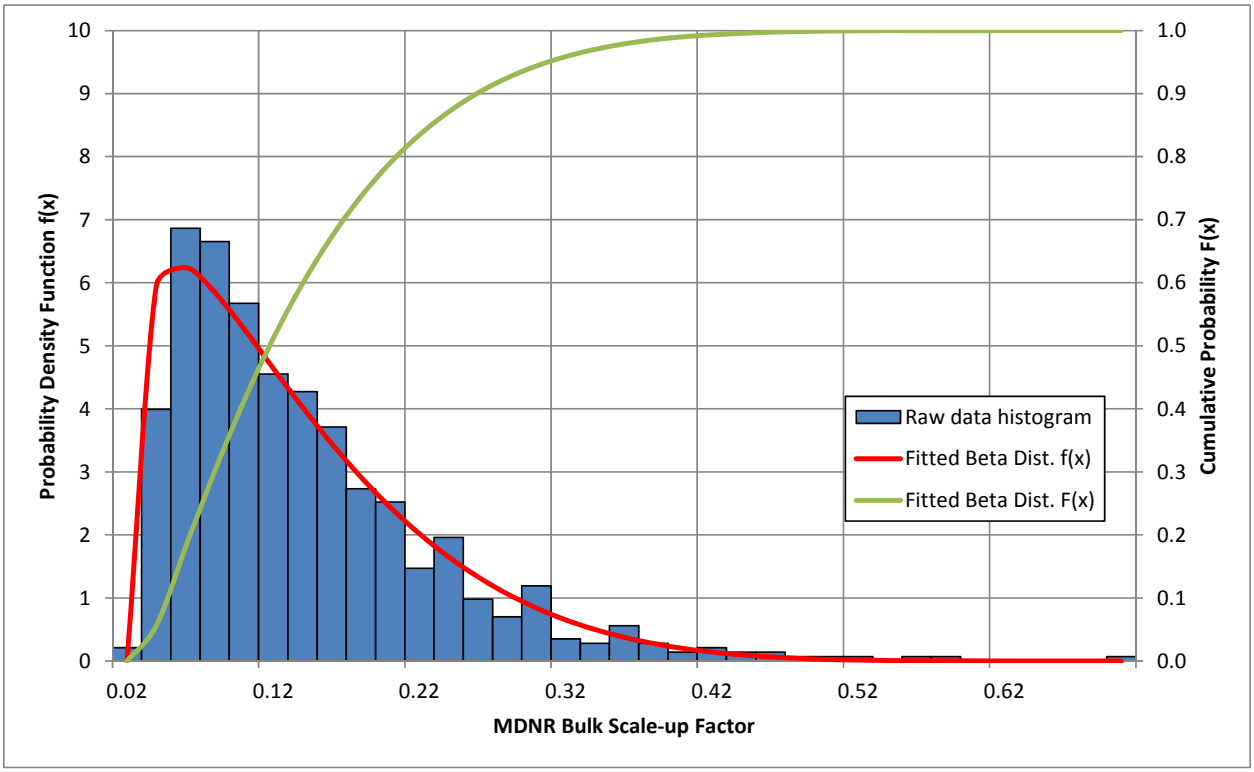

Figure 8-17 MDNR Bulk Scale-up Factor Distribution

As directed by the Co-lead Agencies, this distribution for the Category 1 Waste Rock Stockpile lab to field scale-up has been used for initial modeling of potential project impacts. It is the professional opinion of PolyMet and its consultants that this distribution is overly conservative.

8.3 Concentration Caps

An updated methodology for interpreting the anticipated concentration caps for waste rock is described in Section 4.1.3.1 and Large Table 11, based on the methods presented in Attachment A. Attachment A includes leachate chemistry data from full-scale waste rock stockpiles that have similar characteristics to NorthMet Category 2/3 and Category 4 waste rock and have observed primarily acidic conditions (Whistle and Vangorda mines). These data have been used to complement the MDNR data from the AMAX test piles to develop concentration caps for the waste rock stockpiles.

Concentration caps are used in the water quality modeling to limit concentrations of dissolved metals in effluent from the stockpiles. The term "solubility limits" has been used to describe this modeling input in previous documents; in this and all future documentation the term used will be "concentration caps". The maximum concentrations of dissolved metals observed under field conditions result from multiple competing geochemical processes such as mineral precipitation and dissolution, sorption, desorption, and solubility of secondary minerals. The concentration cap, therefore, is primarily an empirical method for modeling the combined effect of these complex processes in field-scale waste rock stockpiles. The proposed modeling methods in this section make use of available laboratory and field data as well as thermodynamic modeling.

8.3.1 Category 1 Waste Rock

The probability distributions for the Category 1 waste rock concentration caps are shown in Large Table 12 and Large Figure 23 through Large Figure 27.

Differing methods for developing concentration caps for the Category 1 waste rock have been proposed by PolyMet and MDNR (Reference (26). Both methods are described below and the differences are identified; as directed by the Co-lead Agencies, only the values for the method approved by the Co-lead Agencies are shown in Large Table 12 for use in initial modeling of potential project impacts. It is the professional opinion of PolyMet and its consultants that the concentration caps approved by the Co-lead Agencies are overly conservative for many constituents. However, engineering controls have been put in place for the Project based on these results to minimize environmental impacts. Adaptive management is also in place to enable adjustments to water treatment or mitigation needs as the Project develops and monitoring data can be obtained and analyzed.

8.3.1.1 Category 1 Waste Rock Stockpile pH

The long-term pH in the Category 1 Waste Rock Stockpile will be determined by the interaction between the oxidation of sulfide minerals (producing acidity), the weathering of silicate minerals (producing alkalinity), and the additional acidity from dissolved carbon dioxide gas. This interaction will result in the precipitation of secondary minerals (calcite, ferrihydrite, gypsum, kaolinite, magnesite, silica) until a stable pH is attained. While the

Category 1 waste rock is not expected to become acidic, variations in pH in the circum-neutral range (pH 6-8) can cause significant changes in porewater concentrations for some constituents. The uncertainty in the stockpile pH is therefore an important consideration in assessing the uncertainty in the concentration caps assigned to the Category 1 waste rock.

Geochemical modeling of the Category 1 waste rock using Geochemist's Workbench was performed to estimate the stable pH in the porewater at various sulfur contents. The sulfate release rate was defined as a function of sulfur content according to the relationship developed from NorthMet humidity cell data (Section 8.1.1.2). The release rates for the major cations (Ca, Mg, Na) were defined as the average stable (weeks 174-188) release from the six humidity cells with the lowest sulfur contents. These release rates are assumed to represent silicate weathering in the near absence of sulfide minerals, and were not varied with increasing sulfur content or with CO_2 concentration. The partial pressure of carbon dioxide (which controls the amount of CO_2 dissolution) was defined for two different modeling cases: atmospheric CO_2 (log pCO_2 = -3.4, equivalent to 398 ppm) and enriched CO_2 (log pCO_2 = -2.0, equivalent to 1%). The modeled pH for both cases is shown in Figure 8-18.

The stable pH data from laboratory tests on oxidized Category 1 waste rock (Section 4.1.3.1) are also shown in Figure 8-18. These data closely follow the upper line (log pCO_2 = -3.4) and confirm the modeling assumptions and methodology for stable pH in the Category 1 waste rock porewater under atmospheric CO_2 conditions. There is less variation in the empirical data than in the model results, with the pH for lower- and higher-sulfur samples plotting closer to the mean pH than predicted by the geochemical model.

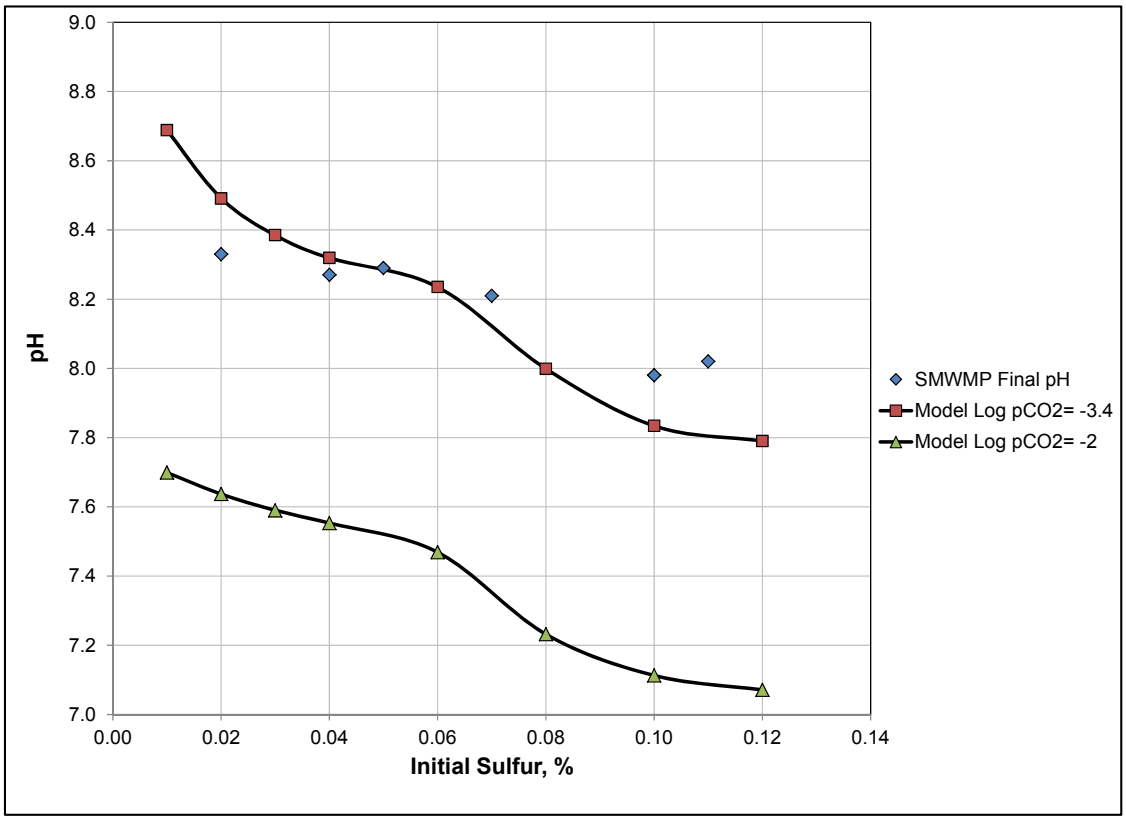

Figure 8-18 Modeled Category 1 Waste Rock pH

Enriched CO_2 conditions have been inferred from MDNR geochemical modeling of taconite tailings (which contain carbonate minerals), and are observed in existing waste rock stockpiles for rock that contains carbonate minerals; carbonate minerals provide a source of carbon for CO_2 enrichment. The Duluth Complex Category 1 waste rock, however, is almost devoid of carbonates and will not in itself be a source of elevated CO_2. Other potential sources of carbon to the Category 1 Waste Rock Stockpile porewater are organic material and the resulting microbial communities present in soil growing layers, which give off CO_2 during respiration. Although the Category 1 Waste Rock Stockpile will not be placed on a constructed liner, the amount of organic material in the stockpile foundations is expected to be minimal due to geotechnical requirements. The vegetated cover that will be placed on the Category 1 Waste Rock Stockpile in closure, however, will have an active growing layer extending around the entire outer shell of the stockpile. Because all infiltration water in closure will pass through this vegetated layer, it is likely that elevated CO_2 levels will exist for some unknown depth below the cover soils. Whether CO_2 enrichment will persist throughout the interior of the stockpile in the absence of carbonate minerals is unknown. Geochemical mechanisms that would tend to reduce CO_2 concentrations (and therefore increase pH) include carbon sequestration in the formation of carbonate mineral precipitates.

To reflect the uncertainty in the long-term behavior of the Category 1 Waste Rock Stockpile porewater, pH is modeled as a probabilistic *input* to the water quality model for the Category 1 Waste Rock Stockpile. It is conservatively assumed that CO_2 enrichment persists to some degree throughout the entire stockpile.

The proposed distribution for the average pH is defined from the range shown in Figure 8-19; a triangular distribution with a minimum value of 7.1, a maximum value of 7.7, and a most likely value of 7.4. This uncertainty reflects the potential for processes that may increase or decrease the mean pH from the modeled value of 7.4, which represents CO_2 enrichment at the mean sulfur content.

The method approved by the Co-lead Agencies for modeling the average stockpile pH is also shown in Figure 8-19 as a uniform distribution between 7.0 and 7.5. It is the professional opinion of PolyMet and its consultants that this range is overly conservative.

In the probabilistic water quality modeling, the randomly-generated pH for each realization is used to determine the concentration caps for some constituents using the methods discussed below.

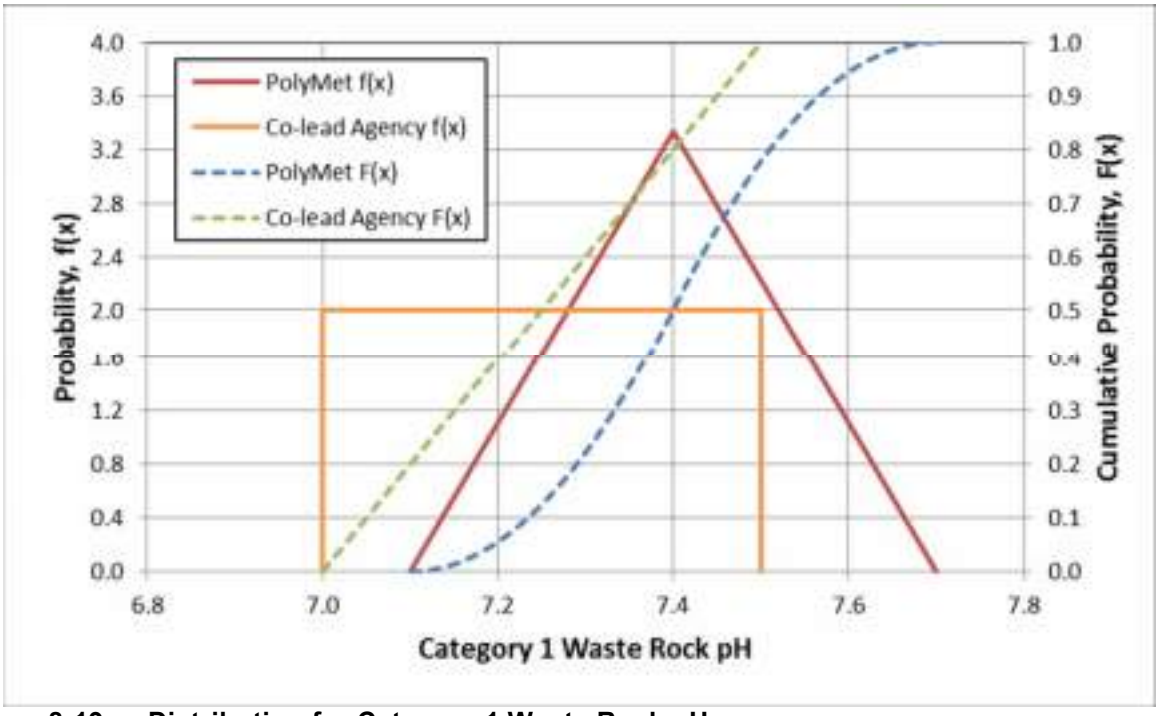

Figure 8-19 Distribution for Category 1 Waste Rock pH

8.3.1.2 Concentration Caps from Category 1 SMWMP Study

The concentration cap study (SMWMP) described in Section 4.1.3.1 provides empirical data on appropriate concentration caps for many constituents for the Category 1 waste rock. For

constituents that were shown to be constrained in the study probability distributions for the solubility limits have been determined based on the range of concentrations observed for the last cycle of leach testing. Although the stable pH in the concentration cap study was higher than the range shown in Figure 8-19 (stable pH of approximately 8), concentration caps for most constituents are not expected to be sensitive to a possible pH shift from 8 to 7. Those metals that are sensitive to pH in this range are discussed in Section 8.3.1.5.

The Category 1 concentration caps approved by the Co-lead Agencies do not use the results of the SMWMP study because of concerns about the conditions and characteristics of the study. The concentration caps discussed in the remainder of this section are for the PolyMet method only.

The constituents that have concentration caps developed from the SMWMP study include silver, aluminum, alkalinity, arsenic, beryllium, iron, potassium, sodium, lead, antimony, thallium, and vanadium. The range of maximum concentrations is used to define the uncertainty in the concentration caps (uniform distribution); the specific values are shown in Large Table 12. Although selenium was not constrained in the study, the selenium concentration was shown to be directly correlated to the observed sulfate concentration. The selenium concentration cap is defined from the modeled sulfate concentration using a regression relationship from the final leaching cycle in the concentration cap study (Equation 8-18).

Selenium: $$Se\left(\frac{mg}{L}\right) = 6.35 \times 10^{-6} \cdot SO_4\left(\frac{mg}{L}\right) + 0.0020 \qquad \textbf{8-18}$$

None of these constituents are expected to have concentration caps that will significantly increase due to pH changes from the range in the Category1 study to the range modeled in Figure 8-19.

As discussed in Section 4.3.2, the sulfur content of the Category 1 Waste Rock Stockpile will vary locally across the range of sulfur contents used in the solubility limit study, but the expected average sulfur distribution of the stockpile as a whole will be more limited and will tend towards the middle of the range of sulfur contents in the solubility limit study. In developing probability distributions for the solubility limits, however, no attempt has been made to weight the available data by sulfur content or otherwise bias the determined solubility towards the expected average sulfur content conditions in the field. This results in conservatively wide ranges for the modeled solubility limits relative to the likely average field conditions.

8.3.1.3 Concentration Caps from Modeled Mineral Solubility

Some of the constituents that were not constrained in the SMWMP study are expected, based on thermodynamic considerations, to be controlled by the solubility of secondary minerals. The limiting equations are shown in Equations 8-19 through 8-23 below (adapted from

Attachment A). The concentration cap equations for these constituents are identical for the PolyMet method and the method approved by the Co-lead Agencies.

For sulfate (gypsum solubility), calcium (gypsum solubility), and magnesium (calcium-magnesium ratio), the concentration cap for each realization in the probabilistic model is calculated based on the release rates of the referenced constituents (converted to molar release rates). A single value for the solubility limit is generated for each model realization. This method results in a major-ion charge balance at the concentration cap.

For barium (barite solubility) and fluoride (fluorite solubility, Reference (27)), the concentration cap for each time step in the probabilistic model is calculated based on the current concentrations of sulfate and calcium, respectively (i.e., the applicable concentration cap for barium is calculated based on the current sulfate concentration).

Sulfate:
$$SO_4\left(\frac{mg}{L}\right) = 1294 \times \frac{[Mg + .5Na + .5K]\left(\frac{mmol}{kg \cdot week}\right)}{Ca\left(\frac{mmol}{kg \cdot week}\right)} + 1760 \qquad \textbf{8-19}$$

Calcium:
$$Ca\left(\frac{mmol}{L}\right) = \frac{2SO_{4,Cap}\left(\frac{mmol}{L}\right) + Alk_{Cap}\left(\frac{mmol}{L}\right) - K_{Cap}\left(\frac{mmol}{L}\right)}{2 + \frac{[2Mg + Na]\left(\frac{mmol}{kg \cdot week}\right)}{Ca\left(\frac{mmol}{kg \cdot week}\right)}} \qquad \textbf{8-20}$$

Magnesium:
$$Mg\left(\frac{mg}{L}\right) = Ca_{Cap}\left(\frac{mg}{L}\right) \cdot \left[\frac{Mg}{Ca}\right]\left(\frac{mg}{kg \cdot week}\right) \qquad \textbf{8-21}$$

Barium:
$$\log\left[Ba\left(\frac{mg}{L}\right)\right] = -0.32 \cdot \log\left[SO_4\left(\frac{mg}{L}\right)\right] - 0.87 \qquad \textbf{8-22}$$

Fluoride:
$$F\left(\frac{mol}{L}\right) = \sqrt{\frac{8.91\text{x}10^{-11}\left(\frac{mol^3}{kg^3}\right)}{Ca\left(\frac{mol}{L}\right)}} \cdot \left[\rho_{water}\left(\frac{\text{kg}}{L}\right)\right]^{1.5} \qquad \textbf{8-23}$$

None of these constituents are expected to have concentration caps that are sensitive to changes in pH in the range of 7 to 8.

8.3.1.4 Concentration Caps from Analog Site Data

For some of the constituents that were not constrained in the SMWMP study, concentration cap data are available from the analog sites detailed in Attachment A. For both boron and chromium, the available data are from the Whistle Mine site under nonacidic conditions (pH 6.9 to 8.1). These constituents were not detected in the data set; the detection limit is used as the concentration cap. No concentration cap is assumed for chloride.

The concentration caps for the method approved by the Co-lead Agencies use data from the Whistle Mine site under nonacidic conditions for several additional constituents: arsenic, lead, and vanadium. These constituents were not detected in the Whistle Mine data set; the detection limit is used as the concentration cap. It is the professional opinion of PolyMet and its consultants that these values are overly conservative.

The concentration caps for the method approved by the Co-lead Agencies use data from samples collected from Dunka Mine Stockpiles for the constituents: silver, beryllium, and thallium. For these constituents, a single sampling event in May 2006 (Reference (28) Appendix C) is used to derive the concentration cap, with the concentration cap being equal to the maximum observed value (or detection limit, where there were no detected values) times a factor of 10, an estimate of the maximum concentration based on differences between Zn and Ni in the May 2006 sample and maximum observed concentration of those constituents in Dunka Mine seepage. For the constituent antimony, the concentration cap is defined as a uniform distribution between the highest concentration observed in uncontaminated tailings HCT and waste rock reactor experiments (lower bound) under the assumption that the highest antimony concentration is not a diluted concentration and an upper limit derived under the assumption that antimony concentrations scale similarly to sulfate concentrations between laboratory and field conditions.

8.3.1.5 Concentration Caps Dependent on pH (PolyMet method)

The modeling of pH-dependent concentration caps differ for the PolyMet method and the method approved by the Co-lead Agencies. The concentration caps discussed in the remainder of this section are for the PolyMet method only.

The constituents with concentration caps that are most likely to be sensitive to changes in the stockpile pH over the expected pH range are cadmium, cobalt, copper, manganese, nickel, and zinc.

Cadmium concentrations in the Project humidity cells are strongly correlated to zinc concentrations (Attachment A); the cadmium to zinc release ratio is used to estimate cadmium release rates (Section 8.1.2.2). This same ratio is used to simulate the concentration cap for cadmium as a function of the zinc concentration cap (below).

Copper and nickel were not constrained in the SMWMP study, and the remaining metals (cobalt, manganese, zinc) were constrained in the SMWMP study at stable pH typically at or above 8.0. Because the concentration caps for these metals are sensitive to pH changes in the range discussed in Section 8.3.1.1, a method has been developed to model the concentration cap as a function of pH. This method is based on the following assumptions:

- the Category 1 SMWMP study results represent likely concentration caps at the measured stable pH for Category 1 waste rock (if concentration caps were indicated)

- the highest observed concentrations in the AMAX test piles (0.64%S) represent likely concentration caps at a range of pH values for higher-sulfur waste rock
- the concentration-pH relationship observed in the AMAX data can be understood from thermodynamic first principles, with a slope that is dependent on the valence state of the dominant metal ions
- the same concentration-pH relationship applies to the Category 1 waste rock, but must be shifted to intersect the Category 1 SMWMP data

The relevant AMAX and Category 1 data for the PolyMet concentration cap method are shown in Large Figure 23 through Large Figure 27. The resulting concentration cap equations that are used in the probabilistic modeling are shown in Equations 8-24 through 8-28 (the equations are used for the pH range shown in Figure 8-19). For cobalt, manganese, nickel and zinc these equations have been developed by a visual fit of log-linear curves to the upper envelope of the AMAX dataset; the slope of each log-linear curve has been pre-defined based on the dominant valence state (ex. Ni^{2+} results in a log slope of -2). For copper, the polynomial equation is fit to a thermodynamic model of tenorite solubility, calibrated to match the upper envelope of the AMAX dataset.

This method provides a consistent means to develop pH-dependent concentration caps based on both AMAX and Category 1 data. Note that the SMWMP data for zinc are of the same magnitude as the AMAX data and the concentration cap relationship is not shifted. Because the SMWMP study did not indicate concentration caps for copper and nickel, the relationship for the AMAX data is used to conservatively estimate concentration caps for the Category 1 waste rock.

Cobalt:

$$log\left[\mathrm{Co}\left(\frac{mg}{L}\right)\right] = -2 \cdot \mathrm{pH} + 12.0 \ \ (for\ pH < 7.5)$$
$$log\left[\mathrm{Co}\left(\frac{mg}{L}\right)\right] = -\mathrm{pH} + 4.5 \ \ (for\ pH \geq 7.5) \qquad \textbf{8-24}$$

Copper:

$$log\left[Cu\left(\frac{mg}{L}\right)\right] = -0.1623 \cdot pH^3 + 4.412 \cdot pH^2$$
$$-39.64 \cdot pH + 116.8 \ \ (for\ 6.9 < pH < 8.2) \qquad \textbf{8-25}$$

Manganese:

$$log\left[\mathrm{Mn}\left(\frac{mg}{L}\right)\right] = -\mathrm{pH} + 6.1 \ \ (for\ pH < 8)$$
$$log\left[\mathrm{Mn}\left(\frac{mg}{L}\right)\right] = -4 \cdot \mathrm{pH} + 30.1 \ \ (for\ pH \geq 8) \quad \textbf{8-26}$$

Nickel:

$$log\left[\mathrm{Ni}\left(\frac{mg}{L}\right)\right] = -2 \cdot \mathrm{pH} + 15.0 \ \ (for\ pH < 7.3)$$
$$log\left[\mathrm{Ni}\left(\frac{mg}{L}\right)\right] = -\mathrm{pH} + 7.7 \ \ (for\ pH \geq 7.3) \quad \textbf{8-27}$$

Zinc:

$$log\left[\mathrm{Zn}\left(\frac{mg}{L}\right)\right] = -2 \cdot \mathrm{pH} + 13.7 \ \ (for\ pH < 7)$$
$$log\left[\mathrm{Mn}\left(\frac{mg}{L}\right)\right] = -\mathrm{pH} + 6.7 \ \ (for\ pH \geq 7) \quad \textbf{8-28}$$

8.3.1.6 Concentration Caps Dependent on pH (MDNR method)

The concentration caps approved by the Co-lead Agencies for the Category 1 waste rock make use of the AMAX pile data wherever possible. For this application, the AMAX data is restricted to the test piles with average sulfur contents of 0.64%. The uncertainty in the concentration cap at a given pH value (as simulated from Figure 8-19) is determined by a uniform distribution between the 95th percentile value and the maximum observed concentration at that pH. This method is used to simulate concentration caps for the following constituents: alkalinity, cobalt, copper, iron, potassium, manganese, sodium, nickel, and zinc. The bounding values from the AMAX data are presented in Large Table 12.

The concentration cap for aluminum is simulated as a function of pH according to Equation 8-29, which is based on modeled gibbsite mineral solubility over the pH range shown in Figure 8-19.

Aluminum:

$$\log\left[Al\left(\frac{mg}{L}\right)\right] = 0.909 \cdot \mathrm{pH} - 9.44 \quad \textbf{8-29}$$

Like the PolyMet method above, the method approved by the Co-lead Agencies for developing a concentration cap for cadmium is to use the cadmium to zinc release ratio and the zinc concentration cap. The method for selenium is similar, with the selenium

concentration cap equal to the sulfate concentration times the selenium to sulfate release ratio.

8.3.2 Duluth Complex Category 2/3/4 Waste Rock and Ore (nonacidic)

The concentration cap methods discussed below for the nonacidic Duluth Complex waste rock and pit walls represent the PolyMet method. For this temporary condition in the stockpiles and pit walls, the concentration caps approved by the Co-lead Agencies for the Category 1 waste rock has been used as directed by the Co-lead Agencies. The assumed pH range for this condition for the pH-dependent constituents described in Section 8.3.1.6 is 6 to 7.5 (uniform distribution), and the data from all AMAX piles is used (0.064% to 1.41% sulfur).

Sulfate concentrations for nonacidic Duluth Complex waste rock and pit walls are calculated based on the solubility of gypsum, as shown in Equation 8-19.

Where applicable, metal concentrations are constrained using AMAX test pile data (constituents include Co, Cu, Mn, Ni, Zn). The AMAX data have been shown to give a reasonable understanding of likely concentration caps for many constituents in the higher-sulfur Duluth Complex waste rock under nonacidic conditions (Attachment A). However, because of the size of the AMAX test piles and the locations from which water quality samples were collected, some samples may be affected by dilution effects from precipitation and runoff. In order to remove this influence from the data set, only observations that are in the upper 5% of the data set (for a given pH range) were used to construct probability distributions.

Within a given pH range (ex. 6.0 to 8.0), the AMAX data set was segregated into bins at 0.1 pH increments. The upper 5% of observed concentrations were extracted from each bin. These extracted observations were re-combined across the specified pH range and probability distributions were developed for the combined data set. Each of this limited set of observations is assumed to be an equally likely representation of concentration-capped conditions throughout the stockpiles.

For constituents not measured in the AMAX pile drainages, data from the Category 1 SMWMP study are used if the maximum concentration was not correlated with the sulfur content of the sample (alkalinity, Ag, Al, Fe, K, Na, Pb, Sb, Tl, V) (discussion of Category 1 study data treatment above). Whistle Mine acidic concentrations are used if the concentration cap study showed that sulfur concentrations are expected to be an important control (discussion of Whistle Mine data treatment below).

The probability distributions for the nonacidic Duluth Complex waste rock concentration caps are shown in Large Table 13 (method approved by the Co-lead Agencies) and Large Figure 28 through Large Figure 31 (PolyMet method).

8.3.3 Duluth Complex Category 2/3/4 Waste Rock and Ore (acidic)

Because the available data from the Whistle Mine are derived from groundwater wells beneath waste rock stockpiles or from leachate collected at the base of stockpiles, the data do not appear to be affected by the dilution issues evident in the AMAX data. All of the available data, therefore, can be used to represent solubility-limiting conditions at the low pH levels observed. Probability distributions have been developed for acidic conditions (all data with pH < 4.5), with the assumption that each observation is an equally likely representation of concentration-capped conditions throughout the stockpiles under acidic conditions. The probability distributions for the acidic Duluth Complex waste rock concentration caps are shown in Large Table 14 and Large Figure 32 through Large Figure 36.

8.3.4 Virginia Formation Category 4 Waste Rock

The Vangorda Mine data have been used to develop concentration caps for the Virginia Formation waste rock for all constituents except Cd, Pb and Zn, which are mineralized in the Vangorda Mine deposit and are not analogous to the NorthMet Virginia Formation waste rock. Concentration caps for these constituents (Cd, Pb, Zn) have been developed using the Whistle Mine data. The Vangorda Mine data are used in the same manner as the Whistle Mine data, with the assumption that each observation is an equally likely representation of concentration-capped conditions throughout the Virginia Formation waste rock. The probability distributions for the Virginia Formation Category 4 waste rock concentration caps are shown in Large Table 15 and Large Figure 37 through Large Figure 41.

8.4 Additional Model Parameters and Considerations

8.4.1 Depletion

The water quality model includes a check for depletion of each modeled constituent from each waste rock stockpile and category of pit wall. This ensures that any simulated very high metal leaching rates do not continue, but consume the available metals within an appropriate time frame and cease.

The weighted average content of each constituent in each waste rock type was determined from the NorthMet drill core data set, using the methods described in Section 8.1.2. The average sulfate content for each waste rock type was obtained from the Block Model. These weighted average values are considered to be deterministic in the water quality modeling and are shown in Table 8-8.

Table 8-8 Average Metal Content from Aqua Regia Data (ppm)

Constituent	Category 1 Waste Rock	Category 2/3 Waste Rock	Duluth Complex Category 4 Waste Rock	Virginia Formation Category 4 Waste Rock	Ore
Silver (Ag)	1.35E-01	2.80E-01	3.63E-01	3.61E-01	1.31E+00
Aluminum (Al)	4.07E+04	3.86E+04	4.04E+04	3.23E+04	3.84E+04
Alkalinity(1)	--	--	--	--	--
Arsenic (As)	2.47E+00	3.52E+00	1.99E+01	3.20E+01	6.92E+00
Boron (B)	7.94E+00	7.32E+00	9.16E+00	8.82E+00	5.02E+00
Barium (Ba)	4.07E+01	4.85E+01	5.57E+01	1.04E+02	4.72E+01
Beryllium (Be)	2.43E-01	2.66E-01	6.15E-01	5.77E-01	1.81E-01
Calcium (Ca)	2.22E+04	2.18E+04	1.79E+04	5.93E+03	2.11E+04
Cadmium (Cd)	4.19E-01	4.59E-01	7.34E-01	1.42E+00	9.72E-01
Chlorine (Cl)(1)	--	--	--	--	--
Cobalt (Co)	4.83E+01	4.99E+01	6.05E+01	2.56E+01	7.48E+01
Chromium (Cr)	1.01E+02	8.74E+01	1.23E+02	1.86E+02	8.26E+01
Copper (Cu)	2.15E+02	7.47E+02	7.18E+02	2.17E+02	3.58E+03
Fluoride (F)(1)	--	--	--	--	--
Iron (Fe)	6.17E+04	5.97E+04	5.47E+04	5.28E+04	7.14E+04
Potassium (K)	1.40E+03	1.72E+03	2.66E+03	8.18E+03	1.75E+03
Magnesium (Mg)	4.00E+04	3.30E+04	2.00E+04	1.37E+04	3.63E+04
Manganese (Mn)	7.01E+02	6.28E+02	3.69E+02	2.43E+02	6.65E+02
Sodium (Na)	5.80E+03	5.12E+03	3.40E+03	9.94E+02	4.87E+03
Nickel (Ni)	2.55E+02	3.29E+02	3.33E+02	1.35E+02	9.72E+02
Lead (Pb)	2.45E+00	2.52E+00	5.10E+00	5.87E+00	6.22E+00
Antimony (Sb)	1.34E+00	1.31E+00	1.48E+00	1.74E+00	1.78E+00
Selenium (Se)(1)	--	--	--	--	--
Sulfur (S)(2)	6.40E+02	2.10E+03	9.50E+03	2.43E+04	9.00E+03
Thallium (Tl)	4.78E+00	4.74E+00	4.75E+00	4.30E+00	3.40E+00
Vanadium (V)	3.32E+01	3.77E+01	9.11E+01	1.36E+02	3.69E+01
Zinc (Zn)	6.83E+01	7.18E+01	1.04E+02	2.51E+02	8.11E+01

(1) Aqua regia data not available for alkalinity, chlorine, fluoride and selenium. No depletion is modeled.

(2) Sulfur values are from the Block Model, not the aqua regia dataset.

8.4.2 Nitrogen in Waste Rock

Nitrogen by-products of explosives used in rock blasting can leach from waste rock and pit walls, potentially causing exceedances of water quality standards. The applicable water quality standards are for nitrate plus nitrite in groundwater (10 mg/L as N) and for un-ionized ammonia in Class 2B surface waters such as the Partridge River (0.040 mg/L as N).

The total quantity of nitrogen in explosives residue is dependent on the nitrogen content of the explosives, the quantity of explosives used and the efficiency of detonation. Empirical relationships have been developed that account for the efficiency of detonation of each type of explosives (Reference (29)). Using these standard methods and PolyMet's expected explosives usage and characteristics (Table 8-9), the total nitrogen available for leaching (i.e., the nitrogen in the explosive residue) from each unit of waste rock can be calculated based on Equations 8-30 and 8-31, with variable names and units as defined in Table 8-9.

N usage: $$N_U = P\ x\ C\ x\ U$$ **8-30**

N residue: $$N_R = 0.94\% N_U^{ANFO} + 5.1\% N_U^{Emulsion}$$ **8-31**

The disposition of nitrogen in the explosive residue that remains with the waste rock is assumed to be expressed as 87% NO_3, 11% NH_3 and 2% NO_2. The resulting loading of each type of nitrogen from the waste rock is shown in Table 8-10.

Table 8-9 Nitrogen Leaching Input Information

Parameter	Symbol	Units	ANFO	Emulsion
Maximum powder factor (explosives per unit rock)	P	kg/mton	0.35	
Explosives N content	C	%	35%	26%
Explosives usage	U	%	30%	70%
N usage	N_U	kg N/mton	0.0368	0.0637
N residue	N_R	kg N/mton	0.00359	

Table 8-10 Nitrogen Leaching Outputs

Parameter	Symbol	Units	Value
Nitrate leached	NO_3	kg as N/mton	0.00313
Ammonia leached	NH_3	kg as N/mton	0.00040
Nitrite leached	NO_2	kg as N/mton	0.00007

It should be noted that nitrogen leaching from explosives residue is a one-time load dependent on the quantity of waste rock and is not expressed as a release rate or ongoing load like the other solutes discussed in this document. If all nitrogen is assumed to leach within the first year after rock is placed in the stockpile, the maximum potential impact from nitrogen leaching for a given waste rock stockpile (maximum leachate concentrations) occurs when the largest quantity of freshly-blasted waste rock is placed in the stockpile with the smallest footprint area (and therefore the lowest infiltration volume for dilution). For the NorthMet stockpiles, this maximum impact will occur in Mine Year 1, when relatively large quantities of waste rock are placed in relatively small stockpiles. Although the total mass of waste rock per acre is greater in later years of mining, the mass per acre *added* in Mine Year 1 is the largest for all stockpiles (Table 8-11).

Table 8-11 Stockpile Nitrogen Leaching Critical Conditions

Stockpile	Maximum New Material (tons/acre)	Mine Year
Category 1	87,800	1
Category 2/3	84,500	1
Category 4	51,700	1
Ore Surge Pile	32,300	1

A scoping-level calculation was performed for the nitrogen leaching from the Category 2/3 Waste Rock Stockpile in Mine Year 1. This stockpile has the largest initial mass of waste rock (M_S = 5,238,766 tons at the end of Mine Year 1) of the NorthMet stockpiles that have geomembrane liners and liner leakage that reports to groundwater flow paths (i.e., excluding the Category 1 Waste Rock Stockpile). At the end of Mine Year 1 the footprint area of the Category 2/3 Waste Rock Stockpile is 62 acres (A_S). This stockpile also has the shortest groundwater flow distance to the property boundary or the Partridge River (L_N = 600 feet), the potential point of compliance with groundwater standards. These conditions combine to make the Category 2/3 Waste Rock Stockpile the worst-case example of nitrogen leaching for the NorthMet stockpiles.

A simple mixing model was constructed for a flow path passing under the Category 2/3 Waste Rock Stockpile and ending at the property boundary. The assumptions for this scoping-level analysis are described in Table 8-12 and the model is shown in Equations 8-32 through 8-34.

Table 8-12 Scoping Model Inputs for Nitrate

Parameter	Symbol	Units	Value	Notes
Stockpile source length	L_S	ft	1,350	Width of stockpile in direction of flow path
Stockpile N loading	$N_{S,nit}$	kg as N/mton	0.0032	Sum of NO_3 and NO_2 from Table 8-10
	$N_{S,amm}$	kg as N/mton	0.0004	NH_3 from Table 8-10
Stockpile infiltration	I_S	in/yr	random	Probabilistic model as defined in Reference (30) Section 4.3
Liner leakage	LL	--	random	Probabilistic model as defined in Reference (30) Section 4.3
Mean natural area recharge	R_N	in/yr	0.5	Partridge River baseflow suggests recharge of between 0.55 and 0.85 in/yr
Mean recharge concentration	$C_{N,nit}$	mg as N/L	1.42	NO_3 + NO_2 , highest value observed at Mine Site, mean is 0.31
	$C_{N,amm}$	mg as N/L	0.42	NH_3, highest value observed at Mine Site, mean is 0.074

Stockpile concentration:
$$C_S = \frac{N_S M_S}{I_S A_S 1year} \quad \textbf{8-32}$$

Leakage to flow path:
$$R_S = I_S LL \quad \textbf{8-33}$$

Mixed concentration:
$$C_{mix} = C_S \left(\frac{R_S L_S}{R_S L_S + R_N L_N}\right) + C_N \left(\frac{R_N L_N}{R_S L_S + R_N L_N}\right) \quad \textbf{8-34}$$

This simple model conservatively assumes that 100% of the nitrogen available to leach from the waste rock deposited in the stockpile during the first year of mining does so evenly throughout a single year. If this leaching process took more time or had a correction for water contact, the resulting concentrations at the bottom of the stockpile would be lower. The model also makes the conservative assumption that the liner leakage from the stockpile is instantly mixed with the recharge water along the flow path, using a lower mean recharge rate and higher mean concentration than indicated by the available data.

Most importantly, the scoping-level model assumes that nitrogen mass is conserved along a groundwater flow path and that there is no net uptake by vegetation or denitrification and loss to the atmosphere. Given the relatively long travel times (decades to breakthrough) associated with flow through the surficial material at the Mine Site, this is a conservative assumption. It is highly likely that any nitrogen released from the waste rock stockpiles will be consumed by biologic activity long before reaching the property boundary or the Partridge River.

In order to quantify the impact of the uncertainty in stockpile hydrology (infiltration and liner leakage), the simple mixing model was run as a Monte Carlo simulation with 5,000 realizations in the GoldSim software (Section 3.0 of Reference (30)). The only inputs to this model that were defined as probability distributions (accounting for uncertainty and variability) were the stockpile hydrology variables—the remaining variables were defined with the conservative assumptions described above.

The intermediate model results for the concentration of nitrate plus nitrite in stockpile seepage (*Cs* in Equation 8-32) are shown in Figure 8-20. Using the conservative assumptions defined here, the concentration leaving the Category 2/3 Waste Rock Stockpile could range from approximately 100 to 300 mg/L (as N), with a mean of 180 mg/L (as N). Compared to nitrate seepage data from other cold-climate (Canada) mines with granitic waste rock (Table 8-13), these values are on the high end of the observed range. This observation confirms the conservative nature of the scoping-level model performed here.

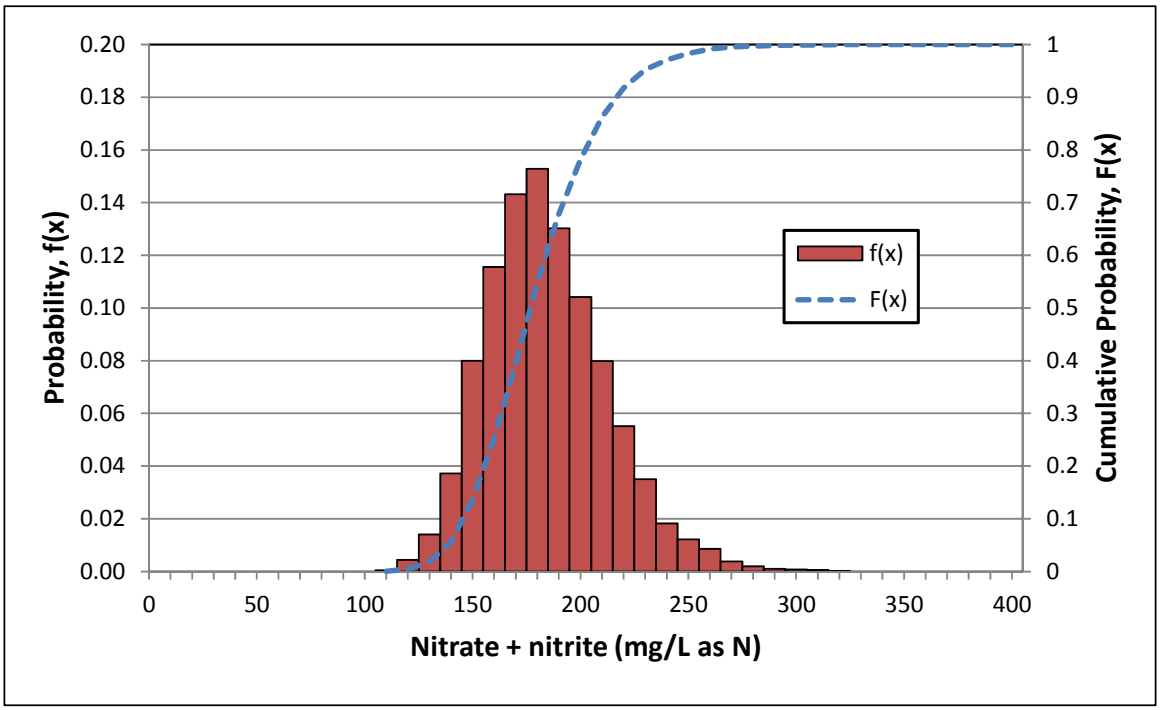

Figure 8-20 Scoping Model Results for Nitrate in Stockpile Seepage

Table 8-13 Cold Climate Waste Rock Seepage Nitrate Concentrations

Mine	Commodity	n	Min (mg/L)	Median (mg/L)	95th %ile (mg/L)	Max (mg/L)
Panda	Diamond	27	< 0.006	23.8	180	218
Misery	Diamond	31	0.334	13.3	81.8	143
Gibraltar	Copper, molybdenum	19	0.009	1.1	16.5	19.7
Mt. Polley	Copper, gold	13	0.013	11.9	31.9	32.3

Source: SRK project files

The model results for the groundwater concentration at the property boundary are shown as probability distributions in Figure 8-21 for nitrate plus nitrite and Figure 8-22 for ammonia.

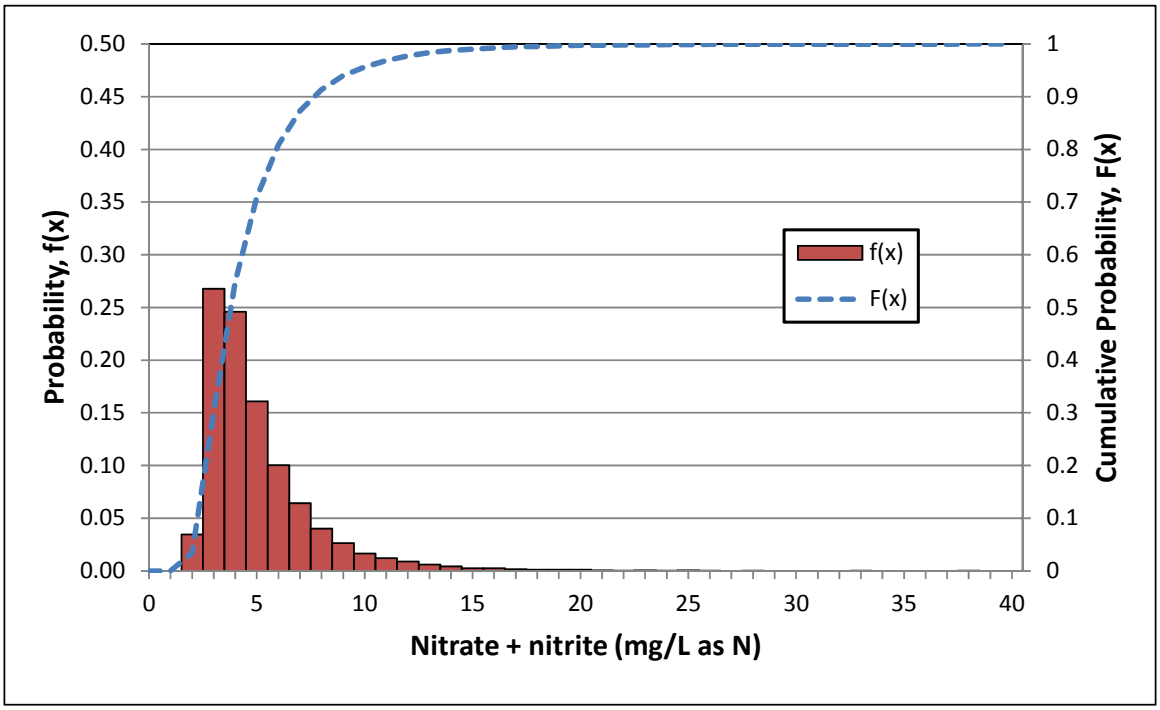

Figure 8-21 Scoping Model Results for Nitrate at the Property Boundary

As shown in Figure 8-21, the scoping-level mixing model shows that nitrate plus nitrite concentrations in groundwater at the NorthMet property boundary are highly likely to be below the applicable groundwater quality standard of 10 mg/L. Despite the conservative nature of the simple mixing model and the conservative input assumptions, less than 5% of

the model simulations had nitrate plus nitrite concentrations greater than 10 mg/L. Given that this model does not include the effects of biologic activity, the likely impacts to groundwater from nitrogen leaching can be considered to be negligible.

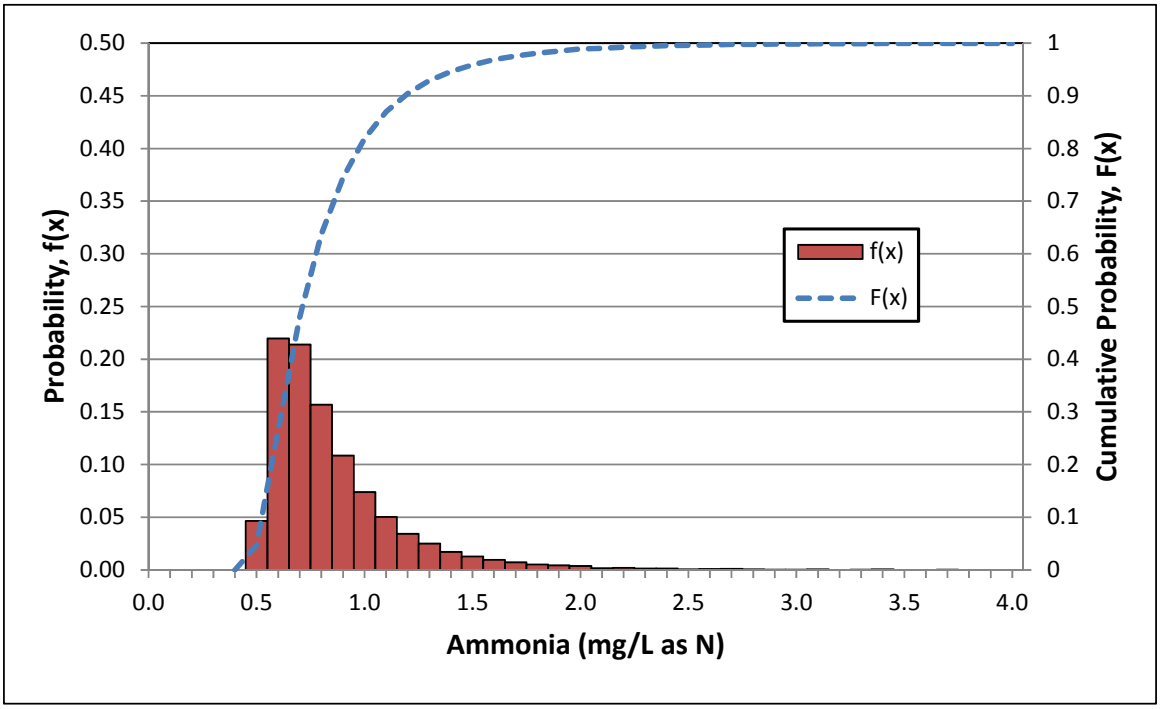

Figure 8-22 Scoping Model Results for Ammonia at the Property Boundary

As shown in Figure 8-22, the scoping-level mixing model shows that ammonia concentrations in groundwater at the NorthMet property boundary have a less than 5% probability of being greater than 1.5 mg/L (as N). At the average surficial groundwater temperature (7.7°C) and pH (7.7), un-ionized ammonia will represent less than 1% of the total ammonia in solution. The resulting un-ionized ammonia concentration at an ammonia concentration of 1.5 mg/L would be approximately 0.01 mg/L (as N), well below the surface water quality standard of 0.04 mg/L. Given the conservative nature of this model and that it does not include the effects of biologic activity, the likely impacts to surface water from nitrogen leaching can also be considered to be negligible.

Nitrogen from explosive residue is therefore not included in the probabilistic modeling of the waste rock stockpiles. Section 9.6.3 for analysis of the potential impacts from the flooded mine pits.

8.4.3 Ore Spillage from Rail Cars

Concerns were raised in the NorthMet Impact Assessment Planning process regarding the potential for water quality impacts due to the spillage of ore from rail cars along the rail

transfer corridor. Spillage of ore and mineral concentrates at other operations has, in some cases, resulted in contamination of soil and water bodies along the associated transportation corridors. For the Project, the proposed transportation of ore to the Plant Site is in rail cars along an existing rail corridor that traverses wetlands and crosses two streams.

Table 8-14 provides an estimate of the quantity of fine ore material that could spill from rail cars within the first 1,000 meters of the transfer corridor. Assuming that all spillage occurs in a 2 meter wide strip along this portion of the rail corridor, it is estimated that approximately 2.78 kg/m^2 could spill annually or 55.7 kg/m^2 over the 20-year life of the project. This is equivalent to 1.25 inches of spilled material over a 2,000 m^2 area.

Rail cars similar to those that will be used for the Project are currently in use at a Carmeuse Limestone operation in Michigan's Upper Peninsula. According to mine representatives, little or no spillage of fine material has been observed along the mainline tracks (Reference (31)). Some spillage occurs in the crusher building due to frequent stops and starts of the rail cars, and some fine material may be washed through the rail car doors by heavy rains. In general, however, there is no evidence of continued spillage along the main rail line. This information suggests that the spillage estimates provided here are conservative maximum values.

Preliminary modeling using the geochemical inputs discussed in this section for the spilled ore suggests that the rainfall contacting spilled ore material has the potential to exceed surface and groundwater quality standards at the source (i.e., including only rainfall over the 2,000 m^2 area and no other dilution). Water quality at points of compliance, such as within wetlands or streams adjacent to the rail corridor, will be better than the water quality at the spilled material due to dilution from other contributing areas, which may be significant.

If the spilled material is deposited evenly over the entire 13-km rail corridor rather than the first 1,000 meters as assumed here, the total quantity of spillage over the 20-year life of the project would be 4.28 kg/m^2 (assuming a 2 meter wide strip), or 0.1 inches of depth. Note that the streams traversed by the rail corridor are farther than 1,000 meters from the Rail Transfer Hopper. This small quantity of expected spilled material will also tend to become depleted of sulfide minerals quite rapidly compared to a waste rock stockpile. Nonetheless, impacts to surface and groundwater from rail car spillage cannot be dismissed.

In order to quantify the potential indirect impacts to wetlands along the rail corridor, the estimated spillage over the entire 13-km corridor is used with the modeling methods presented in this document to estimate the quality of water contacting this spilled material. The spilled ore is modeled according to the following assumptions:

- humidity cell rates as defined for the "ore composite" case in Section 8.1
- water contact factor equal to 1.0 (assumed complete rinsing)

- particle size factor equal to 1.0 (assumed particle size distribution identical to humidity cells)
- temperature factor uncertainty as defined in Section 8.2.4
- acidification factor and time to acidification uncertainty as defined in Section 8.2.5
- concentration caps as defined in Sections 8.3.1.6 and 8.3.3
- depletion as defined in Section 8.4.1
- annual water contact as defined for the bare stockpile case in Reference (30), Section 6.1.1, assuming that annual water contact (infiltration or runoff) equals precipitation minus evaporation

The results of this modeling include an estimate of the quality of water at the source of spillage as well as the additional dilution necessary to bring estimated concentrations below water quality standards. The dilution amount is expressed in terms of distance from the rail centerline, resulting in an estimated width of potential indirect impacts to wetlands.

The data needed for precise estimates of impacts from rail spillage is not available. The amount of likely spillage and the size of the material spilled can only be estimated; the precise location and physical distribution of spillage is unknown; the hydrology of the spilled material and the amount of dilution that could be expected before reaching points of compliance are highly site-specific.

In order to guard against possible adverse impacts from spilled ore, monitoring and mitigation activities can be developed. It is expected that surface water quality sampling in the two streams traversed by the rail line will be included in permit monitoring. Mitigation measures could include alterations to the stream crossings (bridges or culverts) to collect any spilled material or the physical collection of spilled ore from the top of the rail ballast.

Table 8-14 Estimate of Potential Rail Car Ore Spillage

Characteristic and Units	Value
Tons of ore transported per year	**11,680,000**
Size Distribution of Blasted Ore from blast fragmentation model	
% by weight less than 2" from blast fragmentation model	6.00%
Tons of Ore per Year of a size that could fit through potential side door gaps	700,800
% by weight less than 1/2" (assume linear from 2" to zero)	1.50%
Tons of Ore per Year of a size that could fit through potential hinge gaps	175,200
Rail Car Characteristics	
Car length (ft)	33.75
Car width (ft)	8.67
Car side height (ft)	3.88
Car volume (ft^3)	1,133
Hinge gap width (in)	0.50
Side door to carbody end gap (in)	2.00
Car volume within 2" of side door gap (ft^3) [This material would have to be adjacent to gap to fall out but would not move much]	0.43
Percent of car volume within 2" of side door gaps	0.04%
Car volume within 1.5" (3 diameters) of hinge gap (ft^3) [This material would have to be adjacent to gap to fall out and could move in the car]	1.05
Percent of car volume within 1.5" (3 diameters) of hinge gap	0.09%
Computed Mass	
Tons of <2 " ore per year that could reach side door gaps (assuming no size classification) (700,800 tons x 0.04%)	266
Percent of <2 " ore not classified to center of car [Most fines sift straight down - very good size classification on pan feeder plus additional classification as feeder feeds to center of car]	2.00%
Tons of <2 " ore per year close enough to side gap to fall out (266 tons x 2%)	5.32
Tons of <1/2 " ore per year that could reach hinge gaps (assuming no size classification) (172,200 tons x 0.09%)	163
Percent of <1/2 " ore not classified to center of car [Most fines sift straight down - very good size classification on pan feeder plus additional classification as feeder feeds to center of car]	0.50%
Tons of <1/2 " ore per year close enough to hinge gap to fall out (163 tons x 0.5%)	0.82

Characteristic and Units	Value
Potential Spillage	
Tons of ore per year potentially spilled (side gaps and hinge gap)	6.14
Kg of ore per year potentially spilled (side gaps and hinge gap)	5570
Kg of ore per year potentially spilled per square meter along track [assume all spills out in first 1000 meters in 2 meter width]	2.78
Kg per square meter of potential spillage for 20-year mine life	55.70
Approximate total depth of spillage for 20-year mine life (inches)	**1.25**
Approximate annual depth of spillage (inches)	**0.06**

8.4.4 Chloride Release from Waste Rock

Chloride release is included in the probabilistic model of Mine Site water quality in a different manner from the other constituents modeled. Chloride leaching from blasted waste rock is observed as a one-time process with no solubility effects, similar to nitrate leaching. The chloride originates from the bedrock or groundwater within the bedrock but is not continuously released during rock weathering.

Humidity cell data indicates that chloride release for the project samples (all rock types and categories considered together) ranges from less than 1 mg Cl per kg rock (ppm) to more than 70 ppm, with a median of 4.5 ppm. The beta distribution fit to the chloride release from all NorthMet humidity cells is shown in Figure 8-23.

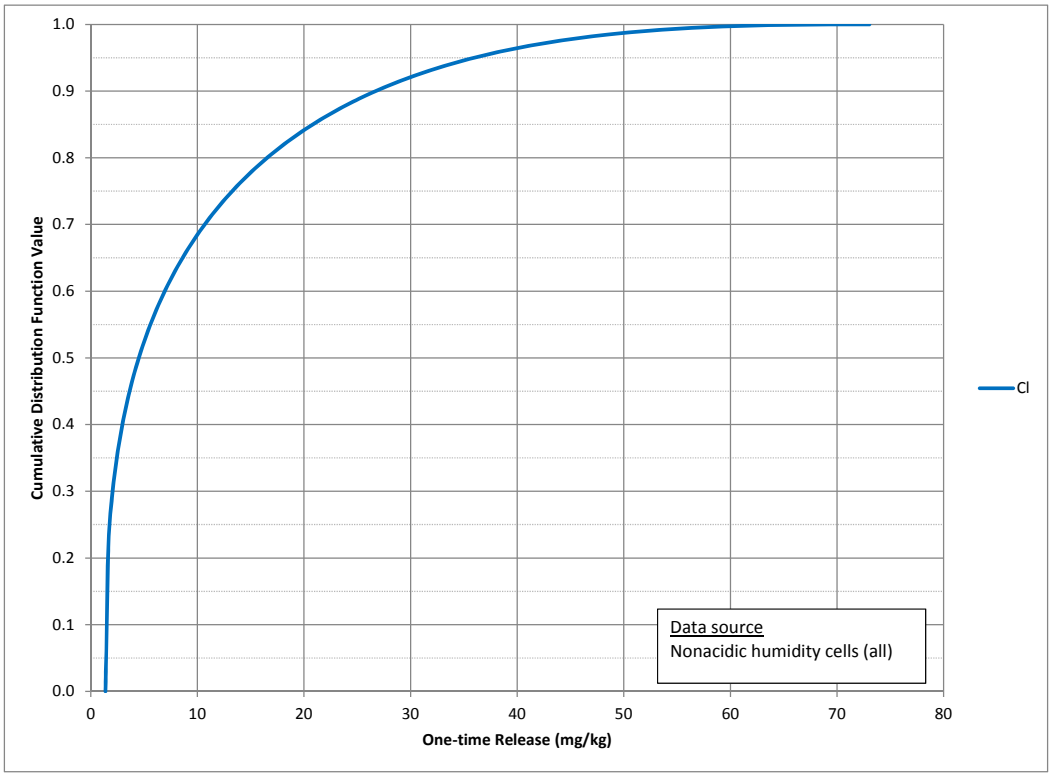

Figure 8-23 Distribution for One-Time Chloride Release

Chloride release in the probabilistic model is converted from a one-time release to a release rate (mg/kg/week, the same as the other constituents) by multiplying the randomly-simulated value from Figure 8-23 by the fractional rate at which waste rock is added to the stockpile (mass added per time period divided by total stockpile mass). Chloride release, therefore, will only occur while new rock mass is added to the stockpile. The release rate is scaled to field conditions to account for differences in water contact and particle size (Section 8.2) between the laboratory and field, but is not scaled by the temperature correction factor.

9.0 Geochemical Parameters – Pit Lake

This section covers geochemical parameters relating to pit wall rock and the pit lake that are used in water quality modeling.

9.1 Laboratory Release Rates

Constituent release rates for the pit wall rock are simulated using identical methods as for the waste rock stockpiles, described in Section 8.1. The simulated release rates are applied to the applicable portions of the pit walls by rock category and geology. Ore portions of the pit walls are simulated as discussed in Section 8.1, with probability distributions drawn from Category 2/3 humidity cells (with the exception of sulfate) and ore drill cores. This is a slight change from previous geochemical modeling of the pit wall rock (References (32) and (12)), in which wall rock classified as ore was modeled identically to Category 2/3 wall rock.

In closure, the Virginia Formation portions of the East Pit highwall would be treated with crushed limestone and capped with overburden and a permanent geomembrane cover (Reference (9)). The geomembrane would reduce the availability of oxygen to the wall rock, which would inhibit the oxidation of sulfide minerals and accompanying constituent release. The amount of limestone would be designed to provide sufficient buffering capacity to create neutral or near-neutral conditions along the wall rock, rather than the acidic conditions common for oxidizing Virginia Formation rock. The combined effect of these treatments would be a dramatic reduction in oxidation rates and constituent release from the covered portions of the East Pit highwall. Because no data exists for Virginia Formation oxidation under nonacidic conditions, however, the constituent release rates for this segment of pit wall are conservatively assumed to remain unchanged from those presented in Section 8.1. This treatment is modeled as primarily affecting the applicable concentration caps for exposed Virginia Formation wall rock (nonacidic caps instead of acidic caps), Section 9.5.1.

9.2 Mass of Reactive Wall Rock

The surface area of each category of wall rock that is subject to oxidation is determined from the Mine Plan as a function of elevation for a given year of mine development. The plan view surface area (looking downward) is used to reflect that the majority of the fractured wall rock is located on horizontal benches rather than on the sloped walls due to the expected blasting patterns (Reference (32), Section 5.3).

In the deterministic modeling, a thickness of 2 meters was used to represent the average thickness of reactive wall rock throughout the entire pit. This value is derived from the depth of expected over-drilling during blasting and wall rock fracturing below the removal zone. In order to reflect the uncertainty in this thickness, the distribution previously proposed is a triangular distribution of ± 50% of the deterministic value, or from 1 to 3 meters. This distribution is applied to all wall rock with the exception of the Virginia Formation Category 4 wall rock. For the Virginia Formation Category 4 wall rock, the distribution for wall

thickness is a triangular distribution from 2 to 6 meters. See Figure 9-1 for these distributions.

In addition to the pit walls blasted into the bedrock, the probabilistic modeling also considers the potential loading from the 20-foot pit rim set back at the top of the rock walls. This rock will not be blasted and is expected to be more competent than bedrock exposed on in-pit benches; however, the removal of some or all of the overlying surficial material will expose this rock to oxidation and freeze-thaw weathering. These areas are modeled identically to the blasted pit walls but with the thickness reduced by a factor of 10. The resulting range of reactive thickness for the pit rim set back is 10 cm to 30 cm for Duluth Complex and 20 cm to 60 cm for the Virginia Formation Category 4 wall rock.

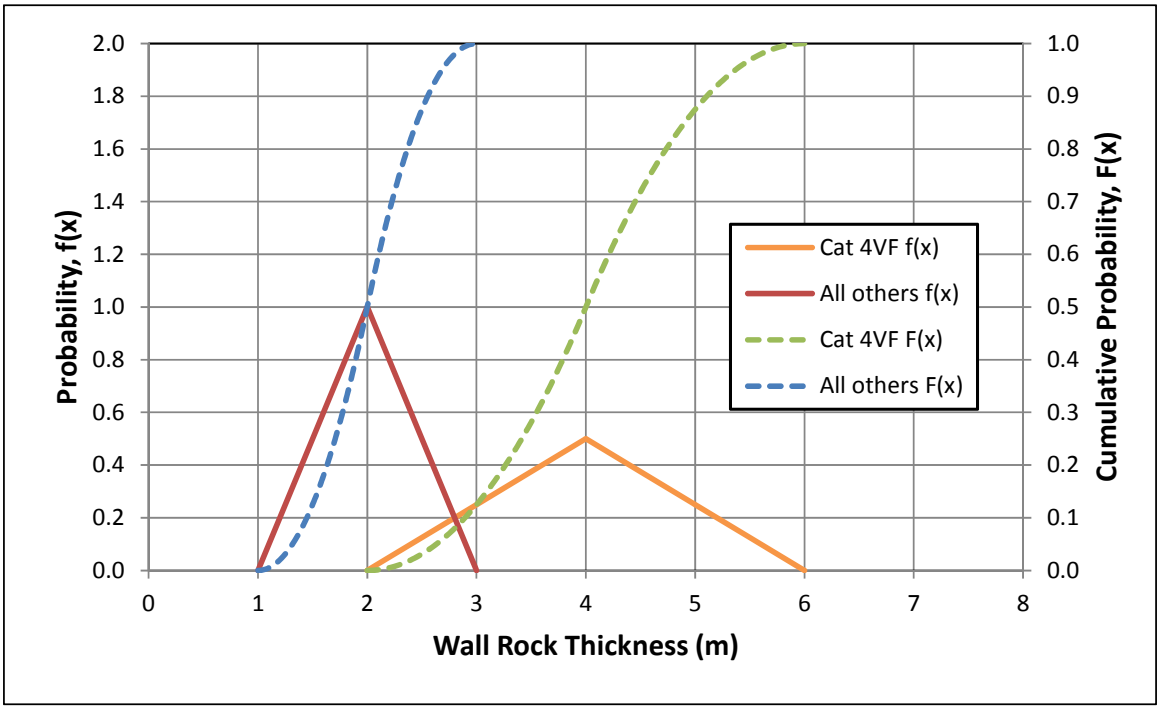

Figure 9-1 Wall Rock Thickness Distribution

9.3 Lab to Field Scale-Up

In general, scaling from laboratory to field scale constituent release for the pit wall rock is performed in a similar manner as for the waste rock. The pit walls are assumed to behave much like stockpiles of waste rock, with competent but fractured bedrock on the benches and pit walls and accumulated talus material that is exposed to oxygen as in the waste rock stockpiles.

The scale-up factor is applied to the nonacidic laboratory release rates (from the humidity cell tests) in order to calculate release rates from full scale wall rock. The bulk scale-up

factor (S_B) is dependent on three sub-factors that represent the effects of differences between laboratory and field conditions with respect to the water contact (k_c), particle size (k_s), and temperature (k_t). Note that unlike the scale-up for waste rock stockpiles there is no acidity factor in Equation 9-1; the effects of acidification and subsequent decay are modeled differently for the pit wall rock as described in Section 9.4.

$$S_B = k_t * k_s * k_c \qquad \textbf{9-1}$$

9.3.1 Water Contact Factor

The water contact factor for the pit wall rock k_c is modeled using the same distribution as discussed in Section 8.2.2 and shown in Figure 8-8 for the waste rock stockpiles. For the pit walls and the waste rock backfilled into the East Pit, the fraction of released constituent mass that is not contacted by percolating water (1 - k_c) is assumed to be entirely flushed into the pit water when each segment of wall or backfill is inundated.

9.3.2 Particle Size Factor

The particle size factor for the pit wall rock k_s was assumed to equal 0.10 in the deterministic modeling. This is half of the value that was used for the waste rock in stockpiles, reflecting the fact that the fractured wall rock is expected to contain fewer fine particles and a larger proportion of very large blocks. The proposed distribution for the wall rock size factor is a uniform distribution of ± 50% of the deterministic value, or from 0.05 to 0.15. This range reflects the relatively large uncertainty in this factor and is shown in Figure 9-2.

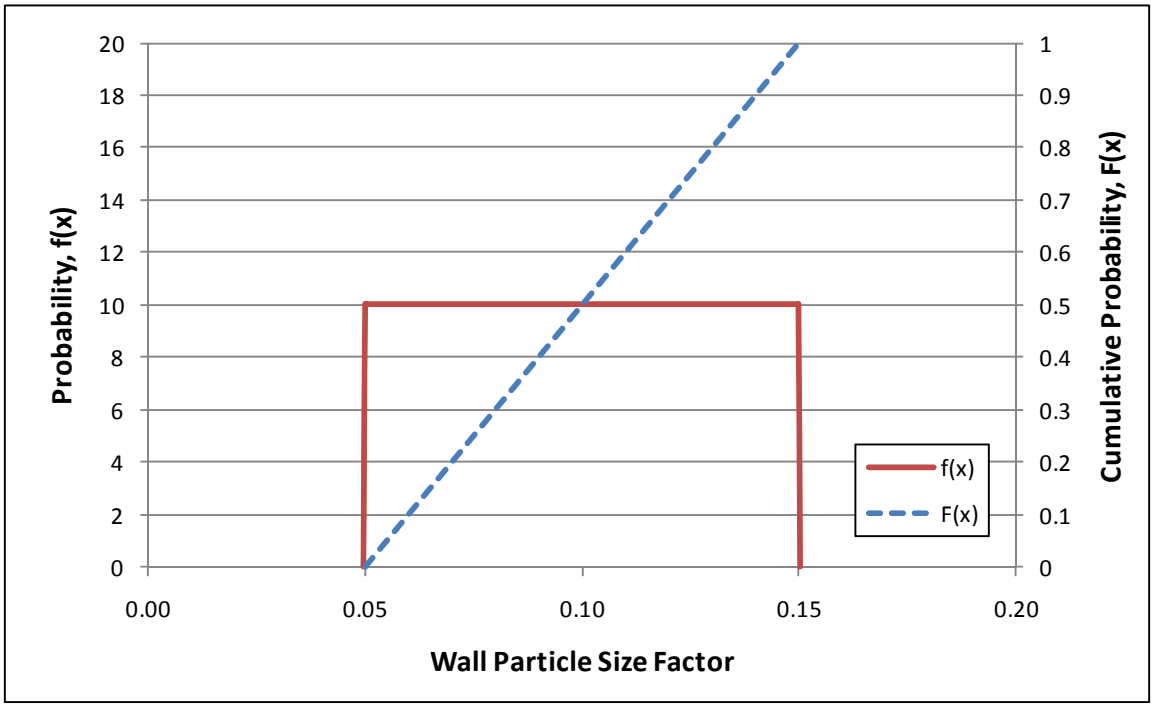

Figure 9-2 Wall Particle Size Factor Distribution

9.3.3 Temperature Factor and Solar Heating

The expected temperature factor k_t for the pit wall rock can be calculated in a manner similar to that for the waste rock stockpiles, using the Arrhenius equation (Equation 8-16). It is assumed that the release of heat from oxidation of the relatively thin veneer of pit wall rock is not sufficient to increase the temperature of the walls. The average temperature of the pit wall rock will then be governed by the distribution for the average air temperature at the site, similar to the Category 1 Waste Rock Stockpile (Figure 8-10). Because the wall rock is geochemically identical to the waste rock, the same distribution for the activation energy applies (Figure 8-11).

Because the proposed mine pits have their long axes oriented east-west, questions have been raised regarding whether the south-facing pit walls (the northern wall of each pit) will have increased temperature and therefore increased oxidation due to solar heating. The following analysis is designed to determine the approximate magnitude of temperature increase (above ambient air temperatures) in this portion of the pit walls due to solar heating. This analysis is based on thermodynamic principles and equations found in Reference (33) and the assumed thermodynamic properties of stone and air shown in Table 9-1.

Table 9-1 Thermodynamic Input Assumptions

Parameter	Symbol	Units	Value
Daily temperature oscillation frequency	ω	1/s	7.27×10^{-5}
Thermal diffusivity of stone(1)	α	m^2/s	7.4×10^{-7}
Stone emissivity(2)	ε	--	0.80
Stone reflectance(3)	α_{solar}	--	0.20
Stone thermal conductivity(1)	k_{stone}	W/mK	1.6
Stefan-Boltzmann constant(1)	σ	W/m^2K^4	5.67×10^{-8}
Air kinematic viscosity at 320°K(1)	υ	m^2/s	1.75×10^{-5}
Air Prandtl number at 320°K(1)	Pr	--	0.708
Air thermal conductivity at 320°K(1)	k_{air}	W/mK	0.0275

(1) Reference (33)
(2) Low-end value for stone taken from literature. See examples for various materials at www.engineeringtoolbox.com/emissivity-coefficients-d_447.html. Typical ε for concrete is 0.85.
(3) Low-end value for stone taken from literature. See examples of (1 - αsolar) for various materials at http://www.engineeringtoolbox.com/solar-radiation-absorbed-materials-d_1568.html. Typical αsolar for concrete is 0.40.

The increase in temperature at the rock surface above mean daily air temperature (ΔT) can be defined by performing an energy balance for the rock face. Energy inflows are short-wave solar radiation (q_s) and long-wave radiation absorbed from the atmosphere (q_a), energy outflows are reflected solar radiation (q_r), long-wave radiation emitted from the rock face (q_e), convective heat transfer from the rock face to the air (q_{conv}) and conduction of heat into the wall (q_{cond}):

$$q_s + q_a = q_r + q_e + q_{conv} + q_{cond} \qquad \textbf{9-2}$$

The energy inflows on the left side of the equation can be determined from regionally-available data. The energy outflows on the right side of the equation are functions of the rock surface temperature and can be used to solve for the rock temperature.

9.3.3.1 Solar Radiation

The solar radiation incident on a horizontal plane surface was measured at St. Paul, Minnesota as part of climatological studies in the 1970s (Reference (34)). The study used measurements from multiple years to determine the median peak daily radiation at the summer solstice in June and at the winter solstice in December (Table 9-2). In addition, the study also measured the solar radiation incident on a south-facing plane inclined 55° to the horizontal. Radiation on this face, which approximates conditions on an inclined pit wall, was less that on the horizontal plane in June and greater in December (i.e., the inclined plane is a more efficient collector of winter solar radiation but less efficient in the summer). Using

these ratios, the median peak daily solar radiation on the pit wall q_s is estimated as shown in Table 9-2.

Table 9-2 Radiation Inflow Data and Calculations

Parameter	Symbol	Units	June	December
Median peak daily solar radiation (horizontal)(1)		W/m^2	700	200
Ratio of inclined to horizontal radiation(1)		--	80%	160%
Average monthly temperature(2)	T_{month}	°F	61.7	8.5
Average daily high air temperature(2)	T_{air}	°F	74.2	18.1
Average dew point temperature(2)	T_{dp}	°F	51.1	6.8
Average wind speed(2)	u	mph	8.2	8.5
Calculated median peak daily solar radiation (inclined)	q_s	W/m^2	560	320
Calculated effective sky temperature	T_{sky}	°F	277	248
Calculated long-wave radiation absorbed from atmosphere	q_a	W/m^2	268	172

(1) Reference (34)
(2) Reference (35)

9.3.3.2 Atmospheric Radiation

For the purposes of these calculations, the pit wall is assumed to behave as a thermodynamic "grey body", with infrared emittance and reflectance (a.k.a. albedo) that are not dependent on temperature or wavelength. The assumed emittance ε and reflectance α_{solar} for the rock wall are shown along with other thermodynamic inputs in Table 9-1.

The long-wave radiation absorbed by the pit wall from the atmosphere can be estimated by treating the sky as a black-body radiator with an effective temperature T_{sky} [°K]:

$$T_{sky} = T_{air}\left[0.711 + 0.0056T_{dp} + 7.3 \times 10^{-5}T_{dp}^2 + cos\left(\frac{2\pi t}{24}\right)\right]^{\frac{1}{4}} \qquad \textbf{9-3}$$

where T_{air} is in °K, T_{dp} is in °C and t is the hour past midnight. Using the temperature and dew point values measured at International Falls and shown in Table 9-2, the resulting calculated values for T_{sky} at noon are shown in Table 9-2.

Using the effective sky temperature, the long-wave radiation absorbed by the pit walls from the atmosphere is calculated from Equation 9-4, where σ is the Stefan-Boltzmann constant from Table 9-1. The resulting energy absorbed is shown in Table 9-2.

$$q_a = \varepsilon\sigma T_{sky}^4 \quad \textbf{9-4}$$

9.3.3.3 Reflected Solar Radiation

The solar radiation reflected by the pit wall is simply calculated as a fraction of the incoming solar radiation, represented by the reflectance α_{solar}. The calculated values for q_r are shown in Table 9-3.

9.3.3.4 Emitted Radiation

The remaining elements of the energy balance (Equation 9-2) for the rock face are functions of the temperature at the surface of the rock and must be solved by iteration. Long-wave radiation emitted by the stone q_e is calculated using a similar equation to the long-wave radiation absorbed from the atmosphere:

$$q_e = \varepsilon\sigma T_{wall}^4 \quad \textbf{9-5}$$

9.3.3.5 Convective Heat Transfer

Convective heat transfer away from the rock face by moving air can be estimated by assuming turbulent flow over a segment of the rock face, which acts as a long plate. The heat removed from the rock is a function of the temperature difference between the rock and the air and the convective heat transfer coefficient$\bar{h}$, which is a function of the properties of the moving air mass over the plate:

$$q_{conv} = \bar{h}(T_{wall} - T_{air}) \quad \textbf{9-6}$$

$$\bar{h} = 0.0370\left(\frac{uL}{\nu}\right)^{0.8} Pr^{0.43}\left(\frac{k_{air}}{L}\right) \quad \textbf{9-7}$$

where u is the velocity of the flowing air [m/s], L is the assumed length of the wall face [m] and υ, Pr and k_{air} are thermodynamic properties of air at the "film temperature" midway between T_{wall} and T_{air} (assumed values shown in Table 9-1). Using the mean wind speed measured at International Falls (Table 9-2) and an assumed unbroken wall face length of 50 m, the resulting values for the heat transfer coefficient are shown in Table 9-3.

9.3.3.6 Heat Conducted into Wall

Precise calculation of the heat conducted into the stone wall is beyond the scope of this analysis. The heat flow into the mass of wall rock is driven by the temperature difference between the surface T_{wall} and the temperature at depth, which is assumed to equal the average monthly temperature T_{month}. This simplified system can be approximated as:

$$q_{cond} = \frac{k_{stone}}{d}(T_{wall} - T_{month}) \qquad \textbf{9-8}$$

where k_{stone} [W/mK] is the thermal conductivity of the rock (Table 9-1) and d [m] is the depth at which the temperature difference applies. It is assumed that temperatures within the rock return to approximately the average monthly temperatures at d of 0.5 meters (this assumption has been validated below).

9.3.3.7 Wall Surface Temperature Increase

The above discussion has shown that the variables on the left side of the energy balance shown in Equation 9-2 can be calculated directly and the variables on the right side can either be calculated directly (q_r) or are functions of the surface temperature T_{wall}. By trial and error, T_{wall} can be determined such that the energy balance closes. The calculated values for T_{wall} and for the remaining portions of the energy balance are shown for June and December conditions in Table 9-3. Because of the simplifying and conservative assumptions used in this analysis, the estimated median peak wall surface temperature in June is higher than expected (45.3 °C or 113.5 °F during daily high temperatures of 74.2 °F).

Table 9-3 Calculated Radiation Outflows and Temperature Changes

Parameter	Symbol	Units	June	December
Reflected solar radiation	qr	W/m^2	112	64
Convective heat transfer coefficient	$\bar{h}$	W/m^2/K	7.3	7.5
Wall surface temperature	Twall	°C	45.3	5.3
Emitted long-wave radiation	qe	W/m^2	466	272
Convective heat loss	qconv	W/m^2	158	97
Conductive heat loss into wall	qcond	W/m^2	92	59
Wall surface temperature increase	ΔTwall	°C	28.8	13.0
Average temperature increase over 2m depth	$\overline{T_{2m}}$	°C	2.05	0.93

The values shown in Table 9-3 for ΔT_{wall} represent the temperature increase at the rock surface due to solar and long-wave radiative heating beyond the temperature increase driven by diurnal air temperature changes (i.e., $T_{wall} - T_{air}$). It is this wave of additional heating that will potentially cause increased chemical reactions in the south-facing pit walls. The influence of this large diurnal temperature variation is limited to surface layer of wall rock and will trigger a pulse of increased temperature that will move back into the rock mass while being dampened. By assuming sinusoidal daily temperature variation at the rock surface and an infinite mass of rock, the envelope (peak of each pulse) of observed rock temperatures above mean daily temperatures can be defined by:

$$T = \Delta T_{wall} \cdot e^{-x\sqrt{\frac{\omega}{2\alpha}}} \qquad \textbf{9-9}$$

where ΔT_{wall} is the temperature amplitude at the surface [°K], x is distance into the rock face [m], ω is the frequency of the temperature oscillation [1/day] and α is the thermal diffusivity of stone (assumed values in Table 9-1).

Using the temperature variation envelope shown in Equation 9-9, the changes in wall rock temperature driven by oscillating radiative heating are shown in Figure 9-3. The effect of the temperature increase is minimal (< 1°C) at depths deeper than 0.5 meters into the rock walls, validating the assumed value of d used in Equation 9-8 above.

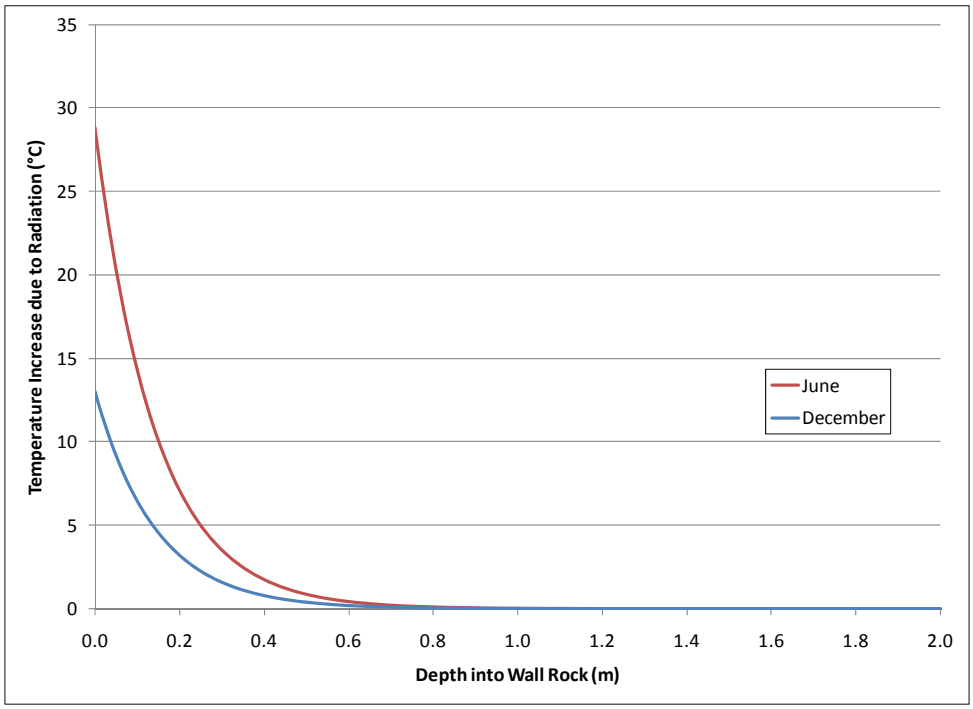

Figure 9-3 Envelope of Increased Wall Rock Temperatures due to Radiation

As discussed in Section 9.2, the geochemical modeling for the pit walls considers the reactive mass of wall rock to be, on average, 2 meters thick (all wall rock except Virginia Formation Category 4, Figure 9-1). The average temperature increase over the surface 2 meters of wall rock can be determined by integrating Equation 9-9. The resulting depth-averaged temperature values for June and December are shown in Table 9-3.

The values shown for $\overline{T_{2m}}$ in Table 9-3 are assumed to represent the bounds of the annual temperature increase due to radiative heating. The average annual temperature increase over the 2 meter thickness of reactive rock (due to radiative heating) is then calculated to be 1.5 °C. This temperature increase has been applied to the temperature used for lab-to-field

scaling of the wall rock reaction rate (Section 8.1), but only for the south-facing portions of the pit walls (approximately 50% of both the West Pit and East Pit).

9.4 Acidification and Long-Term Decay in Constituent Release

As discussed in Attachment A, data from the long-term MDNR tests on Duluth Complex rock provide information on the decrease in sulfate release following the onset of acidic conditions as readily-available sources of sulfide minerals are consumed. As discussed in Section 8.2.5, it has been suggested that the rapid decay observed in the MDNR reactors may over-state the potential decay under field conditions. Decay curves are developed here for both the MDNR reactors and the few NorthMet humidity cells that have exhibited decaying sulfate release. Note that this discussion focuses on acidification and declines after the onset of acidic conditions, and does not apply to Category 1 waste rock.

The modeled decay equation for sulfate release is of the form shown in Equation 9-10. This equation is distinct from a typical first-order decay equation in that it is a log-log relationship between release rate and time; time is not log-transformed in a typical first-order relationship. This log-log fit matches both the MDNR and NorthMet data better than a first order fit and results in a "flatter" long-term release curve. An example of the decay fit for one of the NorthMet humidity cells using Equation 9-10 is shown in Figure 9-4. Note that the value for time in Equation 9-10 is expressed in terms of weeks since the onset of acidic conditions, which has been determined as discussed in Section 8.2.5.

$$SO_4\left(\frac{mg}{kg \cdot week}\right) = 10^{a_1 \times \log time(weeks\ since\ peak) + a_0} \qquad \textbf{9-10}$$

The weekly humidity cell sulfate release data and the fitted decay curves for the NorthMet humidity cells used in this analysis are shown in Figure 9-5.

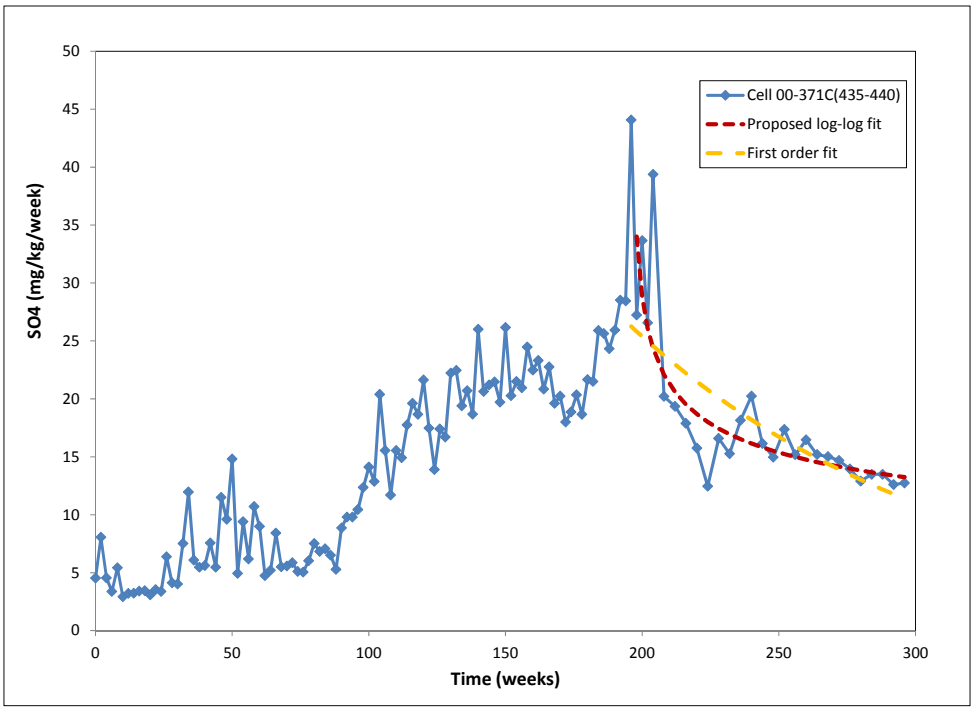

Figure 9-4 Example Sulfate Decay Relationships

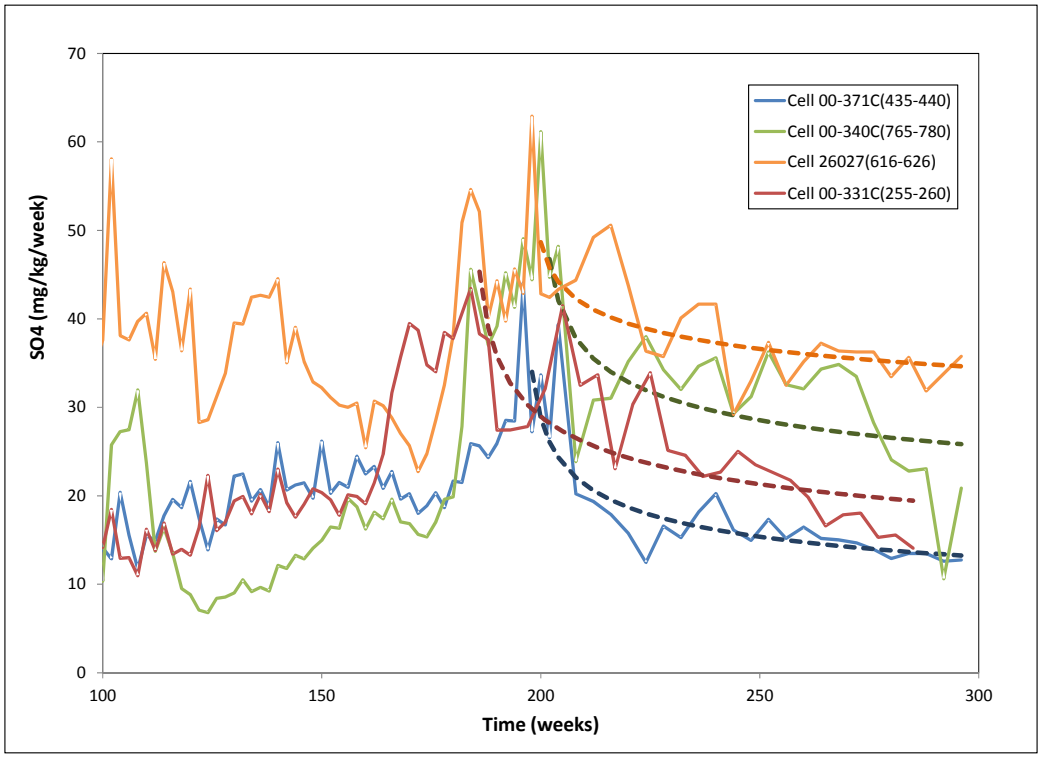

Figure 9-5 Sulfate Decay in NorthMet Humidity Cells

A plot of the parameters a_0 and a_1 developed from the MDNR long-term data and from the NorthMet humidity cell data are shown in Figure 9-6, along with the increase in sulfate release due to acidic conditions discussed in Section 8.2.5. See Table 3 of Attachment A for a list of the specific MDNR reactors used in this analysis; the NorthMet humidity cells used here are identified in Large Table 1. The parameters that describe the magnitude of the peak sulfate release (a_0 and the increase over non-acid release) are highly correlated with the speed of the decay (a_1). In both data sets test cells that have high initial acidic release rates (higher values for a_0 and higher increase factors) experience the most rapid decay (lower, more negative values for a_1). The correlation coefficient between a_0 and a_1 for the combined MDNR and NorthMet data is -0.989; the correlation coefficient between the acidic increase and a_1 is -0.831 (±1 indicates perfect correlation, 0 indicates no correlation).

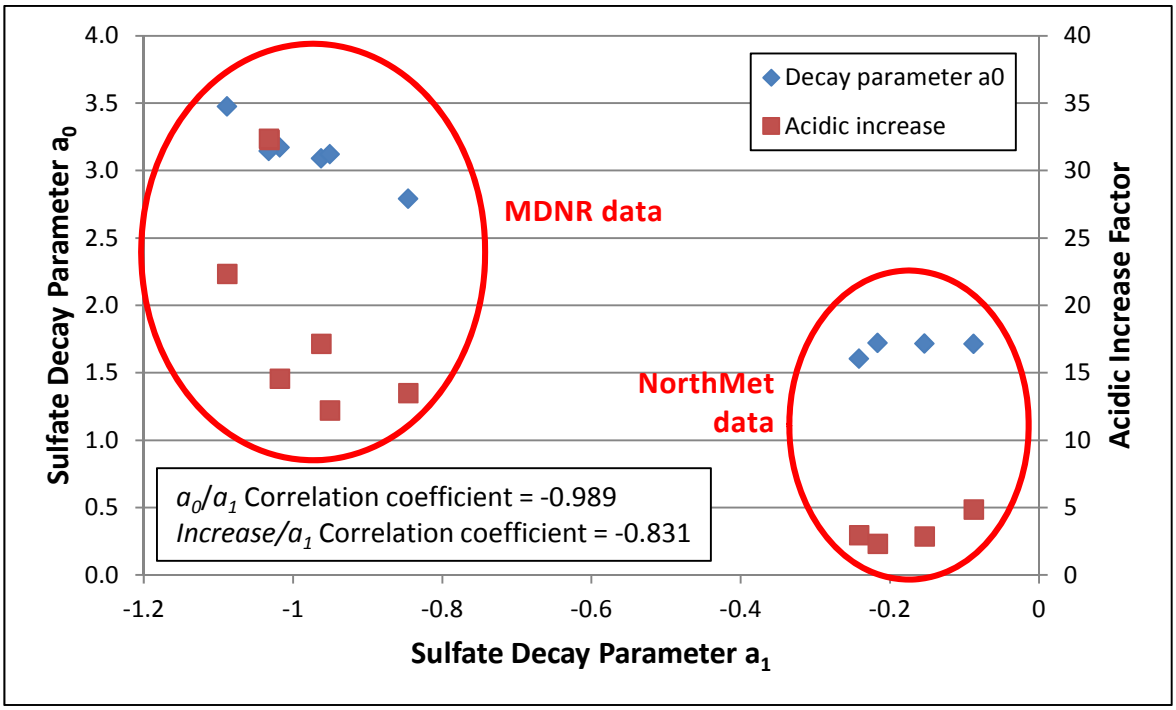

Figure 9-6 Sulfate Decay Parameters from MDNR and NorthMet Data

It is clear from Figure 9-6 that the MDNR reactors showed much greater decrease in sulfate release, as evidenced by lower, more negative values for the decay slope parameter a_1. This observation confirms the hypothesis that the finer material in the MDNR reactors likely over-states the decay in sulfate release that will be observed at field conditions. The decay relationships developed from the NorthMet humidity cells, in contrast, represent the cumulative impact on decaying release from a much wider range of particle sizes. Because of the uncertainty associated with the true long-term performance of the pit walls, the combined data is used to develop probability distributions.

For the majority of the chemical constituents included in the probabilistic water quality modeling, release rates are simulated using ratios to sulfate release (Section 8.1.2). When the rock becomes acidic, sulfate release will increase by a factor as defined in Section 8.2.5 and the release of constituents simulated using ratios will increase accordingly. Similarly, long-term decay in release rates for all constituents is simulated by modeling decay in the sulfate release rate. When sulfate release declines according to Equation 9-10 the release of constituents simulated using ratios will decrease accordingly. No other decay rates have been calculated for other constituents in order to maintain the conceptual model of constituent release driven by sulfide oxidation. Those constituents with release rates not linked to sulfate are assumed to have no decrease in release rates.

Probability distributions for the parameters a_0 and a_1 developed from the combined test data are shown in Figure 9-8 and Figure 9-9. These uniform distributions represent the minimum and maximum parameter values observed in the referenced NorthMet humidity cells and MDNR reactors. The correlation coefficients developed from the combined MDNR and NorthMet data have been used to correlate these two distributions in the probabilistic modeling and to correlate decay to the increase in sulfate release from acidification (Section 8.2.5). These correlations in the modeling maintain the relationships observed in the laboratory: samples with high peaks in sulfate release also had rapid decay.

Unlike the waste rock stockpiles, the relatively thin veneer of wall rock is not expected to generate enough heat from chemical reactions to warm the wall rock above ambient temperatures (Section 9.3.3 for a complete discussion of wall rock temperature scaling). Therefore, the time to acidification determined from MDNR laboratory tests must be scaled to account for the slower rate of chemical reactions at field temperatures. The simulated time determined as in Section 8.2.5 has been divided by the temperature correction factor discussed in Section 9.3.3, effectively extending the period of nonacidic conditions.

The steps in modeling long-term sulfate release from wall rock are as follows:

1) Generate random nonacidic laboratory sulfate release rates (Section 8.1)
2) Generate random values for the time to onset of acidic conditions (Section 8.2.5)
3) Generate random scale-up correction factors for water contact, particle size, and temperature (Section 9.3)
4) Determine first date of wall rock exposure (Section 9.6.1)
5) If the time of wall rock exposure is less than the time to onset of acidic conditions divided by the temperature correction (to represent the longer time to reach peak release at field temperatures), sulfate release is nonacidic
    a. Field sulfate release is calculated from nonacidic laboratory release rates times temperature and particle size corrections and partitioned between contact (flushed by runoff) and non-contact mass
6) If the time of wall rock exposure is equal to or greater than the time to onset of acidic conditions divided by the temperature correction (to represent the longer time to reach peak release at field temperatures):

a. Generate random decay parameters a_1, a_0 (correlated to a_1) and acidic increase factor (correlated to a_1)
b. If the length of time the wall rock has been acidic is greater than zero, laboratory sulfate release is determined from Equation 9-10
c. Field sulfate release is calculated from the result of Equation 9-10 times temperature and particle size corrections and partitioned between contact (flushed by runoff) and non-contact mass

As an example, twenty realizations of the modeling method described here are shown in Figure 9-7 for Category 2/3 wall rock. All other model parameters (nonacidic sulfate release rate, temperature correction factor, etc.) were held constant at their mean values and only the decay parameters a_1 and a_0 were allowed to vary according to the probability distributions shown in Figure 9-8 and Figure 9-9 and the defined correlation coefficient. The resulting range of sulfate release curves includes realizations with long-term sulfate release of more than five times the nonacidic release rate as well as realizations with significant and relatively rapid (within a few decades) decay to release rates below the pre-acidic value.

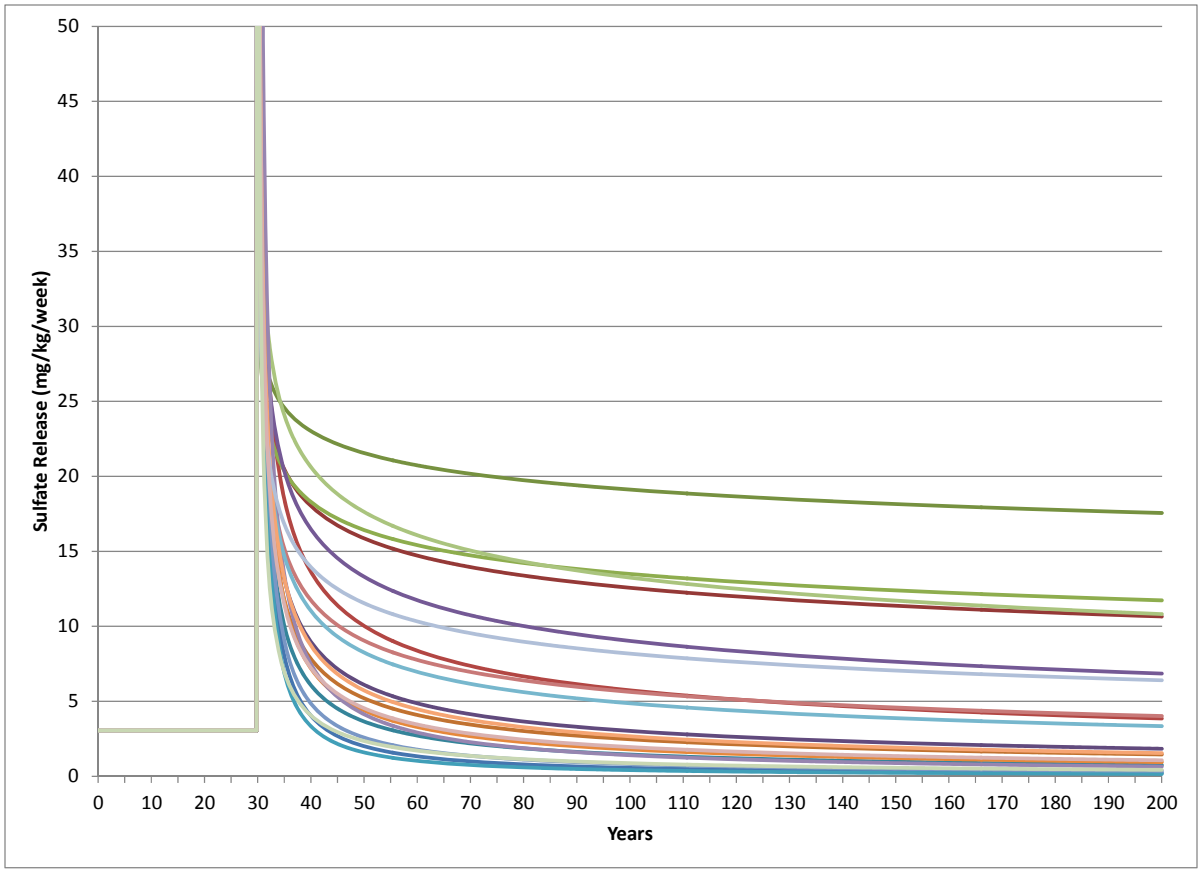

Figure 9-7 Sulfate Decay Example Simulation (20 realizations)

Long-term decay in release rates were not modeled for the Category 1 or for the Virginia Formation portions of the pit wall rock, only for the Duluth Complex Category 2/3, Category 4 and ore wall rock in both mine pits. The nonacidic reactions in the Category 1 wall rock are expected to consume the sulfide minerals too slowly for appreciable decay during the pit filling time, and there is no data on long-term behavior of this material similar to the MDNR reactors. In contrast, the different physical structure of the Virginia Formation wall rock has led to concerns of long-term physical weathering, which could liberate sulfide mineral grains for ongoing oxidation. Therefore it is conservatively assumed that there is no long-term decrease in sulfate release from the Virginia Formation portions of the pit walls.

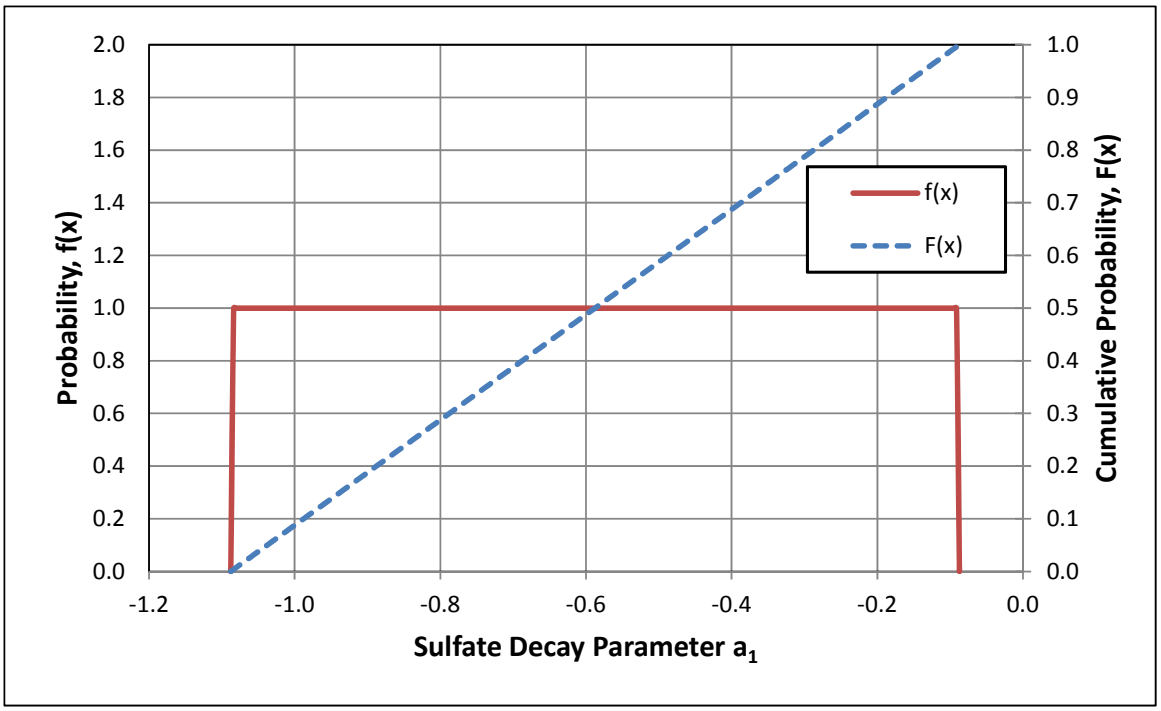

Figure 9-8 Distribution for Sulfate Decay Parameter a_1

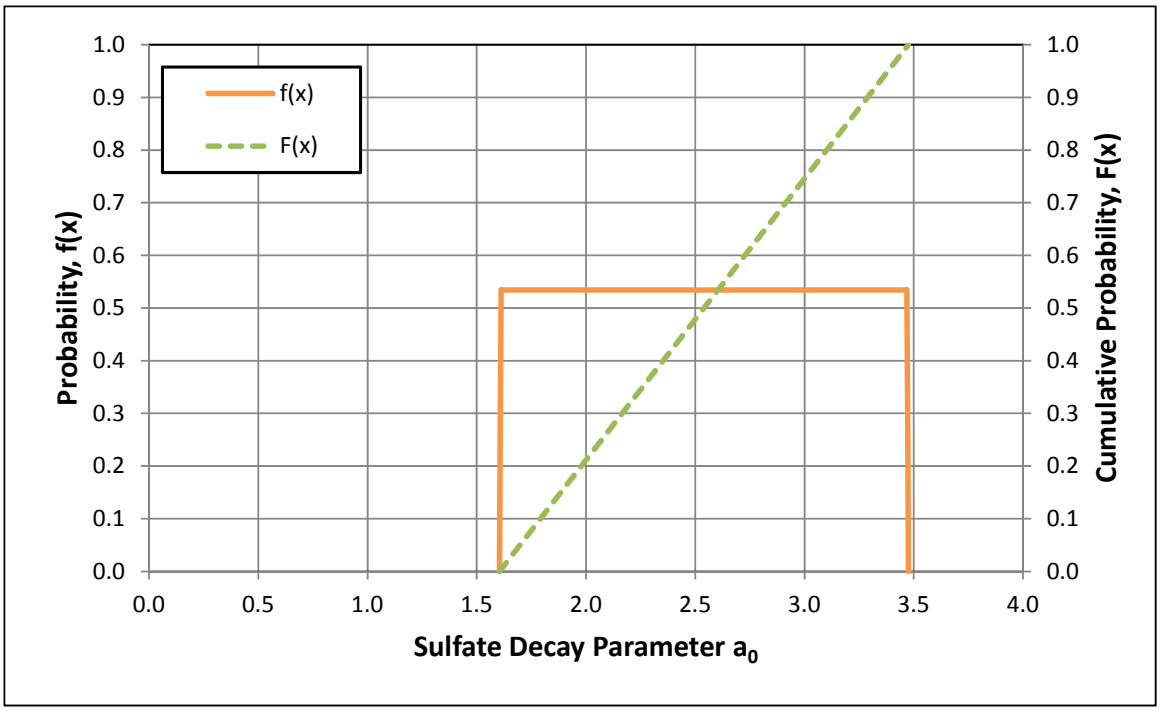

Figure 9-9 Distribution for Sulfate Decay Parameter a_0

9.5 Concentration Caps

9.5.1 Pit Walls

Concentration caps for the water running off the pit wall rock are simulated using identical methods as for the waste rock stockpiles, described in Section 8.3. The simulated concentration caps are applied to the applicable portions of the pit walls by rock category.

As described in Section 9.1, the Virginia Formation highwall of the East Pit would be treated with limestone and capped with overburden and geomembrane in closure. The amount of limestone would be designed to provide sufficient buffering capacity to create neutral or near-neutral conditions along the wall rock, rather than the acidic conditions common for oxidizing Virginia Formation rock. In the probabilistic modeling, therefore, the concentration caps applied to that portion of the pit wall are those developed for the nonacidic Duluth Complex Category 2/3, Category 4 and ore rock. The geomembrane cap placed over the highwall would also have the effect of dramatically reducing the quantity of infiltrating water to contact the wall rock, making the concentration of the seepage water highly likely to reach the applied nonacidic concentration caps and limiting the pollutant mass transport away from the wall rock. The effect of this lower infiltration rate on lower mass transport, however, was conservatively not included in the probabilistic model.

9.5.2 Pit Backfill and Pit Lake

The waste rock and overburden material that is backfilled into the East Pit will be adjusted as needed to maintain circumneutral pH (see Section 2.2.1.1 of Reference (36)). The pH of the backfill porewater is not simulated directly. However, the concentration caps applied to the backfill porewater are those developed for the nonacidic Duluth Complex Category 2/3, Category 4 and ore rock. These nonacidic concentration caps are also applied to the water in the East Pit wetland, which will be in contact with submerged backfill material. The backfill pH control is assumed for the water quality modeling to not add any constituent mass to the East Pit.

For simplicity, the concentration caps developed for the Category 1 waste rock are applied to the West Pit lake to reflect the generally low acidity and metals content in the diluted pit water. This simplified empirical method, however, does not capture the complex geochemical processes in the aqueous system that will cause secondary mineral precipitation and impose solubility controls on some constituents in the pit water. If additional precision is needed for the long-term concentrations of the West Pit lake, a chemical and thermodynamic equilibrium model could be developed as part of permitting.

9.6 Additional Model Parameters and Considerations

This section covers additional modeling parameters relating to pit wall rock and the pit lake.

9.6.1 Age of Pit Walls

As mining progresses, the mine pit boundaries will be extended both vertically downward and horizontally through pushbacks. The typical age of the pit walls at any time will be dependent on the details of the Mine Plan. For the purposes of the probabilistic water quality modeling, all of the pit walls are assumed to be reactive at the commencement of operations (time zero). This assumption results in the over-estimation of loading to the WWTP during operations due to pit dewatering.

The Duluth Complex portions of the pit walls are estimated to become acidic in approximately 20 years (based on the time to acidity distributions presented in Section 8.2.5 and the temperature correction factor distribution presented in Section 9.3). The assumption that the pit walls are reactive at time zero minimizes the amount of nonacidic loading from the pit walls that is retained in the flooding pits during closure, effectively over-estimating the loading from the pit walls as the walls are inundated. This is especially true because the portions of the pit walls that will be exposed latest in mining (near the bottom of the pits) will be the first to be flooded and may not be exposed long enough to become acidic.

The sensitivity of the model results to this assumption will be evaluated during the modeling process to ensure that this assumption is in fact conservative.

9.6.2 Backfill Modeling Parameters

The Mine Plan provides the quantity of waste rock (in tons) that is moved to the East Pit during each year of backfilling. In order to determine the volume of the pit occupied by the backfilled rock, the available pore space and the water required to maintain steady flooding of the backfill, the following parameters are used in the water quality model. These parameters are considered deterministic (known) and do not vary through time.

- waste rock specific gravity: 2.93
- waste rock swell (effective increase in bulk volume, including voids): 30%
- backfill porosity: 0.2255
- desired margin between backfill surface and water table: 5 feet (elevation)

9.6.3 Nitrogen in Mine Pits

As described in Section 8.4.2, standard methods can be used to estimate the quantity of nitrogen residues from explosives that are present on blasted waste rock and may potentially leach to the surrounding environment. For the mine pits, this information can be used to estimate the nitrogen leaching from the pit walls to the flooded pit lakes.

Because the West Pit is a source of water in post-closure to both the Partridge River via a surface overflow and to the surficial groundwater, a scoping-level analysis was prepared to compare potential West Pit water quality with both the surface and groundwater quality standards. This analysis uses the computed ammonia and nitrate plus nitrite loading per metric ton of waste rock shown in Table 8-10. The total mass of wall rock in the West Pit, assuming a conservatively-large thickness of 3 meters (high end of the range shown in Figure 9-1, is 10,730,000 metric tons. The total water volume in the flooded West Pit at overflow is 100,040,000 cubic meters.

The West Pit walls will be exposed to rainfall for at least several years during operations and could potentially leach most of the nitrogen while pit wall runoff and other inflows are collected and treated prior to pit flooding. The pit walls are assumed to leach half of the nitrogen during operations and half during flooding for this scoping-level analysis.

As a result of this analysis, the water quality in the West Pit at overflow is estimated as 0.17 mg/L nitrate plus nitrite (as N) and 0.021 mg/L ammonia (as N). The computed nitrate plus nitrite concentration is significantly less than the groundwater quality standard of 10 mg/L, while the ammonia concentration is approximately half of the surface water quality standard of 0.04 mg/L for un-ionized ammonia. Note that at the expected site temperatures and near-neutral pH of the flooded West Pit water, un-ionized ammonia will represent approximately 1% of the total ammonia in solution and the resulting concentration would be much less than the surface water quality standard. Note that this scoping-level analysis does not consider the

effects of biologic activity and the cycle of nitrogen uptake and denitrification. These effects would tend to reduce the amount of available nitrogen in the flooded pit lake.

The generally low availability of nitrogen in other mine pits in northeastern Minnesota supports the conclusion that exceedances of surface or groundwater standards for nitrogen is not a concern in the mine pits. See Table 9-4 for water quality data from 2008 and 2009 from flooded mine pits at the former LTVSMC mine for an example. Given these data and the analysis included here, the effects of explosives residue on water quality has not been included in the probabilistic model of the mine pits.

Table 9-4 Nearby Pit Lake Nitrogen Concentrations

Parameter	Units	Area 1 Pit (oldest)	Area 2 WX Pit (newest)	Area 6 Pit
Nitrate plus nitrite (as N)	mg/L	0.05	0.11	0.050
Ammonia (as N)	mg/L	0.116	0.05	0.050
pH	s.u.	8.35	8.53	8.31
Calculated un-ionized ammonia (as N)1	mg/L	0.003	0.002	0.001

Source: Reference (37)
(1) Calculated assuming site temperature of 3°C

9.6.4 Subaqueous Oxidation

The modeling methodology for the pit lakes assumes that oxidation of sulfide minerals and the accompanying release of constituents is negligible when waste rock or pit walls are submerged under water. To test this assumption the rate of subaqueous oxidation can be calculated by an approach recommended by the MDNR and discussed in Reference (38). The original method is outlined in Reference (39).

The flux of dissolved oxygen (J, mg $O_2/m^2/s$) into a 1 m^2 column of tailings or waste rock can be estimated from the empirical equation:

$$J = 8.2 \times 10^{-2} DO \sqrt{\frac{(\varphi - \varphi^2)P}{d\tau}} \qquad \textbf{9-11}$$

where DO is the dissolved oxygen concentration in the overlying water [mg/L], φ is the rock porosity, P is the percentage of pyrite in the rock, d is the rock particle diameter [m], and τ is the tortuosity of the oxygen flow path. The rate of pyrite consumption (CP, mg $FeS_2/m_2/s$) is estimated as:

$$CP = \alpha J \qquad \textbf{9-12}$$

where α is a factor based on the molar ratio of oxygen to pyrite in the oxidation of pyrite to ferric hydroxide, and is equal to 1.0.

The input values used for this analysis are shown in Table 9-5.

Table 9-5 Scoping Model Inputs for Subaqueous Oxidation

Parameter	Symbol	Units	Value	Notes
Dissolved oxygen	DO	mg/L	10	Typical surface water value
Wall rock porosity	φ	--	0.1	Assumed value for fractured wall rock
Wall rock diameter	d	m	0.1	Median particle size of fractured rock
Wall rock tortuosity	τ	--	3	Recommended value
Wall rock pyrite(1)	P	%	1.1	See note 1

(1) 1.1% pyrite is equivalent to a sulfur content of 0.6%, representing the high end of the Category 2/3 wall rock. The actual sulfide mineral in the NorthMet walls is pyrrhotite, not pyrite, but the difference is expected to be minimal for this calculation.

The resulting rate of pyrite consumption from Equation 9-12 is 4.7×10^{-6} mg $FeS_2/m^2/s$. This is equivalent to a sulfate release rate of 3.9×10^{-4} mg SO_4/kg/week, assuming an average 2-meter thick column of wall rock (the calculated release is inversely related to the assumed wall thickness, so a smaller wall thickness is more conservative). For comparison, the average sulfate release rate for ore wall rock in the West Pit, as calculated from the release rate relationship in Figure 8-5 and the average particle size in Figure 9-2 (excluding the temperature correction), is 0.84 mg SO_4/kg/week. The estimated sulfate release due to subaqueous oxidation, therefore, is more than three orders of magnitude less than the sulfate release from exposed wall rock.

For the pit walls, this calculation supports the assumption that subaqueous oxidation is a negligible contributor of sulfate and metals to the pit lake. For the West Pit, approximately 2% of the total pit wall surface area will remain exposed above the flooded elevation in post closure, representing more than enough rock mass to overwhelm any contribution from subaqueous wall rock. Furthermore, the assumed dissolved oxygen concentration of 10 mg/L is only applicable for the upper portions of the pit lake; the pit lake is expected to be thermally stratified for the majority of the year and below about 10 meters oxygen supply will be minimal.

For the waste rock backfilled into the East Pit, the quantity of subaqueous waste rock is significant relative to the quantity of exposed wall rock. Once the backfill is complete and the pit is flooded, there will be approximately 1,000 times more rock mass submerged (backfill and pit walls) than exposed to the atmosphere. However, the oxygen supply to the waste rock backfill will be even more limited than in an open pit lake. The waste rock backfill will be covered with a layer of compacted soil to serve as a hydraulic barrier and a wetland will be constructed at the surface; there will be virtually no downward transport of atmospheric oxygen into the backfill. The only mechanism for oxygen transport into the backfill will be groundwater from bedrock, with typical concentrations around 3 mg/L and flow rates around 20 gpm for the entire pit. This small supply of oxygen will result in total sulfate release rates that are at least an order of magnitude less than those from the exposed wall rock.

9.6.5 Dewatered Material Oxidation

Concerns were raised in the NorthMet Impact Assessment Planning process regarding the potential for water quality impacts due to the oxidation of currently-saturated materials as the dewatering of the mine pits proceeds. When water is drawn out of the pores of surficial materials and bedrock as the water table is lowered, air will fill the void spaces and react with the exposed mineral soil surfaces. When dewatering ceases and the water table recovers, any oxidation products such as sulfate or soluble metals in the void spaces will be transported towards the pit lake by groundwater flowing into the cone of depression. The chemical loading from the dewatered materials could be a significant source of load to the flooded mine pits.

Given the information on current conditions and modeling of future conditions that is available at the Mine Site it is not possible to quantify the magnitude of the potential impacts to water quality from oxidation of the in-situ materials around the mine pits. This is because of a lack of information in several key areas:

1) Groundwater modeling: The MODFLOW model that has been developed to estimate pit inflows during dewatering and pit flooding does not provide realistic estimates of the size of the cone of depression around the mine pits. Without this information it is impossible to estimate the quantity of currently-saturated material that will have the potential to generate oxidation products.

2) Oxygen transport: In order to compute the mass of oxidation products released, the amount of oxygen reaching the newly-unsaturated material must be quantified. For this project, the waste rock stockpiles have been assumed to be fully oxygenated at all times. For the unsaturated tailings, an oxygen diffusion model has been constructed to simulate oxygen availability with depth through the moist tailings. The material surrounding the mine pits will be unsaturated but moist, and will not be fully oxygenated. Oxygen transport through the surficial material will be complicated by the presence of layered soils, perched saturated areas (especially in wetlands) and

oxygen consumption by vegetation, biologic activity, decomposition and reaction with mineral soils (including sulfide minerals). There is not sufficient information about the conditions at the Mine Site to adequately simulate oxygen transport.

3) Material reactivity: In order for aerated mineral soils to oxidize to form sulfate and soluble metals, sulfide minerals such as pyrite and pyrrhotite must be present and in contact with the void spaces. The availability of these minerals will determine the rate at which oxygen is consumed oxidation products are formed. There is not sufficient information about the mineralogy in the saturated overburden to accurately determine the reactivity of the aerated soils.

While it is not possible at this time to estimate the impacts to water quality due to oxidation of dewatered material and subsequent reestablishment of the water table, it should be noted that any impacts will be a one-time occurrence and will primarily be to the pit lakes rather than to the surrounding environment. As water levels recover the groundwater flow gradient will be in to the pits, carrying any oxidation products into the flooding pit lakes. Once water levels are stable there will continue to be inward gradients towards the pits in many areas, particularly on the north sides of the pits. The areas with potential impacts to the surrounding environment will be limited to those locations where the final pit water level is above the top of the bedrock and groundwater flows away from the pit.

If the oxidation of dewatered in-situ material and subsequent reestablishment of the water table causes significant loading of sulfate or metals to the pit lakes during pit flooding, opportunities exist for mitigation through in-pit treatment or the Waste Water Treatment Facility, which will remain operational as needed in closure.

10.0 Geochemical Parameters – Flotation Tailings

This section covers geochemical parameters relating to the Flotation Tailings and the underlying LTVSMC tailings that are used in water quality modeling.

10.1 Laboratory Release Rates

An updated methodology for interpreting the results of the NorthMet humidity cell tests is described in Section 5.1.3.2 and Large Table 5. This method builds on and continues the analysis presented in Attachment A. The specific methods used to develop probability distributions for the various constituents and types of tailings material are detailed here. The probability distributions discussed in this section are presented in Large Table 16 through Large Table 19 and shown in Large Figure 42 through Large Figure 60.

Separate methods for developing release rates are presented for the Flotation Tailings (coarse and fine) and the LTVSMC tailings. It is important to note that the tailings modeling methods described in Section 10.2.1 are used to estimate the oxidation of sulfide minerals and the subsequent release of sulfate. For this reason the remaining release ratios and solids ratios, as much as possible, must be linked to sulfate release.

10.1.1 Flotation Tailings

The methods for modeling the uncertainty in laboratory-scale release rates for all constituents are described below. Distribution parameters used in the probabilistic modeling are shown in Large Table 16 and Large Table 17 and Large Figure 42 through Large Figure 49.

10.1.1.1 Release Rates from Humidity Cells

For all constituents with release rate methods identified by “XX Rate” or “XX/XX Rate” in Large Table 5, probability distributions have been developed either directly from the humidity cell data (SO_4) or the ratio of release rates in the NorthMet humidity cells (all others). For each cell a temporal average release rate or release rate ratio has been determined for the long-term average conditions. Humidity cells with material larger than the #200 mesh, previously referred to as “mid” (#100 mesh to #200 mesh) and “coarse” (larger than the #200 mesh) tailings separately, have been analyzed together to represent coarse tailings as determined from the tailings deposition study (Large Table 4 and Section 5.1.3.1). In all cases, only tailings samples generated when the pilot plant was adding copper sulfate have been included in this analysis. For manganese, the correlation to nickel release is much stronger in the tailings humidity cell data than in the waste rock data, therefore the Mn/Ni ratio is used to estimate tailings Mn release (waste rock modeling uses the Mn/SO_4 ratio).

A probability distribution has been fit to the sample data set of average release rates or release rate ratios (one per humidity cell) applicable for the specific tailings portion. The complete range of release rates observed in the humidity cell testing has been assumed to

represent the possible average release rate from the entire mass of NorthMet coarse or fine tailings. No attempt has been made to weight the humidity cell data by sulfur content or otherwise bias the determined release rates towards the expected average conditions in the field.

For sulfate, the observed release rate in the humidity cell tests varies as a function of the remaining sulfur content, with observed release rates declining over time in the tests (see Attachment F). Because the oxygen transport modeling requires a zero-order reaction rate (see Section 10.3), a sulfate release rate that does not change with time or sulfur content must be defined. The humidity cell data were analyzed to compare the sulfate release rate (mg/kg/week) and the computed remaining sulfur content (%S) for each humidity cell, excluding the initial five weeks of testing as directed by the Co-lead Agencies to remove the effects of initial flushing of stored oxidation products. Linear regression was used on these data to estimate the effective sulfate release at the initial measured sulfur content for each humidity cell test. Based on this analysis there are no observed differences between the relationship of sulfate release and sulfur content for the various pilot plant tests. The distributions for sulfate release (fine and coarse tailings separately) have been defined from these initial effective sulfate release rates rather than an average observed release rate.

For all constituents with release rates determined from release ratios, the actual release rate for the constituent is calculated by multiplying the randomly-selected release rate ratio by the simulated release rate of the constituent in the denominator (ex. Mg release rate = Mg/SO_4 release ratio x SO_4 release rate).

For several constituents (Cd, Co, Zn) the proposed release rate ratios are derived from the Category 2/3 waste rock humidity cell tests rather than the tailings humidity cell tests. Similar to the proposed waste rock modeling method, release of these particular metals is observed to be strongly correlated to metal and sulfate release rates in the tailings humidity cells (not the solid metal to sulfur ratio, Attachment A). Due to the larger number of waste rock humidity cells, the waste rock data provides a better understanding of the possible range of variability in these release ratios than do the blended tailings humidity cell samples, therefore the distributions are fit to waste rock humidity cell data. The derived ratios are related ultimately to sulfate release, so the final release rates for these constituents are different than for the waste rock because of the different sulfate release rate.

10.1.1.2 Release Rates from Solid Ratios

For all constituents with release rate methods identified by “XX/XX” and data sources of “Aqua Regia” or “Microprobe” in Large Table 5, probability distributions have been developed from the solids content data in the NorthMet whole tailings testing database. For each constituent, a probability distribution for the solids ratio (ex. ratio of Cu/S in the whole tailings analysis) has been fit to the population of observed solids ratios applicable for the specific tailings portion.

For constituents with "Aqua Regia" identified as the source data, the solids ratios from analysis of the humidity cell samples have been used to develop distributions. As discussed above for the humidity cell release rates, the complete range of solids ratios observed in the whole tailings testing (for coarse and fine tailings separately) has been conservatively assumed to represent the possible average release rate ratio from the entire Flotation Tailings deposit, with no weighting or relationship to sulfur content.

For constituents with "Microprobe" identified as the source data, the solids ratios identified from analysis of individual mineral grains in the waste rock humidity cell tests have been used to develop distributions. Because the minerals present in the Flotation Tailings are the same as those in the NorthMet waste rock, the distributions used here are identical to those developed for the waste rock analysis. Again, the range of solids ratios in the microprobe testing has been assumed to represent the possible average conditions for the entire Flotation Tailings deposit.

The actual release rate for each constituent is calculated by multiplying the randomly-selected solids ratio by the simulated release rate of the constituent in the denominator (ex. Cu release rate = Cu/S ratio x SO_4 release rate in terms of S).

10.1.1.3 Release from Concentration Caps

For all constituents with release rate data sources identified by "Solubility Model" or "Defined Concentration Cap" in Large Table 5, release is not simulated directly in the probabilistic model. Rather, these constituents are assumed to be released so that concentrations are always at the indicated concentration caps. For alkalinity, the concentration cap is calculated based on modeled calcite solubility at a range of CO_2 partial pressures and the resulting pH conditions (Equation 10-1). For fluoride, the concentration cap is calculated based on calcium concentrations as determined for the Category 1 waste rock in Section 8.3.1, reproduced as Equation 10-2 (Reference (27)). For boron and chromium, the concentration cap is modeled as the concentration cap derived for the Category 1 waste rock, as described in Section 10.4.

Alkalinity:
$$\text{Alkalinity}\left(\frac{mg}{L}\right) = 0.110 + 2.42\left[Ca\left(\frac{mg}{L}\right)\right] \quad \textbf{10-1}$$

Fluoride:
$$F\left(\frac{mol}{L}\right) = \sqrt{\frac{8.91\text{x}10^{-11}\left(\frac{mol^3}{kg^3}\right)}{Ca_{limit}\left(\frac{mol}{L}\right)}} \times \left[\rho_{water}\left(\frac{\text{kg}}{L}\right)\right]^{1.5} \quad \textbf{10-2}$$

10.1.2 LTVSMC Tailings

The methods for modeling the uncertainty in laboratory-scale release rates for all constituents are described below. Distribution parameters used in the probabilistic modeling

are shown in Large Table 18 and Large Table 19 and Large Figure 50 through Large Figure 60.

10.1.2.1 Release Rates

For all constituents with release rate data sources identified as "HCT", "Aqua Regia", or "Microprobe" in Large Table 5, distributions for release rates and solids ratios have been determined as discussed above for the Flotation Tailings. All tailings humidity cell and whole tailings data used are for the LTVSMC tailings only, and include more whole tailings samples than those used in the humidity cells.

10.1.2.2 Release from Concentration Caps

For all constituents with release rate data sources identified as "Solubility Model" or "Observed Seepage" in Large Table 5, release is not simulated directly in the probabilistic model. Rather, these constituents are assumed to be released so that concentrations are always at the indicated concentration caps.

For constituents with identified controlling minerals, a solubility model has been developed to define the concentration of the seepage from the LTVSMC tailings material, as shown in Equations 10-3 through 10-5 (for fluoride, Reference (27)). For constituents where the controlling minerals are not well-defined, surrogate concentration caps have been developed from the seepage data collected at the existing LTVSMC basin. The inherent assumption of this method is that for many constituents, conditions in the existing LTVSMC basin represent capped conditions that are expected to continue in the future.

Alkalinity:
$$\text{Alkalinity}\left(\frac{mg}{L}\right) = 0.110 + 2.42\left[Ca\left(\frac{mg}{L}\right)\right] \quad \textbf{10-3}$$

Barium:
$$\log\left[Ba\left(\frac{mg}{L}\right)\right] = -0.32 \times \log\left[SO_4\left(\frac{mg}{L}\right)\right] - 0.87 \quad \textbf{10-4}$$

Fluoride:
$$F\left(\frac{mol}{L}\right) = \sqrt{\frac{8.91\text{x}10^{-11}\left(\frac{mol^3}{kg^3}\right)}{Ca_{limit}\left(\frac{mol}{L}\right)}} \times \left[\rho_{water}\left(\frac{\text{kg}}{L}\right)\right]^{1.5} \quad \textbf{10-5}$$

As discussed in more detail in Section 10.5, the modeling of the FTB assumes no geochemical interaction between the NorthMet and LTVSMC tailings or their seepage water. The application of solubility controls or defined seepage concentrations (rather than generation rates) for the LTVSMC tailings as discussed here is therefore not used to limit the loading originating from the FTB. Chemical loading from the LTVSMC tailings, whether determined from laboratory generation rates or observed seepage concentrations, is additive to the loading from the Flotation Tailings (i.e., the estimated concentration at the toe of the

existing LTVSMC tailings is allowed to be higher than the defined seepage concentration from the LTVSMC tailings alone).

10.1.2.3 Flushing Load

As in previous modeling of the FTB, the probabilistic model includes loading from the flushing of soluble metals when LTVSMC tailings are disturbed to construct the dams. Leaching of soluble metals is assumed to occur in the first year after construction of each lift of the dam. Distributions for the mass loading of all constituents (per mass of disturbed tailings) have been derived from the leach extraction tests on LTVSMC tailings material and are shown in Large Table 19 and Large Figure 56 through Large Figure 60.

10.2 Lab to Field Scale Up

The scale-up method from laboratory release rates to field release rates is very similar to the method used for the waste rock described in Section 8.2. In that section, the scaling factor for waste rock is dependent on differences in temperature, particle size and water contact. The tailings however use a correction factor for temperature and an additional factor for freezing. The tailings used in the humidity cells are the same size as the operational tailings so there is no need to correct for differences in particle size. Additionally, the vertical flow behavior through the tailings will be different from that through a stockpile of waste rock. Water will likely not create highly preferential flow in the tailings meaning the water contact at both field and lab scales will be about the same. Therefore, the scaling factor is only dependent on temperature differences and times of the year when the tailings are frozen.

The scale-up factor equation from Section 8.2 (Equation 8-9) is modified to include only the temperature factor and freezing factor:

$$S = k_T * (1 - k_f) \qquad \textbf{10-6}$$

The method for calculating k_T is described in detail in Section 8.2.4. Figure 10-1 shows the resulting distribution for k_T. The temperature data described in Section 8.2.4 was used to determine the portion of the year that temperatures are below freezing. For this analysis, the daily maximum temperature was used and a count was made of days where the maximum temperature was below freezing for each year (1981-2010). The year with the minimum number of days below freezing was 1987 with 73 days (2.4 months). The year with the maximum number of days below freezing was 1996 with 133 days (4.4 months). The average number of days below freezing was 102.8 days or 3.4 months. A triangular distribution was chosen to represent the frozen period, with a minimum of 2.4 months (k_f = 0.2), a mode of 3.4 months (k_f = 0.283), and a maximum of 4.4 months (k_f = 0.367). Figure 10-2 shows the input distribution for k_f and Figure 10-3 shows the resulting composite scale factor, S.

In the water quality modeling, the randomly generated laboratory release rate of sulfate is multiplied by the scale-up factor to produce the field-scale release rate of sulfate. This scaling applies to both the Flotation Tailings and the LTVSMC tailings.

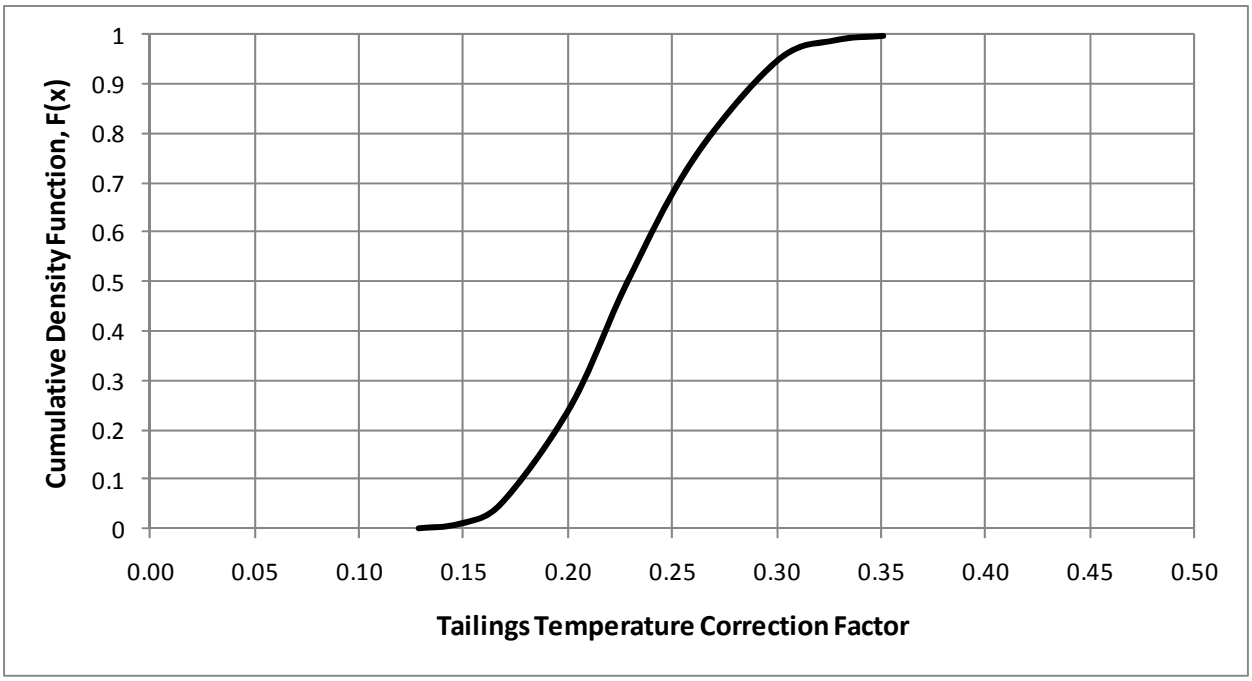

Figure 10-1 Distribution for the Tailings Lab-to-Field Temperature Factor

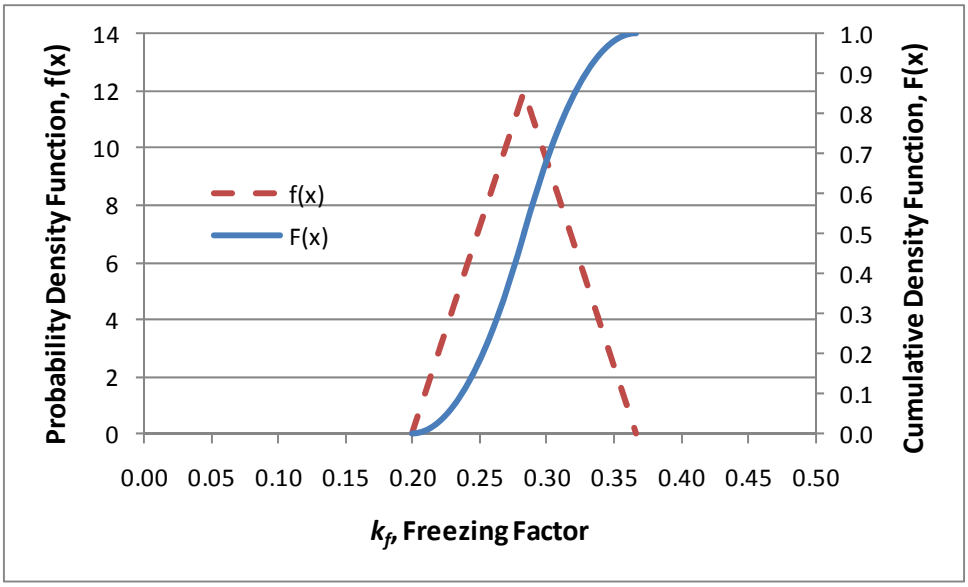

Figure 10-2 Distribution for the Tailings Lab-to-Field Freezing Factor

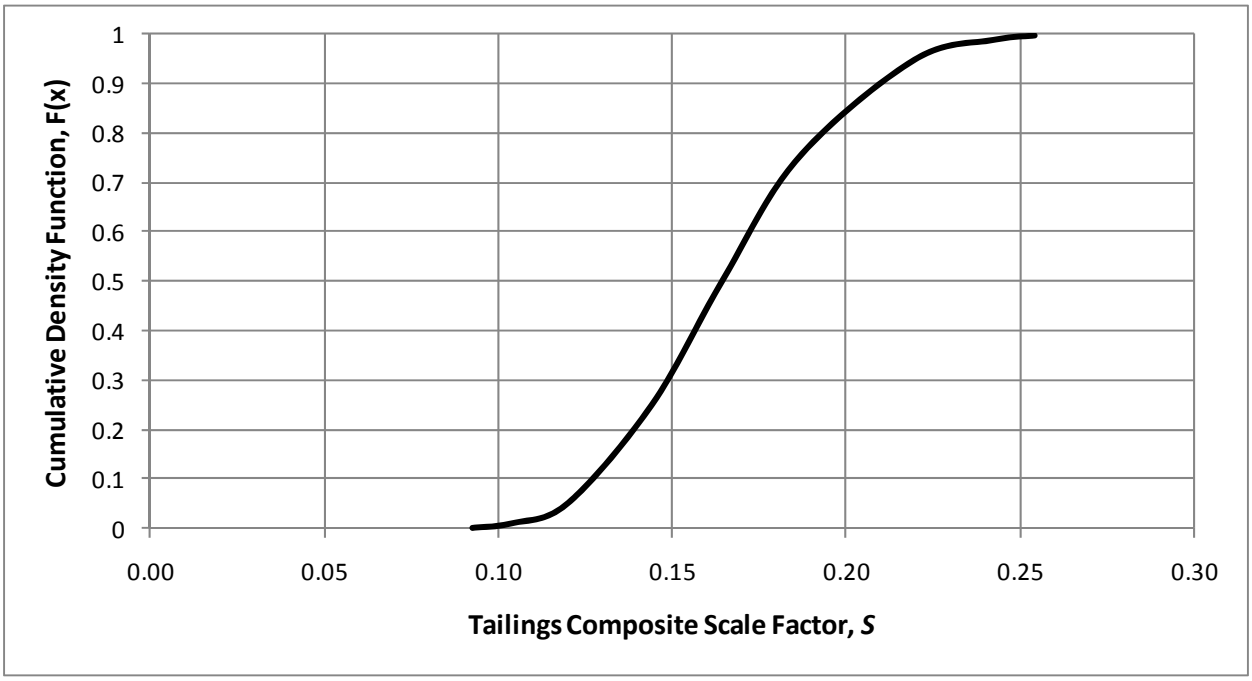

Figure 10-3 Distribution for the Tailings Composite Lab-to-Field Scale Factor

10.2.1 Scaling / Calibration to LTVSMC Field Data

Loading from the Tailings Basin was not modeled for the DEIS. The basin however is included in the modeling for the SDEIS and FEIS. Laboratory release rates from the LTVSMC tailings for sulfate and laboratory release ratios for other constituents are described in detail in Section 10.1.2. There is also existing field data characterizing the quality of the seepage from the existing basin. This section discusses the methods used to modify the laboratory release rates and ratios so that the estimates of the existing conditions model of the Plant Site reasonably reflect the current field conditions at the Plant Site.

Section 10.3 and Section 10.6.1 describe the method by which constituent load is generated within the LTVSMC tailings. Section 5.4.5 of Reference (40) describes the MODFLOW model of the existing Plant Site that is used to describe the existing flow condition (seepage rates, transport direction, etc.). These sections in combination describe the rate at which the generated load within the existing basin and the seepage load from the existing ponds reports to each toe of the Tailings Basin. Furthermore, the total seepage rates and the total loading rates at each toe are used to determine the concentration leaving the basin at each toe for each constituent.

Because the release rates for all constituents are dependent on the sulfate release rate, it is very important to properly capture the sulfate release rate for the different classes of existing LTVSMC tailings. Large Table 18 shows that there is one distribution for the sulfate release

rate for LTVSMC tailings; it does not differentiate by size fraction (i.e., coarse or fine). An existing conditions model was created to determine the loading from the existing LTVSMC basin using the theoretical model described in Section 10.3 and Section 10.6.1.

Several of the wells around the Tailings Basin were chosen to be representative of seepage at each toe of the basin. See Large Figure 5 of Reference (40) for locations of the wells and surface seeps used. See Large Figure 6 of Reference (40) for the delineation of the toes of the basin (e.g., the north toe flows into the north flow path). Water quality data from monitoring locations GW001, GW006, GW007, GW012, SD004, SD026, and PM-7 (another identifier for SD026) were used to establish concentration targets for the GoldSim model calibration. Compared with other monitoring locations, observed concentrations at these locations are relatively similar to those observed at monitoring well GW005, which is completed within the footprint of the Tailings Basin. Based on data collected at GW005, Tailings Basin pore water is characterized by elevated concentrations of total dissolved solids (especially chloride, sulfate, and other major cations and anions). Tailings Basin pore water at GW005 also contains higher concentrations of trace constituents such as molybdenum and fluoride. Monitoring wells GW001, GW006, GW007, and GW012 are completed at the toe of the Tailings Basin and generally exhibit elevated concentrations of constituents characteristic of Tailings Basin pore water. In addition to the water quality at SD004, SD026, and PM-7, which is similar to the Tailings Basin pore water, the physical locations of these monitoring locations allows for minimal influence from the natural environment. Water at SD004 discharges from a steel pipe that emerges from the toe of the Tailings Basin; therefore, it is unlikely that significant dilution from un-impacted water would occur. SD026 is located just downstream of an area where water is observed to be seeping directly out of the Tailings Basin, with minimal potential for dilution. Therefore, as shown in Table 10-1, GW001 and GW012 are used to represent the seepage to the north toe, GW006 is used to represent the seepage to the northwest toe, GW007 and SD004 are used to represent seepage to the west toe, and SD026 and PM-7 are used to represent seepage to the south toe.

The average concentration was used as the target seepage concentration. Table 10-1 shows the four toes of the basin, the sampling locations chosen to represent the basin seepage, and the average concentration of the samples at those locations. The seepage rate from the basin at each toe is determined by summing the infiltration rates throughout the basin and the pond seepage rates, assuming that there is no storage (i.e., steady-state). Mean values were used for all model inputs (i.e., infiltration rates, air temperature, freezing factor, etc.).

The total load leaving the Tailings Basin does not come entirely from oxidizing tailings. A portion of the load comes directly from the seepage from the ponds in Cells 1E and 2E. This load is accounted for by using an average pond SO4 concentration (sampling in 2001-2004) and constant pond seepage rates determined by the calibrated MODFLOW model (Section 5.4.5 of Reference (40)). The seepage rate and sulfate concentration from the pond in Cell 2E to the basin toes is approximately 580 gpm and 130 mg/L respectively. The seepage rate and sulfate concentration from the pond in Cell 1E to the basin toes is

approximately 900 gpm and 95 mg/L respectively. Using the total load leaving the basin from Table 10-1 (4150 kg/day), and the total load seeping from the existing ponds (880 kg/day), the tailings are generating a load of about 3270 kg/day due to oxidation.

Table 10-1 Input values for the target concentrations and loading from the basin

Basin Toe	Sampling Locations	Average SO_4 Seepage Concentration (mg/L)	Number of Samples	Approximate Seepage Rate (gpm)	Average Loading Rate (kg/day)
North	GW001, GW012	224	30	1540	1880
North-West	GW006	505	18	440	1210
West	GW007, SD004	251	42	610	835
South	SD026	181	113	230	225

The results from the existing conditions model showed a significant over-estimation of sulfate concentrations, reflecting the large degree of conservatism already built into the model assumptions (Figure 10-4). The red stars are the model's estimation of seepage concentrations. The bars show the minimum, average, and maximum of the sampled concentrations at each toe.

Therefore, two sulfate release rate correction factors are suggested, one for each of the two different LTVSMC tailings classes (coarse and fine). Modeling the LTVSMC tailings as separate coarse and fine fractions is a similar method to that proposed for the Flotation Tailings in Section 10.1.1. The sulfate correction factors are applied to the generic LTVSMC tailings sulfate release rate, creating two new calibrated sulfate release rates; one for the coarse LTVSMC tailings and one for the fine LTVSMC tailings (the dams of the existing basin are assumed to be constructed with coarse tailings). These factors alter the laboratory sulfate release rate so that the estimated sulfate concentrations at the toes of the existing basin are much closer to the concentrations seen in the current field data.

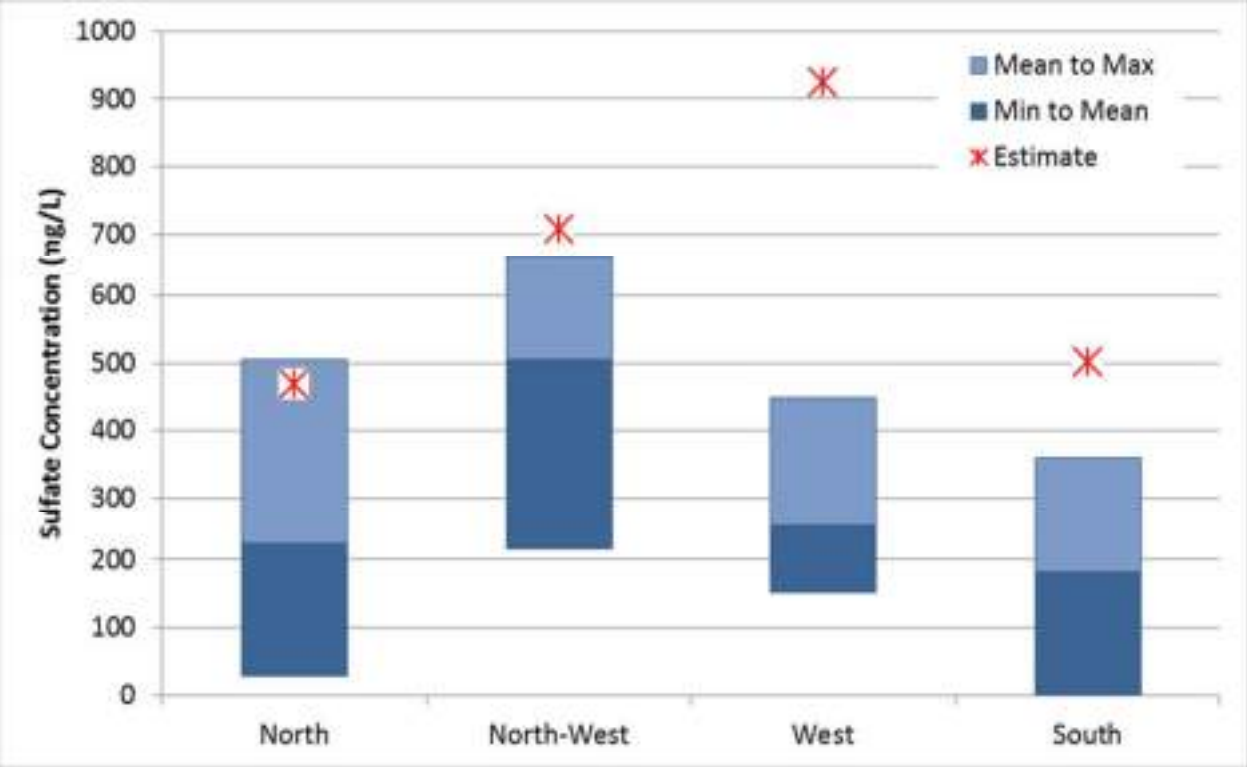

Figure 10-4 Sulfate concentrations of the un-calibrated existing conditions model

To constrain the calibration, the total modeled load leaving the basin using mean values for all inputs was set equal to the total sampled load leaving the basin (Equation 10-7).

$$\left[\sum_{i=1}^{4} \bar{C}_i \bar{Q}_i\right]_{Field} = \left[\sum_{j=1}^{4} \bar{C}_j \bar{Q}_j\right]_{Model} \quad \textbf{10-7}$$

Due to this constraint, one of the calibration factors becomes a function of the other (fine tailings factor is a function of the coarse tailings factor). In other words, because the total load is fixed, if one calibration parameter increases, the other must decrease to maintain the fixed total load. Therefore, only one parameter was adjusted (coarse tailings factor) to minimize the calibration objective function (Equation 10-8).

$$max\left[abs\left(\bar{C}_{i,Field} - \bar{C}_{i,Model}\right) * Q_i\right] \quad \textbf{10-8}$$

Equation 10-8 is simply the greatest loading error between samples and model estimates at the toes. The coarse tailings calibration factor was adjusted and the fine tailings calibration factor was held as a function of the coarse tailings factor. The values that gave the best results were 0.2 and 0.7 for the coarse and fine factors respectively. When the LTVSMC

tailings release rates are multiplied by these factors, the model estimates are much closer to the sampled data (Figure 10-5).

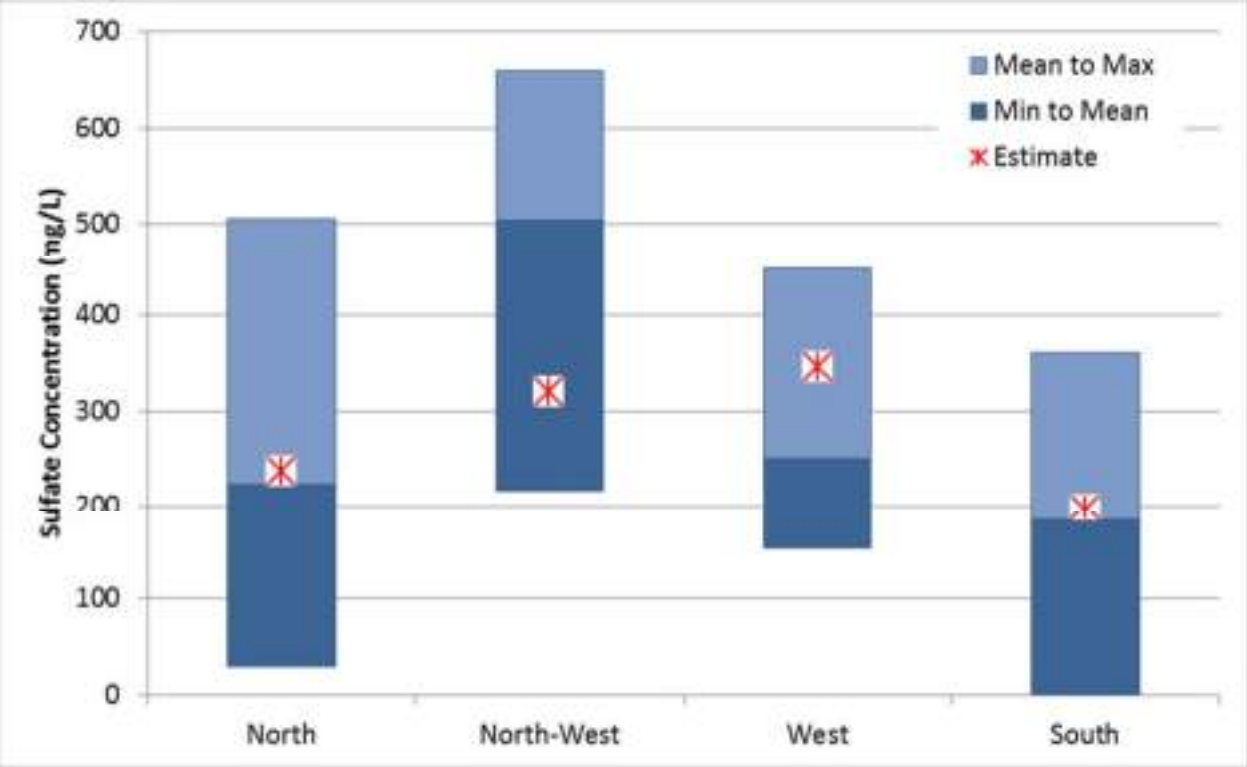

Figure 10-5 Sulfate concentrations of the calibrated existing conditions model

As seen in Figure 10-5, the estimates of the sulfate concentrations at the north and south toes are close to the average sampled in the field. The northwest and west estimates tend to under-estimate and over-estimate the means. However, due to the constraint of Equation 10-7, the total load leaving the Tailings Basin is equal to the average load expected from sampling data. Even though the calibrated model estimates are not perfect, it is clear that calibration factors are needed so that the model reflects the sampled data. Without these calibration factors, the model would simulate more than twice the load that the sampling data suggests is generated.

The existing conditions model was also used to test model estimates of all other model constituents (Cu, Ni, etc.) against field data. Figure 10-6 shows the model estimates using the laboratory release ratio of Co to S (0.041 mg Co / mg S).

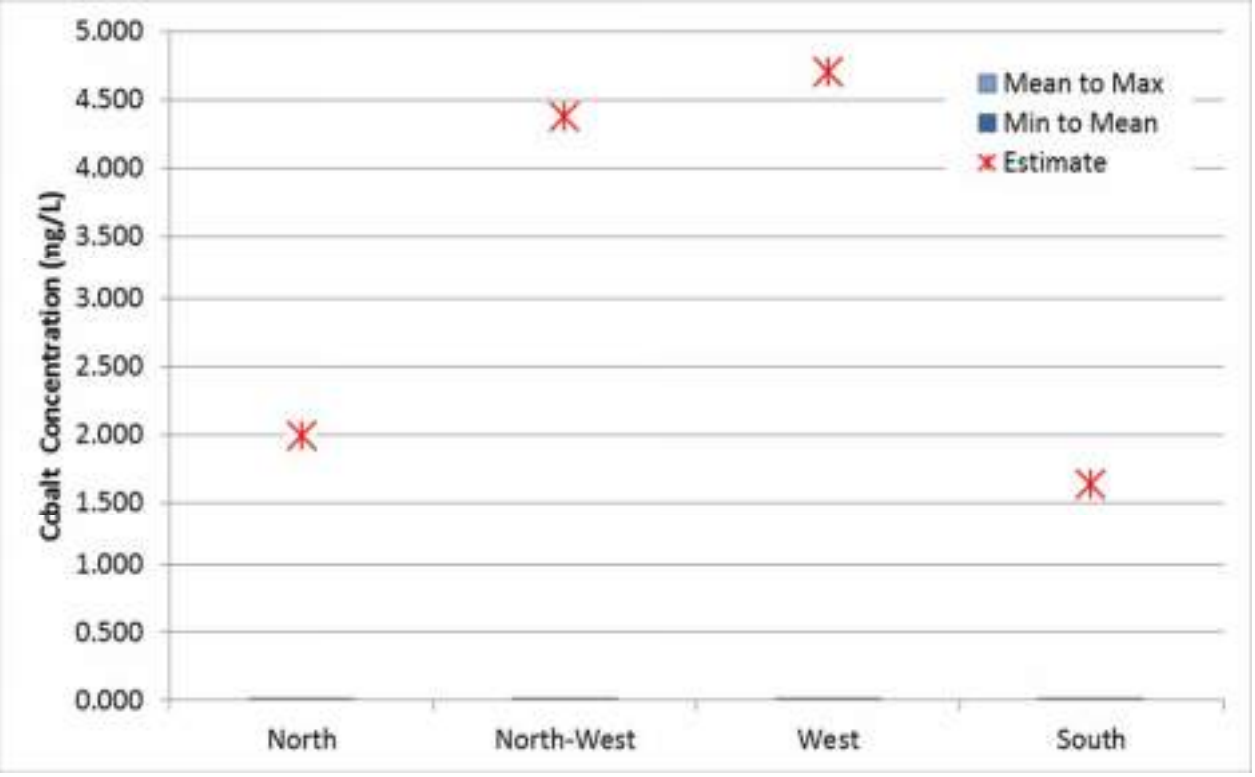

Figure 10-6 Cobalt concentrations of the un-calibrated existing conditions model

Even while using calibrated sulfate release rates, the model over-estimated concentrations of all other constituents, indicating that the metal release ratio for each constituent is also conservatively high. Therefore, an additional constituent-specific correction factor is suggested to be applied to the release ratios. The suggested correction factor alters the release ratio of each constituent so that the concentrations at the toe of the basin in the existing conditions model are reasonably close to the observed field water quality data. For this calibration though, there is only one parameter. Therefore, Equation 10-7 determines this parameter and Equation 10-8 is not used for the calibration of release ratios of constituents other than sulfate. For cobalt, the calibration factor is 0.0005, making the overall release ratio 2.05 $x10^{-5}$ mg Co / mg S. Figure 10-7 shows the model results when the calibration factor is used for cobalt.

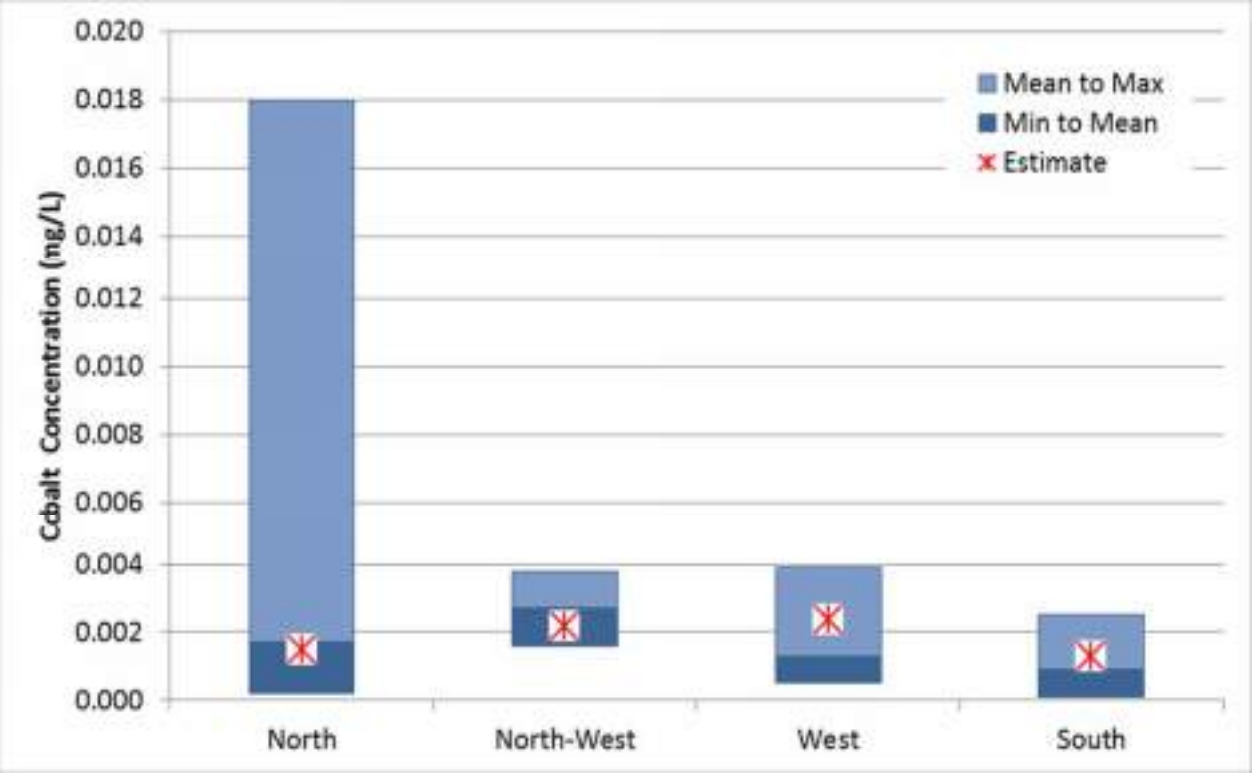

Figure 10-7 Cobalt concentrations of the calibrated existing conditions model

As for the sulfate calibration plot (Figure 10-5), the model results do not perfectly match the data. However, these release ratio factors are clearly needed because model estimates of cobalt concentrations between 2 mg/L and 5 mg/L are not acceptable when the sampled data shows ~2 µg/L.

The correction factors mentioned in this section (one sulfate release factor for coarse tailings, one sulfate release factor for fine tailings, and one release ratio factor for each of the other constituents) only apply to the existing LTVSMC tailings because the field data is only representative of those tailings (Table 1-1 and Table 1-21 of Attachment B to Reference (40). No correction factors have been applied to the Flotation Tailings because field data does not exist to defensibly modify the laboratory release rates.

10.3 Saturation and Oxygen Diffusion

This section explains the governing equations and the input variables that are used in the water quality modeling to determine saturation and oxygen diffusion.

Oxygen is the necessary driver in the pyrrhotite reaction which produces sulfate. Equation 10-9 is the chemical reaction used in the water quality modeling (Reference (7)). It states that for every 9 moles of oxygen consumed, 4 moles of sulfate are produced. This ratio (4:9) is a direct input in the water quality model.

$$4FeS_{(s)} + 9O_2 + 10H_2O \xrightarrow{yields} 4Fe(OH)_{3(s)} + 4H_2SO_4 \qquad \textbf{10-9}$$

The quantity of oxygen within the tailings is controlled by the chemical reaction rate and the rate at which oxygen can diffuse into the tailings from the atmosphere. Equation 10-10 is the assumed governing oxygen transport equation. It is a steady-state, one-dimensional form of Fick's law with a zero-order reaction term.

$$r = D\frac{\partial^2 C}{\partial z^2} \qquad \textbf{10-10}$$

where r is the rate at which oxygen is consumed [mol/m^3/s], D is the diffusion rate [m^2/s], C is the concentration of oxygen at any depth [mol/m^3] and z is depth [m] (z = 0 at the surface and is positive downward). The diffusion rate is largely controlled by the saturation of the tailings. Equation 10-11 is used to calculate D from saturation (Reference (7)).

$$D = \tau D_a[1 - S]^c + \tau S\frac{D_w}{K_H} \qquad \textbf{10-11}$$

where τ is the tortuosity factor [--] and equals 0.273, D_a is the free diffusion coefficient of oxygen in air [m^2/s] and equals 1.80x10^{-5}, c is an empirical coefficient [--] and equals 3.28, D_w is the free diffusion coefficient of oxygen in water [m^2/s] and equals 2.20x10^{-9}, S is the saturation [--] and K_H is Henry's constant for oxygen [--] and equals 33.9 (Reference (7)).

Using this calculation method, the production of sulfate is a function of the available amount of water and the hydraulic properties of the tailings, which together will determine the saturation of the tailings. Because a portion of the tailings is not completely saturated unsaturated flow equations must be used.

10.3.1 Unsaturated Flow

A steady-state Richard's Equation (10-12) is the governing equation for volumetric water content in an unsaturated media and is a modification of Darcy's Law which is used for flow in porous saturated media.

$$0 = \frac{\partial}{\partial z}\left[K\left(\frac{\partial \varphi}{\partial z} + 1\right)\right] \qquad \textbf{10-12}$$

where K is the unsaturated hydraulic conductivity [L/T] ranging from 0 cm/s to the saturated hydraulic conductivity (K_{sat}) and φ is the pressure head [L].

The Van Genuchten model (Equation 10-13) is used to determine the water content as a function of the pressure head and the tailings specific hydraulic properties.

$$\sigma = \theta_r + \frac{\theta - \theta_r}{(1 + \alpha|\varphi|^{\beta})^{\gamma}} \qquad \textbf{10-13}$$

where σ is the volumetric content [cm^3/cm^3], θ_r is the residual moisture content [cm^3/cm^3], θ is the porosity or the saturated moisture content [cm^3/cm^3], α is the air entry suction [1/cm], β is a Van Genuchten parameter [--] and γ is related to β by $\gamma = 1 - 1/\beta$. All of the parameters in the Van Genuchten model are related to the characteristics of the tailings beaches. Using laboratory data, they have been related to the % fines (by mass) and the porosity of the beach, as discussed below.

The *effective* water saturation is related to the moisture content using equation 10-14.

$$S_e = \frac{\sigma - \theta_r}{\theta - \theta_r} \qquad \textbf{10-14}$$

K is related to the *effective* water saturation using Equation 10-15. K is the known infiltration rate which, in unsaturated flow, is naturally less than K_{sat}.

$$K = K_{sat}(S_e^{0.5})\left[1 - \left(1 - S_e^{1/\gamma}\right)^{\gamma}\right]^2 \qquad \textbf{10-15}$$

These four unsaturated flow equations are highly non-linear and must be solved simultaneously using an iterative approach. Once they are solved, the *effective* saturation can be converted to the real saturation (Equation 10-16), which is used in Equation 10-11 to calculate the diffusion rate.

$$S = \frac{\sigma}{\theta} \qquad \textbf{10-16}$$

10.3.2 Hydraulic Parameters for Flotation Tailings

Daniel B. Stephens and Associates (DBS&A) in Albuquerque, New Mexico received a large sample of tailings from pilot plant testing of NorthMet ore. As described in their laboratory report (included as Attachment G), the tailings were separated into coarse and fine fractions (split on a #200 sieve) and re-mixed into four different blends based on percent fine by mass. Each of the four blends was also given three different target porosity values, making a total of twelve samples that were further tested. Table 10-2 summarizes hydraulic parameters of the twelve samples and the results from the tests. The original data is provided in Attachment G.

Table 10-2 Hydraulic Parameter Results from the DBS&A Laboratory Study

Blend	% Fines	θ	Bulk Density	K_{sat}	α	β	θ_r
	[g/g]	[--]	[g/cm^3]	[cm/s]	[1/cm]	[--]	[--]
1	0.77	0.381	1.85	7.4E-05	0.0028	4.4060	0.0210
2	0.77	0.453	1.64	1.4E-04	0.0037	2.3302	0.0204
3	0.77	0.557	1.32	4.9E-04	0.0032	1.5352	0.0000
4	0.56	0.396	1.80	2.4E-04	0.0039	3.4263	0.0155
5	0.56	0.452	1.63	5.2E-04	0.0049	2.3810	0.0136
6	0.56	0.558	1.32	1.4E-03	0.0097	1.5315	0.0020
7	0.37	0.397	1.80	6.9E-04	0.0112	1.9686	0.0208
8	0.37	0.452	1.63	2.0E-03	0.0065	2.6280	0.0117
9	0.37	0.560	1.31	1.3E-03	0.0066	2.4829	0.0118
10	0.21	0.448	1.64	3.7E-03	0.0106	2.7451	0.0081
11	0.21	0.503	1.48	3.9E-03	0.0148	2.7336	0.0107
12	0.21	0.551	1.34	9.8E-03	0.0127	2.6317	0.0095

First, the results for % fines, porosity (θ) and bulk density were used to develop a best-fit for the specific gravity of both the coarse and fine solids. The results from this best-fit study showed that the specific gravity for the coarse and fine solids is 2.97 and 2.99 respectively. DBS&A measured the specific gravity of both the coarse and fine fractions and the results were 2.98 and 3.00. The essentially identical comparison shows the strong confidence in the calculation and measurements and that both fractions can be assumed to have a specific gravity of 3.0.

Second, the results for the van Genuchten parameters were used to create surface functions that solve for these parameters based on the % fines and θ as independent variables. These functions are used in the probabilistic water quality model to determine the hydraulic parameters of the tailings. The hydraulic parameters themselves are not considered random variables in the model. However, the variability in these parameters is accounted for by simulating the uncertainty in the % fines and the porosity.

Equations 10-17 and 10-18 show the format of the surfaces fit to the laboratory data. The coefficients were solved for using best-fits to the data and they are shown in Table 10-3.

$$\log(K_{sat}) = m_m(F)(\theta) + b_m(\theta) + m_b(F) + b_b \quad \textbf{10-17}$$

$$\alpha, \beta, \theta_r = m_m(F)(\theta) + b_m(\theta) + m_b(F) + b_b \quad \textbf{10-18}$$

where F is the % fines in the tailings and θ is the porosity.

Table 10-3 Coefficients used to Determine Hydraulic Parameters (Eq. 10-17 & 10-18)

Value	Units	mm	bm	mb	bb
Ksat	[cm/s]	2.7930	2.4585	-3.6293	-3.1175
α	[1/cm]	0.002036	0.008121	-0.015927	0.010728
β	[--]	-31.3442	8.6015	14.6871	-1.4748
θr	[--]	-0.2417	0.0543	0.1173	-0.0155

Figure 10-8 through Figure 10-11 show the resulting surfaces used to determine hydraulic properties as a function of the random independent variables % fines and porosity. In each case, the surface is colored from blue (low value) to red (high value). The laboratory data are also shown alongside the surface (black circles) to show how the data fit to the surface.

It is important to note that these hydraulic parameters are used primarily for the water quality modeling in the NorthMet beaches. In those areas, the % fines is typically between 0.30 and 0.40 and the porosity is typically between 0.38 and 0.45 as described in Section 5.1.3.1.

Table 10-4 shows the assumed van Genuchten parameters for the bentonite-amended tailings (both NorthMet and LTVSMC) which will be used to cover the dams and the beaches in reclamation. The values are taken from HYDRUS-1D using the default soil properties of a silty-clay.

Table 10-4 Van Genuchten parameters for the bentonite-amended tailings

% Solid Volume is bentonite	Saturated Hydraulic Conductivity	Porosity	Residual Moisture Content	Van Gen. α	Van Gen. β
[--]	[cm/s]	[--]	[--]	[1/cm]	[--]
3%	5.56e-06	0.36	0.07	0.005	1.09

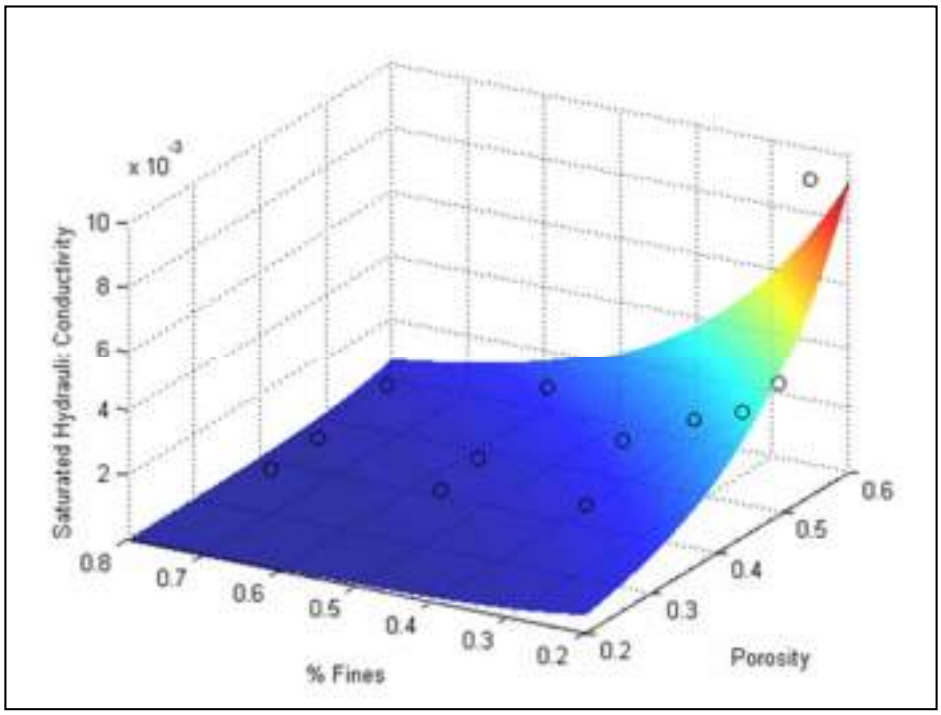

Figure 10-8 Saturated Hydraulic Conductivity (K_{sat}) Surface

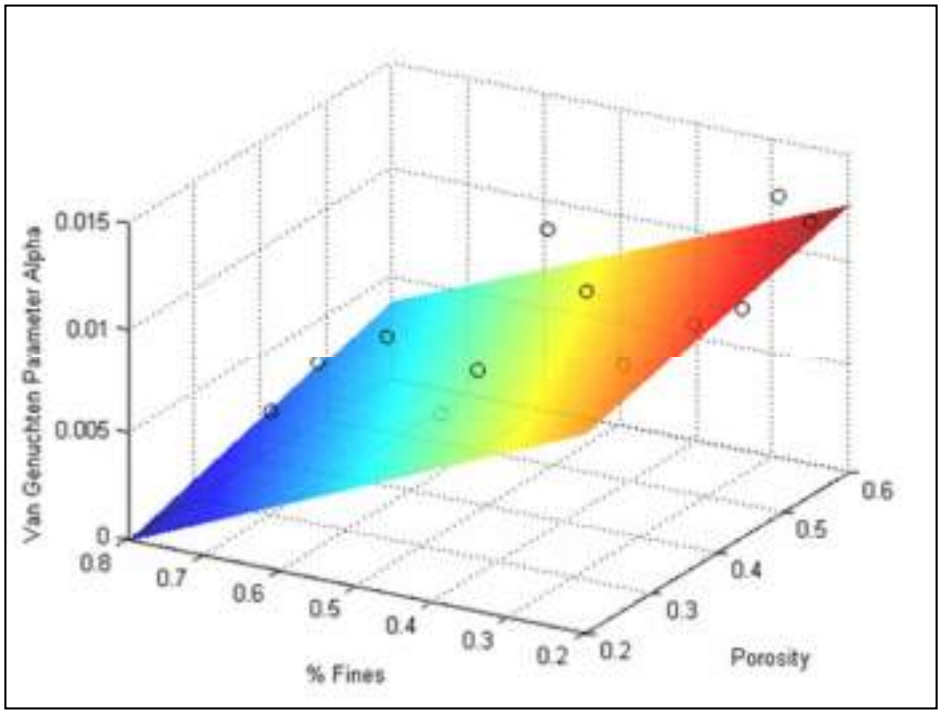

Figure 10-9 Van Genuchten Parameter α Surface

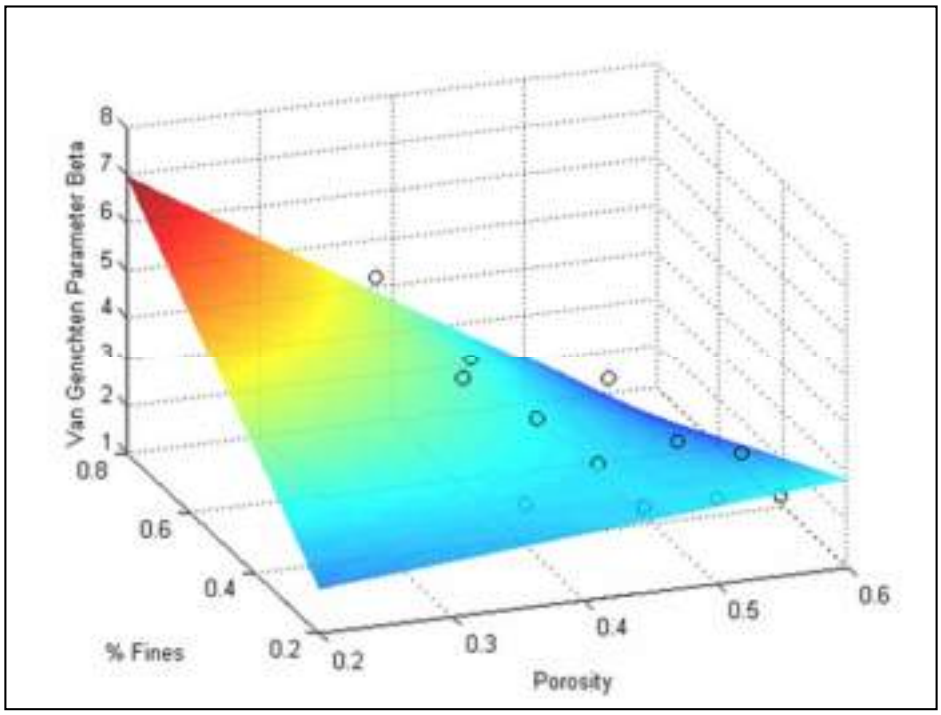

Figure 10-10 Van Genuchten Parameter β Surface

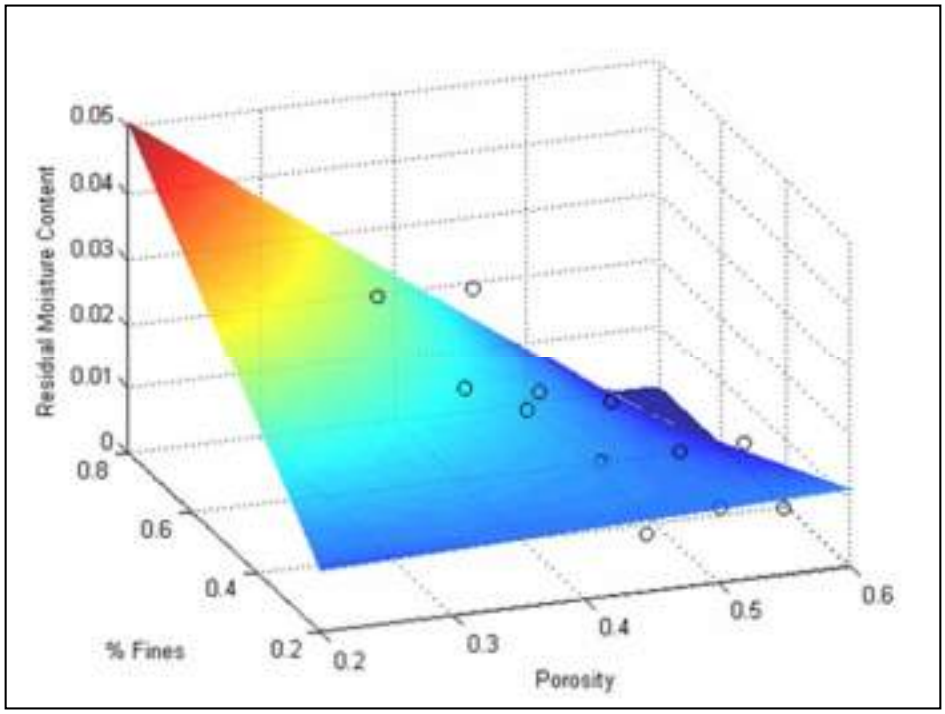

Figure 10-11 Residual Moisture Content (θ_r) Surface

10.3.2.1 Expected Saturation Levels of the FTB Dams and Beaches

In contrast to the modeling of the Flotation Tailings Basin for the DEIS, saturation is not a direct input into the current modeling. Instead, saturation is calculated as shown in Section 10.3.1. This section describes the expected, model-calculated saturation levels in the Flotation Tailings Basin (dams and beaches) based on hydraulic characteristics and available water for infiltration,

During early years of operations (up to Mine Year 7), there is only one dam constructed to form the FTB; the North Dam. During those early years, there is also only one beach which receives all of the Beneficiation Plant discharge. Because of the high discharge rate from the Beneficiation Plant, the beach saturation is maintained at a high level, approximately between 45% and 48% saturation as defined by Equation 10-16. Once the two cells of the FTB combine, the discharge from the Beneficiation Plant is moved around the basin to build up all three dams and beaches. This means that during each year, each beach will go through a period of high saturation (when the Beneficiation Plant is discharging to it) and low saturation (when the Beneficiation Plant is not discharging to it). Finally, upon closure, the beaches do not receive discharge from the Beneficiation Plant and are only wetted by precipitation. Additionally, the beaches are covered by a bentonite-amended tailings layer which, due to its hydraulic properties, maintains a very high saturation level. See Table 10-5 for a full summary of the expected saturation levels.

In Table 10-5, Mine Years 1 and 7 are when only the North Dam exists. Mine Year 19 is towards the end of operations when the FTB Pond is completely surrounded by the NorthMet beach. Mine Year “19 (dry)” represents the times of the year when the Beneficiation Plant is *not* discharging to that particular beach. It makes sense that those values are the same for all beaches because it is only affected by precipitation. Mine Year “19 (wet)” represents the times of the year when the Beneficiation Plant *is* discharging to that particular beach. The saturation during those times is dependent on the duration of discharge.

In the DEIS modeling (Reference (8)), saturation levels of NorthMet bulk tailings during in closure were estimated to be about 58%. Initial modeling of the separate NorthMet coarse and fine tailings estimated saturation levels in closure of 28% and 80%, respectively. The saturation levels of the dams were estimated at 35% in closure (no bentonite amendment), and the bentonite-amended layer of the beaches was estimated at 93%. According to the hydraulic properties described in Section 10.3.2, the saturation levels of the Flotation Tailings beneath the bentonite-amended layer in the SDEIS modeling should be closer to the NorthMet *coarse* tailings of the DEIS modeling. The modeled saturation of the Flotation Tailings is different from the estimated saturation of the NorthMet coarse tailings of the DEIS modeling because the measured material hydraulic properties are different from those assumed for the DEIS modeling (notably a change in the porosity and the saturated hydraulic conductivity).

Table 10-5 Expected Saturation (%) in the Tailings Layers of the FTB

Tailings Layer[1]		Mine Year				
		1	7	19 (dry)	19 (wet)	Closure
North Dam	Beach	45-47	46-48	18-22	36-39	14-22
	Beach-BNT	--	--	--	--	95-97
	Dam	45-53	45-53	45-53	--	45-53
	Dam-BNT	97-98	97-98	97-98	--	97-98
East Dam	Beach	--	--	18-22	25-34	14-22
	Beach-BNT	--	--	--	--	95-97
	Dam	--	--	45-53	--	45-53
	Dam-BNT	--	--	97-98	--	97-98
South Dam	Beach	--	--	18-22	33-41	14-22
	Beach-BNT	--	--	--	--	95-97
	Dam	--	--	45-53	--	45-53
	Dam-BNT	--	--	97-98	--	97-98
Closure	Beach	--	--	18-22	35-38	14-22
	Beach-BNT	--	--	--	--	95-97

(1) "Beach" represents the Flotation Tailings that are not amended with bentonite, "Beach-BNT" represents the layer of bentonite-amended Flotation Tailings (only in closure), "Dam" represents the LTVSMC tailings used to construct the dams, "Dam-BNT" represents the bentonite-amended LTVSMC tailings used to cover the newly constructed dams.

10.3.2.2 Expected Saturation Levels Below the FTB Pond

Upon closure, the Flotation Tailings below the bentonite-amended layer in the FTB Pond bottom will transition from a saturated to an unsaturated state. Although the Flotation Tailings in this area will be unsaturated, the load generation in the tailings below the FTB Pond will be limited by transport of dissolved oxygen in the downward-seeping water (see Section 10.6.1). The estimated saturation levels below the FTB Pond are provided here for reference only and are not used to calculate loading rates in the water quality modeling.

Saturation levels below the bentonite-amended layer were estimated assuming a) steady, downward flow and, hence, a water table far below the seepage face and b) steady flow under the influence of a water table. The Buckingham-Darcy flux law, a modification of Darcy's Law to account for the non-linear function of unsaturated hydraulic conductivity with respect to pressure head was used to solve for saturation under steady, downward flow. At steady state, the Buckingham-Darcy equation is as shown in Equation 10-19:

$$J_w = -K(h)\left[\frac{dh}{dz} + 1\right] \quad \textbf{10-19}$$

where J_w is water flux per unit cross-sectional area per unit time [L/T], K(h) is the unsaturated hydraulic conductivity [L/T] and h is the pressure head [L]. Assuming steady, downward flow with a sufficiently deep water table, the water flux, J_w, simplifies to Equation 10-20:

$$J_w = -K(h) \quad \textbf{10-20}$$

The unsaturated hydraulic conductivity function, K(h), can be defined using the van Genuchten relationship (Equation 10-21), which can also be written in terms of saturation (Equation 10-22):

$$K(S_e) = K_{sat} S_e^l [1 - \left(1 - S_e^{\frac{1}{m}}\right)^m]^2 \quad \textbf{10-21}$$

$$S_e = \frac{\theta - \theta_r}{\theta_s - \theta_r} \quad \textbf{10-22}$$

Where S_e is the effective saturation [unitless], K_{sat} is saturated hydraulic conductivity [L/T], l is a pore connectivity factor [unitless], m is a fitting parameter [unitless], θ is water content [L^3/L^3], θ_r is residual water content [L^3/L^3] and θ_s is saturated water content [L^3/L^3].

The parameters for Equation 10-21 were estimated from the surface functions shown in Figure 10-8 through Figure 10-11 assuming 68% fines and a porosity of 0.52 (Equation 10-17, Equation 10-18, and Table 10-3). These values of 68% fines and 0.52 porosity are based on the assumptions of 35% fines in the Flotation Tailings beaches, 30% delivery of Flotation Tailings delivery to the beaches (70% subaqueously), and 38% coarse material in the feed. The resulting values for K_{sat}, l, m (or $1 - 1/\beta$), θ_s and θ_r were estimated for the Flotation Tailings to be 4.8 x 10^{-4} cm/sec, 0.5, 0.474, 0.52 and 0.007, respectively. Using these parameter values and the design seepage rate of 6.5 inches/year through the bentonite-amended layer (Section 5 of Reference (36)) the Flotation Tailings saturation under steady, downward flow was estimated to be 33% using the analytical solution of the Buckingham-Darcy flux law. This saturation is equivalent to a volumetric water content of about 0.171, resulting in an effective saturation (S_e) of about 0.32 (see Equation 10-22). Assuming free gravity drainage (or sufficient vertical distance above the water table) using Equation 10-21, an effective saturation of 0.32, and the other parameter values mentioned above as a double-check yields a $K(S_e)$ of 5.233 x 10^{-7} cm/sec which is equivalent to an unsaturated flow rate of 6.5 in/yr.

To estimate saturation assuming the presence of a water table below the FTB, a numerical solution to the Richard's Equation, the governing equation for transient, unsaturated flow, was used (Equation 10-23):

$$\frac{\partial \theta(h)}{\partial t} = \frac{\partial}{\partial z}\left[K(h)\left(\frac{\partial h}{\partial z}\right)\right] + \frac{\partial K(h)}{\partial z} - S(h) \qquad \textbf{10-23}$$

where S(h) is a sink function [T^{-1}]. The van Genuchten model relating pressure head to water content was used and the unsaturated flow equations were solved numerically using the HYDRUS-1D software package and the same input parameters used above for the analytical solution. The top model boundary condition was set as a constant flux of 6.5 inches/year, the seepage rate for the long-term closure FTB, and the bottom boundary condition was set at a constant pressure head of 0 m at a depth 17.5 m below the model top, representative of the water table. Using this numerical simulation, the effects of the water table were incorporated and the saturation was found to vary between 33% immediately below the bentonite amended Flotation Tailings to 100% (saturated) at the water table with an average saturation of 44%. As dictated by the model boundary condition, the steady state model resulted in fluxes through the top and bottom model boundaries of 6.5 inches/year.

Although the Flotation Tailings below the FTB Pond will not be fully saturated, oxygen delivery to this material is limited to the dissolved oxygen carried in the seepage through the bentonite-amended layer. Unlike the unsaturated tailings in the FTB dams and beaches, there will be no diffusion of atmospheric oxygen to the Flotation Tailings below the FTB Pond. Constituent release from this material is modeled as discussed in Section 10.6.1 and is not a function of the saturation levels discussed here.

10.3.3 Hydraulic Parameters for LTVSMC Tailings

The existing LTVSMC tailings are included in the probabilistic water quality model. The Van Genuchten parameters for these tailings must be known in order to perform the unsaturated hydraulic modeling. The geotechnical seepage and stability modeling being performed for the Project also incorporates the moisture-release characteristics of the existing LTVSMC tailings. For consistency, the input values used for the seepage and stability modeling inform the values used in the water quality modeling.

The geotechnical seepage and stability model includes the three classes of LTVSMC tailings used in the water quality modeling which are: coarse tailings, fine tailings, and bulk tailings (used to construct the planned dams). The parameters used in the geotechnical modeling are specific gravity, porosity, and residual moisture content. The remaining required Van Genuchten parameters in the water quality model (α and β), are based on the D60 and D10 information obtained from lab tests of the LTVSMC tailings (Table 10-7) and estimates from available literature data (Reference (41)). The saturated hydraulic conductivity values needed were taken from the calibrated existing conditions MODFLOW model described in Section 5.4 of Reference (40) and are shown in Table 10-6.

Table 10-7 shows a summary of the values used for the unsaturated hydraulic modeling of the existing LTVSMC tailings.

Table 10-6 Saturated Hydraulic Conductivity Values for the Existing LTVSMC Tailings

	Cell 1E [cm/s]	Cell 2E [cm/s]	Cell 2W [cm/s]
LTVSMC Coarse Tailings	4.43e-03	1.76e-03	1.42e-03
LTVSMC Fine Tailings	2.12e-05	3.77e-04	7.06e-05

Note:
Values come from the calibrated MODFLOW model (Large Table 1 of Attachment A to Reference (40))

Table 10-7 Van Genuchten Parameters for the Existing LTVSMC Tailings

Fraction	D_{10}(1)	D_{60}(1)	Specific Gravity(1)	Saturated Hydraulic Cond.	Porosity(1)	Residual Moisture Content(1)	Van Gen. α(2)	Van Gen. β(2)
	[mm]	[mm]	[--]	[cm/s]	[--]	[--]	[1/cm]	[--]
Coarse	0.05	0.70	2.80	Table 10-6	0.412	0.041	0.024	2.0
Fine	0.045	0.03	2.90	Table 10-6	0.493	0.059	0.001	1.6
Bulk	0.03	0.25	2.85	8.02e-05(1)	0.440	0.048	0.011	2.0

(1) Values adopted from the geotechnical seepage and stability modeling (Reference (42), Sections 5.2.1, 5.2.2, and 5.8).
(2) Values derived from measured particle size distribution and literature values (Reference (41)).

10.3.4 Oxygen Diffusion

To determine the oxygen profile and the depth to which oxygen is available for consumption, the oxygen transport equation (Equation 10-10) must be solved and the rates at which oxygen is consumed and diffuses into the tailings must be known. The diffusion rate is dependent on saturation, as shown in Equation 10-11. The oxygen consumption rate is dependent on the sulfate production rate which is a random variable in the water quality model (Section 10.1).

Because the relationship between oxygen consumption and sulfate production is known (Equation 10-9), the mass sulfate production rate can be used to determine the molar oxygen consumption rate. For example, if sulfate is produced at a rate of 5 mg of sulfate per kg of tailings per week, the bulk density of the tailings is 1.50, the porosity of the tailings is 0.50 and the concentration of oxygen in air is 8.89 moles per cubic meter of air, then the oxygen consumption rate is 1.83×10^{-7} moles per cubic meter of air per second.

The general form of the solution to Equation 10-10 is shown in Equation 10-24.

$$C(z) = \frac{r}{2D}z^2 + C_1 z + C_2 \qquad \textbf{10-24}$$

The boundary conditions used to solve for the unknown coefficients C_1 and C_2 are that the concentration at the surface is equal to the concentration of oxygen in air (C_o) (Equation 10-25) and that the concentration and the concentration gradient both go to zero at the same depth (Equation 10-26).

$$C(0) = C_o \qquad \textbf{10-25}$$

$$C(z') = \frac{\partial C(z')}{\partial z} = 0 \qquad \textbf{10-26}$$

From the general solution and the boundary conditions, the specific solution is shown in Equation 10-27. An additional limitation is that the tailings are only producing sulfate to the depth where the oxygen profile goes to zero. The depth at which oxygen is completely depleted (z') is shown in Equation 10-28. Figure 10-12 shows an example oxygen profile given that the concentration of oxygen in air is 8.89 mol/m^3, the reaction rate is 1.83x10^{-7} mol/m^3/sec and the diffusion rate is 4.117x10^{-8} m^2/sec.

$$C(z) = \frac{r}{2D}z^2 - \left(\frac{2C_o r}{D}\right)^{0.5}(z) + C_o \qquad \textbf{10-27}$$

$$z' = \left(\frac{2C_o D}{r}\right)^{0.5} \qquad \textbf{10-28}$$

Figure 10-13 shows the effect of the r/D ratio on the oxygen profile. Intuitively, if the rate at which oxygen is consumed is high compared to the rate at which oxygen can diffuse, the penetration depth of oxygen will be quite shallow. At an r/D ratio of 4, the penetration depth is about 2 meters. On the other hand, if oxygen is not being consumed quickly compared to the diffusion rate (r/D = 0.25 for example), then the penetration depth is quite deep, around 8 meters.

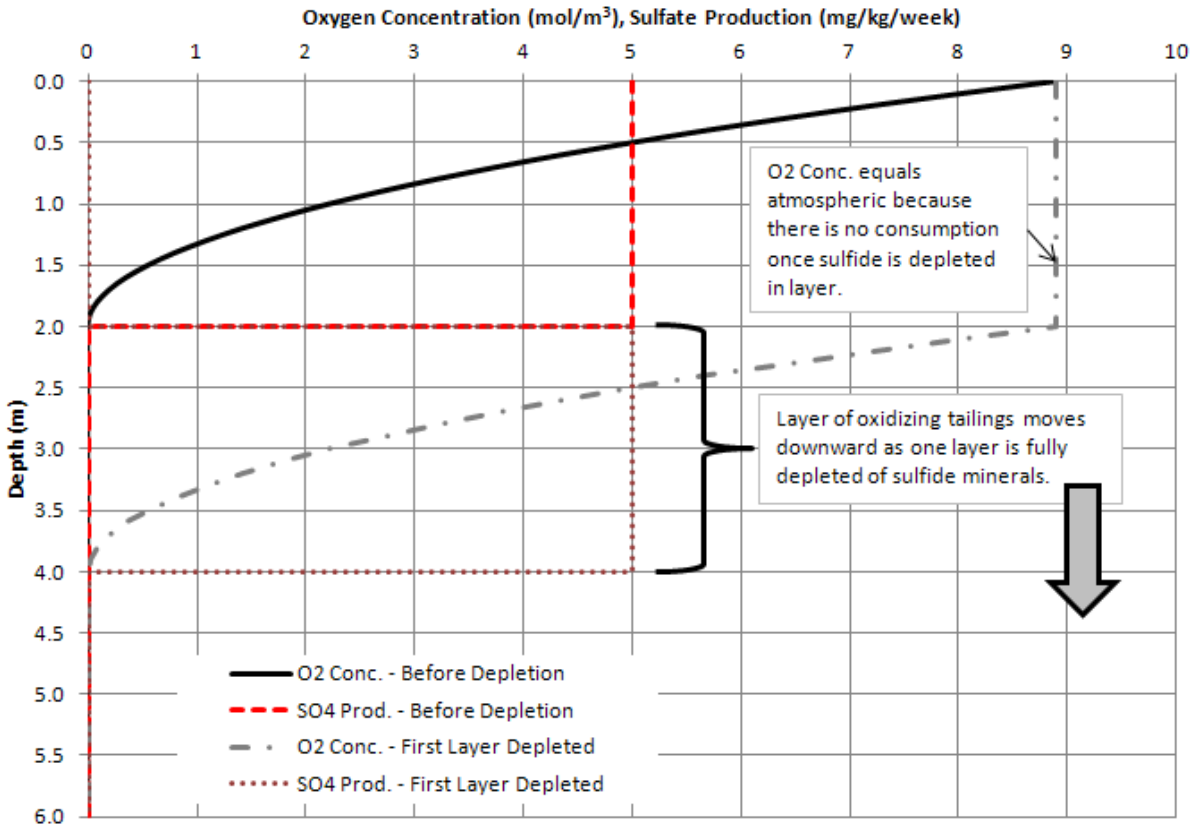

Figure 10-12 Example Oxygen Concentration and Sulfate Production Profile

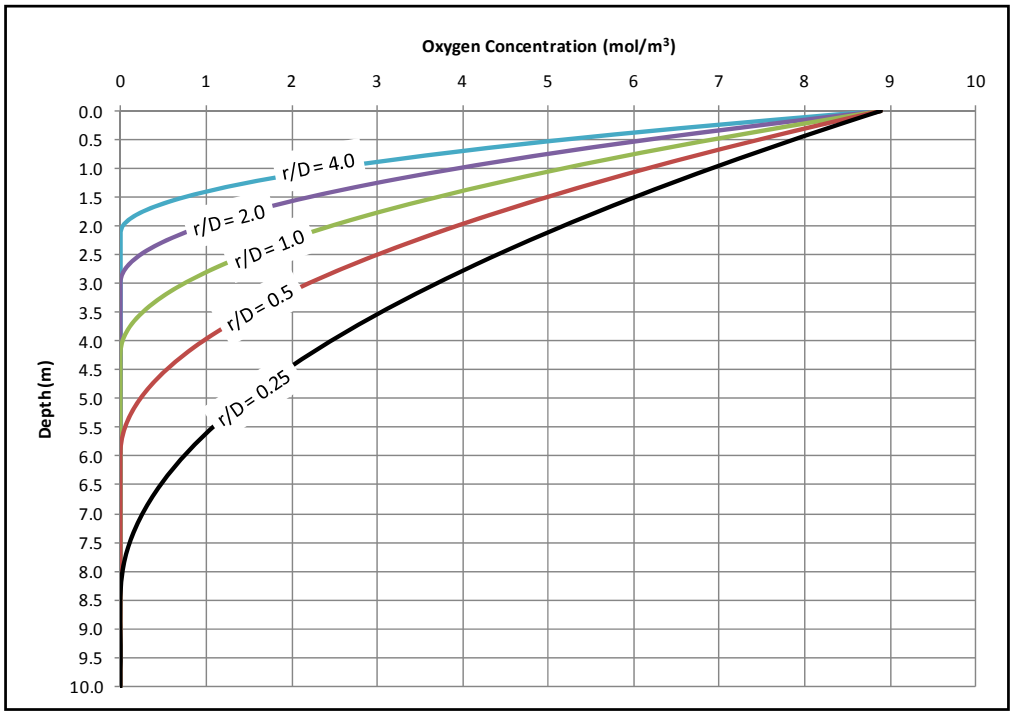

Figure 10-13 Oxygen Profiles at Different *r/D* Ratios

As shown in Figure 10-12, sulfate and other constituents where release is dependent on sulfate release, are only produced to a calculated depth to which oxygen can penetrate at any given time (~ 2 meters in the example of Figure 10-12, calculated by Equation 10-28). Within the oxygenated zone, sulfate (and other constituents) are produced at a constant rate (zero-order Fick's Law, Equation 10-10). Below the oxygenated zone, load of these constituents is not generated at any given time. However, as also shown in Figure 10-12, eventually Flotation Tailings near the surface of the Tailings Basin will be depleted of sulfides and will cease to consume oxygen. At this point, oxygen will be able to fully diffuse through those tailings to a depth where oxygen-consuming sulfides are present. The effect is an advancing front of layers of oxygen-consuming, sulfate-producing tailings. This will occur until the entire Tailings Basin is depleted of modeled constituent mass.

10.4 Concentration Caps

The concentration caps developed for the Category 1 waste rock (Sections 4.1.3.1 and 8.3 are applied to the metal concentrations determined from the above analysis. The use of Category 1 waste rock as an analog for the Flotation Tailings is appropriate because of the similar sulfur content and because the Flotation Tailings are also anticipated to remain nonacidic. As documented in previous waste characterization reports (Appendix D.1 of Reference (3) and Appendix B.1 of Reference (7)), the mineralogy of the Category 1 waste rock and the Flotation Tailings is broadly similar. Both the Category 1 waste rock and the Flotation Tailings are primarily composed of plagioclase (30-88% of WR, 50-80% of FT), with lower

levels of olivine (tr-55% of WR, 10-15% of FT) and clinopyroxene (tr-25% of WR, 4-5% of FT). Neither the waste rock nor the tailings contain carbonates. Sulfide levels in both sets of samples are low, with "trace" levels of sulfides in the waste rock and less than 0.5% in the tailings; sulfides are primarily pyrrhotite and chalcopyrite (in approximately equal proportions in WR, dominated by pyrrhotite in FT).

As discussed in Section 8.3 for waste rock stockpiles, it has been suggested that elevated partial pressures of carbon dioxide (pCO_2) could cause pH depression below what has been observed in the laboratory tests of Category 1 waste rock. The Flotation Tailings (like the Category 1 waste rock) are virtually lacking in carbonates to serve as a source of CO_2 (References (3) and (7)). However, elevated CO_2 has been inferred from MDNR geochemical modeling of the existing LTVSMC tailings (Reference (27)) and in the leach extraction tests on LTVSMC tailings (which contain carbonates). Observed pCO_2 in the LTVSMC tailings is approximately 10^{-2} atm. Because Flotation Tailings will be placed on LTVSMC tailings and LTVSMC tailings will be used for dam construction, this effect may cause changes in the pH of the tailings in certain portions of the basin

The pH observed in the laboratory testing of Category 1 waste rock to develop concentration caps (Attachment B) was approximately 8. If the pH of the Flotation Tailings is depressed below the laboratory-indicated levels, the solubility of several metals could be significantly different from what was observed in the laboratory tests. Based on the humidity cell testing, the relevant metals for the Flotation Tailings are cobalt, manganese, nickel, and zinc. As discussed in Sections 8.3.1.5 and 8.3.1.6, concentration cap relationships for these metals have been developed as a function of pH based on the Category 1 laboratory and AMAX field data. Given the sulfur content of the Flotation Tailings, which will likely be process controlled around 0.12 wt% S, and the elevated pCO_2 assumed (10^{-2} atm), the pH applied to the water in the pore spaces of the Flotation Tailings is a constant 7.1 (Figure 8-18).

The concentration caps discussed here apply to the Flotation Tailings only, concentration caps are not applied to the LTVSMC tailings beyond the defined release concentrations discussed in Section 10.1.2.2.

10.5 Flotation Tailings/LTVSMC Tailings Interaction

As discussed in Reference (7), column tests on the interaction between the seepage from Flotation Tailings and LTVSMC tailings indicate that for many constituents the chemical loading from the Flotation Tailings and LTVSMC tailings is additive. For one of the major constituents of concern, nickel, the tests indicate that nickel leaching from the Flotation Tailings will likely precipitate within the LTVSMC tailings and loading to the environment will be reduced. For all other constituents, the tests do not show attenuation of Flotation Tailings seepage loading within the LTVSMC tailings. There is no indication that seepage from the Flotation Tailings will cause significant dissolution of minerals precipitated in the LTVSMC tailings.

Although the column tests indicate that nickel will likely be removed from the NorthMet seepage water, the tests do not provide sufficient data for definitively quantifying changes in the water chemistry of seepage from Flotation Tailings as it passes through LTVSMC tailings. The probabilistic model, therefore, does not include any change in the seepage chemistry due to sorption or dissolution in the LTVSMC tailings underlying the FTB. This is a conservative assumption with respect to nickel and a proper assumption for all other constituents; nickel concentrations leaving the basin would be lower if the Flotation Tailings/LTVSMC tailings interaction was considered. Constituent loading is added to the seepage water from the unsaturated portions of LTVSMC tailings as described in Sections 10.1 through 10.4.

10.6 Additional Model Parameters

10.6.1 Oxidation of Saturated Tailings

As discussed in Section 10.2.1, oxygen availability is the primary driver for production of sulfate and other metals. In this section, oxygen availability in the saturated tailings immediately beneath the permanent pond under both operating and closure conditions is discussed.

The design for the FTB permanent pond, as a part of the wet cover system, is a wide and shallow pond. The area of the pond is approximately 400 acres during early years of operation and about 1,100 acres in later years of operation and upon closure. The maximum depth of the pond (as designed) is about 8 feet and the slope of the pond around the edge will be approximately 3%. It is conservative to ignore stratification by assuming the pond will be well mixed and that the dissolved oxygen (DO) levels will always be at saturation.

The solubility of oxygen in water is discussed in depth in Standard Methods for the Examination of Water and Wastewater (Reference (43)). This reference includes tables of the saturated DO concentration of water at atmospheric pressure (101.325 kPa) for different temperatures and chlorinity levels and the functions used to create the tables. It is proposed to use the empirical functions and apply appropriate corrections as necessary. Chlorinity only begins to have an important effect on DO at values greater than 1 g/kg or 1000 mg/L chloride. Previous analysis of the pond water quality estimated values near 6 mg/L. Therefore, chlorinity will not have a significant effect on DO and is ignored (i.e., chloride concentration of zero assumed). The altered function for oxygen solubility is then:

$$\ln C = -139.34411 + \left(\frac{1.575701 \times 10^5}{T}\right) - \left(\frac{6.642308 \times 10^7}{T^2}\right) + \left(\frac{1.2438 \times 10^{10}}{T^3}\right) - \left(\frac{8.621949 \times 10^{11}}{T^4}\right) \quad \textbf{10-29}$$

where C is the equilibrium DO concentration at 101.325 kPa [mg/L] and T is the water temperature [°K] (°C + 273.15). See Figure 10-14 for a graph of results from Equation 10-29.

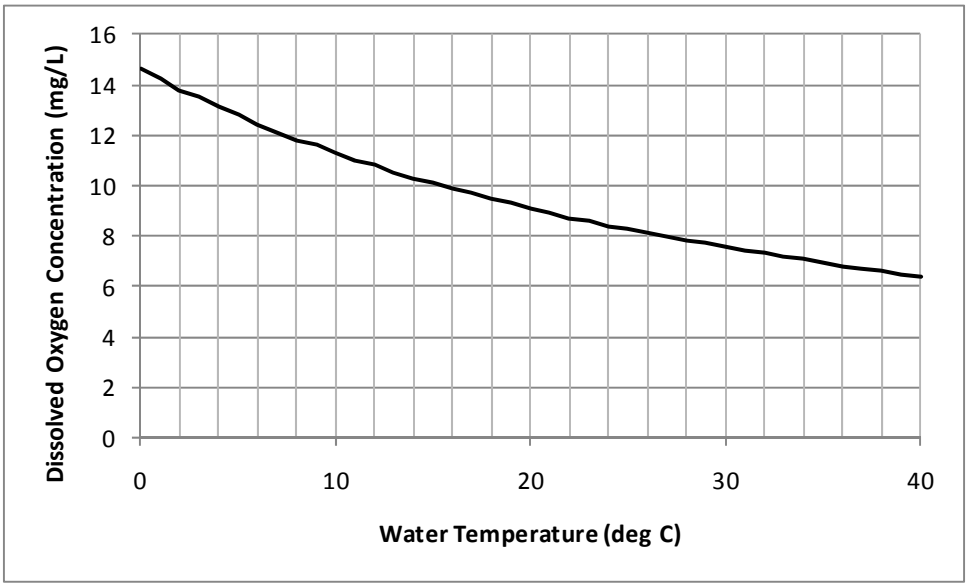

Figure 10-14 Dissolved Oxygen Concentration as a Function of Water Temperature

For the probabilistic water quality modeling the above equation and probabilistically generated average monthly pond water temperatures are used to calculate the dissolved oxygen concentration in the pond at each time step. Monthly pond water temperatures are estimated to range from 1 °C to about 22 °C.

Table 10-8 shows air temperature data from nearby stations over the period 1981 through 2010. Previous studies have shown that the pond temperature will be slightly warmer than the air temperature on site due to the elevated temperature of the tailings transport water from the Beneficiation Plant (Attachment A-5 of Reference (44)). Assuming air temperature is equivalent to pond water temperature is therefore conservative because DO concentration increases as temperature decreases. During months when the air temperature is below freezing, the water temperature is assumed to be 1 ºC (equivalent to a saturated DO of 14.2 mg/L). The proposed distribution for pond DO in any given month is a normal distribution with a mean at the saturated DO for a given mean air temperature and a standard deviation of 0.5 mg/L. This standard deviation was chosen because > 95% of values in a normal distribution will be between +/- two standard deviations around the mean. Therefore, > 95% of values will fall within a month's mean value +/- 1 mg/L. During months (November through March) when the water temperature is limited to 1 ºC because the air temperature is below freezing, the DO concentration in the pond is assumed to be 14.2 mg/L with no variability. Additionally, the distribution for each month is truncated at 14.2 mg/L so that unrealistic values greater than 14.2 mg/L do not occur. Using the mean, min and max temperature values in Table 10-8 and plotting them using Figure 10-14, the distribution parameters appropriately cover the expected range of dissolved oxygen concentrations in any month.

Table 10-8 Nearby Temperature Data (1981 – 2010) and Proposed Distribution Parameters

	Site Air Temperature (deg C)			Proposed Normal Distribution Parameters for Pond DO (mg/L)	
Month	**Mean**	**Min**	**Max**	**μ**	**σ**
January	-15.6	-24.2	-8.2	14.2	0
February	-12.7	-18.7	-4.2	14.2	0
March	-5.5	-11.6	0.3	14.2	0
April	2.8	-1.3	7.2	13.5	0.5
May	9.5	4.7	13.2	11.4	0.5
June	14.5	10.5	18.7	10.2	0.5
July	16.9	12.4	20.5	9.7	0.5
August	16.0	11.0	20.3	9.9	0.5
September	11.0	7.5	13.8	11.0	0.5
October	4.1	-0.9	7.7	13.1	0.5
November	-4.2	-9.7	1.3	14.2	0
December	-12.6	-20.4	-7.1	14.2	0

It is assumed that, as water seeps from the pond into the tailings, all available oxygen is consumed very quickly. Therefore, the production of sulfate and other metals is directly related to the concentration of oxygen in the pond and the rate at which the pond water is seeping into the saturated tailings (Equation 10-30).

$$C(I)\left(A_{pond}\right) = \dot{M} \qquad \textbf{10-30}$$

where C [M/L^3] is the concentration of oxygen in the pond water, I [L/T] is the rate at which water is infiltrating from the pond into the saturated tailings and A_{pond} [L^2] is the surface area of the pond at any given time. M-dot (Equation 10-30) is the mass flux rate of oxygen into the tailings from the pond. Because it is assumed that all of the oxygen is consumed quickly, M-dot is also the consumption rate of oxygen in the tailings. As described by Equation 10-9, the mass production rate of sulfate is calculated using known molecular weights of O_2 and SO_4 and the relationship that 4 moles of sulfate are produced for every 9 moles of oxygen consumed (results in 4 grams of sulfate produced for every 3 grams of oxygen consumed).

For example purposes, if the DO concentration in the pond is 10 mg/L, the infiltration rate is 30 in/yr and the area of the pond is 1,000 acres, then the sulfate production rate is about 9.3 tons/yr or roughly 18.6 pounds/acre/year.

10.6.2 Tailings Weathering

Water falling on exposed tailing beaches can be expected to dissolve soluble products of weathering and oxidation. The contribution of tailings weathering is a function of the exposed area of tailings. Laboratory-measured humidity cell rates were expressed as mg/m^2/month, where m^2 reflects the aerial exposure of tailings. Rates have been generated for both coarse and fine fractions, based on the initial period of testing, corresponding to the length of time any portion of the tailings surface is expected to remain exposed before being covered by subsequent deposition (Table 1-17 of Attachment B to Reference (40)). Section 5.1.3.1 describes the variability in the fractions of coarse and fine grained tailings in the beaches (a range of about 30% to 40% fines in the beaches).

During each month in the model, loads are generated and added to the FTB Pond representing load that has been produced on the beaches and washed off due to rainfall. This load is calculated using the per unit area weathering rates, weighted by the fraction of mass represented by each respective grain size. For example, if at one time, the average % fines in a 100 acre tailings beach is 35% and the weathering rates for coarse and fine grain tailings are 1000 and 2000 mg/m^2/month respectively, then the net load added to the pond (due to precipitation "washing" the beach) would be about 0.6 tons/month.

10.6.3 Process Water Loading to Pond

There are two additional loads that are worth consideration for the water quality model: load leaching from weathered ore and load added by reagents.

10.6.3.1 Ore Leaching Load

The load added to the process water that goes to the FTB is due to processing weathered ore that has not been fully "rinsed" by rain in the field. Section 8.2 describes the load leaching from ore in the field at the Mine Site. This is only a portion of what is generated in the field due to the contact factor (Equation 8-9). Therefore, any load that was generated in the field, but was not contacted by water, is added to the process water load during processing. Equation 10-31 is similar to Equation 8-9 and represents the generated load that clung to the ore in the field. Therefore, all of the same generation rates and correction factors are used to calculate this ore leaching load. The differences are using $(1 - k_C)$ instead of k_C for the contact factor and using the ore processing rate for mass rather than mass of ore present in a stockpile.

$$L = \dot{M} * t_w * k_T * k_s * (1 - k_c) \qquad \textbf{10-31}$$

where L [M] is the total available load still with the ore from the Mine Site, M-dot [M/T] is the mass production rate of ore, t_w [T] is the approximate time that the ore was left in the environment to weather, k_T [--] is the temperature factor, k_S [--] is the size factor and k_C [--] is the water contact factor. M-dot is known from the Mine Plan (30,860 tons per day, Reference (9)) and k_T, k_S and k_C are all described in Section 8.2. The time that ore is allowed

to weather in the environment (t_w) is expected to be less than 6 months according to PolyMet but realistically should not be 0 months. Therefore, due to the modeled monthly time step, a uniform distribution is proposed that ranges from 1 month to 6 months for the variable t_w.

10.6.3.2 Reagent Load

The only reagent that is expected to contribute to metal and sulfate concentrations is copper sulfate. The process will involve addition of approximately 55 g of copper sulfate pentahydrate per ton of ore (applied to the average ore processing rate of 30,860 tons per day, Reference (9)). Copper sulfate is added to the process to activate pyrrhotite and improve flotation by reaction of copper with the pyrrhotite surfaces and bind the copper with the product; therefore only the sulfate portion of the chemical load is added to the process water that goes to the FTB Pond. Pilot plant testing has confirmed that this is an appropriate assumption (Reference (7)).

10.6.4 Effects of Re-suspension by Wind

Concerns were raised in the NorthMet Impact Assessment Planning process regarding the potential for water quality impacts due to the re-suspension of Flotation Tailings within the FTB Pond. Research at facilities with subaqueous sulfide-bearing tailings in Canada (Reference (45)) has shown that tailings basins with shallow ponds can exhibit significant re-suspension of tailings due to wind effects. The re-suspended tailings are exposed to oxygenated water and can oxidize and release sulfate and metals to the pond water. Such release would represent an additional constituent loading to the pond water, beyond the oxidation described in Section 10.6.1.

The degree to which suspended tailings represent a significant source of sulfate to pond water quality is heavily dependent on the sulfate release rate of the tailings themselves. The sulfate oxidation rate for suspended tailings can be estimated from:

$$R_{sus} = C_{sus} * R_{tailings} \qquad \textbf{10-32}$$

where C_{sus} [M tailings/L^3] is the suspended tailings concentration, $R_{tailings}$ [M SO_4/M tailings/T] is the sulfate release rate for tailings from humidity cell testing, and R_{sus} [M SO_4/L^3/T] is the resulting sulfate loading rate per unit volume to the tailings pond.

For the fine Flotation Tailings that will be under the FTB Pond, $R_{tailings}$ ranges from 3.6 to 19.6 mg SO_4/kg tailings/week, with a mean value of 6.0 mg SO_4/kg tailings/week (Large Table 16). Suspended tailings concentrations reported for the basins in Reference (45) are on the order of 10-20 mg/L. Using the high-end values for both of these parameters and assuming that the resulting sulfate load is retained in the FTB Pond for 1 year, the calculation in Equation 10-32 yields an increase in sulfate concentration of 0.02 mg/L. If the average suspended tailings concentration is 100 mg/L, the increase in pond sulfate concentration is at most 0.1 mg/L.

The additional load to the FTB Pond from tailings re-suspension is negligible compared to the other sources of sulfate load. The reagent loading described in Section 10.6.3 will add load equivalent to approximately 23 mg/L sulfate to the process water going to the FTB Pond. The treated water from the Mine Site will have a sulfate concentration of up to 250 mg/L. Because all other loads from the tailings are calculated as a ratio to sulfate load, metal loads to the FTB Pond from re-suspension are also negligible. The effect of re-suspension due to wind is therefore not included in the probabilistic model.

10.6.5 Buttress Material

According to Reference (9), a rock buttress will be placed at the north toe of the existing LTVSMC Cell 2E to ensure sufficient geotechnical stability. The buttress will be constructed of waste rock sourced from the LTVSMC waste rock stockpiles near the Plant Site, with a specific material source location to be determined in consultation with MDNR. The buttress material will be hauled directly from the selected stockpile(s) and will not be crushed or screened.

Because no detailed geochemical information is available for the potential sources of buttress material, the water quality modeling assumes that the buttress rock is equivalent to Category 1 waste rock. Chemical loading from this material is modeled according to the methods described in Section 8.0 for Category 1 waste rock, including scaling factors and concentration caps. This assumption may affect the identification of acceptable buttress waste rock source areas in permitting.

10.6.6 Depletion

The water quality model includes a check for depletion of each modeled constituent from each tailings fraction. This ensures that any simulated very high metal leaching rates do not continue, but consume the available metals within an appropriate time frame and cease.

The average content of each constituent in each tailings fraction type was determined from the Flotation Tailings and LTVSMC tailings data sets. These average values are considered to be deterministic in the water quality modeling and are shown in Table 10-9.

Table 10-9 Average Metal Content from Aqua Regia Data (ppm)

Constituent	Flotation Tailings (fine)	Flotation Tailings (coarse)	LTVSMC Tailings (all)
Silver (Ag)	2.13E-01	1.86E-01	7.33E-02
Aluminum (Al)	3.60E+04	3.56E+04	1.92E+02
Alkalinity[1]	--	--	--
Arsenic (As)	2.19E+00	2.43E+00	2.82E+01
Boron (B)	5.00E+00	5.00E+00	5.15E+00
Barium (Ba)	5.36E+01	4.86E+01	1.03E+01
Beryllium (Be)	1.84E-01	1.87E-01	6.92E-01
Calcium (Ca)	1.98E+04	2.04E+04	1.45E+03
Cadmium (Cd)	6.50E-02	6.29E-02	5.74E-02
Chlorine (Cl)[1]	--	--	--
Cobalt (Co)	4.56E+01	5.51E+01	8.22E+00
Chromium (Cr)	9.89E+01	1.08E+02	8.50E+01
Copper (Cu)	2.22E+02	1.10E+02	9.72E+00
Fluoride (F)[1]	--	--	--
Iron (Fe)	5.39E+04	6.78E+04	9.88E+03
Potassium (K)	1.94E+03	1.83E+03	6.24E+01
Magnesium (Mg)	3.30E+04	4.08E+04	8.09E+02
Manganese (Mn)	6.02E+02	7.52E+02	4.61E+03
Sodium (Na)	4.69E+03	4.53E+03	1.11E+01
Nickel (Ni)	2.46E+02	2.89E+02	4.23E+00
Lead (Pb)	3.21E+00	3.39E+00	1.54E+00
Antimony (Sb)	1.21E-01	1.29E-01	8.08E-02
Selenium (Se)	4.30E-01	5.20E-01	4.94E-01
Sulfur (S)	1.05E+03	1.21E+03	4.64E+01
Thallium (Tl)	1.00E-01	8.86E-02	2.00E-02
Vanadium (V)	3.47E+01	4.54E+01	1.00E+01
Zinc (Zn)	5.79E+01	7.04E+01	9.67E+00

(1) Aqua regia data not available for alkalinity, chlorine, and fluoride. No depletion is modeled.

11.0 Geochemical Parameters – Hydrometallurgical Residue

As discussed in Section 6.0, the hydrometallurgical residue has been well-characterized based on the pilot-testing results. In addition, the HRF is designed as a double-lined storage area with a permanent geomembrane cover in closure (Section 6.2). The leakage of contaminated water from this facility to the surrounding environment will be negligible and the HRF has not been included as a source in the probabilistic water quality modeling (Reference (13)).

Revision History

Date	Version	Description
2/14/2011	1	Initial release
3/21/2011	2	Added Sections for Waste Rock and Pit Lake: Category 1 Concentration caps Study; 2011 Geochemical Update; Geochemical Parameters – Waste Rock; Geochemical Parameters – Pit Lake; Large Table 3 and Large Table 4
5/16/2011	3	Added Sections for Overburden, Waste Rock, Pit Lake and Tailings: Overburden 2011 Update; Sulfur Content of Stockpiles and Pit Walls; Flotation Tailings 2011 Update; Geochemical Parameters – Overburden; Sulfate Release Rates for Category 1, 2/3 and Ore; Depletion; Concentration Caps (Pit Lake); Age of Pit Walls and Backfill Modeling Parameters; Lab to Field Scale Up (Tailings); Saturation and Oxygen Diffusion; Concentration caps (Tailings); Additional Model Parameters (Tailings); Geochemical Parameters – Hydrometallurgical Residue Edited Sections for Waste Rock and Pit Lake: 2009 and 2011 Updates; All Other Release Rates; Effects of Acidification; Lab to Field Scale Up; Concentration Caps; Long-Term Decay in Constituent Release; Large Tables and Figures
7/22/11	4	**Version 4 to accompany submittal of the NorthMet Mine Site Water Modeling Work Plan** Updated formatting and performed minor text edits throughout document Added Sections for Waste Rock, Pit Lake and Tailings: Tailings 2011 Geochemical Update; Nitrogen in Waste Rock; Ore Spillage from Rail Cars; Nitrogen in Mine Pits; Subaqueous Oxidation; Dewatered Material Oxidation; Tailings Laboratory Release Rates; NorthMet/LTVSMC Tailings Interaction Edited Sections for Waste Rock, Pit Lake and Tailings: All Other Release Rates (Waste Rock); Effects of Acidification (Waste Rock); Concentration Caps (Waste Rock); Long-term Decay in Constituent Release; Age of Pit Walls; Saturation and Oxygen Diffusion; Geochemical Parameters – Hydrometallurgical Residue; Large Tables and Figures
8/11/11	5	**Version 5 to accompany submittal of the NorthMet Plant Site Water Modeling Work Plan** Performed minor text edits throughout document Added Sections for Tailings: Scaling/Calibration of LTVSMC Lab Data to Field Data; Effects of Wind Re-Suspension; Buttress Material; Depletion (Tailings) Edited Sections for Tailings: Laboratory Release Rates for Flotation and LTVSMC Tailings; Concentration Caps (Tailings); Process Water Loading to Pond; Large Tables and Figures

Date	Version	Description
11/11/11	6	**Version 6 to accompany submittal of the NorthMet Mine Site Water Modeling Work Plan, Version 4, and the NorthMet Plant Site Water Modeling Work Plan, Version 3** Performed minor text edits throughout document Added Sections for Waste Rock, Pit Lake and Tailings: Humidity Cell Trends Update (Tailings); Nickel Release (Waste Rock); Chloride Release from Waste Rock; Expected Saturation Levels of the FTB Edited Sections for Overburden, Waste Rock, Pit Lake and Tailings: Geochemical Parameters – Overburden; Category 1 Waste Rock Concentration Caps; Nitrogen in Waste Rock; Ore Spillage from Rail Cars; Virginia Formation highwall modeling; Long-term Decay in Constituent Release; Release from Concentration Caps (Tailings); Lab to Field Scale Up (Tailings); Scaling/Calibration to LTVSMC Field Data; Concentration Caps (Tailings)
1/10/12	7	**Version 7 to accompany submittal of the NorthMet Mine Site Water Modeling Work Plan, Version 5, and the NorthMet Plant Site Water Modeling Work Plan, Version 3** Edited Sections for Waste Rock, Pit Lake and Tailings: Category 1 Concentration Caps Study; Flotation Tailings Depositional Study; Tailings Humidity Cell Trends Update; Waste Rock Lab to Field Scale-Up; Category 1 Waste Rock Concentration Caps; Wall Rock Lab to Field Scale-Up; Wall Rock Acidification and Long-Term Decay; Pit Walls and Pit Lake Concentration Caps; Calibration to LTVSMC Field Data; Expected Saturation Levels in the FTB; Tailings Concentration Caps Revised Attachment E, Update on Tailings Humidity Cell Test Data
2/9/12	8	**Version 8 to accompany submittal of the NorthMet Mine Site Water Modeling Work Plan, Version 6, and the NorthMet Plant Site Water Modeling Work Plan, Version 4** Revised Section 8.2.3, Particle Size Factor Edited Section 8.2.8, MDNR Category 1 Scale-Up Edited Section 8.3, Concentration Caps (Waste Rock)
7/2/12	9	**Version 9 to accompany submittal of the NorthMet Mine Site Water Modeling Work Plan, Version 7, and the NorthMet Plant Site Water Modeling Work Plan, Version 6** Updated based on approved Change Definition forms. Revised Sections 4.3.2, 8.1.2.1, 8.2.8, 8.3.1.4, 9.2, 10.2.1, 10.3.2.1, 10.3.2.2, 0, 10.4. Revised Large Tables 3, 4, 7-9, 12, 13 Revised Large Figures 1-2, 5-6, 9-11, 45, 49

Date	Version	Description
3/8/13	10	**Version 10 to accompany submittal of the NorthMet Mine Site Water Modeling Data Package Volume 1 – Mine Site, Version 12, and the NorthMet Plant Site Water Modeling Data Package Volume 2 – Plant Site, Version 9** Updated based on approved Change Definition forms. Revised Sections 10.1.1.1 and 10.3.2.2 Revised Large Tables 16 and 17 Revised Large Figures 45 and 49
12/1/14	11	**Version 11 to accompany submittal of the NorthMet Mine Site Water Modeling Data Package Volume 1 – Mine Site, Version 13, and the NorthMet Plant Site Water Modeling Data Package Volume 2 – Plant Site, Version 10** Revised Sections 4.1.3, 8.1.1, 8.1.2, 9.5.2, 10.1.1.1, 10.2.1, 10.3.2.2, 10.3.3, minor text revisions throughout. Added Sections 4.1.4, 5.1.4, 6.1.4. Added Attachments C and D. Updated Large Tables 1, 5, 7-9. Updated Large Figures 1-2, 5-6, 9-10.
2/13/15	12	**Version 12 to address comments received from the Co-Lead Agencies on Version 11.** Minor text revisions in Sections 4.1.3, 4.1.4, 5.1.4, 8.1.2.1, 9.5.2, 10.3.2, and 10.3.3. Minor edits to Attachment C and Large Table 3.

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List of Attachments

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Attachment B Results of an Experiment to Evaluate Solubility Limits for Category 1 Waste Rock
Attachment C 2015 Update on Kinetic Test Data
Attachment D 2014 Update on Kinetic Test Data
Attachment E Physical Model Study of Fine-Grained Tailings Delta Formation under Various Flow Conditions: Project Report
Attachment F Update on Tailings Humidity Cell Test Data
Attachment G Laboratory Report for Barr Engineering Company (Tailings Samples)

Large Tables

Large Table 1 Summary of NorthMet Project Humidity Cells (Waste Rock)

Rock Type	*Waste Category*	*Sample ID*	*Sulfur (%)*	*Condition 1 Start (weeks)*	*Condition 2 Start (weeks)*	*Condition 3 Start (weeks)*	*Condition 4 Start (weeks)*	*Total Duration (weeks)*
Anorthositic	1	99-320C(830-850)	0.09	4	179	-	-	436
Anorthositic	1	00-361C(345-350)	0.05	6	184	-	-	436
Anorthositic	1	00-366C(185-205)	0.02	0	-	-	-	198
Anorthositic	1	00-366C(230-240)	0.02	4	60	-	-	198
Anorthositic	1	99-320C(165-175)	0.03	0	72	-	-	198
Anorthositic	1	00-334C(30-50)	0.02	4	-	-	-	436
Anorthositic	1	00-368C(125-145)	0.04	0	80	-	-	436
Anorthositic	1	00-368C(20-40)	0.04	0	80	-	-	198
Troctolitic	1	00-340C(595-615)	0.04	0	-	-	-	198
Troctolitic	1	00-334C(580-600)	0.06	1	179	-	-	436
Troctolitic	1	00-334C(640-660)	0.07	12	224	-	-	436
Troctolitic	1	00-347C(795-815)	0.07	0	103	-	-	198
Troctolitic	1	99-318C(250-270)	0.04	0	72	-	-	198
Troctolitic	1	00-373C(95-115)	0.04	0	-	-	-	198
Troctolitic	1	00-373C(75-95)	0.06	0	-	-	-	198
Troctolitic	1	00-357C(110-130)	0.08	10	-	-	-	198
Troctolitic	1	99-320C(315-330)	0.07	4	72	-	-	436
Troctolitic	1	00-366C(35-55)	0.02	0	-	-	-	198
Troctolitic	1	00-334C(110-130)	0.04	0	-	-	-	198
Troctolitic	1	00-347C(155-175)	0.06	0	72	-	-	198
Troctolitic	1	00-347C(280-300)	0.06	16	65	-	-	198
Troctolitic	1	00-367C(50-65)	0.03	0	-	-	-	198
Troctolitic	1	00-367C(260-280)	0.04	0	-	-	-	198
Troctolitic	1	00-367C(290-310)	0.04	0	-	-	-	436
Troctolitic	1	00-370C(20-30)	0.08	10	-	-	-	198
Troctolitic	1	26064(44-54)	0.02	0	-	-	-	436
Troctolitic	1	26064(264+146269+156)	0.06	4	-	-	-	436
Troctolitic	1	26056(110-125)	0.04	0	-	-	-	198
Troctolitic	1	26029(815-825)	0.02	0	-	-	-	194
Troctolitic	1	26056(135-153)	0.05	0	-	-	-	430
Troctolitic	1	00-326C(250-265)	0.08	4	-	-	-	186
Ultramafic	1	00-357C(335-340)	0.08	12	187	-	-	198
Ultramafic	1	00-368C(460-465)	0.06	0	-	-	-	198
Ultramafic	1	26055(940-945)	0.06	16	-	-	-	198
Ultramafic	1	26098+00-337C	0.1	0	-	-	-	198
Ultramafic	1	00-361C(240-245)	0.06	14	184	-	-	436
Ultramafic	1	26039(310-315)	0.06	8	-	-	-	186
Ultramafic	1	00-326C(225-235)	0.12	8	-	-	-	425
Anorthositic	2/3	00-361C(310-320)	0.18	0	111	-	-	436
Anorthositic	2/3	99-320C(400-405)	0.18	14	-	-	-	425
Sedimentary Hornfels	2/3	26030(1047-1052)	0.24	53	-	-	-	436
Sedimentary Hornfels	2/3	26061(1218-1233)	0.44	4	-	-	-	436
Sedimentary Hornfels	2/3	00-340C(990-995)	0.55	0	189	-	-	436
Troctolitic	2/3	00-350C(580-600)	0.19	0	196	-	-	436
Troctolitic	2/3	00-327C(225-245)	0.44	0	182	-	-	198
Troctolitic	2/3	00-369C(335-345)	0.18	4	181	-	-	436
Troctolitic	2/3	00-326C(60-70)	0.14	0	75	164	-	436
Troctolitic	2/3	00-369C(305-325)	0.25	4	187	-	-	436
Troctolitic	2/3	00-369C(20-30)	0.21	0	187	-	-	436
Troctolitic	2/3	00-367C(170-175)	0.51	0	172	-	-	436
Troctolitic	2/3	00-340C(380-390)	0.15	4	-	-	-	198
Troctolitic	2/3	26049+26030	0.59	4	-	-	-	198
Troctolitic	2/3	26056(302-312)	0.23	12	212	-	-	436
Troctolitic	2/3	26142(360+345-365+350)	0.18	0	168	-	-	436
Troctolitic	2/3	99-318C(325-330)	0.17	0	180	-	-	425
Troctolitic	2/3	26056(282-292)	0.32	2	178	-	-	186
Troctolitic	**2/3**	**00-340C(910-925)**	**0.36**	**0**	**72**	**110**	**180**	**425**
Troctolitic	**2/3**	**00-331C(190-210)**	**0.42**	**0**	**48**	**201**	**229**	**425**
Troctolitic	2/3	00-367C(495-500)	0.28	8	114	-	-	425
Ultramafic	2/3	00-326C(680-685)	0.3	0	69	-	-	198
Ultramafic	2/3	00-357C(535-540)	0.2	0	78	194	194	436
Ultramafic	**2/3**	**00-344C(630-635)**	**0.34**	**0**	**51**	**160**	**-**	**186**
Ultramafic	2/3	00-326C(495-505)	0.16	0	-	-	-	186
Anorthositic	4	00-343C(240-250)	0.68	0	161	-	-	198
Anorthositic* **	**4**	**26027(616-626)*** **	**1.83**	**4**	**18**	**24**	**-**	**436**
Anorthositic* **	**4**	**00-331C(255-260)*** **	**0.86**	**0**	**19**	**162**	**184**	**425**
Sedimentary Hornfels	**4**	**00-340C(965-974.5)**	**1.74**	**0**	**25**	**34**	**80**	**198**
Sedimentary Hornfels	**4**	**26043+26027**	**2.47**	**0**	**9**	**26**	**48**	**436**
Sedimentary Hornfels	4	26062+26026	4.46	0	3	3	-	438
Sedimentary Hornfels	4	26058(704-715)	1.46	8	41	-	-	427
Troctolitic* **	**4**	**00-371C(435-440)*** **	**0.88**	**0**	**51**	**90**	**196**	**436**
Troctolitic* **	**4**	**00-340C(765-780)*** **	**1.68**	**0**	**61**	**82**	**200**	**436**
Troctolitic	4	00-367C(395-400)	0.77	0	82	-	-	198
Troctolitic	4	00-340C(725-745)	0.91	6	118	-	-	198
Troctolitic	**4**	**00-367C(400-405)**	**1.37**	**4**	**39**	**78**	**-**	**198**
Ultramafic	4	99-318C(725-735)	0.72	0	96	-	-	198
Ultramafic	4	99-317C(460-470)	1.24	0	39	-	-	198
Ultramafic	**4**	**00-344C(515-520)**	**1.2**	**4**	**47**	**152**	**-**	**198**
Ultramafic	4	00-330C(275-280)	0.75	0	164	-	-	186
Virginia	4	00-361C(737-749)	2	0	39	164	194	436
Virginia	4	00-364C(210-229)	3.79	0	0	5	-	436
Virginia	4	00-337C(510-520)	5.68	0	0	5	-	198
Ore Composite	4	P10	0.86	4	88	-	-	432
Ore Composite	4	P20	0.9	6	88	-	-	434
Ore Composite	4	P30	0.86	6	88	-	-	432

Notes

- Time periods as of December 20, 2013. "--" indicates condition not observed.
- Condition 1: relatively stable sulfate release, leachate pH above about 7
- Condition 2: relatively stable sulfate release, leachate pH below about 7 and nickel release unstable and typically increasing (i.e., neutral conditions)
- Condition 3: sulfate release increasing and variable, pH decreasing
- Condition 4: sulfate release decreasing following peak, acidic pH
- Humidity cells in **bold** are used to develop the acidity factor in Section 8.2.5
- Humidity cells marked with (***) are used to develop decay relationships in Section 9.4. Although several cells had not reached Condition 4 as of February 24, 2011, decay relationships were evident in further analysis as of July 2011.

Large Table 2 Proposed Approach for Deriving Non-Acidic Release Rates (Waste Rock)

		Duluth Complex Category 1				*Duluth Complex Category 2/3, 4 and Ore*				*Virginia Formation Category 4*			
Constituent	***Goldschmidt Classification***	***Mineral Sources***	***Ion***	***Method***	***Source***	***Mineral Sources***	***Ion***	***Method***	***Source***	***Mineral Sources***	***Ion***	***Method***	***Source***
Primary Ions													
SO_4	Chalcophile	Chalcopyrite > pyrrhotite	SO_4^{2-}	SO_4 Rate	HCT	Pyrrhotite > chalcopyrite > pentlandite	SO_4^{2-}	SO_4 Rate	HCT	Pyrrhotite	SO_4^{2-}	SO_4 Rate	HCT
Alkalinity	Atmophile	Calcite, dolomite	CO_3^{2-}	Alkalinity Rate	HCT	Calcite, dolomite	CO_3^{2-}	Alkalinity Rate	HCT	None	CO_3^{2-}	None	--
Ca	Lithophile	Anorthite, calcite	Ca^{2+}	Ca Rate	HCT	Anorthite, calcite	Ca^{2+}	Ca/SO_4 Rate, Condition 2	HCT	Anorthite	Ca^{2+}	Ca/SO_4 Rate, Condition 3	HCT
Cl	Nonmetallic	Unknown	Cl^-	First flush release	HCT	Unknown	Cl^-	First flush release	HCT	Unknown	Cl^-	First flush release	HCT
Mg	Lithophile	Olivine	Mg^{2+}	Mg Rate	HCT	Olivine	Mg^{2+}	Mg/SO_4 Rate, Condition 2	HCT	Mg silicates	Mg^{2+}	Mg/SO_4 Rate, Condition 3	HCT
Na	Lithophile	Albite	Na^+	Na Rate	HCT	Albite	Na^+	Na/SO_4 Rate, Condition 2	HCT	Albite	Na^+	Na/SO_4 Rate, Condition 3	HCT
K	Lithophile	Biotite	K^+	K Rate	HCT	Biotite	K^+	K/SO_4 Rate, Condition 2	HCT	Biotite	K^+	K/SO_4 Rate, Condition 3	HCT
K	Lithophile	Biotite	K^+	K Rate	HCT	Biotite	K^+	K/SO_4 Rate, Condition 3	HCT	Biotite	K^+	K/SO_4 Rate, Condition 4	HCT
Secondary Ions													
Ag	Chalcophile	Sulfide	Ag^+	Ag rate	HCT	Sulfide	Ag^+	Ag/S	Aqua Regia	Sulfide	Ag^+	Ag/S	Aqua Regia
Al	Lithophile	Anorthite, albite	Al^{3+}	Al/Ca (anorthite) + Al/Na (albite)	Mineral forumulae	Anorthite, albite	Al^{3+}	Al/Ca (anorthite) + Al/Na (albite)	Mineral forumulae	Anorthite, albite	Al^{3+}	Al/Ca (anorthite) + Al/Na (albite)	Mineral forumulae
As	Chalcophile	Sulfide	$HAsO_4^{2-}$	As rate	HCT	Sulfide	$HAsO_4^{2-}$ /$H_2AsO_4^-$	As/S	Aqua Regia	Sulfide	$HAsO_4^{2-}$ /$H_2AsO_4^-$	As/S	Aqua Regia
B	Lithophile	Unknown	H_3BO_3	B Rate	HCT	Unknown	H_3BO_3	B Rate	HCT	Unknown	H_3BO_3	B Rate	HCT
Ba	Lithophile	Biotite	Ba^{2+}	Ba/K Ratio	Aqua Regia	Biotite	Ba^{2+}	Ba/K Ratio	Aqua Regia	Biotite	Ba^{2+}	Ba/K Ratio	Aqua Regia
Be	Lithophile	Biotite	Be^{2+}	Be rate	HCT	Biotite	Be^{2+}	Be/K Ratio	Aqua Regia	Unknown	Be^{2+}	Be/K Ratio	Aqua Regia
Cd	Chalcophile	Sulfide	Cd^{2+}	Cd/Zn Rate, Condition 2	HCT	Sulfide	Cd^{2+}	Cd/Zn Rate, Condition 2	HCT	Pyrrhotite	Cd^{2+}	Cd/S	Aqua Regia
Co	Siderophile	Olivine	Co^{2+}	Co/Ni Rate, Condition 2	HCT	Olivine	Co^{2+}	Co/Ni Rate, Condition 2	HCT	Pyrrhotite	Co^{2+}	Co/S	Aqua Regia
Cr	Lithophile	Unknown	$CrOH^{2+}$	Cr Rate	HCT	Unknown	$CrOH^{2+}$	Cr Rate	HCT	Unknown	$CrOH^{2+}$	Cr Rate	HCT
Cu	Chalcophile	Chalcopyrite	Cu^{2+}	Cu/S	Aqua Regia	Chalcopyrite	Cu^{2+}	Cu/S	Aqua Regia	Pyrrhotite	Cu^{2+}	Cu/S	Aqua Regia
F	Lithophile	Apatite	F^-	F rate	HCT	Apatite	F^-	F rate	HCT	Apatite	F^-	F rate	HCT
Fe	Siderophile	Sulfides, silicates	Fe^{3+}	Fe/S (pyrrhotite) + Fe/Mg (olivine)	Microprobe	Sulfides, silicates	Fe^{3+}	Fe/S (pyrrhotite) + Fe/Mg (olivine)	Microprobe	Sulfides	Fe^{3+}	Fe/SO_4 Rate	HCT
Mn	Siderophile	Sulfide	Mn^{2+}	Mn/SO_4 Rate, Condition 2	HCT	Sulfide	Mn^{2+}	Mn/SO_4 Rate, Condition 2	HCT	Pyrrhotite	Mn^{2+}	Mn Rate	HCT
Ni	Siderophile	Sulfides, olivine	Ni^{2+}	Ni/S (Cat 4) + Ni/Mg (olivine)	Aqua Regia Microprobe	Sulfides, olivine	Ni^{2+}	Ni/S (Cat 4 or ore) + Ni/Mg (olivine)	Aqua Regia Microprobe	Pyrrhotite	Ni^{2+}	Ni/S	Aqua Regia
Pb	Chalcophile	Sulfide	Pb^{2+}	Pb rate	HCT	Sulfide	Pb^{2+}	Pb/S	Aqua Regia	Pyrrhotite	Pb^{2+}	Pb/S	Aqua Regia
Sb	Chalcophile	Sulfide	Oxyanion	Sb rate	Reactor	Sulfide	Oxyanion	Sb/S	Aqua Regia	Pyrrhotite	Oxyanion	Sb/S	Aqua Regia
Se	Chalcophile	Chalcopyrite	SeO_4^{2-}	Se/SO_4 Rate, Condition 2	HCT	Chalcopyrite	SeO_4^{2-}	Se/SO_4 Rate, Condition 2	HCT	Pyrrhotite	SeO_4^{2-}	Se/SO_4 Rate, Condition 2	HCT
Tl	Chalcophile	Unknown	Tl^+	Tl Rate	HCT	Unknown	Tl^+	Tl Rate	HCT	Pyrrhotite	Tl^+	Tl Rate	HCT
V	Lithophile	Biotite	$VO_2(OH)^{2-}$	V rate	HCT	Biotite	$VO_2(OH)^{2-}$	V/K	Aqua Regia	Biotite	VO_2^+	V/K	Aqua Regia
Zn	Chalcophile	Olivine	Zn^{2+}	Zn/Mg	Aqua Regia	Pyrrhotite, olivine	Zn^{2+}	Zn/Ni Rate, Condition 2	Aqua Regia	Pyrrhotite	Zn^{2+}	Zn/S	Aqua Regia

Notes

- "Rate" indicates a release rate developed from humidity cells for primary ions or non-correlated metals, or a ratio of release rates used to estimate metal release from other release rates (ex. Cd/Zn Rate for Category 1). For Sb only the smaller MDNR-style reactors were used to estimate a release rate for Category 1 waste rock.
- Metal ratios with a data source of "Aqua Regia" or "Microprobe" indicate a solids ratio that is used to estimate metal release from the primary ion release rates)
- "Condition 2" refers to a time period in the humidity cells with relatively stable sulfate release, leachate pH below about 7 and nickel release unstable and typically increasing (i.e., neutral conditions, see Large Table 1)
- Acidic release rates will be determined as described in Section 8.1. For humidity cell release rates or release ratios not linked to SO_4, acidic release rates will be determined based on Condition 3 for the Duluth Complex Category 2/3/4 and Ore.
- For nickel, S ratio from Duluth Complex Category 4 and ore aqua regia data represents the effect of all sulfide minerals combined. See example calculation in Section 8.1.2.3.

Large Table 3 Stockpile Rock from Block Model Compared to Waste Characterization Tests

		Block Model Proposed Total Waste Rock					Waste Characterization Tests Drill Cores Within Preliminary Pit Shell				
Category	*Unit*	*Tons**	*% of Rock*	*Minimum %S*	*Average %S*	*Maximum %S*	*Samples*	*% of Samples*	*Minimum %S*	*Average %S*	*Maximum %S*
Category 1	1	1,153,645	0.37%	0.06	0.1	0.12	10	11.24%	0.02	0.07	0.09
Category 1	2+	110,439,546	35.84%	0.01	0.07	0.12	15	16.85%	0.02	0.06	0.12
Category 1	3+						8	8.99%	0.02	0.04	0.06
Category 1	4+	46,077,080	14.95%	0.01	0.06	0.12	5	5.62%	0.03	0.05	0.08
Category 1	5+						2	2.25%	0.02	0.04	0.06
Category 1	6	38,800,965	12.59%	0.01	0.05	0.12	2	2.25%	0.04	0.05	0.05
Category 1	7	20,223,481	6.56%	0.02	0.06	0.12					
Total Category 1		216,694,717	70.33%		0.06		42	47.19%		0.05	
Category 2/3	1	9,355,612	3.04%	0.13	0.32	0.6	13	14.61%	0.16	0.31	0.55
Category 2/3	2+	44,659,052	14.49%	0.13	0.19	0.6	3	3.37%	0.15	0.17	0.18
Category 2/3	3+						5	5.62%	0.14	0.30	0.59
Category 2/3	4+	23,807,049	7.73%	0.13	0.2	0.6	2	2.25%	0.21	0.36	0.51
Category 2/3	5+						2	2.25%	0.23	0.28	0.32
Category 2/3	6	2,558,344	0.83%	0.13	0.17	0.51	1	1.12%	0.18	0.18	0.18
Category 2/3	7	2,402,286	0.78%	0.13	0.17	0.46					
Total Category 2/3		82,782,343	26.87%		0.21		26	29.21%		0.05	
Category 4	1	1,111,358	0.36%	0.61	0.85	2.31	15	16.85%	0.68	1.50	4.46
Category 4	3+	1,872,851	0.61%	0.61	0.99	3.04					
Category 4	4+	94,604	0.03%	0.62	0.81	1.14	2	2.25%	0.77	1.07	1.37
Category 4	5+										
Category 4	6	4	≈0.00%	0.65	0.65	0.65					
Category 4	VF	5,557,813	1.80%	0.34	2.43	4.94	4	4.49%	2.00	3.82	5.68
Total Category 4		8,636,630	2.80%		1.90		21	23.60%		1.90	
Total All Categories		308,113,690			0.15		89				

Notes

* Total of rock to stockpiles and East Pit

+ Units 2 and 3 are combined and Units 4 and 5 are combined in the Block Model

Large Table 4 Summary of NorthMet Project Humidity Cells (Tailings)

Tailings Portion	*Tailings Source*	*Size (Mesh)*	*HCT ID*	*Sulfur (%)*	*Total Duration (weeks)*
Fine	Parcel 1-2	< #200	T10	0.09	413
Fine	Parcel 3	< #200	T13	0.14	413
Fine	Pilot Plant 2	< #200	T55	0.09	276
Fine	Pilot Plant 3	< #200	T59	0.08	288
Fine	SCAV	< #200	T63	0.11	216
Fine	SCAV	< #200	T67	0.14	216
Fine	SCAV	< #200	T71	0.13	216
Coarse	Parcel 1-2	#200 - #100	T9	0.1	413
Coarse	Parcel 3	#200 - #100	T12	0.14	413
Coarse	Pilot Plant 2	#200 - #100	T54	0.06	288
Coarse	Pilot Plant 3	#200 - #100	T58	0.08	288
Coarse	SCAV	#200 - #100	T62	0.09	216
Coarse	SCAV	#200 - #100	T66	0.14	216
Coarse	SCAV	#200 - #100	T70	0.1	216
Coarse	Parcel 1-2	> #100	T8	0.11	413
Coarse	Parcel 3	> #100	T11	0.11	413
Coarse	Pilot Plant 2	> #100	T53	0.08	288
Coarse	Pilot Plant 3	> #100	T57	0.1	288
Coarse	SCAV	> #100	T61	0.1	216
Coarse	SCAV	> #100	T65	0.11	216
Coarse	SCAV	> #100	T69	0.1	216
LTVSMC	2E North Embankment	Whole LTVSMC	T73	0.03	181
LTVSMC	1E/2E Separator	Whole LTVSMC	T75	0.04	181
LTVSMC	1E South Beach	Whole LTVSMC	T76	0.01	181
LTVSMC	2W North Embankment	Whole LTVSMC	T77	0.06	181

Notes

- Time periods as of January 17, 2014.
- All Flotation Tailings samples included the use of $CuSO_4$ in the pilot plant processing
- All humidity cells referenced here were conducted according to ASTM methods
- Material retained on mesh #100 (previously referred to as “coarse”) and material passing mesh #100 but retained on mesh #200 (previously referred to as “mid”) are collectively considered as coarse material (retained on mesh #200) according to the modeling methodology described in Section 5.1.3.1

Large Table 5 Proposed Approach for Deriving Release Rates (Tailings)

		Flotation Tailings (Coarse and Fine)				*LTVSMC Tailings (Whole)*			
Constituent	***Goldschmidt Classification***	***Mineral Sources***	***Ion***	***Method***	***Source***	***Mineral Sources***	***Ion***	***Method***	***Source***
Primary Ions									
SO_4	Chalcophile	Chalcopyrite > pyrrhotite	SO_4^{2-}	SO_4 Rate	Tailings HCT	Pyrite	SO_4^{2-}	SO_4 Rate	Tailings HCT
Alkalinity	Atmophile	Calcite, dolomite	CO_3^{2-}	$CaCO_3$ control at atmospheric PCO_2	Solubility Model	Carbonate Minerals	CO_3^{2-}	$CaCO_3$ control at atmospheric PCO_2	Solubility Model
Ca	Lithophile	Anorthite, calcite	Ca^{2+}	Ca/Na Rate	Tailings HCT	Carbonate Minerals	Ca^{2+}	Concentration capped	Observed Seepage
Cl	Nonmetallic	Unknown	Cl^-	No release	None	Unknown	Cl^-	Concentration capped	Observed Seepage
Mg	Lithophile	Olivine	Mg^{2+}	Mg/SO_4 Rate	Tailings HCT	Carbonate Minerals	Mg^{2+}	Mg/Ca Ratio	Observed Seepage
Na	Lithophile	Albite	Na^+	Na/SO_4 Rate	Tailings HCT	Albite	Na^+	Concentration capped	Observed Seepage
K	Lithophile	Biotite	K^+	K/SO_4 Rate	Tailings HCT	Mica	K^+	Concentration capped	Observed Seepage
Secondary Ions									
Ag	Chalcophile	Sulfide	Ag^+	Ag/S	Aqua Regia	Pyrite	Ag^+	Ag/S	Aqua Regia
Al	Lithophile	Anorthite, albite	Al^{3+}	Al/Ca (anorthite) + Al/Na (albite)	Mineral forumulae	Feldspar	Al^{3+}	Concentration capped	Observed Seepage
As	Chalcophile	Sulfide	$HAsO_4^{2-}$ /$H_2AsO_4^-$	As/S	Aqua Regia	Pyrite	$HAsO_4^{2-}$ /$H_2AsO_4^-$	As/S	Aqua Regia
B	Lithophile	Unknown	H_3BO_3	Solubility control	Defined concentration cap	Unknown	H_3BO_3	Concentration capped	Observed Seepage
Ba	Lithophile	Biotite	Ba^{2+}	Ba/K Ratio	Aqua Regia	Mica	Ba^{2+}	$BaSO_4$ control	Solubility Model
Be	Lithophile	Biotite	Be^{2+}	Be/K Ratio	Aqua Regia	Mica	Be^{2+}	Concentration capped	Observed Seepage
Cd	Chalcophile	Sulfide	Cd^{2+}	Cd/Zn Rate, Condition 2	Cat 2/3 Waste Rock HCT	Pyrite	Cd^{2+}	Cd/S	Aqua Regia
Co	Siderophile	Olivine	Co^{2+}	Co/Ni Rate, Condition 2	Cat 2/3 Waste Rock HCT	Pyrite	Co^{2+}	Co/S	Aqua Regia
Cr	Lithophile	Unknown	$CrOH^{2+}$	Solubility control	Defined concentration cap	Unknown	$CrOH^{2+}$	Concentration capped	Observed Seepage
Cu	Chalcophile	Chalcopyrite	Cu^{2+}	Cu/S	Aqua Regia	Chalcopyrite	Cu^{2+}	Cu/S	Aqua Regia
F	Lithophile	Apatite	F^-	CaF_2 control	Solubility Model	Apatite	F^-	CaF_2 control	Solubility Model
Fe	Siderophile	Sulfides, silicates	Fe^{3+}	Fe/S (pyrrhotite) + Fe/Mg (olivine)	Microprobe	Pyrite, carbonates	Fe^{3+}	Fe/S (pyrite)	Microprobe
Mn	Siderophile	Sulfide	Mn^{2+}	Mn/Ni Rate	Tailings HCT	Pyrite	Mn^{2+}	Concentration capped	Observed Seepage
Ni	Siderophile	Sulfides / olivine	Ni^{2+}	Ni/S (pyrrhotite)	Microprobe	Pyrite	Ni^{2+}	Ni/S	Aqua Regia
Pb	Chalcophile	Sulfide	Pb^{2+}	Pb/S	Aqua Regia	Pyrite	Pb^{2+}	Pb/S	Aqua Regia
Sb	Chalcophile	Sulfide	Oxyanion	Sb/S	Aqua Regia	Pyrite	Oxyanion	Sb/S	Aqua Regia
Se	Chalcophile	Chalcopyrite	SeO_4^{2-}	Se/SO_4 Rate	Tailings HCT	Pyrite	SeO_4^{2-}	Se/SO_4 Rate	Tailings HCT
Tl	Chalcophile	Unknown	Tl^+	Tl/S	Aqua Regia	Pyrite	Tl^+	Tl/S	Aqua Regia
V	Lithophile	Biotite	$VO_2(OH)^{2-}$	V/K	Aqua Regia	Mica	VO_2^+	Concentration capped	Observed Seepage
Zn	Chalcophile	Olivine	Zn^{2+}	Zn/Ni Rate, Condition 2	Cat 2/3 Waste Rock HCT	Pyrite	Zn^{2+}	Zn/SO_4 Rate	Tailings HCT

Notes

- “Rate” indicates a release rate developed from humidity cells for primary ions or a ratio of release rates used to estimate metal release from other release rates (ex. Mn/Ni Rate for NorthMet tailings)
- Metal ratios with a data source of “Aqua Regia” or “Microprobe” indicate a solids ratio that is used to estimate metal release from the primary ion release rates
- “Condition 2” refers to a time period in the waste rock humidity cells with relatively stable sulfate release, leachate pH below about 7 and nickel release unstable and typically increasing (i.e., neutral conditions)
- “Solubility model”, “defined concentration cap" and “observed seepage” indicate constituents that are assumed to be at the referenced concentration caps (modeled or inferred from site or analog data) at all times (i.e., no release rate is defined)

Large Table 6 Distribution Parameters for Category 1 Waste Rock Release

Distribution from Regression Analysis of Humidity Cell Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
SO_4	Rate Regression	HCT (1+2)	mg SO_4/kg/week/%S	Normal	13.92	0.581	--	--

Distribution Fit to Humidity Cell Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Ag	Rate	HCT (1+2)*	mg Ag/kg/week	Lognormal	6.46E-06	2.11E-02	--	--
Alkalinity	Rate	HCT (1+2)	mg Alk/kg/week	Beta	4.92E+00	2.21E+00	2.63E+00	1.15E+01
As	Rate	HCT (1+2)*	mg As/kg/week	Lognormal	1.85E-04	1.84E-04	--	--
B	Rate	HCT (1+2)*	mg B/kg/week	Lognormal	4.33E-03	1.27E-02	--	--
Be	Rate	HCT (1+2)*	mg Be/kg/week	Lognormal	6.37E-06	4.61E-05	--	--
Ca	Rate	HCT (1+2)	mg Ca/kg/week	Beta	1.15E+00	3.48E-01	5.78E-01	2.34E+00
Cd	Zn rate ratio	HCT (2)	mg Cd / mg Zn	Beta	2.03E-02	5.10E-03	1.44E-02	4.44E-02
Cl	First flush	HCT (all)	mg Cl / kg rock	Beta	9.78E+00	1.17E+01	1.38E+00	7.30E+01
Co	Ni rate ratio	HCT (2)	mg Co / mg Ni	Beta	1.55E-01	5.11E-02	7.28E-02	3.11E-01
Cr	Rate	HCT (1+2)*	mg Cr/kg/week	Lognormal	5.90E-05	2.80E-05	--	--
F	Rate	HCT (1+2)	mg F/kg/week	Beta	2.33E-02	1.08E-03	1.99E-02	2.52E-02
K	Rate	HCT (1+2)	mg K/kg/week	Beta	2.14E-01	9.17E-02	1.02E-01	4.98E-01
Mg	Rate	HCT (1+2)	mg Mg/kg/week	Beta	3.14E-01	2.04E-01	1.31E-01	1.10E+00
Mn	SO_4 rate ratio	HCT (2)	mg Mn / mg SO_4	Beta	1.96E-03	9.73E-04	1.15E-03	5.95E-03
Na	Rate	HCT (1+2)	mg Na/kg/week	Beta	4.13E-01	4.02E-01	1.28E-01	2.50E+00
Pb	Rate	HCT (1+2)*	mg Pb/kg/week	Lognormal	6.56E-06	6.44E-06	--	--
Sb	Rate	Reactor*	mg Sb/kg/week	Lognormal	4.36E-04	4.81E-04	--	--
Se	SO_4 rate ratio	HCT (2)*	mg Se / mg SO_4	Lognormal	1.90E-05	2.32E-04	--	--
Tl	Rate	HCT (1+2)*	mg Tl/kg/week	Lognormal	9.23E-07	1.77E-05	--	--
V	Rate	HCT (1+2)*	mg V/kg/week	Lognormal	1.52E-04	1.68E-04	--	--

Distribution from Aqua Regia Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Ba	K ratio	Aqua Regia	mg Ba / mg K	Normal	2.90E-02	1.98E-04	--	--
Cu	S ratio	Aqua Regia	mg Cu / mg S	Normal	5.87E-01	2.51E-02	--	--
Zn	Mg ratio	Aqua Regia	mg Zn / mg Mg	Normal	1.81E-03	1.35E-05	--	--

Distribution from Other Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Al	Ca ratio	Anorthite Formula	mg Al / mg Ca	Constant	1.35E+00	--	--	--
	Na ratio	Albite Formula	mg Al / mg Na	Constant	1.17E+00	--	--	--
Fe	S ratio	Pyrrhotite microprobe	mg Fe / mg S	Beta	1.62E+00	8.72E-02	1.49E+00	1.92E+00
	Mg ratio	Olivine microprobe	mg Fe / mg Mg	Beta	1.87E+00	6.75E-01	1.19E+00	4.51E+00
Ni	S ratio	Cat 4 Aqua Regia	mg Ni / mg S	Normal	3.06E-02	1.86E-03	--	--
	Mg ratio	Olivine microprobe	mg Ni / mg Mg	Beta	4.59E-03	1.95E-03	1.10E-04	7.43E-03

Notes

- Humidity cell data used through February 2011 unless noted otherwise.
- HCT (1+2) indicates average rates from humidity cells over Condition 1 and Condition 2, as defined in Large Table 1.
- HCT (2) indicates average rates from humidity cells over Condition 2, as defined in Large Table 1.
- ***** indicates average rates from humidity cells over conditions noted with refined modeling of non-detects (see Section 8.1.2.1). Data used though December 2013.
- For Sb only the smaller MDNR-style reactors were used to estimate a release rate, including refined modeling of non-detects. Data used through February 2007.
- Except for SO_4, all distributions from humidity cell data represent the full range of the observed values, with no weighting. Distributions are shown in Large Figures 1-4.
- Distributions from aqua regia data represent the uncertainty in the average ratios, weighted by geologic unit.
- Distributions from microprobe data represent the full range of the observed ratios for each mineral, with no weighting. Distributions are shown in Large Figures 21-22.
- For nickel, S ratio from Duluth Complex Category 4 aqua regia data represents the effect of all sulfide minerals combined. See example calculation in Section 8.1.2.3.
- For chloride, release is a one-time event per unit rock mass, developed from all Project humidity cells. See Section 8.4.4.

Large Table 7 Distribution Parameters for Category 2/3 Waste Rock Release

Distribution from Regression Analysis of Humidity Cell Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
SO_4	Rate Regression	HCT (1+2)	mg SO_4/kg/week/%S	Normal	13.92	0.581	--	--

Distribution Fit to Humidity Cell Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Alkalinity	Nonacidic rate	HCT (1+2)	mg Alk/kg/week	Beta	4.50E+00	2.59E+00	1.45E+00	1.10E+01
	Acidic rate	None	mg Alk/kg/week	Constant	0	--	--	--
B	Nonacidic rate	HCT (1+2)*	mg B/kg/week	Lognormal	5.84E-03	1.10E-03	--	--
	Acidic rate	HCT (3)	mg B/kg/week	Triangular	4.58E-04	--	4.58E-04	1.61E-02
Ca	SO_4 rate ratio	HCT (1+2)	mg Ca / mg SO_4	Beta	6.81E-01	4.29E-01	2.61E-01	2.59E+00
Cd	Zn rate ratio	HCT (2)	mg Cd / mg Zn	Beta	1.65E-02	1.20E-02	1.01E-03	5.84E-02
Cl	First flush	HCT (all)	mg Cl / kg rock	Beta	9.78E+00	1.17E+01	1.38E+00	7.30E+01
Co	Ni rate ratio	HCT (2)	mg Co / mg Ni	Beta	8.29E-02	3.91E-02	2.24E-02	2.06E-01
Cr	Nonacidic rate	HCT (1+2)*	mg Cr/kg/week	Lognormal	5.49E-05	2.19E-05	--	--
	Acidic rate	HCT (3)	mg Cr/kg/week	Triangular	9.17E-05	--	9.17E-05	1.06E-04
F	Nonacidic rate	HCT (1+2)	mg F/kg/week	Beta	2.36E-02	1.45E-03	2.04E-02	2.74E-02
	Acidic rate	HCT (3)	mg F/kg/week	Triangular	2.29E-02		2.27E-03	2.29E-02
K	SO_4 rate ratio	HCT (1+2)	mg K / mg SO_4	Beta	1.29E-01	8.62E-02	5.39E-02	4.00E-01
Mg	SO_4 rate ratio	HCT (1+2)	mg Mg / mg SO_4	Beta	1.39E-01	1.06E-01	3.37E-02	4.96E-01
Mn	SO_4 rate ratio	HCT (2)	mg Mn / mg SO_4	Beta	2.81E-03	2.56E-03	4.36E-04	1.10E-02
Na	SO_4 rate ratio	HCT (1+2)	mg Na / mg SO_4	Beta	1.33E-01	9.29E-02	3.54E-02	4.51E-01
Se	SO_4 rate ratio	HCT (2)	mg Se / mg SO_4	Beta	3.54E-05	1.67E-05	1.30E-05	9.16E-05
Tl	Nonacidic rate	HCT (1+2)*	mg Tl/kg/week	Lognormal	2.73E-06	8.15E-06	--	--
	Acidic rate	HCT (3)	mg Tl/kg/week	Triangular	9.17E-06	--	9.17E-06	2.29E-05
Zn	Ni rate ratio	HCT (2)	mg Zn / mg Ni	Beta	3.35E-01	3.71E-01	3.31E-02	1.60E+00

Distribution from Aqua Regia Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Ag	S ratio	Aqua Regia	mg Ag / mg S	Normal	1.32E-04	4.54E-06	--	--
As	S ratio	Aqua Regia	mg As / mg S	Normal	1.67E-03	1.28E-04	--	--
Ba	K ratio	Aqua Regia	mg Ba / mg K	Normal	2.93E-02	5.69E-04	--	--
Be	K ratio	Aqua Regia	mg Be / mg K	Normal	1.87E-04	3.77E-06	--	--
Cu	S ratio	Aqua Regia	mg Cu / mg S	Normal	3.59E-01	8.84E-03	--	--
Pb	S ratio	Aqua Regia	mg Pb / mg S	Normal	1.24E-03	5.95E-05	--	--
Sb	S ratio	Aqua Regia	mg Sb / mg S	Normal	6.53E-04	2.81E-05	--	--
V	K ratio	Aqua Regia	mg V / mg K	Normal	2.32E-02	7.29E-04	--	--

Distribution from Other Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Al	Ca ratio	Anorthite Formula	mg Al / mg Ca	Constant	1.35E+00	--	--	--
	Na ratio	Albite Formula	mg Al / mg Na	Constant	1.17E+00	--	--	--
Fe	S ratio	Pyrrhotite microprobe	mg Fe / mg S	Beta	1.62E+00	8.72E-02	1.49E+00	1.92E+00
	Mg ratio	Olivine microprobe	mg Fe / mg Mg	Beta	1.87E+00	6.75E-01	1.19E+00	4.51E+00
Ni	S ratio	Cat 4 Aqua Regia	mg Ni / mg S	Normal	3.06E-02	1.86E-03	--	--
	Mg ratio	Olivine microprobe	mg Ni / mg Mg	Beta	4.59E-03	1.95E-03	1.10E-04	7.43E-03

Notes

- Humidity cell data used through February 2011 unless noted otherwise.
- HCT (1+2) indicates average rates from humidity cells over Condition 1 and Condition 2, as defined in Large Table 1.
- HCT (2) indicates average rates from humidity cells over Condition 2, as defined in Large Table 1.
- HCT (3) indicates average rates from humidity cells over Condition 3, as defined in Large Table 1.
- ***** indicates average rates from humidity cells over conditions noted with refined modeling of non-detects (see Section 8.1.2.1). Data used though December 2013.
- Except for SO_4, all distributions from humidity cell data represent the full range of the observed values, with no weighting. Distributions are shown in Large Figures 5-8.
- Acidic release rate for SO_4 to be determined from nonacidic rate times an acidic increase factor, as discussed in Section 8.2.5.
- Distributions from aqua regia data represent the uncertainty in the average ratios, weighted by geologic unit.
- Distributions from microprobe data represent the full range of the observed ratios for each mineral, with no weighting. Distributions are shown in Large Figures 21-22.
- For nickel, S ratio from Duluth Complex Category 4 aqua regia data represents the effect of all sulfide minerals combined. See example calculation in Section 8.1.2.3.
- For chloride, release is a one-time event per unit rock mass, developed from all Project humidity cells. See Section 8.4.4.

Large Table 8 Distribution Parameters for Duluth Complex Category 4 Waste Rock Release

Distribution Fit to Humidity Cell Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Alkalinity	Nonacidic rate	HCT (1+2)	mg Alk/kg/week	Beta	4.43E+00	2.60E+00	1.47E+00	1.10E+01
	Acidic rate	None	mg Alk/kg/week	Constant	0	--	--	--
B	Nonacidic rate	HCT (1+2)*	mg B/kg/week	Lognormal	9.99E-03	6.37E-03	--	--
	Acidic rate	HCT (3)	mg B/kg/week	Beta	2.52E-03	2.49E-03	5.06E-04	1.00E-02
Ca	SO_4 rate ratio	HCT (1+2)	mg Ca / mg SO_4	Beta	3.56E-01	1.26E-01	1.80E-01	7.91E-01
Cd	Zn rate ratio	HCT (2)	mg Cd / mg Zn	Beta	9.16E-03	5.39E-03	2.70E-03	3.15E-02
Cl	First flush	HCT (all)	mg Cl / kg rock	Beta	9.78E+00	1.17E+01	1.38E+00	7.30E+01
Co	Ni rate ratio	HCT (2)	mg Co / mg Ni	Beta	1.56E-01	7.51E-02	7.79E-02	4.64E-01
Cr	Nonacidic rate	HCT (1+2)*	mg Cr/kg/week	Lognormal	4.34E-05	7.03E-05	--	--
	Acidic rate	HCT (3)	mg Cr/kg/week	Beta	1.07E-04	1.20E-05	9.34E-05	1.47E-04
F	Nonacidic rate	HCT (1+2)	mg F/kg/week	Beta	4.68E-02	4.78E-02	2.16E-02	3.37E-01
	Acidic rate	HCT (3)	mg F/kg/week	Beta	2.57E-02	4.30E-03	2.25E-02	4.19E-02
K	SO_4 rate ratio	HCT (1+2)	mg K / mg SO_4	Beta	1.00E-01	5.61E-02	2.61E-04	2.45E-01
Mg	SO_4 rate ratio	HCT (1+2)	mg Mg / mg SO_4	Beta	6.61E-02	4.17E-02	2.92E-02	2.00E-01
Mn	SO_4 rate ratio	HCT (2)	mg Mn / mg SO_4	Beta	2.94E-03	2.15E-03	5.94E-04	9.00E-03
Na	SO_4 rate ratio	HCT (1+2)	mg Na / mg SO_4	Beta	1.06E-01	1.02E-01	1.43E-02	4.51E-01
Se	SO_4 rate ratio	HCT (2)	mg Se / mg SO_4	Beta	1.87E-05	9.12E-06	9.15E-06	4.91E-05
SO_4	Nonacidic rate	HCT (1+2)	mg SO_4/kg/week	Beta	1.27E+01	8.37E+00	3.74E+00	5.50E+01
Tl	Nonacidic rate	HCT (1+2)*	mg Tl/kg/week	Lognormal	7.36E-06	6.40E-06	--	--
	Acidic rate	HCT (3)	mg Tl/kg/week	Beta	1.54E-05	7.94E-06	9.73E-06	4.26E-05
Zn	Ni rate ratio	HCT (2)	mg Zn / mg Ni	Beta	4.42E-01	6.79E-01	3.47E-02	3.50E+00

Distribution from Aqua Regia Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Ag	S ratio	Aqua Regia	mg Ag / mg S	Normal	3.30E-05	3.21E-06	--	--
As	S ratio	Aqua Regia	mg As / mg S	Normal	1.40E-03	1.13E-04	--	--
Ba	K ratio	Aqua Regia	mg Ba / mg K	Normal	2.46E-02	1.17E-03	--	--
Be	K ratio	Aqua Regia	mg Be / mg K	Normal	3.30E-04	3.04E-05	--	--
Cu	S ratio	Aqua Regia	mg Cu / mg S	Normal	6.81E-02	4.76E-03	--	--
Pb	S ratio	Aqua Regia	mg Pb / mg S	Normal	3.97E-04	4.33E-05	--	--
Sb	S ratio	Aqua Regia	mg Sb / mg S	Normal	1.30E-04	9.01E-06	--	--
V	K ratio	Aqua Regia	mg V / mg K	Normal	4.33E-02	3.24E-03	--	--

Distribution from Other Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Al	Ca ratio	Anorthite Formula	mg Al / mg Ca	Beta	1.35E+00	--	--	--
	Na ratio	Albite Formula	mg Al / mg Na	Beta	1.17E+00	--	--	--
Fe	S ratio	Pyrrhotite microprobe	mg Fe / mg S	Beta	1.62E+00	8.72E-02	1.49E+00	1.92E+00
	Mg ratio	Olivine microprobe	mg Fe / mg Mg	Beta	1.87E+00	6.75E-01	1.19E+00	4.51E+00
Ni	S ratio	Cat 4 Aqua Regia	mg Ni / mg S	Normal	3.06E-02	1.86E-03	--	--
	Mg ratio	Olivine microprobe	mg Ni / mg Mg	Beta	4.59E-03	1.95E-03	1.10E-04	7.43E-03

Notes

- Humidity cell data used through February 2011 unless noted otherwise.
- HCT (1+2) indicates average rates from humidity cells over Condition 1 and Condition 2, as defined in Large Table 1.
- HCT (2) indicates average rates from humidity cells over Condition 2, as defined in Large Table 1.
- HCT (3) indicates average rates from humidity cells over Condition 3, as defined in Large Table 1.
- ***** indicates average rates from humidity cells over conditions noted with refined modeling of non-detects (see Section 8.1.2.1). Data used though December 2013.
- All distributions from humidity cell data represent the full range of the observed values, with no weighting. Distributions are shown in Large Figures 9-12.
- Acidic release rate for SO_4 to be determined from nonacidic rate times an acidic increase factor, as discussed in Section 8.2.5.
- Distributions from aqua regia data represent the uncertainty in the average ratios, weighted by geologic unit.
- Distributions from microprobe data represent the full range of the observed ratios for each mineral, with no weighting. Distributions are shown in Large Figures 21-22.
- For nickel, S ratio from Duluth Complex Category 4 aqua regia data represents the effect of all sulfide minerals combined. See example calculation in Section 8.1.2.3.
- For chloride, release is a one-time event per unit rock mass, developed from all Project humidity cells. See Section 8.4.4.

Large Table 9 Distribution Parameters for Ore Release

Distribution from Regression Analysis of Humidity Cell Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
SO_4	Rate Regression	HCT (1+2)	mg SO_4/kg/week/%S	Normal	13.92	0.581	--	--

Distribution Fit to Humidity Cell Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Alkalinity	Nonacidic rate	HCT (1+2)	mg Alk/kg/week	Triangular	1.52E+00	--	1.37E+00	1.52E+00
	Acidic rate	None	mg Alk/kg/week	Constant	0	--	--	--
B	Nonacidic rate	HCT (1+2)	mg B/kg/week	Triangular	5.85E-03	--	5.09E-03	1.49E-02
Ca	SO_4 rate ratio	HCT (1+2)	mg Ca / mg SO_4	Triangular	2.16E-01	--	2.16E-01	2.18E-01
Cd	Zn rate ratio	HCT (2)	mg Cd / mg Zn	Triangular	5.76E-03	--	5.76E-03	6.72E-03
Cl	First flush	HCT (all)	mg Cl / kg rock	Beta	9.78E+00	1.17E+01	1.38E+00	7.30E+01
Co	Ni rate ratio	HCT (2)	mg Co / mg Ni	Triangular	4.86E-02	--	4.86E-02	6.08E-02
Cr	Nonacidic rate	HCT (1+2)	mg Cr/kg/week	Triangular	1.10E-04	--	1.10E-04	1.18E-04
F	Nonacidic rate	HCT (1+2)	mg F/kg/week	Triangular	2.39E-02	--	2.39E-02	2.96E-02
K	SO_4 rate ratio	HCT (1+2)	mg K / mg SO_4	Triangular	3.97E-02	--	3.22E-02	4.16E-02
Mg	SO_4 rate ratio	HCT (1+2)	mg Mg / mg SO_4	Triangular	7.29E-02	--	7.29E-02	8.22E-02
Mn	SO_4 rate ratio	HCT (2)	mg Mn / mg SO_4	Triangular	5.89E-03	--	5.45E-03	6.27E-03
Na	SO_4 rate ratio	HCT (1+2)	mg Na / mg SO_4	Triangular	1.21E-02	--	1.21E-02	2.96E-01
Se	SO_4 rate ratio	HCT (2)	mg Se / mg SO_4	Triangular	4.01E-05	--	4.01E-05	4.42E-05
Tl	Nonacidic rate	HCT (1+2)	mg Tl/kg/week	Triangular	2.22E-05	--	1.74E-05	2.22E-05
Zn	Ni rate ratio	HCT (2)	mg Zn / mg Ni	Triangular	2.28E-02	--	2.26E-02	3.00E-02

Distribution from Aqua Regia Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Ag	S ratio	Aqua Regia	mg Ag / mg S	Normal	1.87E-04	2.80E-06	--	--
As	S ratio	Aqua Regia	mg As / mg S	Normal	9.20E-04	3.48E-05	--	--
Ba	K ratio	Aqua Regia	mg Ba / mg K	Normal	2.77E-02	1.06E-04	--	--
Be	K ratio	Aqua Regia	mg Be / mg K	Normal	1.22E-04	1.97E-06	--	--
Cu	S ratio	Aqua Regia	mg Cu / mg S	Normal	5.04E-01	5.62E-03	--	--
Pb	S ratio	Aqua Regia	mg Pb / mg S	Normal	1.05E-03	4.85E-05	--	--
Sb	S ratio	Aqua Regia	mg Sb / mg S	Normal	3.38E-04	1.17E-05	--	--
V	K ratio	Aqua Regia	mg V / mg K	Normal	2.19E-02	3.36E-04	--	--

Distribution from Other Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Al	Ca ratio	Anorthite Formula	mg Al / mg Ca	Constant	1.35E+00	--	--	--
	Na ratio	Albite Formula	mg Al / mg Na	Constant	1.17E+00	--	--	--
Fe	S ratio	Pyrrhotite microprobe	mg Fe / mg S	Beta	1.62E+00	8.72E-02	1.49E+00	1.92E+00
	Mg ratio	Olivine microprobe	mg Fe / mg Mg	Beta	1.87E+00	6.75E-01	1.19E+00	4.51E+00
Ni	S ratio	Ore Aqua Regia	mg Ni / mg S	Normal	1.53E-01	3.26E-03	--	--
	Mg ratio	Olivine microprobe	mg Ni / mg Mg	Beta	4.59E-03	1.95E-03	1.10E-04	7.43E-03

Notes

- Humidity cell data used through February 2011 unless noted otherwise.
- HCT (1+2) indicates average rates from humidity cells over Condition 1 and Condition 2, as defined in Large Table 1.
- HCT (2) indicates average rates from humidity cells over Condition 2, as defined in Large Table 1.
- Except for SO4, all distributions from humidity cell data represent the full range of the observed values in the humidity cells, with no weighting. Distributions are shown in Large Figures 13-16. Distributions from humidity cells shown here are only used for the blended OSP; Category 2/3 humidity cells (see Large Table 8) are used to capture the full range of variability in the ore wall rock. Distributions are shown in Large Figures 13-16.
- Acidic release rate for SO_4 to be determined from nonacidic rate times an acidic increase factor, as discussed in Section 8.2.5.
- Distributions from aqua regia data represent the uncertainty in the average ratios, weighted by geologic unit.
- Distributions from microprobe data represent the full range of the observed ratios for each mineral, with no weighting. Distributions are shown in Large Figures 21-22.
- For nickel, S ratio from ore aqua regia data represents the effect of all sulfide minerals combined. See example calculation in Section 8.1.2.3.
- For B, Cr, F, and Tl no increase in release rates due to acidic conditions is indicated by laboratory data.
- For chloride, release is a one-time event per unit rock mass, developed from all Project humidity cells. See Section 8.4.4.

Large Table 10 Distribution Parameters for Virginia Formation Category 4 Waste Rock Release

Distribution Fit to Humidity Cell Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Alkalinity	Acidic rate	None	mg Alk/kg/week	Constant	0	--	--	--
B	Acidic rate	HCT (3)	mg B/kg/week	Triangular	6.70E-03	--	6.70E-03	1.70E-02
Ca	SO_4 rate ratio	HCT (3)	mg Ca / mg SO_4	Triangular	2.32E-02	--	2.32E-02	2.50E-01
Cl	First flush	HCT (all)	mg Cl / kg rock	Beta	9.78E+00	1.17E+01	1.38E+00	7.30E+01
Cr	Acidic rate	HCT (3)	mg Cr/kg/week	Triangular	1.11E-04	--	9.14E-05	1.28E-04
F	Acidic rate	HCT (3)	mg F/kg/week	Triangular	2.50E-02	--	2.50E-02	4.98E-02
Fe	SO_4 rate ratio	HCT (3)	mg Fe / mg SO_4	Triangular	5.80E-02	--	3.98E-02	3.16E-01
K	SO_4 rate ratio	HCT (3)	mg K / mg SO_4	Triangular	8.03E-03	--	8.03E-03	1.79E-02
Mg	SO_4 rate ratio	HCT (3)	mg Mg / mg SO_4	Triangular	5.32E-02	--	2.93E-02	7.83E-02
Mn	Acidic rate	HCT (3)	mg Mn/kg/week	Triangular	7.11E-02	--	3.49E-02	1.56E-01
Na	SO_4 rate ratio	HCT (3)	mg Na / mg SO_4	Triangular	5.64E-03	--	5.64E-03	1.79E-02
Se	SO_4 rate ratio	HCT (3)	mg Se / mg SO_4	Triangular	8.52E-06	--	4.86E-06	9.20E-06
SO_4	Acidic rate	HCT (3)	mg SO_4/kg/week	Triangular	5.76E+01	--	4.44E+01	5.76E+01
Tl	Acidic rate	HCT (3)	mg Tl/kg/week	Triangular	1.11E-05	--	9.92E-06	1.21E-05

Distribution from Aqua Regia Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Ag	S ratio	Aqua Regia	mg Ag / mg S	Normal	3.42E-05	2.23E-06	--	--
As	S ratio	Aqua Regia	mg As / mg S	Normal	2.87E-03	1.28E-04	--	--
Ba	K ratio	Aqua Regia	mg Ba / mg K	Normal	1.51E-02	5.79E-04	--	--
Be	S ratio	Aqua Regia	mg Be / mg S	Normal	1.02E-04	1.02E-05	--	--
Cd	S ratio	Aqua Regia	mg Cd / mg S	Normal	1.88E-04	5.11E-05	--	--
Co	S ratio	Aqua Regia	mg Co / mg S	Normal	4.26E-03	6.15E-04	--	--
Cu	S ratio	Aqua Regia	mg Cu / mg S	Normal	2.51E-02	2.59E-03	--	--
Ni	S ratio	Aqua Regia	mg Ni / mg S	Normal	1.76E-02	1.39E-03	--	--
Pb	S ratio	Aqua Regia	mg Pb / mg S	Normal	9.23E-04	3.07E-04	--	--
Sb	S ratio	Aqua Regia	mg Sb / mg S	Normal	2.70E-04	2.28E-05	--	--
V	K ratio	Aqua Regia	mg V / mg K	Normal	2.18E-02	1.07E-03	--	--
Zn	S ratio	Aqua Regia	mg Zn / mg S	Normal	3.03E-02	2.88E-03	--	--

Distribution from Other Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Al	Ca ratio	Anorthite Formula	mg Al / mg Ca	Constant	1.35E+00	--	--	--
	Na ratio	Albite Formula	mg Al / mg Na	Constant	1.17E+00	--	--	--

Notes

- Humidity cell data used through February 2011 unless noted otherwise.
- HCT (3) indicates average rates from humidity cells over Condition 3, as defined in Large Table 1.
- All distributions from humidity cell data represent the full range of the observed values, with no weighting. Distributions are shown in Large Figures 17-20.
- Distributions from aqua regia data represent the uncertainty in the average ratios, weighted by geologic unit.
- For chloride, release is a one-time event per unit rock mass, developed from all Project humidity cells. See Section 8.4.4.

Large Table 11 Proposed Approach for Deriving Concentration Caps

Constituent	*Duluth Complex Category 1*		*Non-Acidic Duluth Complex Category 2/3, 4 and Ore*		*Acidic Duluth Complex Category 2/3, 4 and Ore*		*Virginia Formation Category 4*	
	Data Sources	*Comments*	*Data Sources*	*Comments*	*Data Sources*	*Comments*	*Data Sources*	*Comments*
Primary Ions								
SO_4	Gypsum solubility		Gypsum solubility		Whistle Mine		Vangorda Mine	
Alkalinity	AMAX Pile pH 7 to 7.5	pH-dependent	AMAX Pile pH 6 to 7.5	pH-dependent	No alkalinity		No alkalinity	
Ca	Gypsum solubility		Gypsum solubility		Whistle Mine		Vangorda Mine	
Cl	None	No cap	None	No cap	None	No cap	None	No cap
Mg	Mg/Ca release ratio		Mg/Ca release ratio		Whistle Mine		Vangorda Mine	
Na	AMAX Pile pH 7 to 7.5	pH-dependent	AMAX Pile pH 6 to 7.5	pH-dependent	Whistle Mine		Vangorda Mine	
K	AMAX Pile pH 7 to 7.5	pH-dependent	AMAX Pile pH 6 to 7.5	pH-dependent	Whistle Mine		Vangorda Mine	
Secondary Ions								
Ag	Dunka Seep		Dunka Seep		Whistle Mine		Vangorda Mine	
Al	Gibbsite solubility	pH-dependent	Gibbsite solubility	pH-dependent	Whistle Mine		Whistle Mine	Higher than Vangorda Mine
As	Whistle Mine		Whistle Mine		Whistle Mine		Vangorda Mine	
B	Whistle Mine		Whistle Mine		Whistle Mine		Vangorda Mine	
Ba	Barite solubility		Barite solubility		Barite solubility		Vangorda Mine	
Be	Dunka Seep		Dunka Seep		Whistle Mine		Vangorda Mine	
Cd	Cd/Zn release ratio	Limited by Cd/Zn ratio	Cd/Zn release ratio	Limited by Cd/Zn ratio	Whistle Mine		Whistle Mine	Vangorda Cd mineralized
Co	AMAX Pile pH 7 to 7.5	pH-dependent	AMAX Pile pH 6 to 7.5	pH-dependent	Whistle Mine	Comparable to AMAX	Vangorda Mine	
Cr	Whistle Mine		Whistle Mine		Whistle Mine		Vangorda Mine	
Cu	AMAX Pile pH 7 to 7.5	pH-dependent	AMAX Pile pH 6 to 7.5	pH-dependent	AMAX Pile pH 3 to 4	Higher than Whistle Mine	Vangorda Mine	
F	Fluorite solubility		Fluorite solubility		Fluorite solubility		Fluorite solubility	
Fe	AMAX Pile pH 7 to 7.5	pH-dependent	AMAX Pile pH 6 to 7.5	pH-dependent	Whistle Mine		Vangorda Mine	
Mn	AMAX Pile pH 7 to 7.5	pH-dependent	AMAX Pile pH 6 to 7.5	pH-dependent	Whistle Mine	Comparable to AMAX	Vangorda Mine	
Ni	AMAX Pile pH 7 to 7.5	pH-dependent	AMAX Pile pH 6 to 7.5	pH-dependent	Whistle Mine	Comparable to AMAX	Vangorda Mine	
Pb	Whistle Mine		Whistle Mine		Whistle Mine		Whistle Mine	Vangorda Pb mineralized
Sb	NorthMet Lab Data		NorthMet Lab Data		Whistle Mine		Vangorda Mine	
Se	Se/SO_4 release ratio	Limited by Se/SO4 ratio	Se/SO_4 release ratio	Limited by Se/SO4 ratio	Whistle Mine		Vangorda Mine	
Tl	Dunka Seep		Dunka Seep		Vangorda Mine	No data for Whistle or AMAX	Vangorda Mine	
V	Whistle Mine		Whistle Mine		Whistle Mine		Vangorda Mine	
Zn	AMAX Pile pH 7 to 7.5	pH-dependent	AMAX Pile pH 6 to 7.5	pH-dependent	Whistle Mine	Comparable to AMAX	Whistle Mine	Vangorda Zn mineralized

Notes
- AMAX indicates the concentrations observed from the MDNR AMAX test piles at the referenced pH values.
- Whistle Mine indicates the concentrations observed from the Whistle Mine in Ontario, Canada for acidic & nonacidic waters as presented in Attachment A.
- Vangorda Mine indicates the concentrations observed from the Anvil Range Mine Complex in Yukon, Canada for acidic waters as presented in Attachment A.
- Dunka Seep indicates the highest observed concentration or detection limit from the available Dunka Mine data (a single sampling event in May 2006 at Seep X) multiplied by a factor of 10.
- NorthMet Lab Data indicates a range between the highest observed concentration in the NorthMet tailings humidity cells and an estimated field-scale value developed by MDNR.

Large Table 12 Distribution Parameters for Category 1 Waste Rock Concentration Caps

Cap Value From Various Data Sources

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*Std. Deviation*	*Min.*	*Max.*
Ag	Limit	Dunka Seep	mg/L	Constant	2.00E-04	--	--	--
As	Limit	Whistle Mine	mg/L	Constant	1.00E-01	--	--	--
B	Limit	Whistle Mine	mg/L	Constant	1.00E-01	--	--	--
Be	Limit	Dunka Seep	mg/L	Constant	4.00E-04	--	--	--
Cr	Limit	Whistle Mine	mg/L	Constant	1.00E-02	--	--	--
Pb	Limit	Whistle Mine	mg/L	Constant	1.00E-01	--	--	--
Sb	Limit	NorthMet Lab Data	mg/L	Uniform	--	--	8.30E-03	1.00E-01
Tl	Limit	Dunka Seep	mg/L	Constant	2.00E-04	--	--	--
V	Limit	Whistle Mine	mg/L	Constant	1.00E-02	--	--	--

pH-based Range from AMAX Data

(95th percentile values, all units mg/L)

pH	*Alkalinity*	*Co*	*Cu*	*Fe*	*K*	*Mn*	*Na*	*Ni*	*Zn*
8.1	5.00E+01	3.00E-02	3.00E-02	2.00E-02	4.00E+01	1.40E-01	2.40E+02	3.60E-01	2.00E-02
8.0	4.50E+01	1.00E-02	2.00E-02	2.00E-02	4.30E+01	1.40E-01	1.15E+02	2.00E-01	5.20E-02
7.9	4.00E+01	7.58E-02	5.73E-02	3.80E-02	4.80E+01	2.88E-01	3.90E+02	5.26E-01	8.88E-02
7.8	4.20E+01	6.00E-02	1.31E-01	5.50E-02	3.95E+01	2.05E-01	3.70E+02	3.75E-01	6.50E-02
7.7	4.50E+01	4.36E-02	1.23E-01	6.35E-02	4.37E+01	3.19E-01	4.68E+02	4.85E-01	1.15E-01
7.6	5.07E+01	4.00E-02	1.54E-01	7.75E-02	4.72E+01	2.10E-01	3.10E+02	4.55E-01	1.19E-01
7.5	4.82E+01	5.00E-02	1.00E-01	4.00E-02	4.60E+01	2.27E-01	2.18E+02	9.05E-01	9.64E-02
7.4	4.92E+01	7.00E-02	9.68E-02	4.20E-02	4.28E+01	1.72E-01	2.19E+02	1.28E+00	7.00E-02
7.3	3.59E+01	9.30E-02	2.00E-01	5.00E-02	5.04E+01	2.00E-01	2.31E+02	1.62E+00	1.33E-01
7.2	3.55E+01	1.36E-01	1.78E-01	1.01E-01	4.28E+01	1.75E-01	1.73E+02	2.08E+00	1.70E-01
7.1	3.45E+01	2.33E-01	2.85E-01	7.50E-02	4.61E+01	3.86E-01	1.38E+02	4.31E+00	2.93E-01
7.0	2.60E+01	2.80E-01	5.20E-01	4.00E-02	3.99E+01	3.08E-01	1.32E+02	5.91E+00	4.05E-01

pH-based Range from AMAX Data

(maximum values, all units mg/L)

pH	*Alkalinity*	*Co*	*Cu*	*Fe*	*K*	*Mn*	*Na*	*Ni*	*Zn*
8.1	7.00E+01	4.00E-02	4.00E-02	6.00E-02	4.60E+01	1.60E-01	3.17E+02	4.60E-01	2.50E-02
8.0	5.50E+01	1.00E-02	2.00E-02	2.00E-02	4.30E+01	1.40E-01	1.15E+02	2.00E-01	5.20E-02
7.9	4.00E+01	9.00E-02	6.00E-02	4.00E-02	4.90E+01	2.90E-01	3.95E+02	5.65E-01	9.00E-02
7.8	5.90E+01	7.00E-02	1.70E-01	6.00E-02	4.00E+01	2.40E-01	3.72E+02	4.20E-01	7.00E-02
7.7	5.10E+01	5.20E-02	1.31E-01	7.00E-02	5.00E+01	3.40E-01	5.55E+02	5.90E-01	1.20E-01
7.6	5.90E+01	6.00E-02	1.90E-01	2.10E-01	5.20E+01	2.30E-01	3.39E+02	1.07E+00	1.34E-01
7.5	5.27E+01	5.00E-02	1.30E-01	7.00E-02	6.00E+01	2.40E-01	3.13E+02	1.70E+00	1.00E-01
7.4	5.40E+01	8.00E-02	1.80E-01	6.00E-02	5.32E+01	1.90E-01	3.22E+02	1.35E+00	1.12E-01
7.3	3.60E+01	1.20E-01	2.60E-01	6.00E-02	5.90E+01	3.00E-01	2.60E+02	2.29E+00	2.30E-01
7.2	4.50E+01	1.50E-01	3.40E-01	7.00E-01	4.43E+01	2.40E-01	2.00E+02	3.42E+00	2.30E-01
7.1	4.10E+01	3.10E-01	7.50E-01	8.00E-02	4.80E+01	9.70E-01	5.91E+02	7.02E+00	3.70E-01
7.0	4.30E+01	6.20E-01	2.30E+00	4.00E-02	4.30E+01	3.80E-01	2.60E+02	1.30E+01	5.50E-01

Notes

• All distributions from AMAX data represent a uniform distribution between the 95th percentile and maximum observed value at the referenced pH for AMAX piles with 0.64% S. Data for pH values above 7.5 are used for Flotation Tailings as discussed in Section 10.4 (not for Category 1 waste rock).

• Whistle Mine indicates the concentrations observed from the Whistle Mine in Ontario, Canada for acidic & nonacidic waters as presented in Attachment A.

• Vangorda Mine indicates the concentrations observed from the Anvil Range Mine Complex in Yukon, Canada for acidic waters as presented in Attachment A.

• Dunka Seep indicates the highest observed concentration or detection limit from the available Dunka Mine data (a single sampling event in May 2006 at Seep X) multiplied by a factor of 10.

• NorthMet Lab Data indicates a range between the highest observed concentration in the NorthMet tailings humidity cells and an estimated field-scale value developed by MDNR.

• Concentration caps for all constituents not shown are calculated from the equations shown in Section 8.3.1.

Large Table 13 Distribution Parameters for Duluth Complex Category 2/3, 4, and Ore Concentration Caps (nonacidic)

Cap Value From Various Data Sources

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*Std. Deviation*	*Min.*	*Max.*
Ag	Limit	Dunka Seep	mg/L	Constant	2.00E-04	--	--	--
As	Limit	Whistle Mine	mg/L	Constant	1.00E-01	--	--	--
B	Limit	Whistle Mine	mg/L	Constant	1.00E-01	--	--	--
Be	Limit	Dunka Seep	mg/L	Constant	4.00E-04	--	--	--
Cr	Limit	Whistle Mine	mg/L	Constant	1.00E-02	--	--	--
Pb	Limit	Whistle Mine	mg/L	Constant	1.00E-01	--	--	--
Sb	Limit	NorthMet Lab Data	mg/L	Uniform	--	--	8.30E-03	1.00E-01
Tl	Limit	Dunka Seep	mg/L	Constant	2.00E-04	--	--	--
V	Limit	Whistle Mine	mg/L	Constant	1.00E-02	--	--	--

pH-based Range from AMAX Data

(95th percentile values, all units mg/L)

pH	*Alkalinity*	*Co*	*Cu*	*Fe*	*K*	*Mn*	*Na*	*Ni*	*Zn*
7.5	4.79E+01	2.48E-01	1.30E-01	7.45E-02	4.60E+01	1.40E+00	4.68E+02	1.50E+00	1.00E-01
7.4	4.90E+01	2.04E-01	1.47E-01	5.90E-02	4.21E+01	1.49E+00	3.94E+02	1.58E+00	9.73E-02
7.3	3.59E+01	9.30E-02	2.00E-01	5.00E-02	5.04E+01	2.00E-01	2.31E+02	1.62E+00	1.33E-01
7.2	3.53E+01	1.89E-01	2.33E-01	1.68E-01	4.25E+01	1.72E+00	3.47E+02	3.21E+00	1.82E-01
7.1	3.45E+01	2.31E-01	2.84E-01	8.00E-02	4.60E+01	6.46E-01	1.85E+02	4.31E+00	2.91E-01
7.0	2.60E+01	5.08E-01	5.59E-01	5.00E-02	3.96E+01	2.48E+00	2.41E+02	7.40E+00	4.09E-01
6.9	2.80E+01	1.02E+00	3.70E+00	1.78E-01	4.18E+01	1.90E+00	1.82E+02	1.98E+01	7.30E-01
6.8	2.16E+01	1.45E+00	5.02E+00	7.00E-02	5.06E+01	1.13E+00	1.50E+02	2.98E+01	1.24E+00
6.7	2.18E+01	1.24E+00	4.30E+00	1.02E-01	4.80E+01	3.61E+00	1.69E+02	2.06E+01	8.78E-01
6.6	1.44E+01	1.05E+00	5.44E+00	1.26E-01	5.07E+01	2.91E+00	2.05E+02	2.46E+01	8.66E-01
6.5	1.60E+01	1.52E+00	6.50E+00	6.00E-02	4.65E+01	1.39E+00	1.42E+02	3.15E+01	1.26E+00
6.4	1.53E+01	2.10E+00	7.09E+00	1.86E-01	4.88E+01	3.45E+00	1.78E+02	5.08E+01	1.51E+00
6.3	1.17E+01	2.11E+00	8.85E+00	8.40E-02	5.04E+01	3.03E+00	2.38E+02	4.75E+01	1.29E+00
6.2	6.90E+00	2.56E+00	1.02E+01	4.00E-02	5.37E+01	4.01E+00	4.39E+02	7.00E+01	1.87E+00
6.1	9.90E+00	3.13E+00	1.49E+01	5.85E-02	6.15E+01	3.26E+00	1.27E+02	8.35E+01	2.33E+00
6.0	9.40E+00	1.42E+00	8.56E+00	3.00E-02	4.97E+01	3.40E+00	1.64E+02	3.02E+01	1.60E+00

pH-based Range from AMAX Data

(maximum values, all units mg/L)

pH	*Alkalinity*	*Co*	*Cu*	*Fe*	*K*	*Mn*	*Na*	*Ni*	*Zn*
7.5	5.27E+01	2.80E-01	1.70E-01	1.50E-01	6.00E+01	1.68E+00	7.00E+02	1.70E+00	1.74E-01
7.4	5.40E+01	2.16E+00	1.80E-01	7.00E-02	5.32E+01	2.40E+00	4.91E+02	2.15E+01	3.96E-01
7.3	3.60E+01	1.20E-01	2.60E-01	6.00E-02	5.90E+01	3.00E-01	2.60E+02	2.29E+00	2.30E-01
7.2	4.50E+01	8.10E-01	3.40E-01	7.00E-01	4.43E+01	2.14E+00	8.62E+02	6.70E+00	2.30E-01
7.1	4.10E+01	3.10E-01	7.50E-01	1.20E-01	4.80E+01	1.64E+00	1.11E+03	7.02E+00	3.70E-01
7.0	4.30E+01	1.24E+00	2.30E+00	6.00E-02	4.30E+01	3.05E+00	2.69E+02	1.30E+01	5.50E-01
6.9	5.03E+01	1.71E+00	6.24E+00	3.00E-01	5.52E+01	2.28E+00	2.13E+02	4.50E+01	1.15E+00
6.8	3.30E+01	2.41E+00	7.25E+00	1.20E-01	5.80E+01	1.74E+00	3.13E+02	4.40E+01	1.65E+00
6.7	3.30E+01	1.41E+00	5.01E+00	1.30E-01	4.84E+01	5.57E+00	3.30E+02	4.10E+01	1.17E+00
6.6	3.90E+01	3.22E+00	1.10E+01	1.02E+00	8.40E+01	3.23E+00	2.40E+02	8.00E+01	2.25E+00
6.5	2.10E+01	1.87E+00	6.95E+00	6.00E-02	5.60E+01	1.89E+00	3.04E+02	4.30E+01	1.53E+00
6.4	2.20E+01	3.24E+00	7.57E+00	3.90E-01	5.10E+01	4.07E+00	2.70E+02	7.95E+01	1.69E+00
6.3	1.36E+01	2.30E+00	1.70E+01	1.00E-01	5.20E+01	3.32E+00	2.49E+02	6.70E+01	1.56E+00
6.2	6.90E+00	3.65E+00	1.20E+01	4.00E-02	5.40E+01	4.10E+00	6.09E+02	9.10E+01	2.01E+00
6.1	9.90E+00	3.34E+00	1.70E+01	6.00E-02	6.35E+01	3.36E+00	1.30E+02	9.10E+01	2.58E+00
6.0	1.11E+01	1.60E+00	1.10E+01	3.00E-02	5.20E+01	3.40E+00	2.01E+02	3.20E+01	1.61E+00

Notes

- All distributions from AMAX data represent a uniform distribution between the 95th percentile and maximum observed value at the referenced pH for all AMAX piles (0.64% S to 1.4% S).
- Whistle Mine indicates the concentrations observed from the Whistle Mine in Ontario, Canada for acidic & nonacidic waters as presented in Attachment A.
- Vangorda Mine indicates the concentrations observed from the Anvil Range Mine Complex in Yukon, Canada for acidic waters as presented in Attachment A.
- Dunka Seep indicates the highest observed concentration or detection limit from the available Dunka Mine data (a single sampling event in May 2006 at Seep X) multiplied by a factor of 10.
- NorthMet Lab Data indicates a range between the highest observed concentration in the NorthMet tailings humidity cells and an estimated field-scale value developed by MDNR.
- Concentration caps for all constituents not shown are calculated from the equations shown in Section 8.3.1.

Large Table 14 Distribution Parameters for Duluth Complex Category 2/3, 4, and Ore Concentration Caps (acidic)

Distribution Fit to AMAX, Whistle, and Vangorda Mine Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Ag	Cap	Whistle Mine	mg/L	Beta	4.20E-02	4.62E-03	3.40E-02	5.00E-02
Al	Cap	Whistle Mine	mg/L	Beta	4.33E+02	2.68E+02	1.13E+02	1.00E+03
As	Cap	Whistle Mine	mg/L	Constant	1.00E-01	--	--	--
B	Cap	Whistle Mine	mg/L	Beta	2.19E-01	9.45E-02	9.23E-02	5.00E-01
Be	Cap	Whistle Mine	mg/L	Beta	1.62E-02	4.31E-03	5.26E-03	2.21E-02
Ca	Cap	Whistle Mine	mg/L	Beta	4.09E+02	4.15E+01	2.62E+02	5.54E+02
Cd	Cap	Whistle Mine	mg/L	Beta	1.47E-01	8.84E-02	5.35E-02	4.51E-01
Co	Cap	Whistle Mine	mg/L	Beta	3.04E+01	1.01E+01	8.68E+00	4.14E+01
Cr	Cap	Whistle Mine	mg/L	Beta	1.60E-02	5.77E-04	1.50E-02	1.70E-02
Cu	Cap	AMAX pH 3-4	mg/L	Beta	1.49E+02	1.30E+01	9.79E+01	1.79E+02
Fe	Cap	Whistle Mine	mg/L	Beta	9.57E+01	5.56E+01	1.61E+00	4.32E+02
K	Cap	Whistle Mine	mg/L	Beta	2.92E+01	9.52E+00	9.39E+00	1.53E+02
Mg	Cap	Whistle Mine	mg/L	Beta	9.92E+02	3.92E+02	4.82E+02	2.11E+03
Mn	Cap	Whistle Mine	mg/L	Beta	5.48E+01	2.32E+01	1.75E+01	1.03E+02
Na	Cap	Whistle Mine	mg/L	Beta	8.75E+01	6.32E+01	2.48E+01	7.17E+02
Ni	Cap	Whistle Mine	mg/L	Beta	6.41E+02	1.90E+02	9.97E+01	8.41E+02
Pb	Cap	Whistle Mine	mg/L	Beta	3.64E-01	1.36E-01	1.28E-01	6.00E-01
Sb	Cap	Whistle Mine	mg/L	Beta	2.00E+00	5.77E-01	1.00E+00	3.00E+00
Se	Cap	Whistle Mine	mg/L	Constant	1.00E-01	--	--	--
SO4	Cap	Whistle Mine	mg/L	Beta	9.52E+03	3.39E+03	3.29E+03	1.81E+04
Tl	Cap	Vangorda Mine	mg/L	Beta	4.47E-02	1.22E-01	2.00E-03	2.18E+00
V	Cap	Whistle Mine	mg/L	Beta	5.50E-02	2.89E-03	5.00E-02	6.00E-02
Zn	Cap	Whistle Mine	mg/L	Beta	1.54E+01	1.27E+01	6.34E+00	6.00E+01

Notes

- All distributions from Whistle and Vangorda Mine data represent the full range of the observed values.
- All distributions from AMAX data represent the full range of the highest 5% of observed values in each 0.1 pH increment over the indicated pH range.
- Concentration caps for all constituents not shown are calculated from the equations shown in Section 8.3.1.
- Distributions shown as constant indicate zero detections in the referenced data set, the detection limit is set as the concentration cap.
- Beta distributions are shown in Large Figures 32-36.

Large Table 15 Distribution Parameters for Virginia Formation Category 4 Waste Rock Concentration Caps

Distribution Fit to AMAX, Whistle, and Vangorda Mine Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Ag	Cap	Vangorda Mine	mg/L	Beta	5.86E-02	7.07E-02	6.24E-03	8.65E-01
Al	Cap	Whistle Mine	mg/L	Beta	4.33E+02	2.68E+02	1.13E+02	1.00E+03
As	Cap	Vangorda Mine	mg/L	Beta	4.21E-01	6.39E-01	1.13E-02	2.50E+00
B	Cap	Vangorda Mine	mg/L	Beta	1.39E+00	9.80E-01	1.30E-02	3.27E+00
Ba	Cap	Vangorda Mine	mg/L	Beta	2.61E-01	3.60E-01	4.96E-03	1.92E+00
Be	Cap	Vangorda Mine	mg/L	Beta	4.59E-02	6.82E-02	5.24E-03	3.20E-01
Ca	Cap	Vangorda Mine	mg/L	Beta	4.09E+02	4.85E+01	3.28E+02	4.98E+02
Cd	Cap	Whistle Mine	mg/L	Beta	1.47E-01	8.84E-02	5.35E-02	4.51E-01
Co	Cap	Vangorda Mine	mg/L	Beta	1.53E+01	6.86E+00	6.98E+00	3.08E+01
Cr	Cap	Vangorda Mine	mg/L	Beta	9.19E-02	1.52E-01	9.60E-03	8.70E-01
Cu	Cap	Vangorda Mine	mg/L	Beta	1.37E-01	1.03E-01	3.06E-02	6.08E-01
Fe	Cap	Vangorda Mine	mg/L	Beta	8.60E+02	1.23E+03	6.00E+00	5.08E+03
K	Cap	Vangorda Mine	mg/L	Beta	1.26E+01	8.42E+00	6.00E-01	3.00E+01
Mg	Cap	Vangorda Mine	mg/L	Beta	2.03E+03	1.48E+03	5.75E+02	6.20E+03
Mn	Cap	Vangorda Mine	mg/L	Beta	1.55E+03	1.13E+03	3.30E+02	4.32E+03
Na	Cap	Vangorda Mine	mg/L	Beta	1.67E+01	1.01E+01	7.39E+00	1.22E+02
Ni	Cap	Vangorda Mine	mg/L	Beta	1.08E+01	5.45E+00	4.17E+00	2.33E+01
Pb	Cap	Whistle Mine	mg/L	Beta	3.64E-01	1.36E-01	1.28E-01	6.00E-01
Sb	Cap	Vangorda Mine	mg/L	Beta	3.25E+00	2.78E+00	1.00E-03	1.60E+01
Se	Cap	Vangorda Mine	mg/L	Beta	4.34E-01	6.24E-01	7.33E-02	3.20E+00
SO4	Cap	Vangorda Mine	mg/L	Beta	2.23E+04	2.21E+04	3.54E+03	1.00E+05
Tl	Cap	Vangorda Mine	mg/L	Beta	4.47E-02	1.22E-01	2.00E-03	2.18E+00
V	Cap	Vangorda Mine	mg/L	Beta	6.00E-02	1.11E-01	3.00E-03	5.15E-01
Zn	Cap	Whistle Mine	mg/L	Beta	1.54E+01	1.27E+01	6.34E+00	6.00E+01

Notes

- All distributions from Whistle and Vangorda Mine data represent the full range of the observed values.
- Concentration caps for all constituents not shown are calculated from the equations shown in Section 8.3.1.
- Beta distributions are shown in Large Figures 37-41.

Large Table 16 Distribution Parameters for Flotation Fine Tailings Release

Distribution Fit to Humidity Cell Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Ca	SO_4 rate ratio	HCT	mg Ca / mg SO_4	Beta	1.18E+00	3.03E-01	8.17E-01	3.45E+00
K	SO_4 rate ratio	HCT	mg K / mg SO_4	Beta	2.63E-01	6.37E-02	1.71E-01	7.51E-01
Mg	SO_4 rate ratio	HCT	mg Mg / mg SO_4	Beta	2.18E-01	4.69E-02	1.62E-01	7.94E-01
Mn	Ni rate ratio	HCT	mg Mn / mg Ni	Beta	4.68E+00	2.25E+00	2.07E+00	9.31E+00
Na	SO_4 rate ratio	HCT	mg Na / mg SO_4	Beta	8.20E-02	1.77E-02	6.03E-02	2.64E-01
Se	SO_4 rate ratio	HCT	mg Se / mg SO_4	Beta	1.79E-05	5.29E-06	1.29E-05	6.09E-05
SO_4	Rate	HCT*	mg SO_4/kg/week	Beta	1.88E+01	2.87E+00	2.66E+00	2.32E+01

Distribution Fit to Aqua Regia Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Ag	S ratio	Aqua Regia	mg Ag / mg S	Beta	1.54E-04	1.49E-05	1.35E-04	2.54E-04
As	S ratio	Aqua Regia	mg As / mg S	Beta	1.96E-03	2.53E-04	1.67E-03	4.89E-03
Ba	K ratio	Aqua Regia	mg Ba / mg K	Beta	2.66E-02	1.27E-03	1.83E-02	3.06E-02
Be	K ratio	Aqua Regia	mg Be / mg K	Beta	1.03E-04	1.51E-05	8.13E-05	2.32E-04
Cu	S ratio	Aqua Regia	mg Cu / mg S	Beta	9.30E-02	1.46E-02	5.29E-02	1.46E-01
Pb	S ratio	Aqua Regia	mg Pb / mg S	Beta	2.67E-03	6.16E-04	1.93E-03	9.32E-03
Sb	S ratio	Aqua Regia	mg Sb / mg S	Beta	1.08E-04	3.50E-05	6.67E-05	1.99E-04
Tl	S ratio	Aqua Regia	mg Tl / mg S	Beta	7.15E-05	7.35E-06	5.97E-05	1.41E-04
V	K ratio	Aqua Regia	mg V / mg K	Beta	2.53E-02	2.61E-03	7.01E-03	3.17E-02

Distribution Fit to Waste Rock Humidity Cell Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Cd	Zn rate ratio	2/3 HCT (2)	mg Cd / mg Zn	Beta	1.65E-02	1.20E-02	1.01E-03	5.84E-02
Co	Ni rate ratio	2/3 HCT (2)	mg Co / mg Ni	Beta	8.29E-02	3.91E-02	2.24E-02	2.06E-01
Zn	Ni rate ratio	2/3 HCT (2)	mg Zn / mg Ni	Beta	3.35E-01	3.71E-01	3.31E-02	1.60E+00

Distribution Fit to Microprobe Data or Mineral Formula

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Al	Ca ratio	Anorthite Formula	mg Al / mg Ca	Constant	1.35E+00	--	--	--
	Na ratio	Albite Formula	mg Al / mg Na	Constant	1.17E+00	--	--	--
Fe	S ratio	Pyrrhotite microprobe	mg Fe / mg S	Beta	1.62E+00	8.72E-02	1.49E+00	1.92E+00
	Mg ratio	Olivine microprobe	mg Fe / mg Mg	Beta	1.87E+00	6.75E-01	1.19E+00	4.51E+00
Ni	S ratio	Pyrrhotite microprobe	mg Ni / mg S	Beta	5.63E-03	6.65E-03	5.65E-04	4.00E-02

Distribution From Defined Concentration Cap

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Cl	No release	N/A	mg/L	Constant	0	--	--	--
B	Cap	Whistle Mine	mg/L	Constant	1.00E-01	--	--	--
Cr	Cap	Whistle Mine	mg/L	Constant	1.00E-02	--	--	--

Notes

- HCT indicates average rates from tailings humidity cells over the entire testing period. Data used through April 2011.
- For sulfate, the release rate is the estimated release rate at the initial sulfur content for each humidity cell.
- Aqua Regia indicates ratios from whole tailings testing.
- Cat 2/3 HCT (2) indicates average rates from Category 2/3 humidity cells over Condition 2, as defined in Large Table 1.
- All distributions from humidity cell data and aqua regia data represent the full range of the observed values, with no weighting. Distributions are shown in Large Figure 42 to Large Figure 45.
- Distributions from microprobe data represent the full range of the observed ratios for each mineral, with no weighting. Distributions are shown in Large Figure 21 and Large Figure 22.
- Constituents not shown above are modeled according to the mineral solubility methods described in Section 10.1.1.

Large Table 17 Distribution Parameters for Flotation Coarse Tailings Release

Distribution Fit to Humidity Cell Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Ca	SO_4 rate ratio	HCT	mg Ca / mg SO_4	Beta	9.58E-01	3.34E-01	3.00E-01	1.60E+00
K	SO_4 rate ratio	HCT	mg K / mg SO_4	Beta	2.60E-01	8.16E-02	0.00E+00	4.91E-01
Mg	SO_4 rate ratio	HCT	mg Mg / mg SO_4	Beta	1.82E-01	3.32E-02	9.68E-02	5.46E-01
Mn	Ni rate ratio	HCT	mg Mn / mg Ni	Beta	3.37E+00	1.32E+00	1.80E+00	1.00E+01
Na	SO_4 rate ratio	HCT	mg Na / mg SO_4	Beta	6.86E-02	2.40E-02	3.58E-02	2.57E-01
Se	SO_4 rate ratio	HCT	mg Se / mg SO_4	Beta	1.75E-05	3.51E-06	0.00E+00	2.41E-05
SO_4	Rate	HCT	mg SO_4/kg/week	Beta	1.19E+01	2.55E+00	4.37E+00	2.13E+01

Distribution Fit to Aqua Regia Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Ag	S ratio	Aqua Regia	mg Ag / mg S	Beta	2.05E-04	3.41E-05	1.42E-04	5.45E-04
As	S ratio	Aqua Regia	mg As / mg S	Beta	1.82E-03	3.31E-04	9.17E-04	5.09E-03
Ba	K ratio	Aqua Regia	mg Ba / mg K	Beta	2.74E-02	1.81E-03	2.01E-02	4.02E-02
Be	K ratio	Aqua Regia	mg Be / mg K	Beta	9.77E-05	9.41E-06	5.71E-05	1.53E-04
Cu	S ratio	Aqua Regia	mg Cu / mg S	Beta	2.11E-01	5.25E-02	2.95E-03	7.00E-01
Pb	S ratio	Aqua Regia	mg Pb / mg S	Beta	2.88E-03	7.68E-04	1.18E-03	1.08E-02
Sb	S ratio	Aqua Regia	mg Sb / mg S	Beta	1.10E-04	3.06E-05	5.45E-05	2.50E-04
Tl	S ratio	Aqua Regia	mg Tl / mg S	Beta	9.44E-05	1.27E-05	6.67E-05	1.86E-04
V	K ratio	Aqua Regia	mg V / mg K	Beta	1.81E-02	2.66E-03	1.81E-03	3.00E-02

Distribution Fit to Waste Rock Humidity Cell Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Cd	Zn rate ratio	2/3 HCT (2)	mg Cd / mg Zn	Beta	1.65E-02	1.20E-02	1.01E-03	5.84E-02
Co	Ni rate ratio	2/3 HCT (2)	mg Co / mg Ni	Beta	8.29E-02	3.91E-02	2.24E-02	2.06E-01
Zn	Ni rate ratio	2/3 HCT (2)	mg Zn / mg Ni	Beta	3.35E-01	3.71E-01	3.31E-02	1.60E+00

Distribution Fit to Microprobe Data or Mineral Formula

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Al	Ca ratio	Anorthite Formula	mg Al / mg Ca	Constant	1.35E+00	--	--	--
	Na ratio	Albite Formula	mg Al / mg Na	Constant	1.17E+00	--	--	--
Fe	S ratio	Pyrrhotite microprobe	mg Fe / mg S	Beta	1.62E+00	8.72E-02	1.49E+00	1.92E+00
	Mg ratio	Olivine microprobe	mg Fe / mg Mg	Beta	1.87E+00	6.75E-01	1.19E+00	4.51E+00
Ni	S ratio	Pyrrhotite microprobe	mg Ni / mg S	Beta	5.63E-03	6.65E-03	5.65E-04	4.00E-02

Distribution From Defined Concentration Cap

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Cl	No release	N/A	mg/L	Constant	0	--	--	--
B	Cap	Whistle Mine	mg/L	Constant	1.00E-01	--	--	--
Cr	Cap	Whistle Mine	mg/L	Constant	1.00E-02	--	--	--

Notes

- HCT indicates average rates from tailings humidity cells over the entire testing period. Data used through April 2011.
- For sulfate, the release rate is the estimated release rate at the initial sulfur content for each humidity cell.
- Aqua Regia indicates ratios from whole tailings testing.
- Cat 2/3 HCT (2) indicates average rates from Category 2/3 humidity cells over Condition 2, as defined in Large Table 1.
- All distributions from humidity cell data and aqua regia data represent the full range of the observed values, with no weighting. Distributions are shown in Large Figure 46 to Large Figure 49.
- Distributions from microprobe data represent the full range of the observed ratios for each mineral, with no weighting. Distributions are shown in Large Figure 21 and Large Figure 22.
- Constituents not shown above are modeled according to the mineral solubility methods described in Section 10.1.1.

Large Table 18 Distribution Parameters for LTVSMC Tailings Release

Distribution Fit to Humidity Cell Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Se	SO_4 rate ratio	HCT	mg Se / mg SO_4	Beta	7.22E-05	4.63E-05	3.04E-05	3.04E-04
SO_4	Rate	HCT	mg SO_4/kg/week	Beta	1.87E+00	5.02E-01	8.13E-01	2.54E+00
Zn	SO_4 rate ratio	HCT	mg Zn / mg SO_4	Beta	5.32E-05	9.20E-06	4.28E-05	8.33E-05

Distribution Fit to Aqua Regia Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Ag	S ratio	Aqua Regia	mg Ag / mg S	Beta	1.85E-04	1.51E-04	3.47E-05	1.99E-03
As	S ratio	Aqua Regia	mg As / mg S	Beta	1.11E-01	5.43E-02	2.85E-02	8.75E-01
Cd	S ratio	Aqua Regia	mg Cd / mg S	Beta	7.69E-05	6.83E-05	8.21E-06	4.62E-03
Co	S ratio	Aqua Regia	mg Co / mg S	Beta	4.10E-02	3.17E-02	9.94E-03	3.75E-01
Cu	S ratio	Aqua Regia	mg Cu / mg S	Beta	4.26E-02	3.66E-02	7.95E-03	7.00E-01
Ni	S ratio	Aqua Regia	mg Ni / mg S	Beta	1.71E-02	1.10E-02	3.46E-03	1.92E-01
Pb	S ratio	Aqua Regia	mg Pb / mg S	Beta	6.66E-03	3.95E-03	1.12E-03	4.17E-02
Sb	S ratio	Aqua Regia	mg Sb / mg S	Beta	3.44E-04	2.34E-04	8.93E-05	2.92E-03
Tl	S ratio	Aqua Regia	mg Tl / mg S	Beta	9.04E-05	7.48E-05	1.95E-05	8.33E-04

Distribution Fit to Microprobe Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Fe	S ratio	Pyrite microprobe	mg Fe / mg S	Beta	8.85E-01	1.36E-02	8.50E-01	9.06E-01

Distribution Fit to Observed Seepage Data

Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Al	Cap	Well Data	mg/L	Uniform	--	--	5.00E-03	2.50E-02
B	Cap	Well Data	mg/L	Beta	3.39E-01	1.03E-01	2.50E-02	5.65E-01
Be	Cap	Well Data	mg/L	Uniform	--	--	1.00E-04	2.50E-04
Ca	Cap	Well Data	mg/L	Beta	7.86E+01	3.79E+01	1.39E+01	1.77E+02
Cl	Cap	Well Data	mg/L	Beta	2.04E+01	7.74E+00	9.25E-01	2.97E+01
Cr	Cap	Well Data	mg/L	Beta	3.65E-03	4.90E-03	4.46E-04	2.81E-02
K	Cap	Well Data	mg/L	Beta	5.58E+00	3.87E+00	1.54E+00	2.02E+01
Mg	Ca ratio	Well Data	mg Mg / mg Ca	Beta	1.07E+00	4.57E-01	5.80E-01	2.10E+00
Mn	Cap	Well Data	mg/L	Beta	9.34E-01	9.92E-01	4.46E-02	6.54E+00
Na	Cap	Well Data	mg/L	Beta	6.01E+01	2.15E+01	4.18E+01	1.51E+02
V	Cap	Well Data	mg/L	Uniform	--	--	5.00E-04	1.00E-03

Notes

• HCT indicates average rates from tailings humidity cells over the entire testing period. Data used through April 2011.

• Aqua Regia indicates ratios from whole tailings testing.

• Cat 2/3 HCT (2) indicates average rates from Category 2/3 humidity cells over Condition 2, as defined in Large Table 1.

• All distributions from humidity cell data, aqua regia and microprobe data represent the full range of the observed values, with no weighting. Distributions are shown in Large Figure 50 to Large Figure 52.

• All distributions from well data represent the full range of observed values for wells GW-001, GW-006, GW-007, GW-008, and GW-012. Distributions are shown in Large Figure 53 to Large Figure 55.

• Constituents not shown above are modeled according to the mineral solubility methods described in Section 10.1.2.

Large Table 19 Distribution Parameters for LTVSMC Tailings Disturbed Flushing Load

Distribution Fit to Leach Extraction Test Data

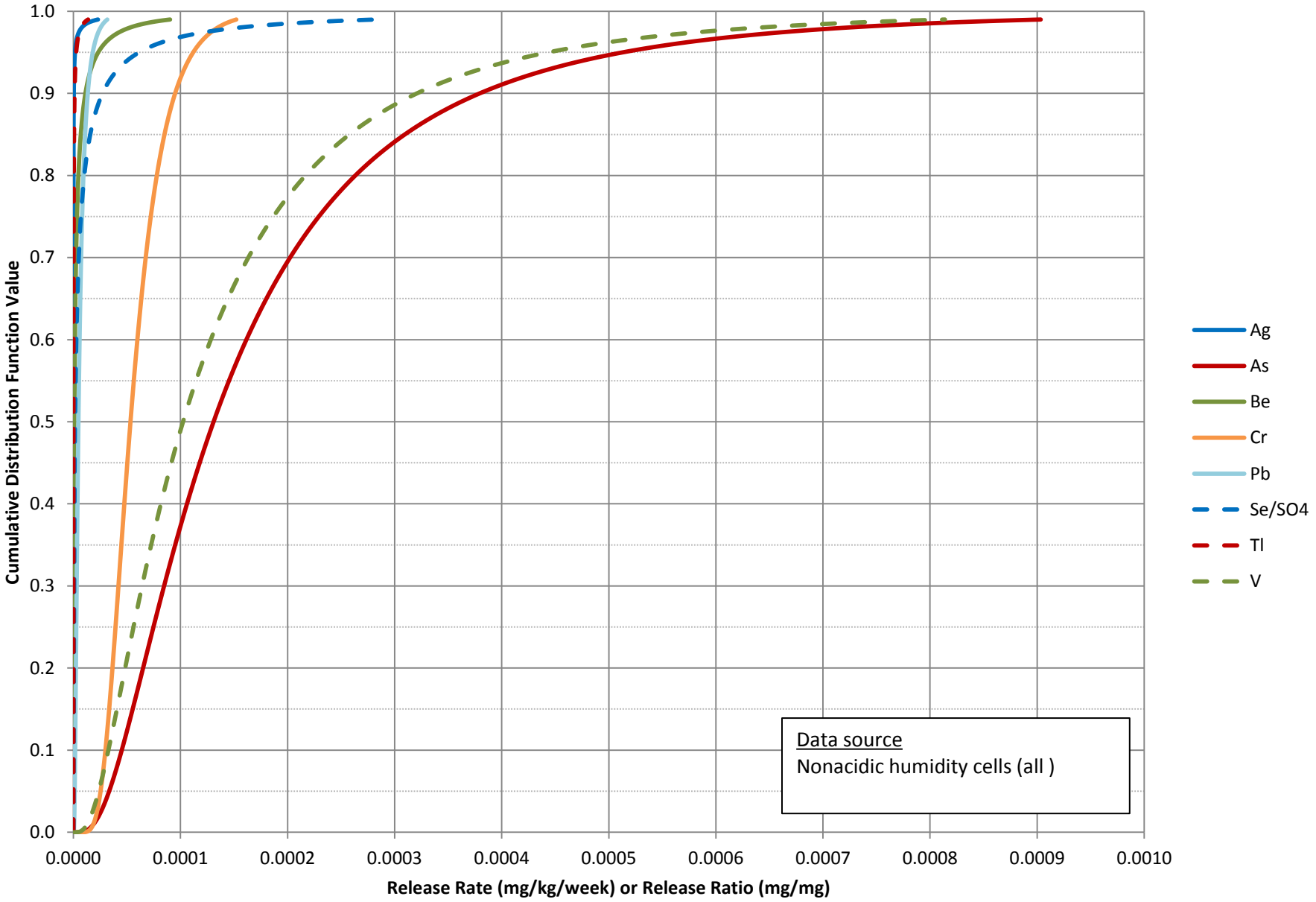
Constituent	*Method*	*Source*	*Units*	*Distribution*	*Mean/Mode*	*St. Dev.*	*Minimum*	*Maximum*
Ag	Load	Leach tests	mg/kg tailings	Beta	2.09E-05	4.85E-06	1.16E-05	3.73E-05
Al	Load	Leach tests	mg/kg tailings	Beta	2.16E-03	1.25E-03	1.26E-04	7.43E-03
Alkalinity	Load	Leach tests	mg/kg tailings	Beta	9.88E+01	2.62E+01	0.00E+00	1.27E+02
As	Load	Leach tests	mg/kg tailings	Beta	2.10E-03	2.96E-03	1.56E-04	2.15E-02
B	Load	Leach tests	mg/kg tailings	Beta	5.51E-02	1.98E-02	3.04E-02	1.90E-01
Ba	Load	Leach tests	mg/kg tailings	Beta	1.86E-03	2.96E-03	5.00E-05	2.00E-02
Be	Load	Leach tests	mg/kg tailings	Beta	7.50E-05	1.44E-05	5.00E-05	1.00E-04
Ca	Load	Leach tests	mg/kg tailings	Beta	1.79E+01	6.21E+00	9.30E+00	4.21E+01
Cd	Load	Leach tests	mg/kg tailings	Beta	1.50E-05	2.89E-06	1.00E-05	2.00E-05
Co	Load	Leach tests	mg/kg tailings	Beta	1.38E-04	1.00E-04	3.95E-05	4.97E-04
Cr	Load	Leach tests	mg/kg tailings	Beta	6.08E-04	6.87E-04	6.56E-05	4.00E-03
Cu	Load	Leach tests	mg/kg tailings	Beta	1.77E-03	1.13E-03	6.61E-04	8.00E-03
F	Load	Leach tests	mg/kg tailings	Beta	2.52E-01	2.08E-01	5.40E-02	1.53E+00
Fe	Load	Leach tests	mg/kg tailings	Beta	1.66E-02	1.20E-02	2.12E-03	4.88E-02
K	Load	Leach tests	mg/kg tailings	Beta	2.02E+00	2.20E+00	4.17E-01	1.00E+01
Mg	Load	Leach tests	mg/kg tailings	Beta	1.64E+01	8.20E+00	1.56E+00	6.28E+01
Mn	Load	Leach tests	mg/kg tailings	Beta	2.43E-02	3.33E-02	4.72E-04	2.51E-01
Na	Load	Leach tests	mg/kg tailings	Beta	3.67E+00	6.70E+00	2.33E-01	4.03E+01
Ni	Load	Leach tests	mg/kg tailings	Beta	5.98E-04	3.61E-04	1.91E-04	1.70E-03
Pb	Load	Leach tests	mg/kg tailings	Beta	3.75E-05	2.82E-05	1.67E-05	2.00E-04
Sb	Load	Leach tests	mg/kg tailings	Beta	7.50E-05	5.52E-05	3.33E-05	3.19E-04
Se	Load	Leach tests	mg/kg tailings	Beta	6.61E-04	6.73E-04	9.70E-05	4.93E-03
SO_4	Load	Leach tests	mg/kg tailings	Beta	2.14E+01	3.09E+01	1.27E+00	1.92E+02
Tl	Load	Leach tests	mg/kg tailings	Beta	7.50E-06	1.44E-06	5.00E-06	1.00E-05
V	Load	Leach tests	mg/kg tailings	Beta	8.01E-05	1.51E-05	3.74E-05	1.02E-04
Zn	Load	Leach tests	mg/kg tailings	Beta	1.08E-03	8.49E-04	4.00E-04	4.00E-03

Notes

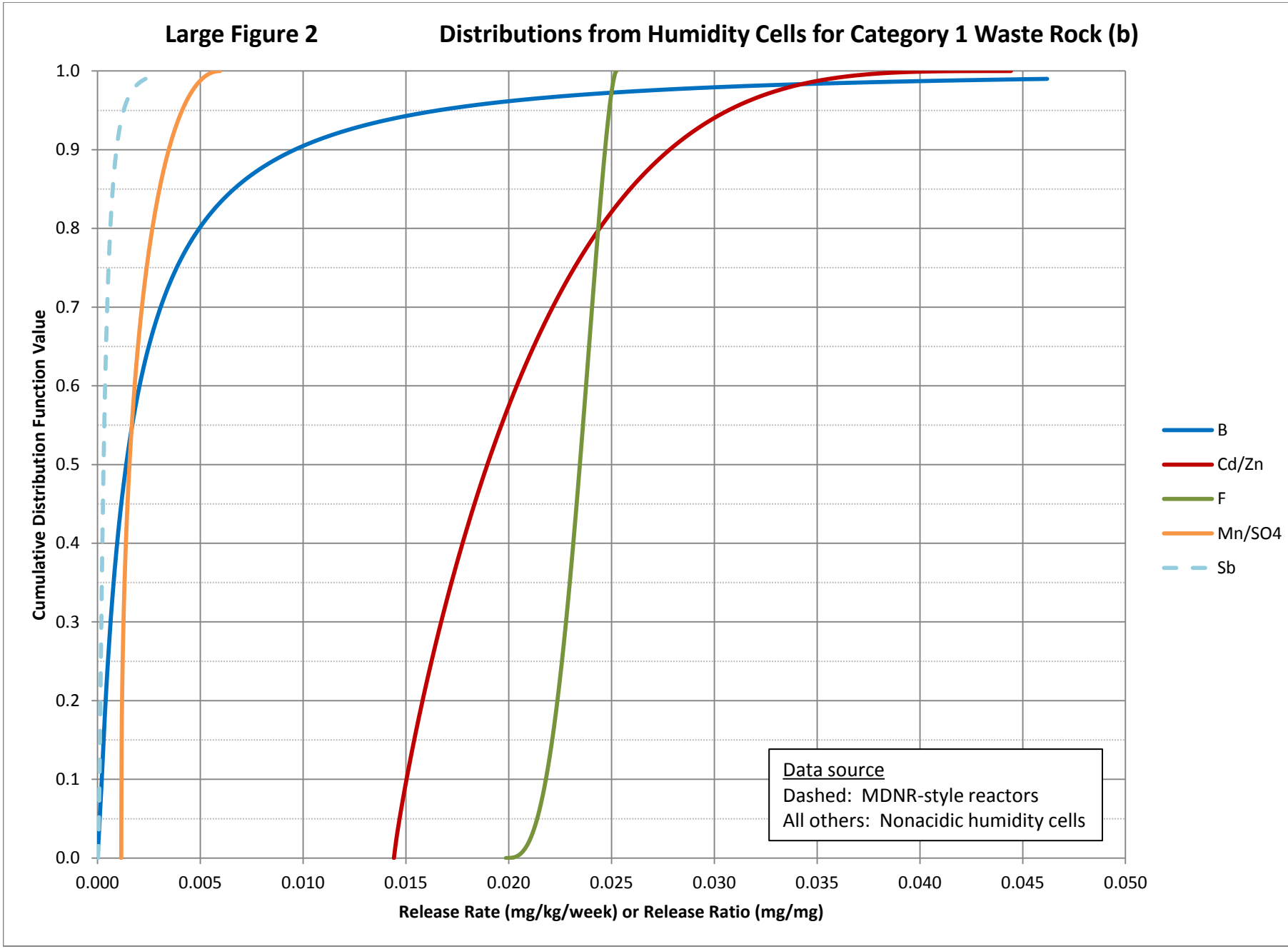
- All distributions from leach extraction testing represent the full range of observed data.
- Distributions for constituents with no detections range from LOD/2 to LOD with a uniform distribution.
- Distributions are shown in Large Figure 56 to Large Figure 60.

Large Figures

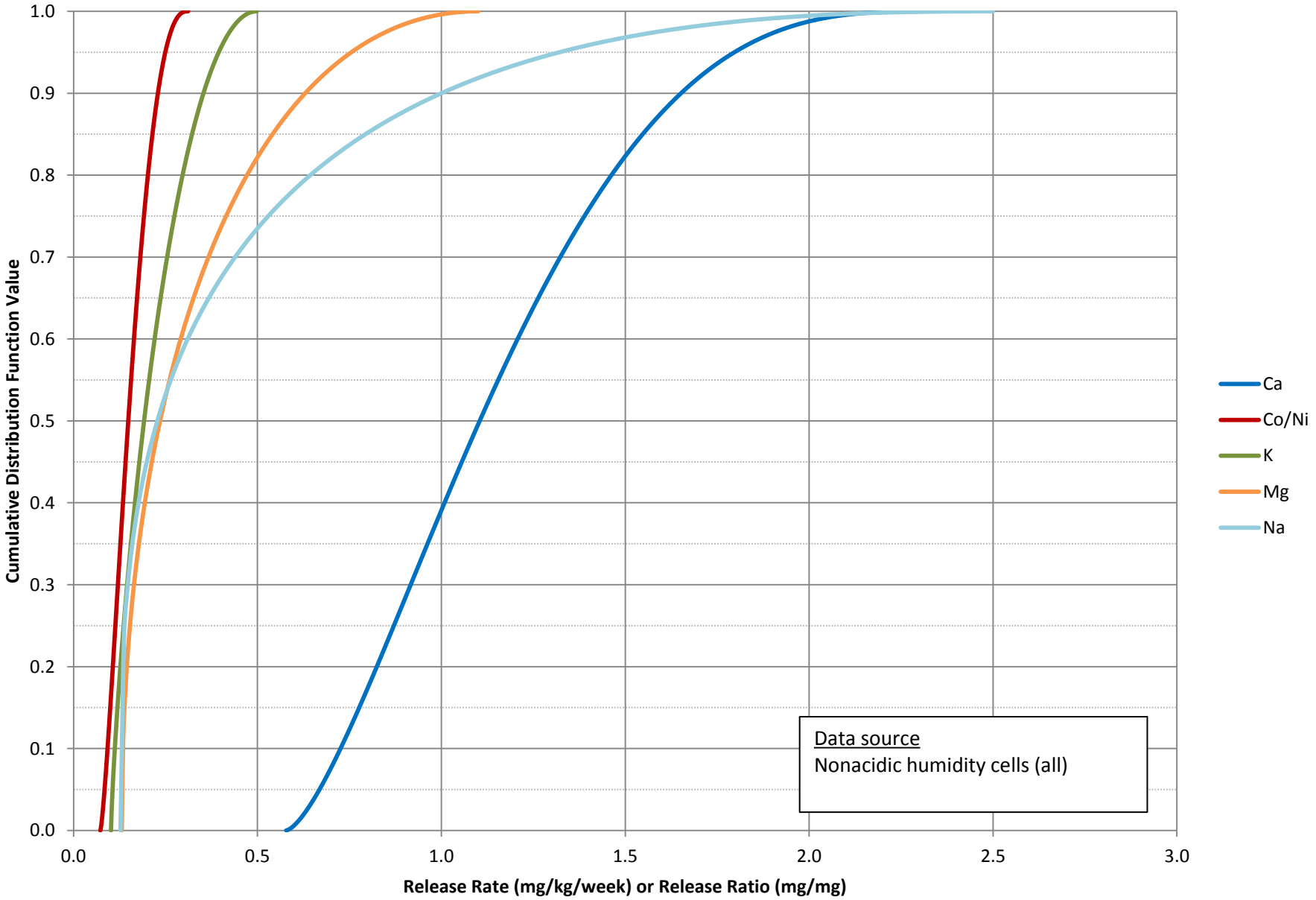
Large Figure 1 Distributions from Humidity Cells for Category 1 Waste Rock (a)



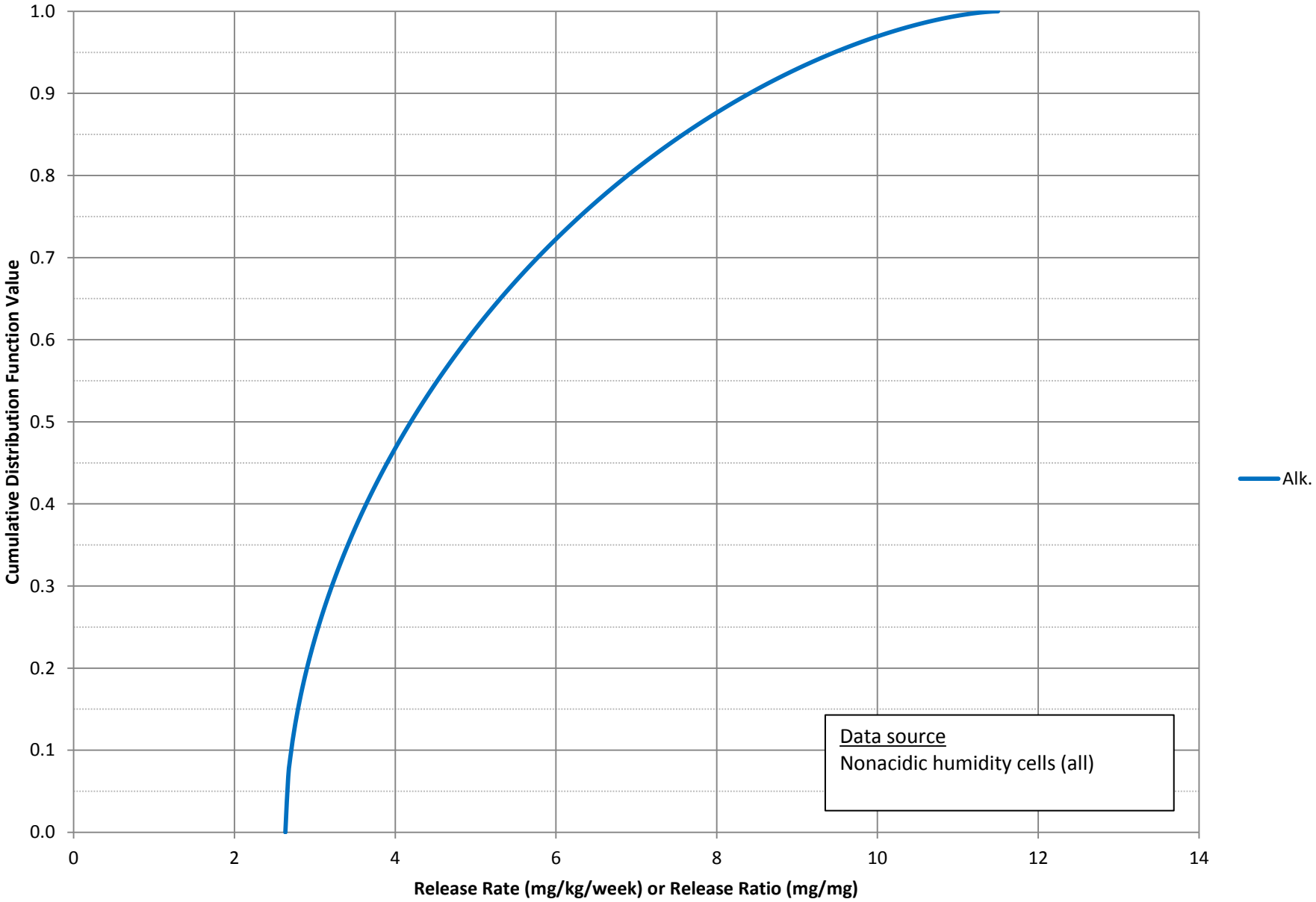
Large Figure 2 **Distributions from Humidity Cells for Category 1 Waste Rock (b)**



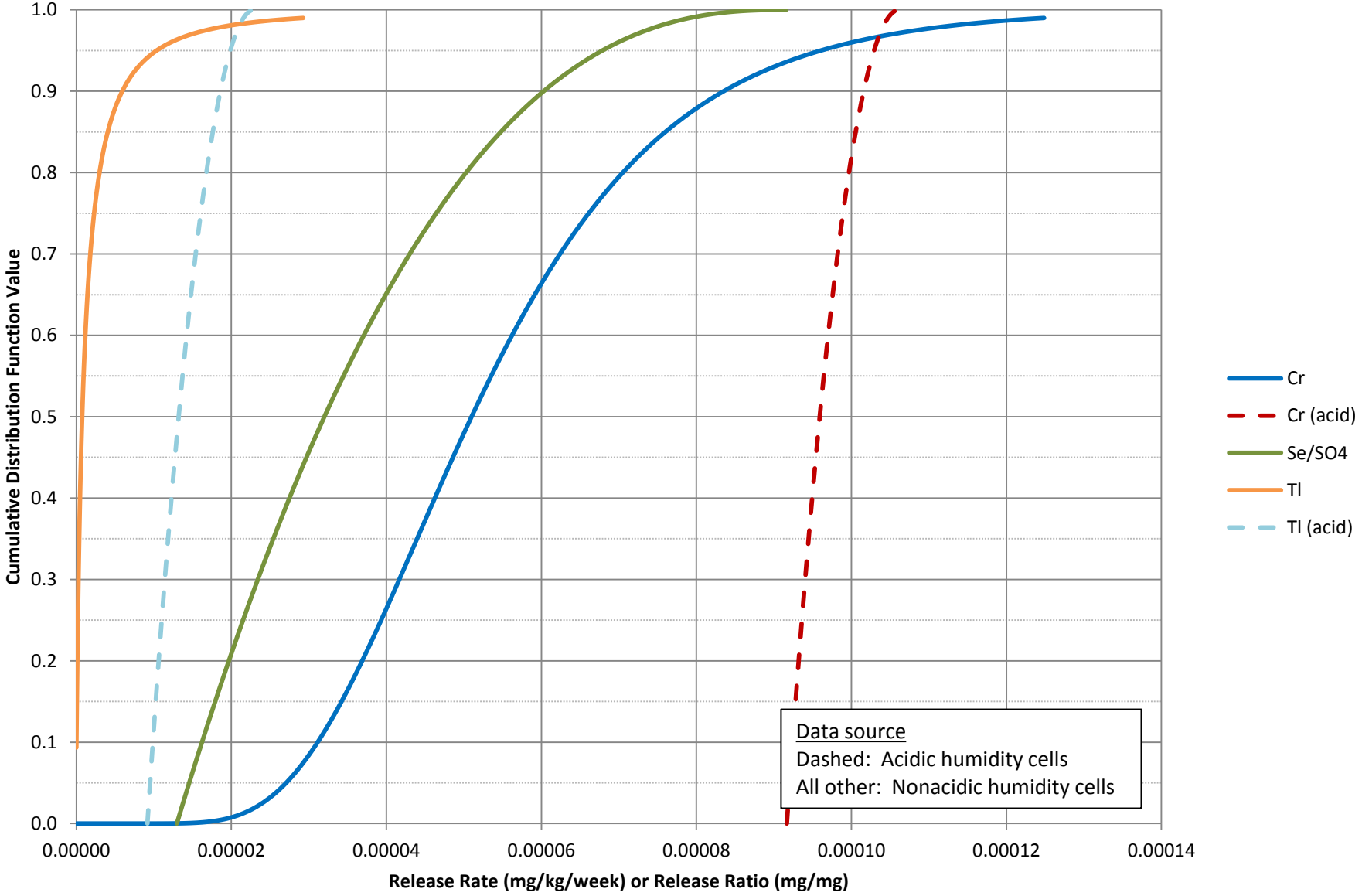
Large Figure 3 Distributions from Humidity Cells for Category 1 Waste Rock (c)



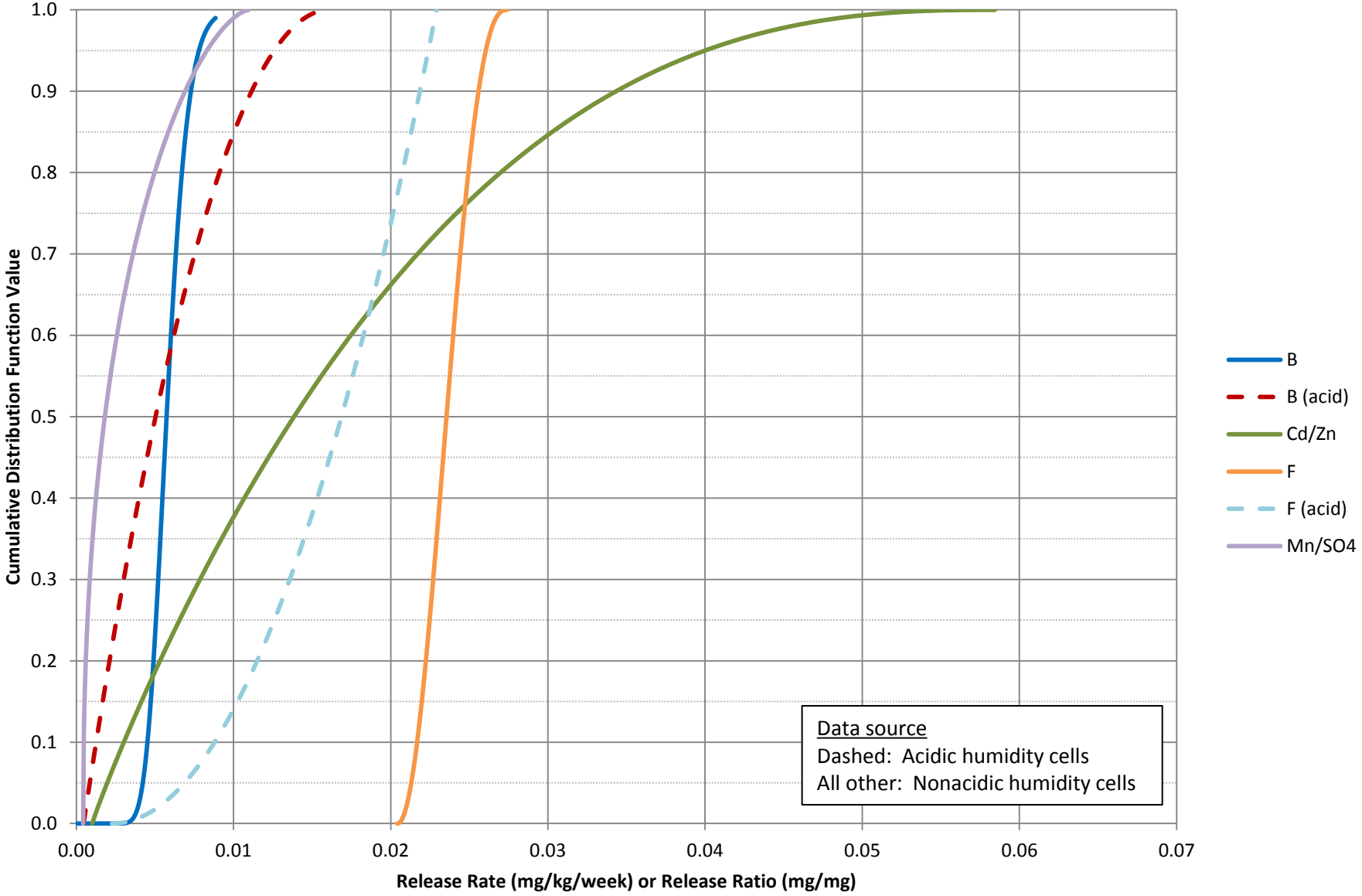
Large Figure 4 Distributions from Humidity Cells for Category 1 Waste Rock (d)



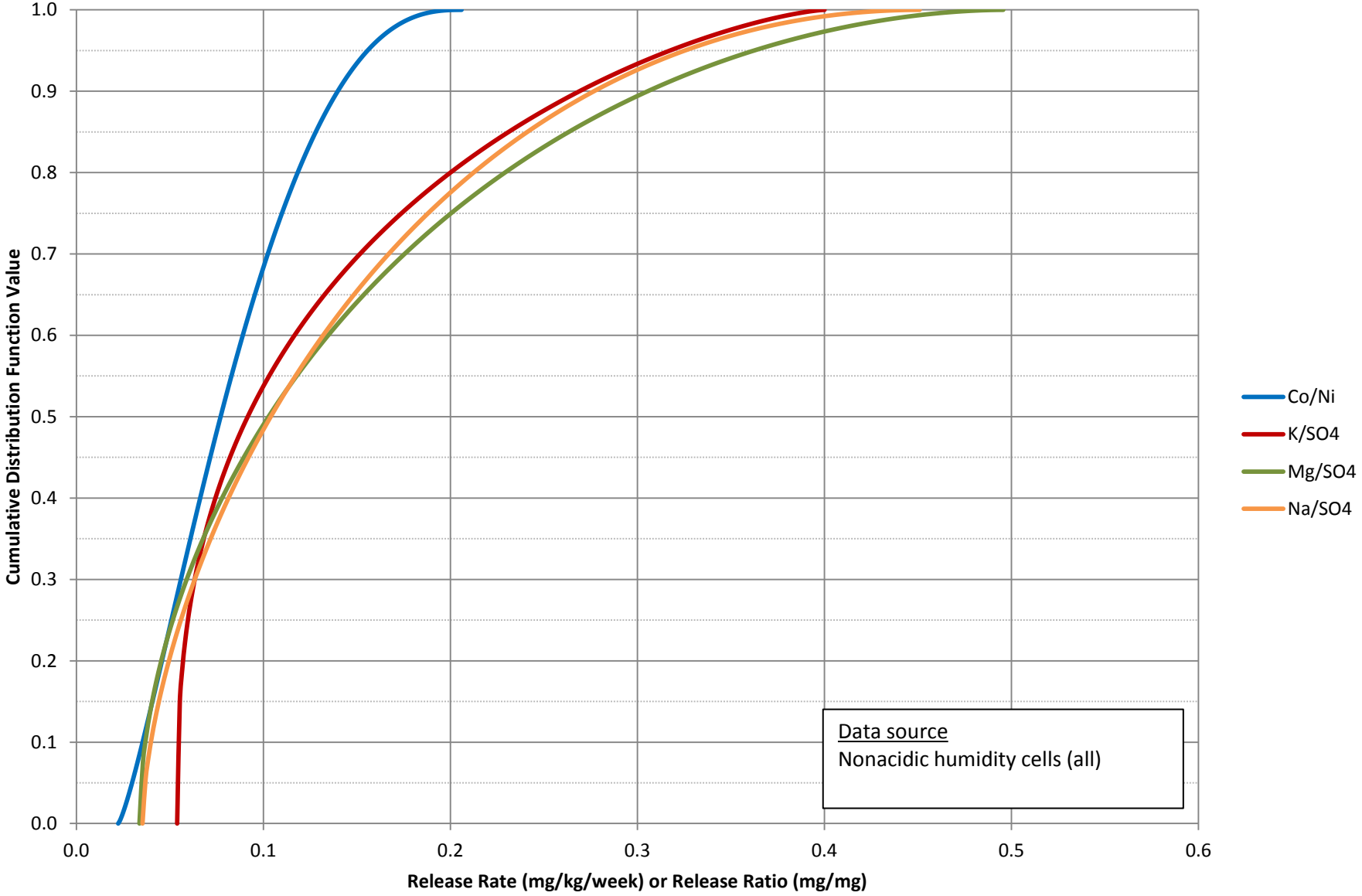
Large Figure 5 **Distributions from Humidity Cells for Category 2/3 Waste Rock and Ore Wall Rock (a)**



Large Figure 6 **Distributions from Humidity Cells for Category 2/3 Waste Rock and Ore Wall Rock (b)**

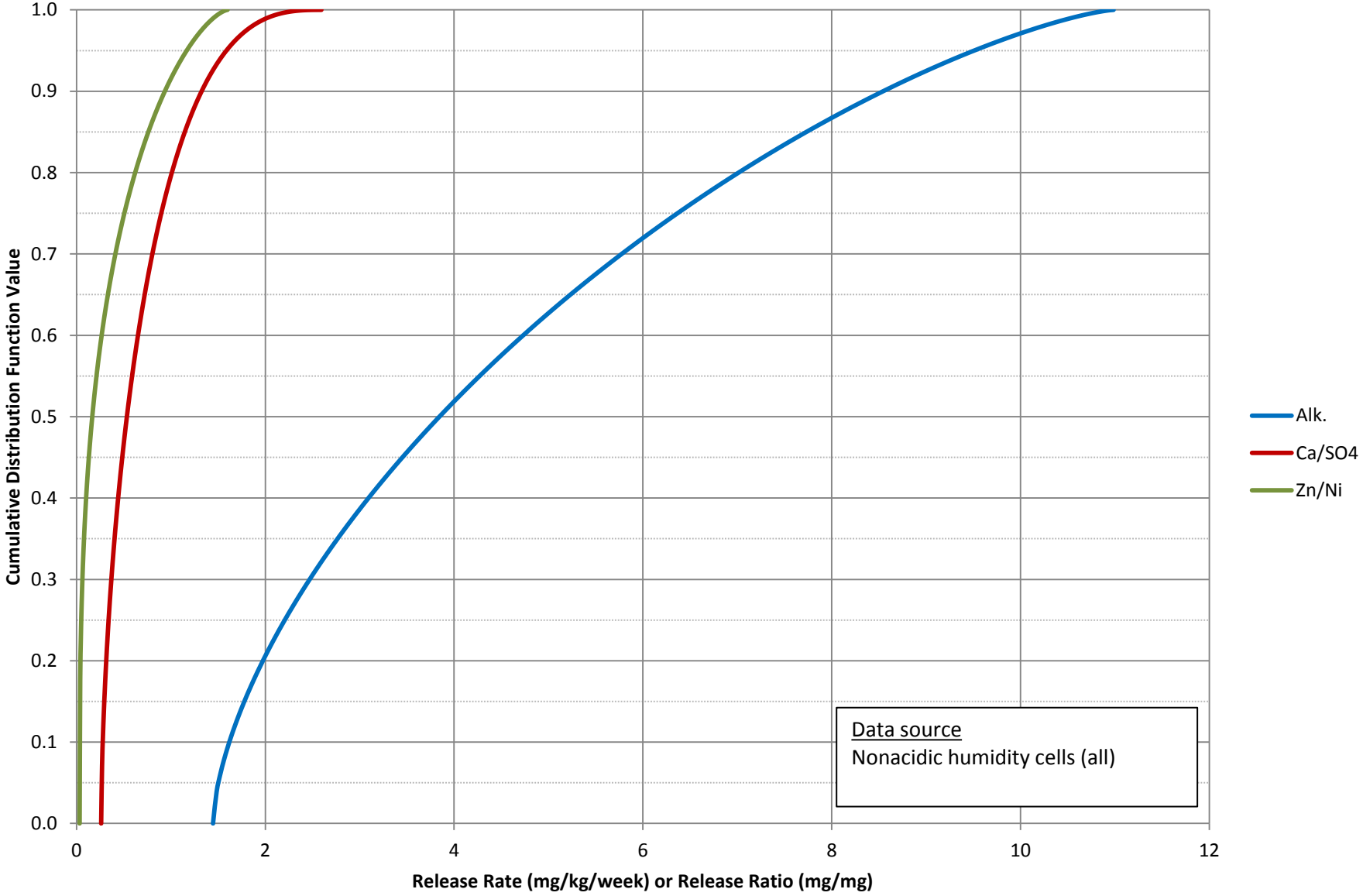


Large Figure 7 **Distributions from Humidity Cells for Category 2/3 Waste Rock and Ore Wall Rock (c)**

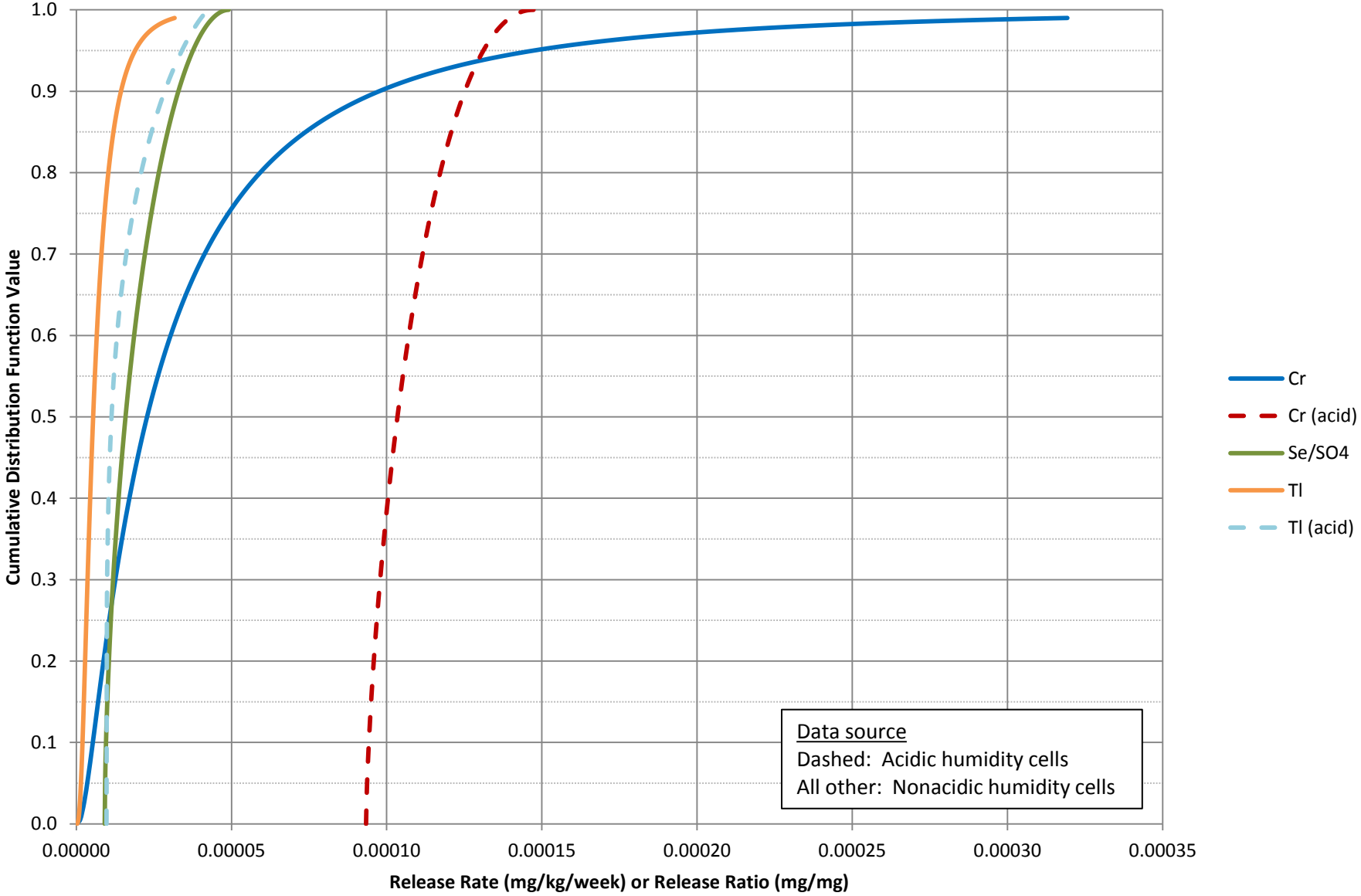


Large Figure 8

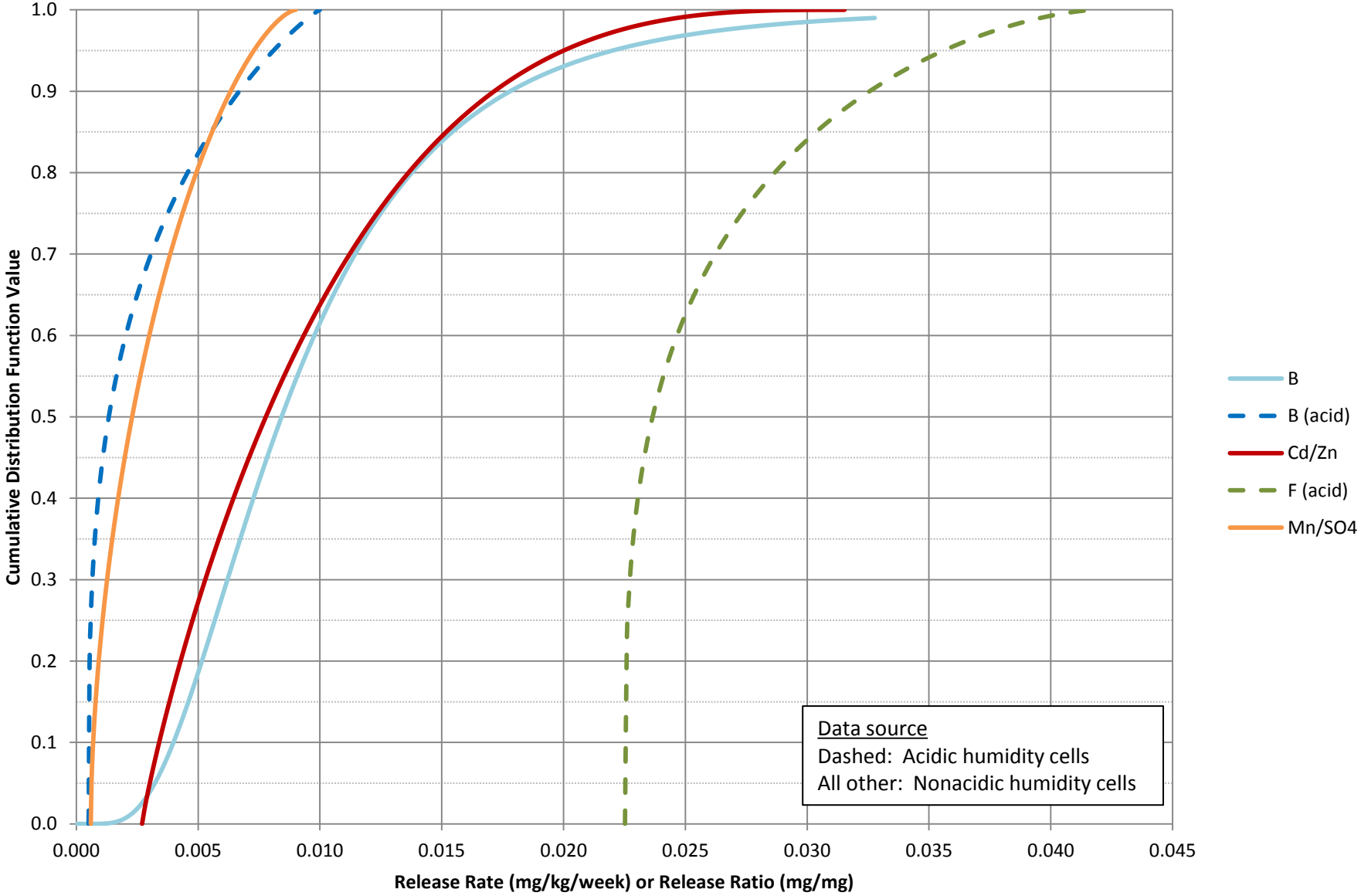
Distributions from Humidity Cells for Category 2/3 Waste Rock and Ore Wall Rock (d)



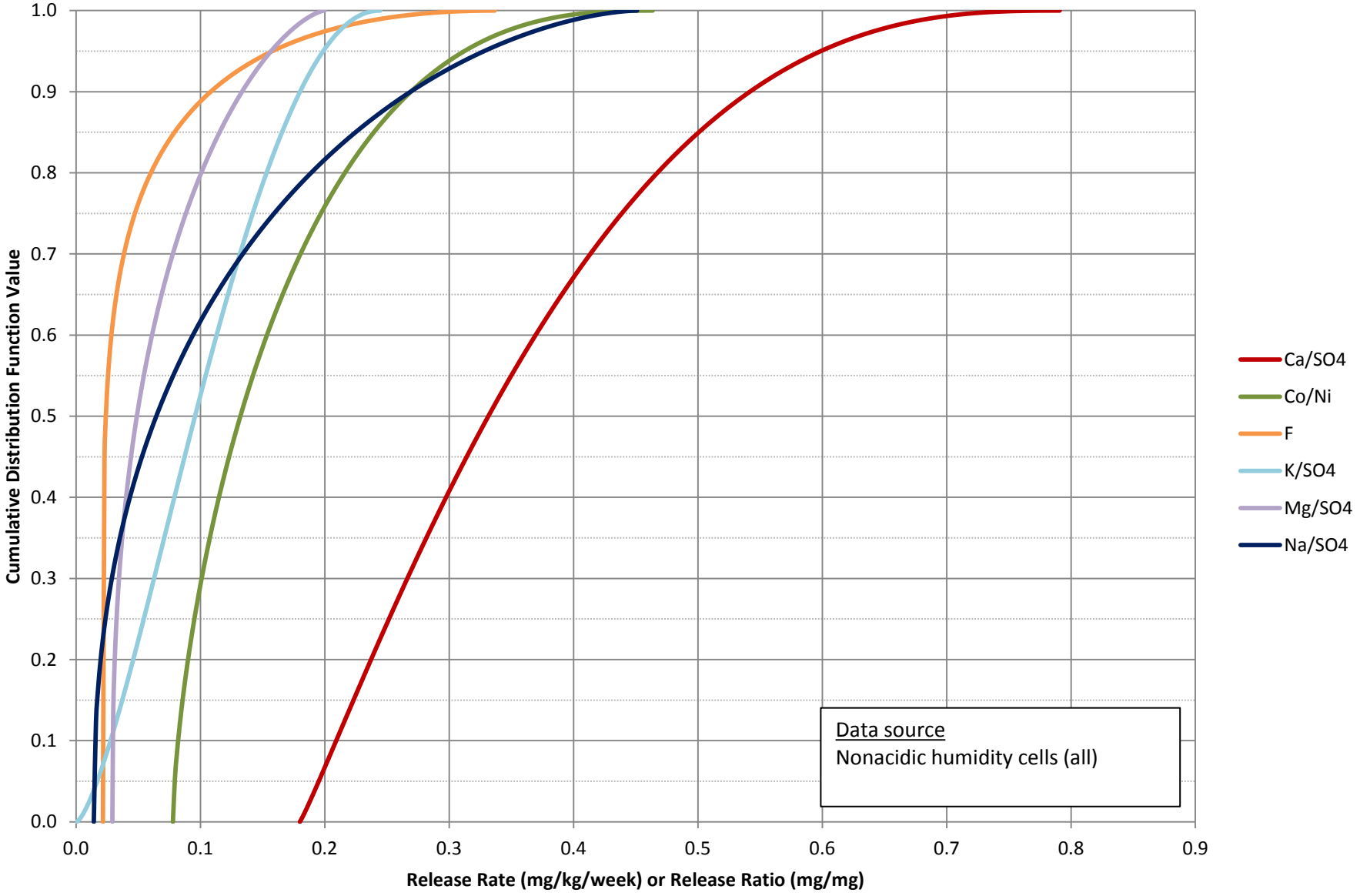
Large Figure 9 Distributions from Humidity Cells for Duluth Complex Category 4 Waste Rock (a)



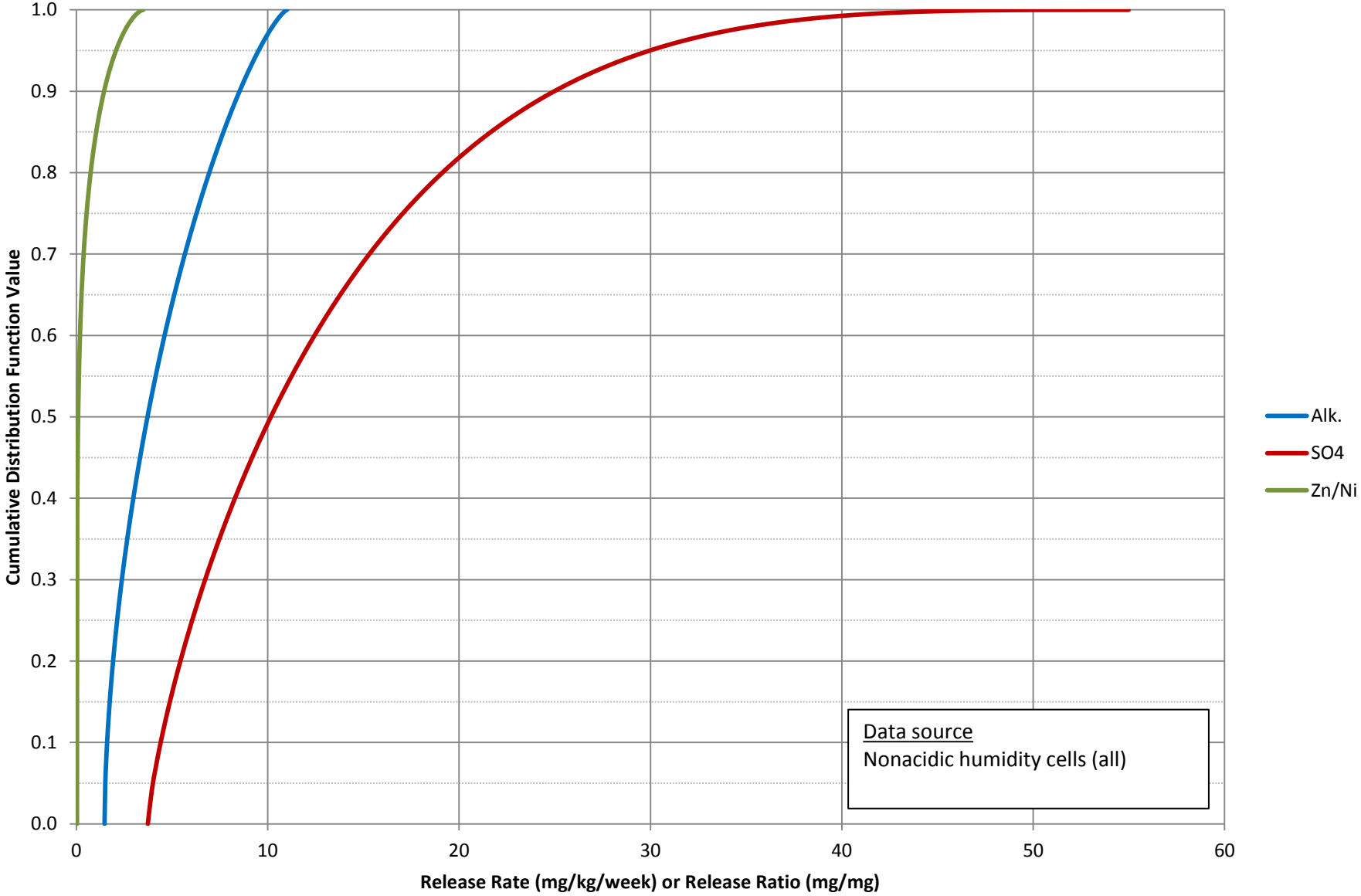
Large Figure 10 **Distributions from Humidity Cells for Duluth Complex Category 4 Waste Rock (b)**



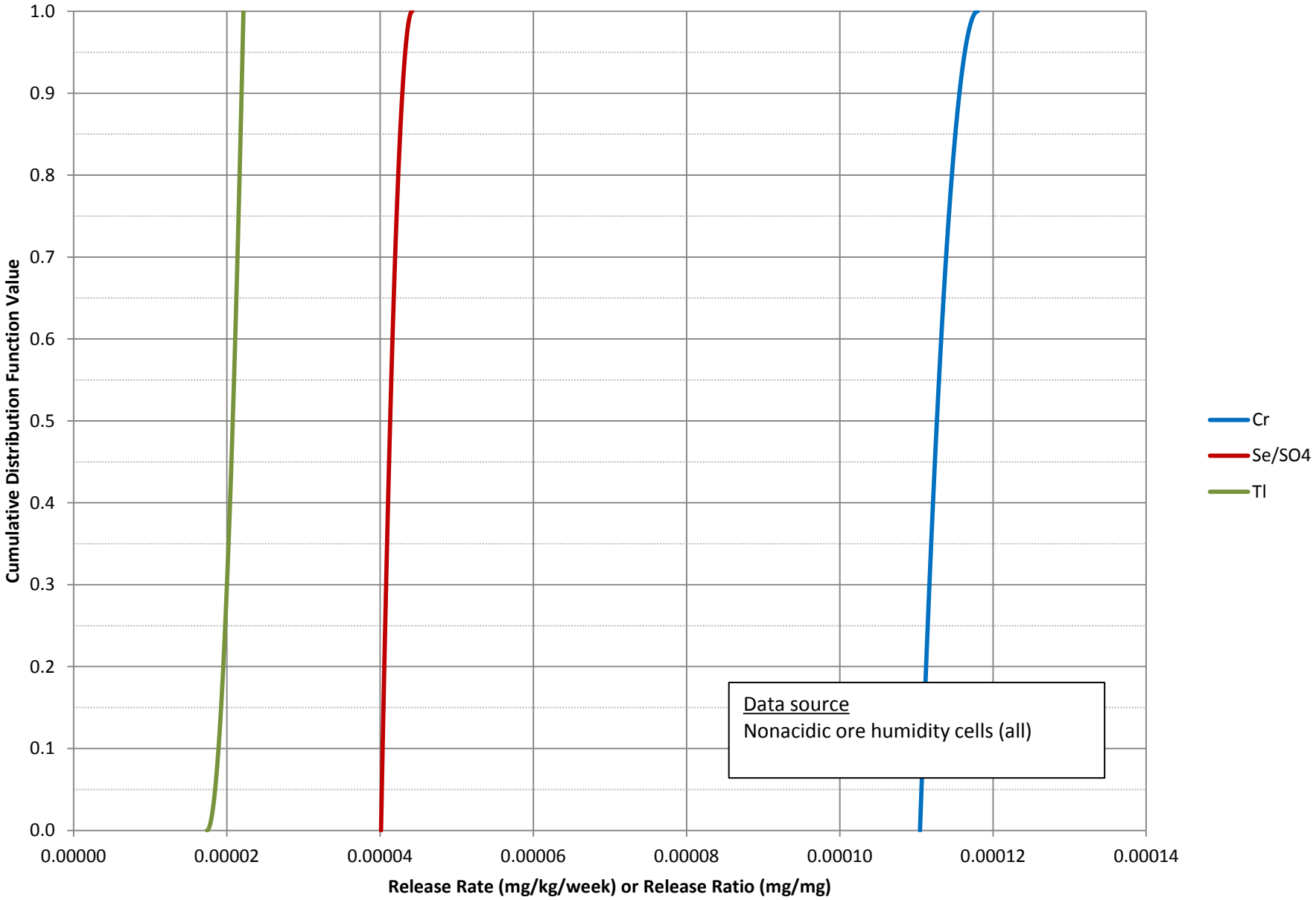
Large Figure 11 **Distributions from Humidity Cells for Duluth Complex Category 4 Waste Rock (c)**



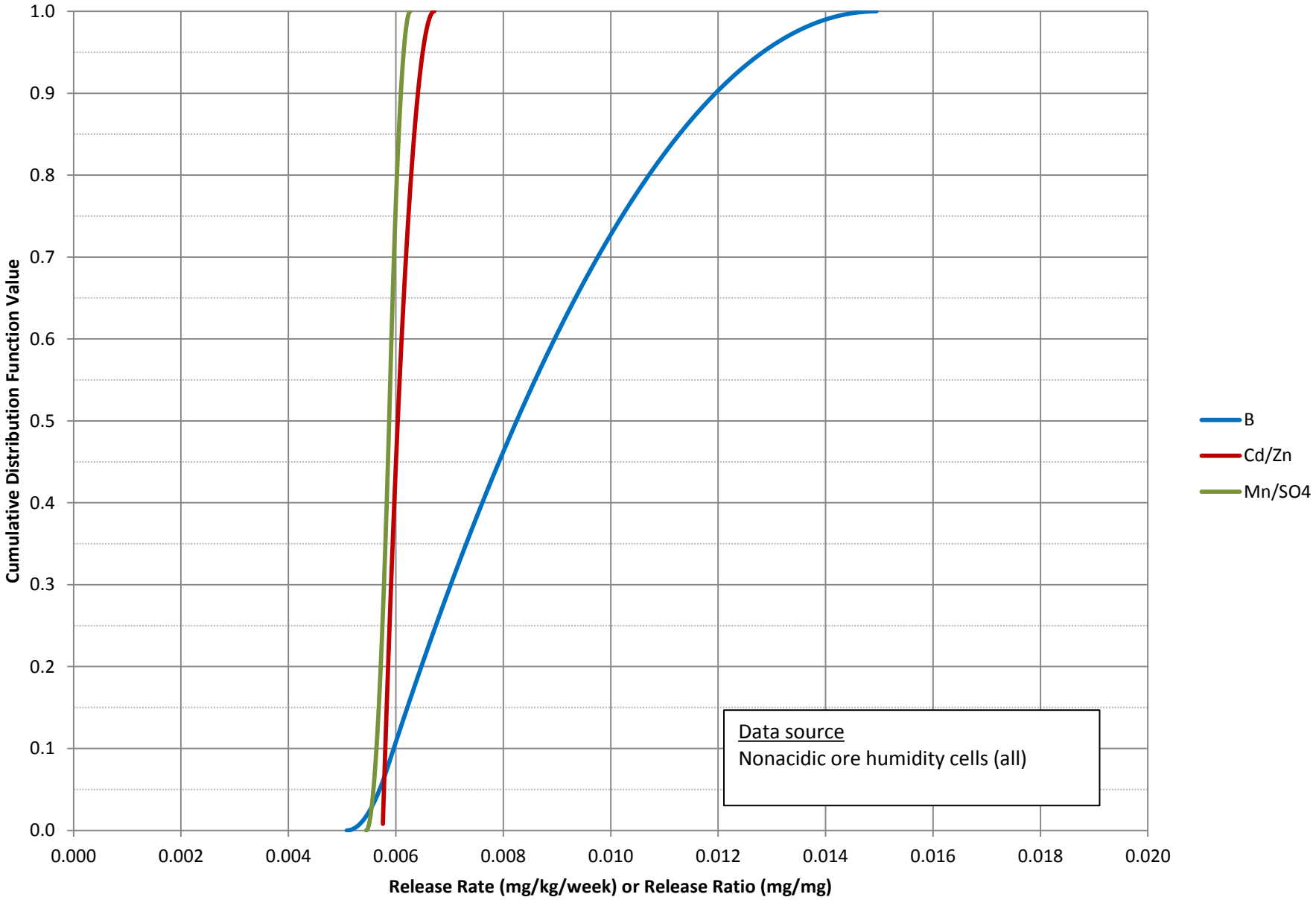
Large Figure 12 **Distributions from Humidity Cells for Duluth Complex Category 4 Waste Rock (d)**



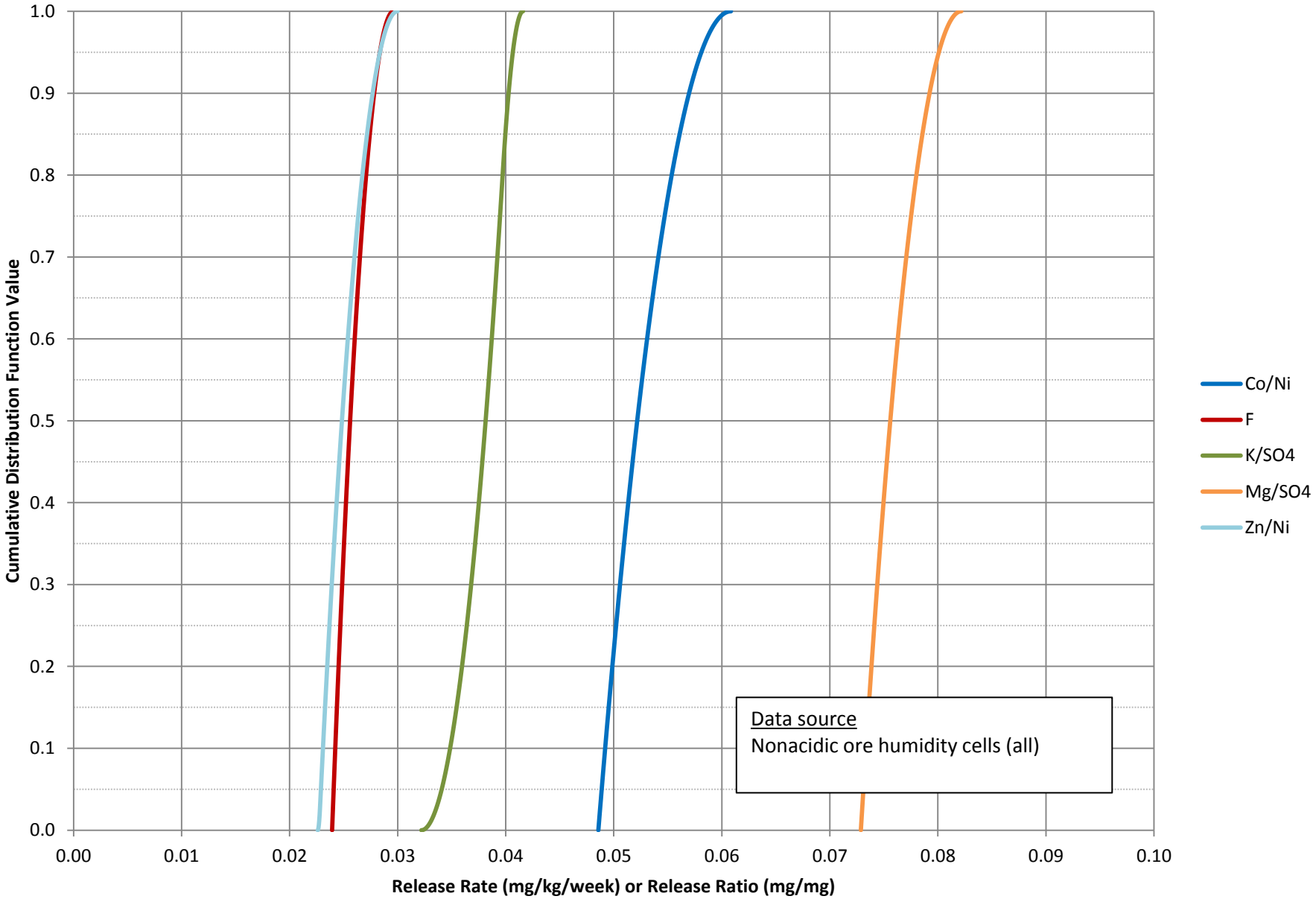
Large Figure 13 Distributions from Humidity Cells for Ore in Surge Pile (a)



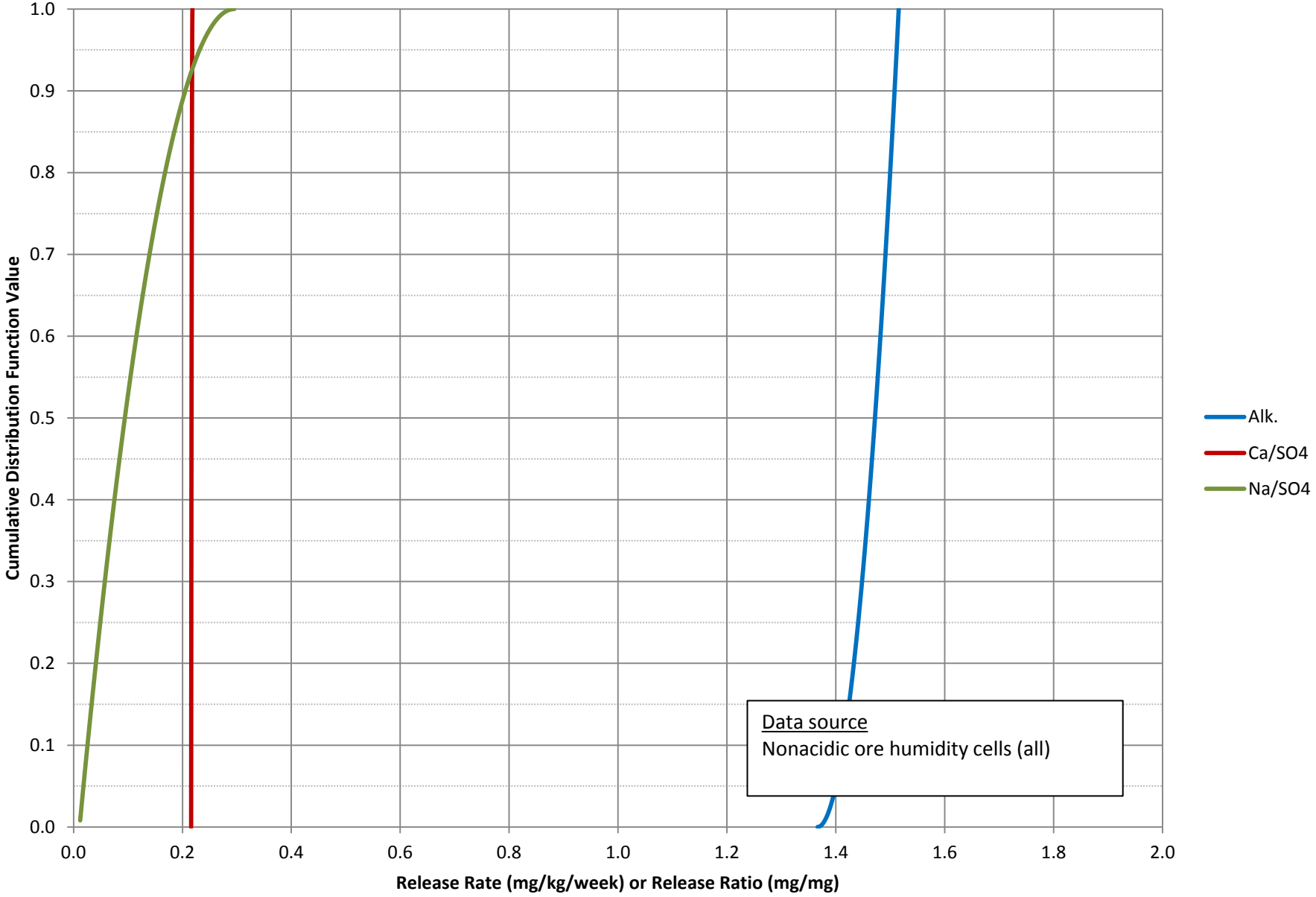
Large Figure 14 Distributions from Humidity Cells for Ore in Surge Pile (b)



Large Figure 15 Distributions from Humidity Cells for Ore in Surge Pile (c)



Large Figure 16 Distributions from Humidity Cells for Ore in Surge Pile (d)



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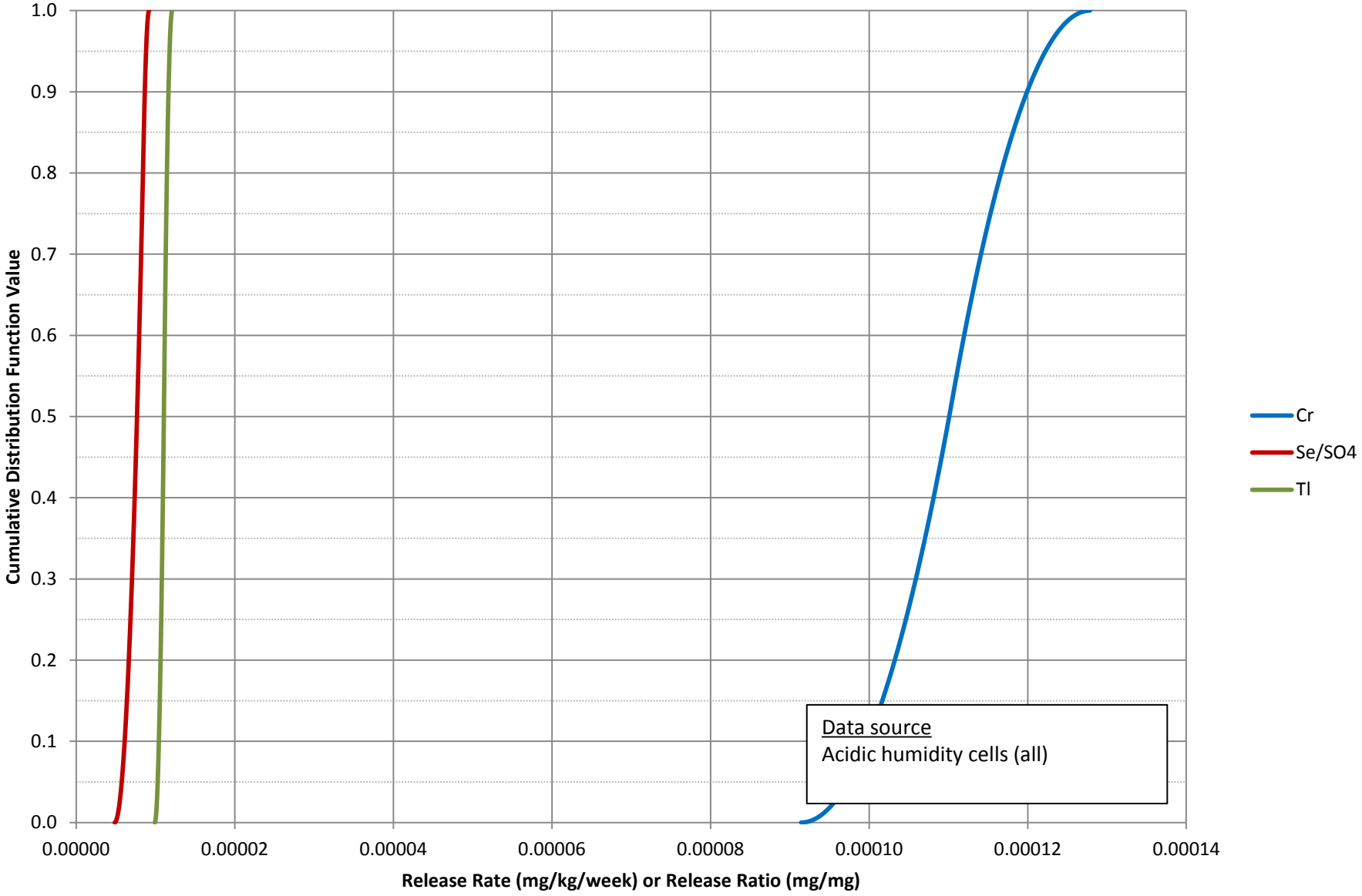
Large Figure 17 **Distributions from Humidity Cells for Virginia Formation Category 4 Waste Rock (a)**



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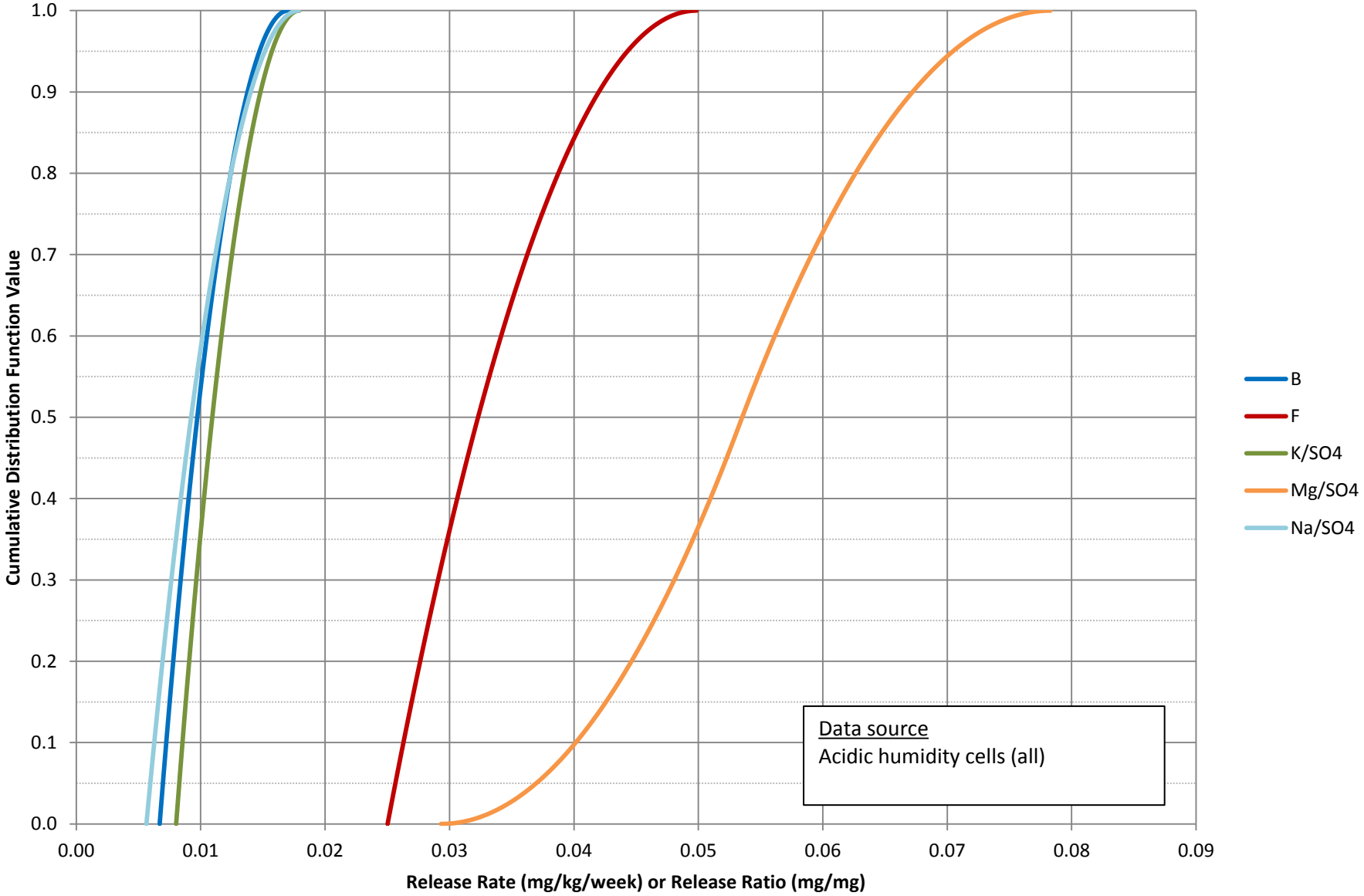
Large Figure 18 **Distributions from Humidity Cells for Virginia Formation Category 4 Waste Rock (b)**



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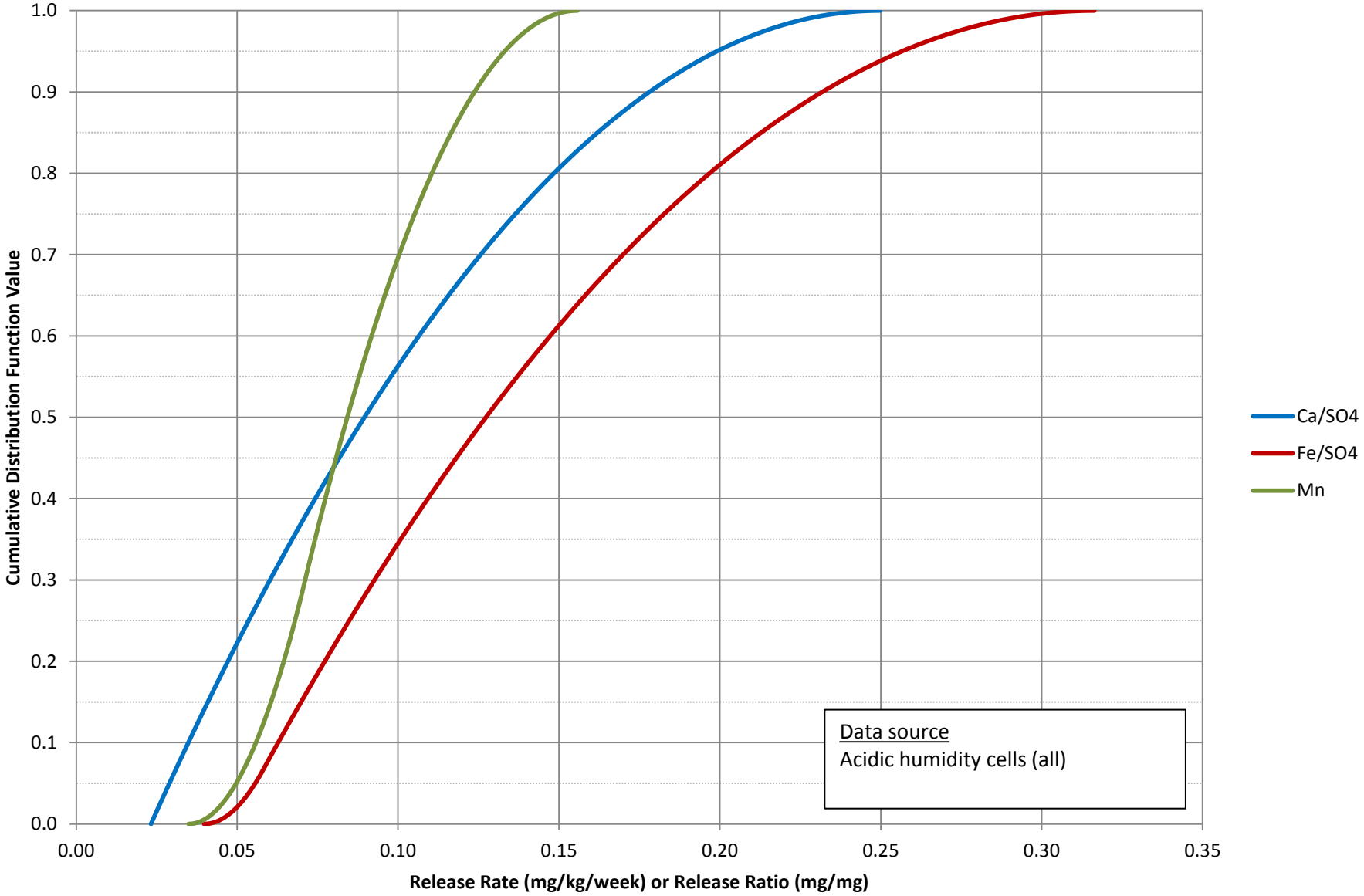
Large Figure 19 **Distributions from Humidity Cells for Virginia Formation Category 4 Waste Rock (c)**



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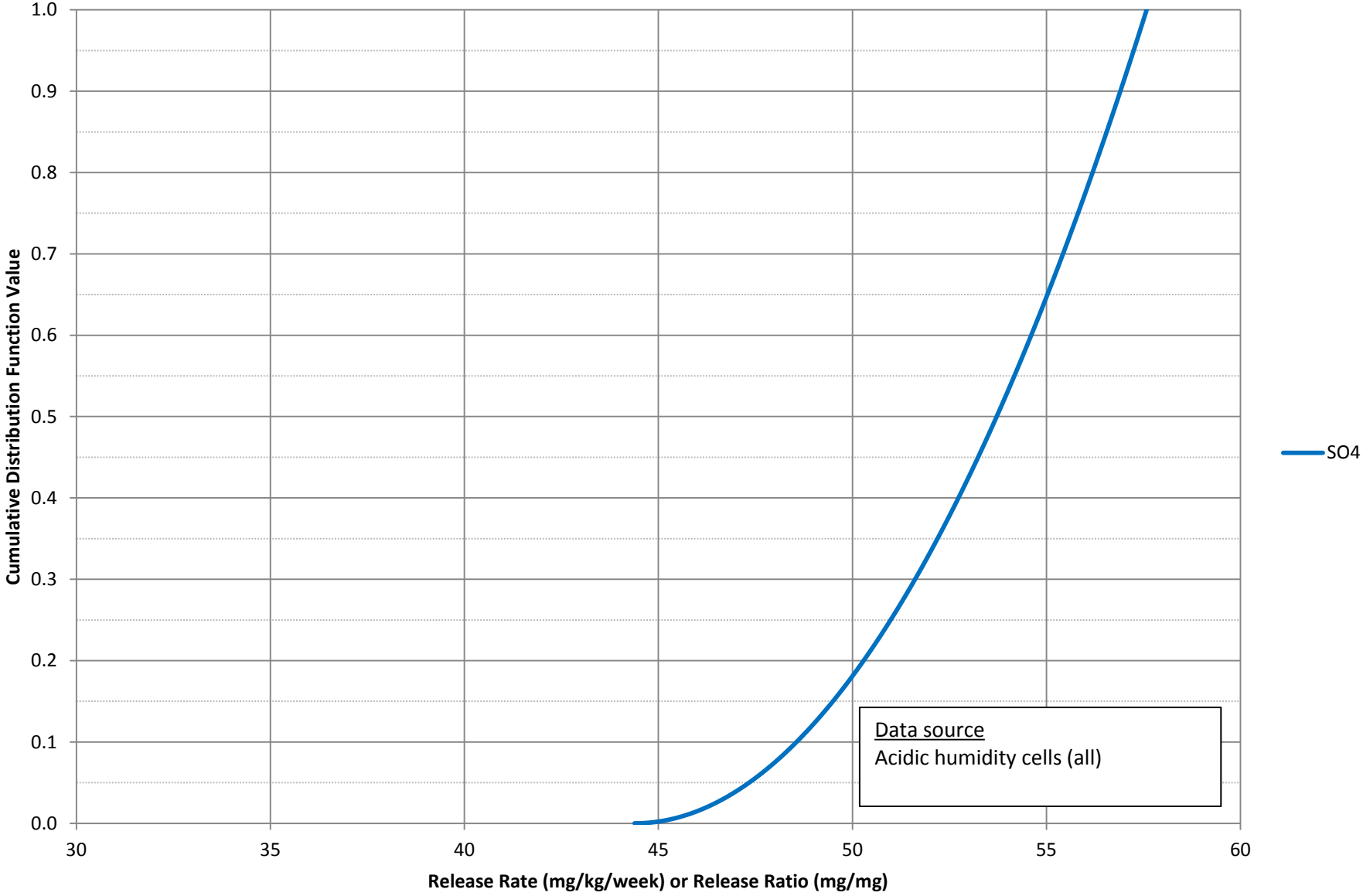
Large Figure 20 **Distributions from Humidity Cells for Virginia Formation Category 4 Waste Rock (d)**



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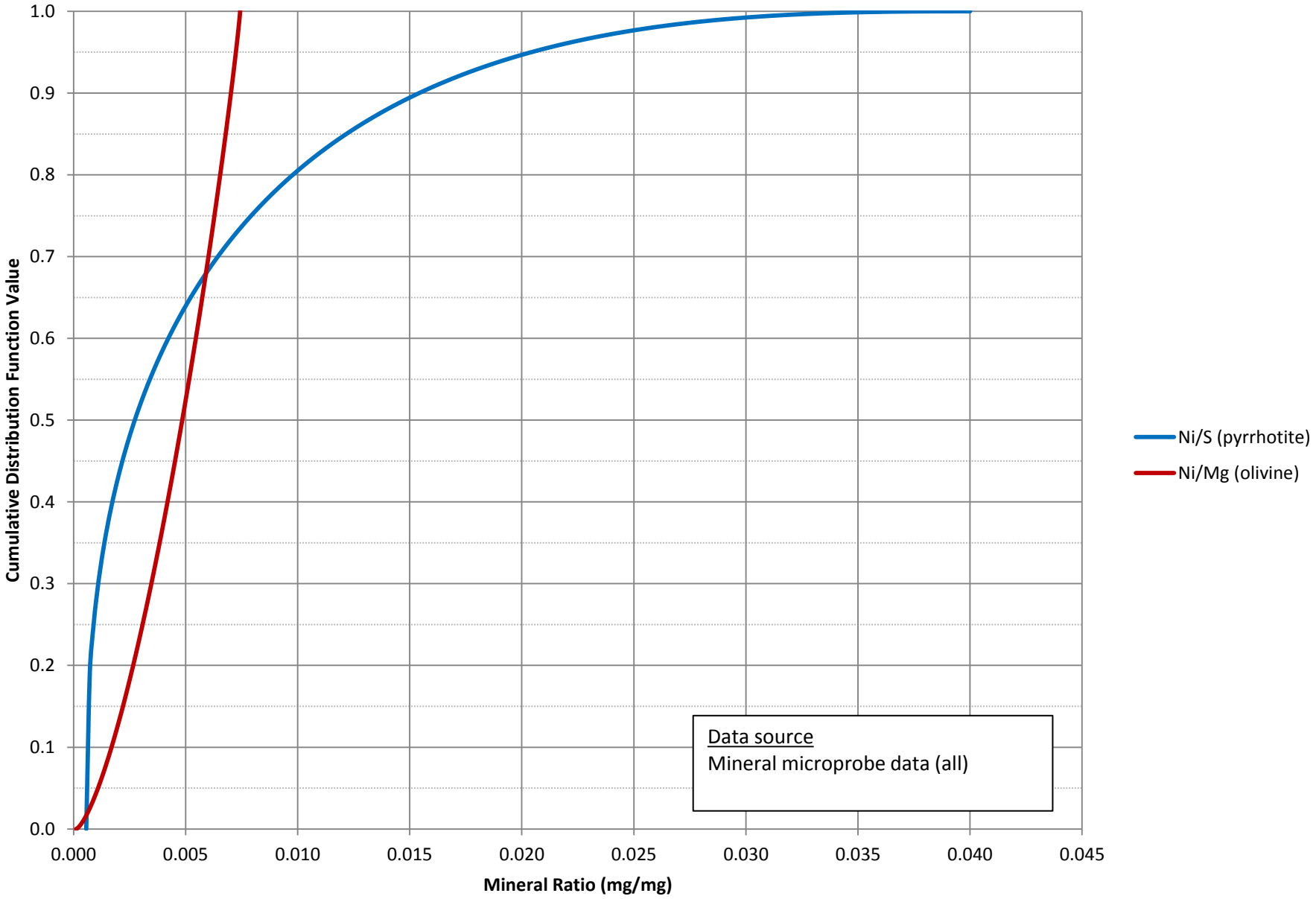
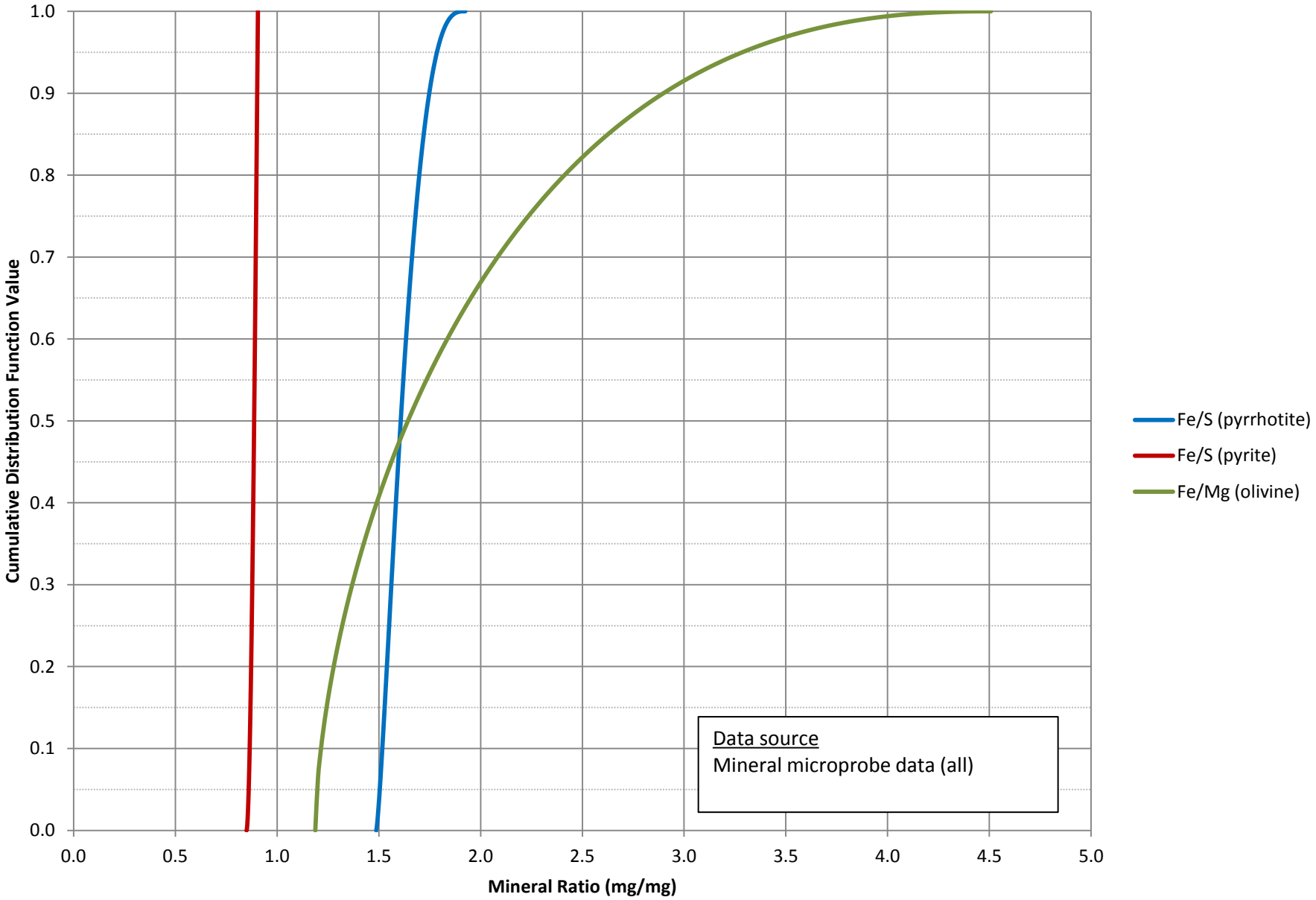
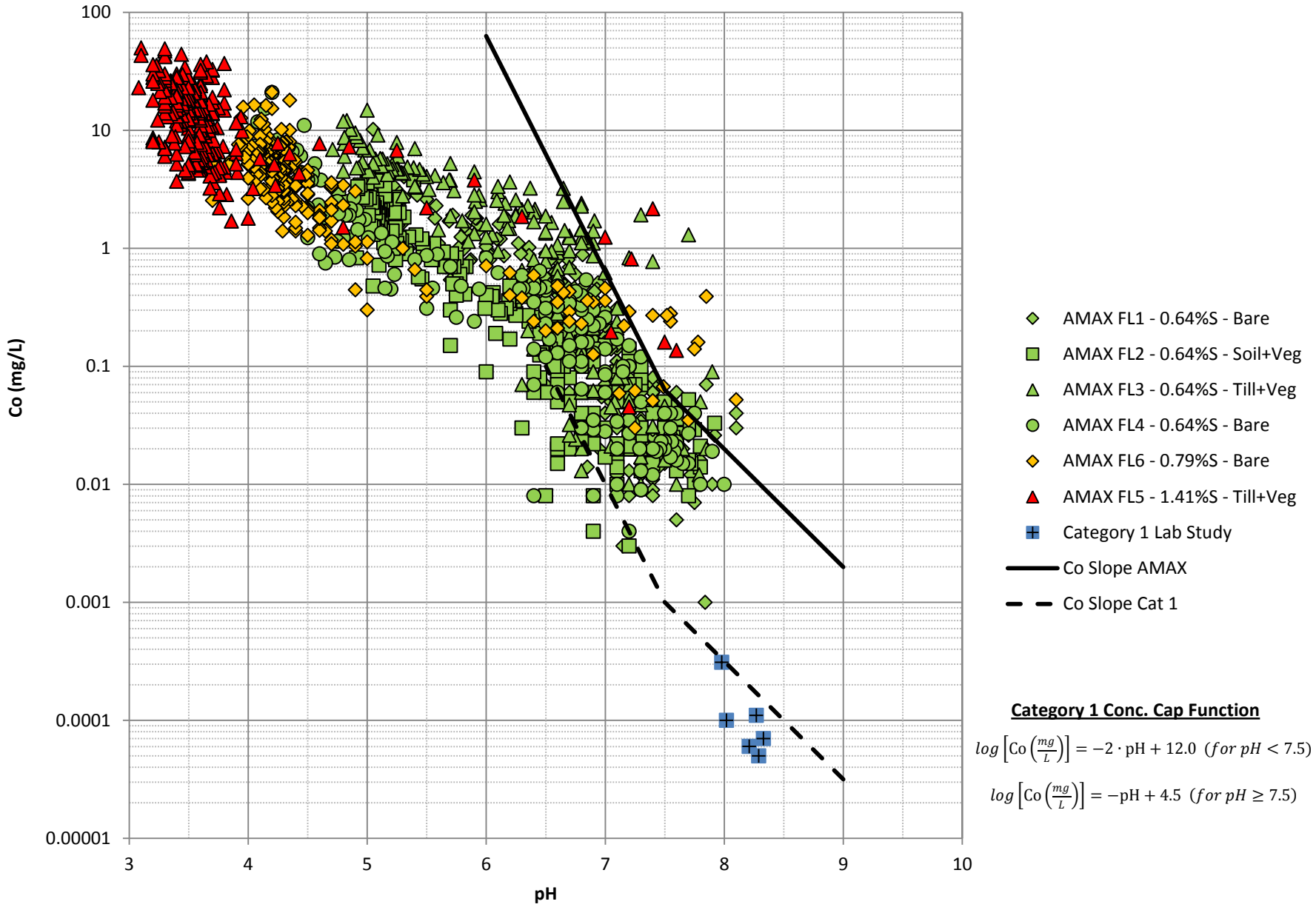
Large Figure 21 Distributions from Microprobe Data for Nickel



Large Figure 22 Distributions from Microprobe Data for Iron



Large Figure 23: Category 1 Concentration Cap Function (Co)



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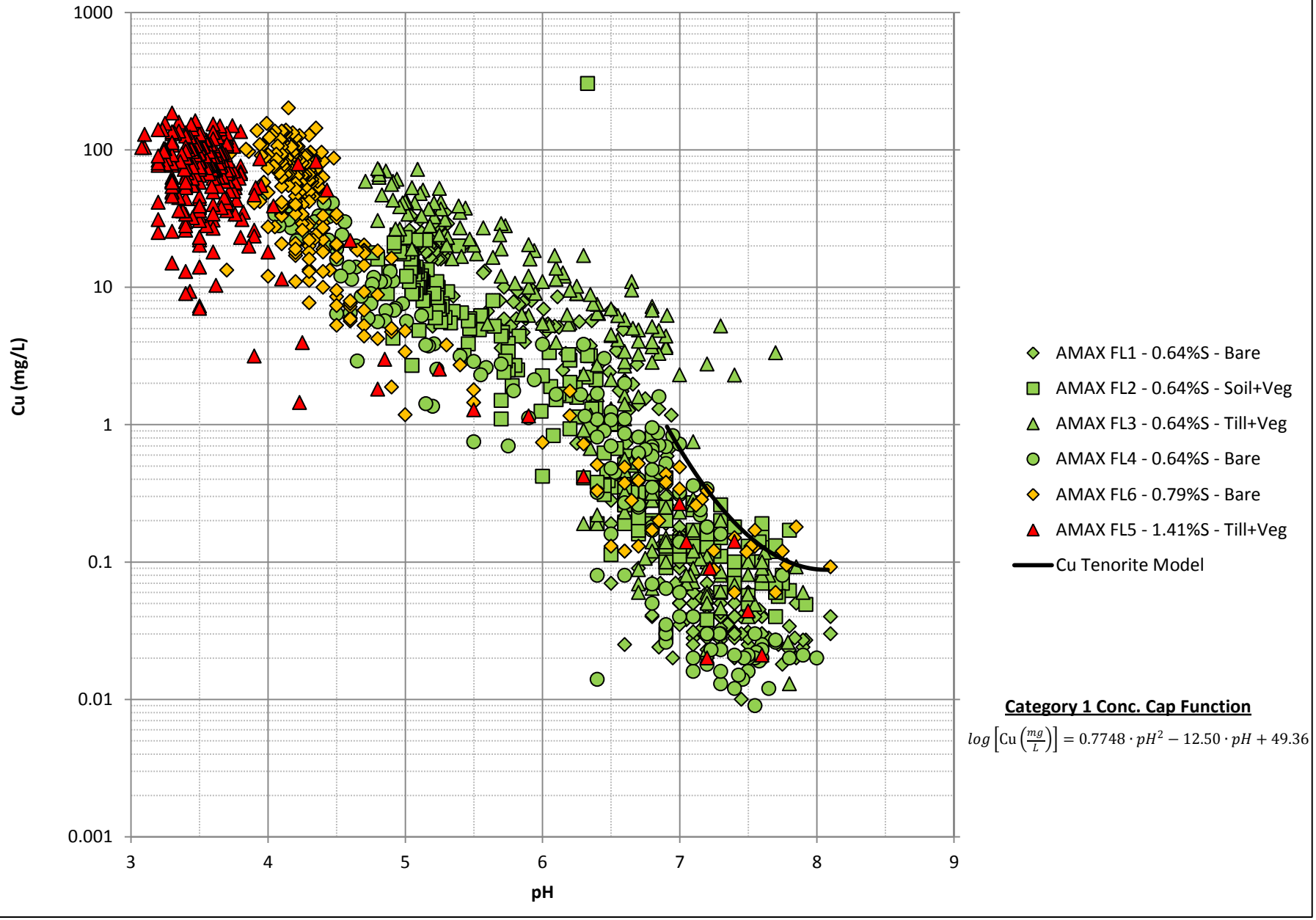
Large Figure 24: Category 1 Concentration Cap Function (Cu)



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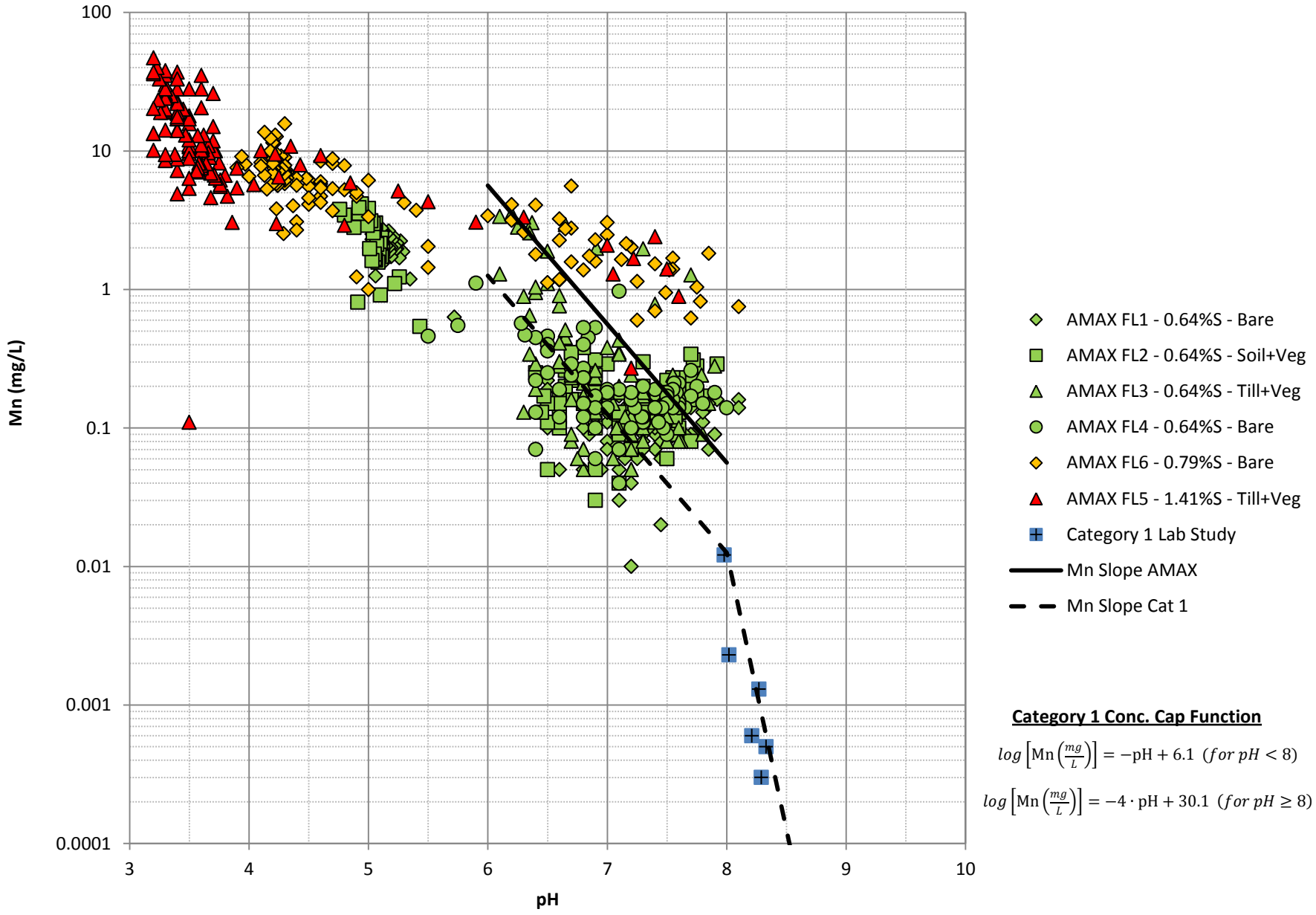
Large Figure 25: Category 1 Concentration Cap Function (Mn)

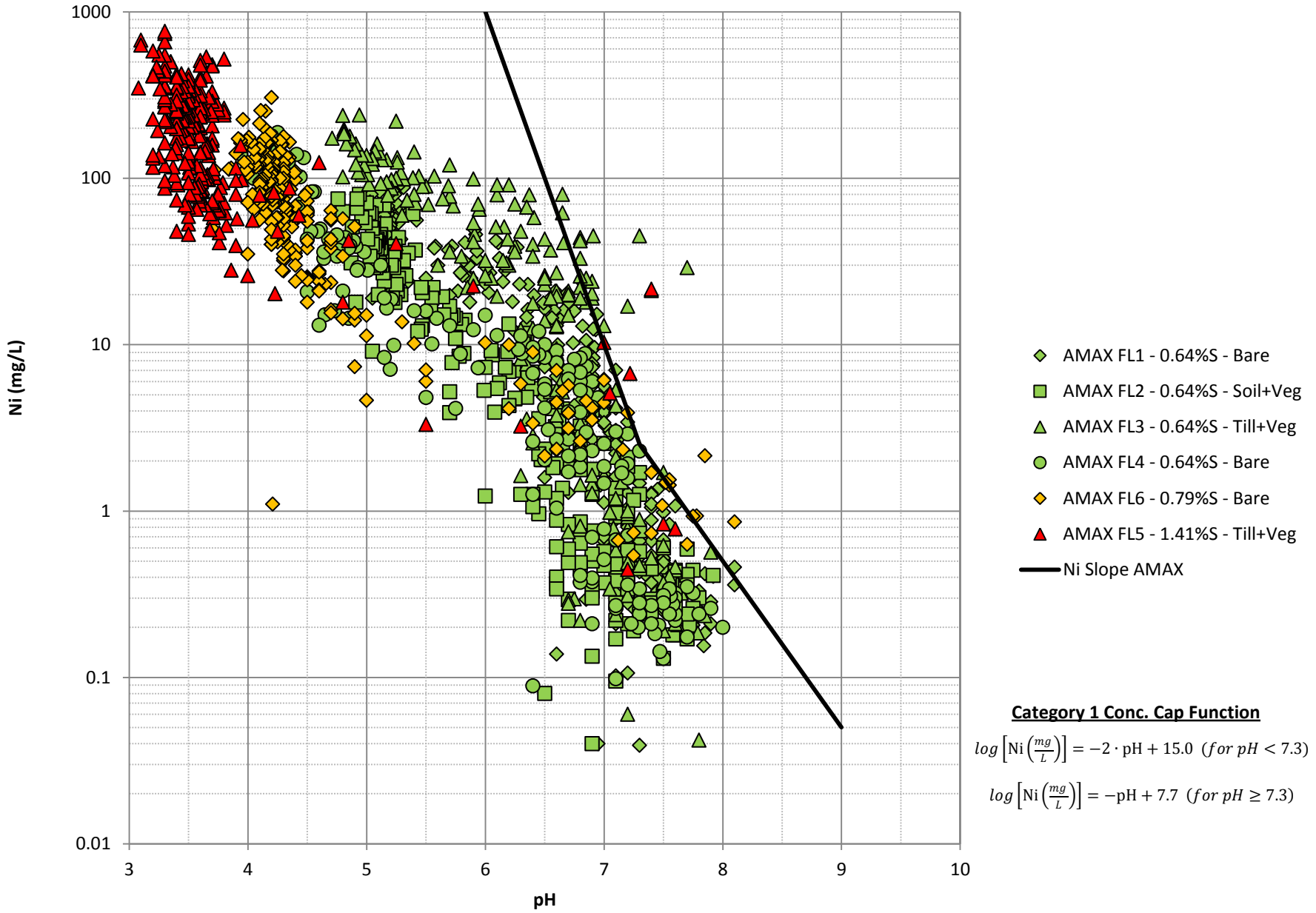


Category 1 Conc. Cap Function

$$log\left[\mathrm{Mn}\left(\frac{mg}{L}\right)\right] = -\mathrm{pH} + 6.1 \quad (for\ pH < 8)$$

$$log\left[\mathrm{Mn}\left(\frac{mg}{L}\right)\right] = -4 \cdot \mathrm{pH} + 30.1 \quad (for\ pH \geq 8)$$

Large Figure 26: Category 1 Concentration Cap Function (Ni)



Category 1 Conc. Cap Function

$$log\left[\mathrm{Ni}\left(\frac{mg}{L}\right)\right] = -2 \cdot \mathrm{pH} + 15.0 \quad (for\ pH < 7.3)$$

$$log\left[\mathrm{Ni}\left(\frac{mg}{L}\right)\right] = -\mathrm{pH} + 7.7 \quad (for\ pH \geq 7.3)$$

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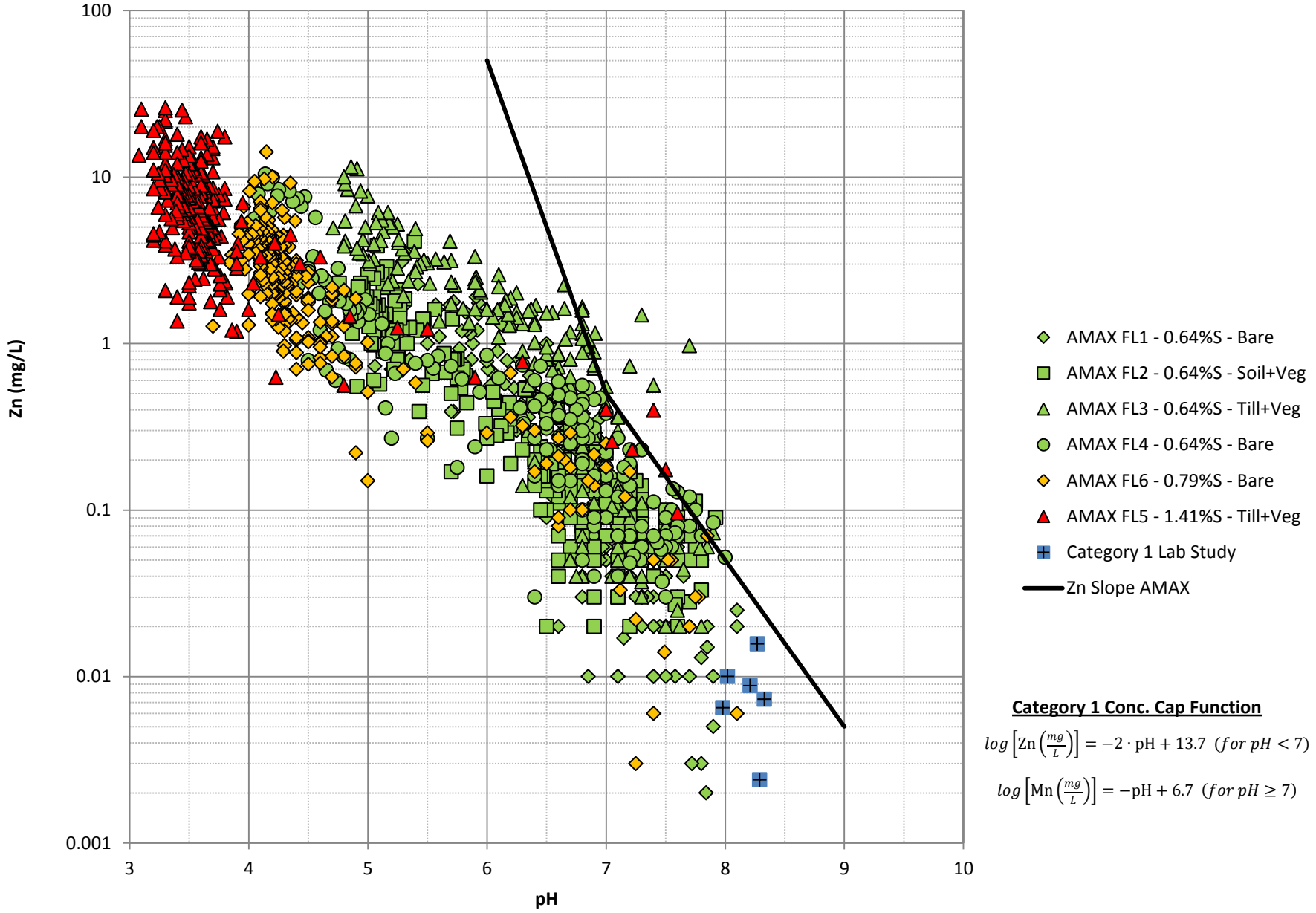
Large Figure 27: Category 1 Concentration Cap Function (Zn)

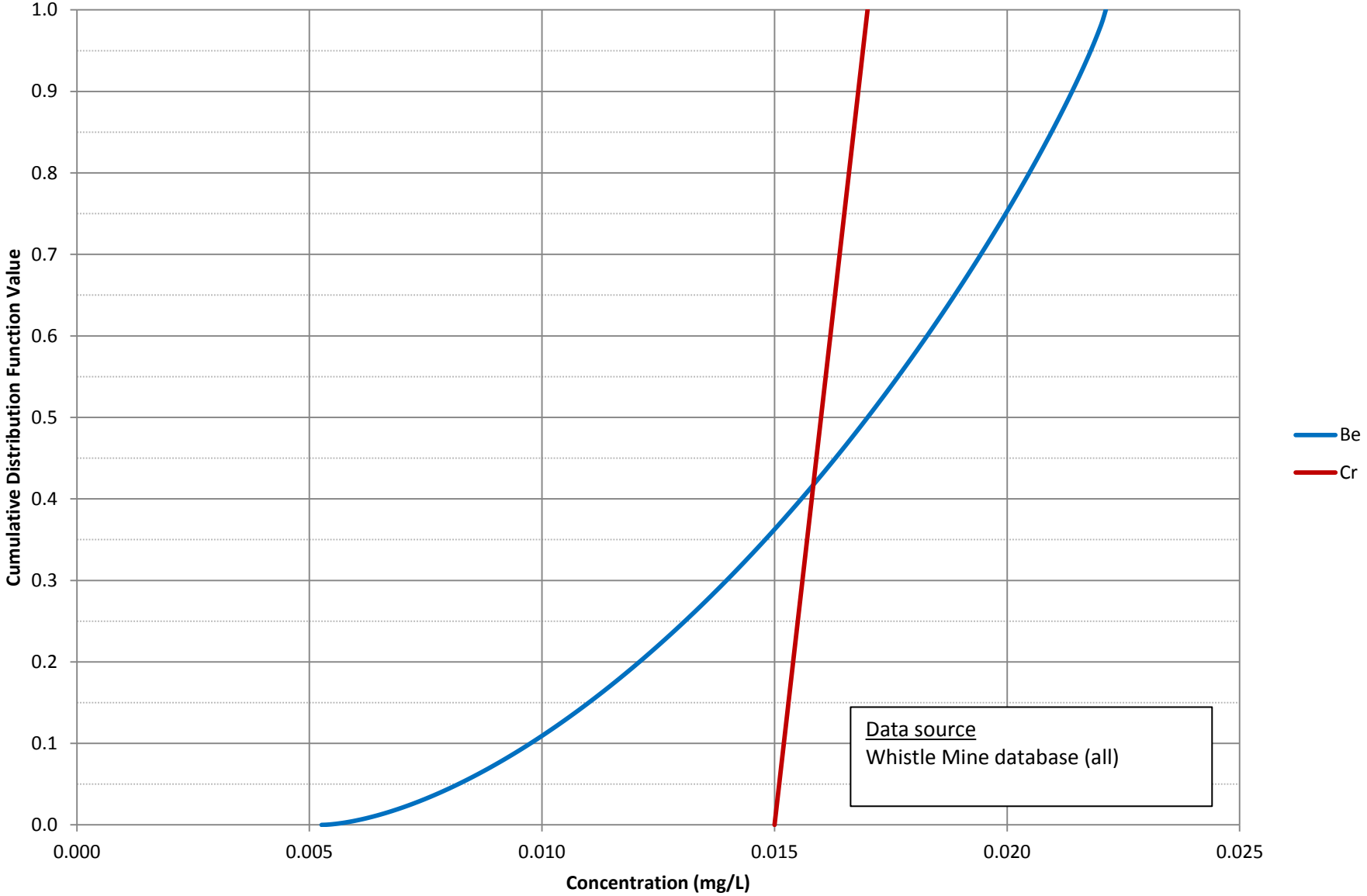


Category 1 Conc. Cap Function

$$log\left[\mathrm{Zn}\left(\frac{mg}{L}\right)\right] = -2 \cdot \mathrm{pH} + 13.7 \quad (for\ pH < 7)$$

$$log\left[\mathrm{Mn}\left(\frac{mg}{L}\right)\right] = -\mathrm{pH} + 6.7 \quad (for\ pH \geq 7)$$

Large Figure 28 Distributions for Duluth Complex Category 2/3, 4 and Ore Concentration Caps (nonacidic) (a)



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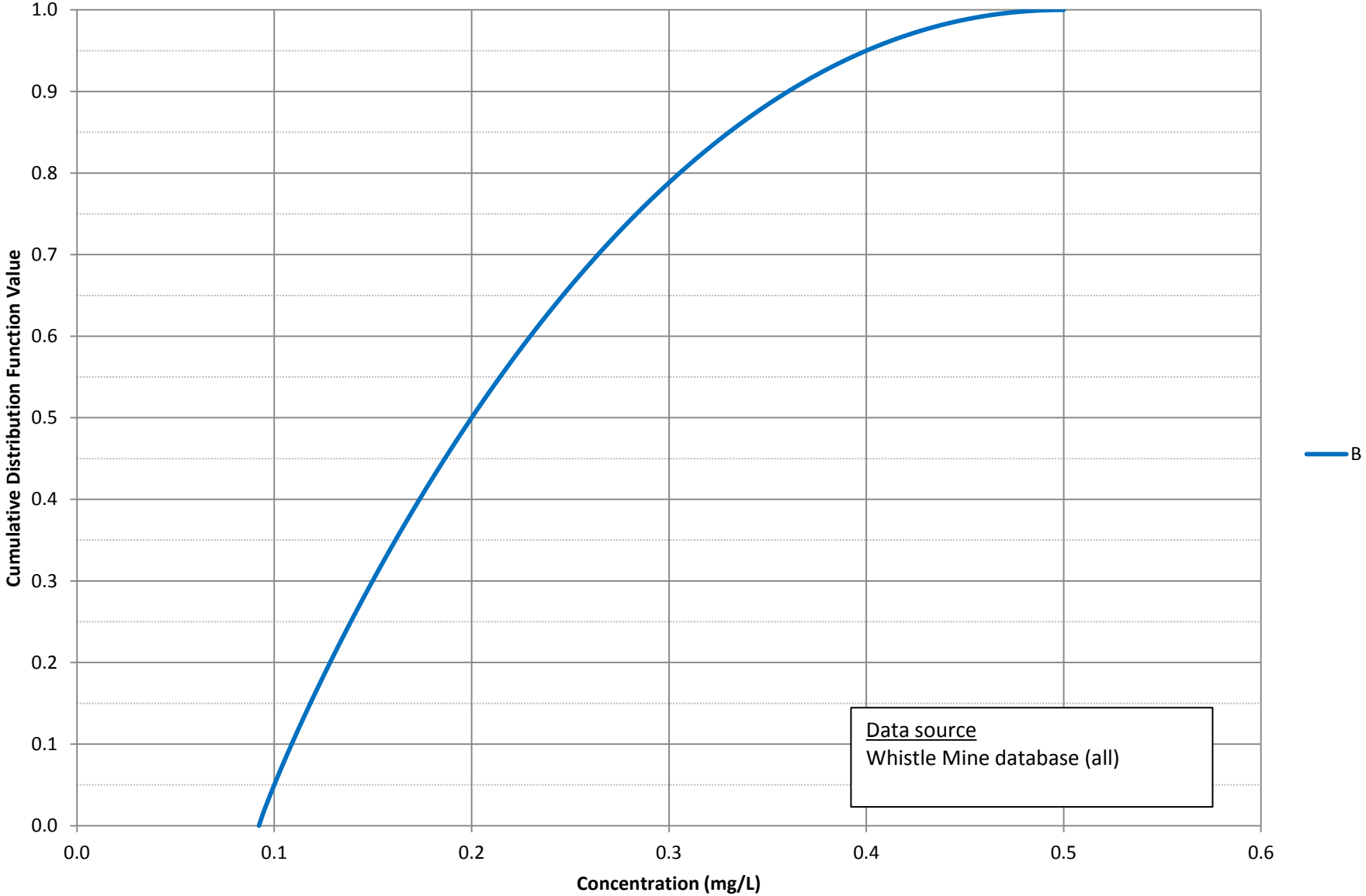
Large Figure 29 **Distributions for Duluth Complex Category 2/3, 4 and Ore Concentration Caps (nonacidic) (b)**



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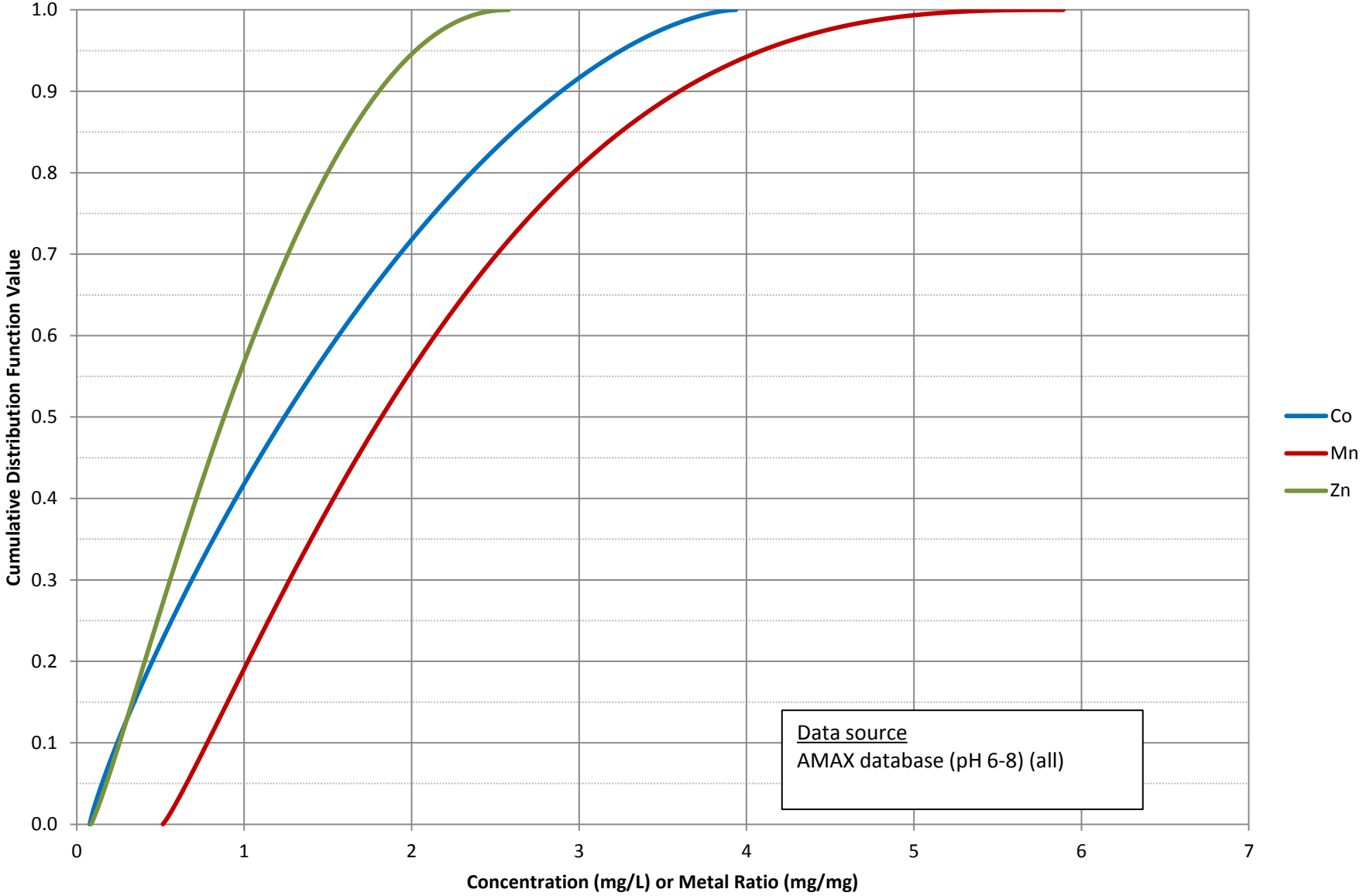
Large Figure 30 Distributions for Duluth Complex Category 2/3, 4 and Ore Concentration Caps (nonacidic) (c)



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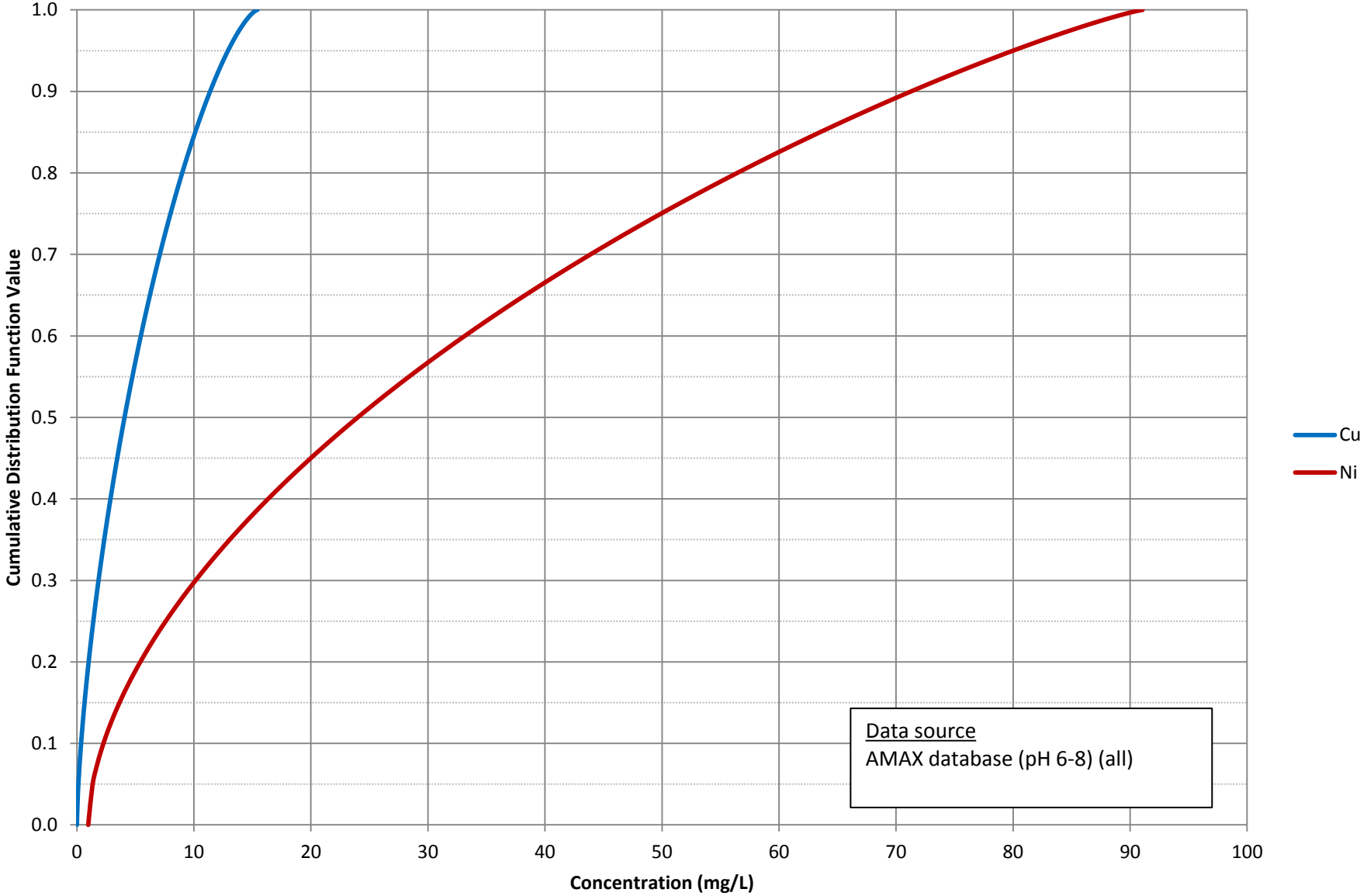
Large Figure 31 **Distributions for Duluth Complex Category 2/3, 4 and Ore Concentration Caps (nonacidic) (d)**



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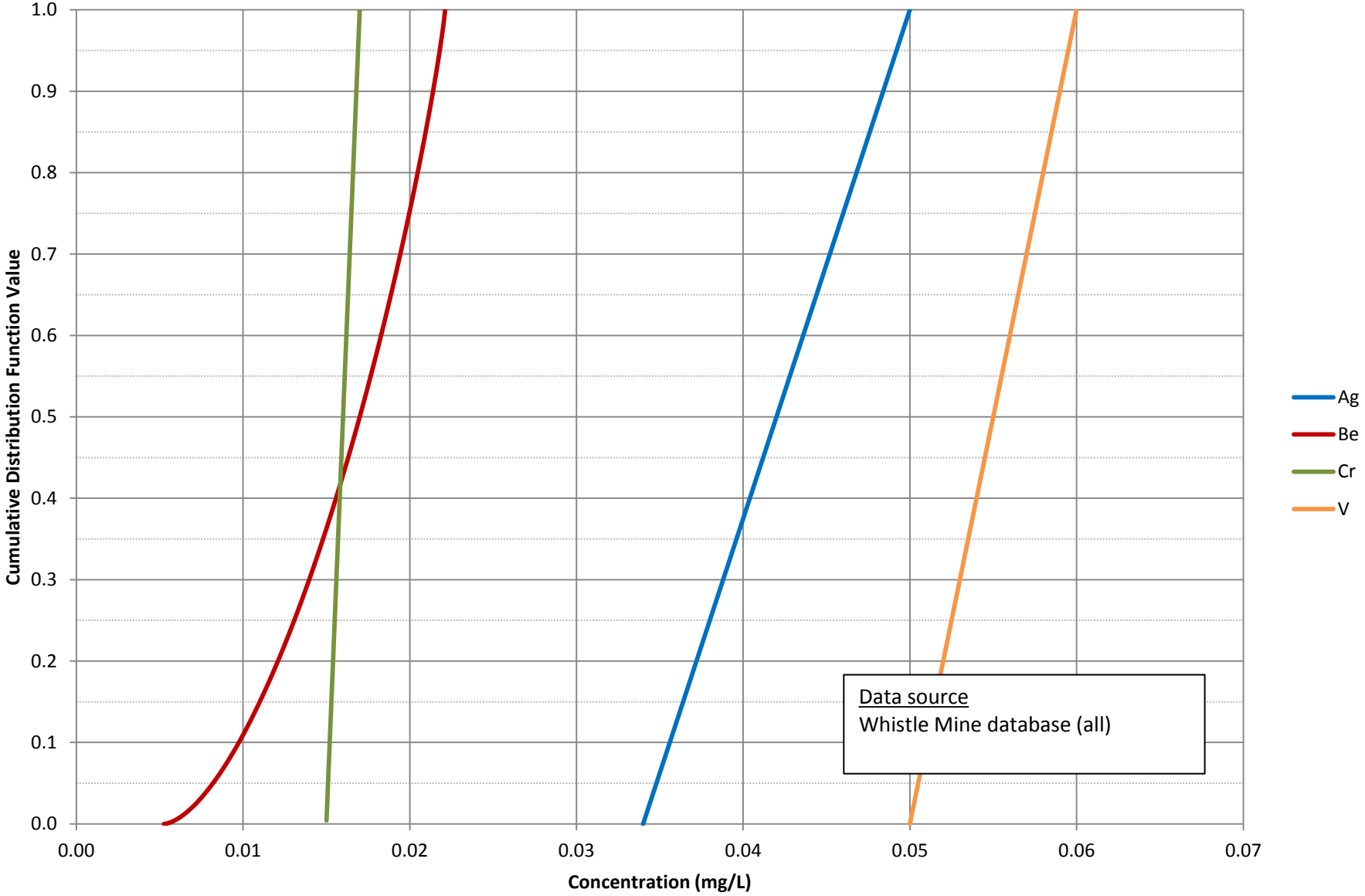
Large Figure 32 **Distributions for Duluth Complex Category 2/3, 4 and Ore Concentration Caps (acidic) (a)**



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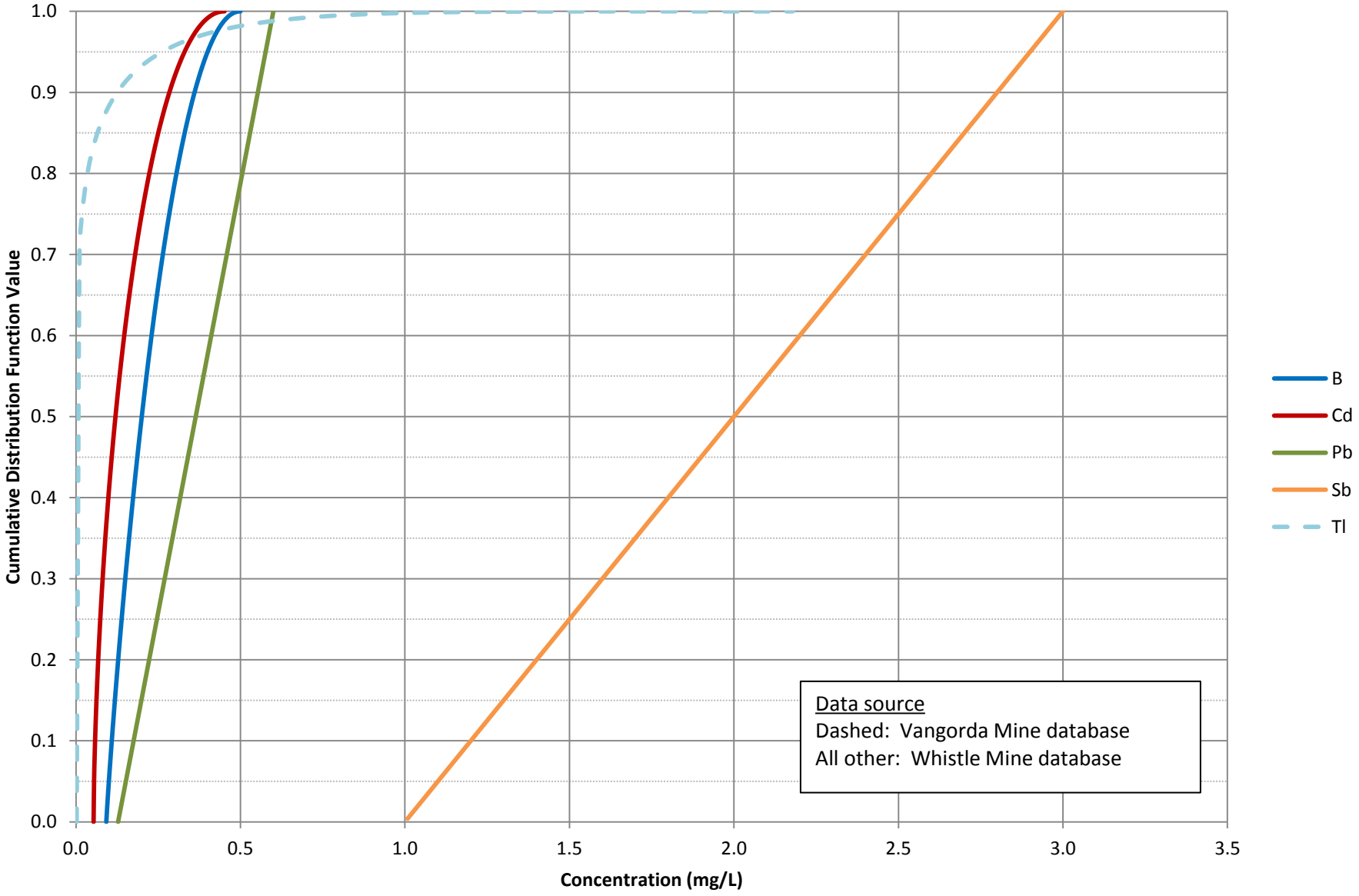
Large Figure 33 Distributions for Duluth Complex Category 2/3, 4 and Ore Concentration Caps (acidic) (b)



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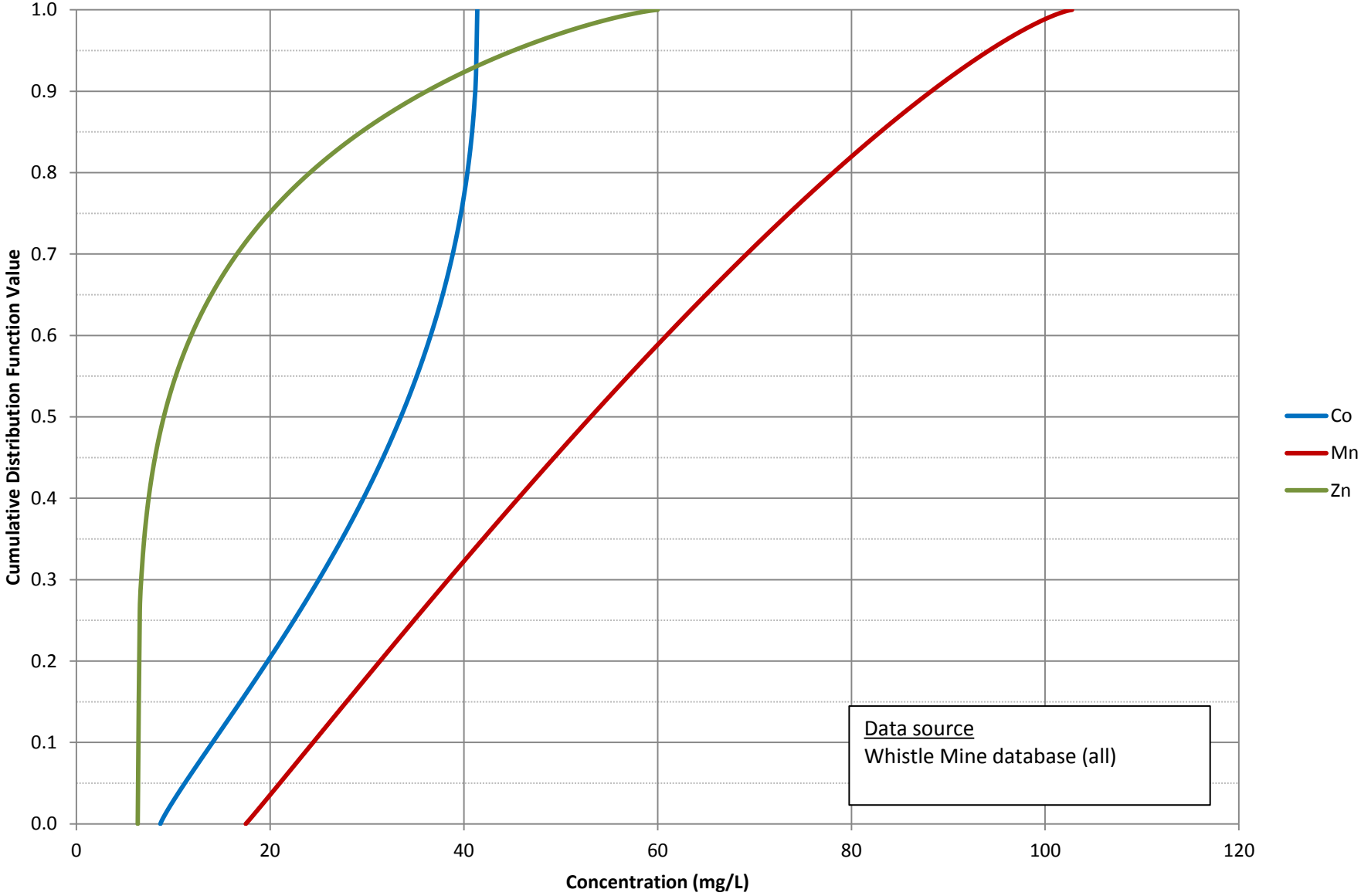
Large Figure 34 **Distributions for Duluth Complex Category 2/3, 4 and Ore Concentration Caps (acidic) (c)**



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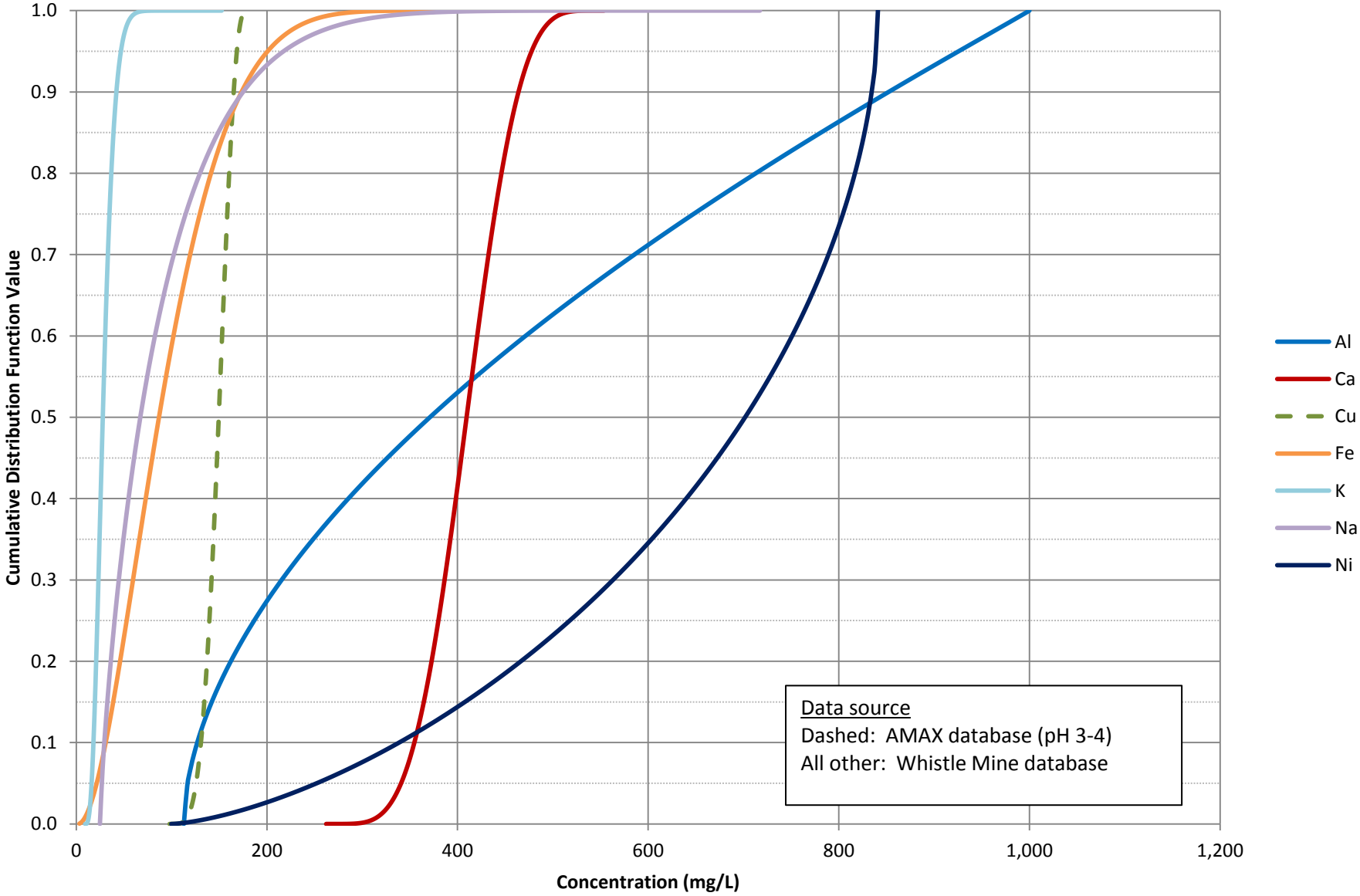
Large Figure 35 **Distributions for Duluth Complex Category 2/3, 4 and Ore Concentration Caps (acidic) (d)**



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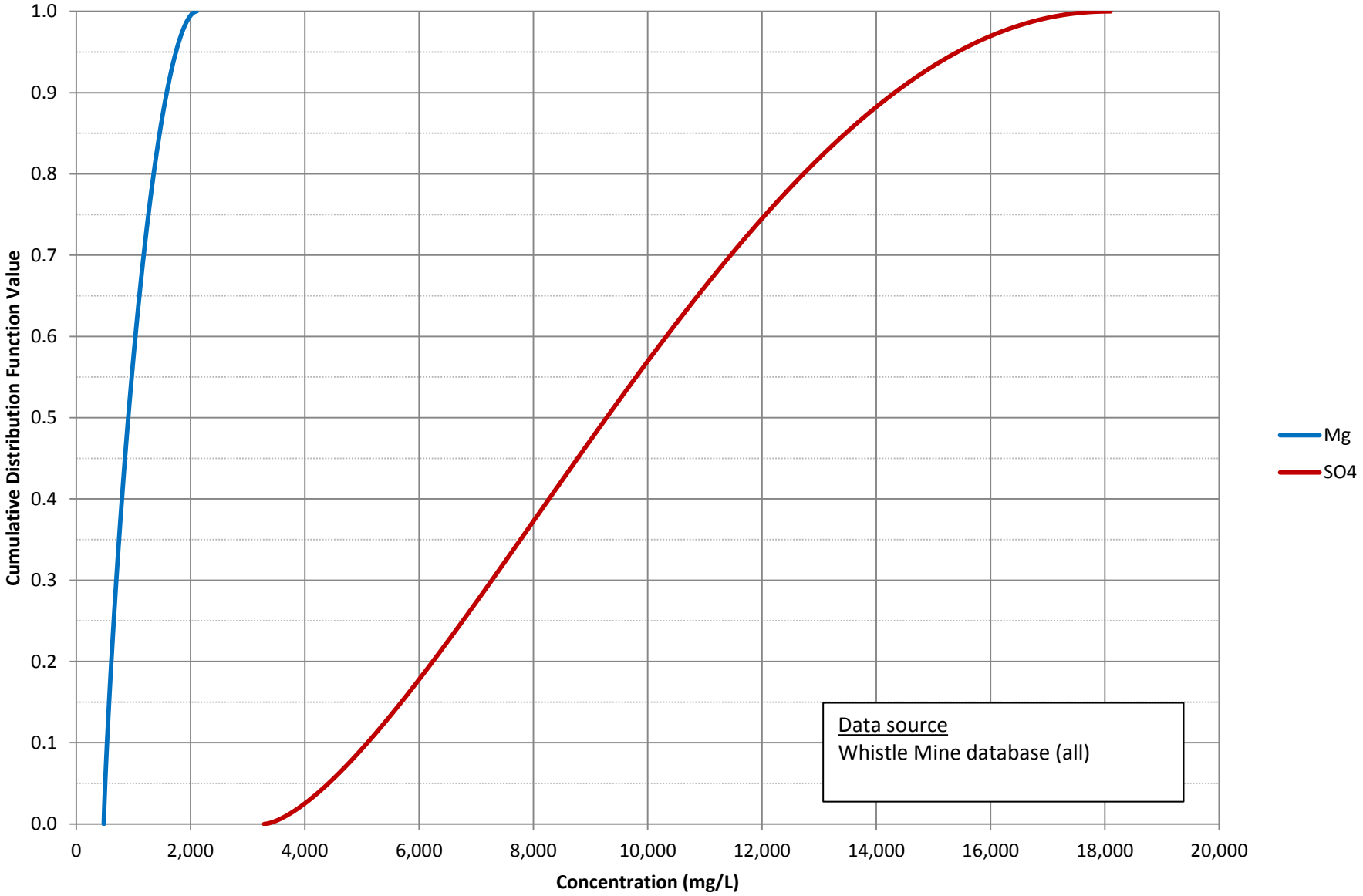
Large Figure 36 Distributions for Duluth Complex Category 2/3, 4 and Ore Concentration Caps (acidic) (e)



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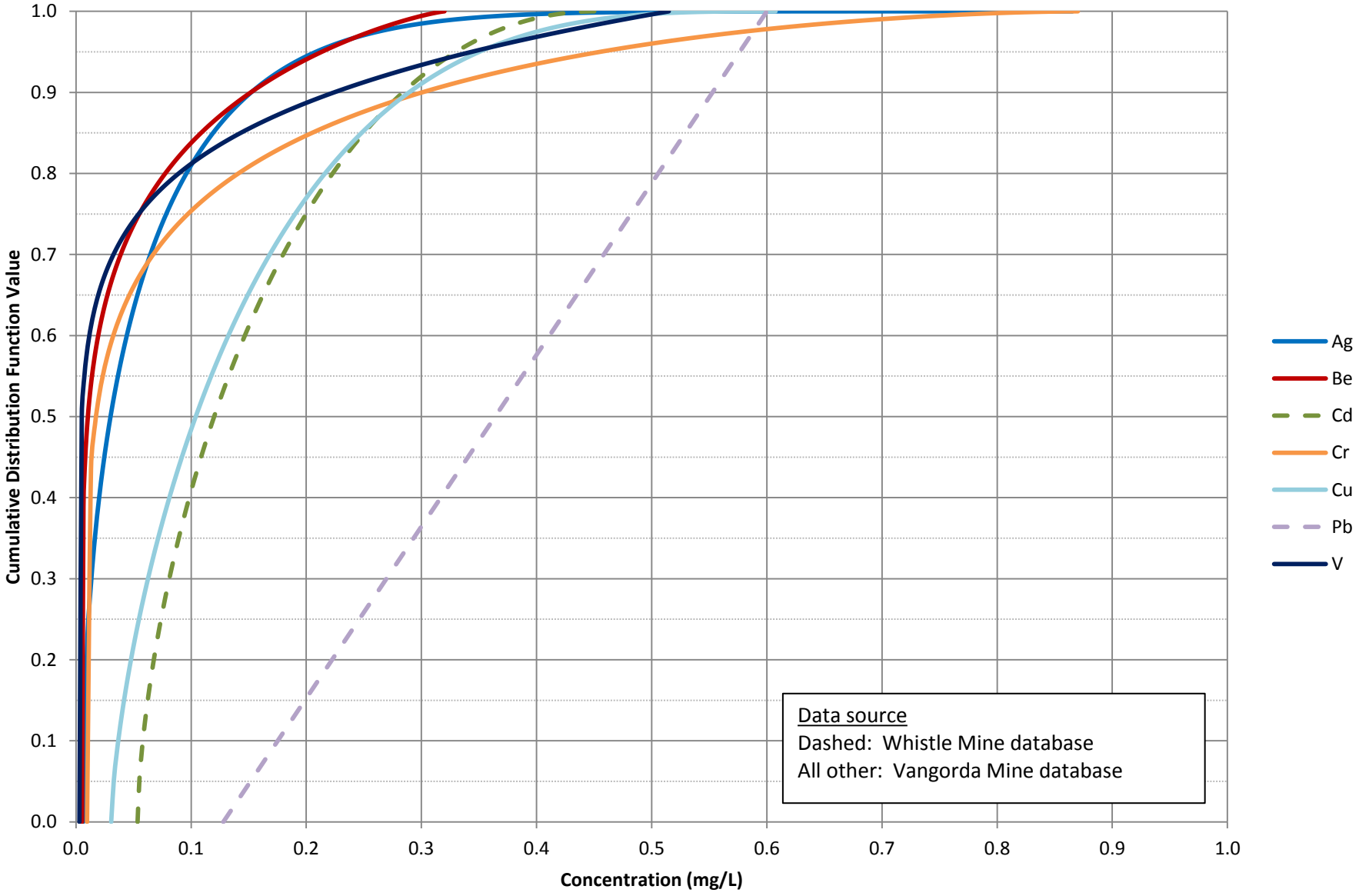
Large Figure 37 **Distributions for Virginia Formation Category 4 Concentration Caps (a)**



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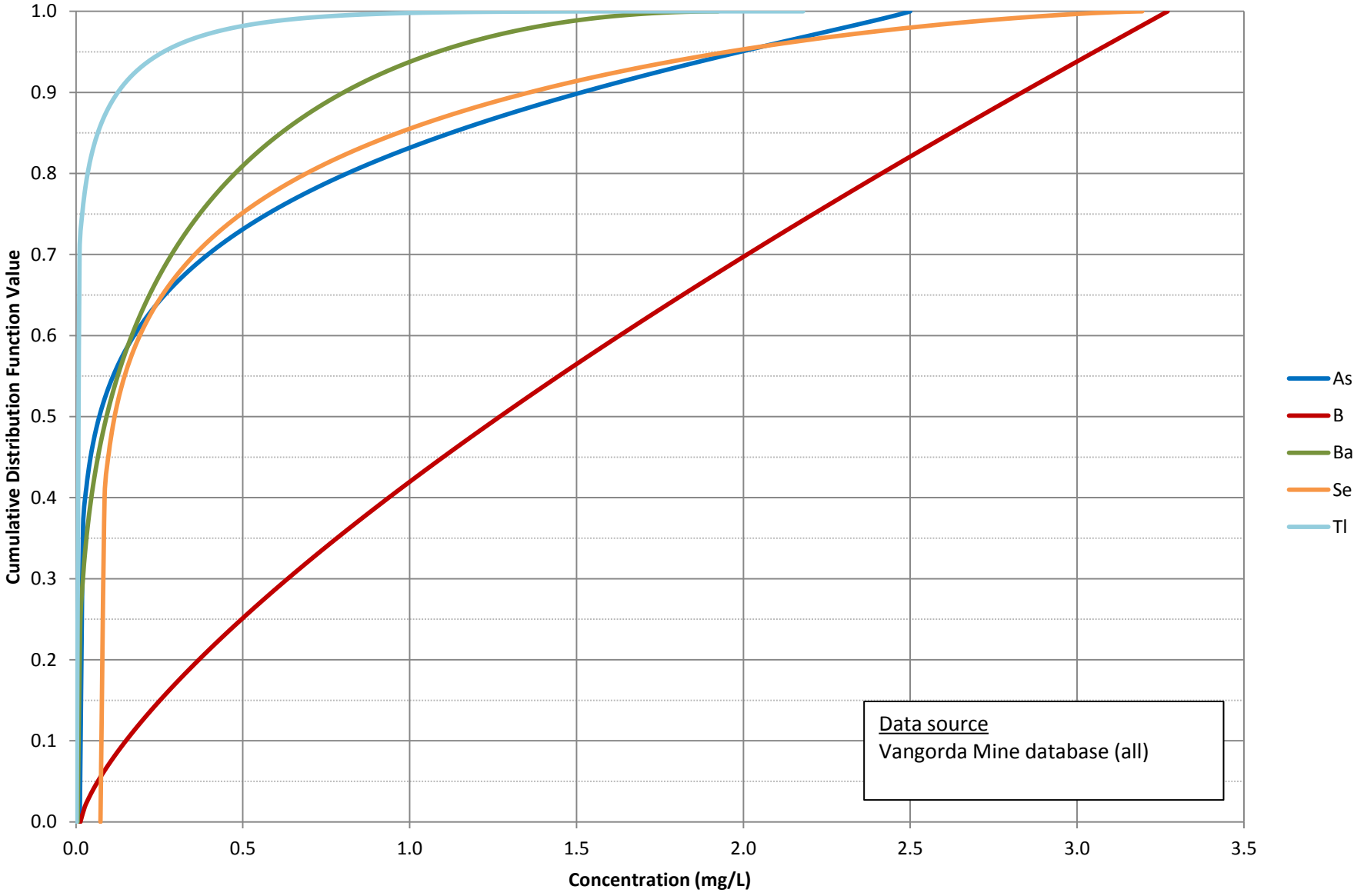
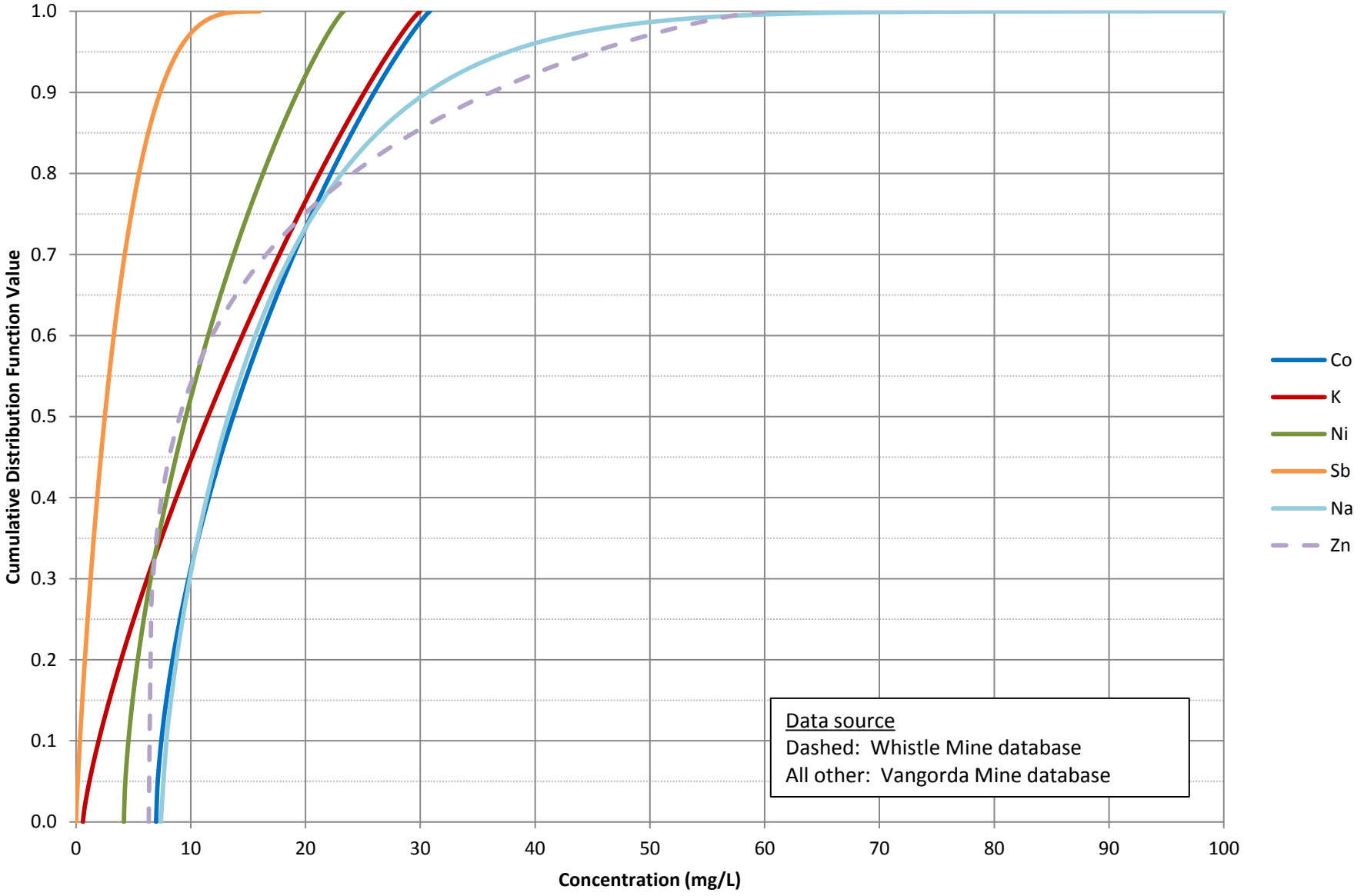
Large Figure 38 **Distributions for Virginia Formation Category 4 Concentration Caps (b)**



Large Figure 39 **Distributions for Virginia Formation Category 4 Concentration Caps (c)**



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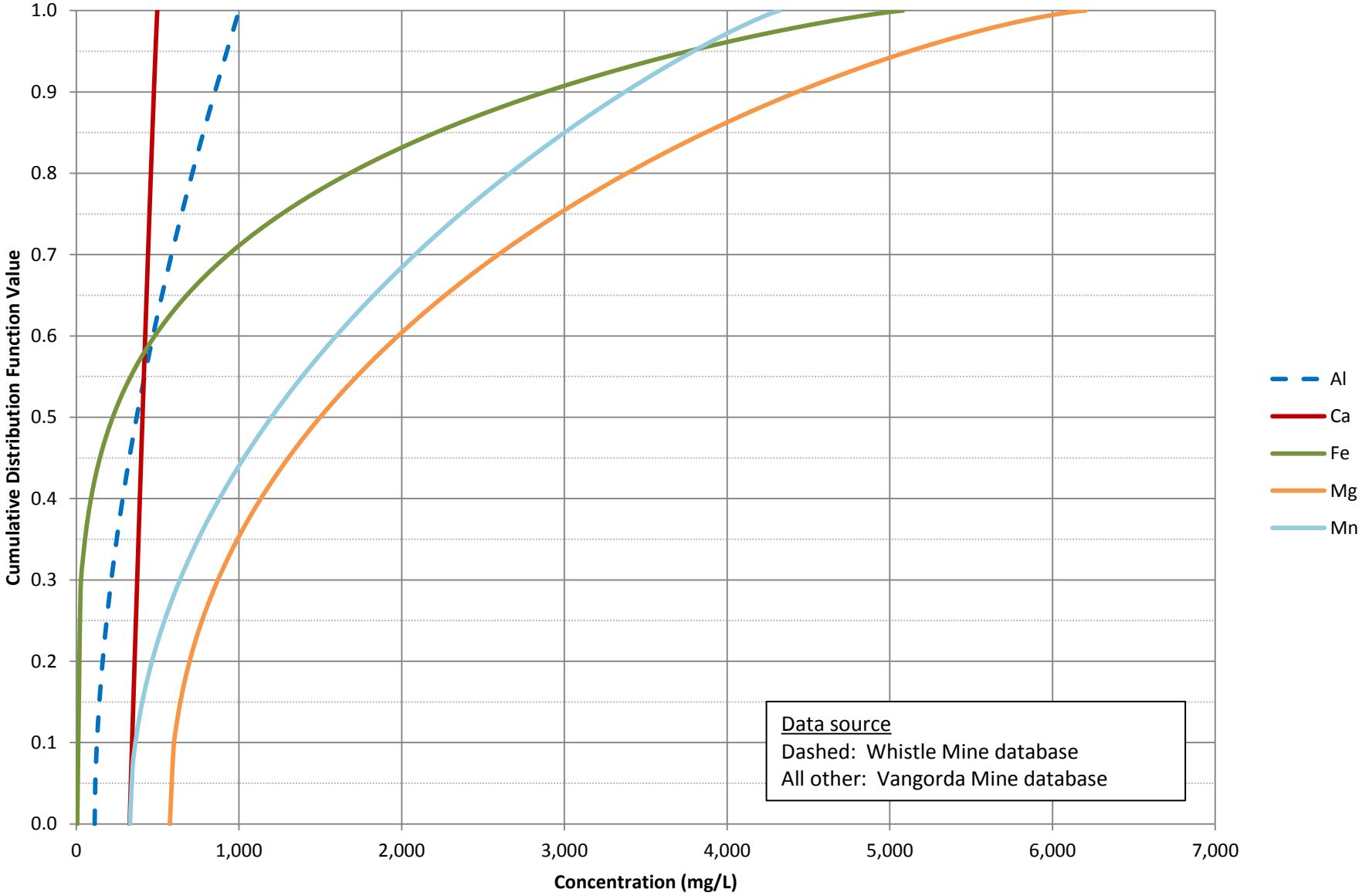
Large Figure 40 Distributions for Virginia Formation Category 4 Concentration Caps (d)



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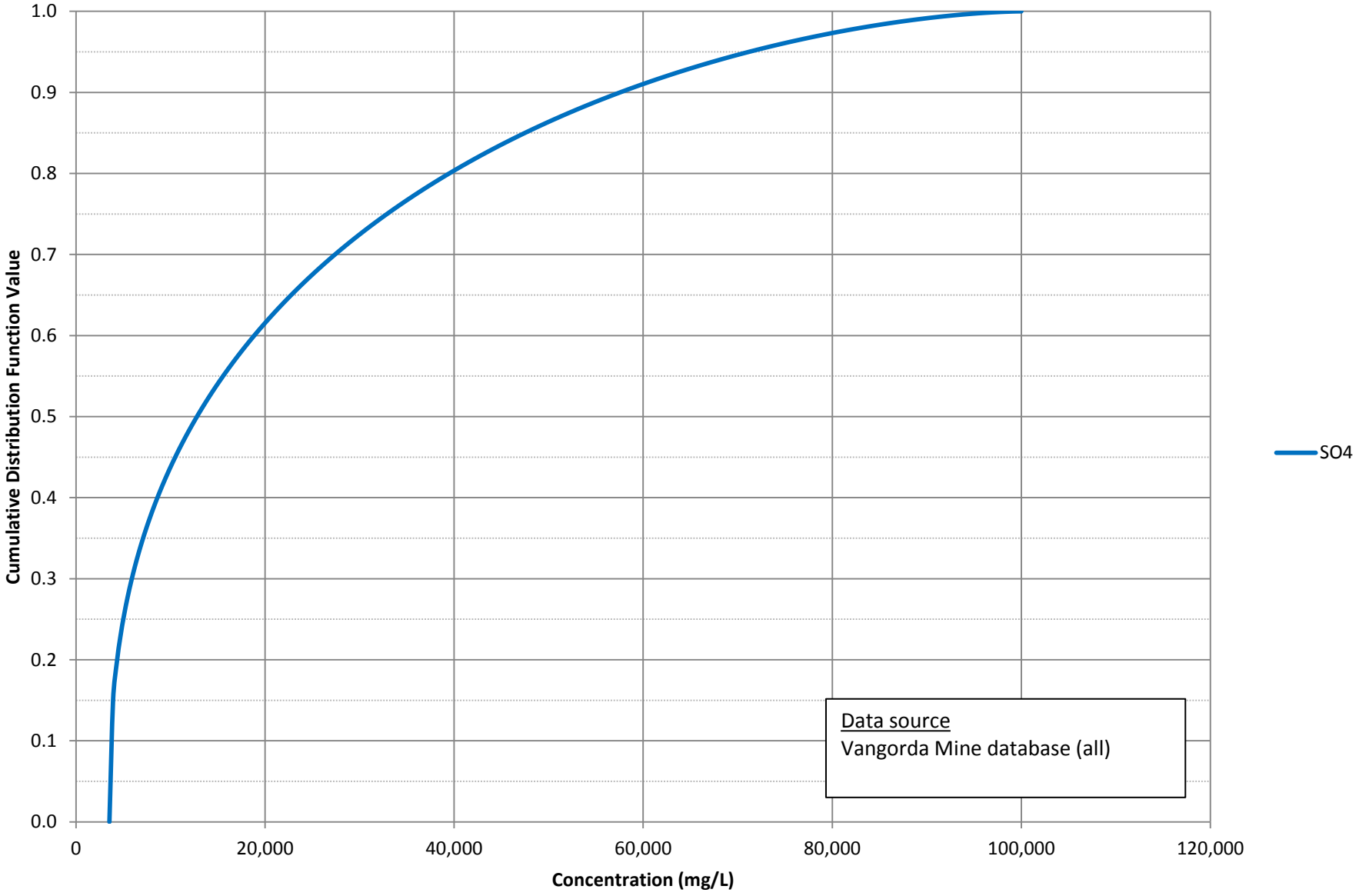
Large Figure 41 **Distributions for Virginia Formation Category 4 Concentration Caps (e)**



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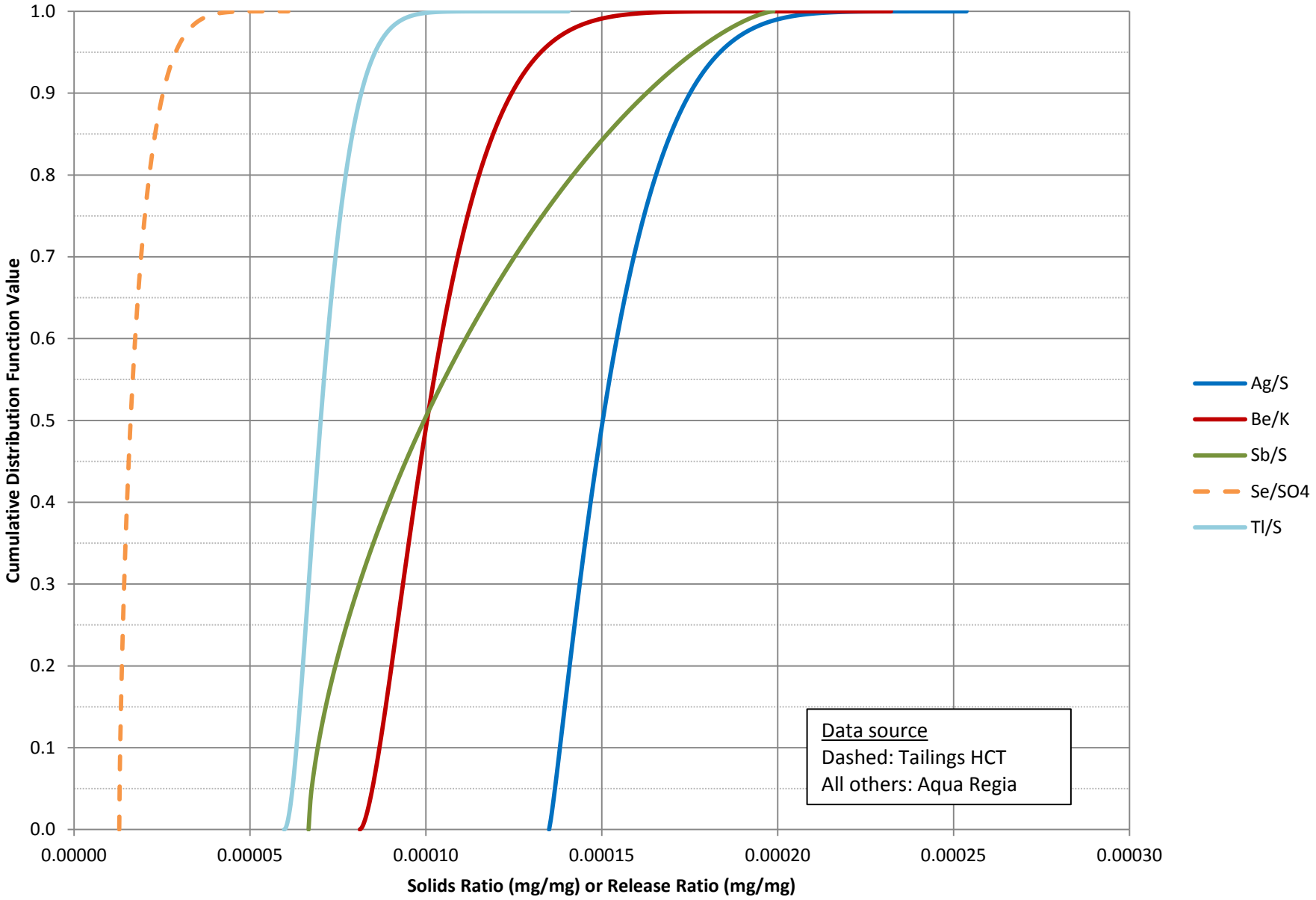
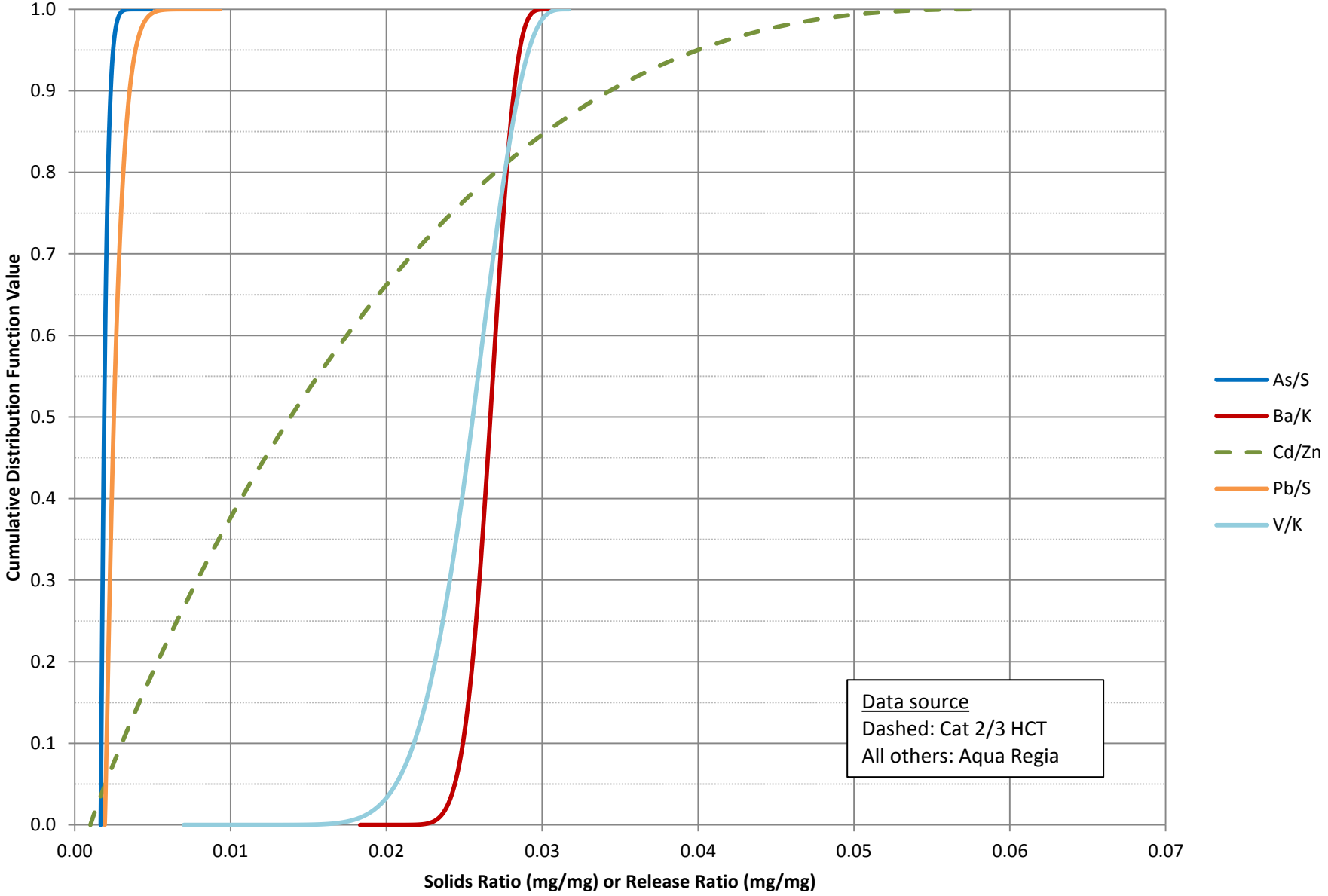
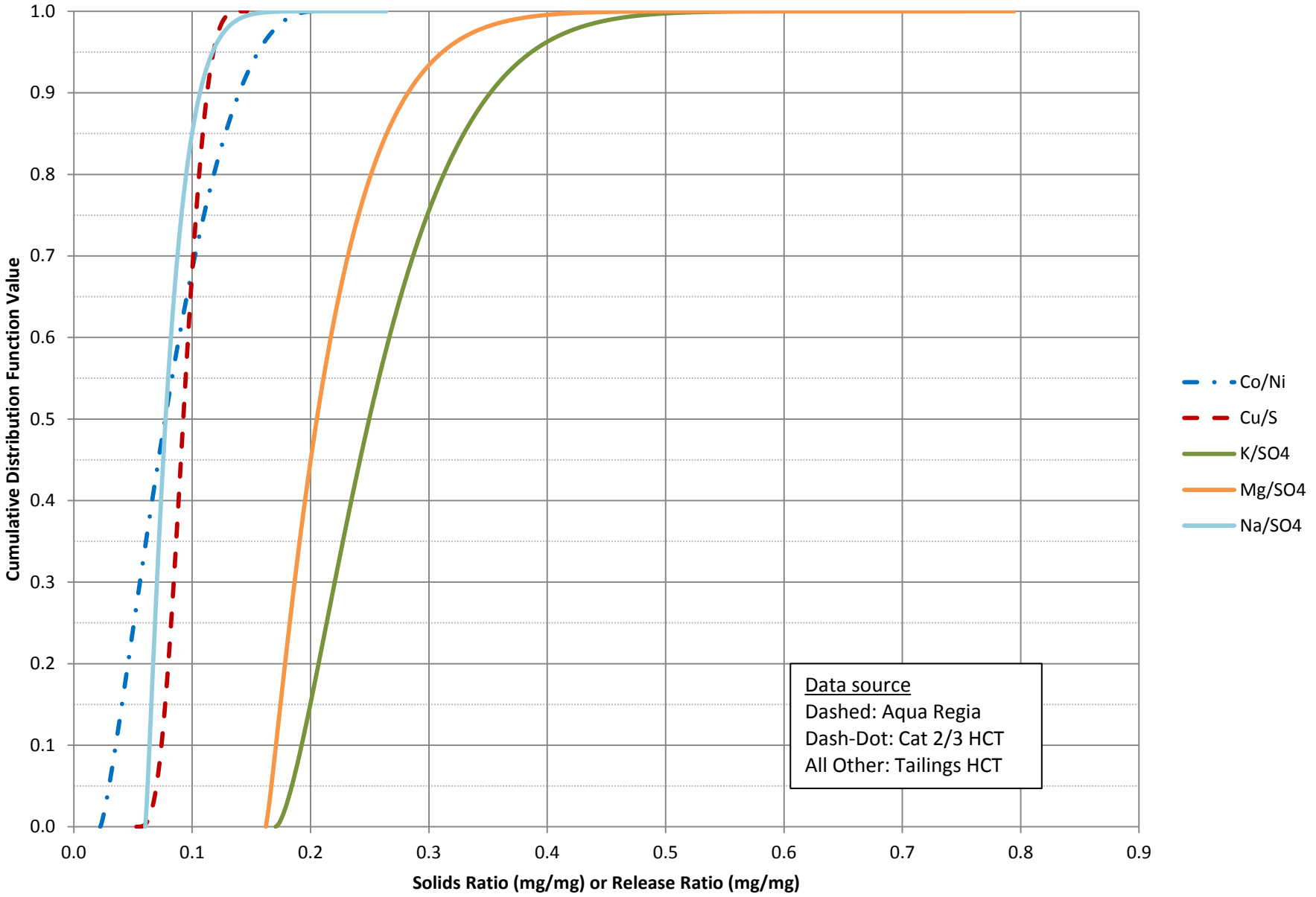
Large Figure 42 Distributions for Flotation Fine Tailings Release (a)

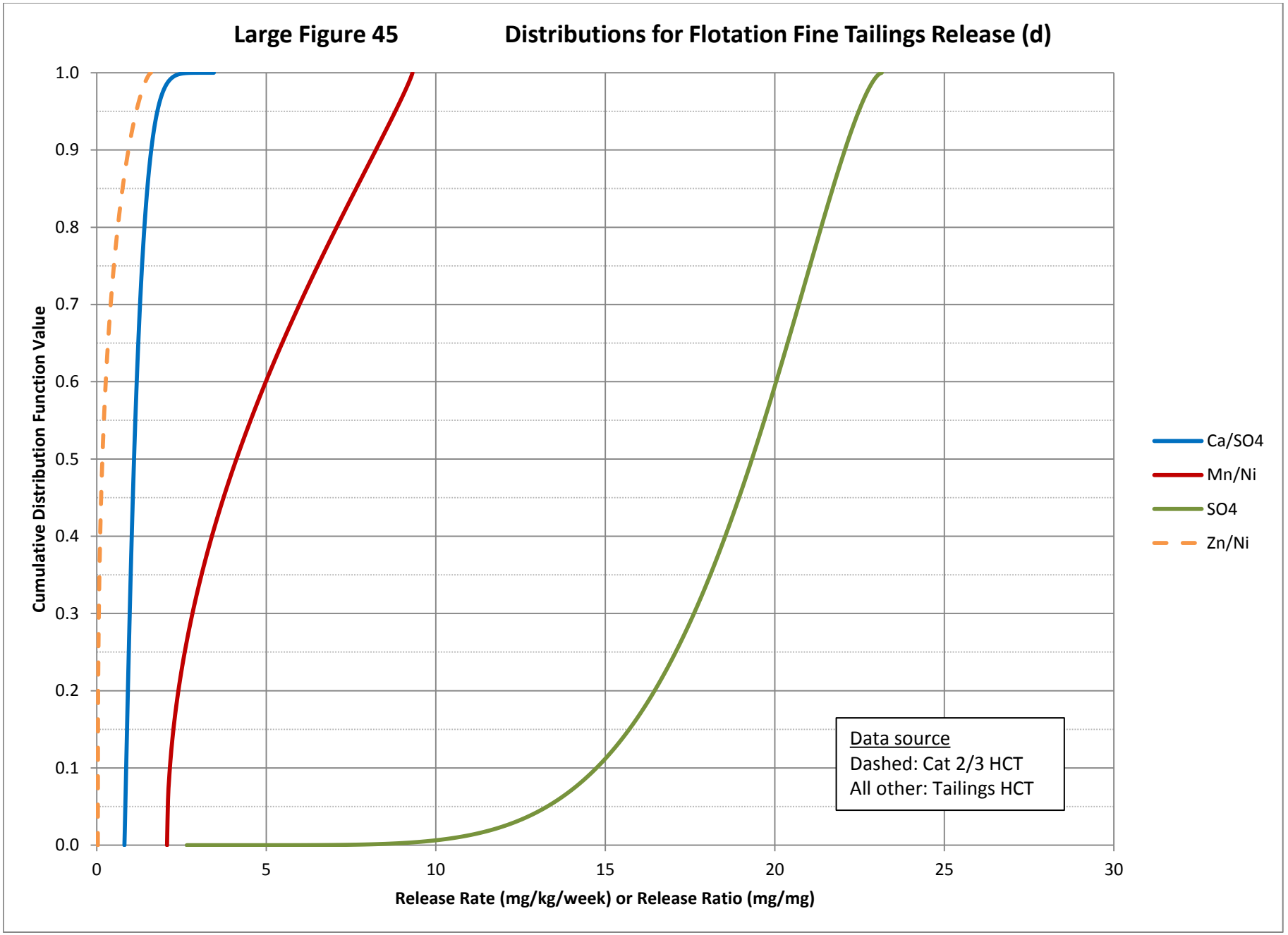


Large Figure 43 Distributions for Flotation Fine Tailings Release (b)

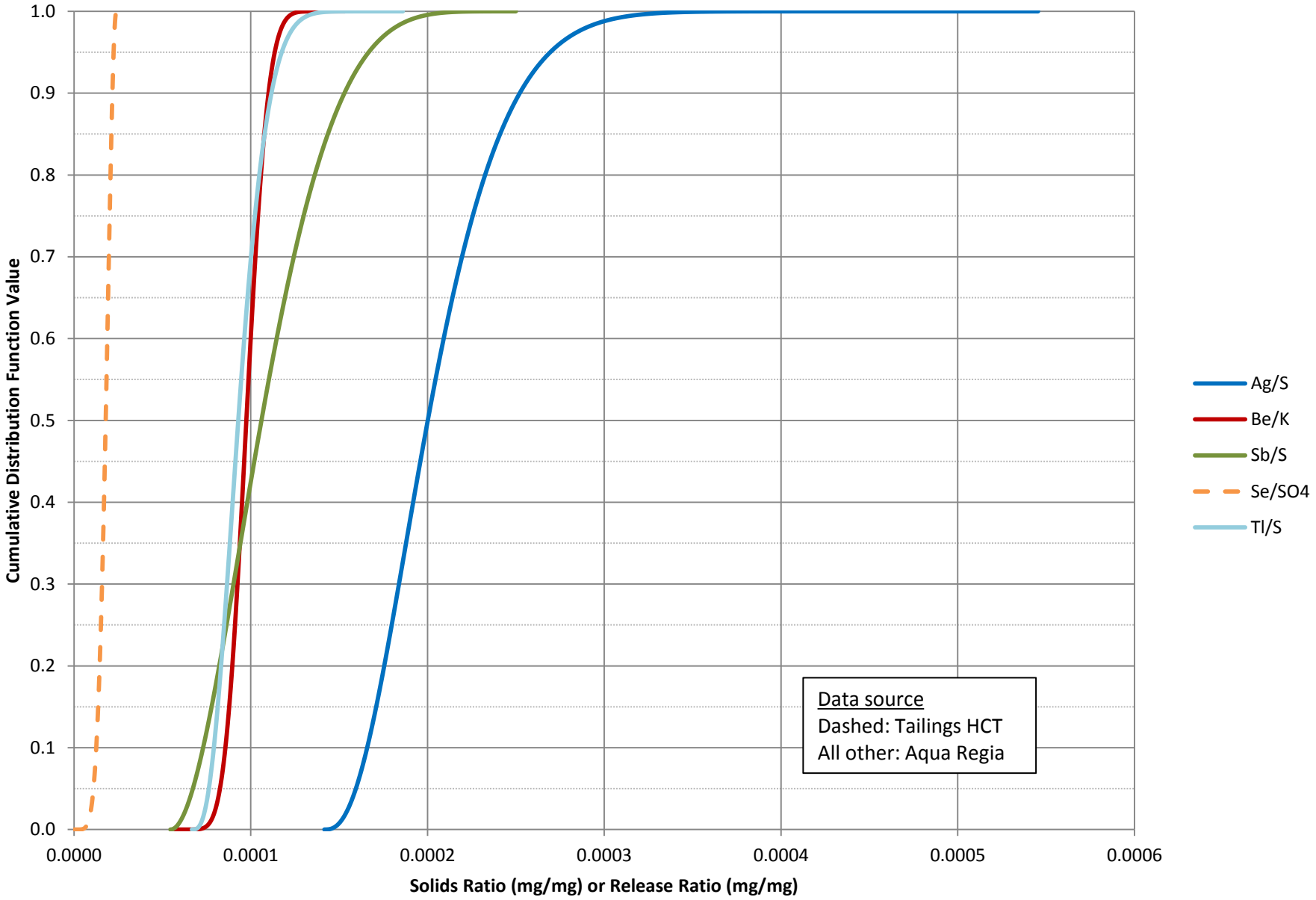


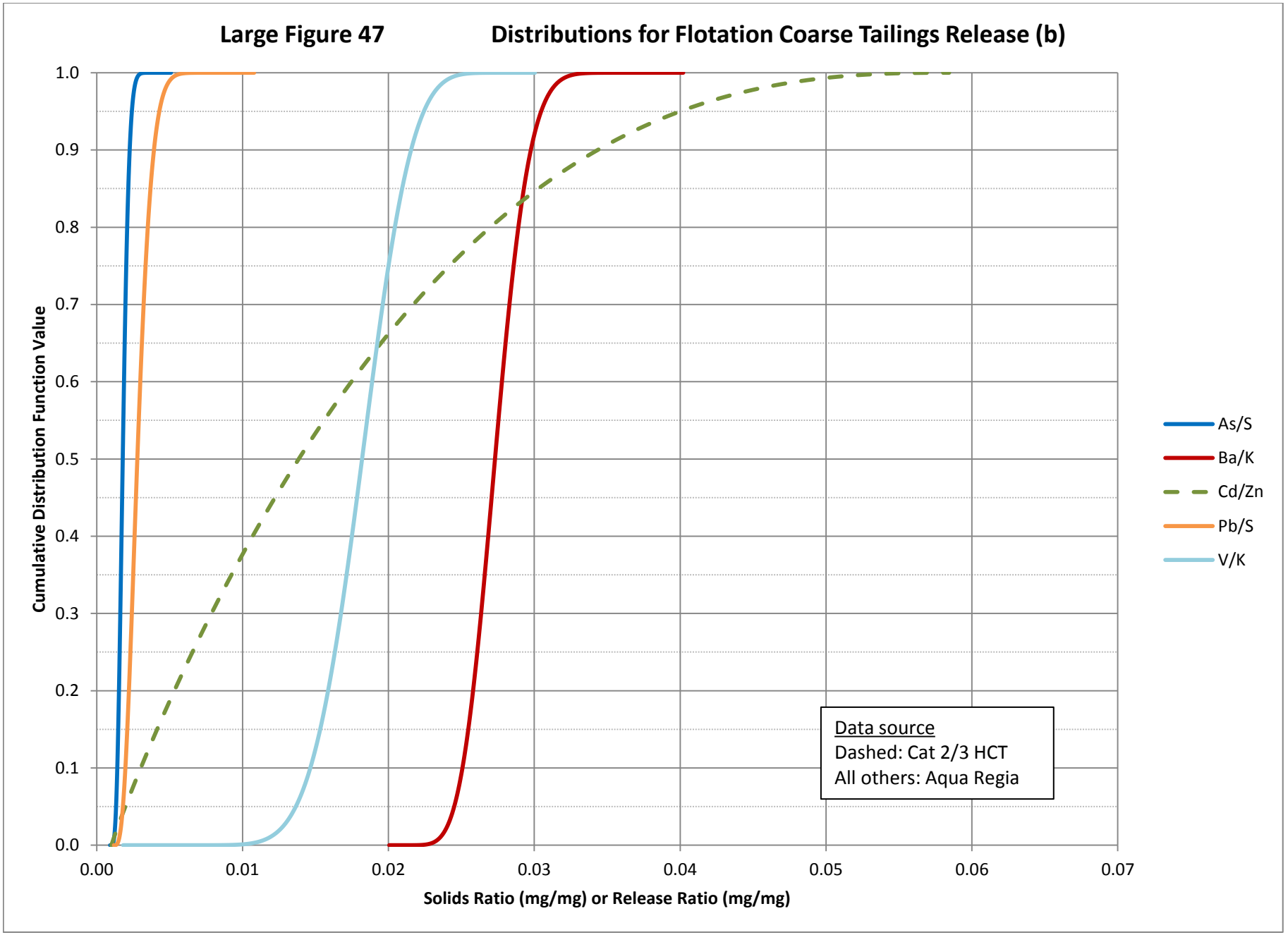
Large Figure 44 Distributions for Flotation Fine Tailings Release (c)

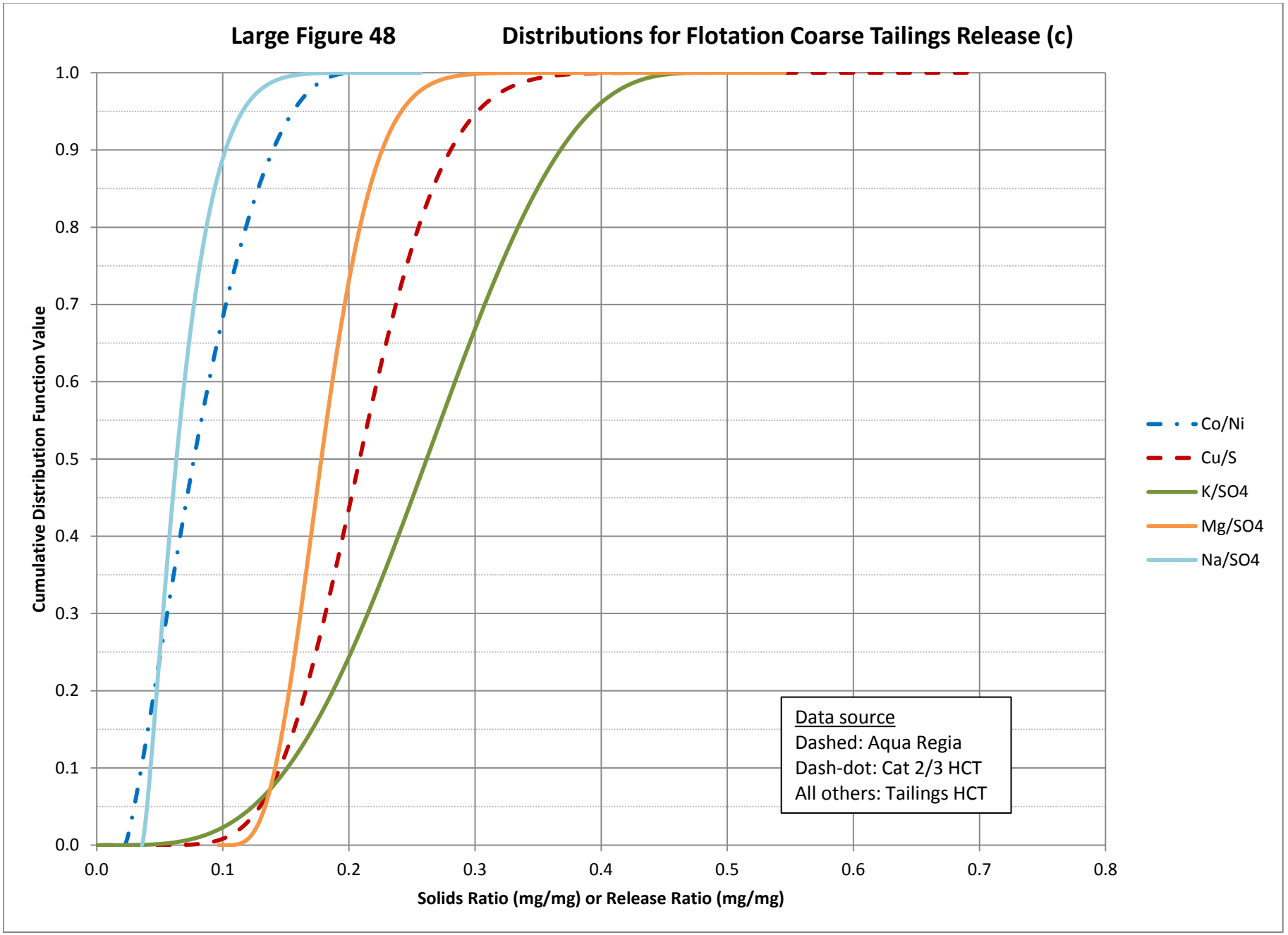


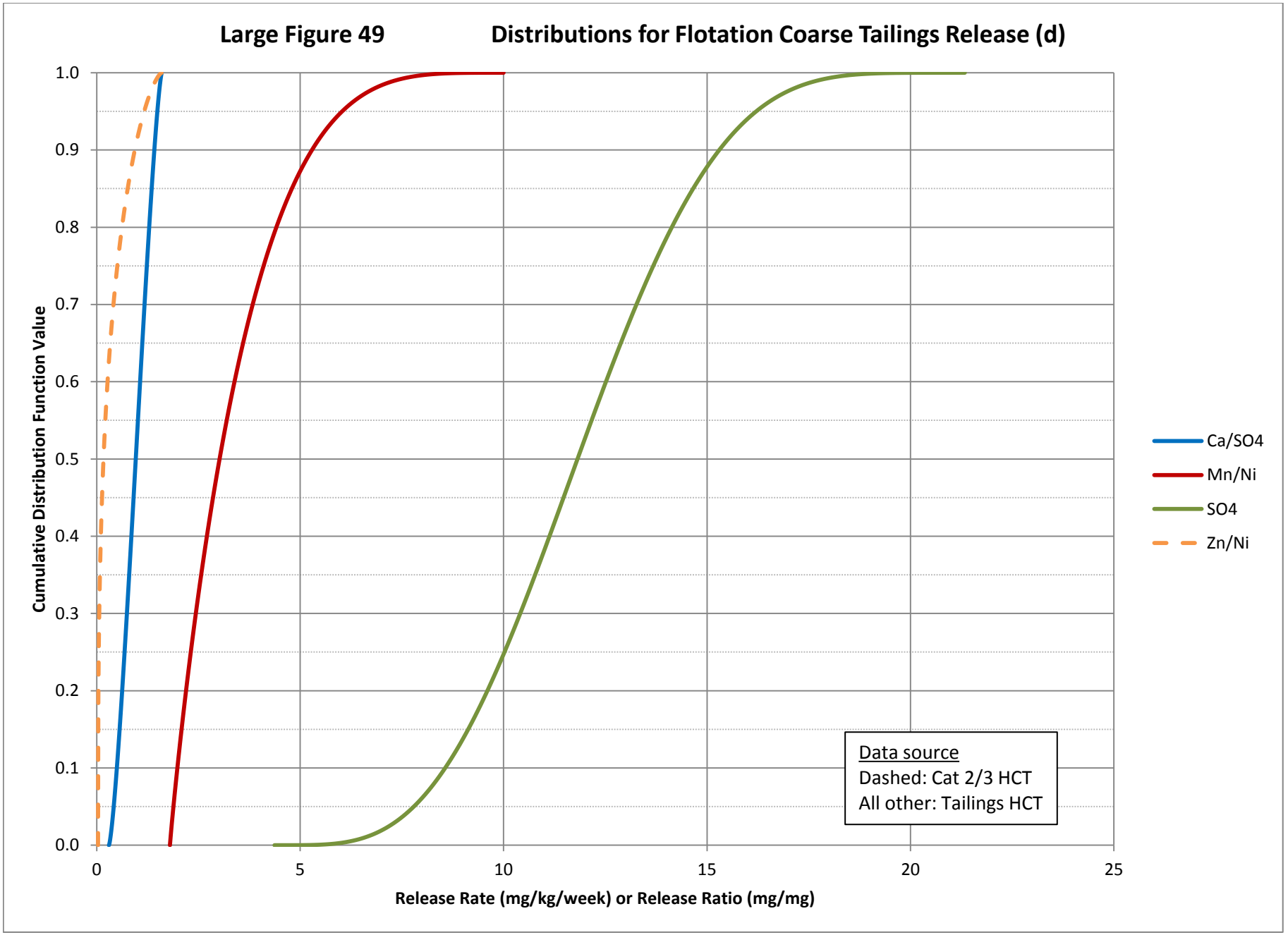

Large Figure 45
Distributions for Flotation Fine Tailings Release (d)
Cumulative Distribution Function Value
1.0
0.9
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
0.0
0
5
10
15
20
25
30
Release Rate (mg/kg/week) or Release Ratio (mg/mg)
Ca/SO4
Mn/Ni
SO4
Zn/Ni
Data source
Dashed: Cat 2/3 HCT
All other: Tailings HCT

Large Figure 46 Distributions for Flotation Coarse Tailings Release (a)

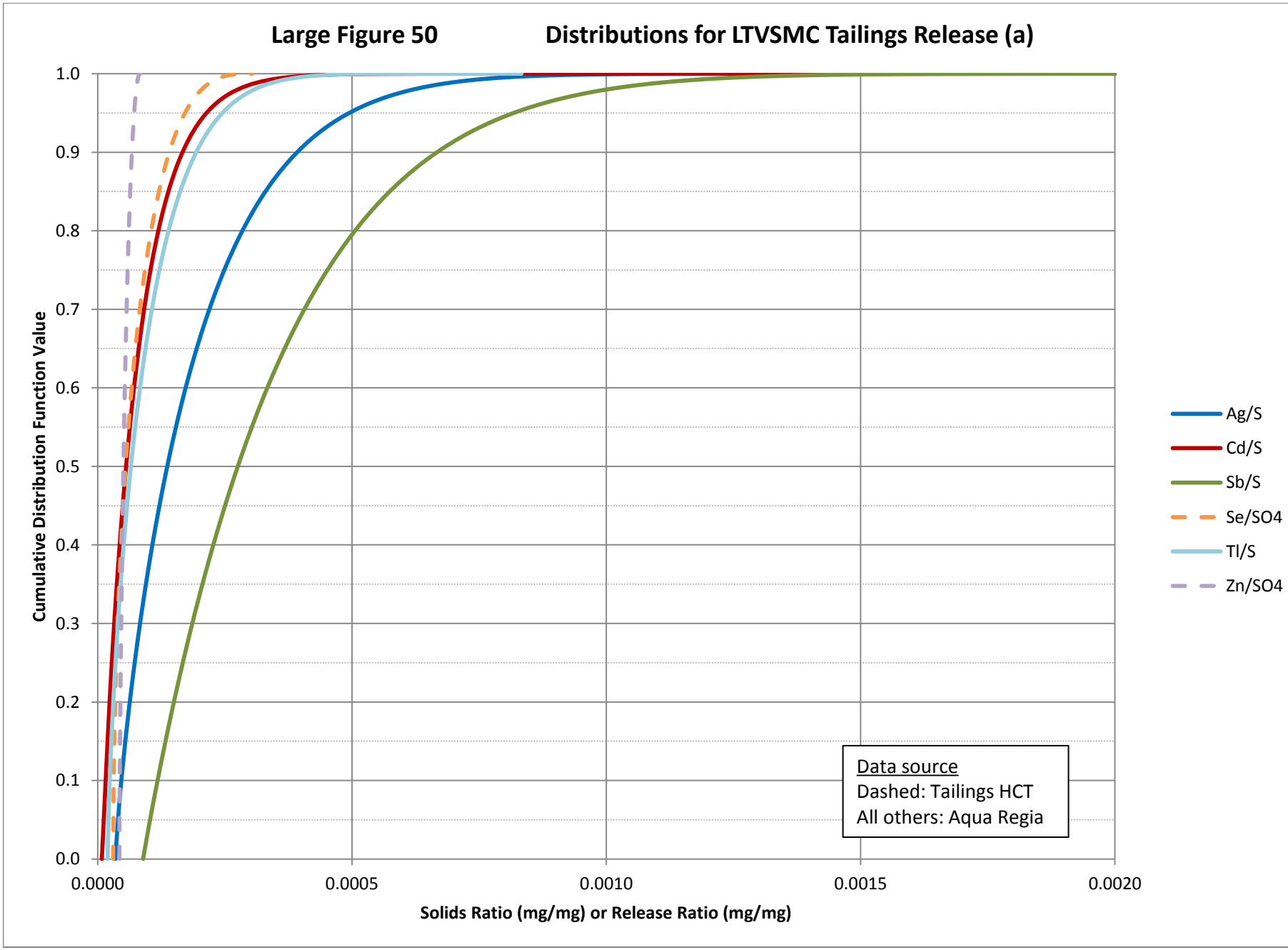



Large Figure 47
Distributions for Flotation Coarse Tailings Release (b)
Cumulative Distribution Function Value
1.0
0.9
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
0.0
0.00
0.01
0.02
0.03
0.04
0.05
0.06
0.07
Solids Ratio (mg/mg) or Release Ratio (mg/mg)
As/S
Ba/K
Cd/Zn
Pb/S
V/K
Data source
Dashed: Cat 2/3 HCT
All others: Aqua Regia


Large Figure 48
Distributions for Flotation Coarse Tailings Release (c)
Cumulative Distribution Function Value
1.0
0.9
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
0.0
0.0
0.1
0.2
0.3
0.4
0.5
0.6
0.7
0.8
Solids Ratio (mg/mg) or Release Ratio (mg/mg)
Co/Ni
Cu/S
K/SO4
Mg/SO4
Na/SO4
Data source
Dashed: Aqua Regia
Dash-dot: Cat 2/3 HCT
All others: Tailings HCT


Large Figure 49
Distributions for Flotation Coarse Tailings Release (d)
Cumulative Distribution Function Value
1.0
0.9
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
0.0
0
5
10
15
20
25
Release Rate (mg/kg/week) or Release Ratio (mg/mg)
Ca/SO4
Mn/Ni
SO4
Zn/Ni
Data source
Dashed: Cat 2/3 HCT
All other: Tailings HCT

Large Figure 50 Distributions for LTVSMC Tailings Release (a)



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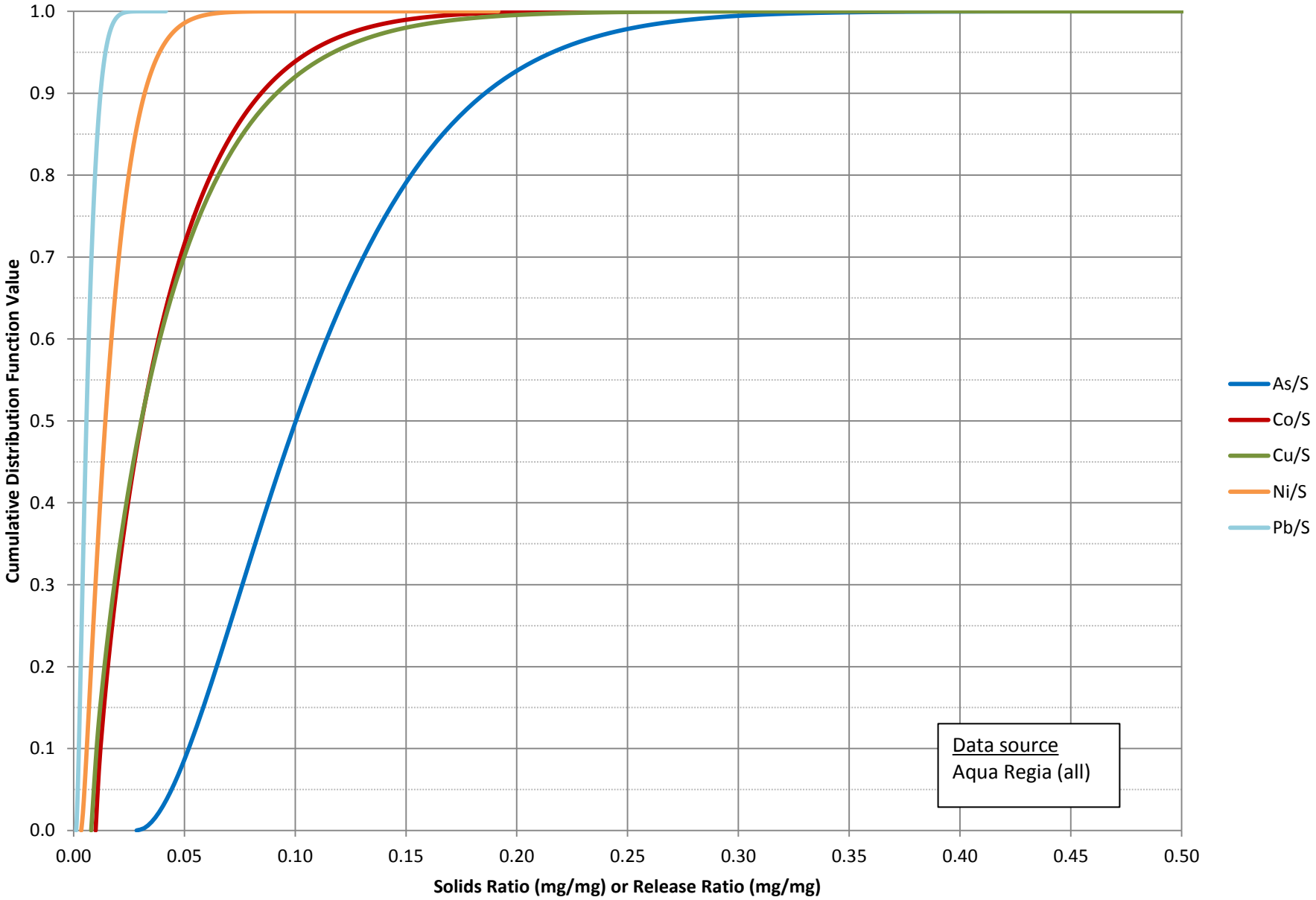
Large Figure 51 Distributions for LTVSMC Tailings Release (b)



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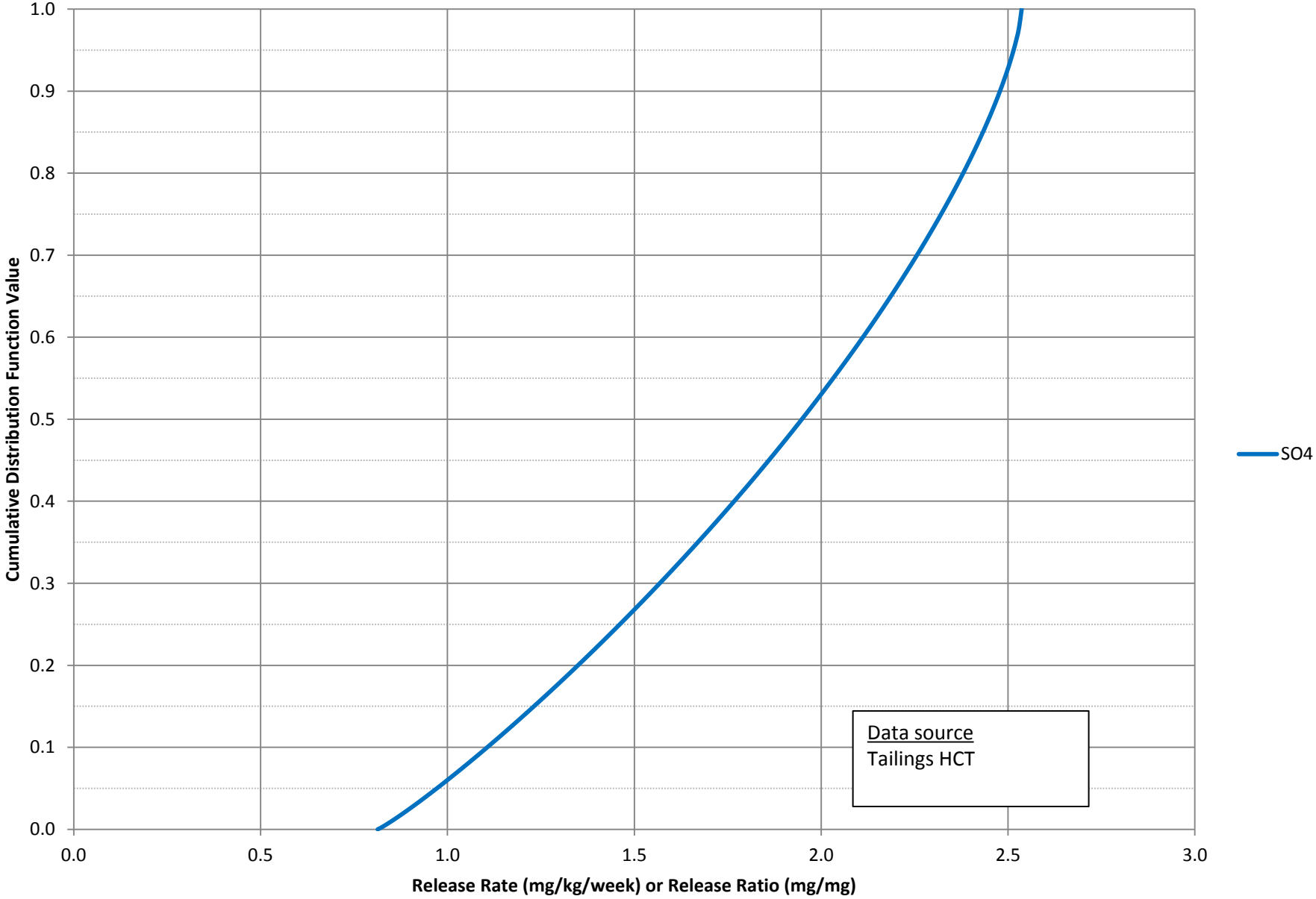
Large Figure 52 Distributions for LTVSMC Tailings Release (c)



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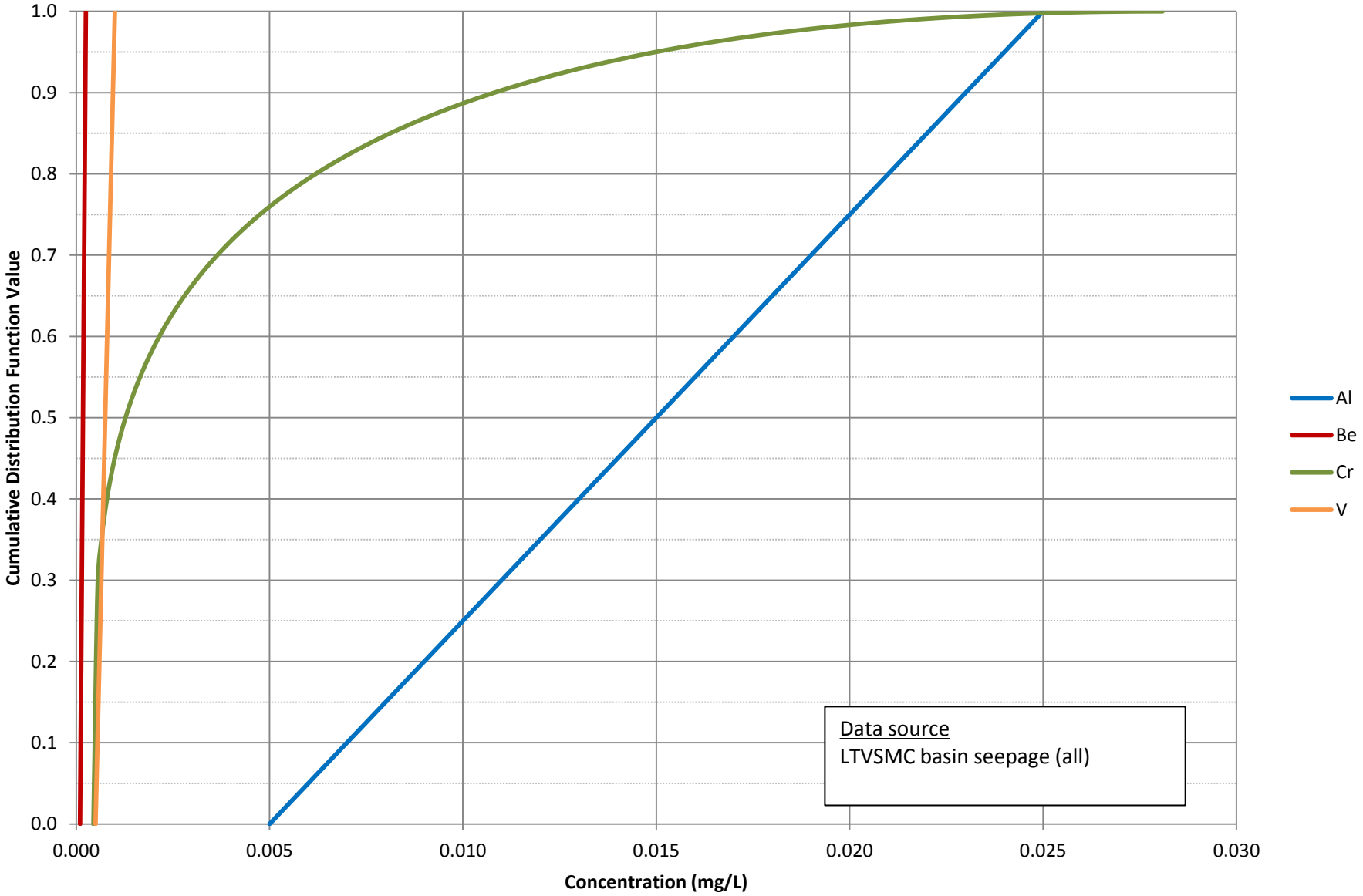
Large Figure 53 **Distributions for LTVSMC Tailings Concentration-Controlled Release (a)**



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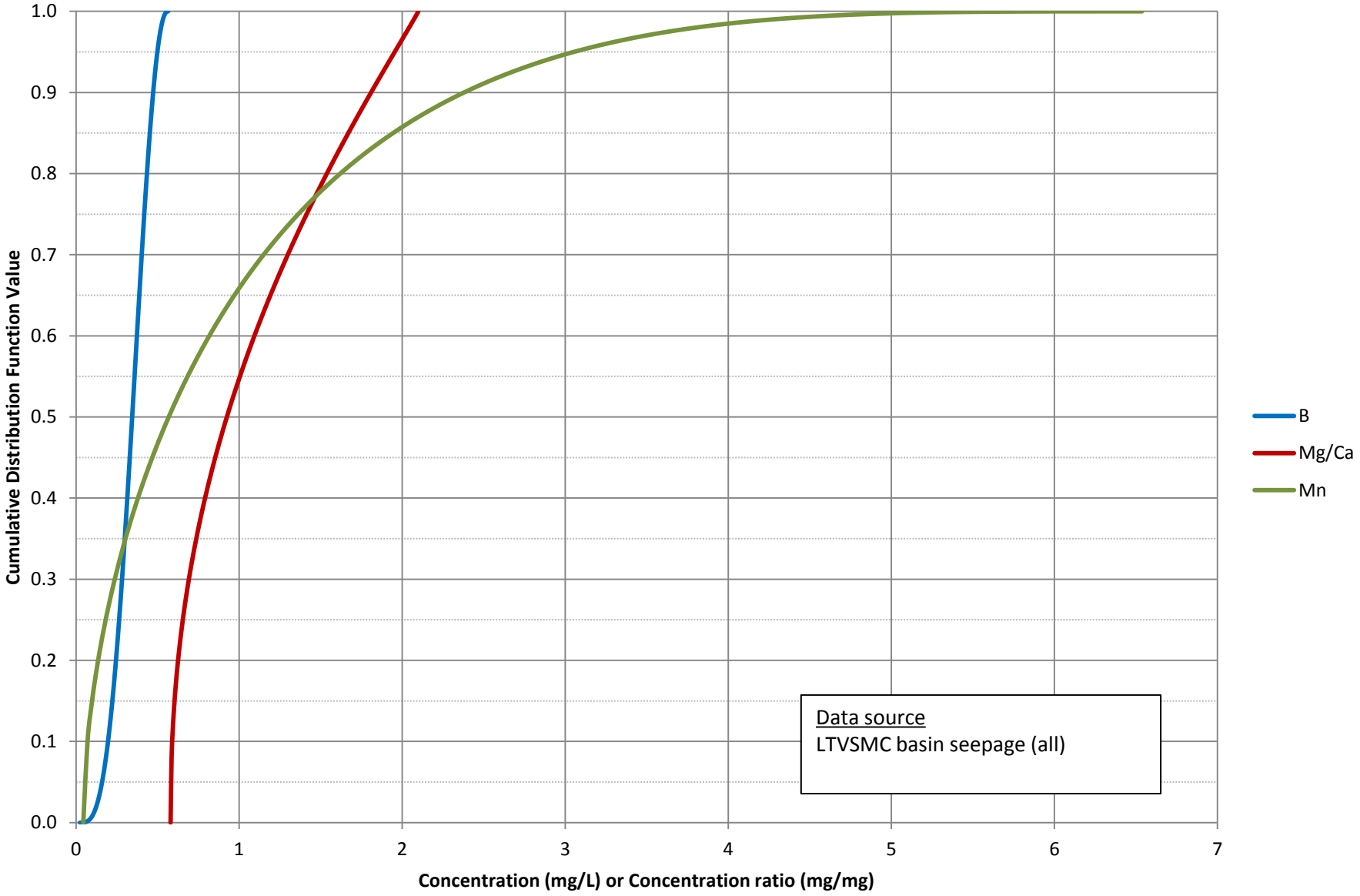
Large Figure 54 **Distributions for LTVSMC Tailings Concentration-Controlled Release (b)**



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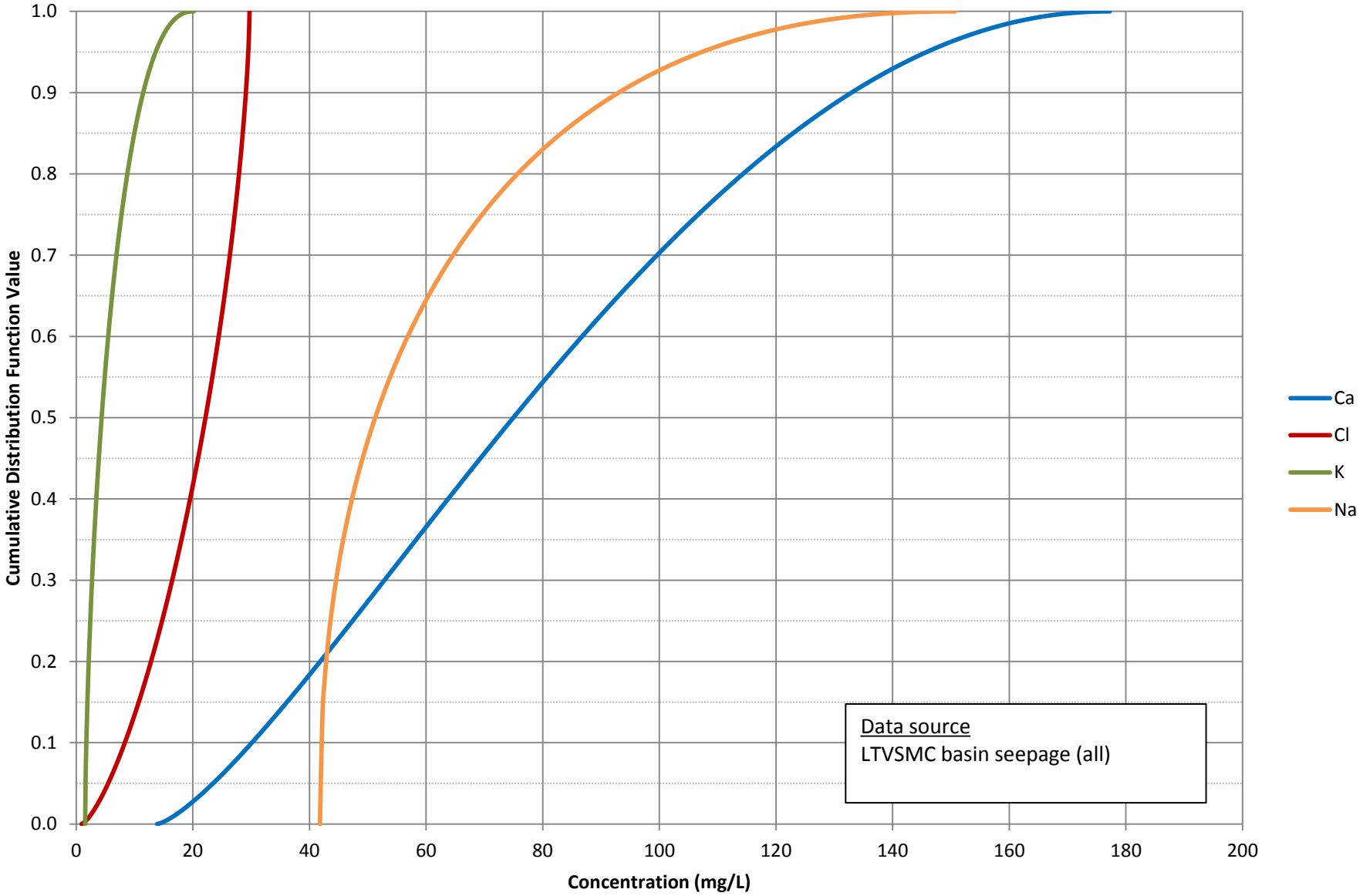
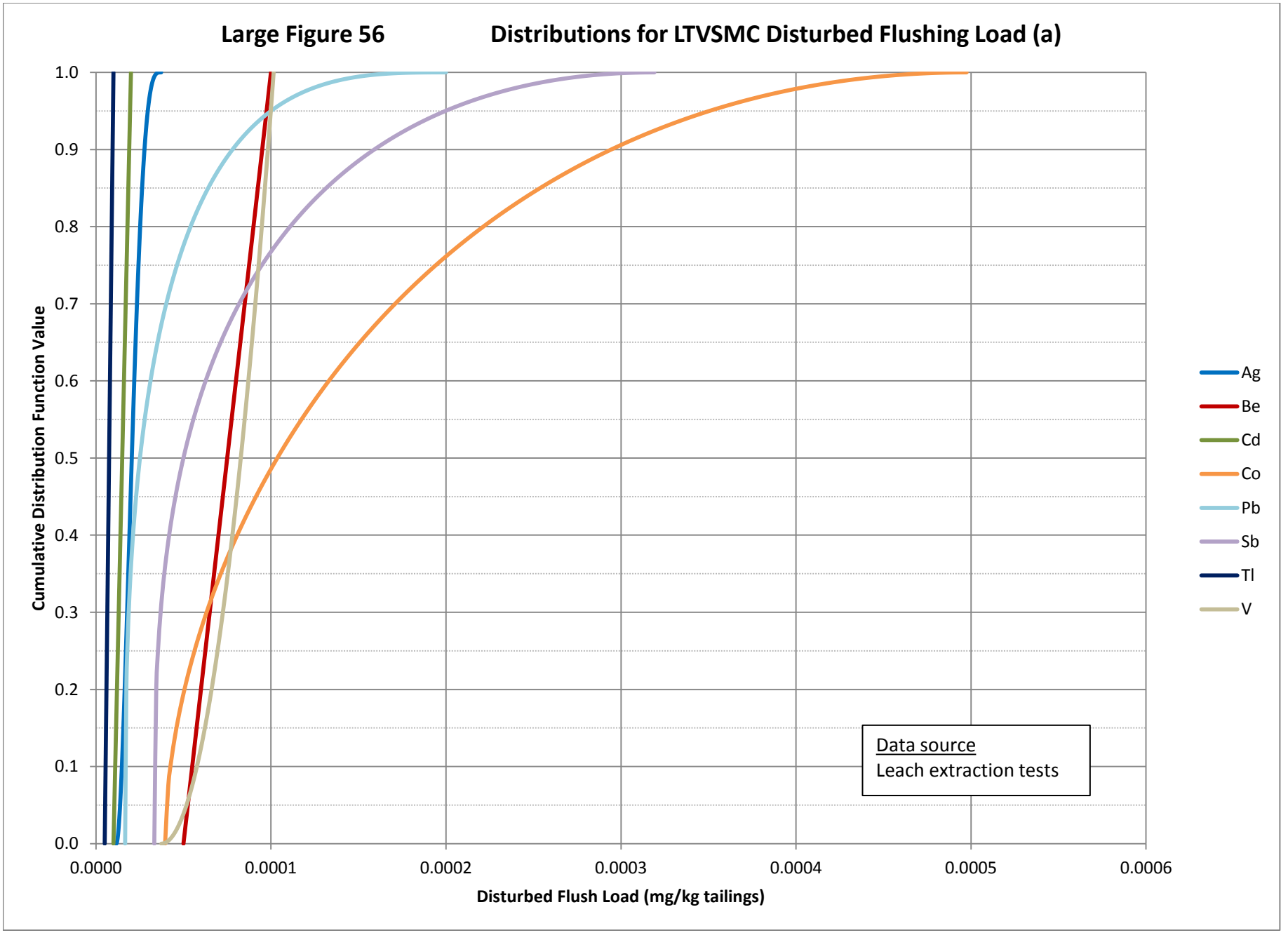
Large Figure 55 **Distributions for LTVSMC Tailings Concentration-Controlled Release (c)**



Large Figure 56 Distributions for LTVSMC Disturbed Flushing Load (a)



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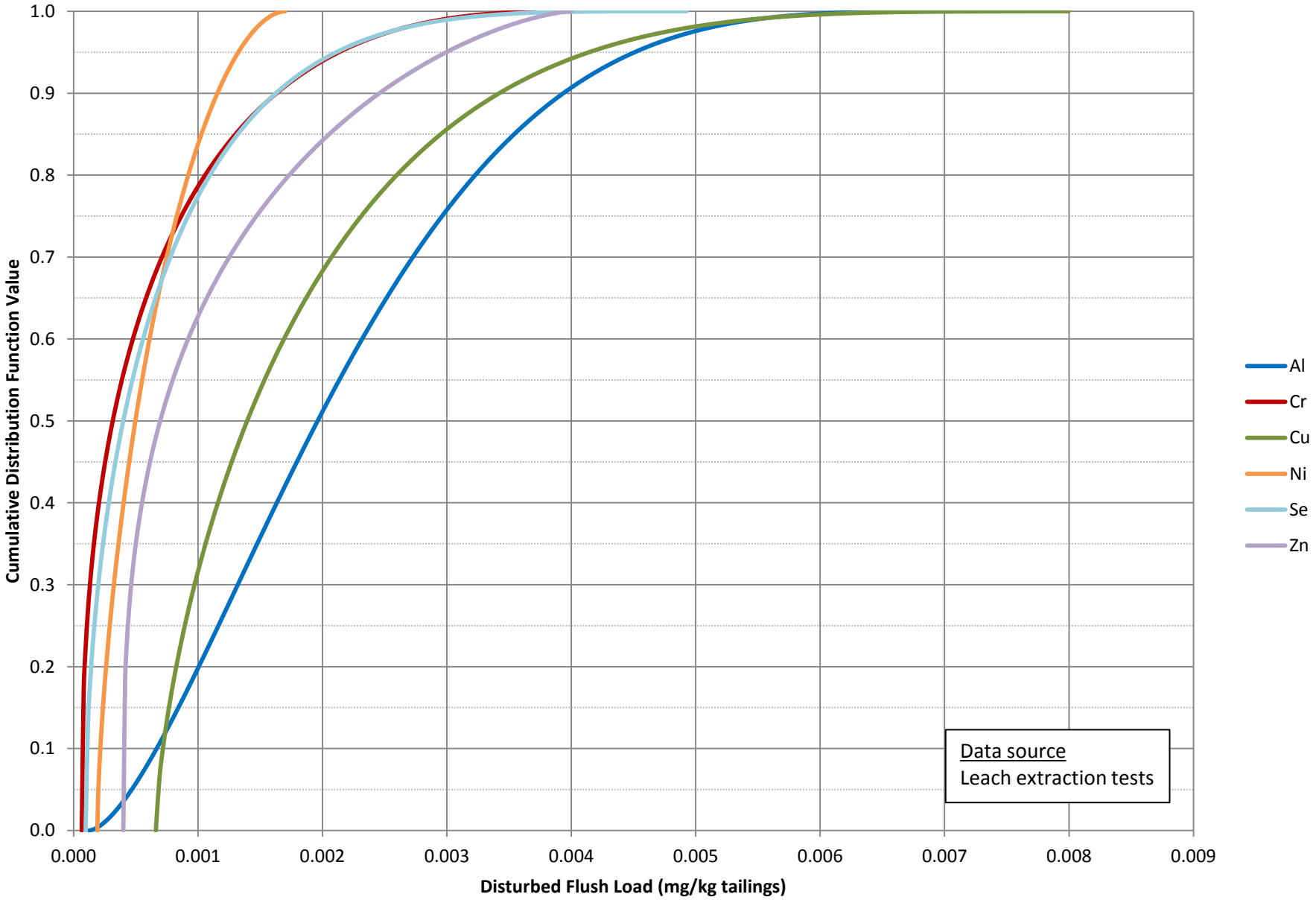
Large Figure 57 Distributions for LTVSMC Disturbed Flushing Load (b)



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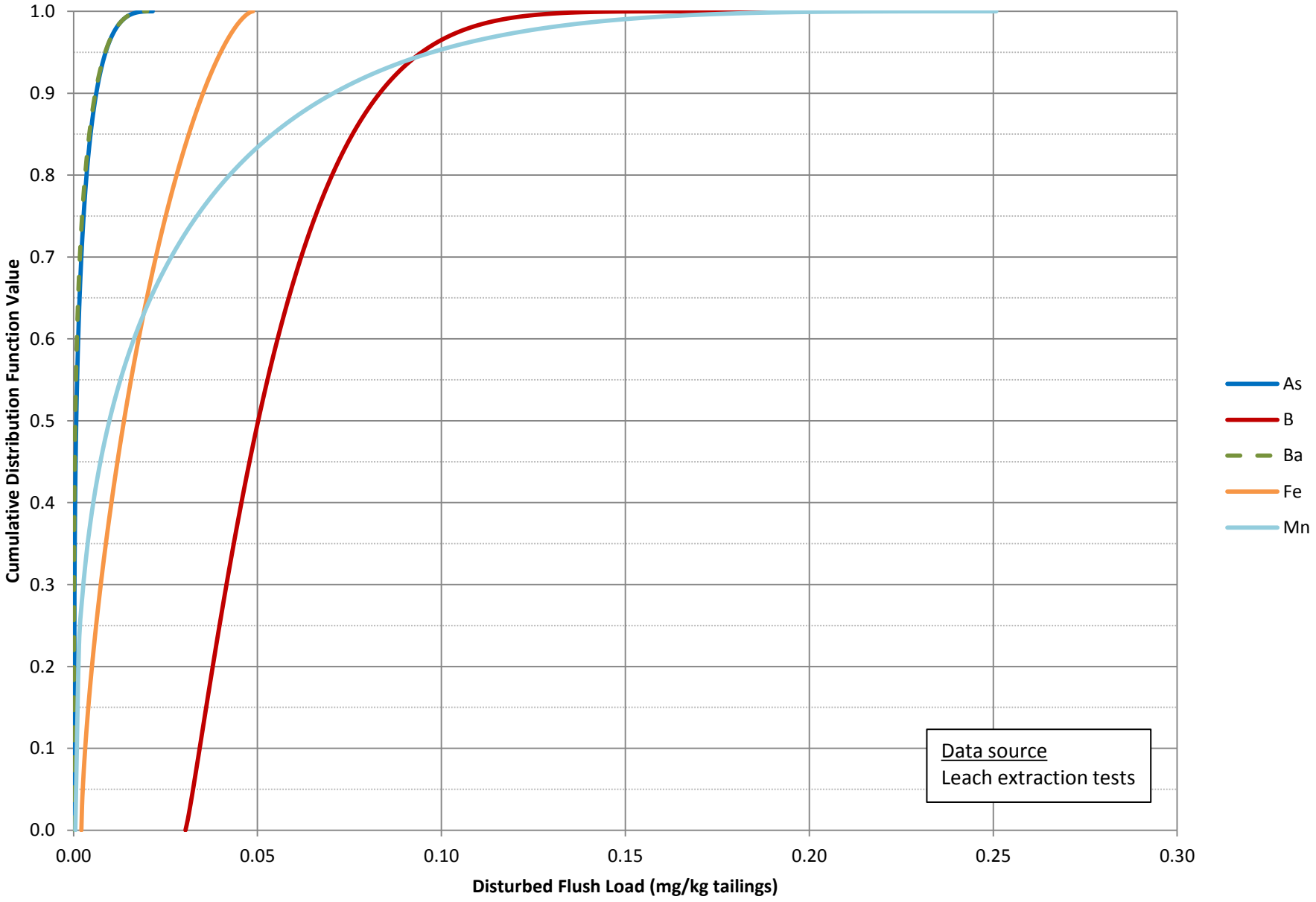
Large Figure 58 Distributions for LTVSMC Disturbed Flushing Load (c)



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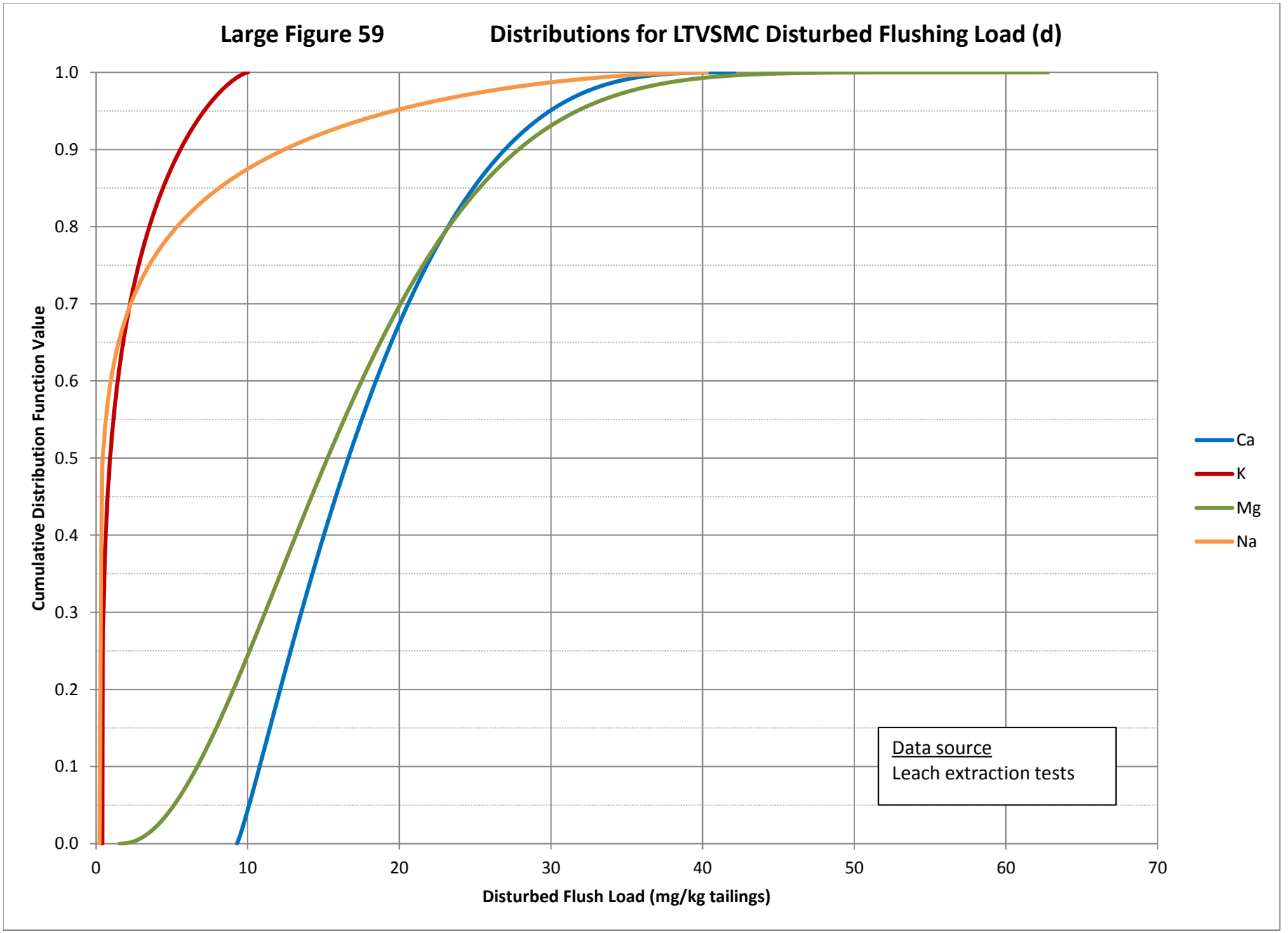

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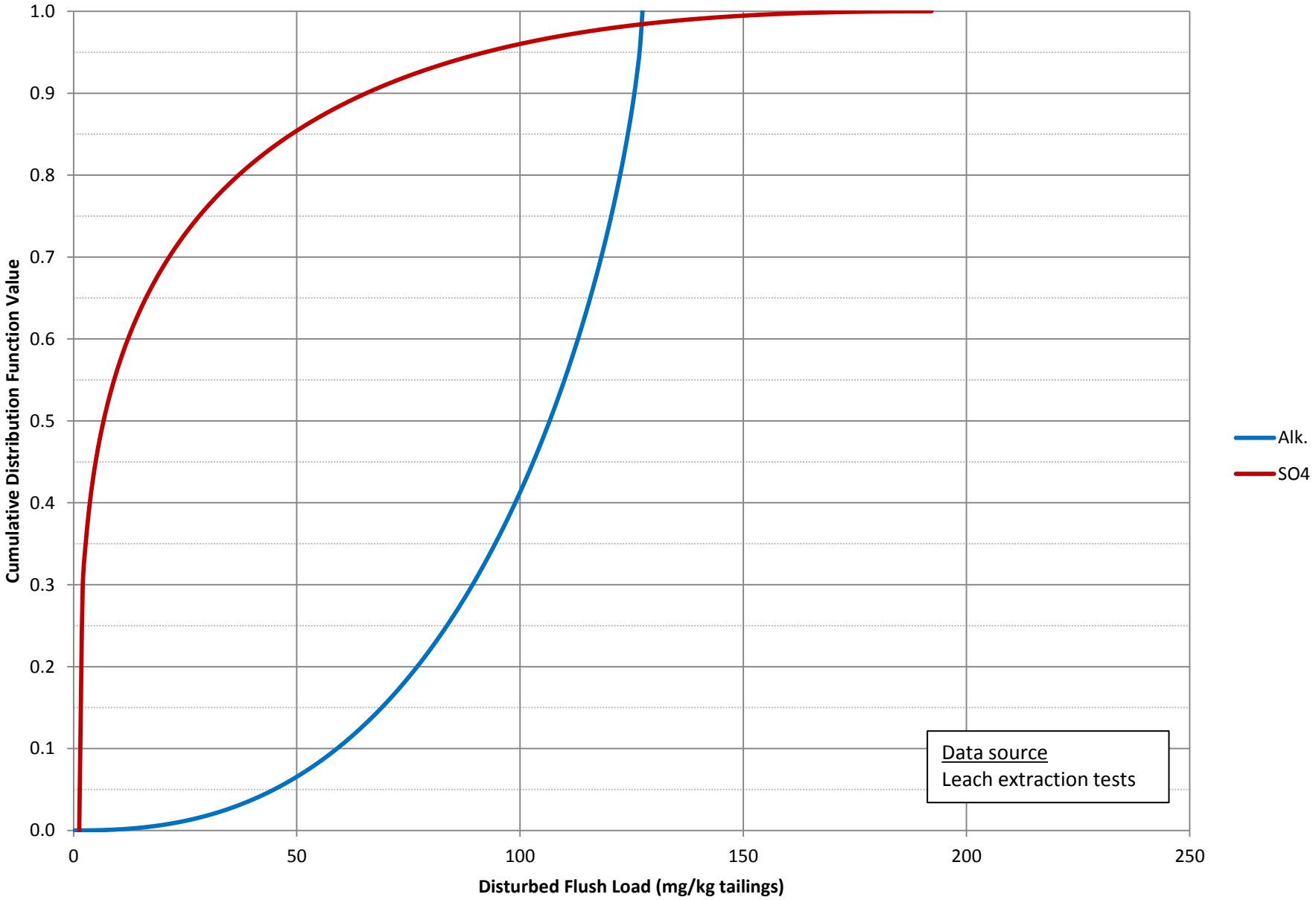
Large Figure 60 Distributions for LTVSMC Disturbed Flushing Load (e)



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Attachments

Attachment A

Water Quality Modeling for Waste Rock and Pit Walls

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Memo

To:	Jim Scott, PolyMet Peter Hinck, Barr	**Date:**	March 15, 2011
cc:		**From:**	Stephen Day
Subject:	Water Quality Modeling for Waste Rock and Pit Walls NorthMet Project – DRAFT 4	**Project #:**	1UP005.001

1 Background

Water quality modeling for waste rock involves consideration of the coupled processes of contaminant release by weathering of minerals and dissolution of the resulting weathering products by contact waters. The finite solubility of secondary minerals typically limits their dissolution so that leaching rates are lower than release rates on average.

The underlying input for these calculations has been rates indicated by humidity cells scaled to reflect site conditions. To this point, it has been assumed that rates of leaching indicated by humidity cells are largely free of solubility limitations due to the high liquid-to-solid ratios used in laboratory kinetic tests in order to obtain sufficient leachate for analysis. Data obtained from the experiments performed for the NorthMet Project demonstrate this is not the case. For example, nickel leaching can increase by orders of magnitude even as the oxidation and weathering of the sources of nickel decrease. Previous modeling did not fully separate contaminant release from solid phases[1] from solubility effects for the rates used in inputs to water quality calculations.

At meetings with MDNR representatives in St. Paul, MN on January 12th and 13th, 2011, SRK presented an approach to allow the processes of release and leaching to be clearly modeled separately. This memorandum provides details of the proposed approach.

2 Calculation of Solid Phase Release Rates

2.1 Conceptualization

Potential contaminants are released from their primary mineralogical hosts by weathering or dissolution processes. For example:

- Metals contained in sulfide minerals are released by oxidation to form metal sulfates and hydroxides.
- Metals contained in silicate minerals are released by carbonic acid weathering to form metal silicates and hydroxides.
- Metals contained in silicates can also be released when they are dissolved by sulfuric acid produced by sulfide oxidation.

[1] In this document, "Release Rates" refers to the conversion of potential contaminants in primary solid phases into solid secondary weathering products which are then available for leaching. "Leaching Rates" refers to the transfer of potential contaminants from the secondary weathering products to the aqueous phase.

In each of the above examples, major mineral components are released along with the trace levels of potential contaminants. In the case of pyrrhotite (iron sulfide), iron and sulfur are released along with elements such as cobalt and nickel. It is reasonable to assume that the iron, cobalt and nickel will be released from the mineral in proportion to the metal to sulfur ratios in pyrrhotite. Since sulfate is readily soluble, pyrrhotite oxidation can be quantified by measuring sulfate release (R_S); however, the metals may form less soluble oxides and hydroxides particularly under non-acidic conditions. The rate of release of metals (R_M) to secondary minerals in response to oxidation is therefore more appropriately represented by:

$$R_M = R_S \cdot \frac{[M]_{solid}}{[S]_{solid}}$$

where $[M]_{solid}$ and $[S]_{solid}$ are the metal and sulfur concentration in the solid pyrrhotite, respectively. Similarly, dissolution of olivine can be expected to release magnesium, silica and trace elements. Magnesium is expected to be readily soluble but trace elements may form less soluble secondary minerals. Release of metals from olivine can therefore be represented by:

$$R_{metal} = R_{Mg} \cdot \frac{[M]_{solid}}{[Mg]_{solid}}$$

To implement these concepts for modeling, the following components are needed:

- Determination of major mineral hosts for potential contaminants;
- Within mineral hosts, identification of readily soluble ions that allow oxidation and weathering processes to be quantified; and
- Quantification of solid ratios.

Each aspect is described in the following sections.

2.2 Mineral Hosts for Potential Contaminants

Mineral hosts for potential contaminants were evaluated using five sources:

- The "Goldschmidt Classification" of elements broadly groups elements into lithophile (elements with a strong affinity for oxygen and therefore associated with oxides and silicates in the earth's crust), siderophile (elements with affinity for metallic iron), chalcophile (elements with low affinity for oxygen and tending to bond with sulfur to form sulfides) and atmophile (elements found mainly as gases).
- Literature on mineral partition coefficients (K_d) (EarthRef.org 2009[2]).
- Evaluation of relationships between elements determined by near whole rock analysis. The analysis used was ICP data following aqua regia digestion. Element relationships are provided in Attachment 1.
- Evaluation of mineral microprobe analysis of individual mineral grains. This was used principally for understanding the distribution of nickel in pyrrhotite. These data were provided previously by SRK (2007).
- Evaluation of release rate correlations for humidity cell leachates. Explanation of the use of humidity cell data is provided in Section 2.5 below. Specific relationships used to understand release of cadmium, cobalt and manganese are shown in Figure 1.

Results of the various assessments are shown as the mineral sources in Table 1.

[2] EarthRef.org. 2009. Geochemical Earth Reference Model. http://earthref.org/GERM/.

The dominant mineral hosts are the major sulfide minerals (pyrrhotite and chalcopyrite) for the chalcophile and siderophile elements, olivine for the siderophile elements, and biotite for some lithophile elements. The latter was a result of the finding that concentrations of barium, beryllium and vanadium were correlated with potassium concentrations.

No mineral hosts were specifically identified for boron, chromium, molybdenum and thallium. Chromium is likely hosted by oxides such as magnetite with low solubility. Molybdenum and thallium are probably associated with sulfides.

While plagioclase is a major rock component which is important in buffering acidity, it does not appear to be an important source of trace elements.

Table 1: Proposed Approach to Derivation of Inputs to Release Rates. Refer to Sections 2.3 to 2.6 for explanation of methods and sources.

		Duluth Complex Category 1				Duluth Complex Category 2/3 and 4				Virginia Formation Category 4			
	Goldschmidt Classification	**Mineral Sources**	**Ion**	**Method**	**Source**	**Mineral Sources**	**Ion**	**Method**	**Source**	**Mineral Sources**	**Ion**	**Method**	**Source**
SO_4	Chalcophile	Chalcopyrite > pyrrhotite	SO_4^{2-}	SO_4 Rate	HCT	Pyrrhotite > chalcopyrite > pentlandite	SO_4^{2-}	SO_4 Rate	HCT	Pyrrhotite	SO_4^{2-}	SO_4 Rate	HCT
Alkalinity	Atmophile	Calcite, dolomite	CO_3^{2-}	Alkalinity Rate	HCT	Calcite, dolomite	CO_3^{2-}	Alkalinity Rate	HCT	-	-	-	-
Ca	Lithophile	Anorthite, calcite	Ca^{2+}	Ca Rate	HCT	Anorthite, calcite	Ca^{2+}	Ca/SO_4 Rate, Condition 2	HCT	Anorthite	Ca^{2+}	Ca/SO_4 Rate, Condition 3	HCT
Mg	Lithophile	Olivine	Mg^{2+}	Mg Rate	HCT	Olivine	Mg^{2+}	Mg/SO_4 Rate, Condition 2	HCT	Mg silicates	Mg^{2+}	Mg/SO_4 Rate, Condition 3	HCT
Na	Lithophile	Albite	Na^{+}	Na Rate	HCT	Albite	Na^{+}	Na/SO_4 Rate, Condition 2	HCT	Albite	Na^{+}	Na/SO_4 Rate, Condition 3	HCT
K	Lithophile	Biotite	K^{+}	K Rate	HCT	Biotite	K^{+}	K/SO_4 Rate, Condition 2	HCT	Biotite	K^{+}	K/SO_4 Rate, Condition 3	HCT
Ag	Chalcophile	Sulfide	Ag^{+}	Ag/S	Aqua Regia	Sulfide	Ag^{+}	Ag/S	Aqua Regia	Sulfide	Ag^{+}	Ag/S	Aqua Regia
Al	Lithophile	Anorthite, albite	Al^{3+}	Al/Ca (anorthite) + Al/Na (albite)	Mineral forumulae	Anorthite, albite	Al^{3+}	Al/Ca (anorthite) + Al/Na (albite)	Mineral forumulae	Anorthite, albite	Al^{3+}	Al/Ca (anorthite) + Al/Na (albite)	Mineral forumulae
As	Chalcophile	Sulfide	$HAsO_4^{2-}$	As/S	Aqua Regia	Sulfide	$HAsO_4^{2-}$ /$H_2AsO_4^{-}$	As/S	Aqua Regia	Sulfide	$HAsO_4^{2-}$ /$H_2AsO_4^{-}$	As/S	Aqua Regia
B	Lithophile	Unknown	H_3BO_3	B Rate	HCT	Unknown	H_3BO_3	B Rate	HCT	Unknown	H_3BO_3	B Rate	HCT
Ba	Lithophile	Biotite	Ba^{2+}	Ba/K Ratio	Aqua Regia	Biotite	Ba^{2+}	Ba/K Ratio	Aqua Regia	Biotite	Ba^{2+}	Ba/K	Aqua Regia
Be	Lithophile	Biotite	Be^{2+}	Be/K Ratio	Aqua Regia	Biotite	Be^{2+}	Be/K Ratio	Aqua Regia	Unknown	Be^{2+}	Be/S	Aqua Regia
Cd	Chalcophile	Sulfide	Cd^{2+}	Cd/Zn Rate, Condition 2	HCT	Sulfide	Cd^{2+}	Cd/Zn Rate, Condition 2	HCT	Pyrrhotite	Cd^{2+}	Cd/S	Aqua Regia
Co	Siderophile	Olivine	Co^{2+}	Co/Ni Rate, Condition 2	HCT	Olivine	Co^{2+}	Co/Ni Rate, Condition 2	HCT	Pyrrhotite	Co^{2+}	Co/S	Aqua Regia
Cr	Lithophile	Unknown	$CrOH^{2+}$	Cr Rate	HCT	Unknown	$CrOH^{2+}$	Cr Rate	HCT	Unknown	$CrOH^{2+}$	Cr Rate	HCT
Cu	Chalcophile	Chalcopyrite	Cu^{2+}	Cu/S	Aqua Regia	Chalcopyrite	Cu^{2+}	Cu/S	Aqua Regia	Pyrrhotite	Cu^{2+}	Cu/S	Aqua Regia
F	Lithophile	Apatite	F^{-}	F rate	HCT	Apatite	F^{-}	F rate	HCT	Apatite	F^{-}	F Rate	HCT
Fe	Siderophile	Sulfides, silicates	Fe^{3+}	Fe/S (pyrrhotite) + Fe/Mg (olivine)	Microprobe	Sulfides, Silicates	Fe^{3+}	Fe/S (pyrrhotite) + Fe/Mg (olivine)	Microprobe	Sulfides	Fe^{3+}	Fe/SO_4 Rate	HCT
Hg	Chalcophile	Sulfide	Hg^{2+}	Hg/S	Aqua Regia	Sulfide	Hg^{2+}	Hg/S	Aqua Regia	Pyrrhotite	Hg^{2+}	Hg Rate	HCT
Mn	Siderophile	Sulfide	Mn^{2+}	Mn/SO_4 Rate, Condition 2	HCT	Sulfide	Mn^{2+}	Mn/SO_4 Rate, Condition 2	HCT	Pyrrhotite	Mn^{2+}	Mn Rate	HCT
Mo	Siderophile	Molybdenite	MoO_4^{2-}	Mo/S	Aqua Regia	Molybdenite	MoO_4^{2-}	Mo/S	Aqua Regia	Pyrrhotite	H_2MoO_4	Mo/S	Aqua Regia
Ni	Siderophile	Pyrrhotite (assume more reactive than chalcopyrite) / olivine	Ni^{2+}	Ni/S (pyrrhotite) + Ni/Mg (olivine)	Microprobe	Pyrrhotite (assume more reactive than chalcopyrite) / olivine	Ni^{2+}	Ni/S (pyrrhotite)	Microprobe	Pyrrhotite	Ni^{2+}	Ni/S	Aqua Regia
Pb	Chalcophile	Sulfide	Pb^{2+}	Pb/S	Aqua Regia	Sulfide	Pb^{2+}	Pb/S	Aqua Regia	Pyrrhotite	Pb^{2+}	Pb/S	Aqua Regia
Sb	Chalcophile	Sulfide	Oxyanion	Sb/S	Aqua Regia	Sulfide	Oxyanion	Sb/S	Aqua Regia	Pyrrhotite	Oxyanion	Sb/S	Aqua Regia
Se	Chalcophile	Chalcopyrite	SeO_4^{2-}	Se/S	Aqua Regia	Chalcopyrite	SeO_4^{2-}	Se/S	Aqua Regia	Pyrrhotite	H_2SeO_3	Se/S	Aqua Regia
Tl	Chalcophile	Unknown	Tl^{+}	Tl Rate	HCT	Unknown	Tl^{+}	Tl Rate	HCT	Pyrrhotite	Tl^{+}	Tl Rate	HCT
V	Lithophile	Biotite	$VO_2(OH)^{2-}$	V/K	Aqua Regia	Biotite	$VO_2(OH)_2^{-}$	V/K	Aqua Regia	Biotite	VO_2^{+}	V/K	Aqua Regia
Zn	Chalcophile	Olivine	Zn^{2+}	Zn/Mg	Aqua Regia	Pyrrhotite, olivine	Zn^{2+}	Zn/Ni Rate, Condition 2	HCT	Pyrrhotite	Zn^{2+}	Zn/S	Aqua Regia

Source: G:\PolyMet Mining\1UP005.01_Northmet_project_2004\Water_Quality_Predictions\Waste_Rock\2010-12_Scale-Up_Factor_Discussion\Modelling_Approach\[ModelApproachByParameter_1UP005001_SJD_REV00.xlsx]

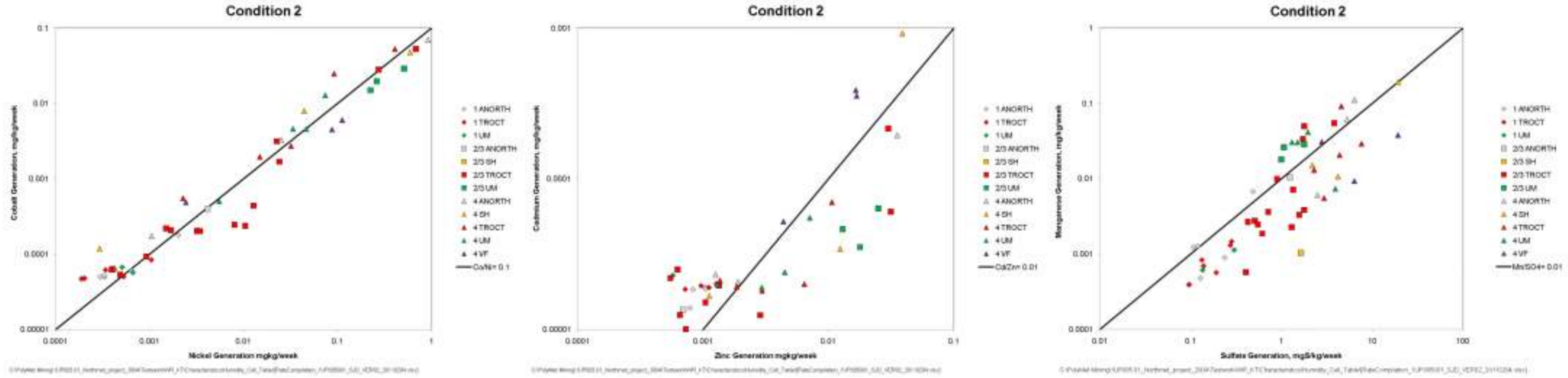

Figure 1. Specific Relationships from Humidity Cells Used to Understand Mineral Hosts. Refer to Section 2.5 for explanation of leachate condition used to generate plots.

2.3 Identification of Soluble Ion for Ratios

Based on the above mineralogical conclusions, sulfate, magnesium and potassium were identified as primary ions that are readily soluble components of major mineral hosts that will be released as weathering products.

A few other strong secondary relationships were identified based on correlations between release rates in humidity cells, as shown in Figure 1. Cadmium and zinc release are correlated indicating a Cd/Zn ratio of about 0.001 (mg/mg). Likewise cobalt and nickel release are also strongly correlated showing Co/Ni of about 0.1 (mg/mg). Cadmium is therefore predicted from zinc release and the observed ratio, cobalt is predicted from nickel release and the observed ratio. Manganese and sulfate release are correlated with a Mn/SO_4 ratio of about 0.01 (mg/mg), and manganese is predicted from sulfate release and the observed ratio (not the solid Mn/S ratio).

2.4 Quantification of Solid Ratios for Calculation of Release Rates

Table 1 shows the approach proposed to obtain solids ratios for each parameter in each category.

For Category 1, the low sulfide content indicates that acid generation from sulfide oxidation may be subordinate to general silicate weathering as a source of metals contained in both silicates and sulfides. Weathering of silicates is therefore considered separately for Category 1.

Primary release rates for Category 1 are based on sulfate, magnesium and potassium as indicated by humidity cell results. For most elements, metal to sulfur, metal to magnesium, and metal to potassium ratios indicated by aqua regia digestion of rock samples will be used to predict metal release based on the primary release rates. Nickel and iron release will be calculated from metal to sulfur and metal to magnesium ratios indicated by microprobe data for pyrrhotite and olivine. This is preferred over bulk solids analysis because it provides direct measurements of the minerals and the data are already available. A similar approach will be used for aluminum, based on the mineral formulae for anorthite and albite rather than microprobe data. For boron, chromium and thallium, direct rates for humidity cells will be used due to the lack of correlation with any of the known major mineral hosts.

For Category 2/3 and 4 Duluth Complex rocks, by definition, acid generation from sulfide oxidation is the major process driving these rock types to become acidic at some point. As a result, dissolution of silicates is linked to acid generation. As shown in Table 1, magnesium and potassium release rates are correlated with sulfate release and therefore would be calculated from sulfate. Other solids ratios are based on the same mineralogical assumptions as Category 1. For zinc, a strong correlation with nickel leaching is apparent in acidic humidity cells, and zinc release would be based on the zinc to nickel ratio.

For Virginia Formation Category 4, solids ratios would largely be based on the strong relationships between sulfur content and metal content. The solids ratios are not necessarily same as for Duluth Complex Category 4 due to the fundamental difference in rock type and mineralogy.

2.5 Release Rates

Humidity cell data have been completely re-interpreted to obtain data for this assessment. Trends for each humidity cell were examined to understand oxidation rates under changing weathering conditions. Five different leachate conditions were recognized and used to calculate rates:

- Condition 0: Brief (a few weeks) initial flushing of weathering products accumulated in storage. No rates were calculated for this period because a time period cannot be assigned to reflect core and sample storage prior to testing.
- Condition 1: Sulfate release relatively stable and leachate pH above about 7.
- Condition 2: Sulfate release relatively stable, leachate pH below about 7, nickel release unstable and typically increasing.
- Condition 3: Sulfate release increasing and variable, leachate pH decreasing further.
- Condition 4: Sulfate release decreasing following a peak usually under acidic conditions.

An example of a test showing all five conditions is shown in Figure 2. Condition 1 begins after week 0 as the initial flushing effect was relatively minor. During Condition 1, leachate pH decreased. Condition 2 begins at week 39 roughly marking when pH dropped below a level which allowed nickel release to being increasing. The beginning of Condition 3 started when sulfate release increased rapidly (along with pH decrease) reaching a peak. Condition 4 is defined as starting when peak sulfate release was reached.

Charts showing assignment of conditions to all tests are provided in Attachment 2. Table 2 summarizes identified conditions.

As shown in Figure 2, Condition 2 is considered to be sufficiently acidic to allow elements such as cadmium, cobalt, manganese and zinc to be released without significant attenuation as secondary minerals. Experiments by both MDNR and PolyMet show that release of these elements from kinetic tests accelerate as pH decreases below 7.

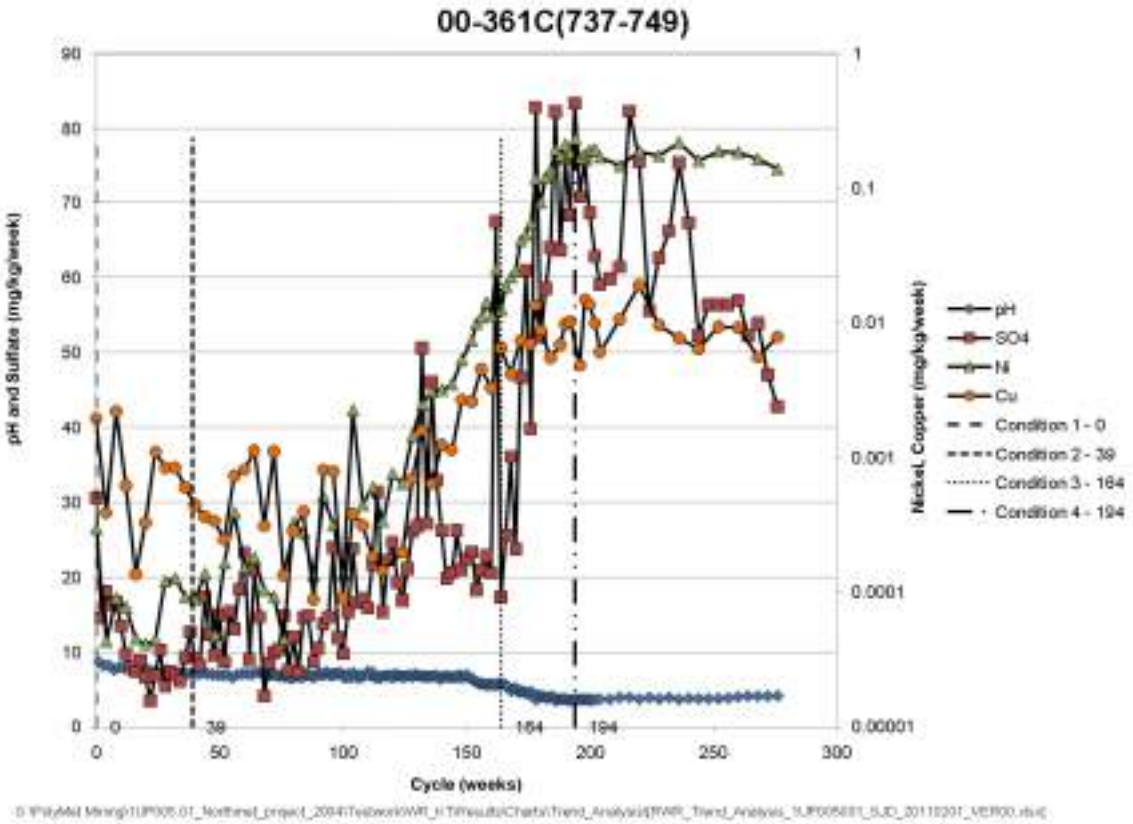

Figure 2. Example of Leachate Conditions for a Humidity Cell

Table 2. Summary of Conditions Applied to NorthMet Project Humidity Cells

Rock Type	Waste Category	Sample ID	S	Condition 1 Start	Condition 2 Start	Condition 3 Start	Condition 4 Start	Total Duration
			%	week	week	week	week	week
Anorthositic	1	99-320C(830-850)	0.09	4	179	-	-	284
Anorthositic	1	00-361C(345-350)	0.05	6	184	-	-	284
Anorthositic	1	00-366C(185-205)	0.02	0	-	-	-	198
Anorthositic	1	00-366C(230-240)	0.02	4	60	-	-	198
Anorthositic	1	99-320C(165-175)	0.03	0	72	-	-	198
Anorthositic	1	00-334C(30-50)	0.02	4	-	-	-	284
Anorthositic	1	00-368C(125-145)	0.04	0	80	-	-	284
Anorthositic	1	00-368C(20-40)	0.04	0	80	-	-	198
Troctolitic	1	00-340C(595-615)	0.04	0	-	-	-	198
Troctolitic	1	00-334C(580-600)	0.06	1	179	-	-	284
Troctolitic	1	00-334C(640-660)	0.07	12	224	-	-	284
Troctolitic	1	00-347C(795-815)	0.07	0	103	-	-	198
Troctolitic	1	99-318C(250-270)	0.04	0	72	-	-	198
Troctolitic	1	00-373C(95-115)	0.04	0	-	-	-	198
Troctolitic	1	00-373C(75-95)	0.06	0	-	-	-	198
Troctolitic	1	00-357C(110-130)	0.08	10	-	-	-	198
Troctolitic	1	99-320C(315-330)	0.07	4	-	-	-	284
Troctolitic	1	00-366C(35-55)	0.02	0	-	-	-	198
Troctolitic	1	00-334C(110-130)	0.04	0	-	-	-	198
Troctolitic	1	00-347C(155-175)	0.06	0	72	-	-	198
Troctolitic	1	00-347C(280-300)	0.06	16	65	-	-	198
Troctolitic	1	00-367C(50-65)	0.03	0	-	-	-	198
Troctolitic	1	00-367C(260-280)	0.04	0	-	-	-	198
Troctolitic	1	00-367C(290-310)	0.04	0	-	-	-	284
Troctolitic	1	00-370C(20-30)	0.08	10	-	-	-	198
Troctolitic	1	26064(44-54)	0.02	0	-	-	-	284
Troctolitic	1	26064(264+146269+156)	0.06	4	-	-	-	284
Troctolitic	1	26056(110-125)	0.04	0	-	-	-	198
Troctolitic	1	26029(815-825)	0.02	0	-	-	-	194
Troctolitic	1	26056(135-153)	0.05	0	-	-	-	278
Troctolitic	1	00-326C(250-265)	0.08	4	-	-	-	186
Ultramafic	1	00-357C(335-340)	0.08	12	187	-	-	198
Ultramafic	1	00-368C(460-465)	0.06	0	-	-	-	198
Ultramafic	1	26055(940-945)	0.06	16	-	-	-	198
Ultramafic	1	26098+00-337C	0.1	0	-	-	-	198
Ultramafic	1	00-361C(240-245)	0.06	14	184	-	-	284
Ultramafic	1	26039(310-315)	0.06	8	-	-	-	186
Ultramafic	1	00-326C(225-235)	0.12	8	-	-	-	273
Anorthositic	2/3	00-361C(310-320)	0.18	0	111	-	-	284
Anorthositic	2/3	99-320C(400-405)	0.18	14	-	-	-	273
Sedimentary Hornfels	2/3	26030(1047-1052)	0.24	53	-	-	-	284
Sedimentary Hornfels	2/3	26061(1218-1233)	0.44	4	-	-	-	284
Sedimentary Hornfels	2/3	00-340C(990-995)	0.55	0	189	-	-	284
Troctolitic	2/3	00-350C(580-600)	0.19	0	196	-	-	284
Troctolitic	2/3	00-327C(225-245)	0.44	0	182	-	-	198
Troctolitic	2/3	00-369C(335-345)	0.18	4	181	-	-	284
Troctolitic	2/3	00-326C(60-70)	0.14	0	75	-	-	284
Troctolitic	2/3	00-369C(305-325)	0.25	4	187	-	-	198
Troctolitic	2/3	00-369C(20-30)	0.21	0	187	-	-	284
Troctolitic	2/3	00-367C(170-175)	0.51	0	172	-	-	284
Troctolitic	2/3	00-340C(380-390)	0.15	4	-	-	-	198
Troctolitic	2/3	26049+26030	0.59	4	-	-	-	198
Troctolitic	2/3	26056(302-312)	0.23	12	212	-	-	284
Troctolitic	2/3	26142(360+345-365+350)	0.18	0	168	-	-	284
Troctolitic	2/3	99-318C(325-330)	0.17	0	180	-	-	273
Troctolitic	2/3	26056(282-292)	0.32	2	178	-	-	186
Troctolitic	2/3	00-340C(910-925)	0.36	0	72	110	180	273
Troctolitic	2/3	00-331C(190-210)	0.42	0	48	201	-	273
Troctolitic	2/3	00-367C(495-500)	0.28	8	114	-	-	273
Ultramafic	2/3	00-326C(680-685)	0.30	0	69	-	-	198
Ultramafic	2/3	00-357C(535-540)	0.2	0	78	-	-	284
Ultramafic	2/3	00-344C(630-635)	0.34	0	51	160	-	186
Ultramafic	2/3	00-326C(495-505)	0.16	0	-	-	-	186
Anorthositic	4	00-343C(240-250)	0.68	0	161	-	-	198
Anorthositic	4	26027(616-626)	1.83	4	18	24	-	284
Anorthositic	4	00-331C(255-260)	0.86	0	19	162	184	273
Sedimentary Hornfels	4	00-340C(965-974.5)	1.74	0	25	34	80	198
Sedimentary Hornfels	4	26043+26027	2.47	0	9	26	48	284
Sedimentary Hornfels	4	26062+26026	4.46	0	3	3	-	284
Sedimentary Hornfels	4	26058(704-715)	1.46	8	41	-	-	273
Troctolitic	4	00-371C(435-440)	0.88	0	51	90	196	284
Troctolitic	4	00-340C(765-780)	1.68	0	61	82	-	284
Troctolitic	4	00-367C(395-400)	0.77	0	82	-	-	198
Troctolitic	4	00-340C(725-745)	0.91	6	118	-	-	198
Troctolitic	4	00-367C(400-405)	1.37	4	39	78	-	198
Ultramafic	4	99-318C(725-735)	0.72	0	96	-	-	198
Ultramafic	4	99-317C(460-470)	1.24	0	39	-	-	198
Ultramafic	4	00-344C(515-520)	1.2	4	47	152	-	198
Ultramafic	4	00-330C(275-280)	0.75	0	164	-	-	186
Virginia	4	00-361C(737-749)	2	0	39	164	194	284
Virginia	4	00-364C(210-229)	3.79	0	0	5	-	198
Virginia	4	00-337C(510-520)	5.68	0	0	5	-	198
Ore Composite	4	P10	0.86	4	88	-	-	268
Ore Composite	4	P20	0.9	6	88	-	-	268
Ore Composite	4	P30	0.86	6	88	-	-	268

Source: G:\PolyMet Mining\1UP005.01_Northmet_project_2004\Water_Quality_Predictions\Waste_Rock\2011-02_Collected_Inputs_for_Modeling\HCT_Data\[Rates_1UP005001_SJD_VER00_20110301.xlsx]

Notes: "-" condition not observed.

2.6 Incorporation of Acidification Effects into Release

Acidification effects for Cat 2/3 and 4 will be incorporated using the previous approach to estimate acceleration of oxidation and acid release rates. These effects are mainly quantified through reference to the MDNR's long term testwork and include:

- Time to acceleration of sulfide oxidation (time from start of test to start of Condition 3).
- Ratio of peak of sulfide oxidation rate to pre-peak oxidation rates (Condition 4 to Condition 2 ratio).
- Decay curve for sulfide oxidation rates post-peak (decrease after start of Condition 4).

Data from NorthMet Project humidity cells are beginning to provide information on the increase in oxidation rates as pH decreases though the increases are less than the ten times factors observed in DNR tests (Figure 3). DNR tests used to characterize these features are shown in Table 3.

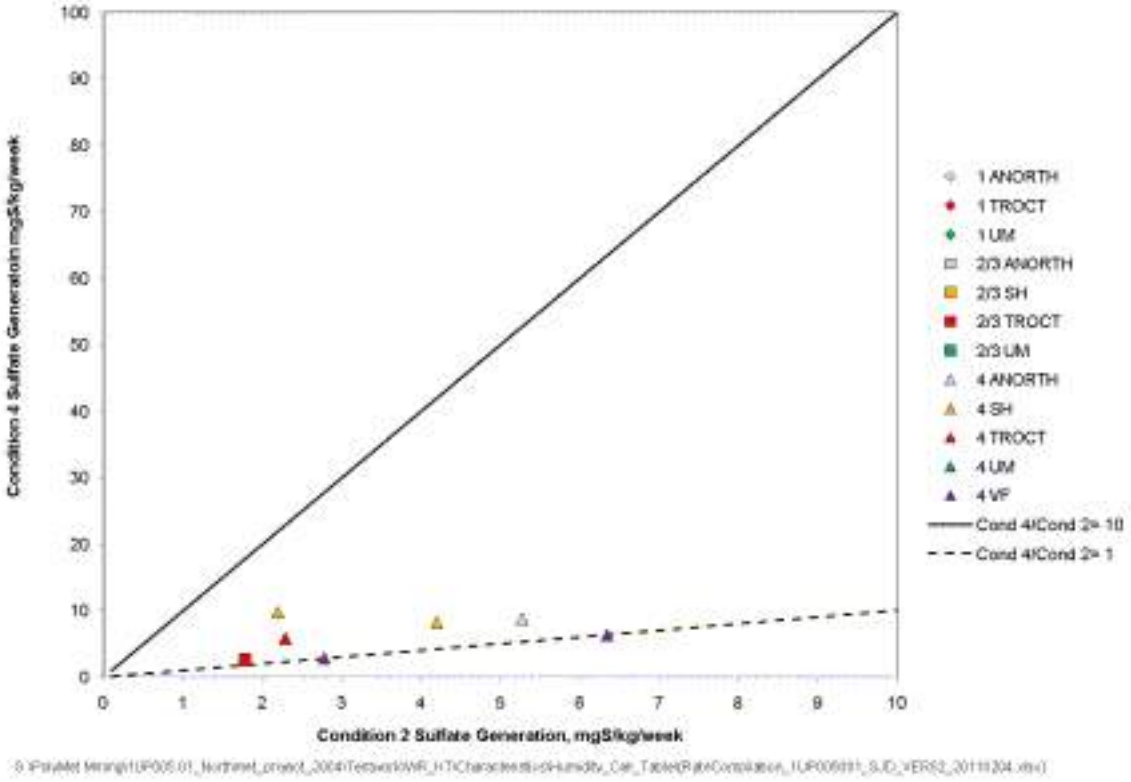

Figure 3. Comparison of Condition 2 and Condition 4 Rates for NorthMet Project Humidity Cells

Table 3: MDNR's Long Term Testwork Used to Characterize Delay to Onset, Peak Oxidation Rates and Decay Curves

Dunka Blast Hole Sample Reactor	S	NorthMet Waste Rock Category	Time to Acceleration and Peak to Pre-Peak	Decay Curve
	%			
7	0.41	2/3	X	X
9	0.51	2/3	X	X
10	0.51	2/3	X	
12	0.54	2/3	X	
11	0.54	2/3	X	
14	0.57	2/3	X	
13	0.57	2/3	X	
15	0.58	2/3	X	X
16	0.58	2/3	X	
39	0.67	4	X	
17	0.71	4	X	X
35	1.12	4	X	
29	1.16	4	X	X
37	1.40	4	X	
33	1.44	4	X	X
19	1.63	4	X	X
31	1.64	4	X	

Source: \\Van-svr0.van.na.srk.ad\ge_projects\PolyMet Mining\1UP005.01_Northmet_project_2004\Background_Docs\MDNR_Testwork\F.Dunka_Blast_Hole_Small_Reactors\Data_analysis\[Decay_Curves_1UP005001_ver02_SJD.xlsx]
\\Van-svr0.van.na.srk.ad\ge_projects\PolyMet Mining\1UP005.01_Northmet_project_2004\Background_Docs\MDNR_Testwork\F.Dunka_Blast_Hole_Small_Reactors\Data_analysis\[SO4_Acceleration_Factors_1UP005001_VER02.xls]

Since the majority of release rates are linked explicitly to sulfate release which tracks release from source minerals, other rates will also change by the same factors. For parameters not linked to sulfate, actual measured rates from humidity cells (defined using observed Condition 2 and Condition 3) will be used.

3 Quantification of Leaching

3.1 Conceptualization

As indicated above, release refers to the transfer of potential contaminants from primary solid phase to weathering products. Leaching refers to the transfer to the aqueous phase and leaching from the facility. Leaching is modeled by consideration of the solubility of the weathering products.

For the expected well-oxygenated conditions in the waste rock piles and pit walls, the dominant control on leaching is expected to be pH with oxidation-reduction potential being a lesser or negligible effect. An additional consideration is the kinetics of leaching which may result in concentrations below the theoretical or maximum solubility of a weathering product.

3.2 Data Sources

3.2.1 Background

Due to the limitation of thermodynamic models for the majority of trace contaminants, and limited understanding of the kinetics of leaching, the previous approach has been to use analog data to provide solubility limits. The only true analog data available for the site is the AMAX stockpiles;

however, concerns have been raised about the suitability of these due to their relatively small size (potentially lower concentrations than full scale), high commodity element content (higher concentrations than waste rock particularly for Category 1) and lack of data on the full suite of elements needed for water quality modeling.

To evaluate these concerns, the following specific searches were completed:

- Means to simulate Category 1 type material.
- Examples of comparisons of small tests with full scale piles.
- Relevant geological analogs particularly for acid rock drainage.

3.2.2 Category 1 Solubility Controls

A previous search for drainage chemistry from Category 1 type waste materials was not fruitful. One site (Lac des Iles Mine, Ontario, Canada) was identified as having similar waste rock; however, the operator does not monitor drainage chemistry. The recommended approach is in general to use the results of the SMWMP experiment reported separately (SRK 2011[3]). Some elements were not constrained by the experiment and would be constrained from mineral solubility indicated by thermodynamic modeling:

- Sulfate – Calculated from SO_4 (mg/L) = 1294.(Mg+0.5.Na+0.5K)/Ca + 1760. The ion ratio is expressed as moles and is obtained from the release rates indicated by Condition 2 of humidity cells. This equation indicates gypsum solubility of 1760 mg/L if Mg, Na and K are not present. Median solubility is 2700 mg/L.
- Calcium – Calculated from sulfate (mg/L), sodium (mmol/L) and potassium (mmol/L) concentrations using the above equation: Ca (mmol/L) = 1294(Mg+0.5Na+0.5K)/(SO_4-1760)
- Barium – based on barite solubility as a sulfate ($\log_{10}Ba = -0.32.\log_{10}SO_4-0.87$).
- Copper – tenorite solubility as a function of pH (0.04 to 0.07 mg/L between pH 8 and 7.5).
- Fluoride – based on fluorite solubility as a function of Ca ($\log_{10}F = -0.298.\log_{10}Ca-0.817$).
- Molybdenum – 1.1 mg/L based on powellite solubility at typical calcium concentrations.
- Selenium – Calculated from Se (mg/L) = $6.35x10^{-6}.SO_4 + 0.0020$.

3.2.3 Comparisons of Acidic Drainage Solubility Controls

Opportunities to compare tests performed at different scales were sought to evaluate the suitability of the AMAX stockpiles as analogs for full scale waste rock stockpiles. SRK has access to three datasets that include a full range of comparisons between laboratory tests, different fields scale tests and full scale. Two of the datasets are from coal mines with no ARD and therefore considered to have limited value to the NorthMet Project. The comparisons showed that small scale (eg <1 tonne) field tests show the same chemistry as full scale facilities.

To support drainage chemistry prediction for another project (see Day and Rees 2006[4]), SRK compiled data for porphyry open pit mines in Western Canada. These sites are not analogs for copper-nickel mining in the Duluth Complex but there are some examples of comparisons of smaller scale tests with full scale monitoring. The dataset consists of seven sites:

- Three are older sites with acid rock drainage and data are for monitoring of waste rock stockpiles.
- Two are older sites without ARD again with monitoring of waste rock stockpiles.

[3] SRK Consulting. 2011. Results of an experiment to evaluate solubility limits for Category 1 waste rock, NorthMet Project. Draft Memorandum, SRK Project 1UP005.001. February 16, 2011.

[4] Day, S. and Rees, B. 2006. Geochemical Controls on Waste-Rock Dump Seepage Chemistry at Several Porphyry Mines in The Canadian Cordilleran. Paper was presented at the 2006, 7th ICARD, March 26-30, 2006, St Louis MO. Published by ASMR, 3134 Montavesta Road, Lexington, KY 40502.

- Two sites are more recently developed. One of these has a long term column test which has generated ARD. No comparable full scale data are available because the mine has been designed to mitigate ARD potential. One site has numerous small-scale field pads. Some of these have generated ARD.

Data are provided as charts in Attachment 3.

The dataset indicates that for the gross parameters indicating acid generation and buffering reactions, sulfate concentrations were with the same order-of-magnitude for each scale though slightly higher at full scale, and iron and aluminum concentrations were similar at all scales. Trace element concentrations were variable and are probably affected by site mineralogy. Copper concentrations in the column leachate were very similar to those at full scale for two sites where copper is the main commodity. Leach pad copper concentrations were over an order of magnitude lower than these concentrations though copper concentrations are low at this site. Nickel data were limited but the column and leach pads both yielded higher concentrations than seen in ARD for one mine. Zinc concentrations in leach pads and columns were comparable to the full scale mines but likely reflect the variable occurrence of zinc sulfide in porphyry copper deposits.

In general, the comparison shows that concentration differences were not reflective of scale and water-to-rock differences. For example, the range of sulfate concentrations in ARD would be expected to span several orders of magnitude if test size were a major control on ARD potential. This does not appear to be the case.

The finding indicates that AMAX test stockpile drainage chemistry provides a reasonable indication of drainage chemistry because scale effects appear to be relatively minor.

A limitation of the AMAX test stockpile data is that the dataset likely includes low concentrations which do not reflect solubility constraints but are a result of dilution of drainage waters by snowmelt and rainstorm events. Higher concentrations in the dataset are more likely to reflect solubility constraints based on the above comparisons.

3.2.4 Relevant Geological Analogs

Duluth Complex Waste Rock (Category 2, 3 and 4)

SRK has sought drainage chemistry data for waste rock produced by mining in similar geological settings (layered mafic intrusions) with particular emphasis on locating chemistry data for trace elements not determined in the drainage from the AMAX test stockpiles.

Information in the literature was very limited, and contact with mining companies was unsuccessful. A 5 million tonne waste rock stockpile at the Whistle Mine in the Sudbury Basin in Ontario, Canada was studied under the MEND program (Vos et al. 1997[5]). Waste rock at the site is norite which is similar in composition to gabbro. The main rock type in the Duluth Complex is troctolite which is a type of gabbro containing less pyroxene than gabbro. Construction of the waste rock stockpile started in 1988. All waste rock was backfilled to the open pit between July 2000 and December 2001[6].

Sampling of the stock pile indicated average sulfur concentrations of about 2% though the pile also contains rock that is semi-massive sulfide. Drainage chemistry monitoring in 1995 and 1996

[5] Vos, K.J., Pettit, C., Martin, J., Knapp, R.A. and Jansons, K.J. Whistle Mine Waste Rock Study. MEND Report 1.41.1. October, 1997.

[6] Ayres, B., Lanteigne, L., and O'Kane, M., 2005. Design and construction of the backfilled pit cover system at Whistle Mine, Canada: a case study. Securing the Future 2005, International Conference on Mining and the Environment, Skelleftea, Sweden, June 27-July 1, 2005.

included sampling of seeps and groundwater in close proximity to the pile. As a result, concentrations are not expected to have a strong effect from diluting events such as snowmelt and rainstorms. Water quality is expected to come close to source water concentrations. The data are provided in Attachment 4.

Table 4 summarizes data for acidic (pH<4.5) waters. For trace elements, detection limits were higher than available currently and variable through the study. The summary provided in Table 4 is based on only reported values above the lowest detection limit. The distributions do not include the higher detection limits, hence the number of values available is not the same for each parameter.

For comparison, Table 4 also includes the deterministic values in RS42 to represent solubility at pH 3.5. The Whistle Mine 95th percentile values and the RS42 values were within the same order of magnitude for most parameters. Exceptions are aluminum, copper, barium, beryllium, cadmium, chromium, lead, antimony and selenium. The elevated aluminum concentrations may reflect the abundance of anorthite in norite compared to gabbro. Copper concentrations were greater in the AMAX stockpiles drainage compared to the Whistle Mine waste rock.

For some of the remaining elements, the variable detection limits may partly be a factor leading to elevated but poorly quantified values. For chromium, detections were not normal and most values were below 0.01 mg/L. For lead, concentrations if detected tended to be near the detection limit in use at the time and concentrations were typically below 0.5 mg/L. The maximum lead value in the dataset was 11 mg/L which was more than an order of magnitude higher than the next nearest value of 0.6 mg/L. Antimony concentrations were always near the detection limit which was commonly 1 mg/L. The apparent concentrations in the 2 to 3 mg/L range appear to be a function of uncertainty near the detection limits. Selenium was never detected and the lowest detection limit was 0.1 mg/L.

Table 4: Summary of Acidic (pH<4.5) Seepage and Groundwater Data for Whistle Mine Waste Rock Stockpile (Vos et al. 1997).

							RS42 (Table 7-2) Deterministic Values	
Parameter	**Units**	**DL**	**n**	**P5**	**Median**	**P95**	**AMAX Pile**	**Other**
pH	mg/L	-	30	3.8	4.2	4.5	-	
Eh	mg/L	-	30	87	260	400	-	
Conductivity	mg/L	-	28	4200	8700	11000	-	
SO_4	mg/L	-	29	4300	9200	18000	9600	
Ag	mg/L	0.005	4	0.036	0.05	0.05	-	
Al	mg/L	0.5	22	120	370	930	83	
As	mg/L	0.1	9	<0.1	<0.1	<0.1		0.71
B	mg/L	0.01	12	0.1	0.2	0.4		0.76
Ba	mg/L	0.005	9	0.0091	0.015	0.019		0.19
Be	mg/L	0.005	22	<0.005	<0.005	0.02		0.0023
Cd	mg/L	0.01	12	0.055	0.12	0.33		0.015
Co	mg/L	0.01	29	11	33	42	44	
Cr	mg/L	0.01	9	<0.01	<0.01	0.015		0.0015
Cu	mg/L	0.01	25	1.2	11	28	200	
Fe	mg/L	0.1	29	3.3	130	250	240	
Mn	mg/L	0.01	29	22	55	93	47	
Ni	mg/L	0.05	29	250	680	820	760	
Pb	mg/L	0.05	3	0.15	0.36	0.58		0.053
Sb	mg/L	0.1	9	1.4	2	3		0.00001
Se	mg/L	0.1	1	-	<0.1	-		0.0029
V	mg/L	0.005	3	0.051	0.055	0.06	-	
Zn	mg/L	0.01	28	6.3	9	43	26	

Source: \\Van-svr0.van.na.srk.ad\ge_projects\PolyMet Mining\1UP005.01_Northmet_project_2004\Water_Quality_Predictions\Waste_Rock\2011-01_Waste_Rock_Analogs\Analog_Sites\Whistle_Mine\[Whistle_Mine_1UP005001_SJD_VER00.xlsx]

Virginia Formation

Leaching of the Virginia Formation waste rock and pit walls is expected to occur differently from Duluth Complex waste rock due to the different waste rock matrix and higher sulfur content. Since the protolith for this rock type is a black shale, waste rock analogs for this types of rock were sought. Direct analogs (i.e. slates containing pyrrhotite but not other types of mineralization) were not identified in the public record.

SRK has access to drainage chemistry data for three sediment hosted lead-zinc deposits. The Anvil Range Mine Complex (Vangorda Mine) is under public management to address acid rock drainage and therefore drainage chemistry data are available in the public record[7]. Table 5 summarizes distribution statistics for ARD from the waste rock stockpile which is collected from drains under the stockpile. Like Whistle Mine, these waters are expected to be dominated by contact waters rather than dilute surface waters.

Data are provided for monitoring location V32 in Attachment 5. Other monitoring locations are not acidic. Due to the presence of lead and zinc mineralization in the waste rock, zinc, lead and cadmium

[7] Gartner Lee Limited, 2007. Anvil Range Mine Complex 2006 Annual Environmental Report Water Licence QZ03-059. Prepared for Deloitte & Touche Inc. February 2007.

concentrations are very high and considered non-representative for the Virginia Formation. For these parameters, it is proposed to use data from the AMAX stockpiles.

Table 5. Summary of Acidic Seepage Data for Vangorda Waste Rock Stockpile

Parameter	Unit	n	P5	P50	P95
pH	-	33	2.8	3.7	5.4
SO_4	mg/L	34	3600	13000	71000
Ag	mg/L	19	0.0064	0.03	0.21
Al	mg/L	31	1.8	7.9	420
As	mg/L	24	0.011	0.069	2
Ba	mg/L	30	0.005	0.092	1.1
B	mg/L	12	0.066	1.3	3.1
Be	mg/L	28	0.0052	0.011	0.24
Cd	mg/L	31	0.92	4.8	17
Co	mg/L	31	7.1	14	28
Cr	mg/L	13	0.0096	0.017	0.84
Cu	mg/L	30	0.033	0.1	0.35
Fe	mg/L	31	6	220	3800
Mn	mg/L	29	340	1200	3800
Ni	mg/L	31	4.3	9.6	21
Pb	mg/L	29	0.3	1.5	11
Sb	mg/L	25	0.001	2.5	8.9
Se	mg/L	14	0.073	0.12	2
Tl	mg/L	9	0.0079	0.012	0.018
V	mg/L	11	0.003	0.005	0.34
Zn	mg/L	31	840	3000	11000

Source: G:\PolyMet Mining\1UP005.01_Northmet_project_2004\Water_Quality_Predictions\Waste_Rock\2011-01_Waste_Rock_Analogs\Analog_Sites\Vangorda\[Copy of Vangorda_DumpDrainChemistry_dbm_rev00.xlsx]

3.3 Summary of Solubility Constraints

Table 6 summarizes the proposed approach to define solubility constraints for each parameter based on the above discussion.

Category 1 values are based mainly on the SMWMP results referenced in Section 3.2.2 with the exception of elements which were not constrained by the procedure. Most of these non-constrained elements would be constrained by concentrations indicated by thermodynamic modeling. Sulfate would be calculated based on the solubility of gypsum varying as a function of ion ratios. Nickel would be constrained using the AMAX pile data for basic pH conditions indicated by the SMWMP results.

For non-acidic drainage from Category 2/3 waste rock, sulfate concentrations would be calculated based on the solubility of gypsum. Where data were available, metal concentrations would be constrained using AMAX pile data. For parameters not determined for the AMAX pile drainages, SMWMP data would be used if the experiment did not show concentrations varying as a function of sulfur content of the sample (Pb, Sb, Tl, V). Whistle Mine acidic concentrations would be used if the SMWMP data showed that sulfur concentrations are expected to be an important control.

For acidic drainage from Duluth Complex Category 2/3 waste rock and Category 4 mine walls, the main sources of constraints are the AMAX piles and Whistle Mine.

For Virginia Formation Category 4 waste rock and mine walls, the Vangorda Mine provides the constraints with the exception of Cd, Pb and Zn which would be assigned the values used for the Whistle Mine.

In terms of using solubility for the probabilistic analysis, the following approach is proposed for development of distributions:

- When using SMWMP data, distributions would be developed based on the concentrations indicated by the last leach cycle of the test data.
- When using Whistle Mine and Vangorda Mine data, the entire dataset in the indicated pH range will be used in each case based on the assumption that water chemistry reflects mainly contact conditions rather than dilution.
- When using AMAX stockpile data, the data would be assigned to 0.1 pH bins in the pH range. In each bin, concentrations at the 95^{th} percentile and above would be extracted from the dataset, combined and used to determine overall concentration distribution. This approach removes waters from the dataset that may be dilute, respects the likely correlation of solubility constraints with pH and considers the uncertainty in pH range.

Table 6. Summary of Solubility Constraint Approach for Each Parameter

	Category 1		Non Acidic Duluth Category 2/3		Acidic Duluth Category 2/3/4		Virginia Formation (Cat 4)	
	Source	**Comment**	**Source**	**Comment**	**Source**	**Comment**	**Source**	**Comment**
SO_4	Gypsum solubility		Gypsum solubility		Whistle Mine		Vangorda Mine	
Alkalinity	SMWMP		SMWMP		No alkalinity		No alkalinity	
Ca	Gypsum solubility		Gypsum solubility	See text	Whistle Mine		Vangorda Mine	
Mg	From Mg/Ca in SMWMP Leachate		From Mg/Ca SMWMP Leachate		Whistle Mine		Vangorda Mine	
Na	SMWMP		SMWMP		Whistle Mine		Vangorda Mine	
K	SMWMP		SMWMP		Whistle Mine		Vangorda Mine	
Ag	SMWMP		SMWMP	No S correlation	Whistle Mine		Vangorda Mine	
Al	SMWMP		SMWMP		Whistle Mine		Whistle Mine	
As	SMWMP		Whistle Mine		Whistle Mine		Vangorda Mine	
B	Whistle Mine		Whistle Mine		Whistle Mine		Vangorda Mine	
Ba	Barite solubility		Barite solubility		Barite solubility		Vangorda Mine	
Be	SMWMP		Whistle Mine		Whistle Mine		Vangorda Mine	
Cd	SMWMP		Cd/Zn ratio		Whistle Mine		Whistle Mine	No Cd, Pb, and Zn mineralization
Co	SMWMP		AMAX Pile pH 6 to 8		Whistle Mine	Comparable to AMAX	Vangorda Mine	
Cr	SMWMP		Whistle Mine		Whistle Mine		Vangorda Mine	
Cu	Model tenorite at pH 7.5 to 8		AMAX Pile pH 6 to 8		AMAX Pile pH 3 to 4	Higher than Whistle Mine	Vangorda Mine	
F	Fluorite solubility		Fluorite solubility		Fluorite solubility		Fluorite solubility	
Fe	SMWMP		SMWMP		Whistle Mine		Vangorda Mine	
Hg	Rock sorption data		Rock sorption data		Rock sorption data		Rock sorption data	
Mn	SMWMP		AMAX Pile pH 6 to 8		Whistle Mine	Comparable to AMAX	Vangorda Mine	
Mo	Model powellite		Model powellite		Model powellite		Vangorda Mine	
Ni	AMAX Pile pH 7 to 8		AMAX Pile pH 6 to 8		Whistle Mine	Comparable to AMAX	Vangorda Mine	
Pb	SMWMP		SMWMP	No S correlation	Whistle Mine		Whistle Mine	No Cd, Pb, and Zn mineralization
Sb	SMWMP		SMWMP	No S correlation	Whistle Mine		Vangorda Mine	
Se	Sequestered by gypsum		Sequestered by gypsum		Whistle Mine		Vangorda Mine	
Tl	SMWMP		SMWMP	No S correlation	Vangorda Mine	No data for Whistle Mine or AMAX piles	Vangorda Mine	
V	SMWMP		SMWMP	No S correlation	Whistle Mine		Vangorda Mine	
Zn	SMWMP		AMAX Pile pH 6 to 8		Whistle Mine	Comparable to AMAX	Whistle Mine	No Cd, Pb, and Zn mineralization

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]

4 Conclusions

The proposed method for calculating source water quality is based on the separation of contaminant release and leaching processes.

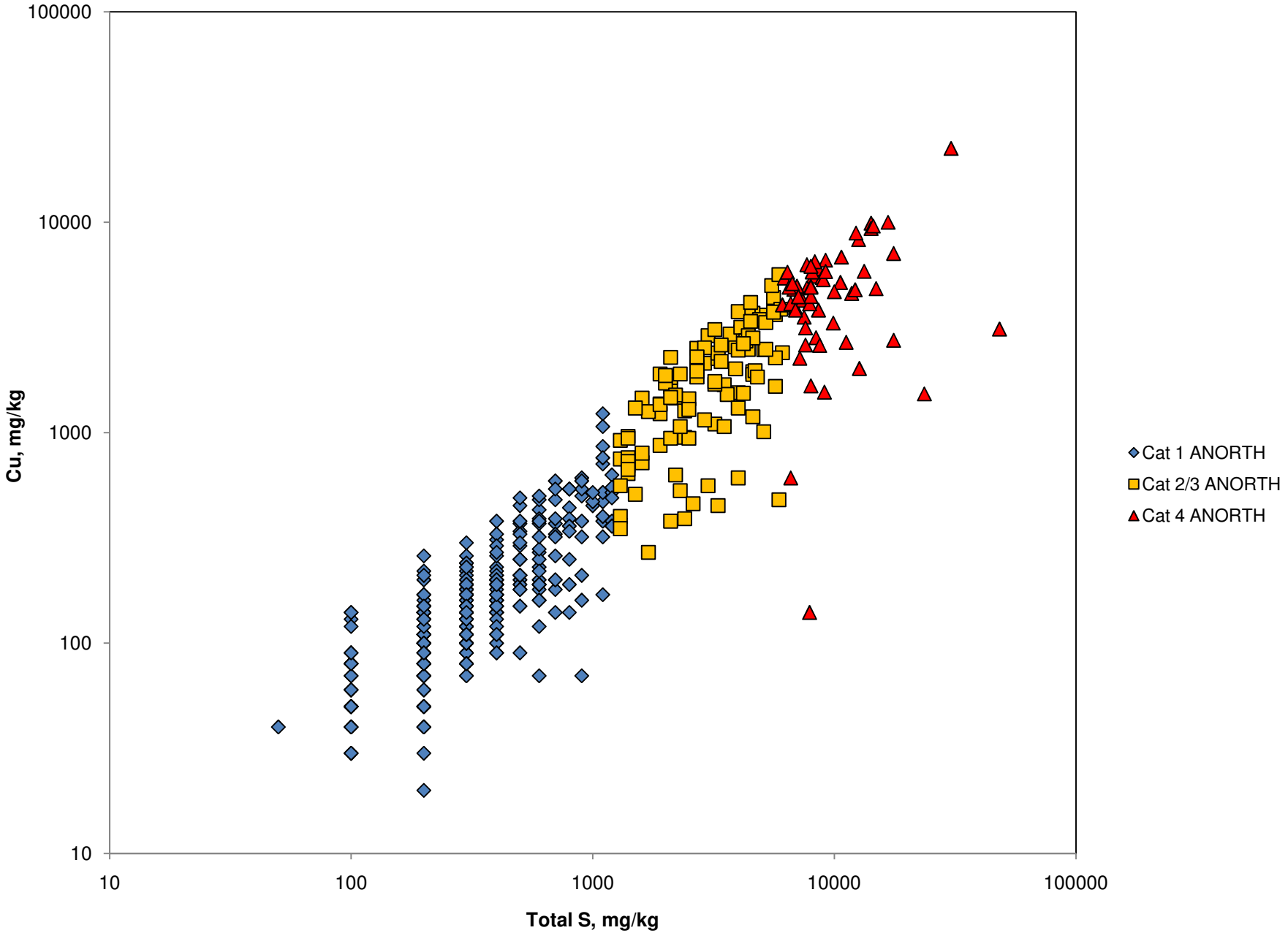
Release rates would be estimated from the parameters indicating weathering of source minerals which are mainly predicted to be sulfides, olivine and bioite, and the contaminant content of the minerals as evaluated using rock analyses and data obtained under acidic conditions from humidity cells.

Leaching processes depend on the solubility of secondary minerals most of which are not pure forms. As a result, leaching would be characterized using the SMWMP experiment designed to understand contaminant solubility, and analog data. The latter include the AMAX stockpiles, data from the Whistle Mine, near Sudbury, Ontario and Vangorda Mine, near Faro, Yukon Territory. For some contaminants, leaching would be evaluated using thermodynamic calculations.

Uncertainty in the calculations would be based on the distributions shown by the individual datasets.

Attachments

Attachment 1
Relationships Between Total Sulfur and Other Elements Determined by Aqua Regia



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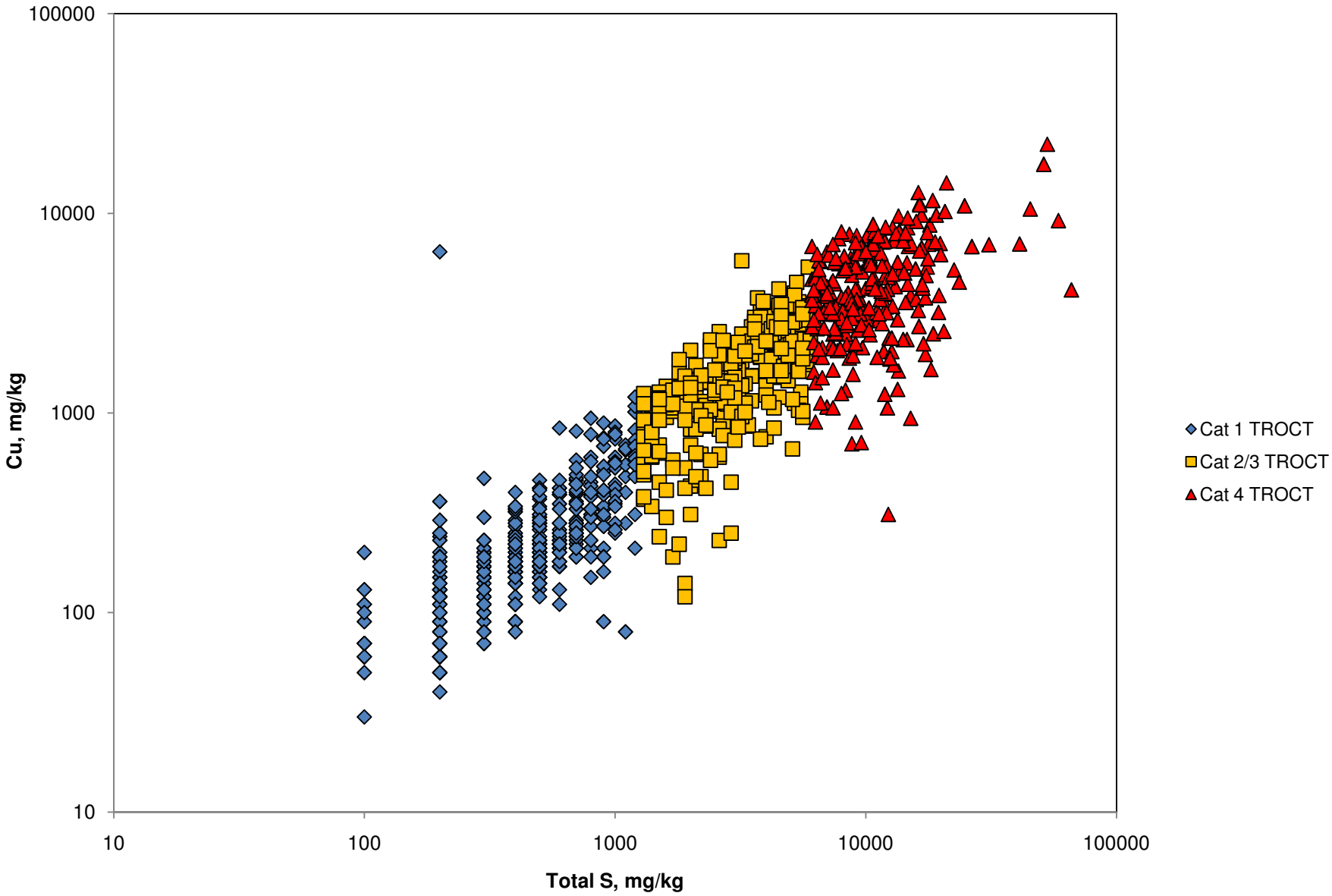

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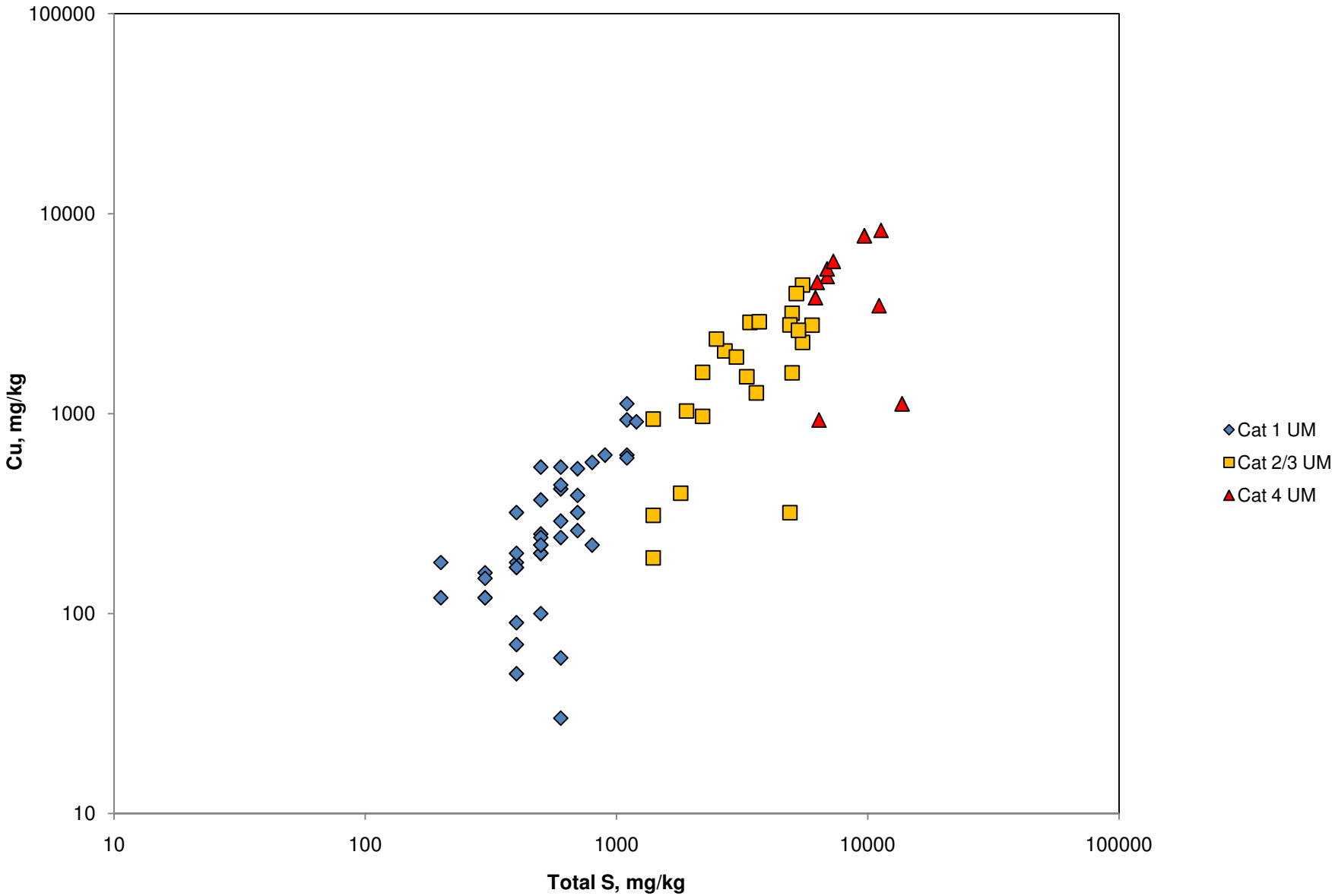

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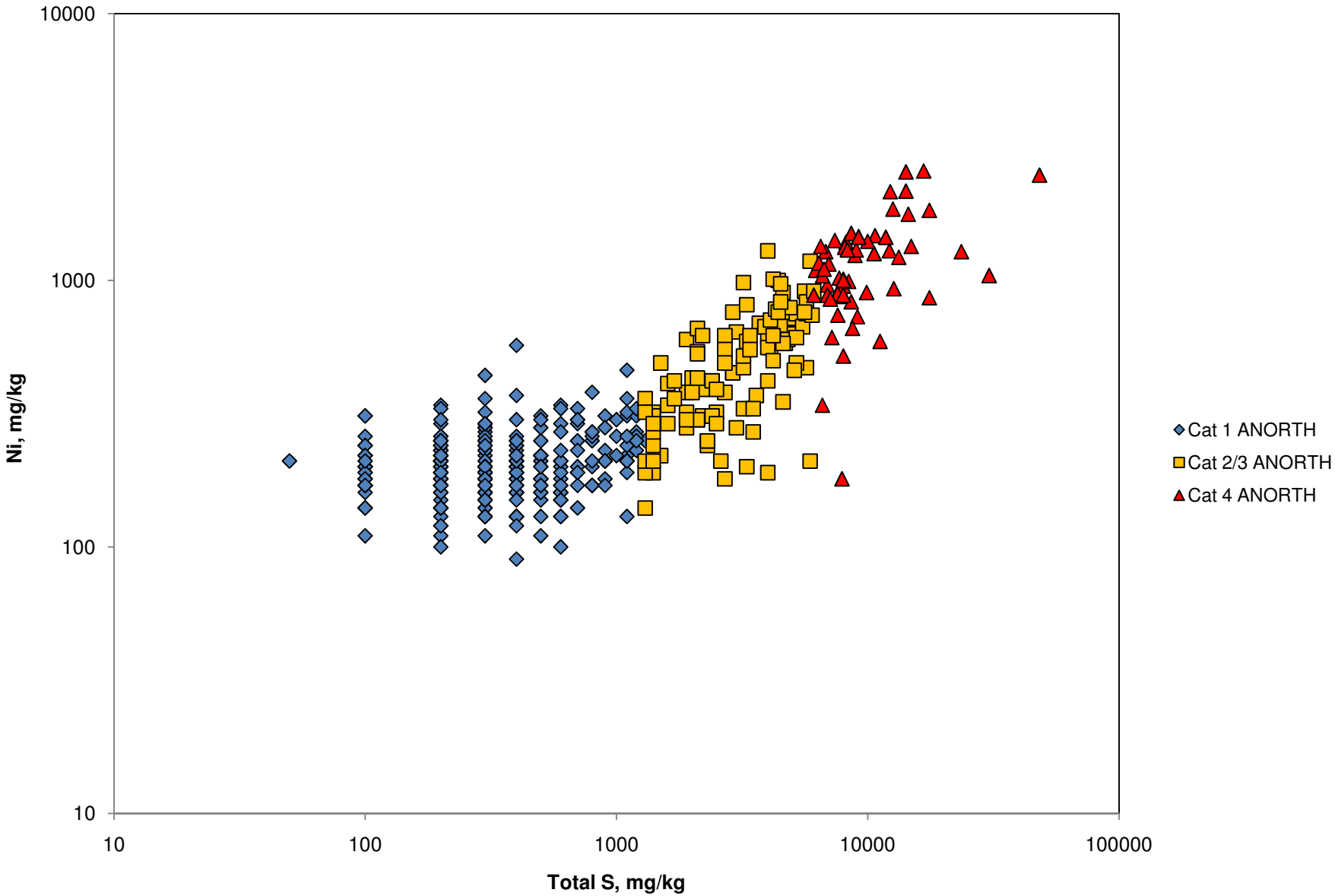

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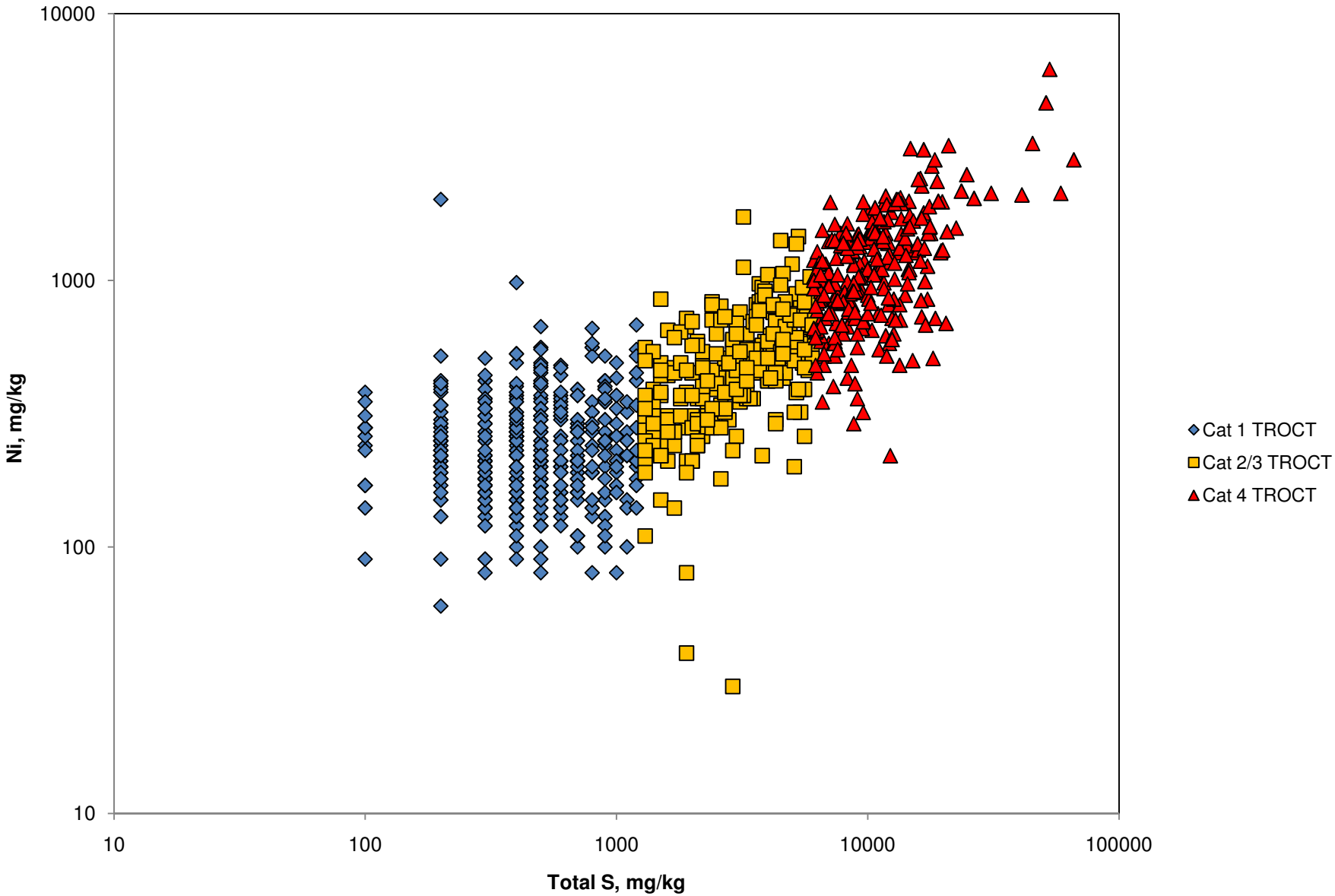

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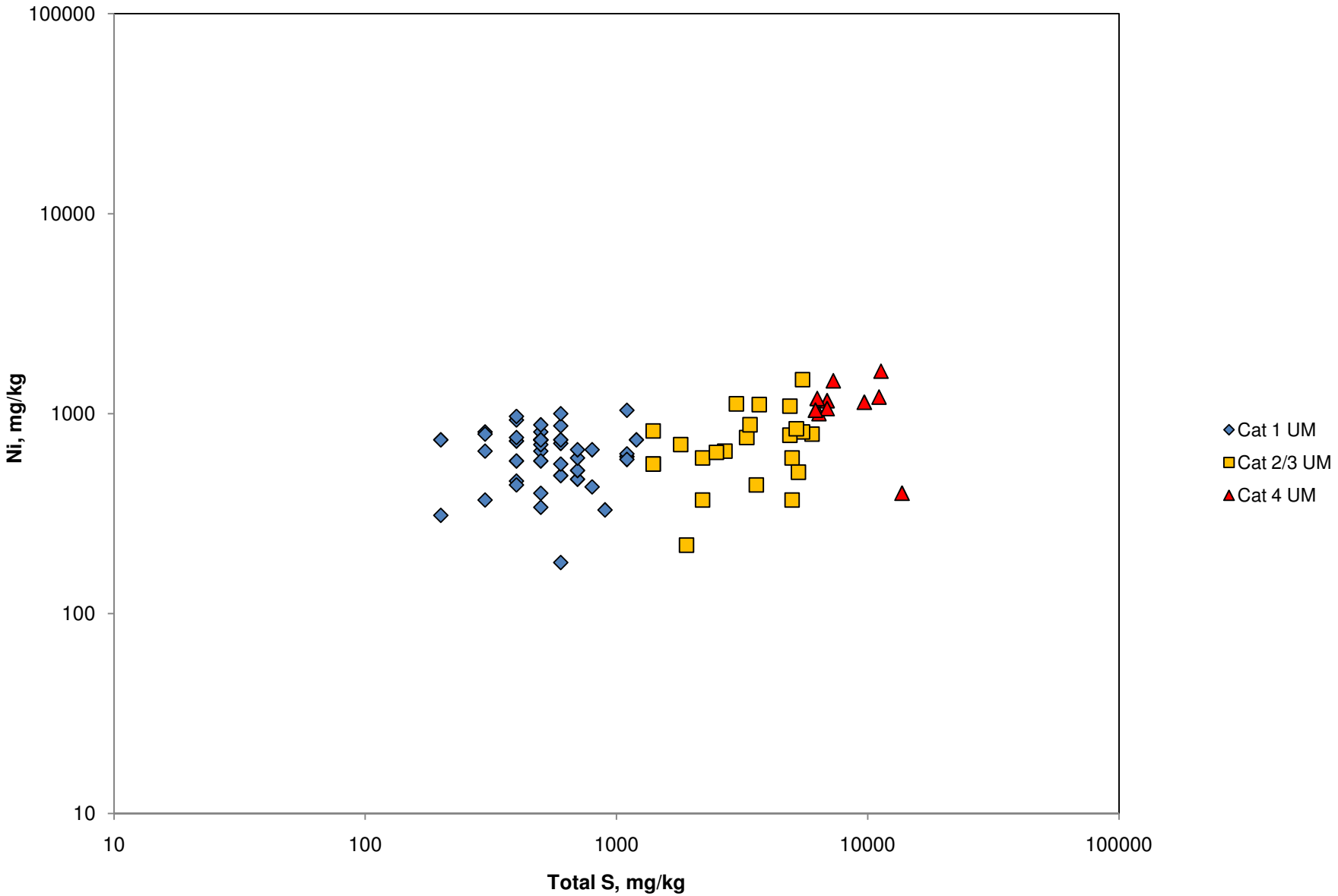

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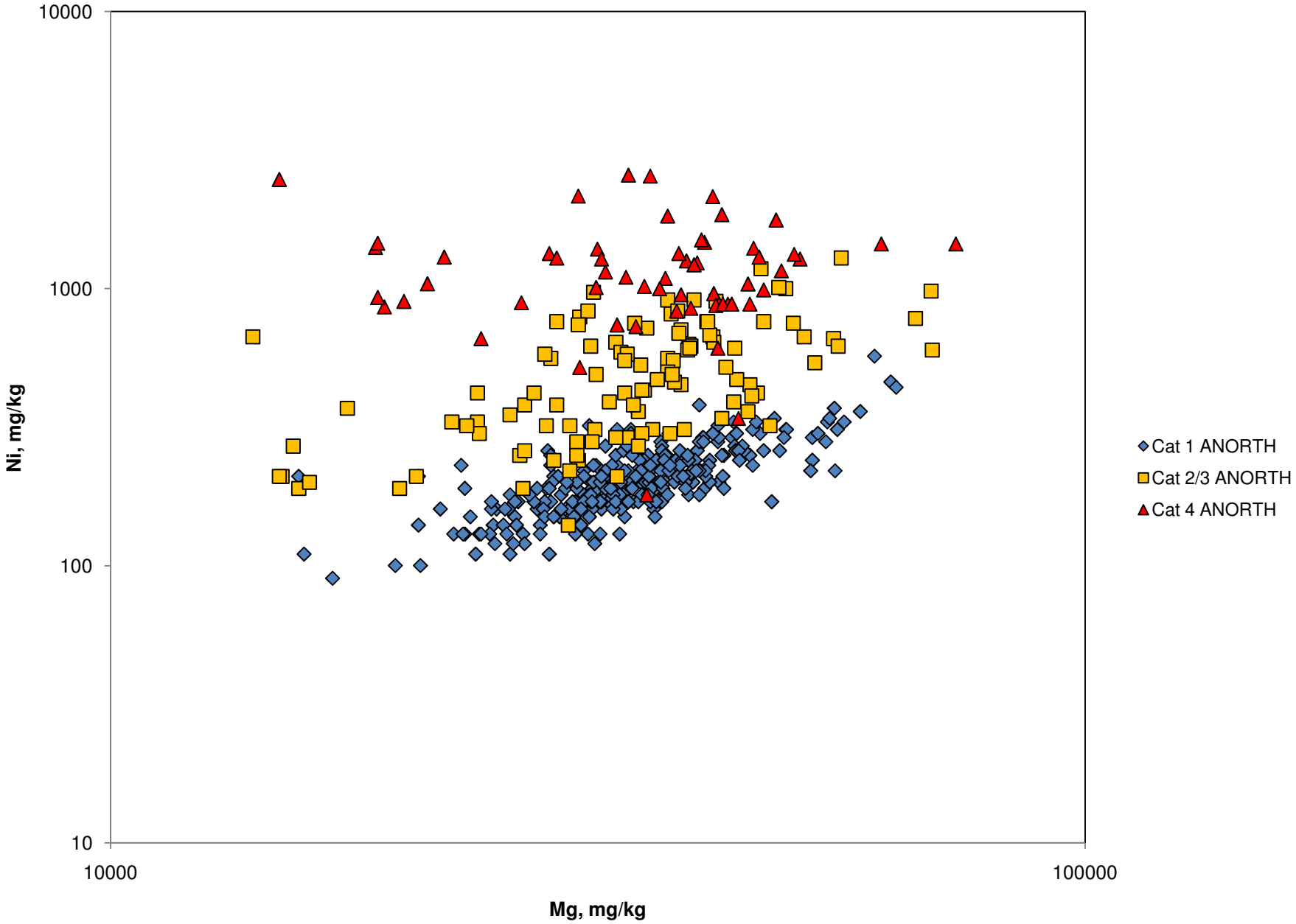

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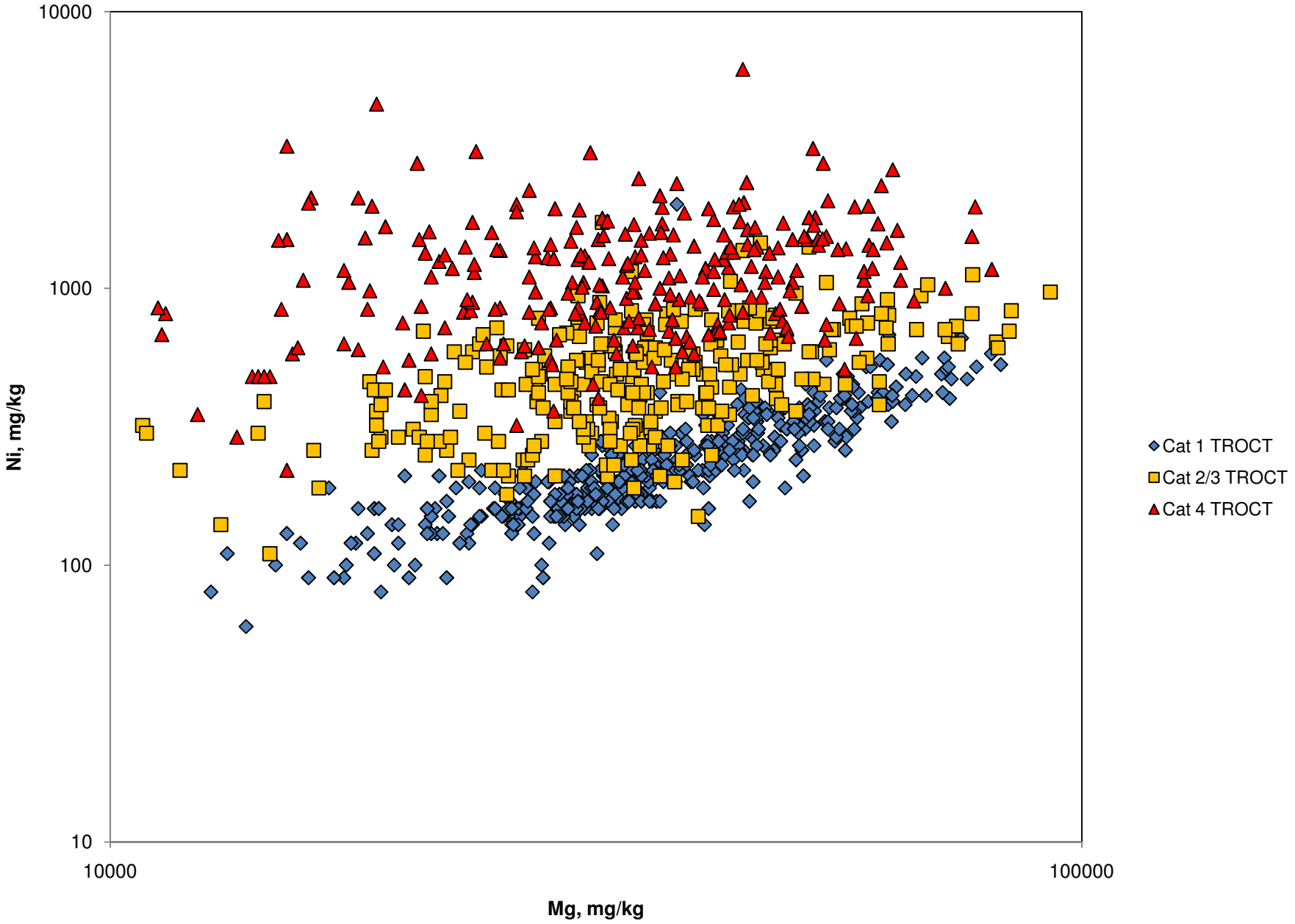

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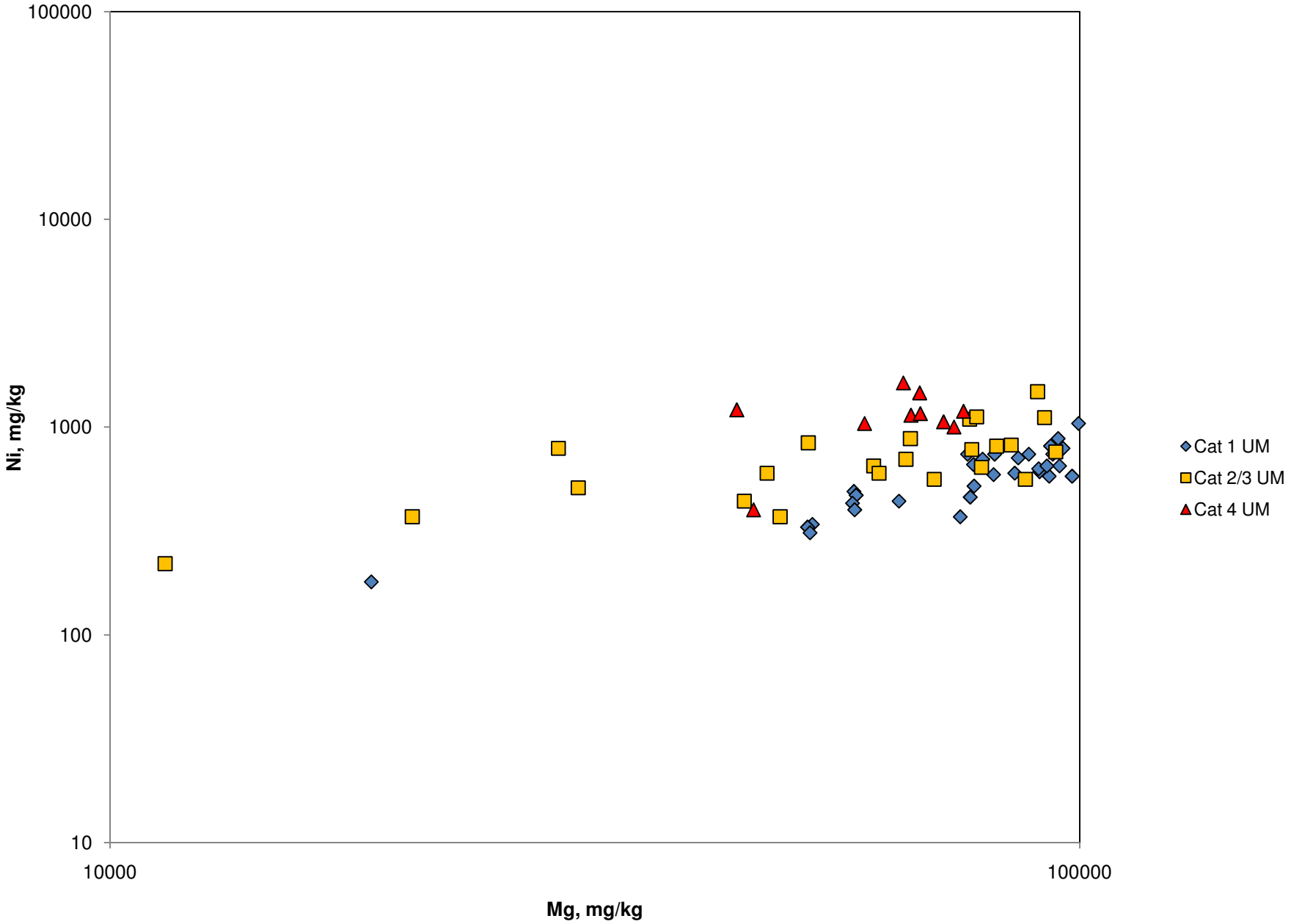

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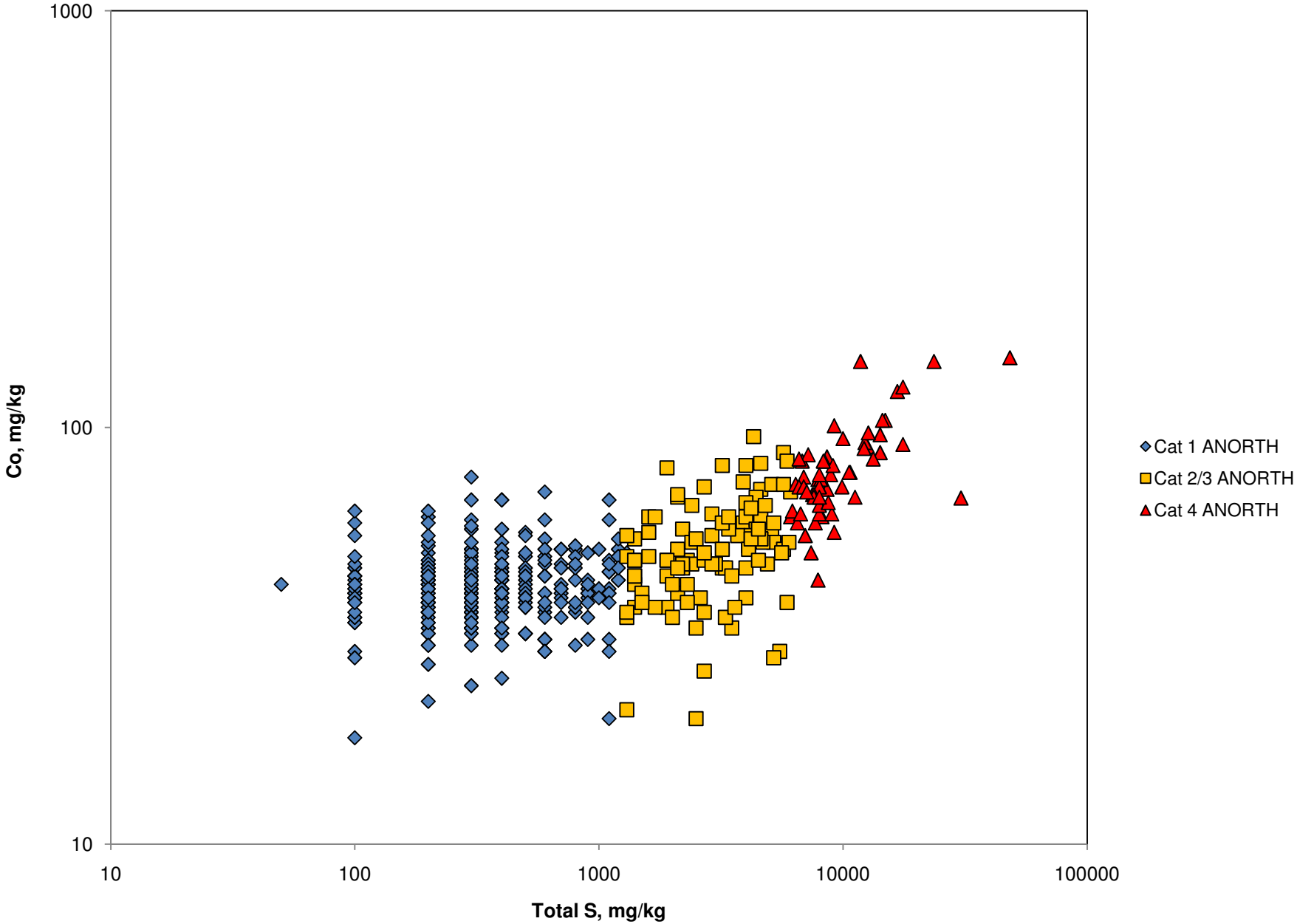

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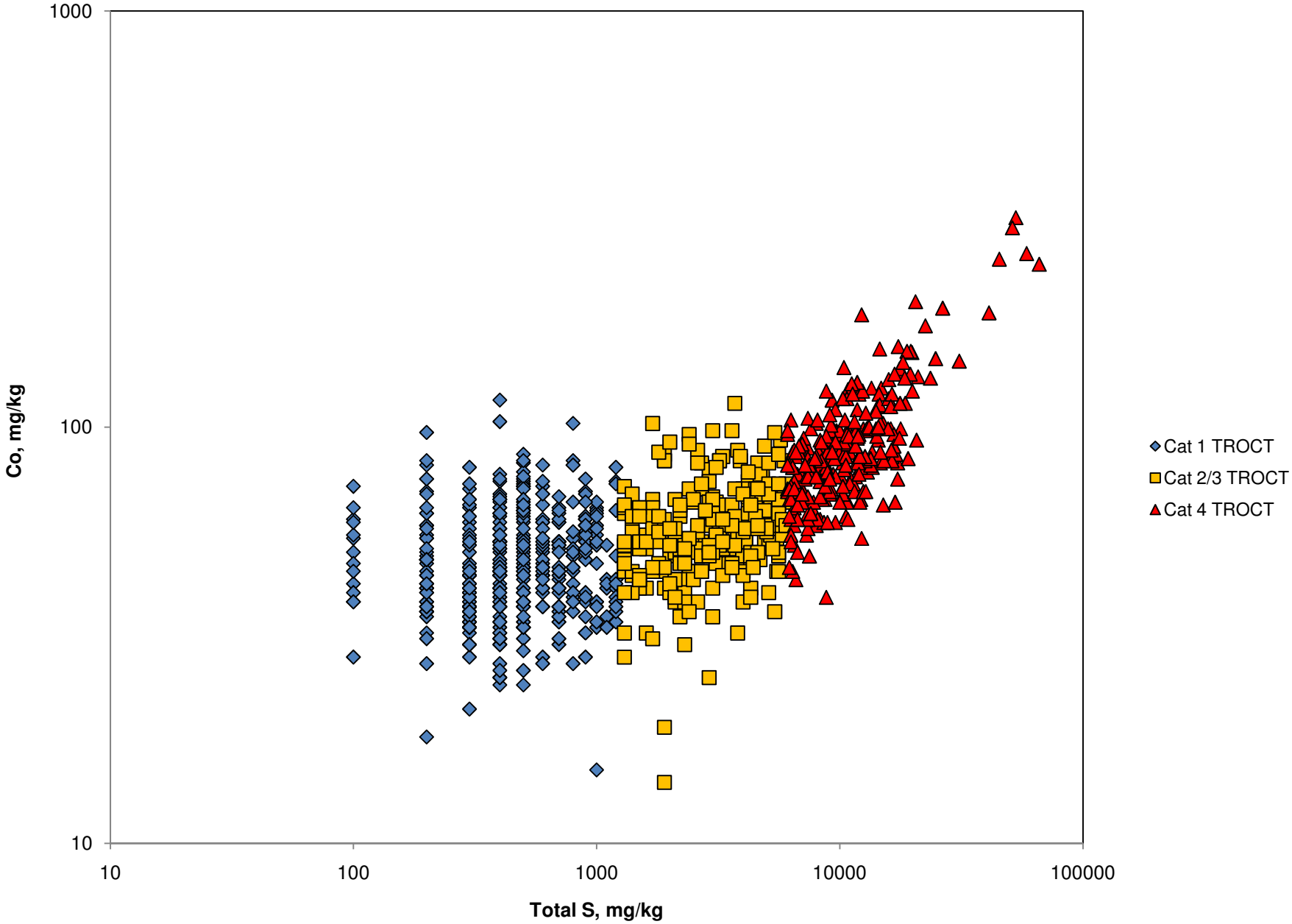

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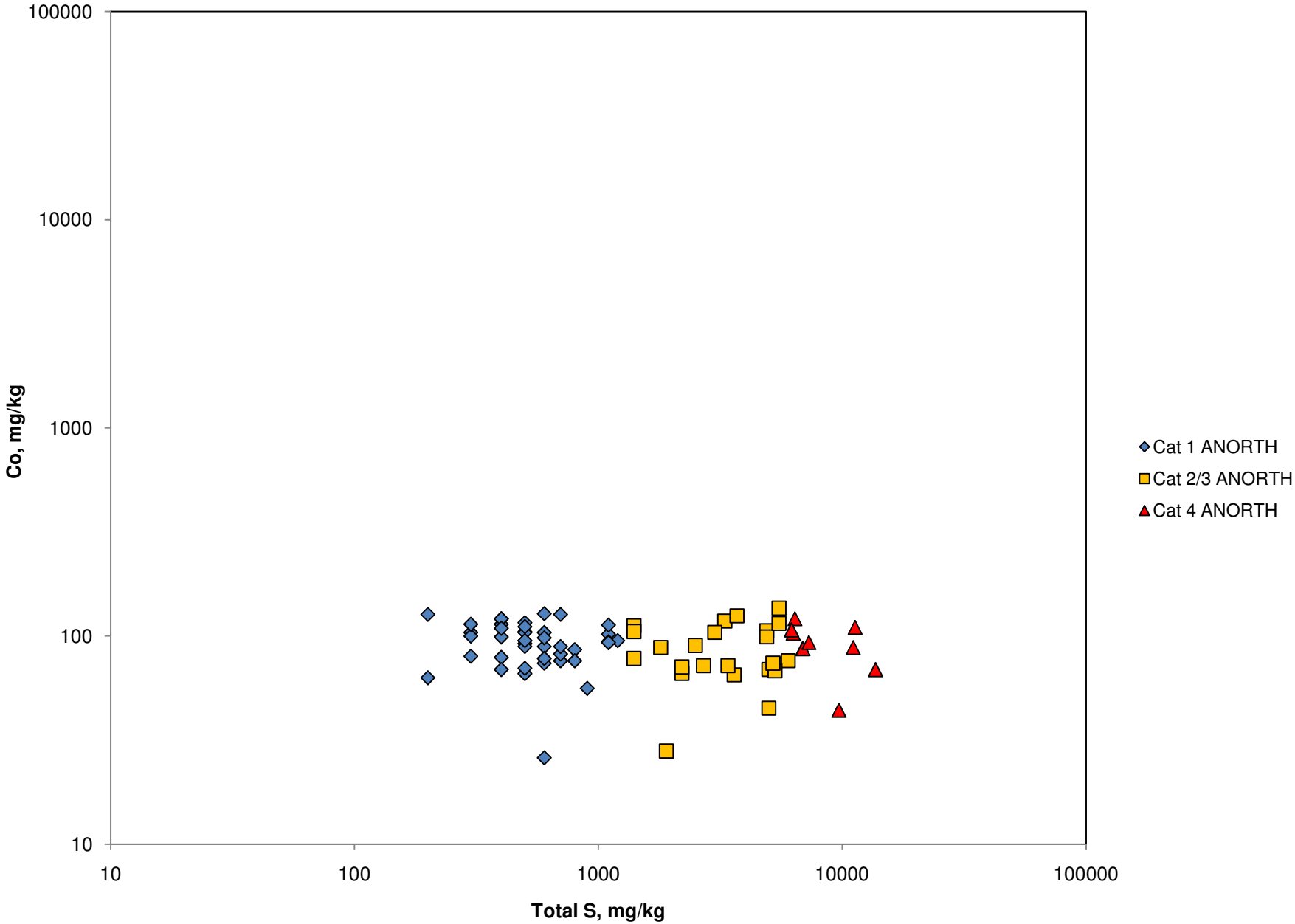

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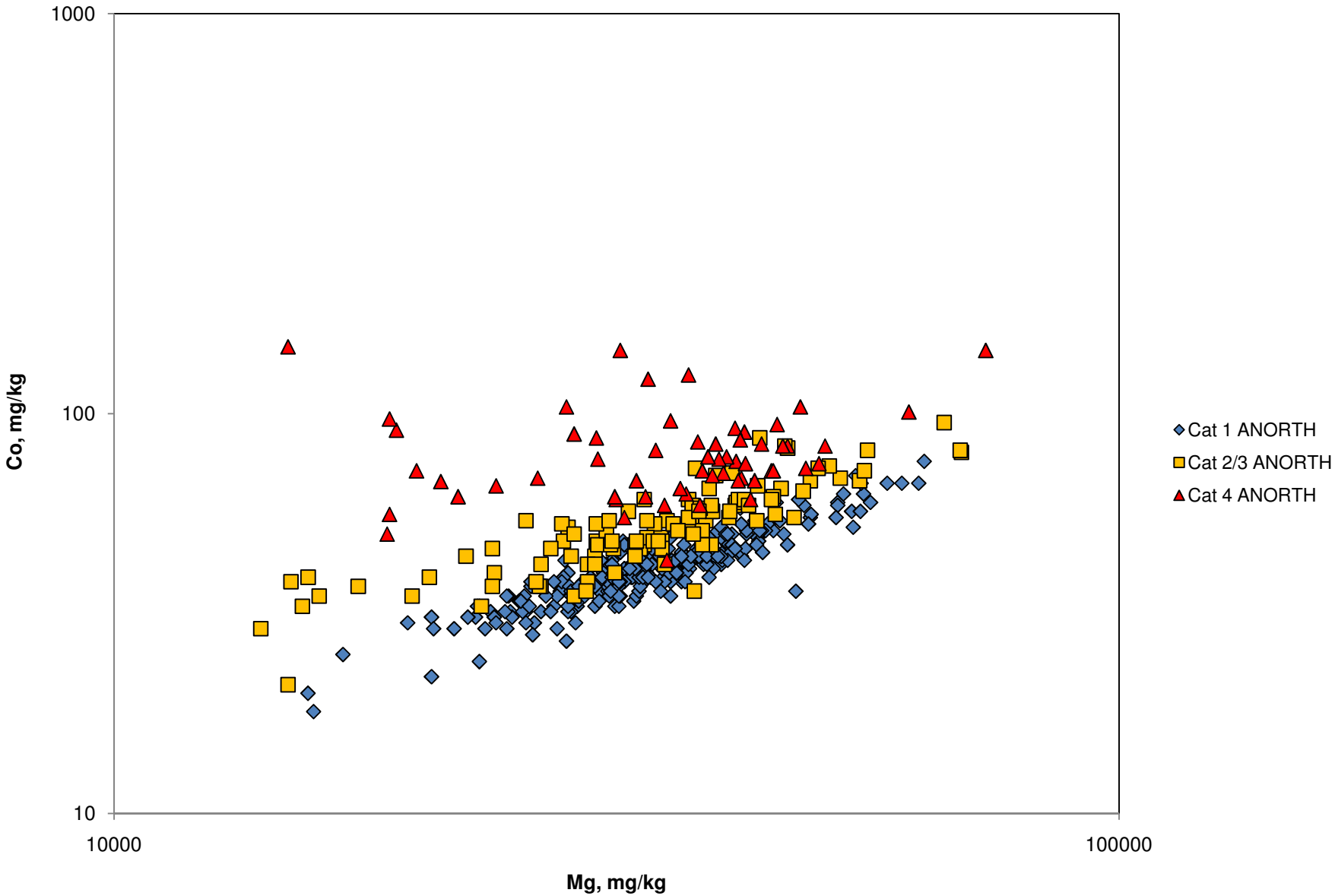

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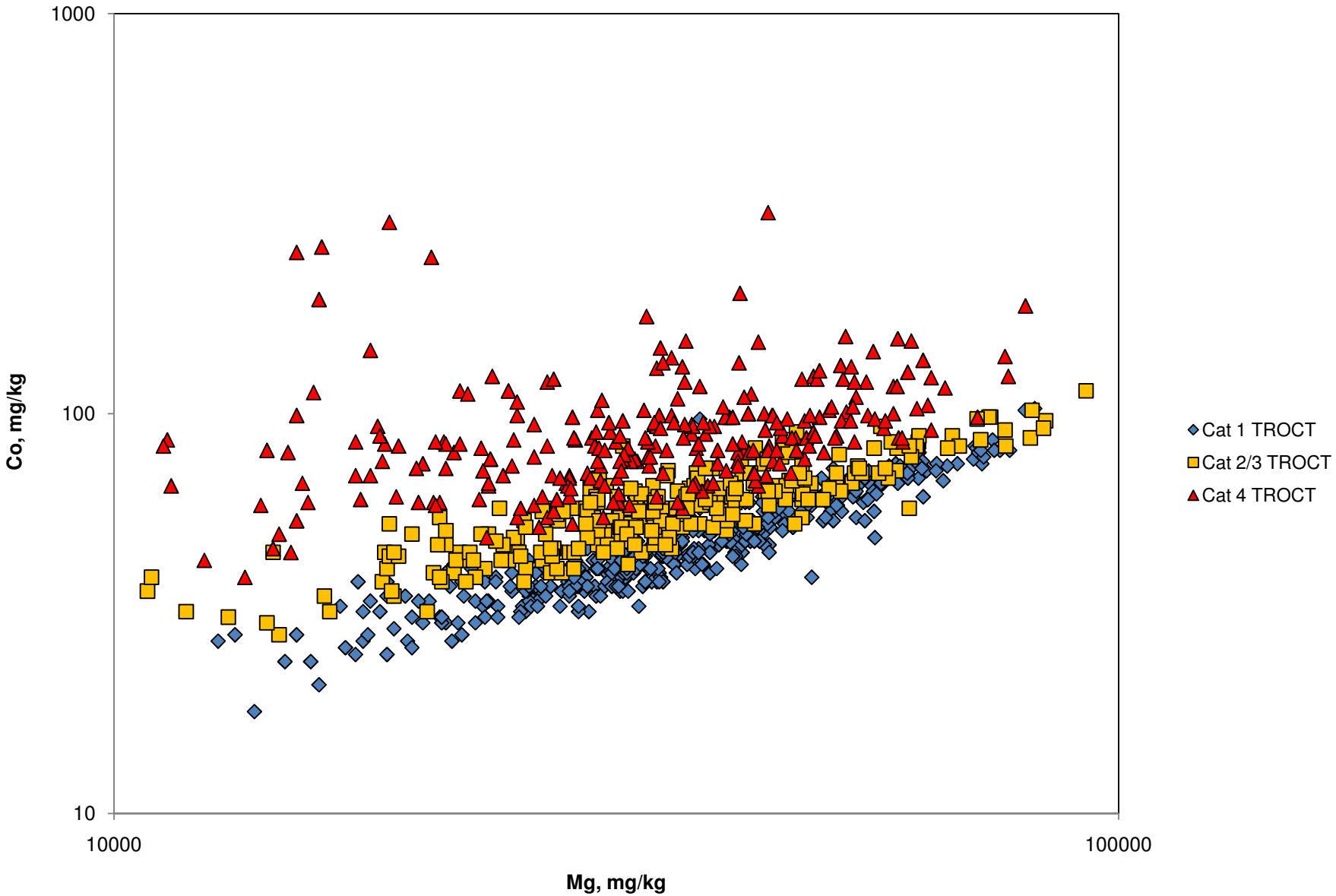

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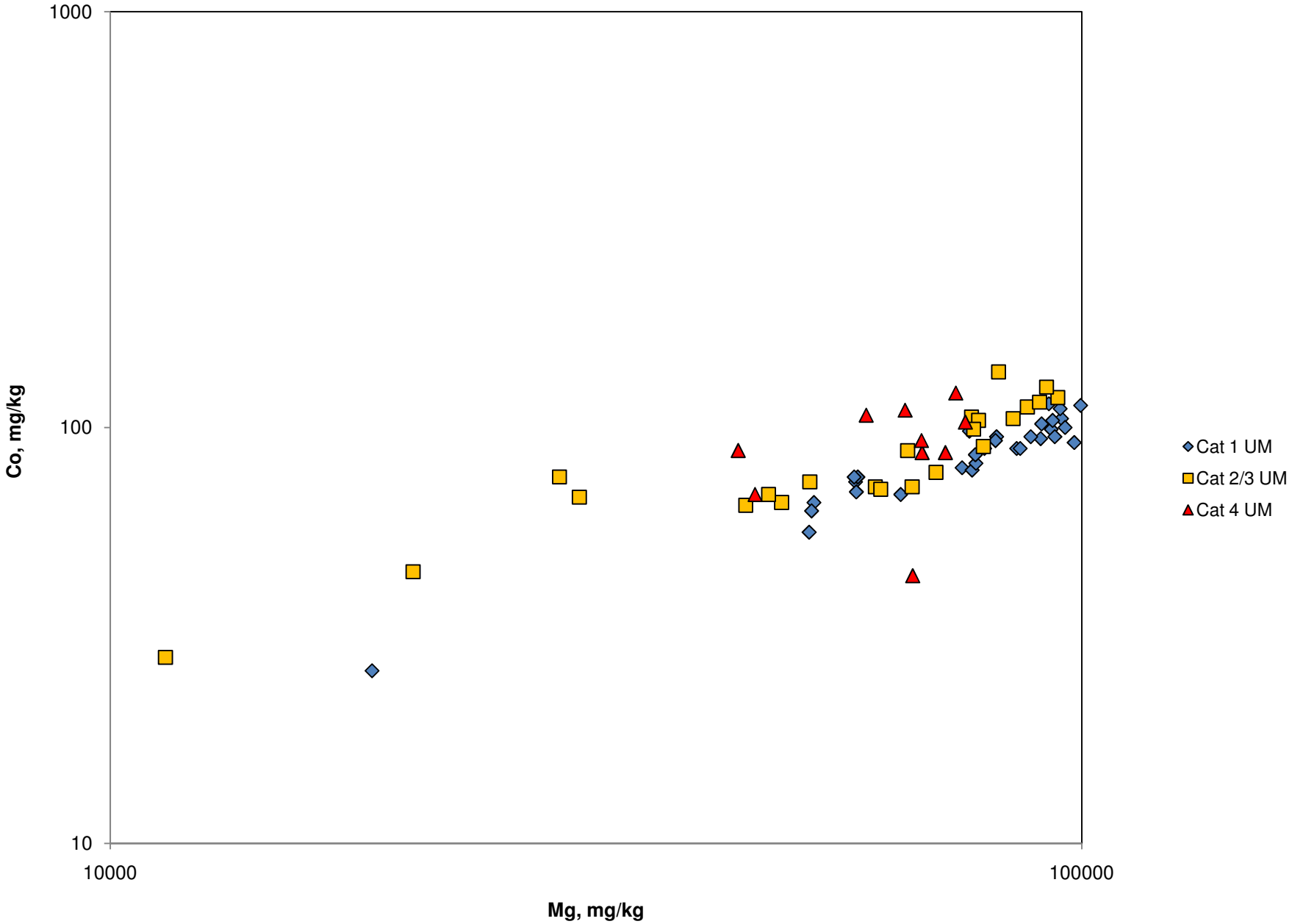

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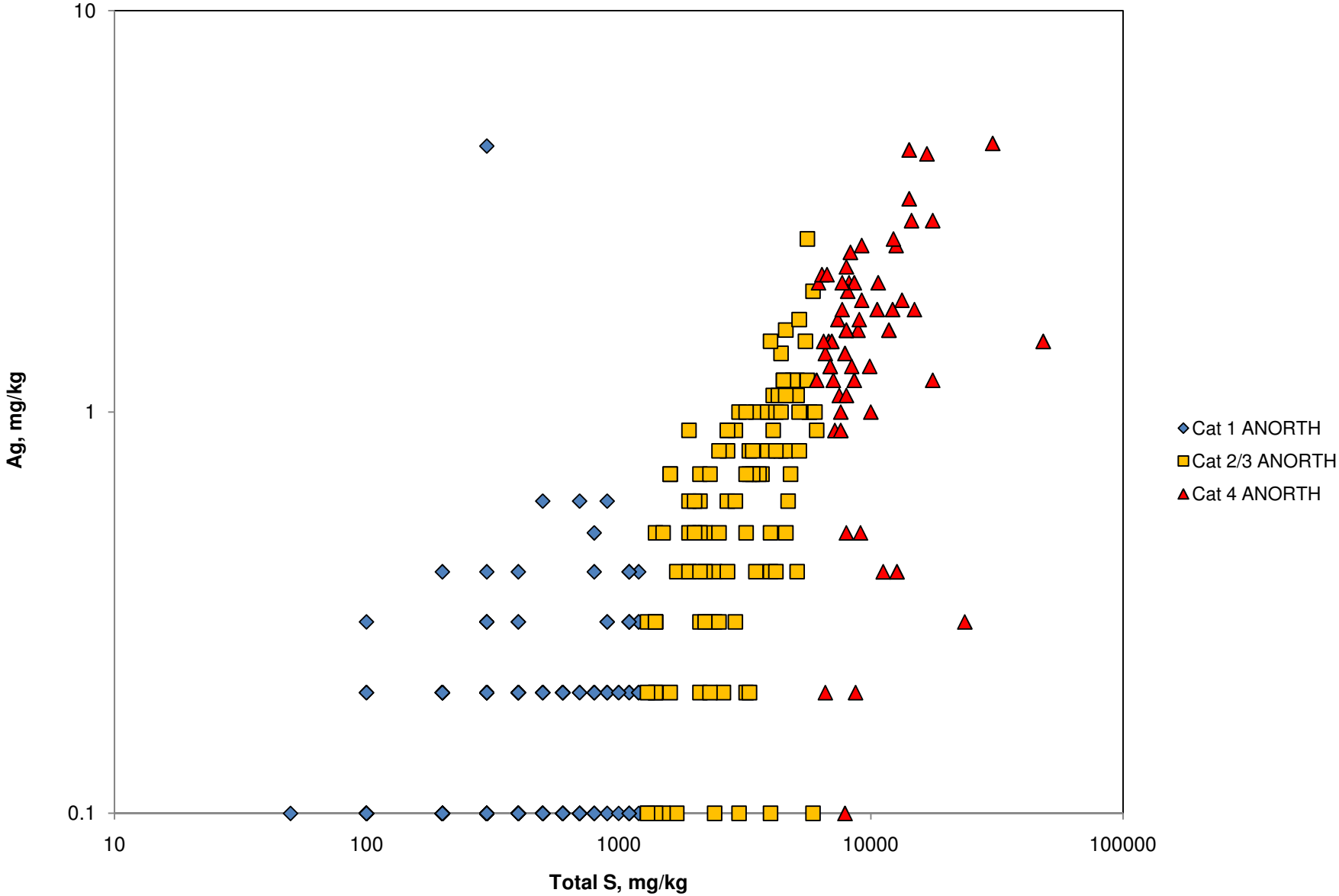

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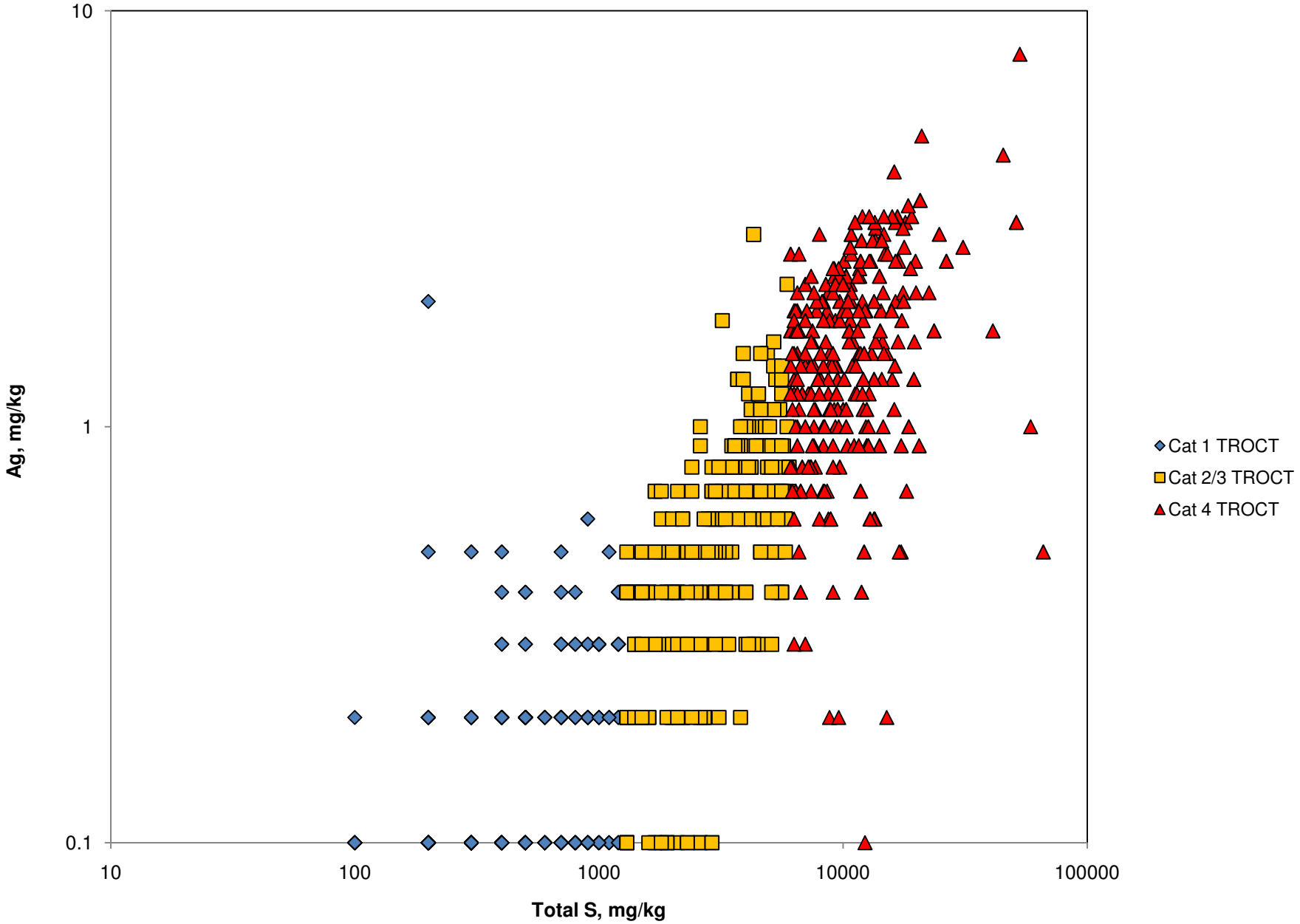

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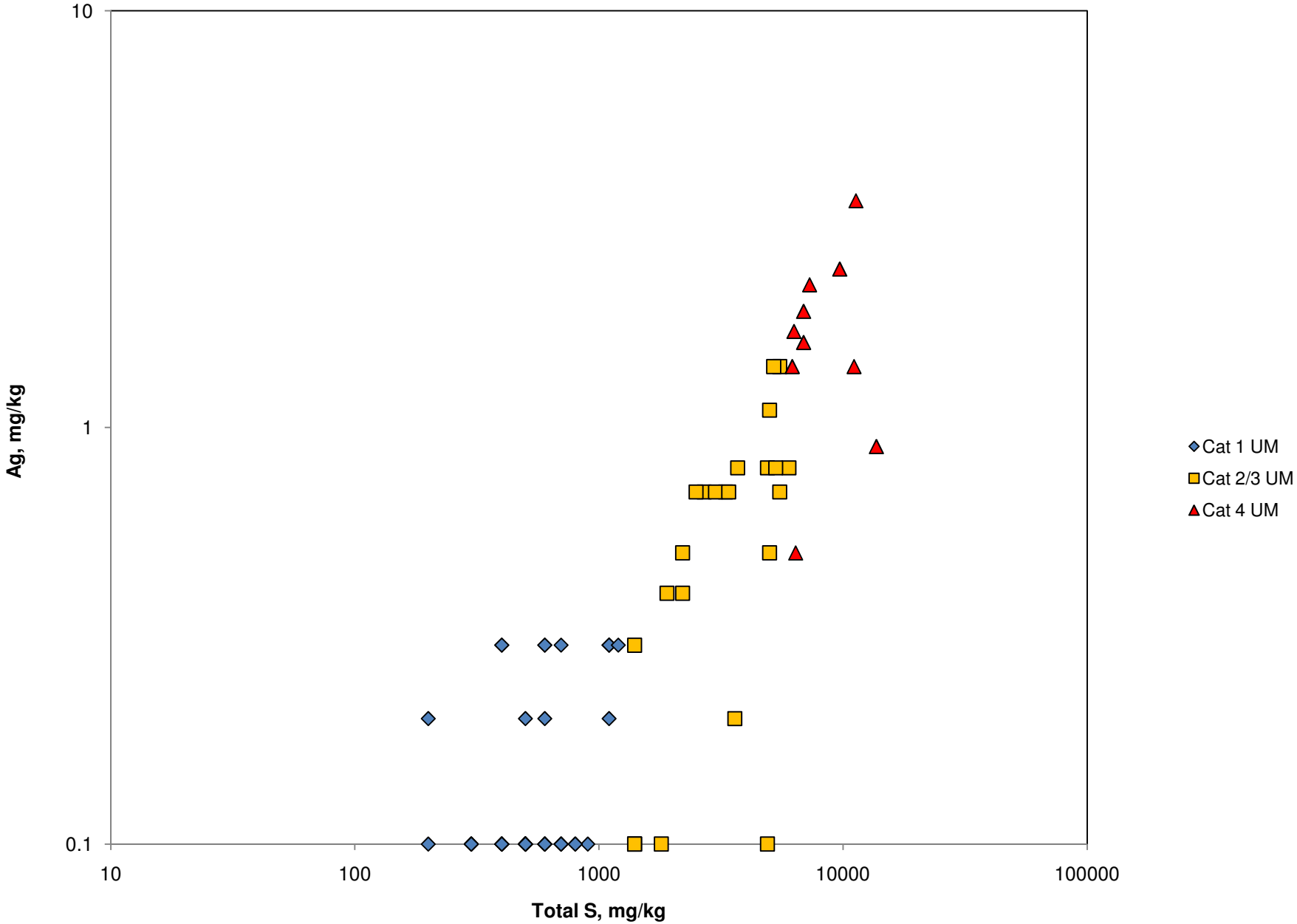

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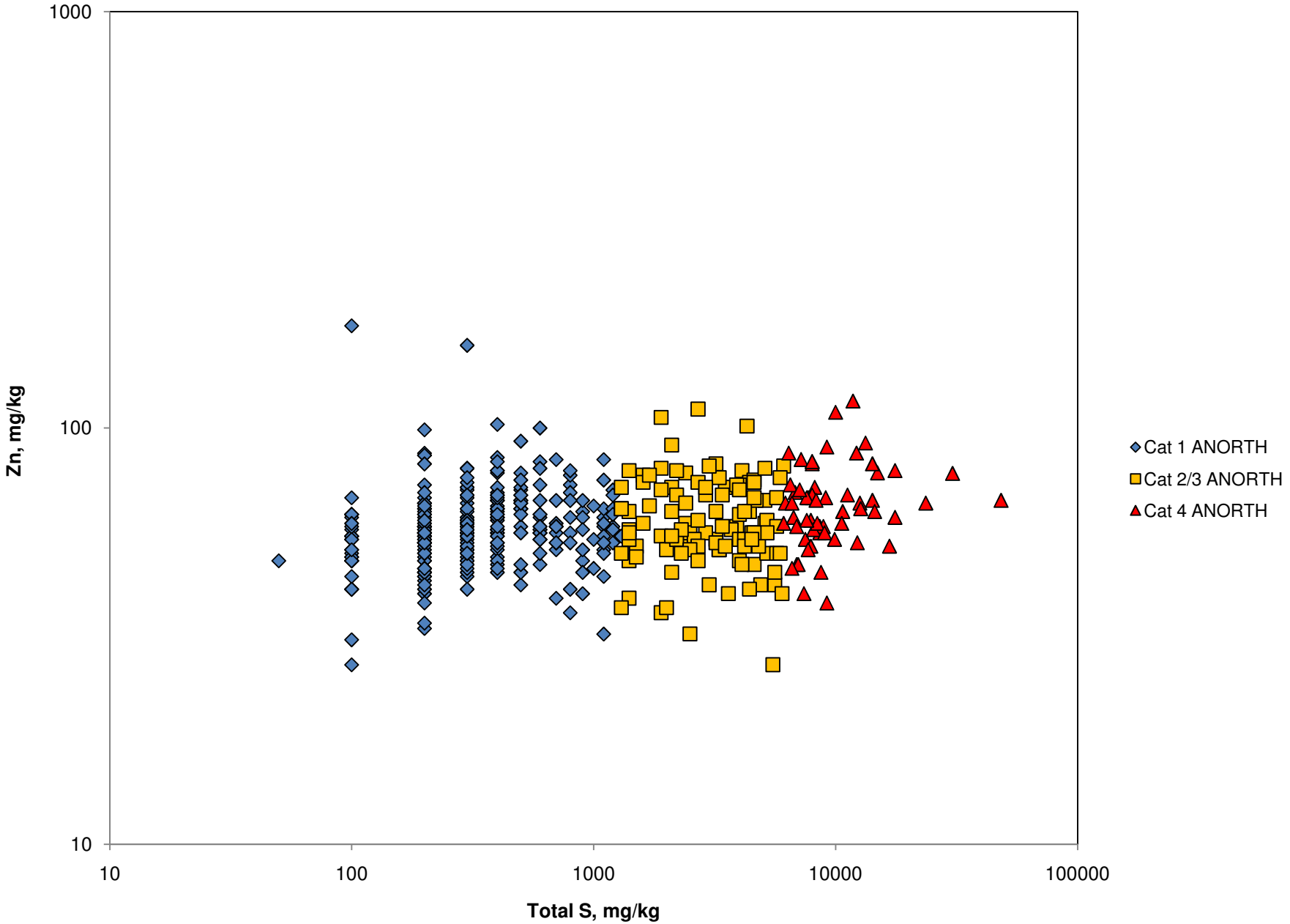

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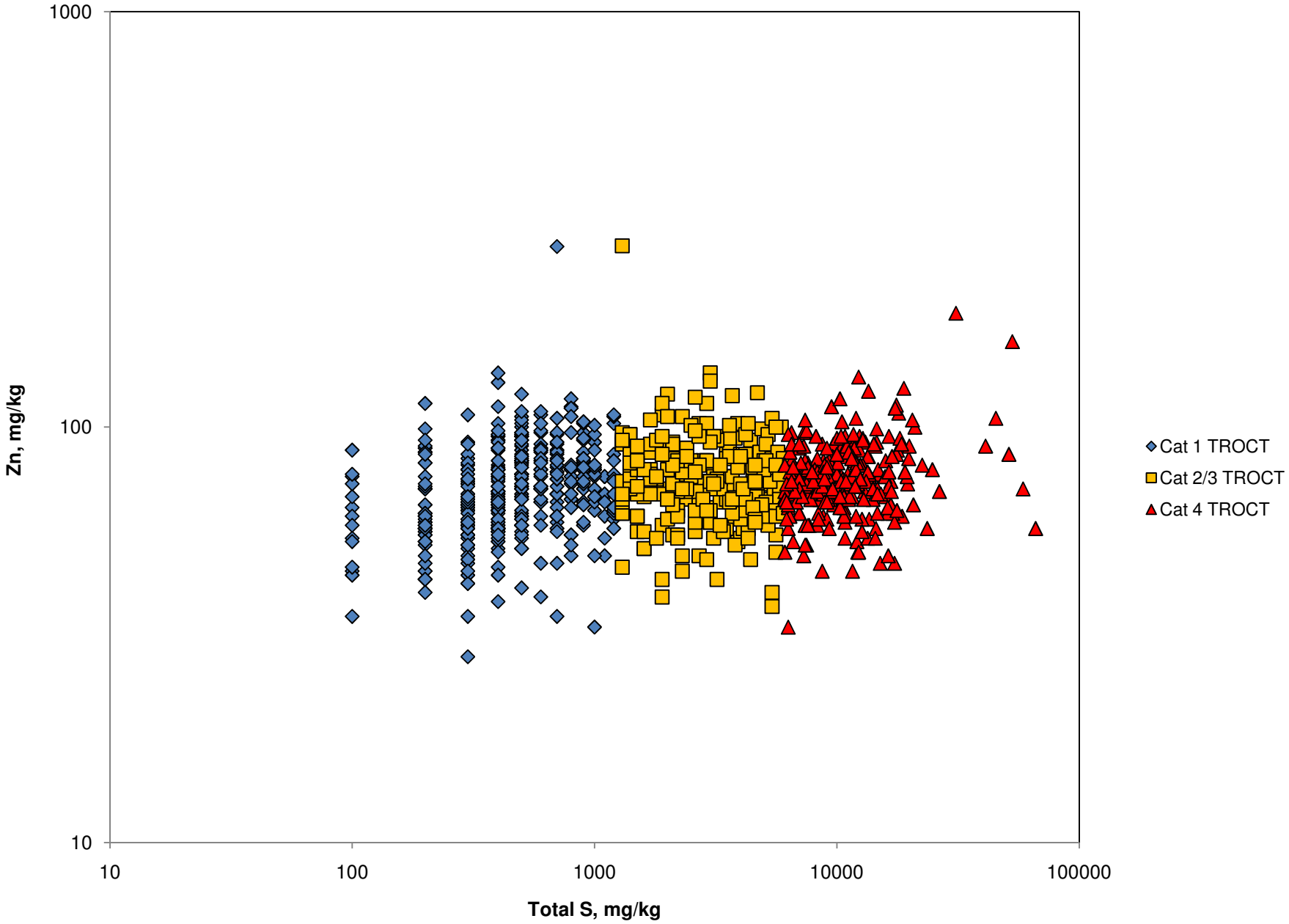

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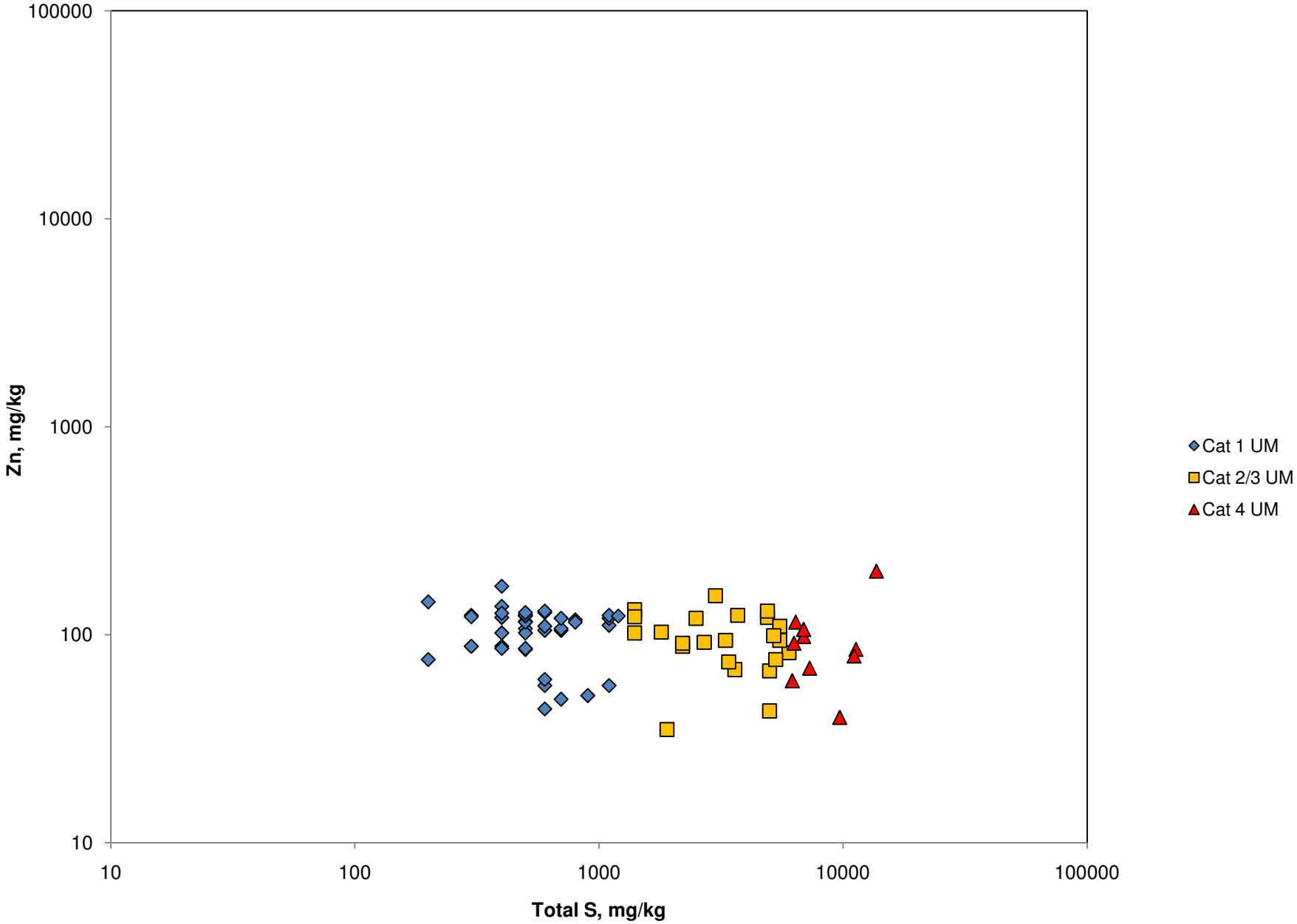

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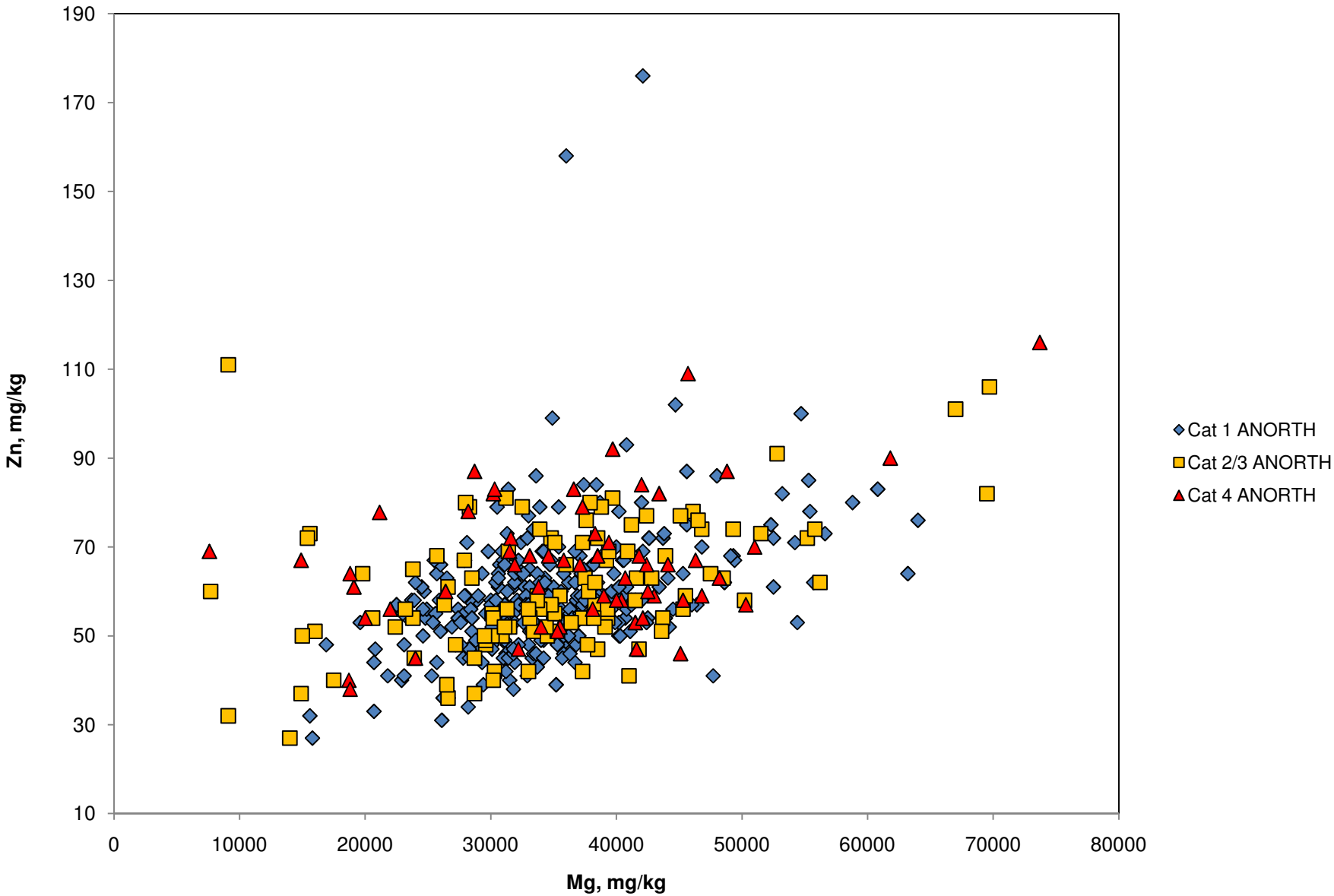

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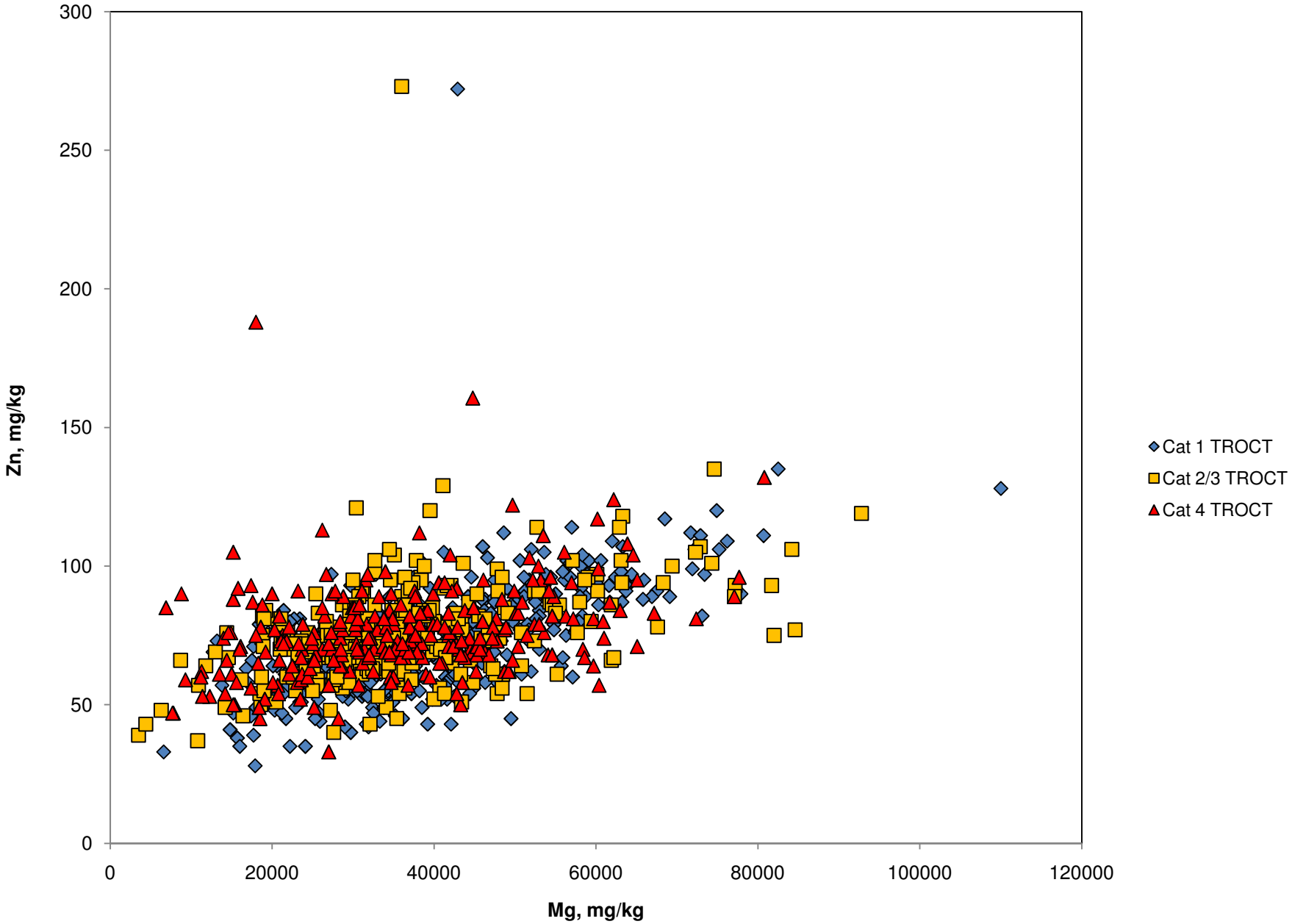

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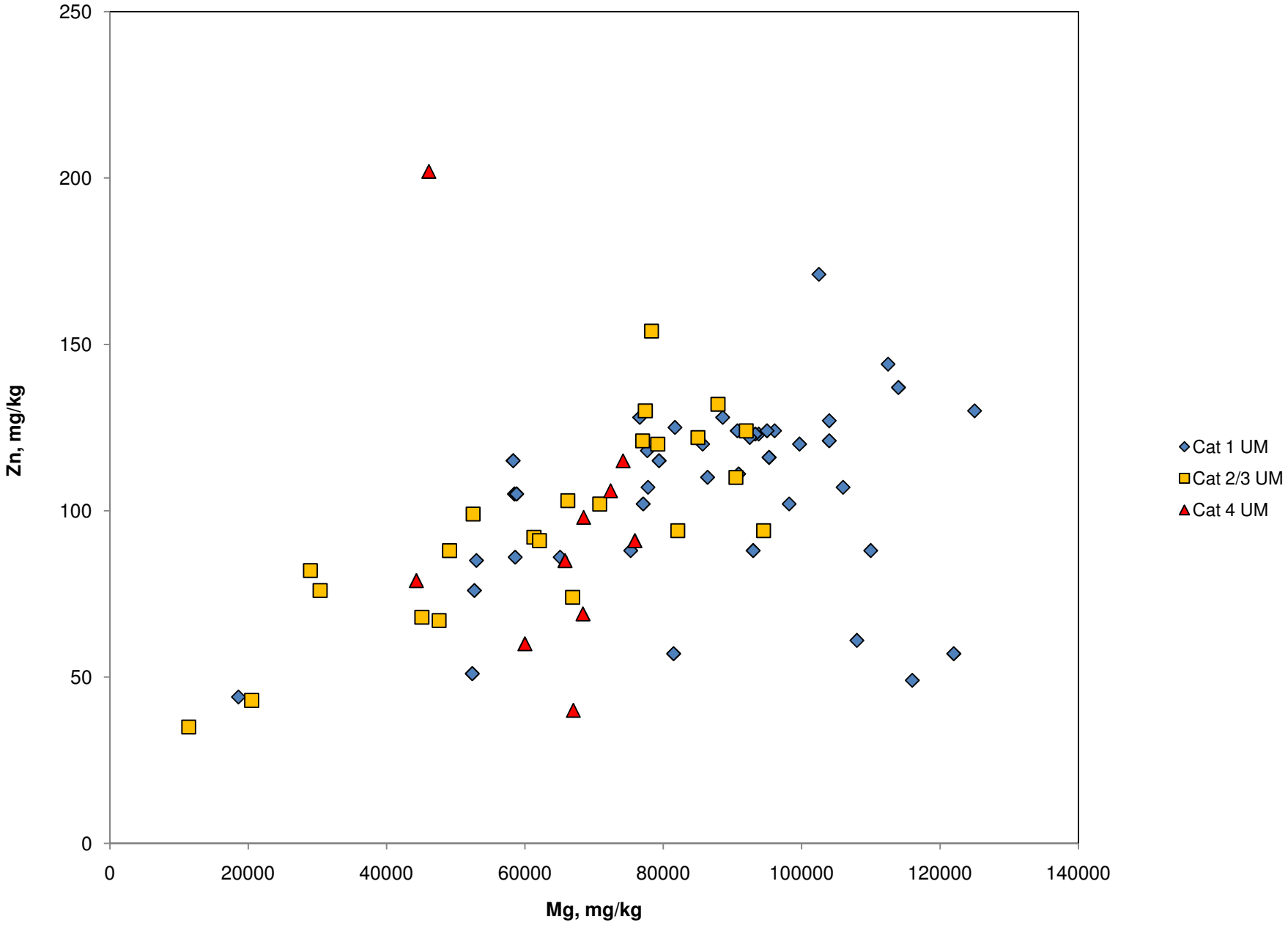

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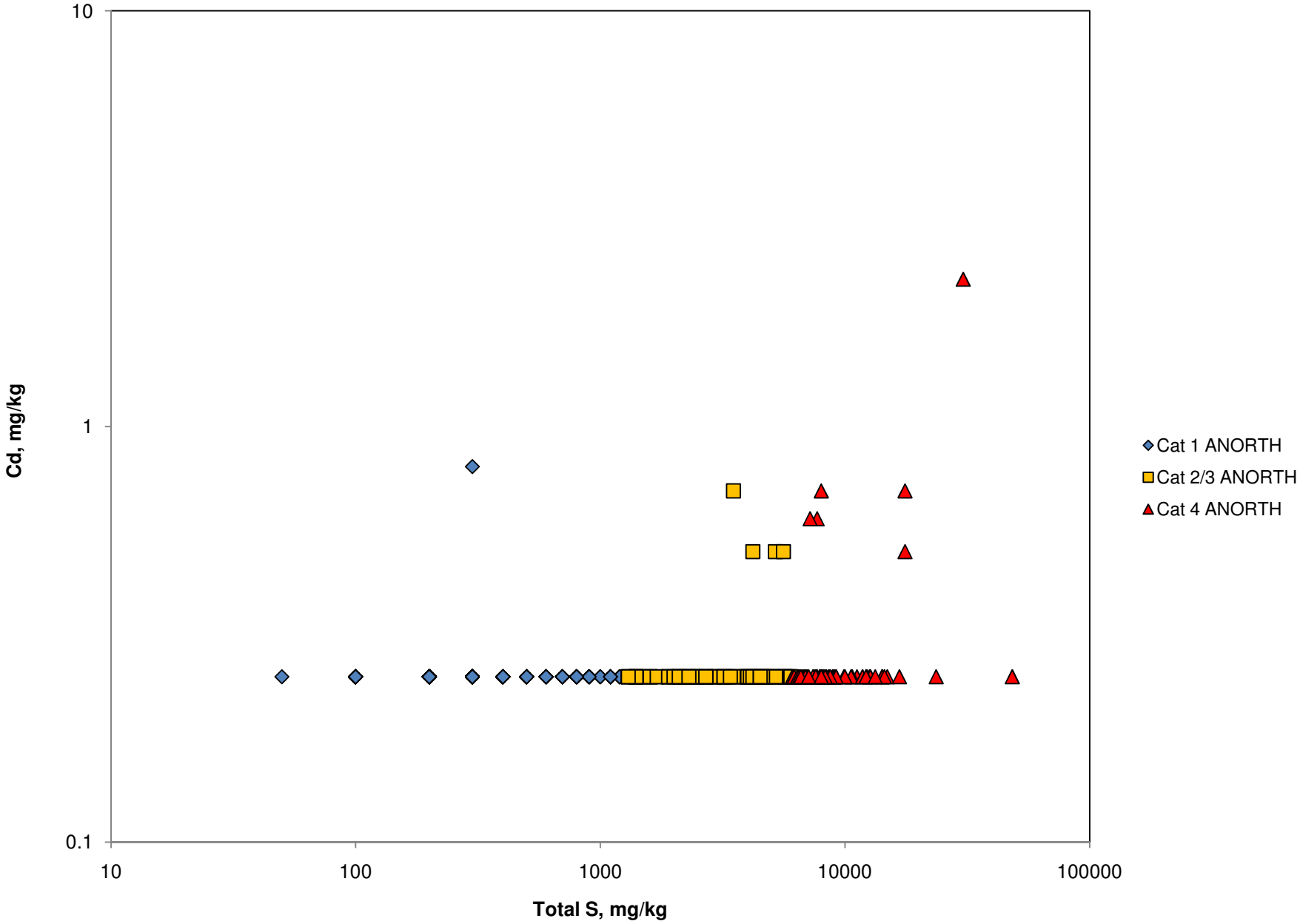

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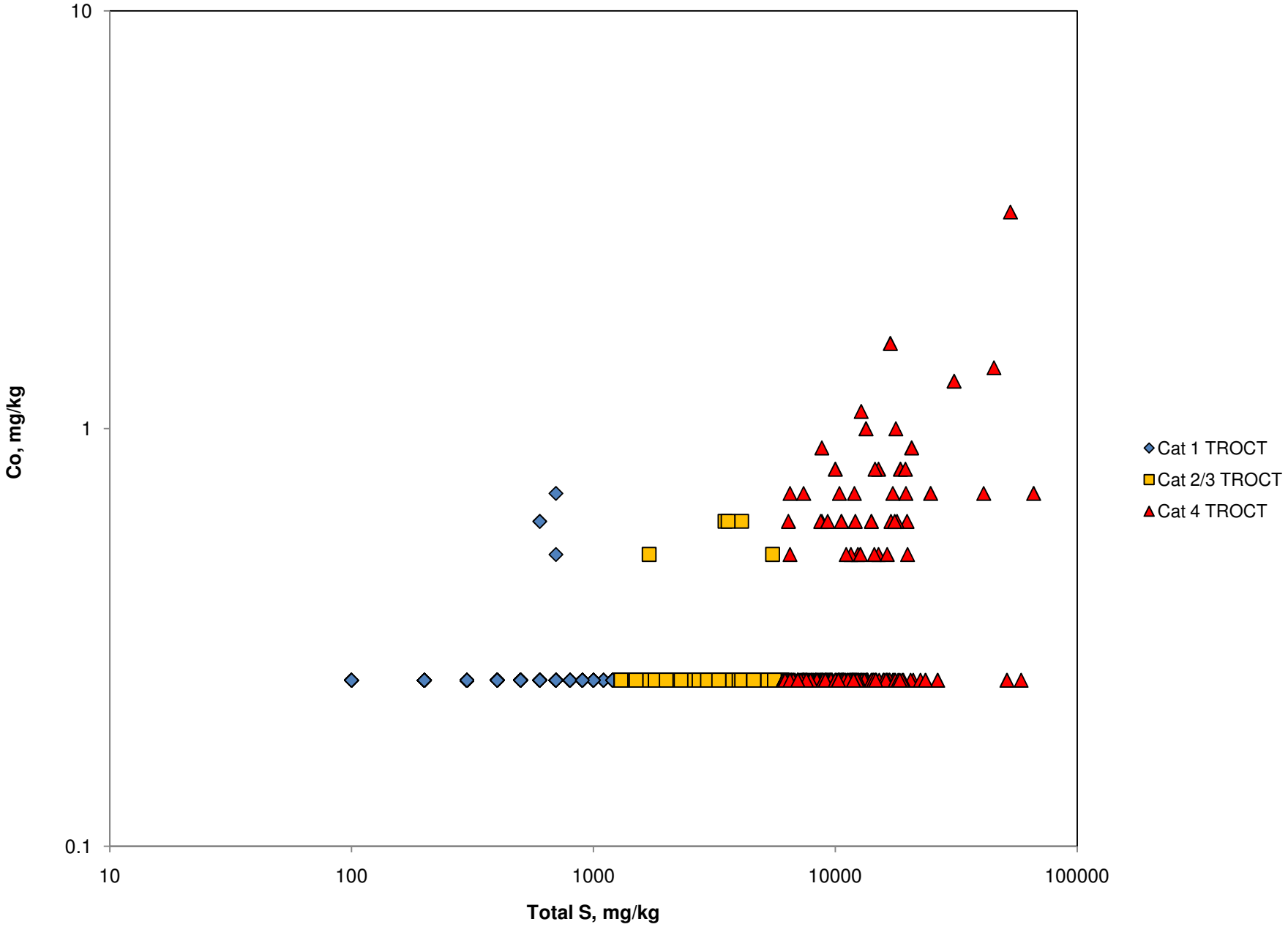
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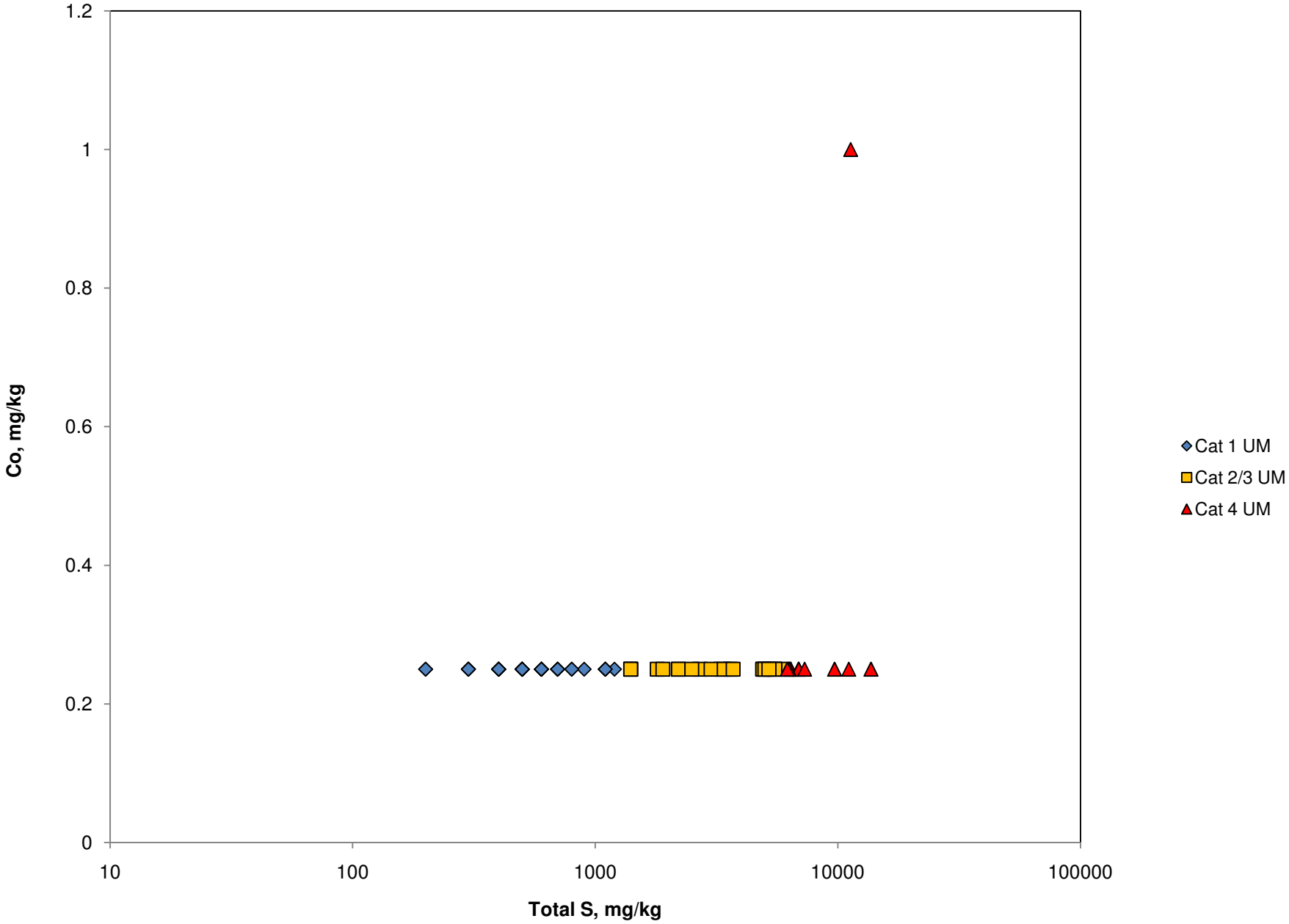

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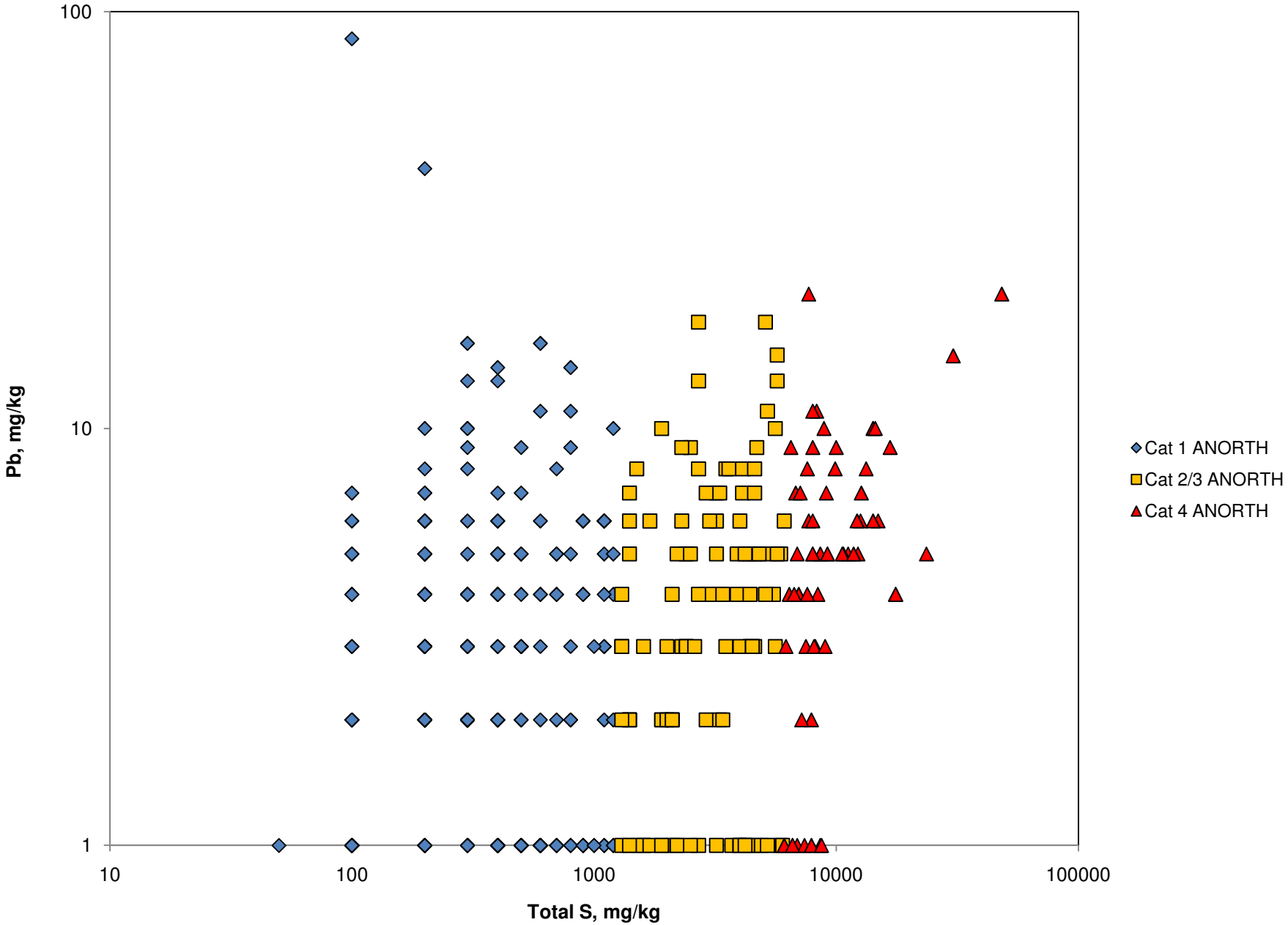

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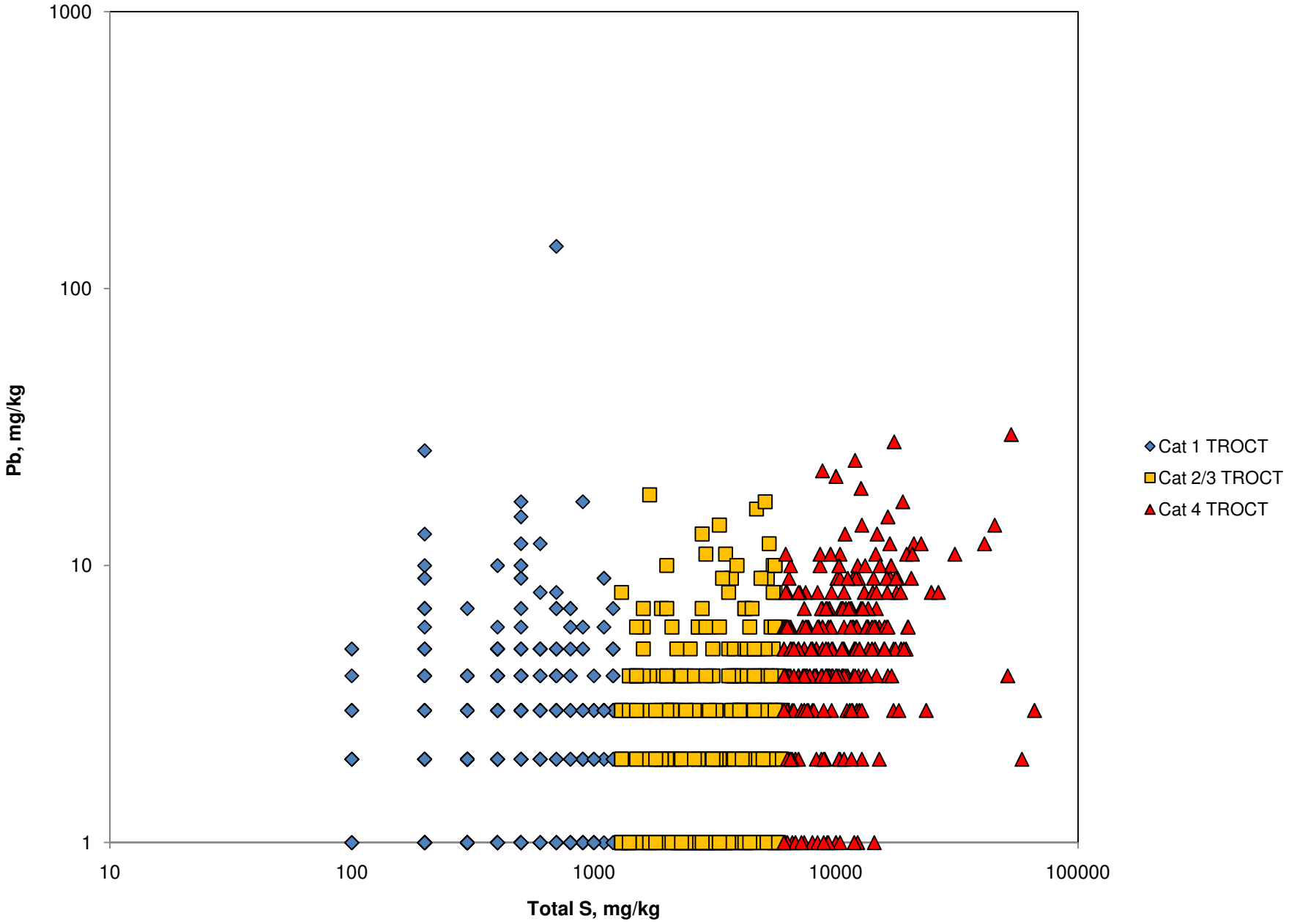

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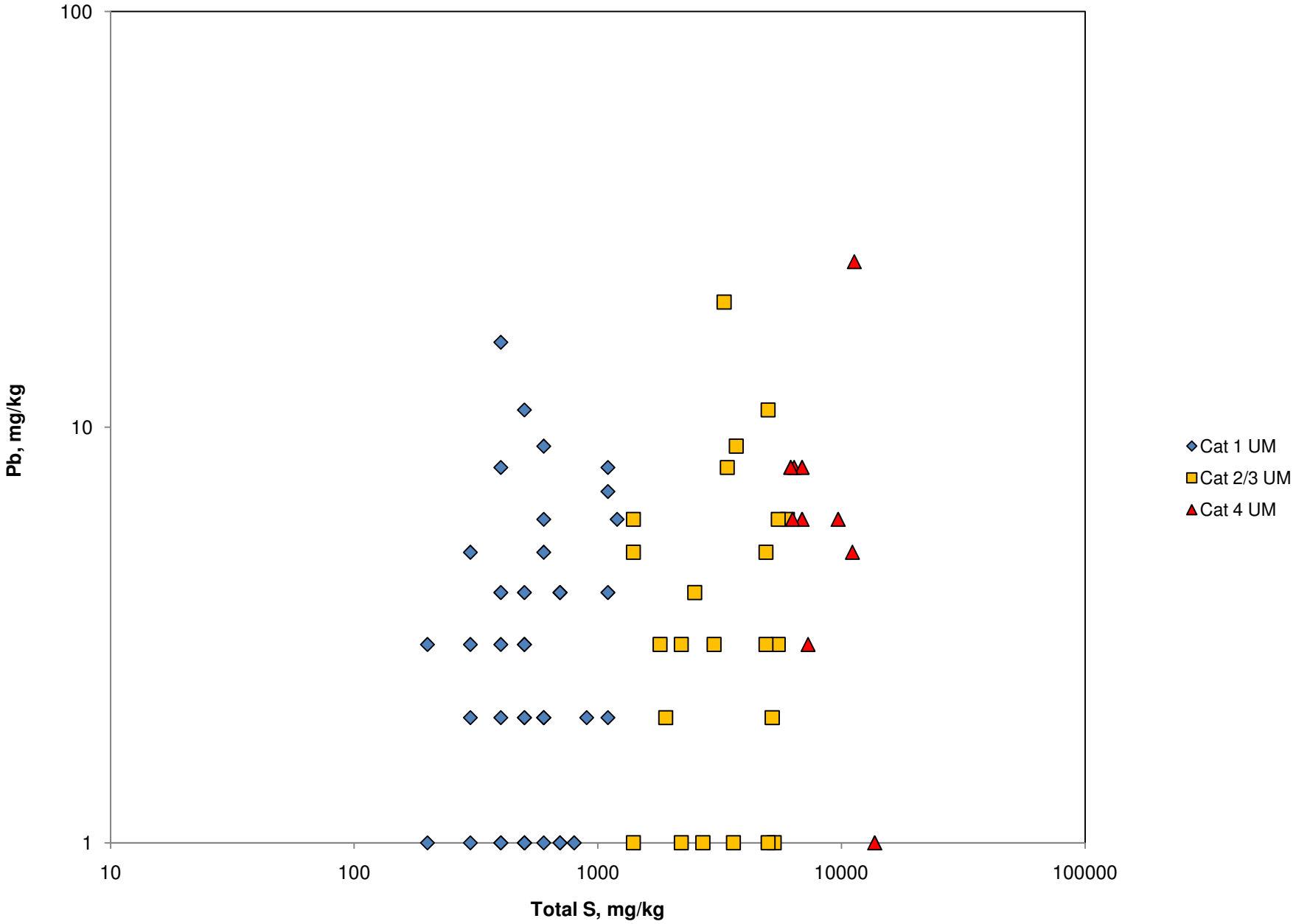

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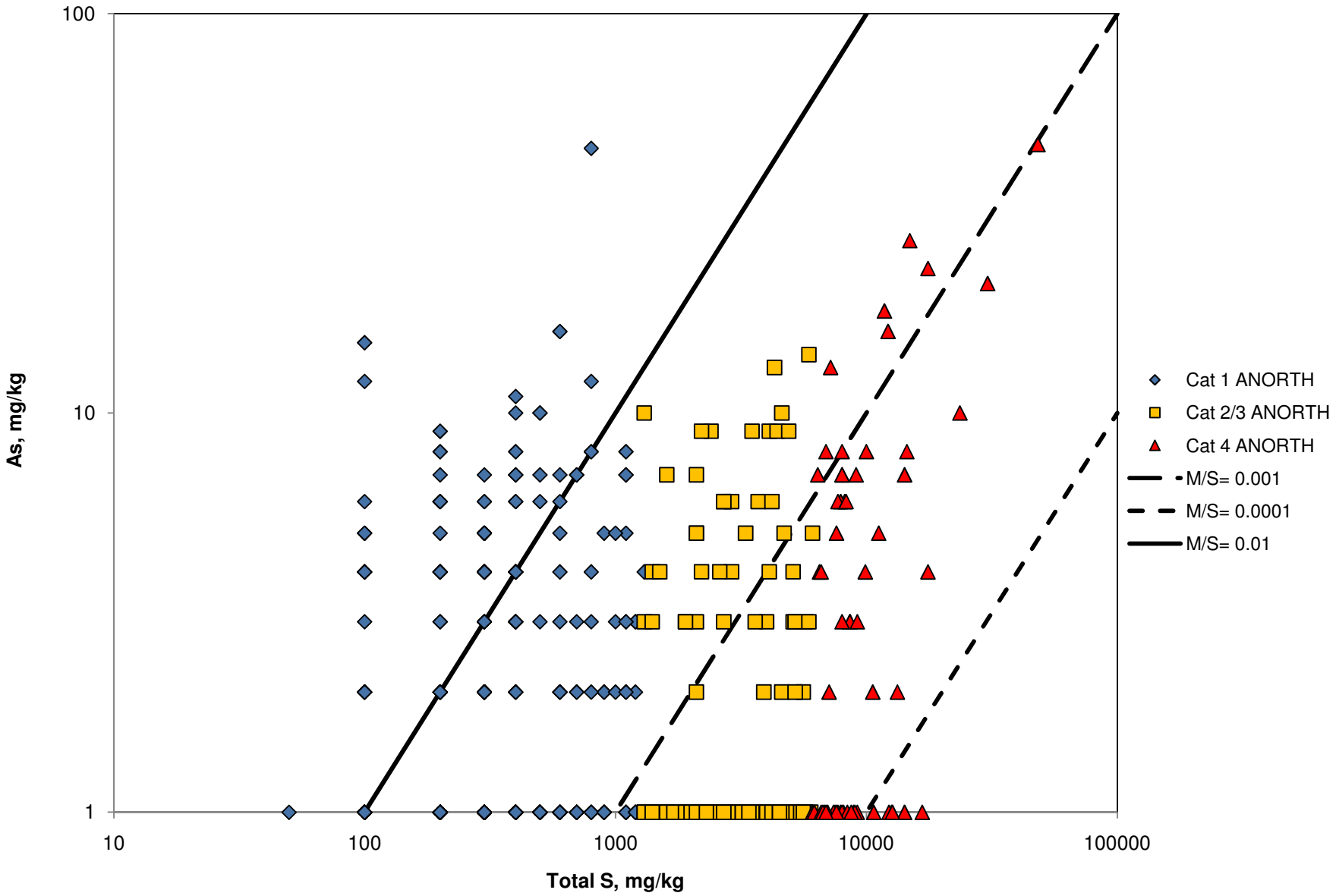

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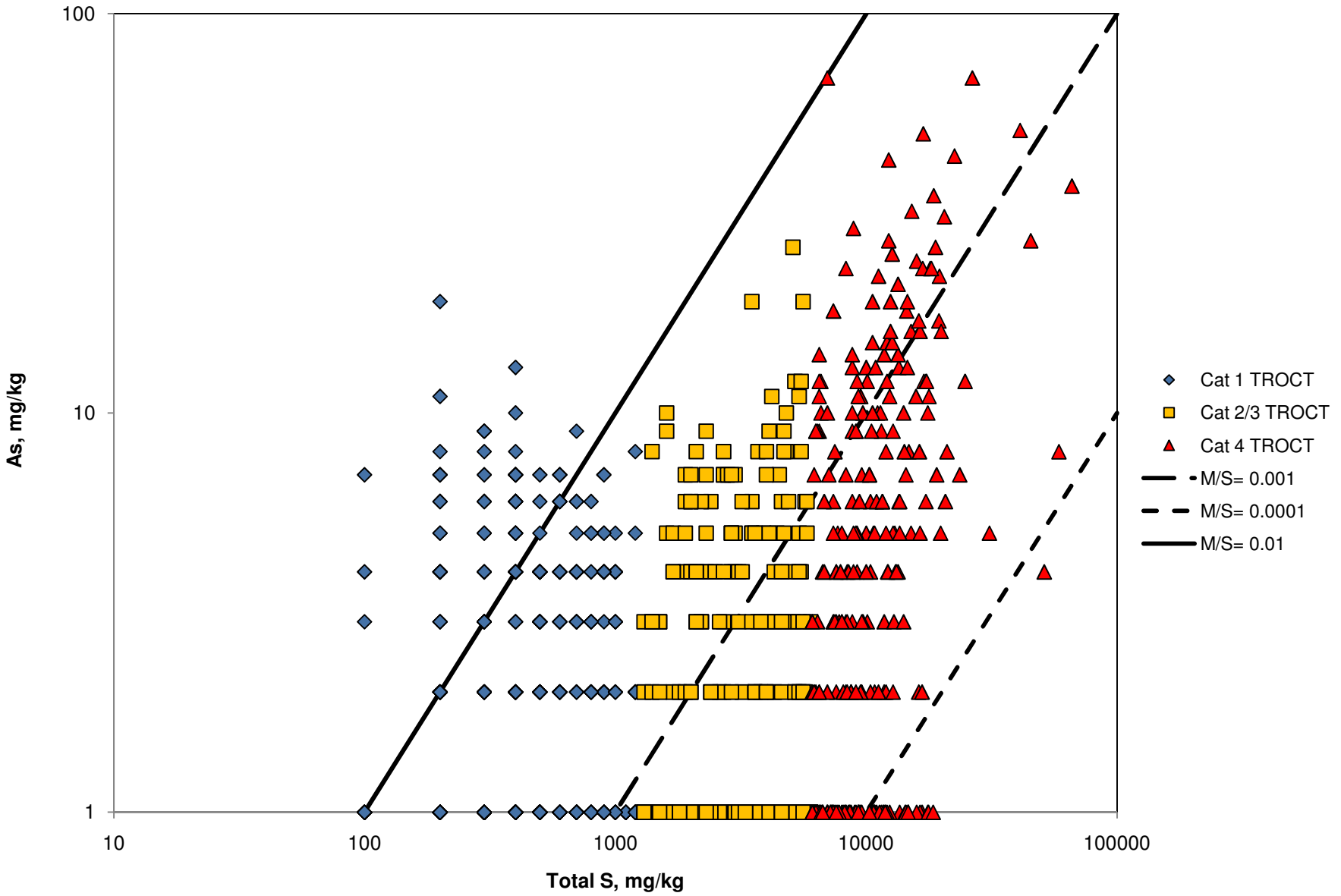

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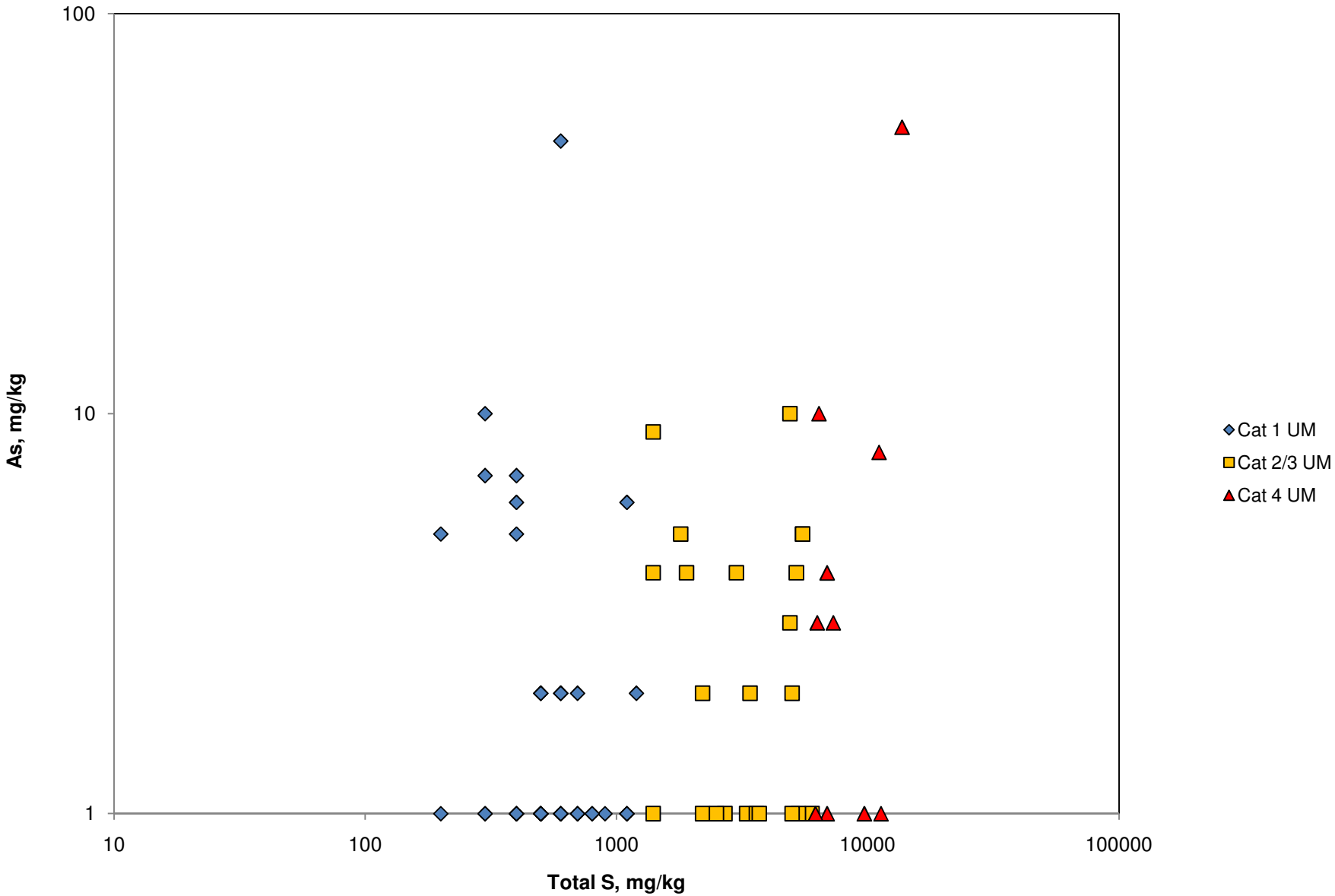

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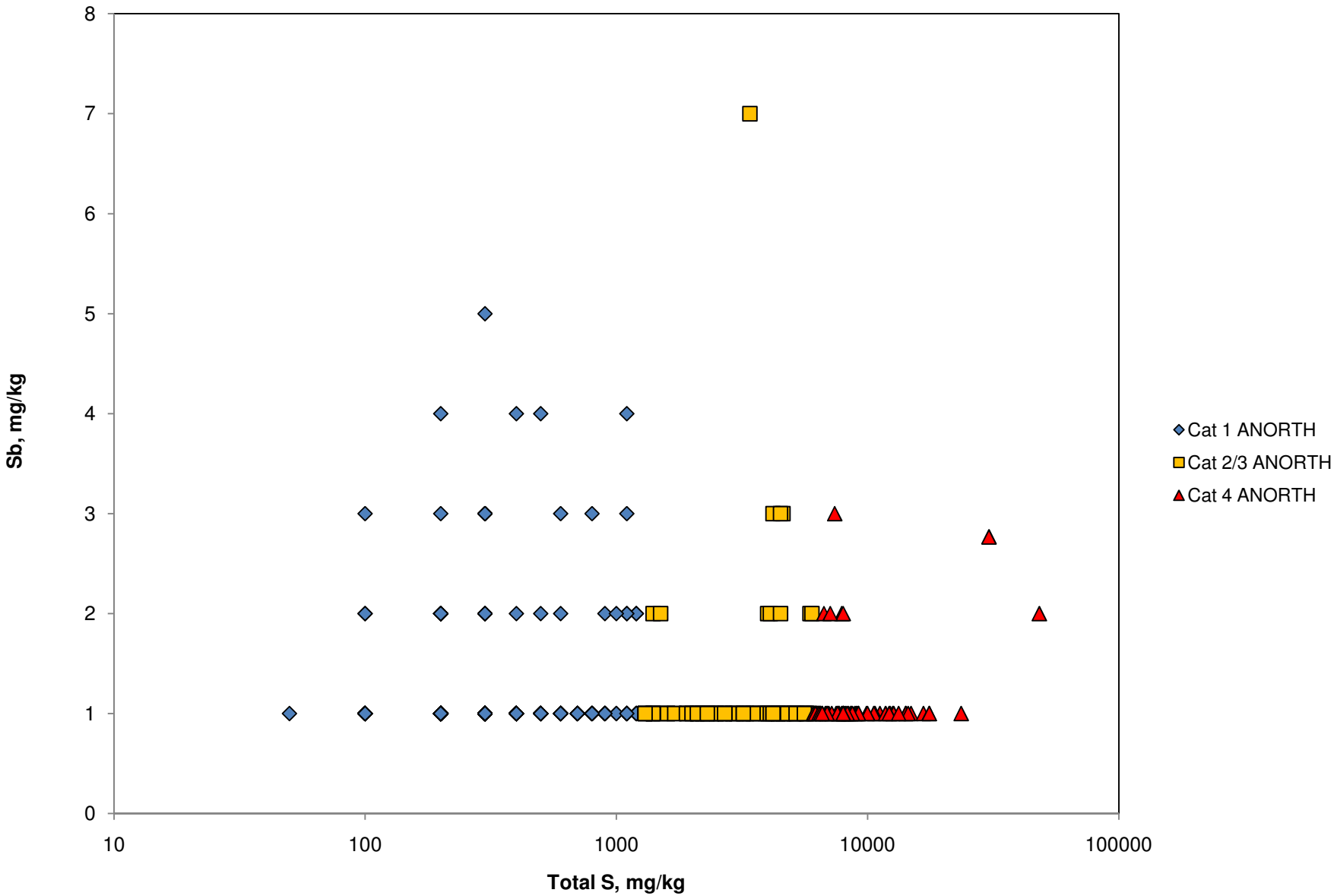

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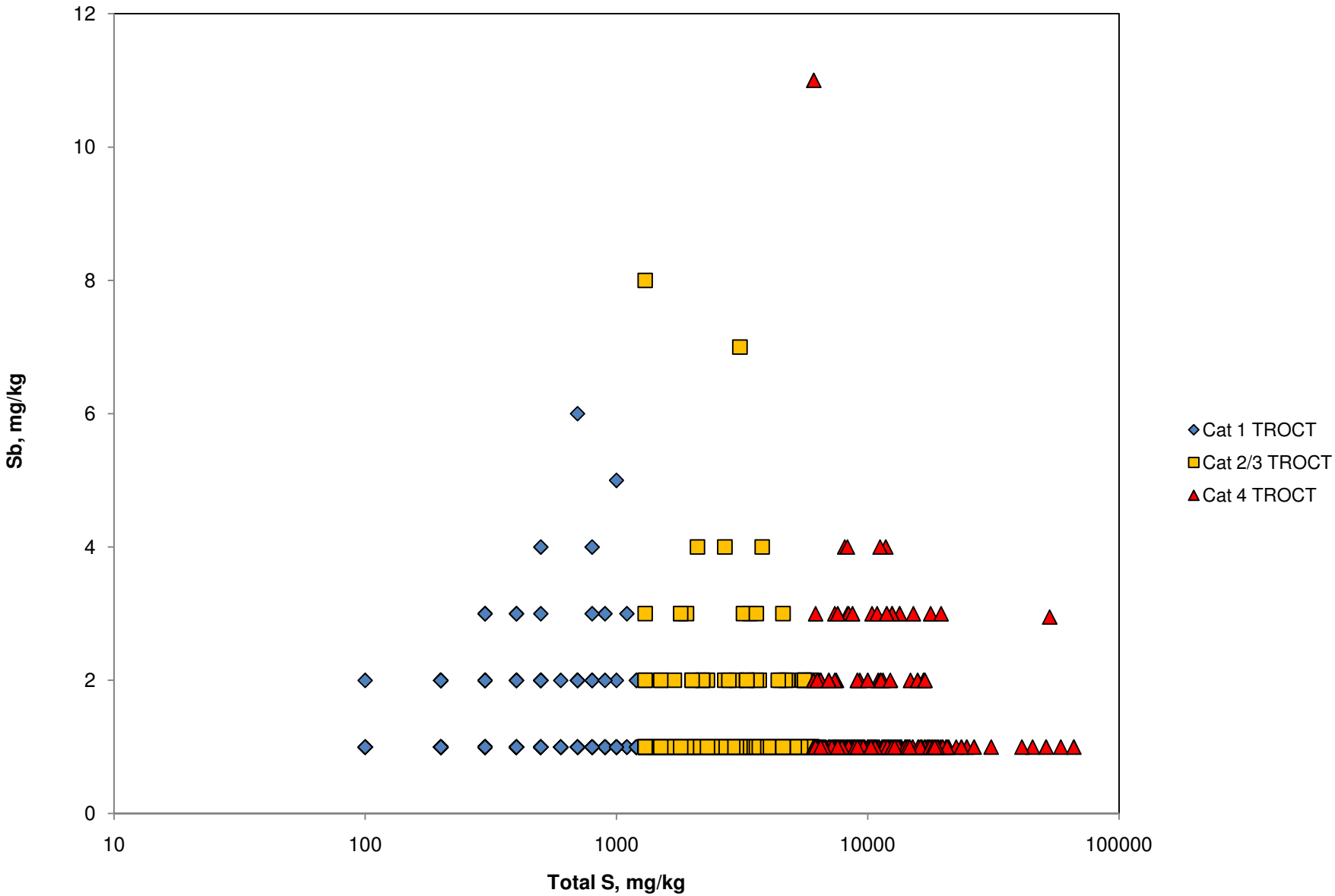

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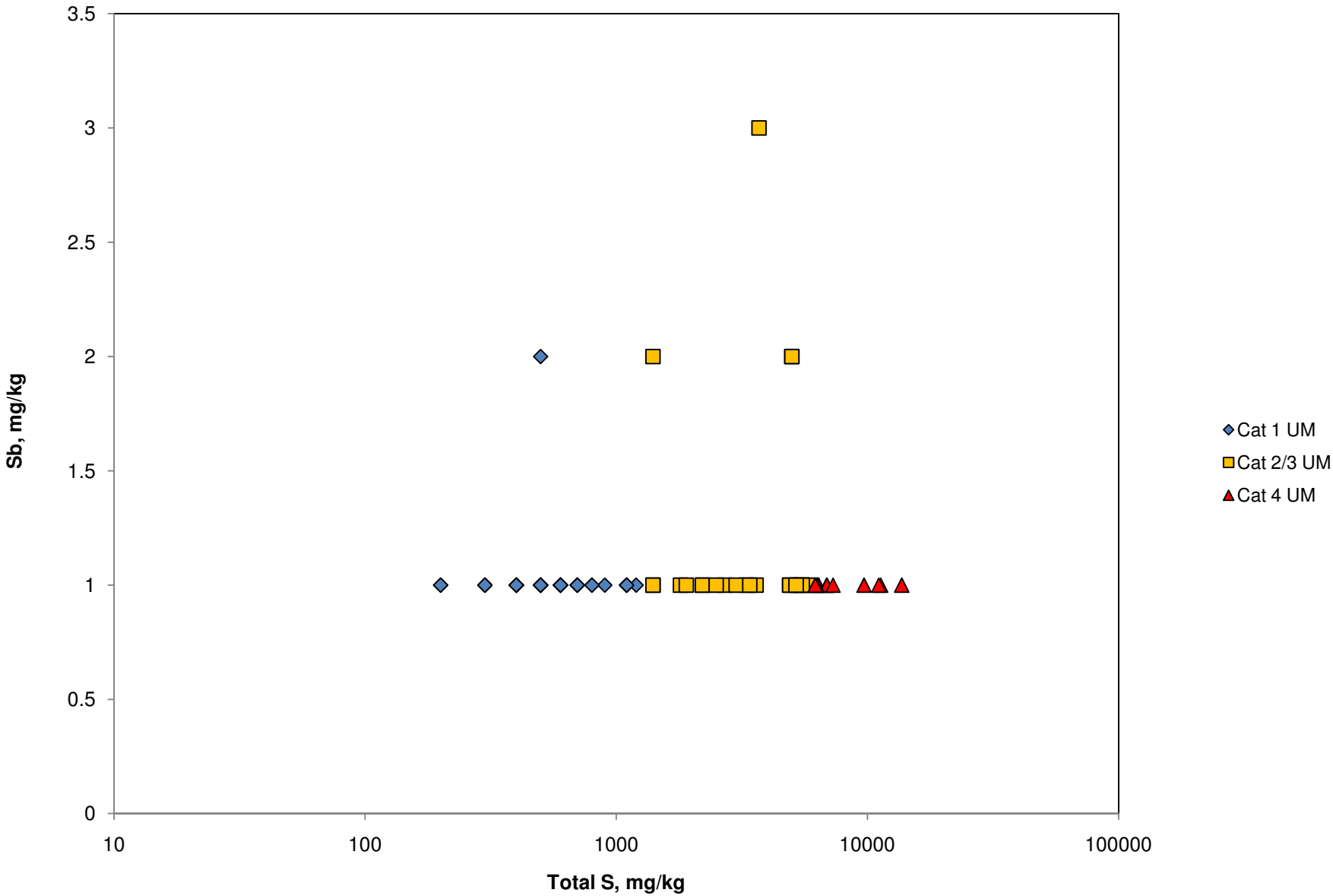

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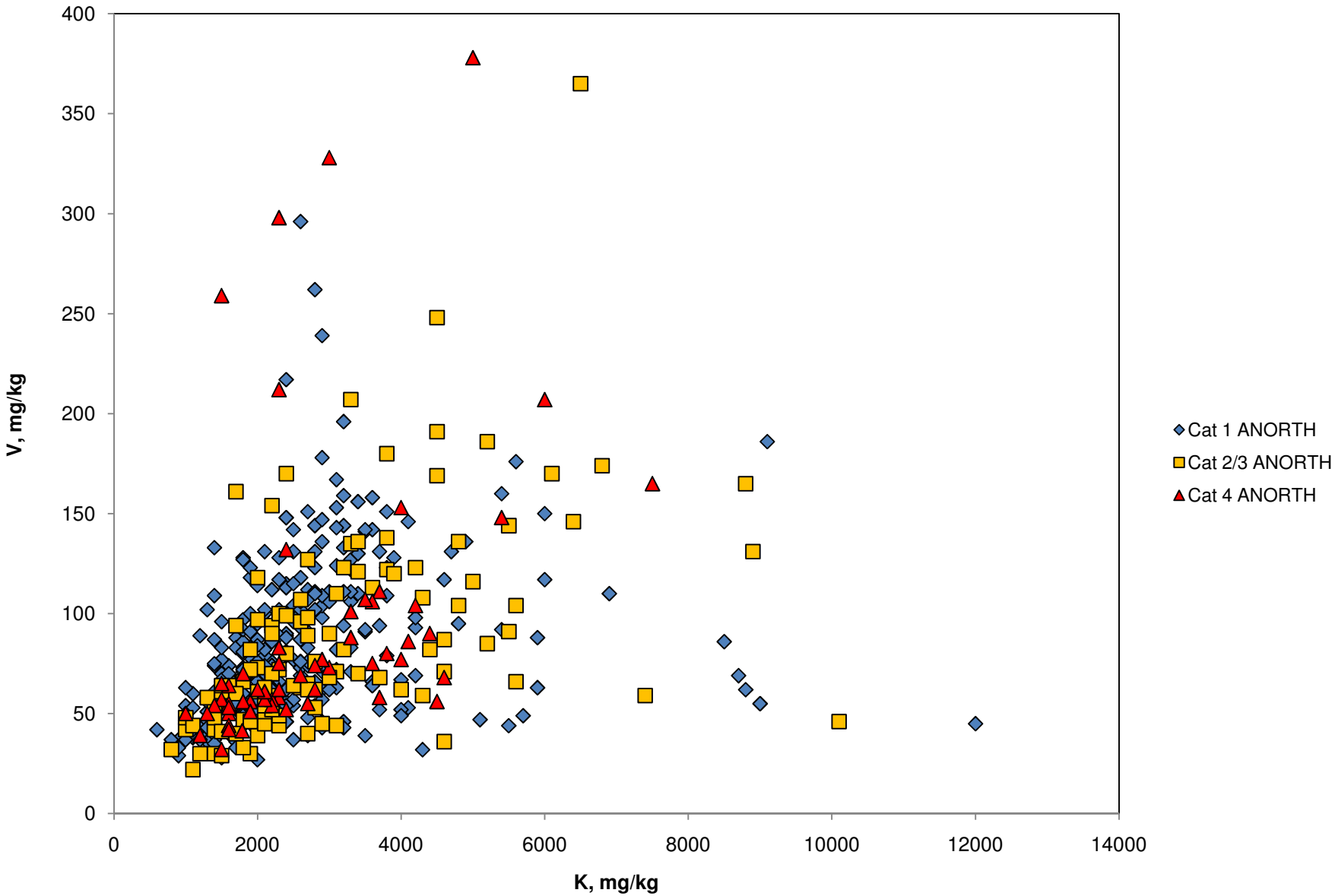

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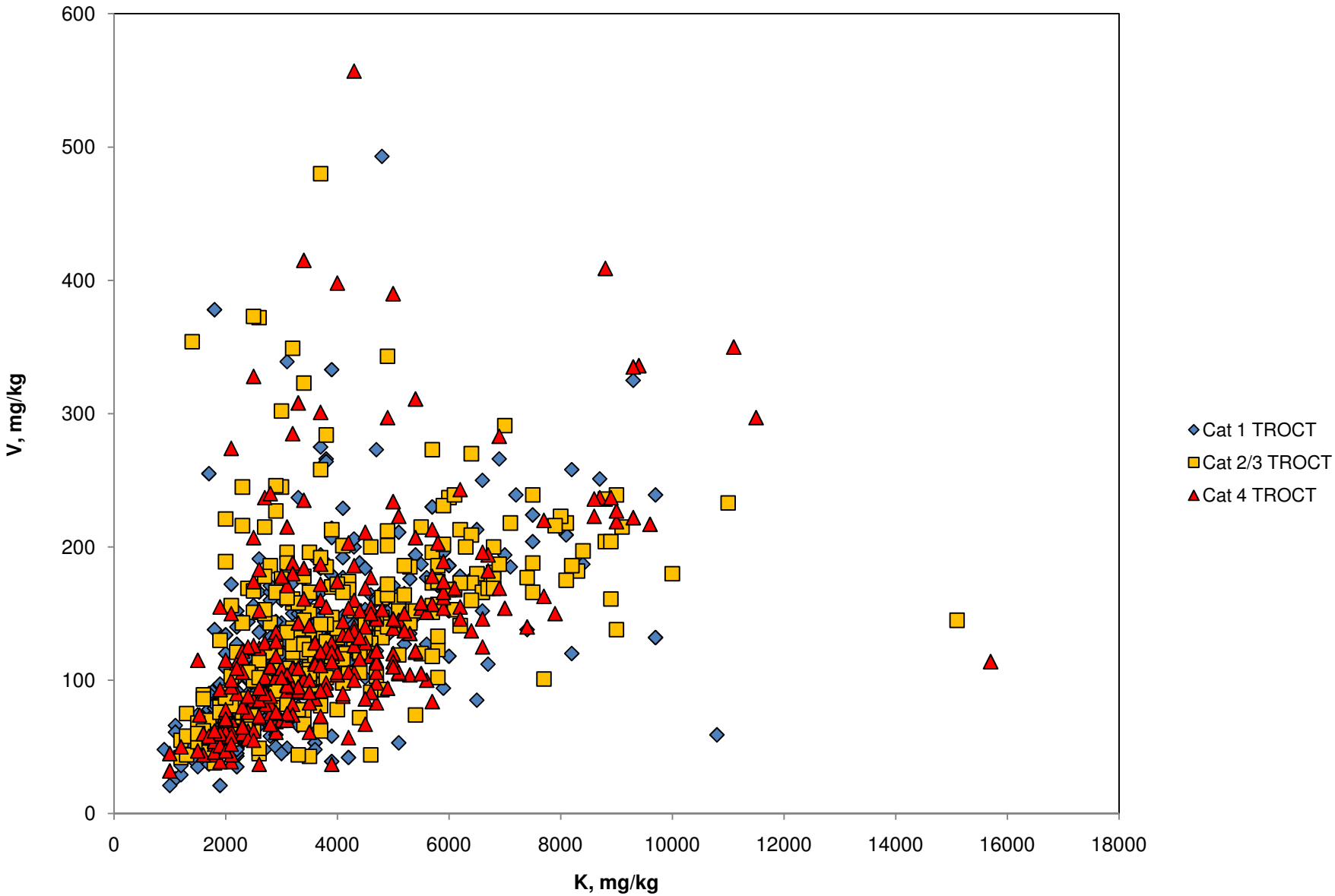

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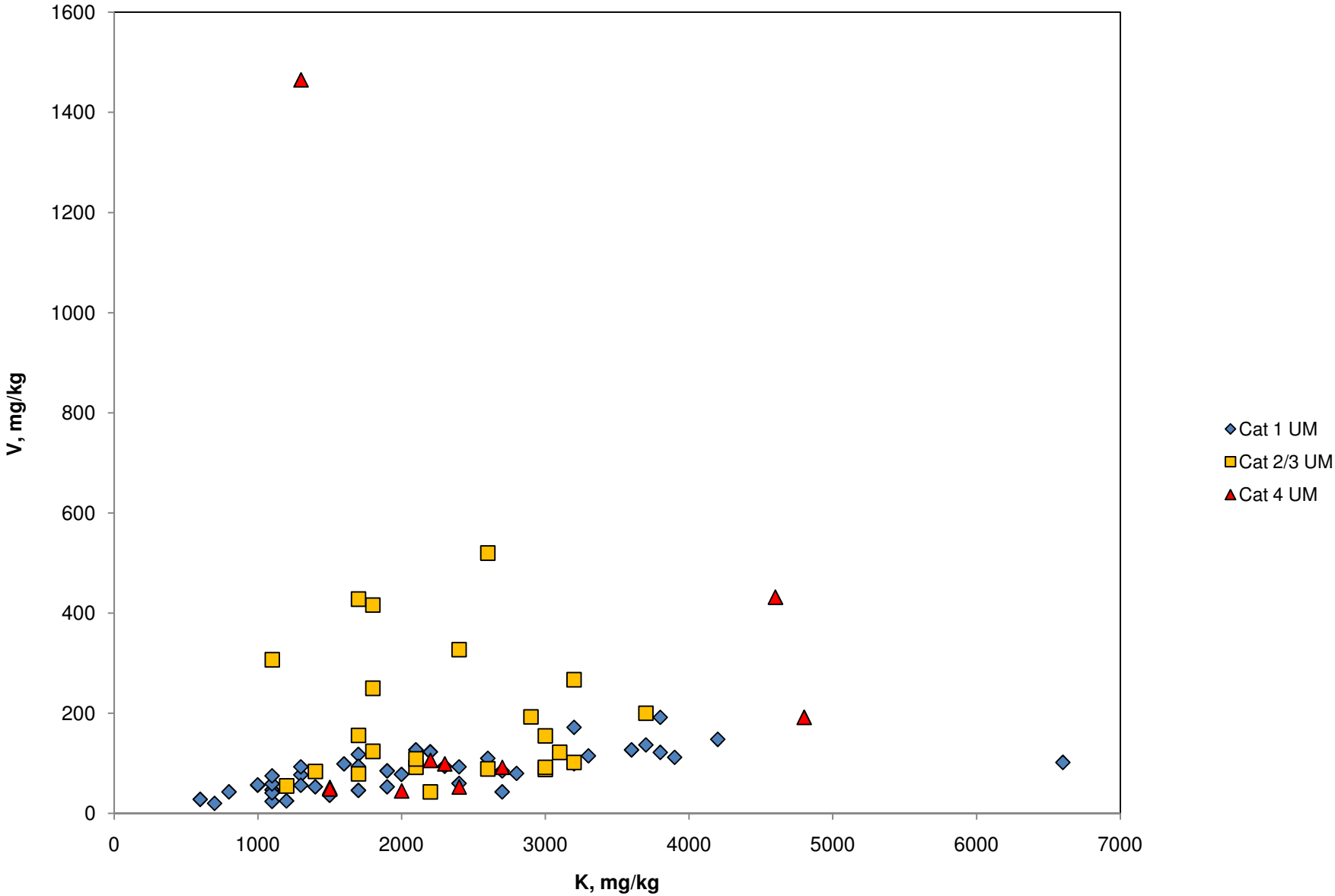

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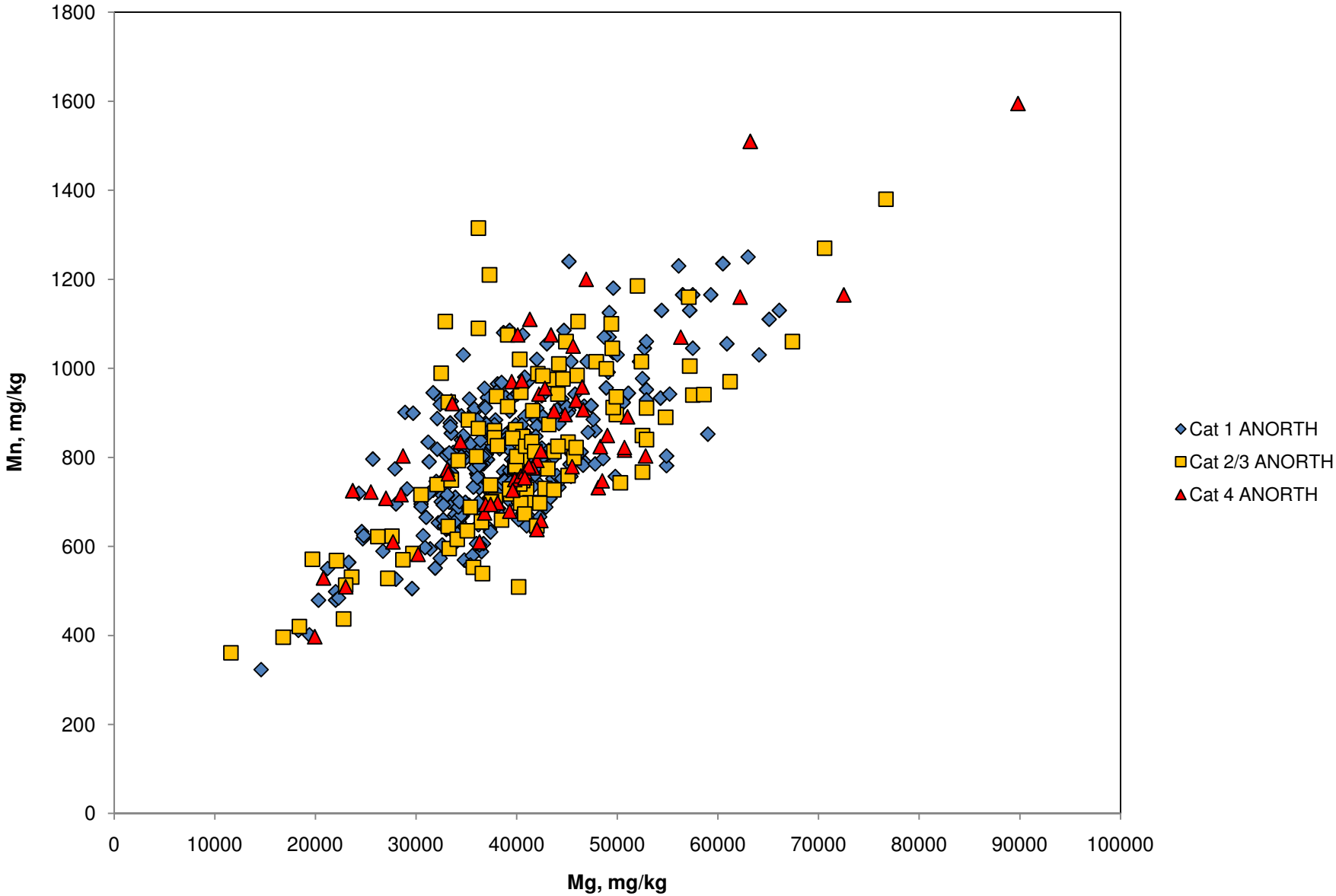

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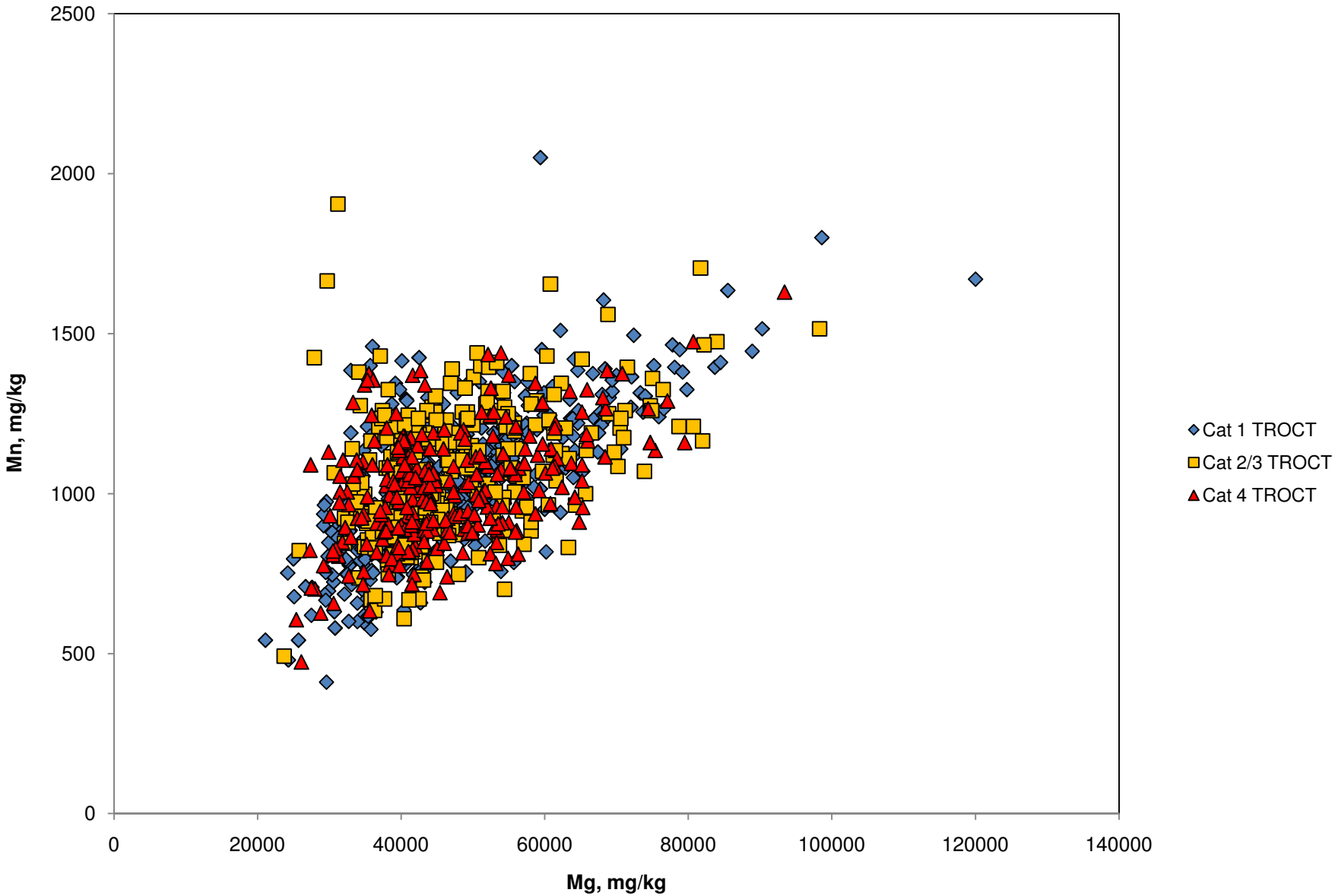

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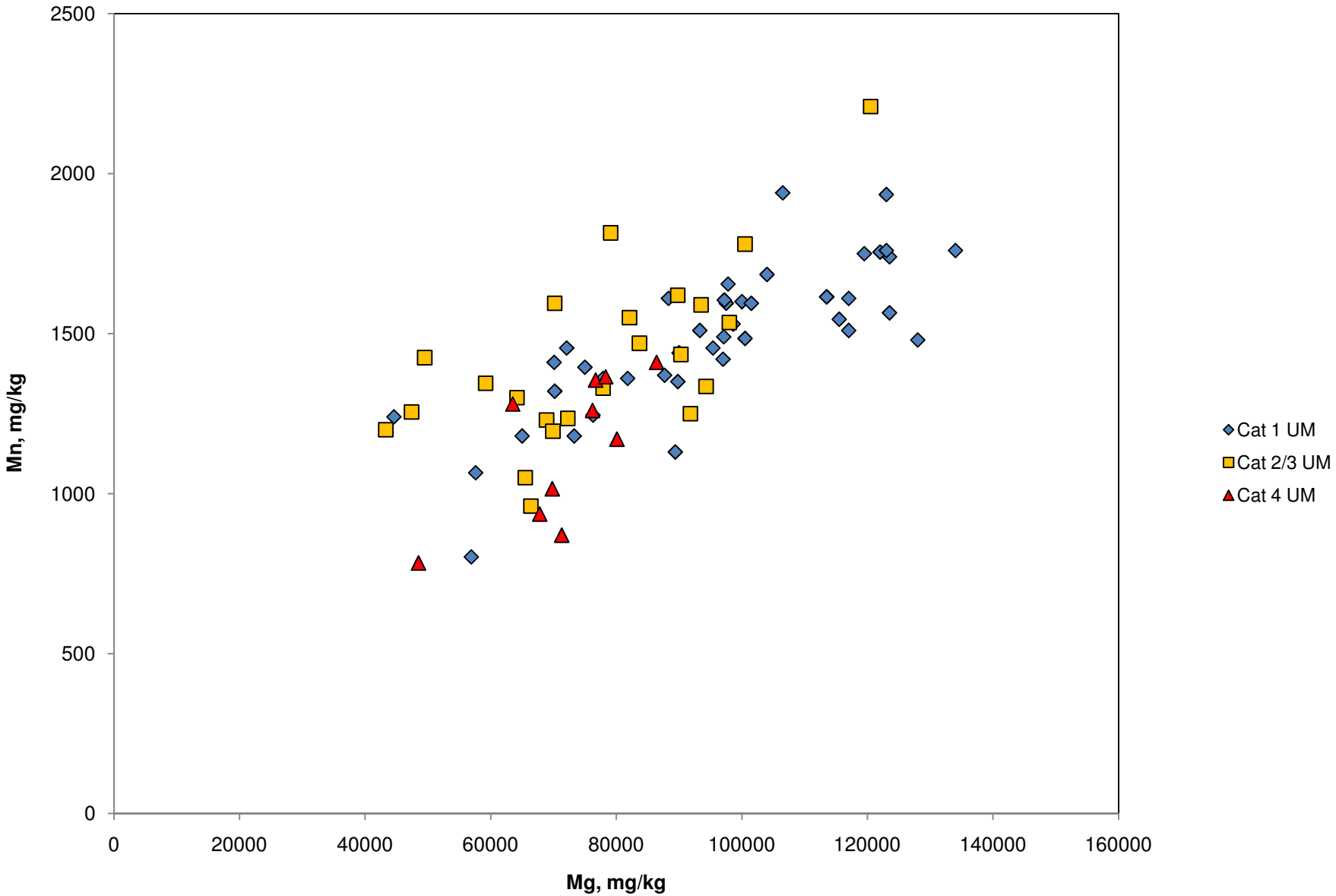

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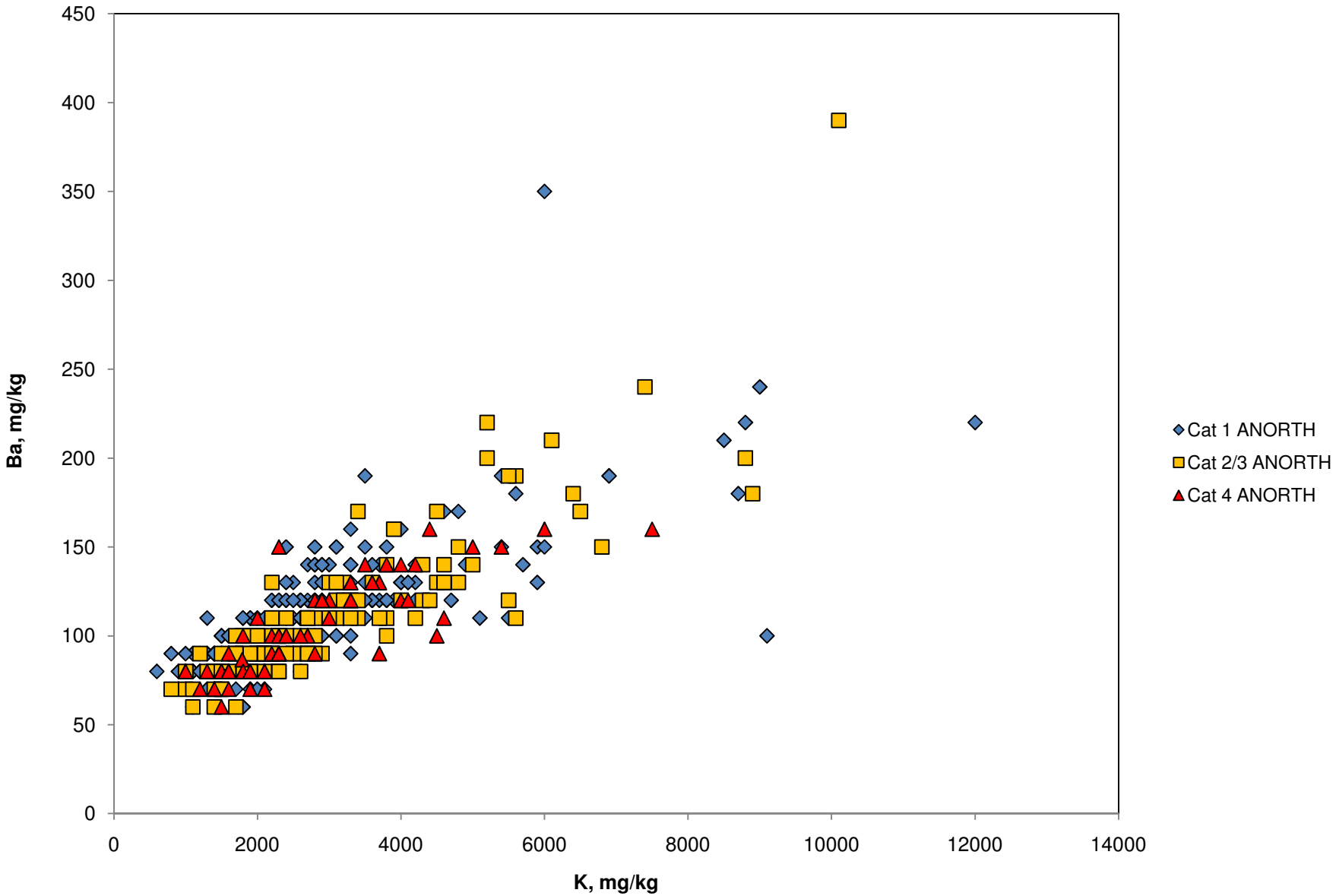

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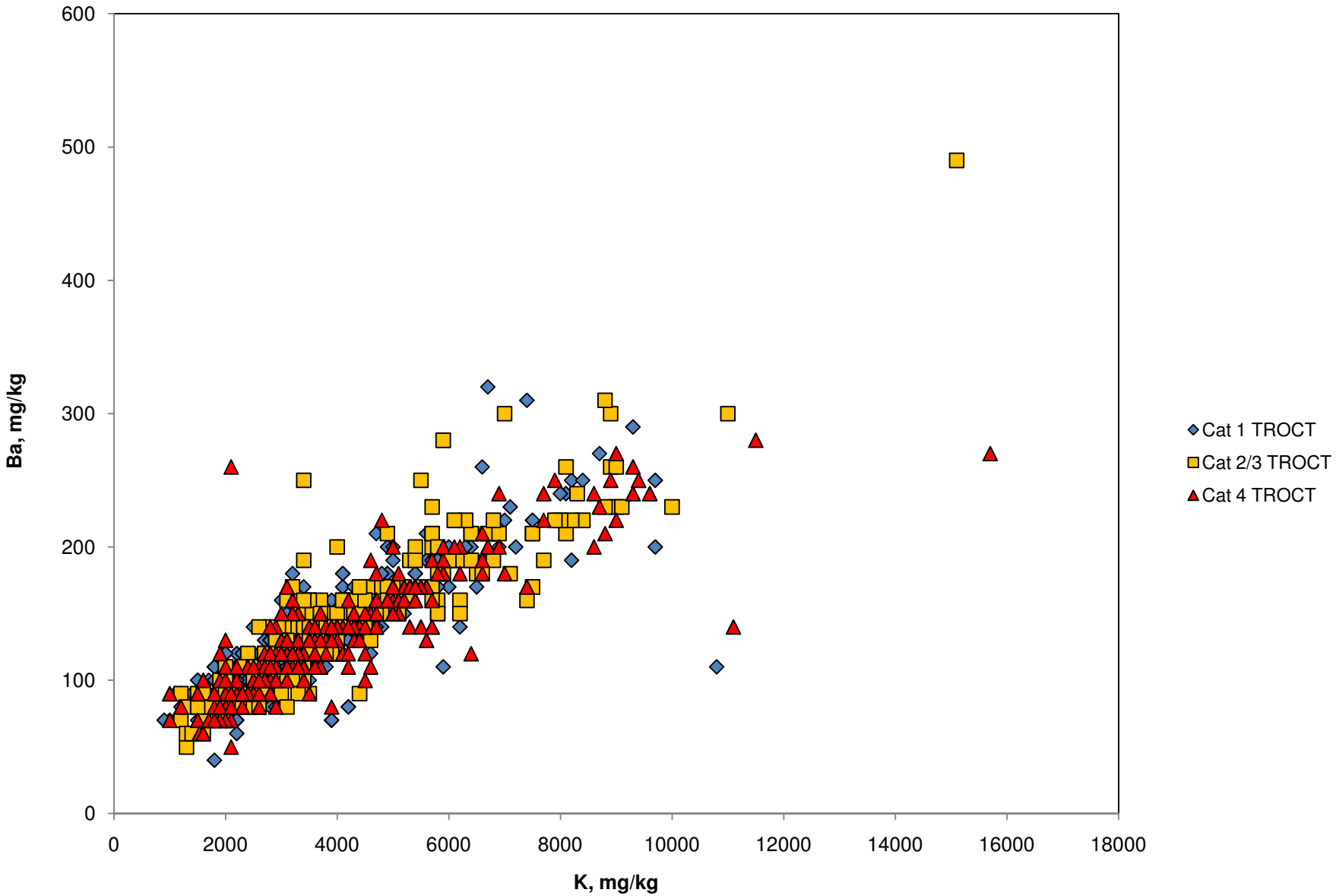

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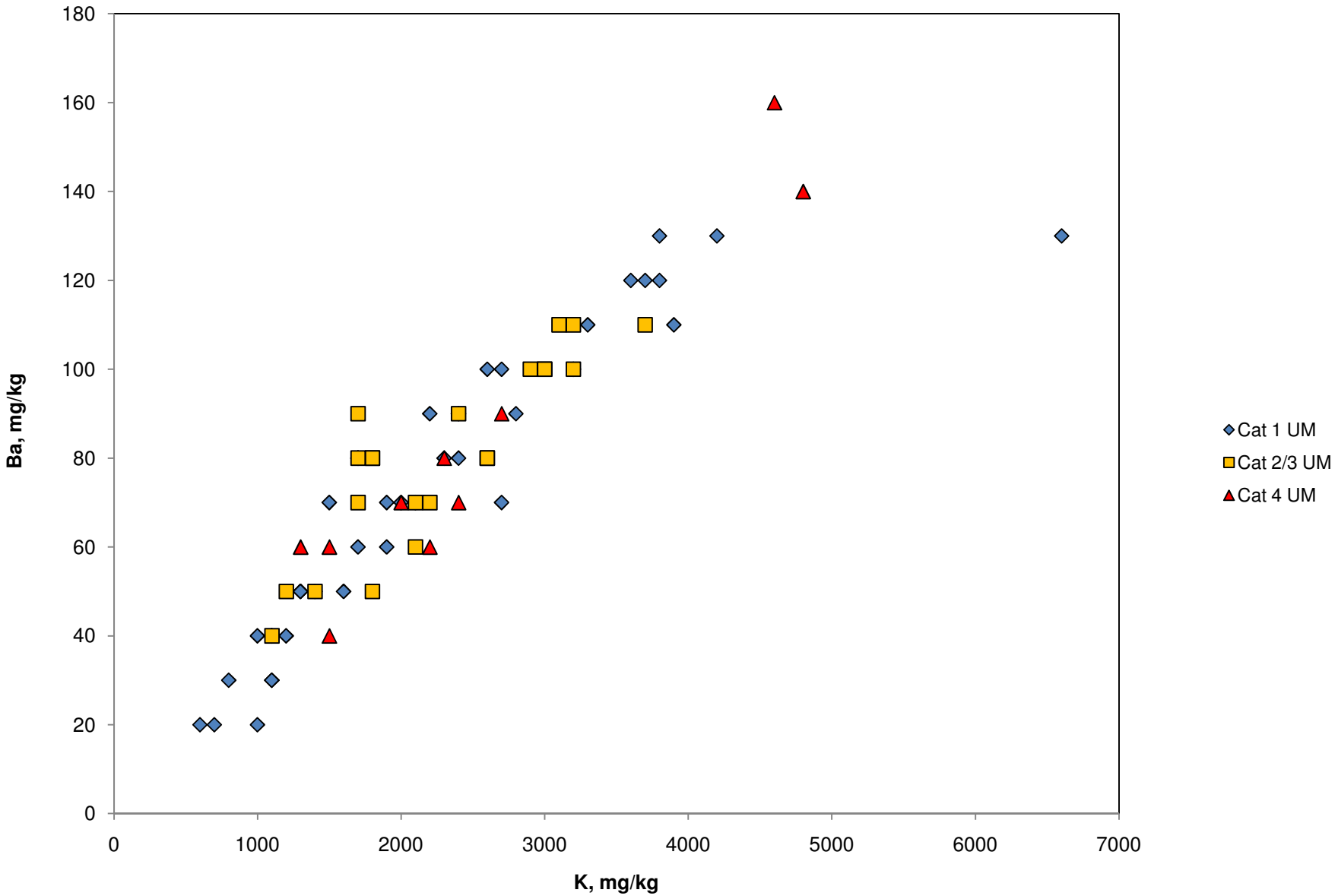

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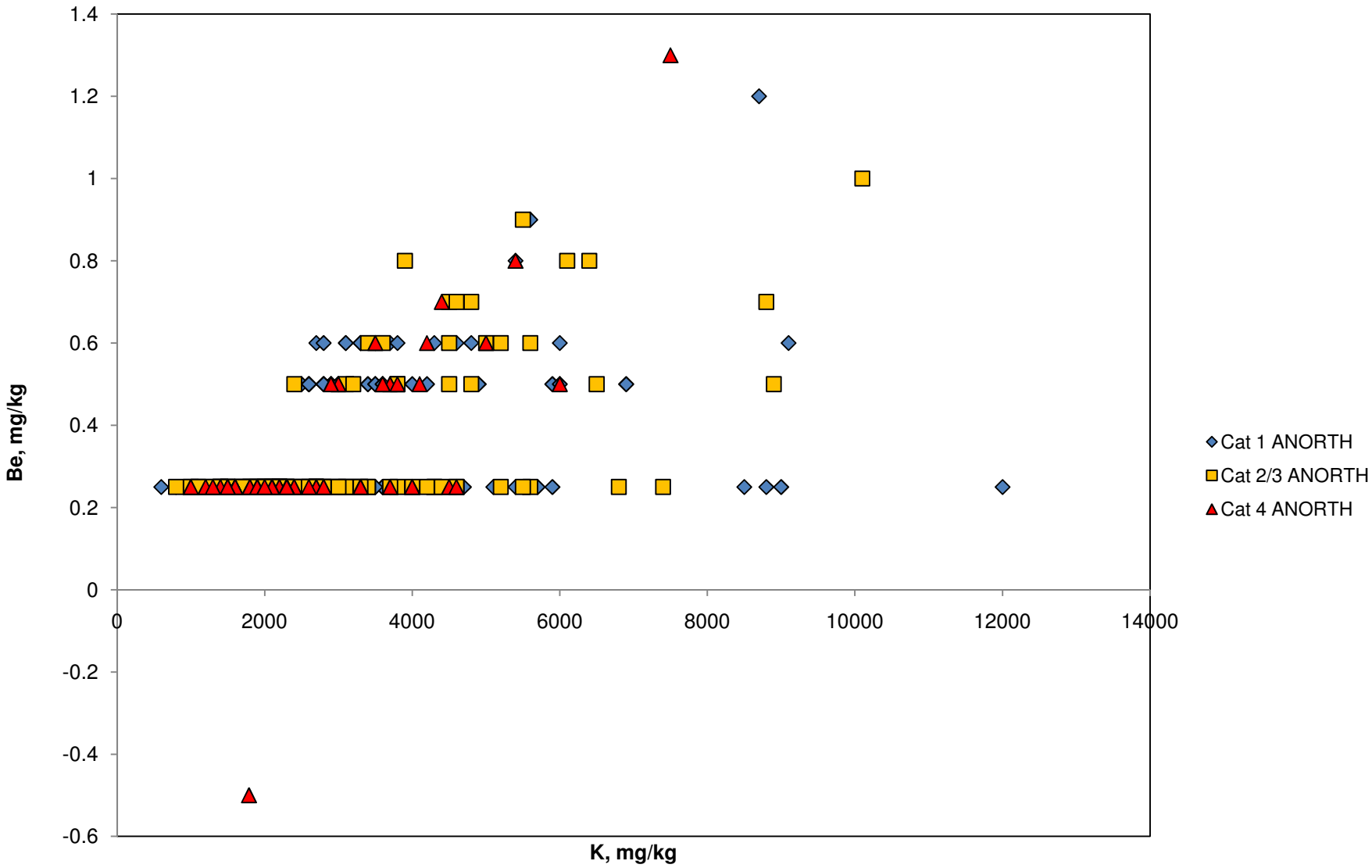

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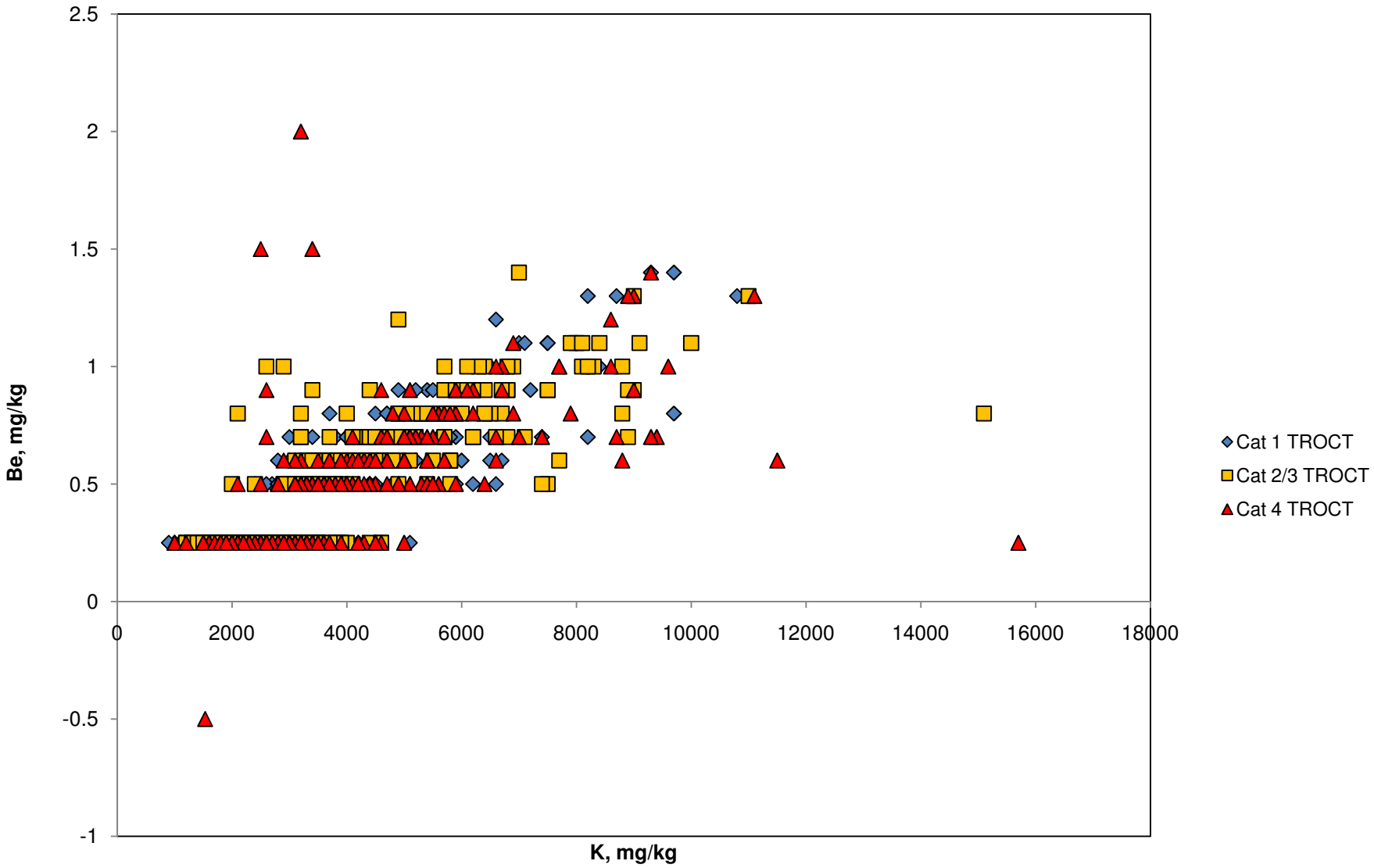

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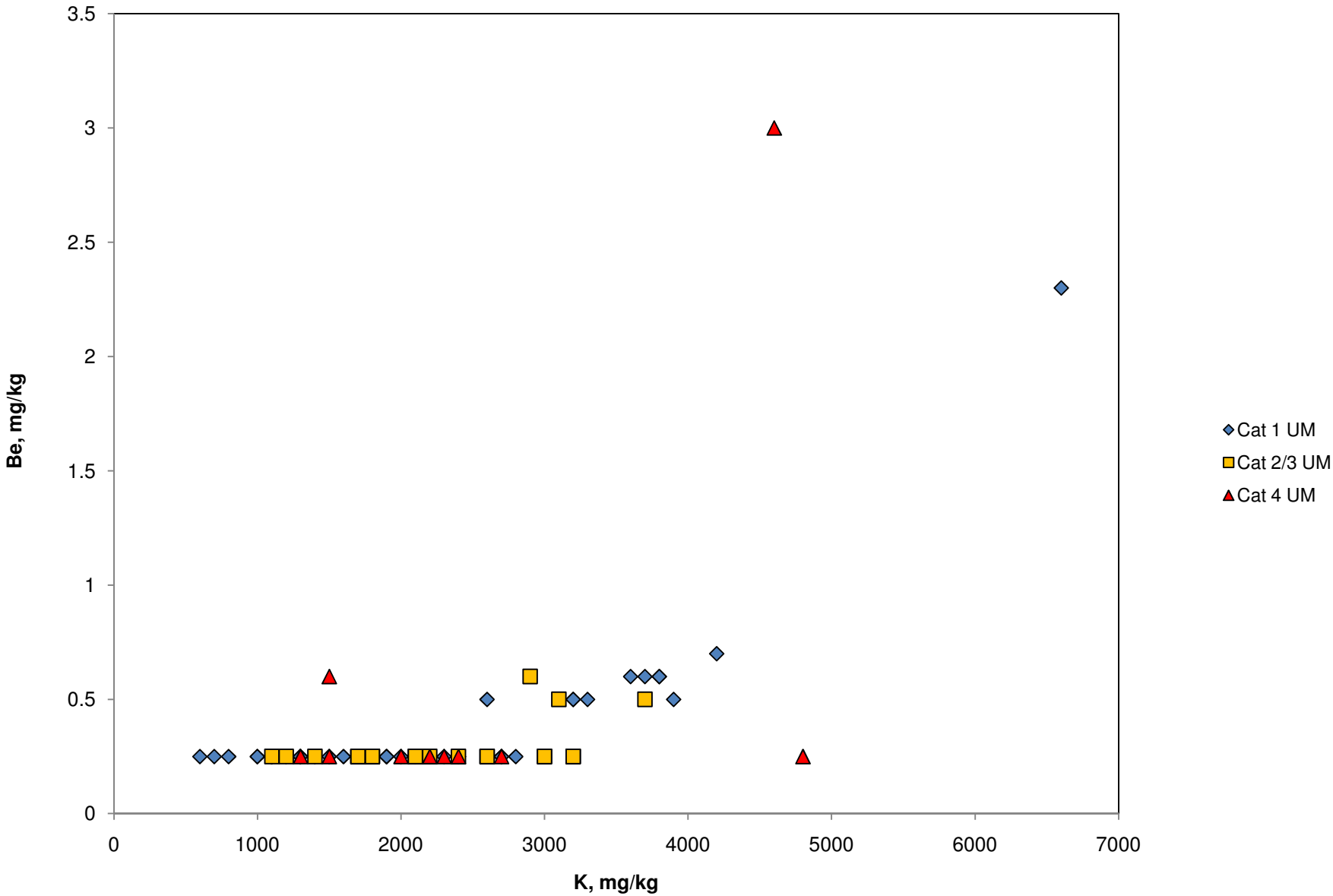

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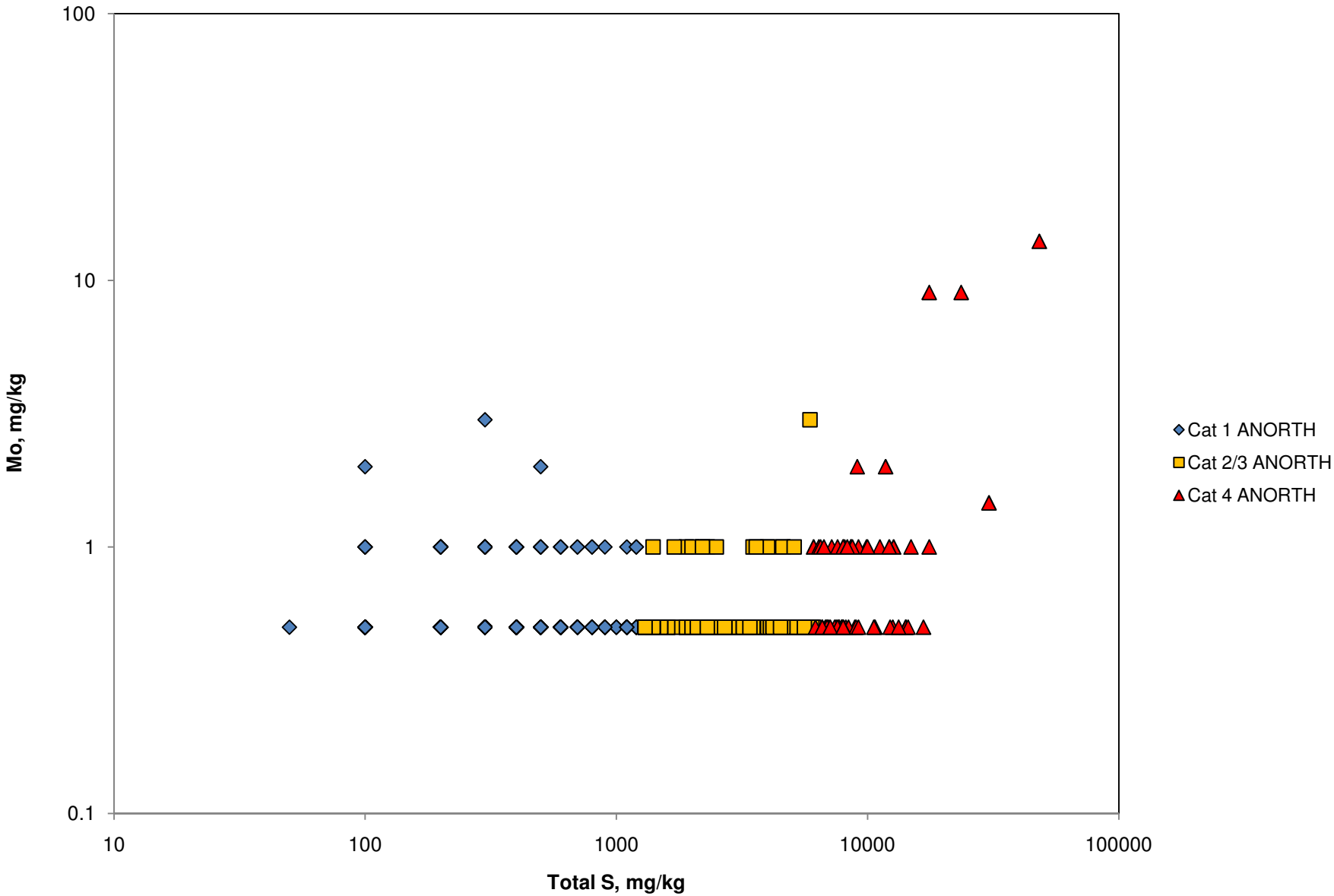

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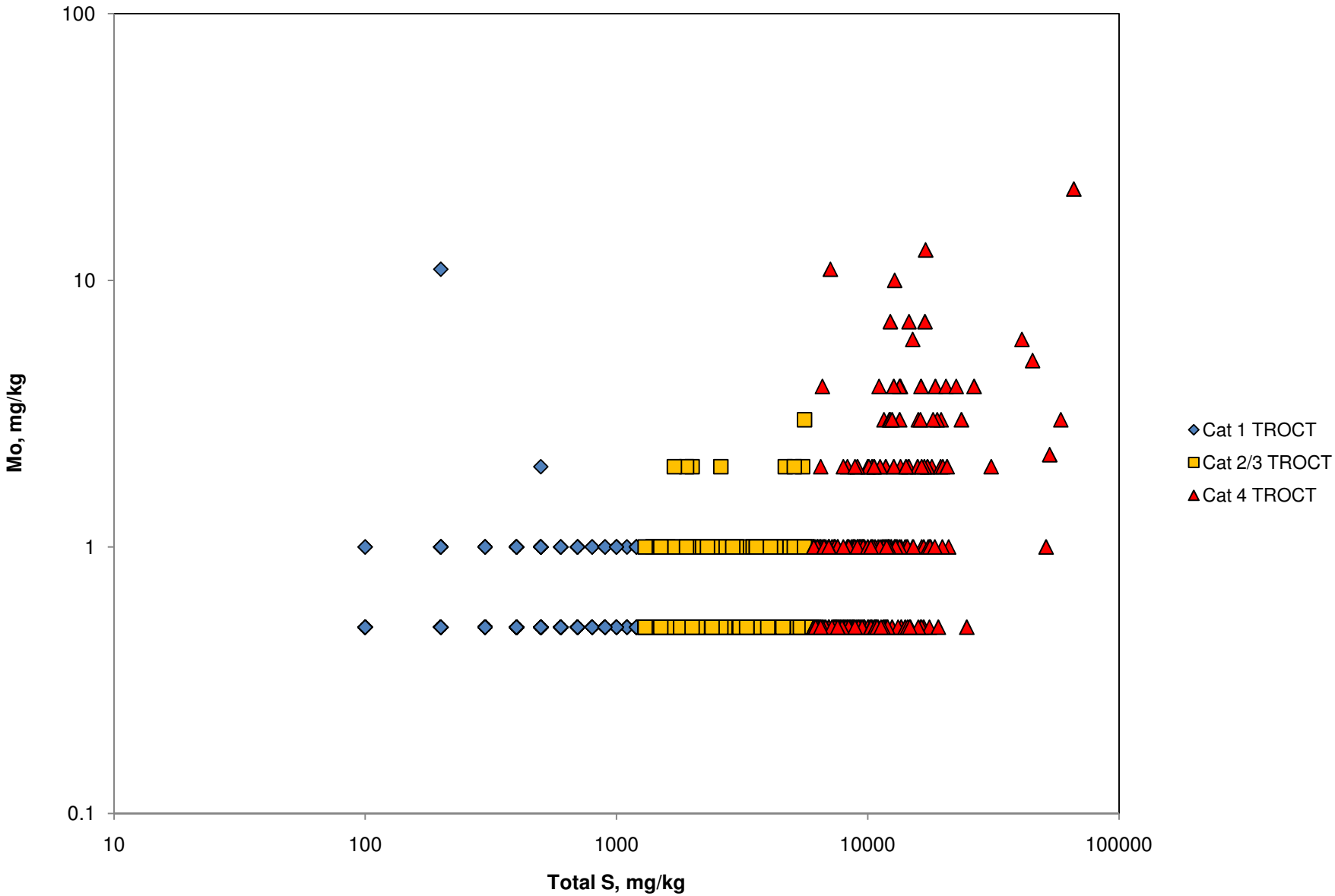

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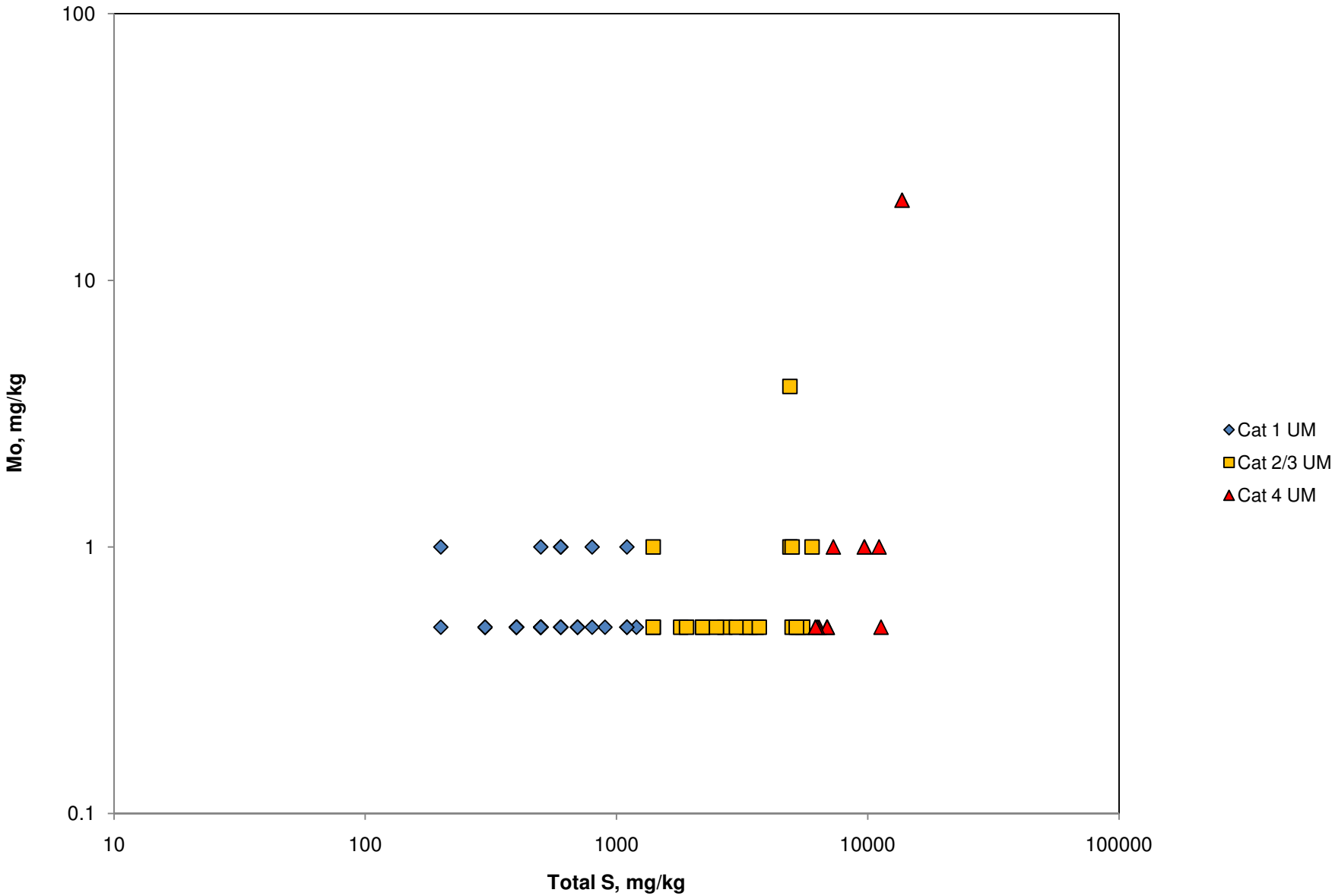

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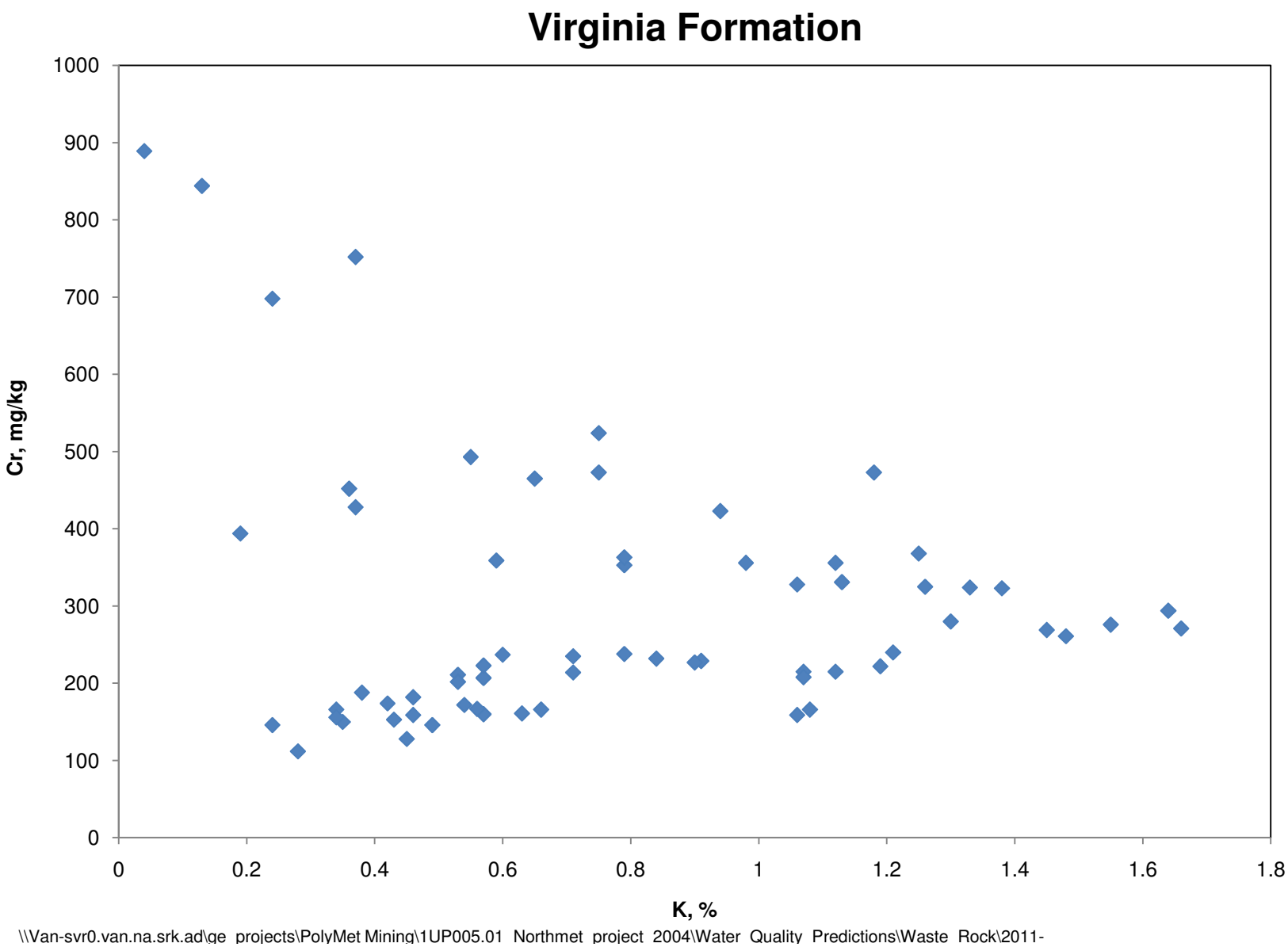

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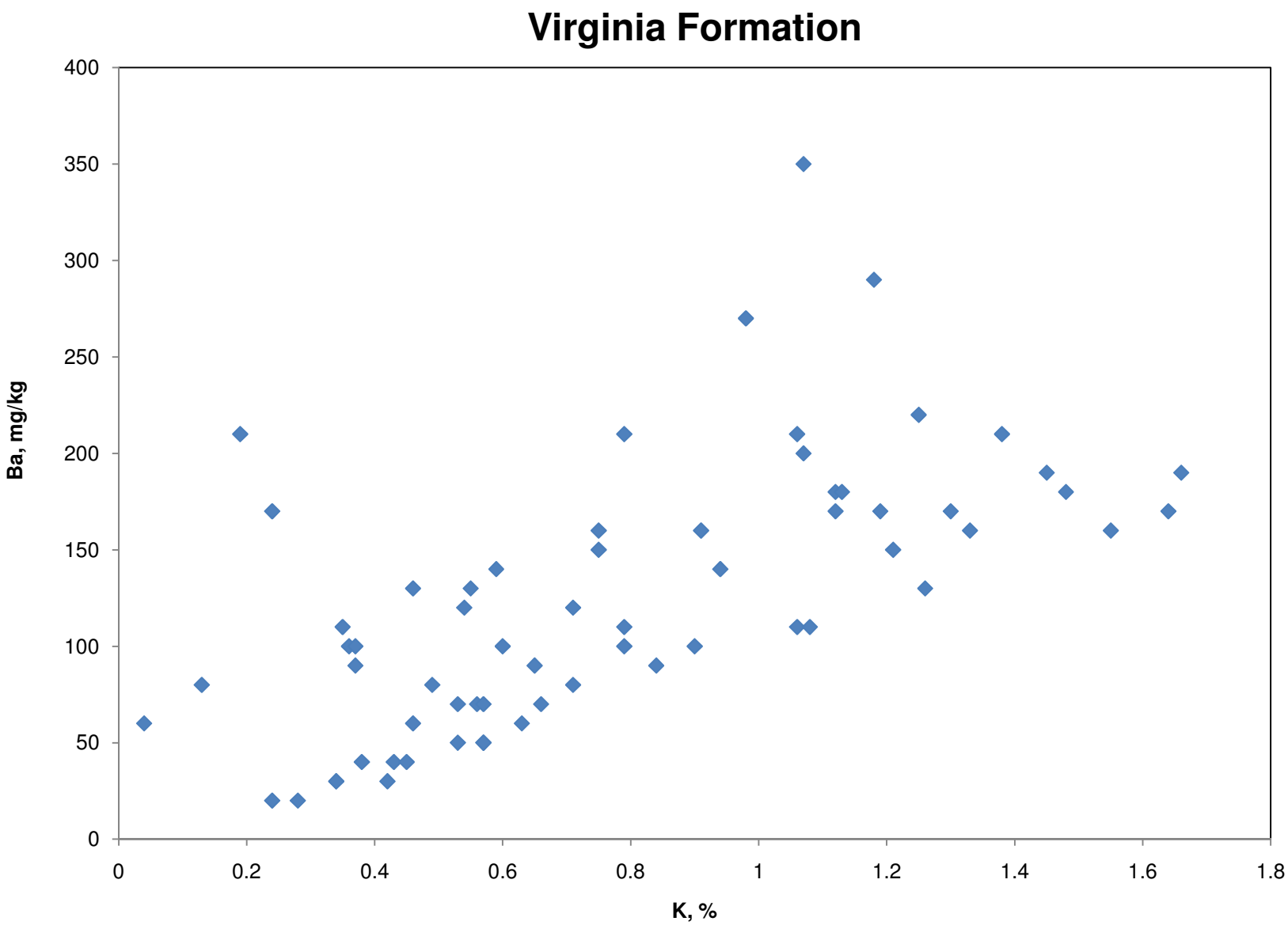

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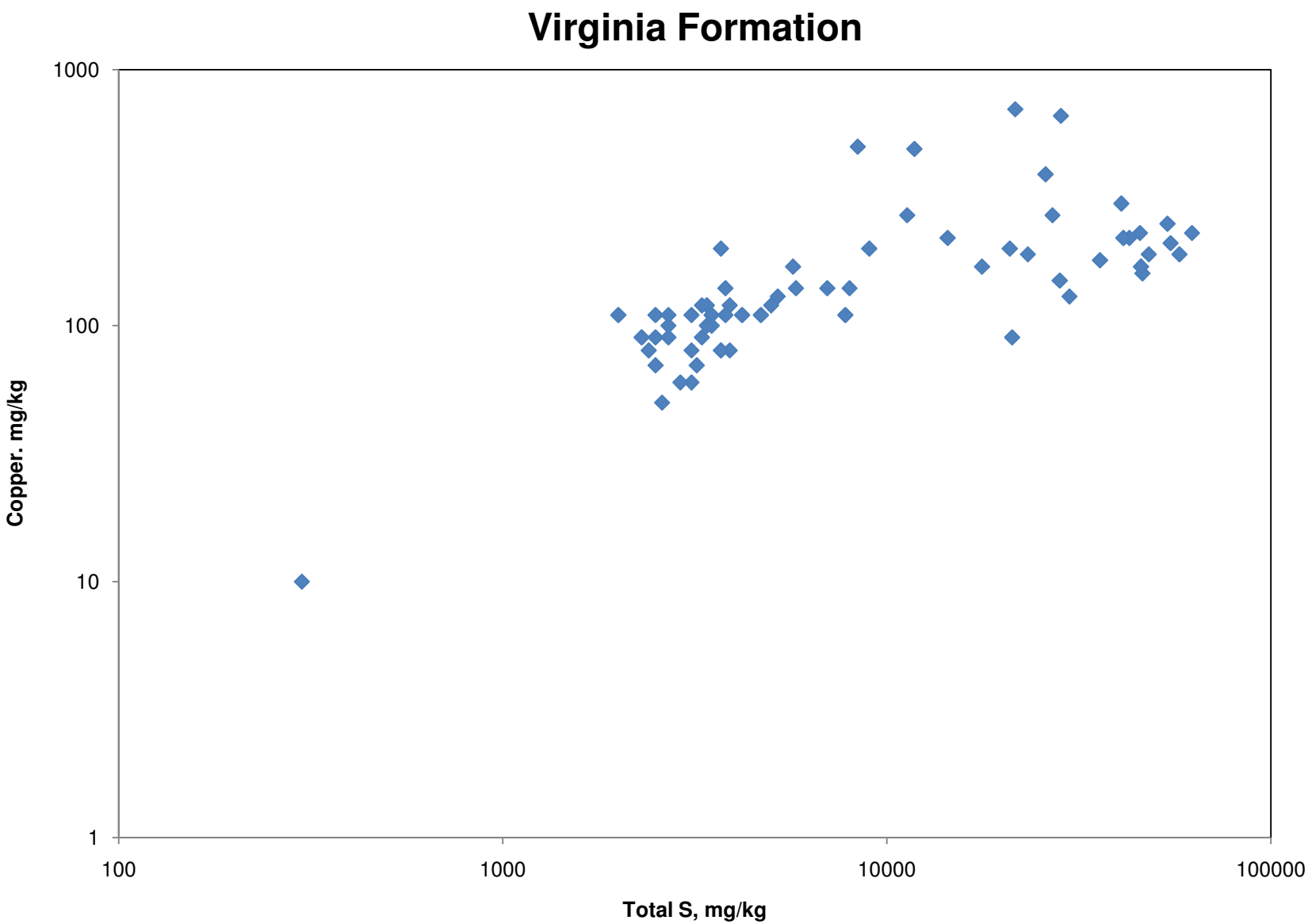

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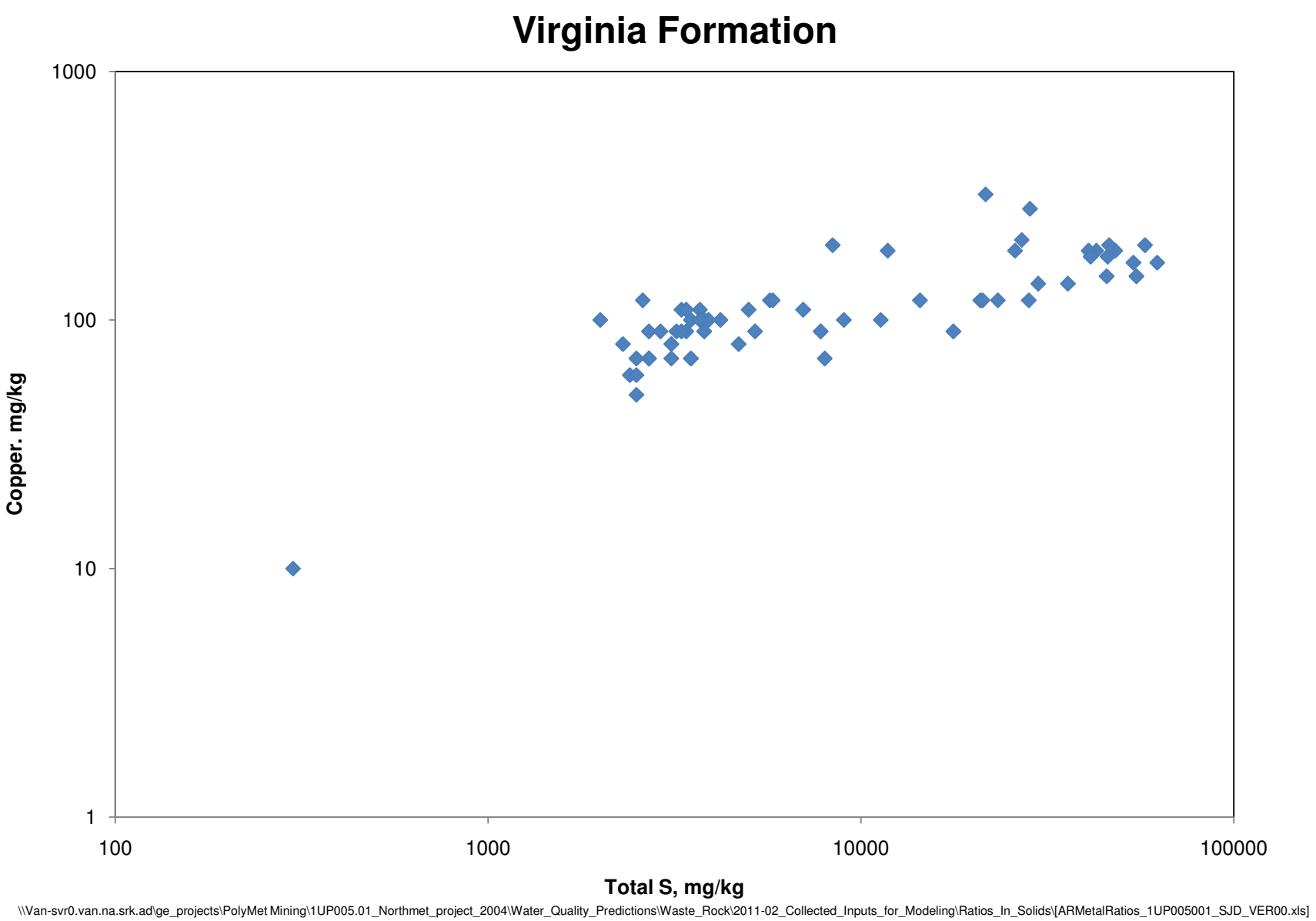
Virginia Formation
1000
100
10
1
Copper. mg/kg
100
1000
10000
100000
Total S, mg/kg
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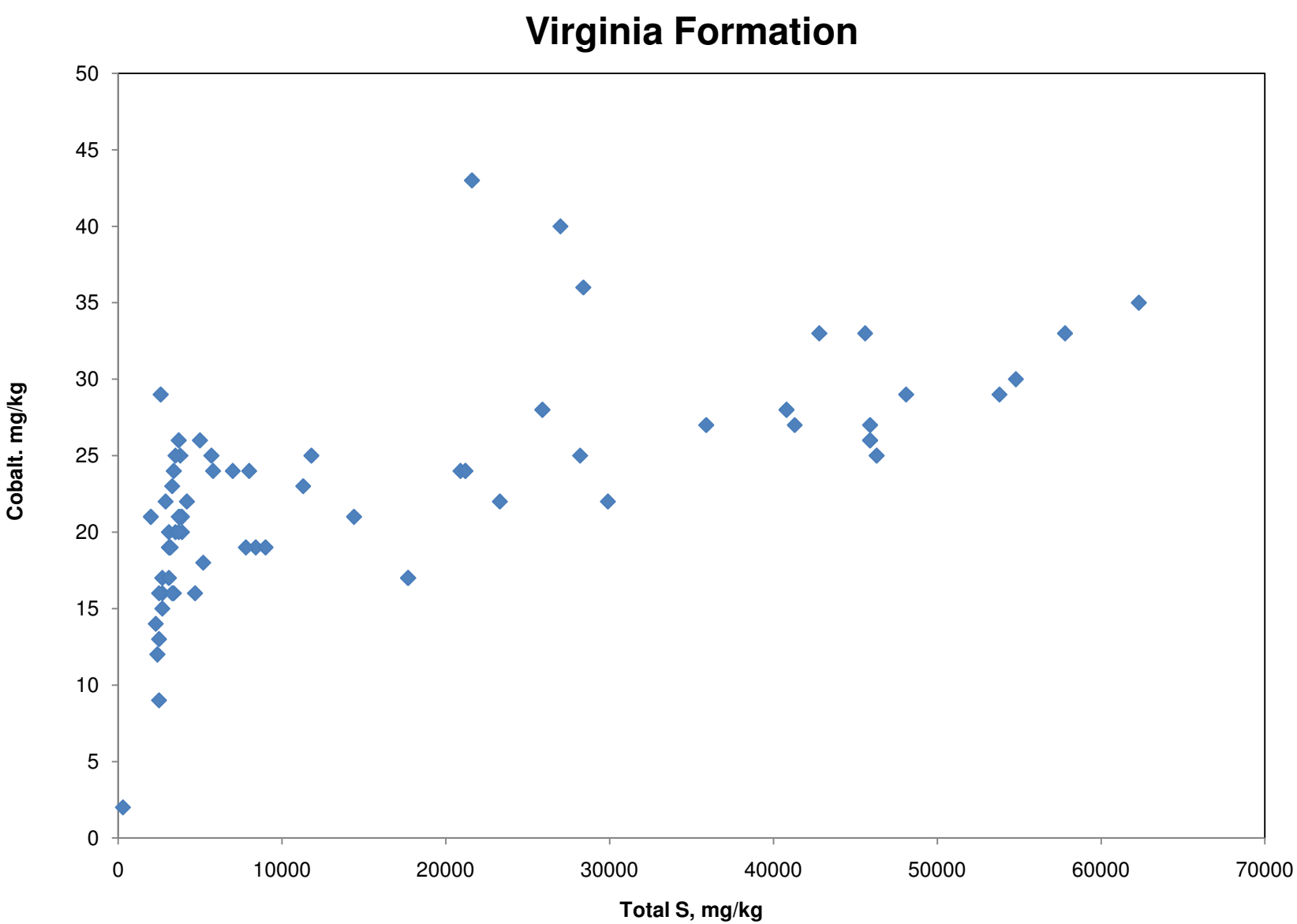

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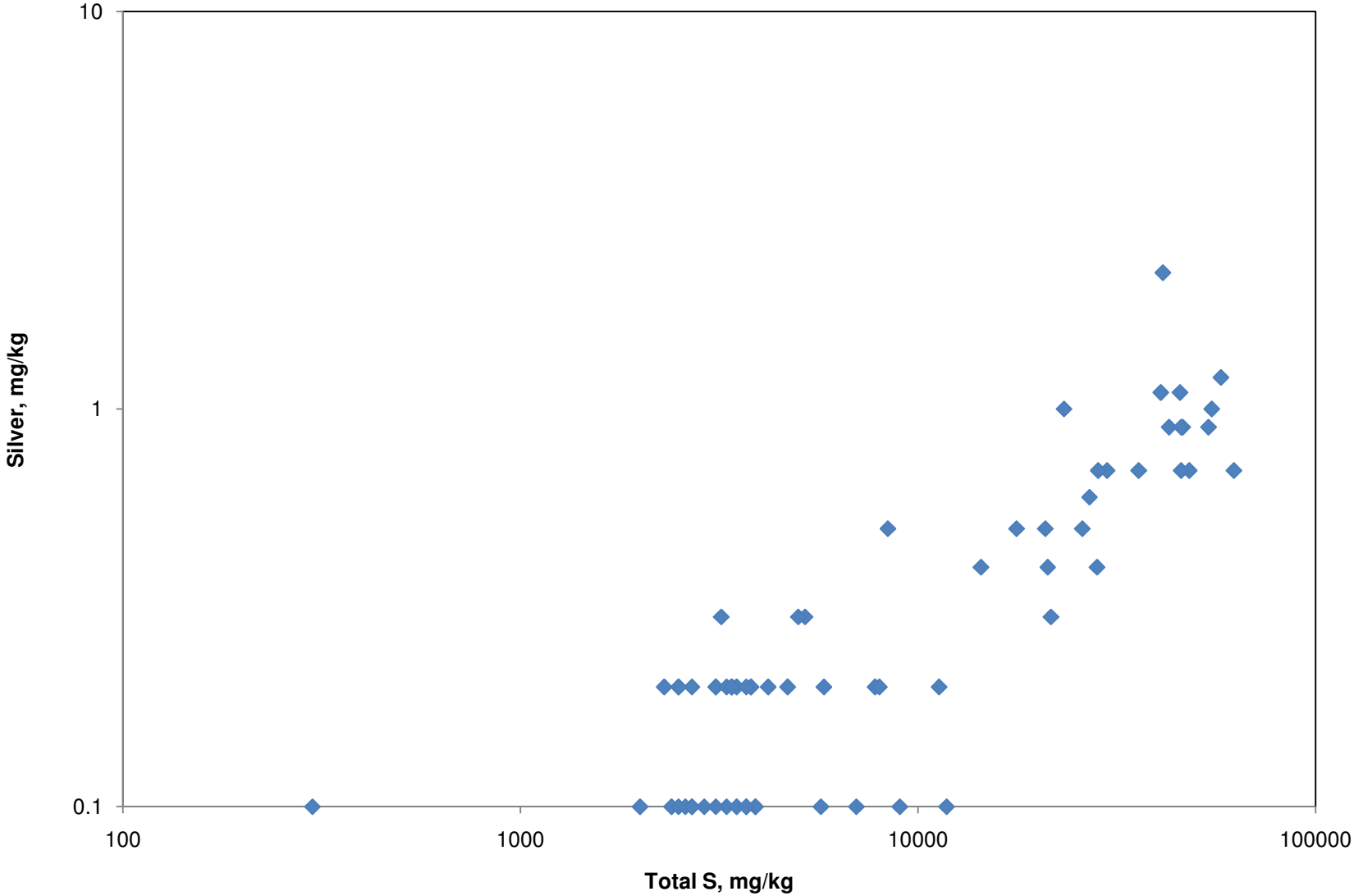
Virginia Formation



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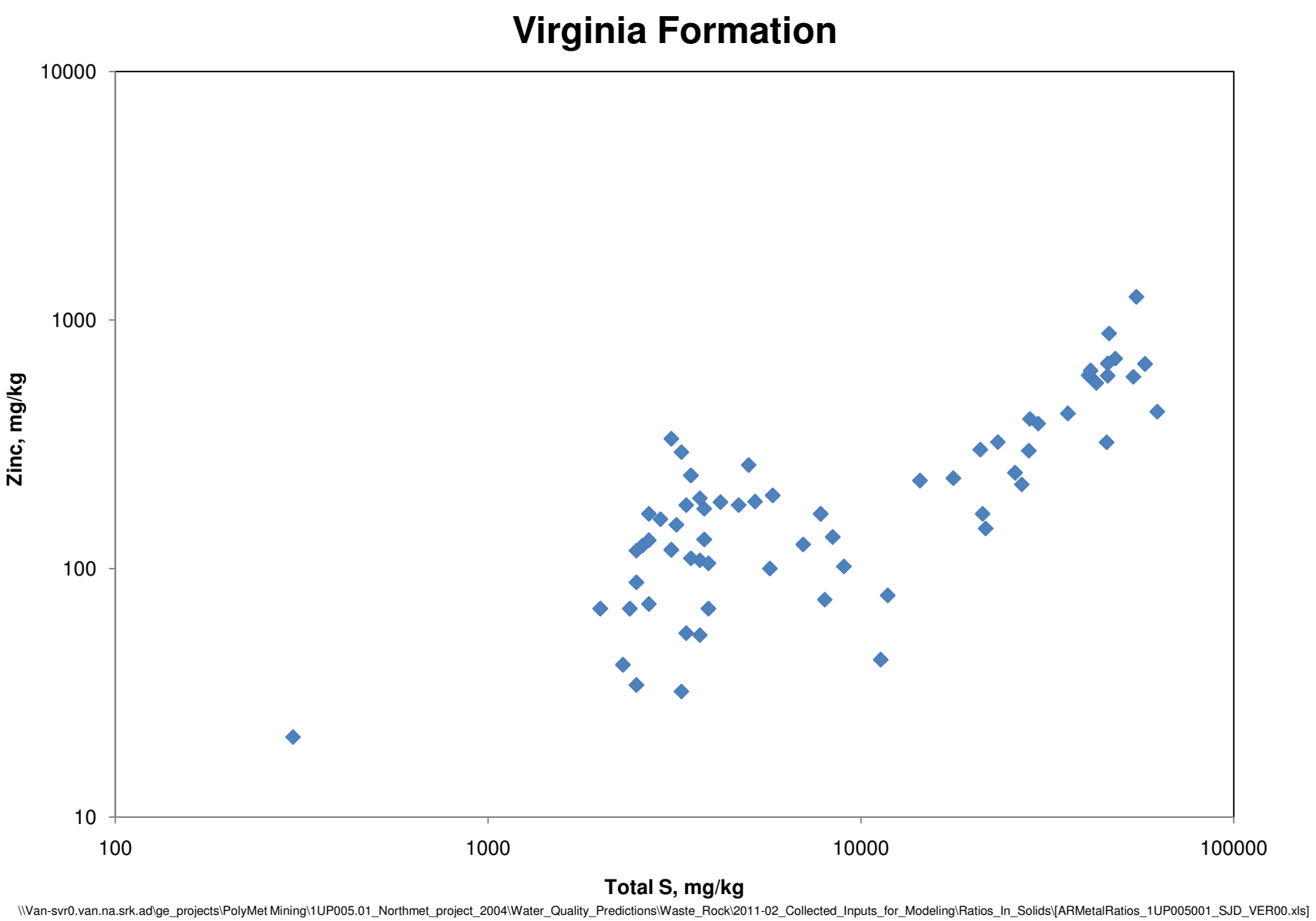
Virginia Formation
10000
1000
100
10
Zinc, mg/kg
100
1000
10000
100000
Total S, mg/kg
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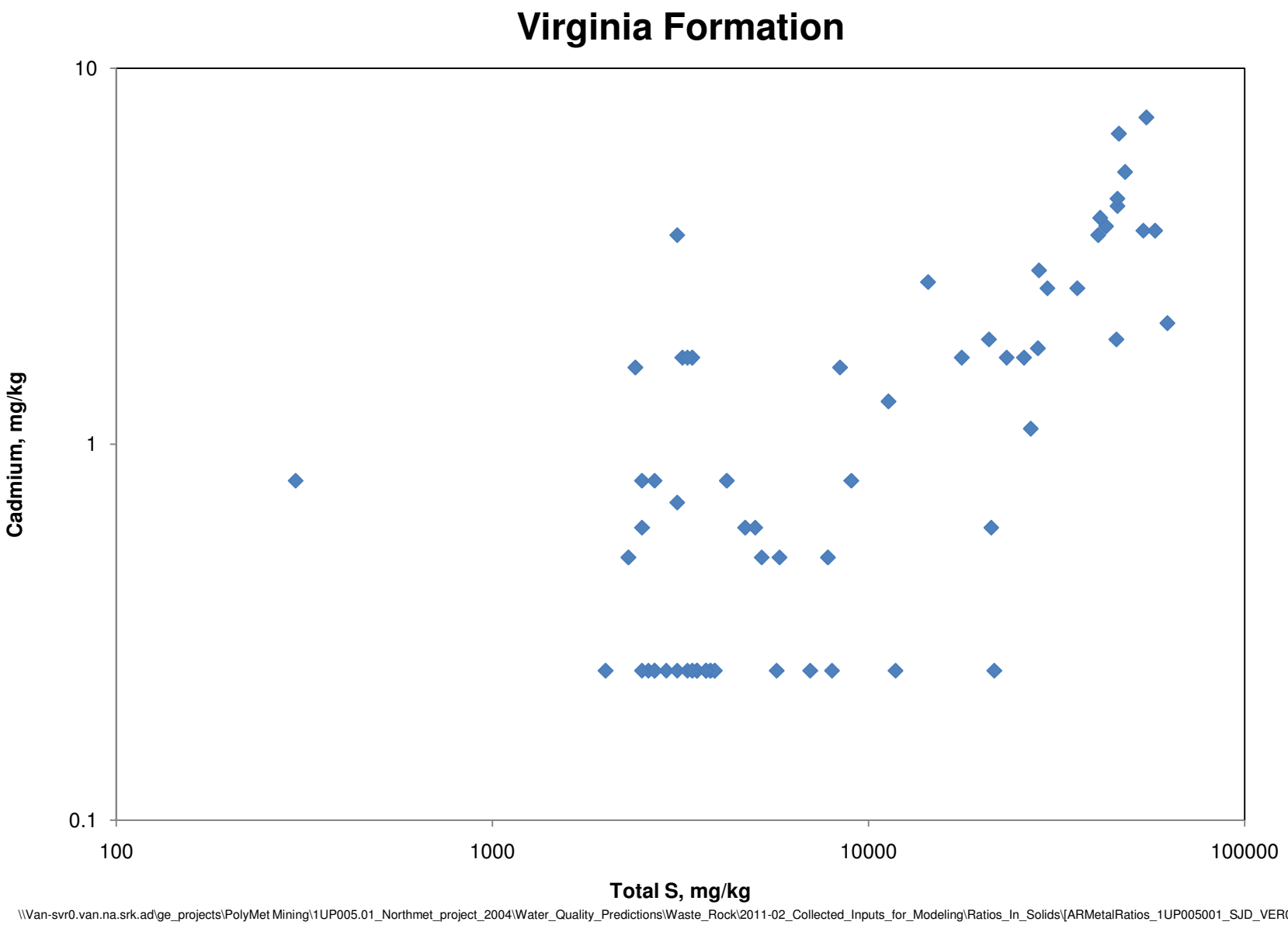
Virginia Formation
10
1
0.1
Cadmium, mg/kg
100
1000
10000
100000
Total S, mg/kg
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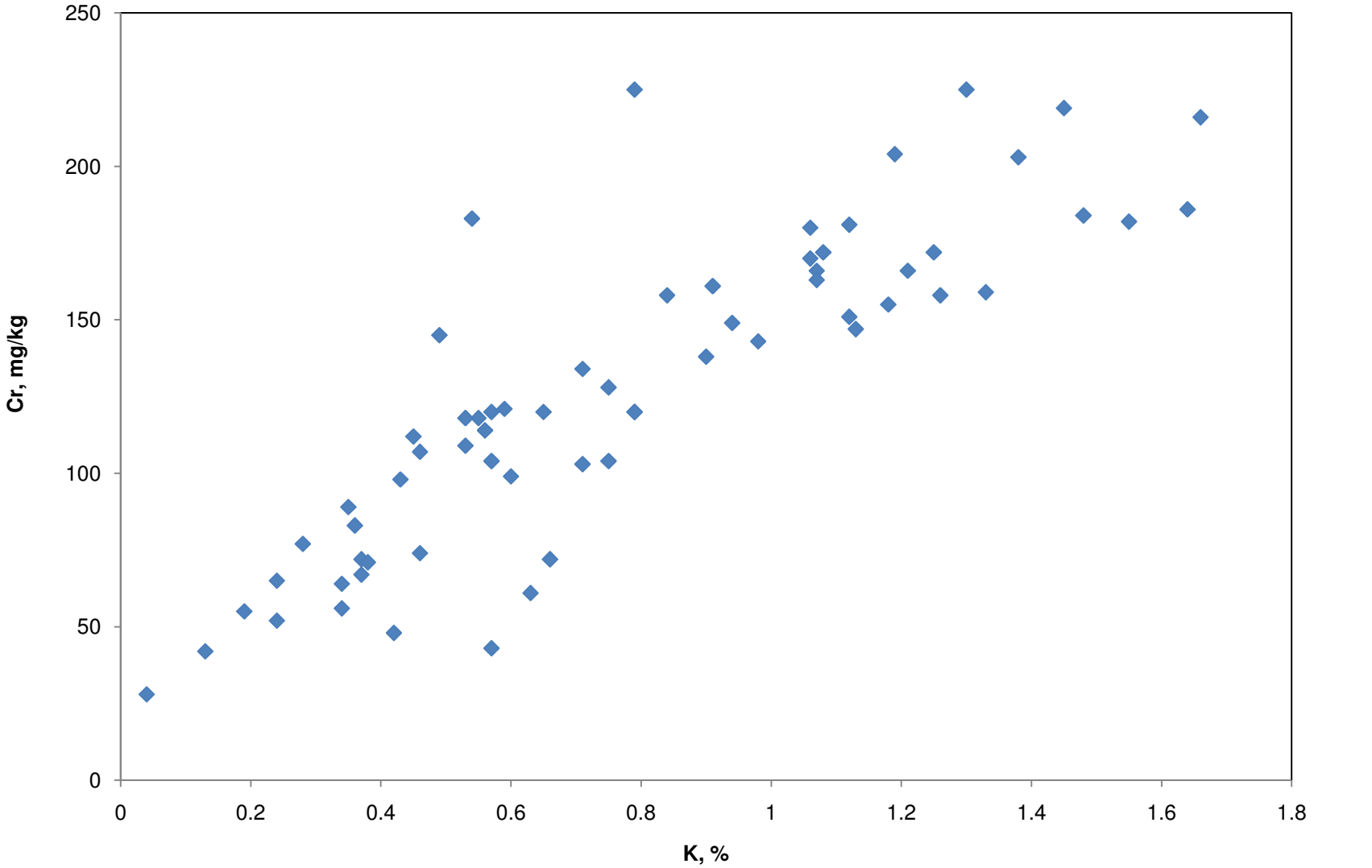
Virginia Formation



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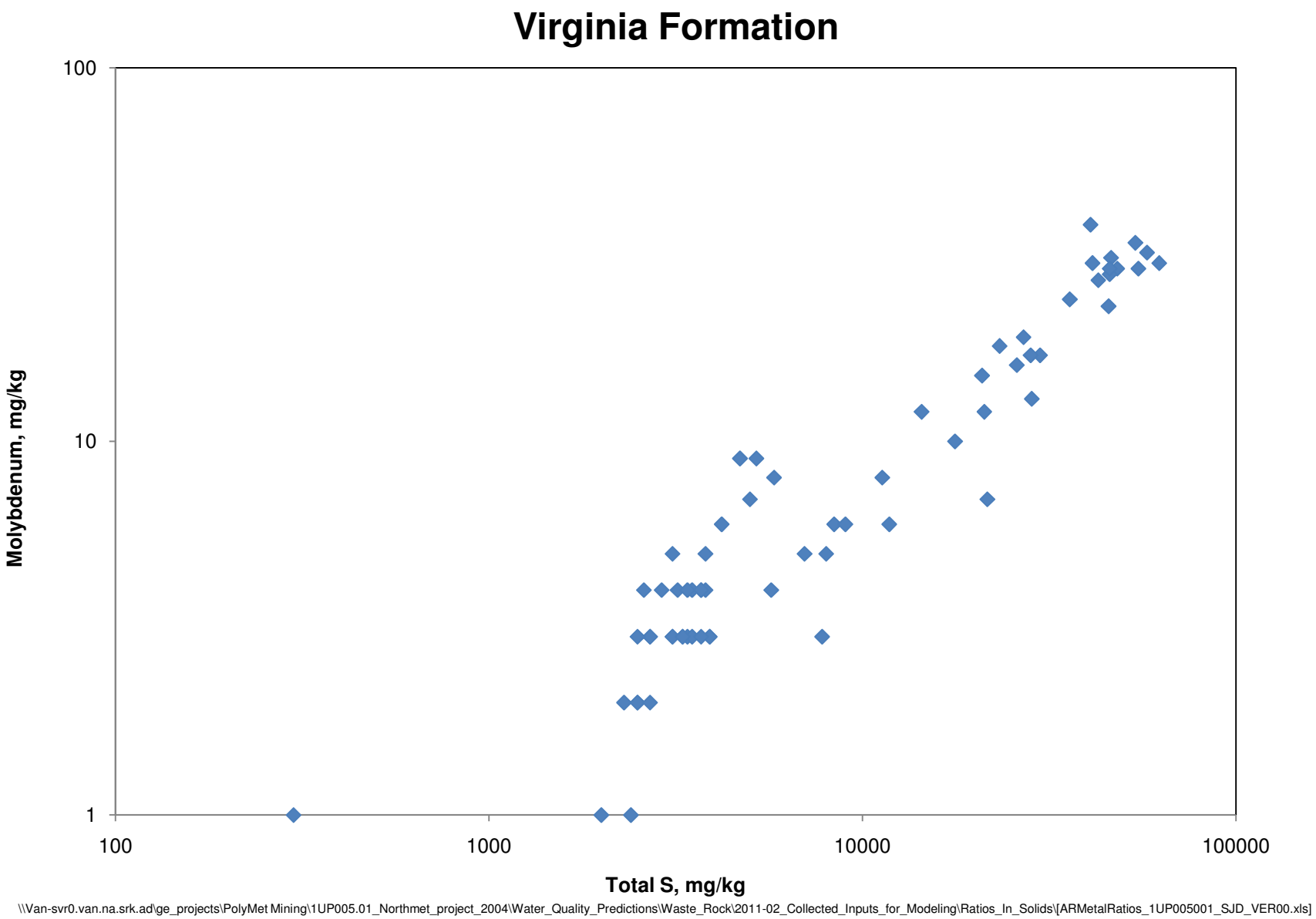
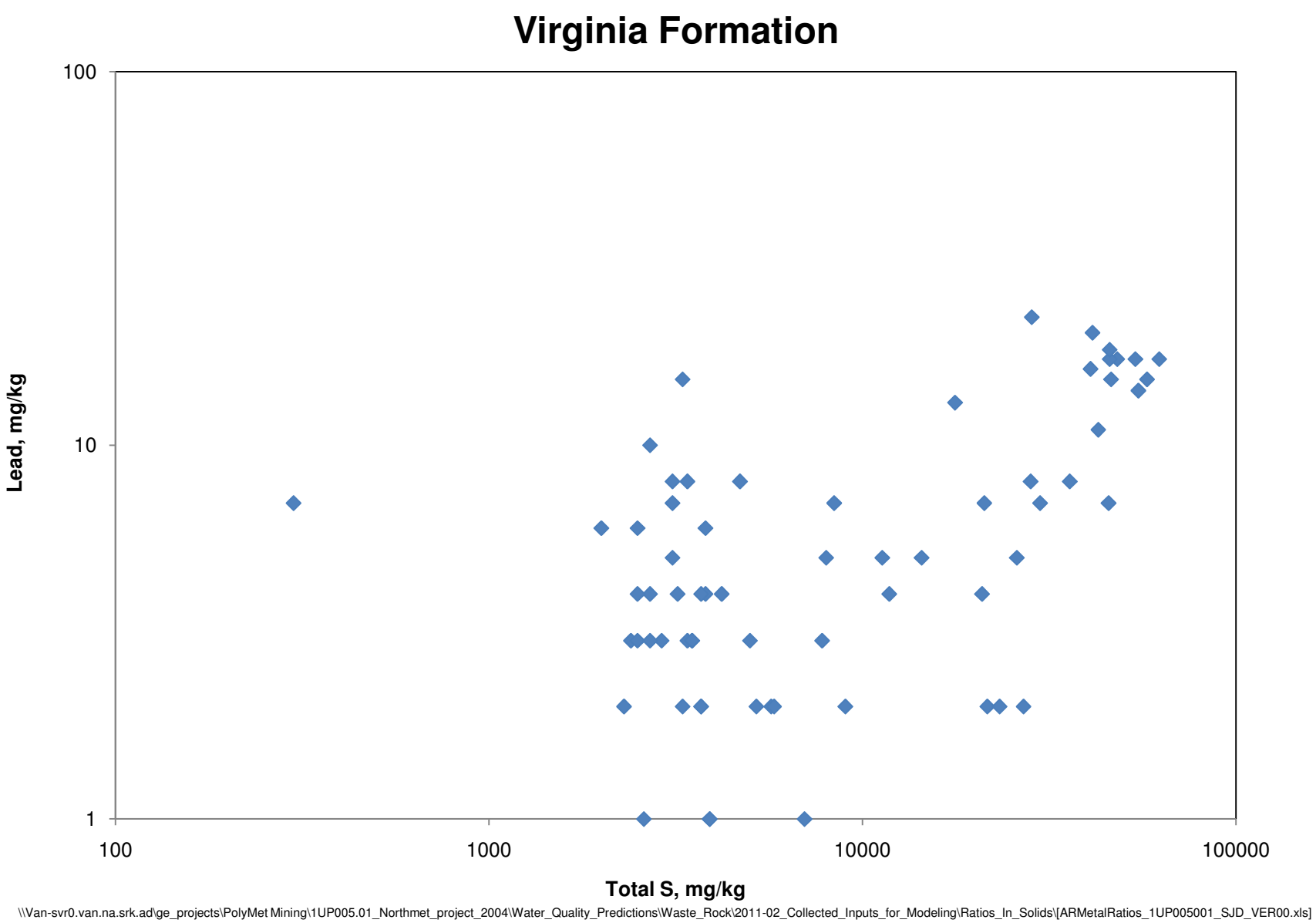
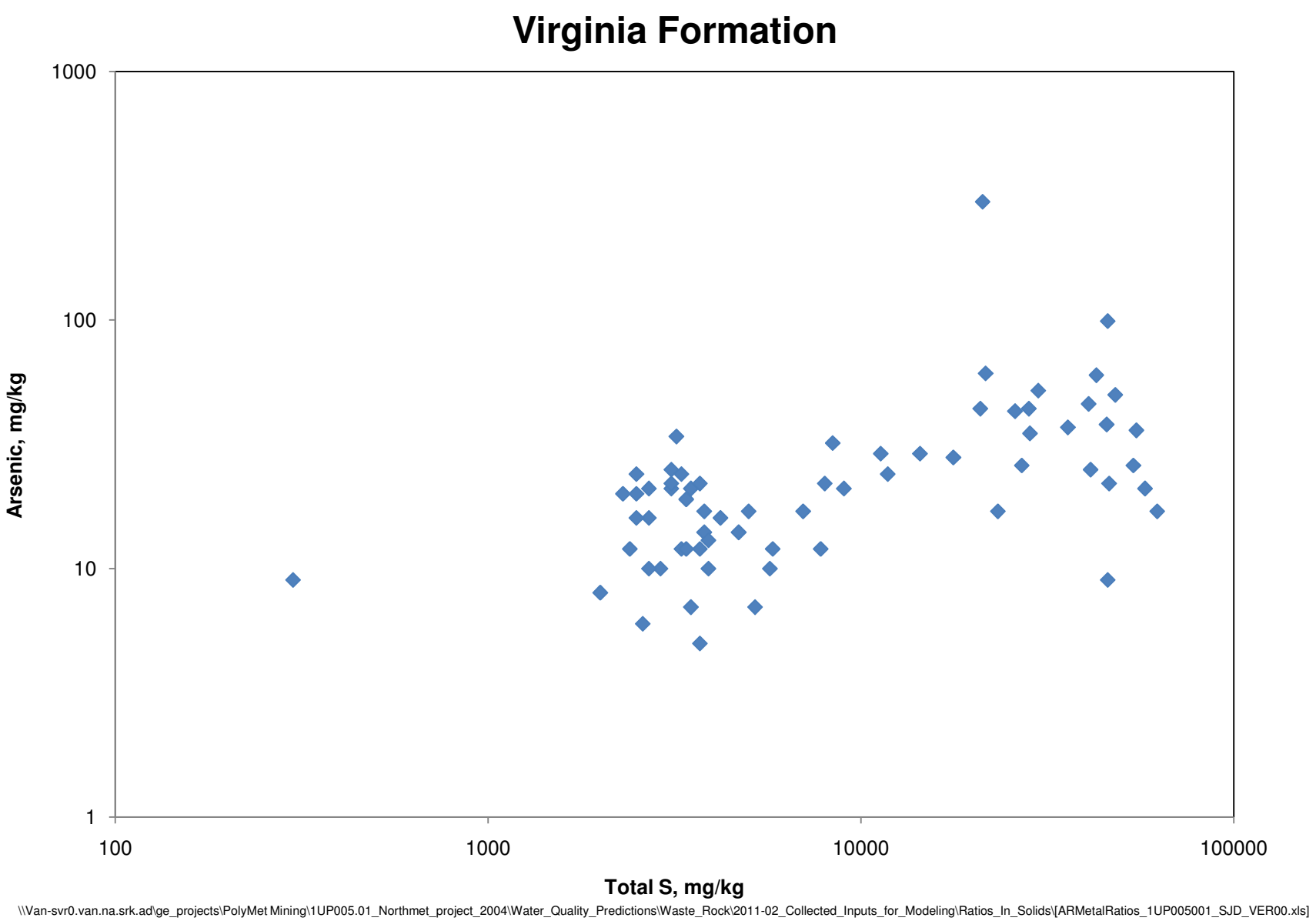

Virginia Formation
100
10
1
Molybdenum, mg/kg
100
1000
10000
100000
Total S, mg/kg
\\Van-svr0.van.na.srk.ad\ge_projects\PolyMet Mining\1UP005.01_Northmet_project_2004\Water_Quality_Predictions\Waste_Rock\2011-02_Collected_Inputs_for_Modeling\Ratios_In_Solids\[ARMetalRatios_1UP005001_SJD_VER00.xls]


Virginia Formation
100
10
1
Lead, mg/kg
100
1000
10000
100000
Total S, mg/kg
\\Van-svr0.van.na.srk.ad\ge_projects\PolyMet Mining\1UP005.01_Northmet_project_2004\Water_Quality_Predictions\Waste_Rock\2011-02_Collected_Inputs_for_Modeling\Ratios_In_Solids\[ARMetalRatios_1UP005001_SJD_VER00.xls]


Virginia Formation
1000
100
10
1
Arsenic, mg/kg
100
1000
10000
100000
Total S, mg/kg
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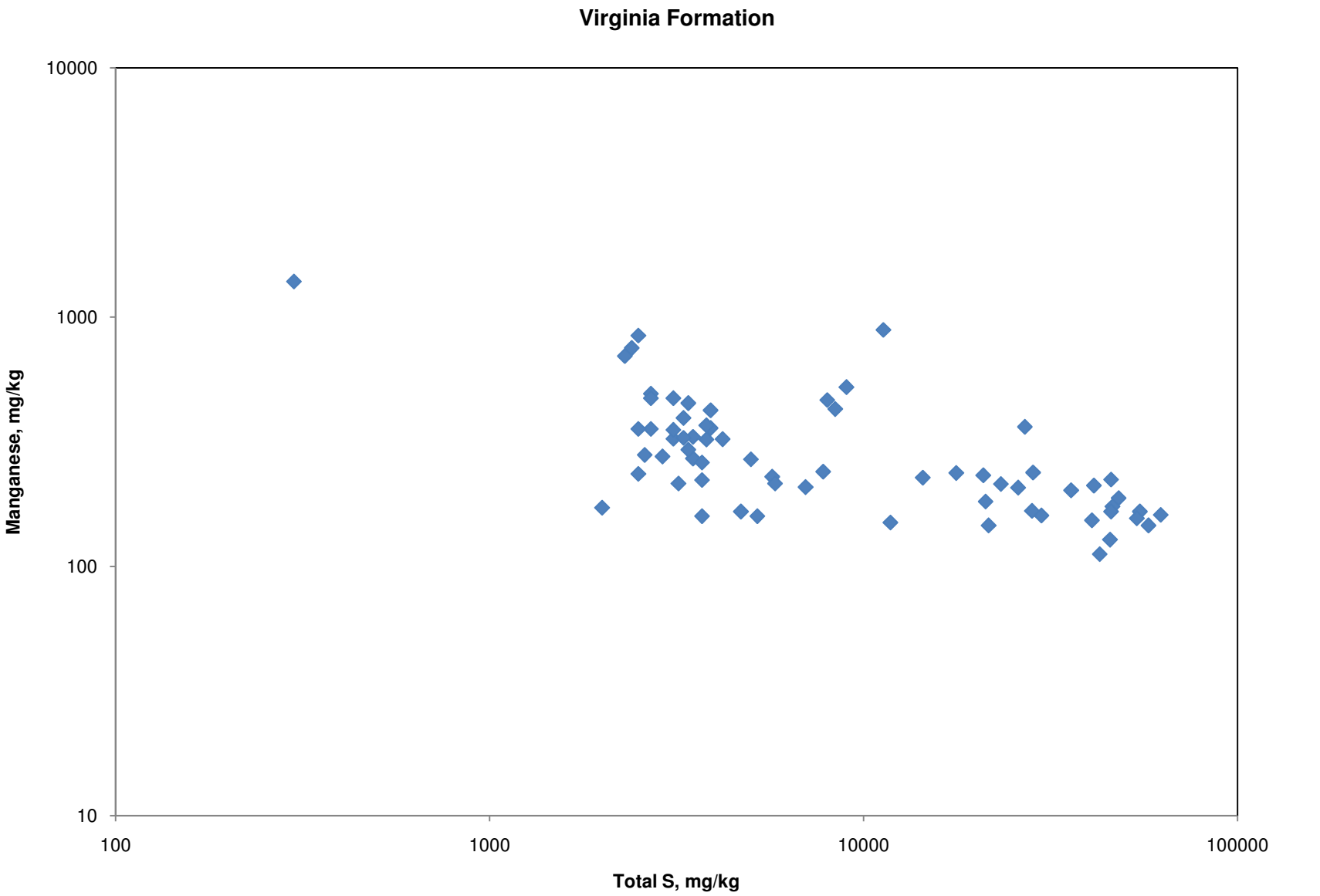

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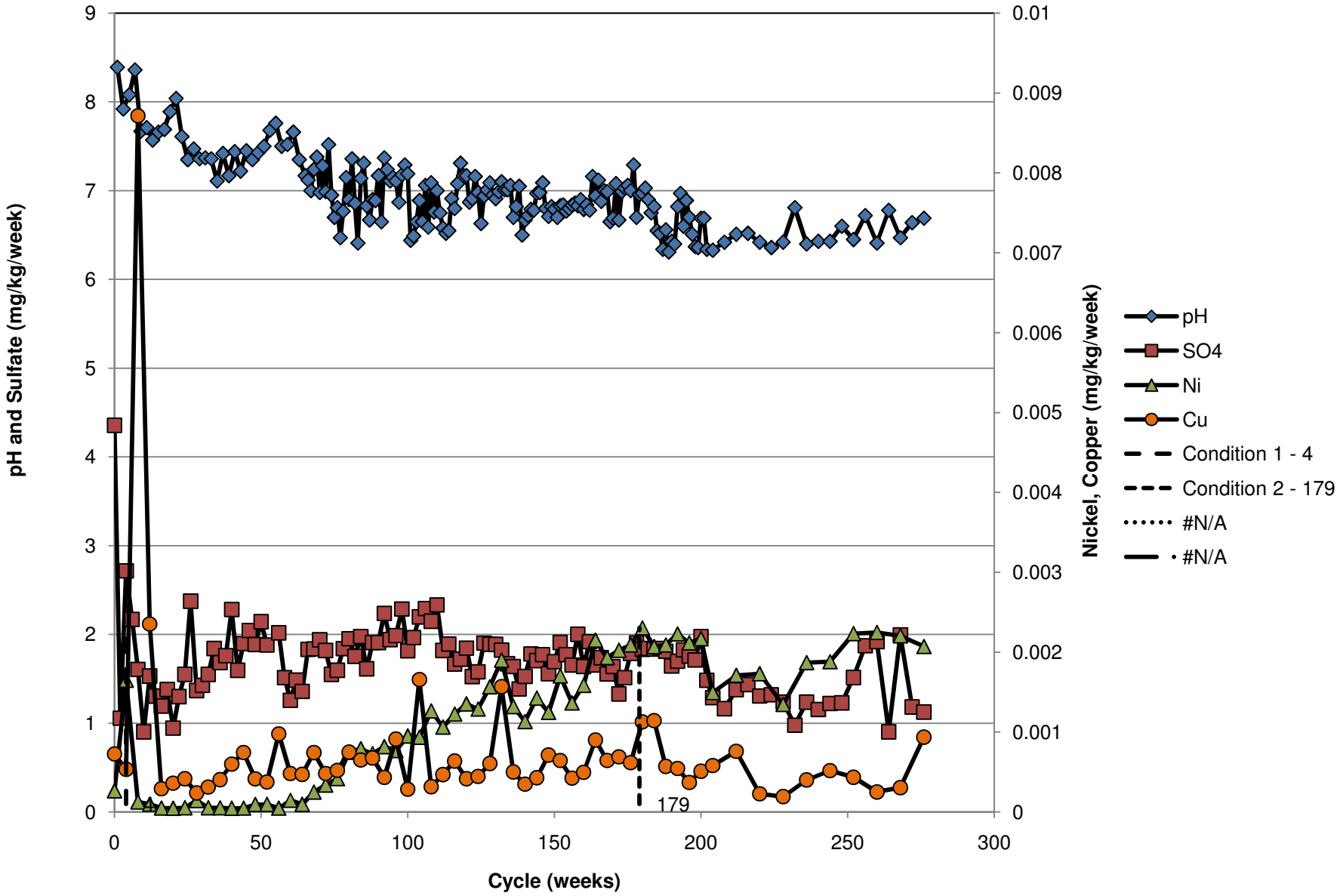
Attachment 2
Trend Analysis for Rock Humidity Cells

a. Category 1

Waste Category:	1
Rock Type:	Anorthositic
Geology:	Duluth Complex
Sulfur:	0.09%

99-320C(830-850)



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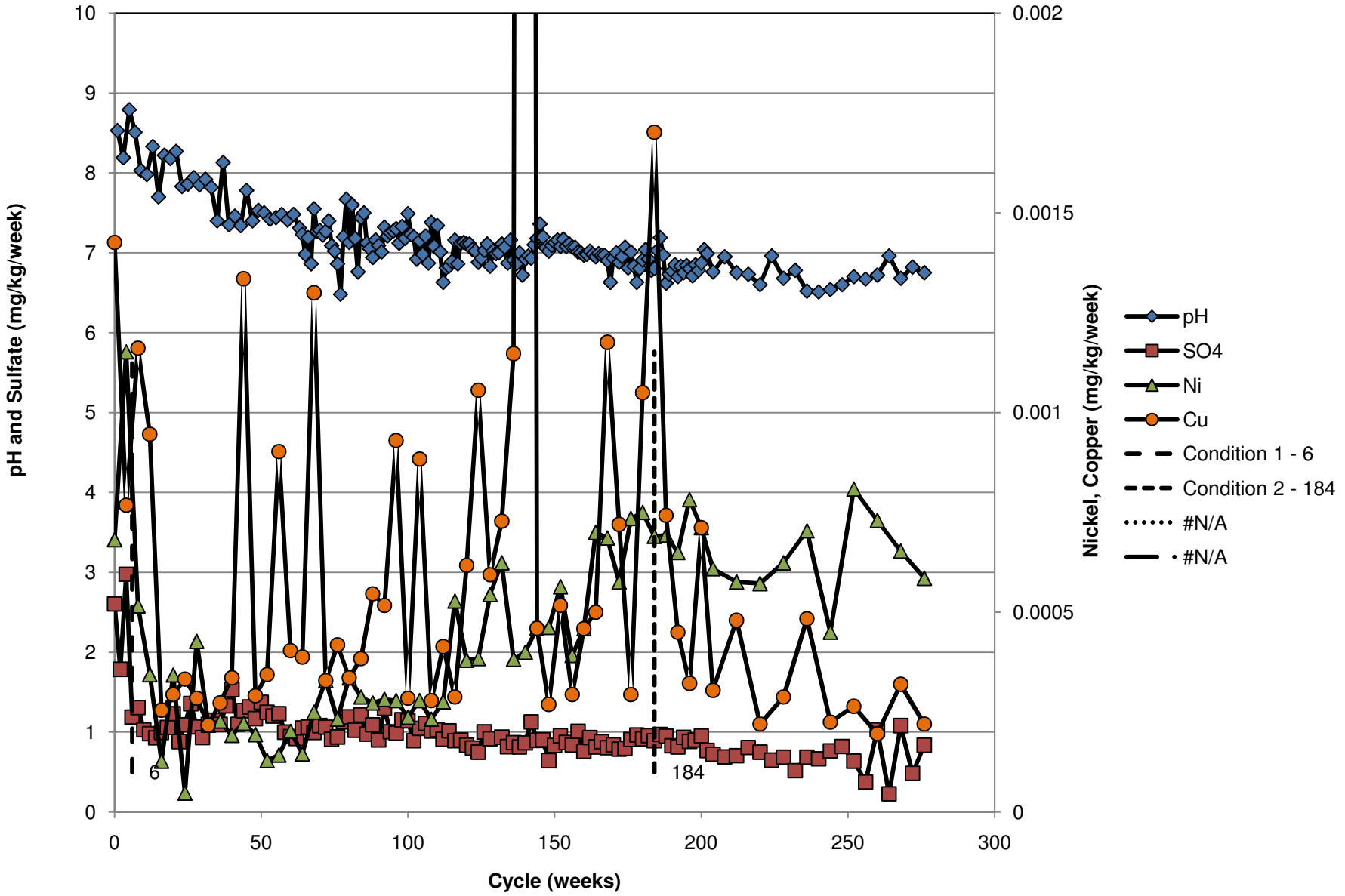
Waste Category:	1
Rock Type:	Anorthositic
Geology:	Duluth Complex
Sulfur:	0.05%

00-361C(345-350)



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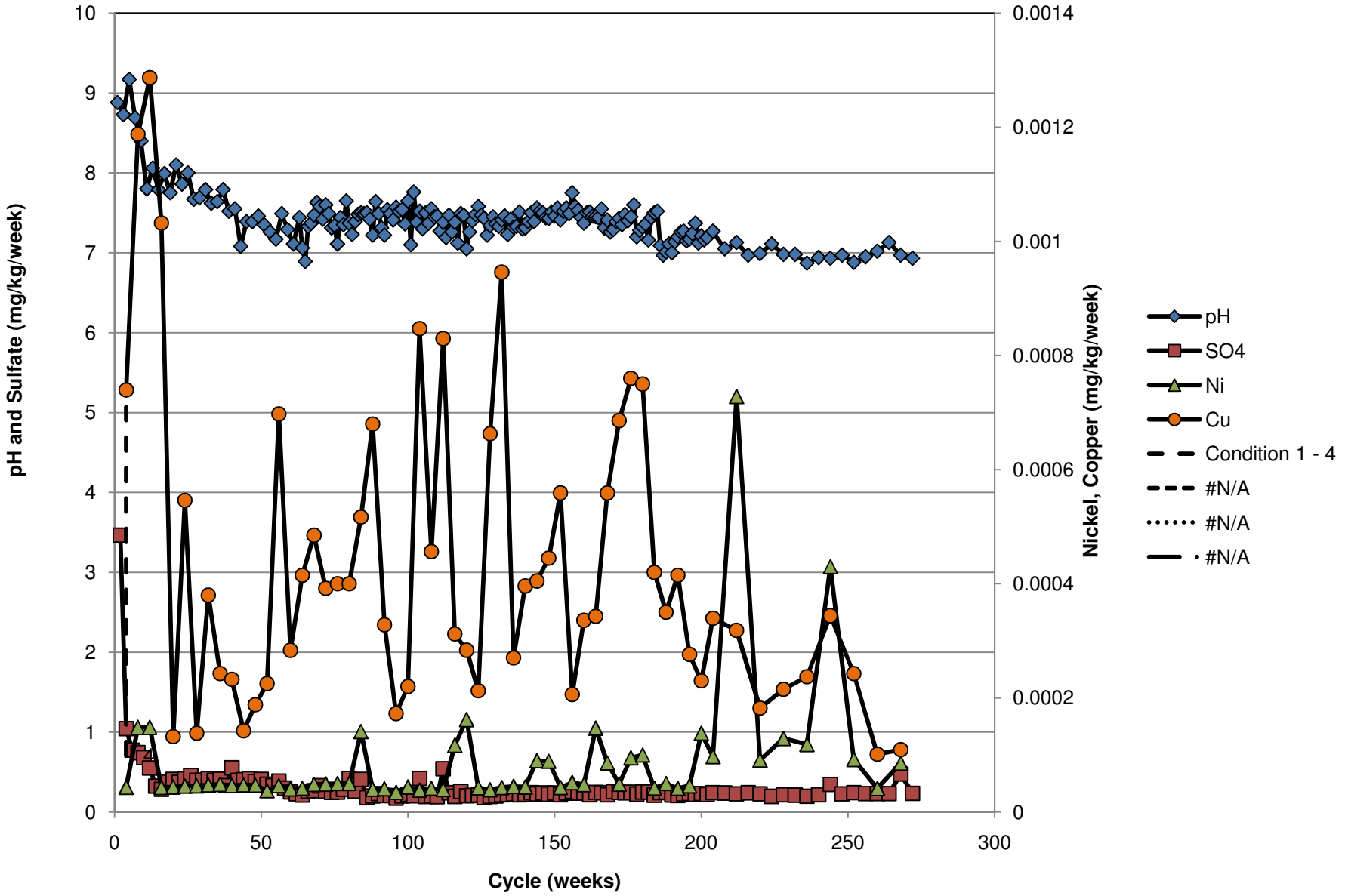
Waste Category:	1
Rock Type:	Anorthositic
Geology:	Duluth Complex
Sulfur:	0.02%

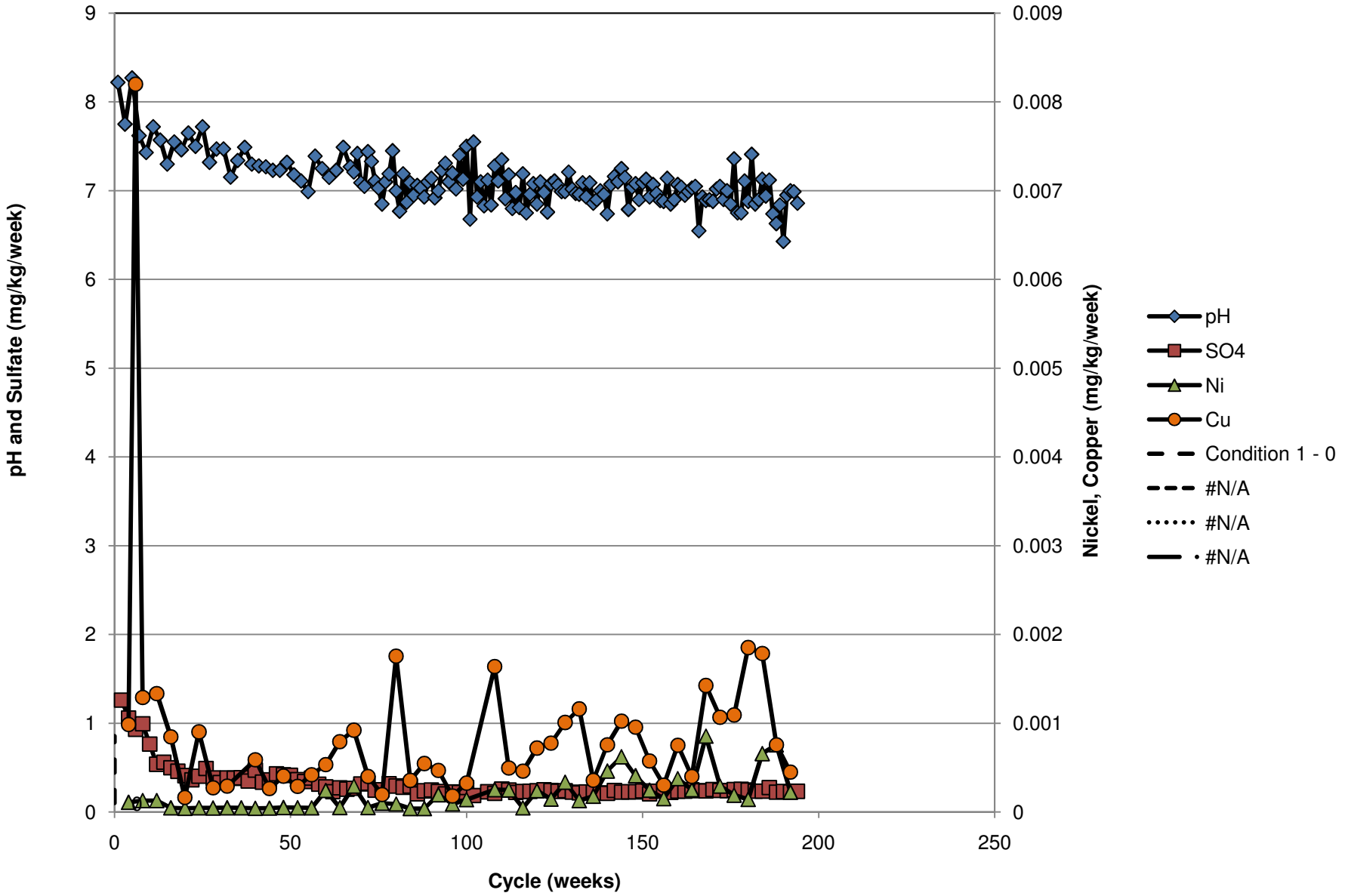
00-334C(30-50)



\\Van-svr0.van.na.srk.ad\ge_projects\PolyMet

Waste Category:	1
Rock Type:	Anorthositic
Geology:	Duluth Complex
Sulfur:	0.02%

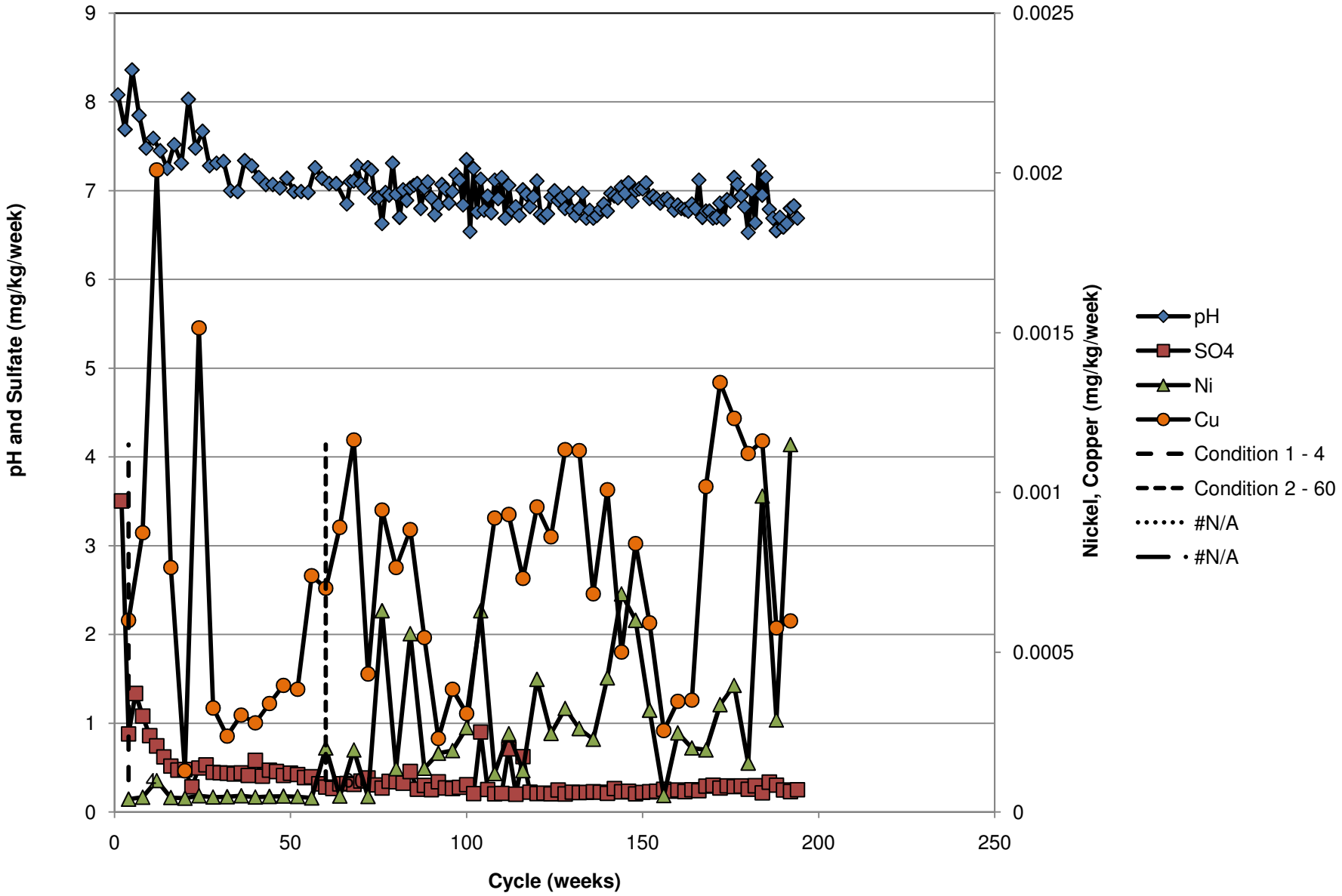
00-366C(185-205)



\\Van-svr0.van.na.srk.ad\ge_projects\PolyMet

Waste Category:	1
Rock Type:	Anorthositic
Geology:	Duluth Complex
Sulfur:	0.02%

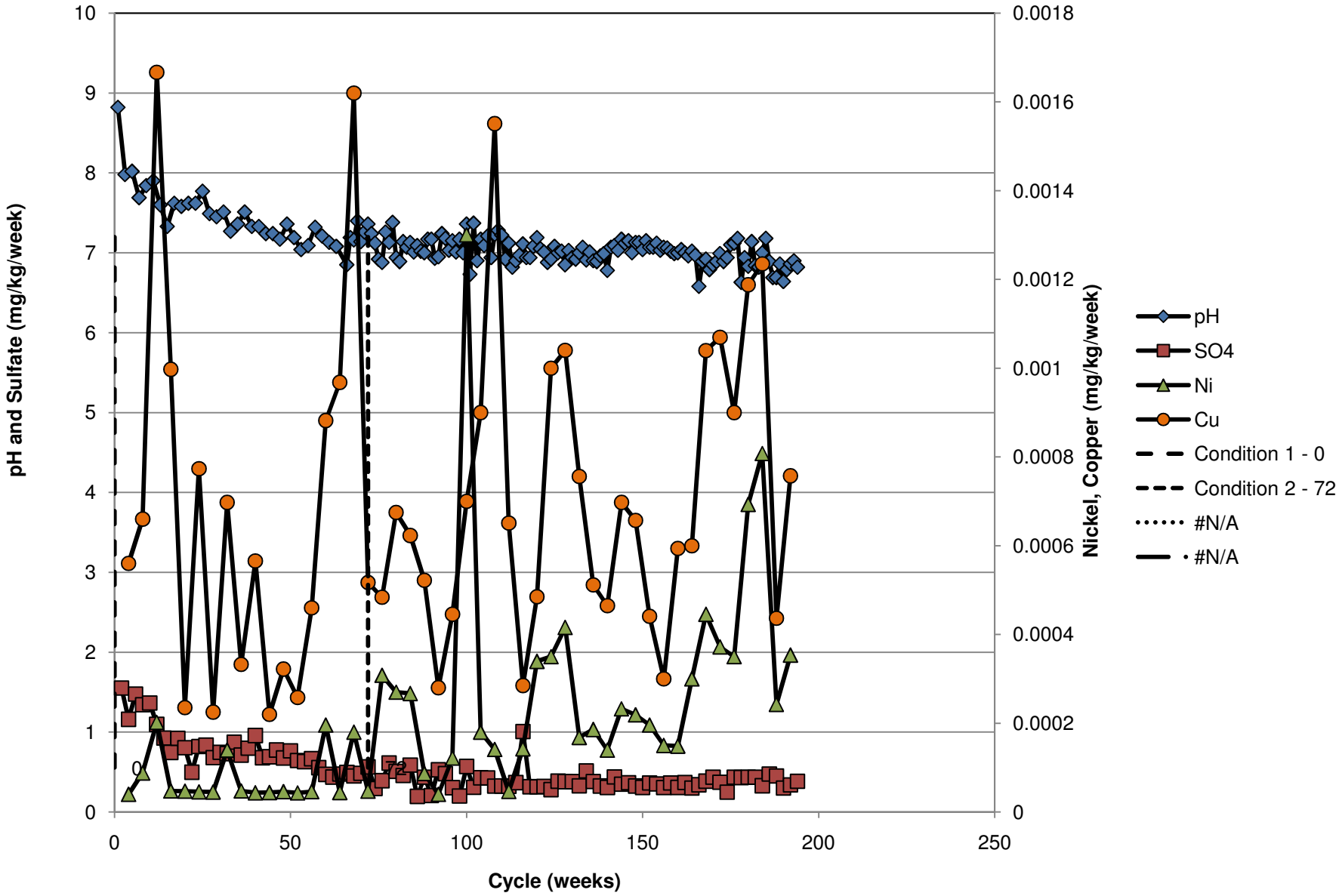
00-366C(230-240)



\\Van-svr0.van.na.srk.ad\ge_projects\PolyMet

Waste Category:	1
Rock Type:	Anorthositic
Geology:	Duluth Complex
Sulfur:	0.03%

99-320C(165-175)



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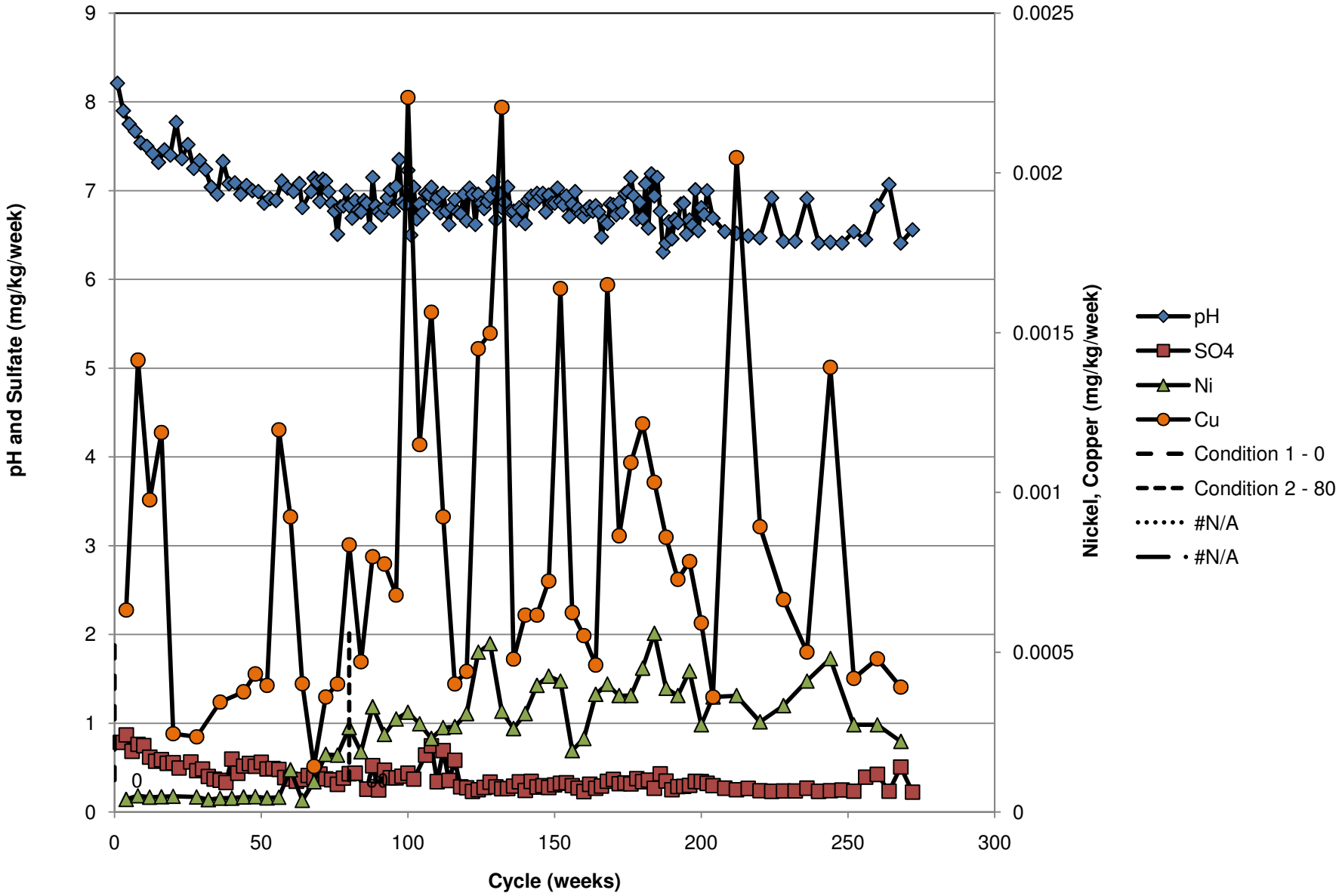
Waste Category:	1
Rock Type:	Anorthositic
Geology:	Duluth Complex
Sulfur:	0.04%

00-368C(125-145)



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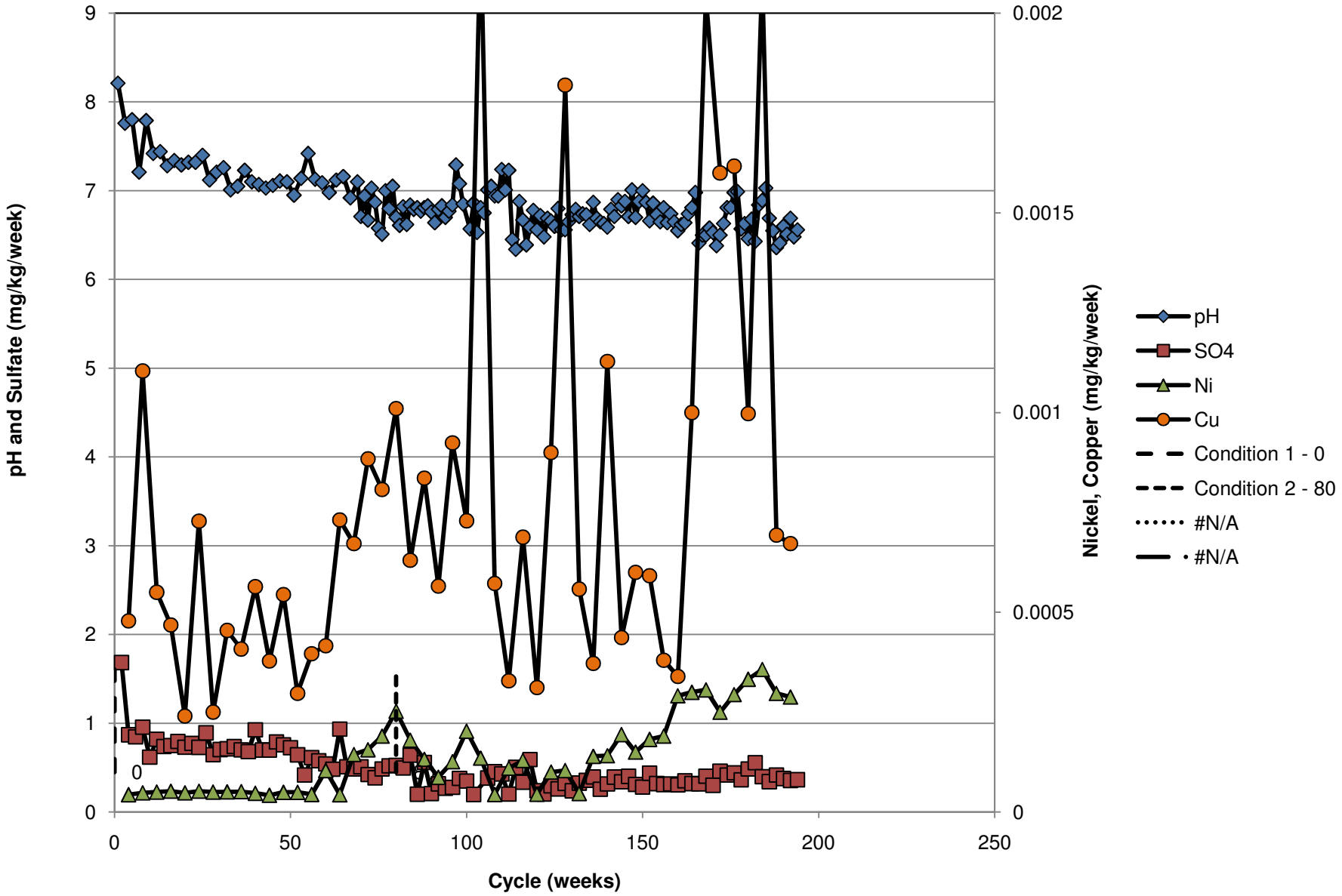
Waste Category:	1
Rock Type:	Anorthositic
Geology:	Duluth Complex
Sulfur:	0.04%

00-368C(20-40)



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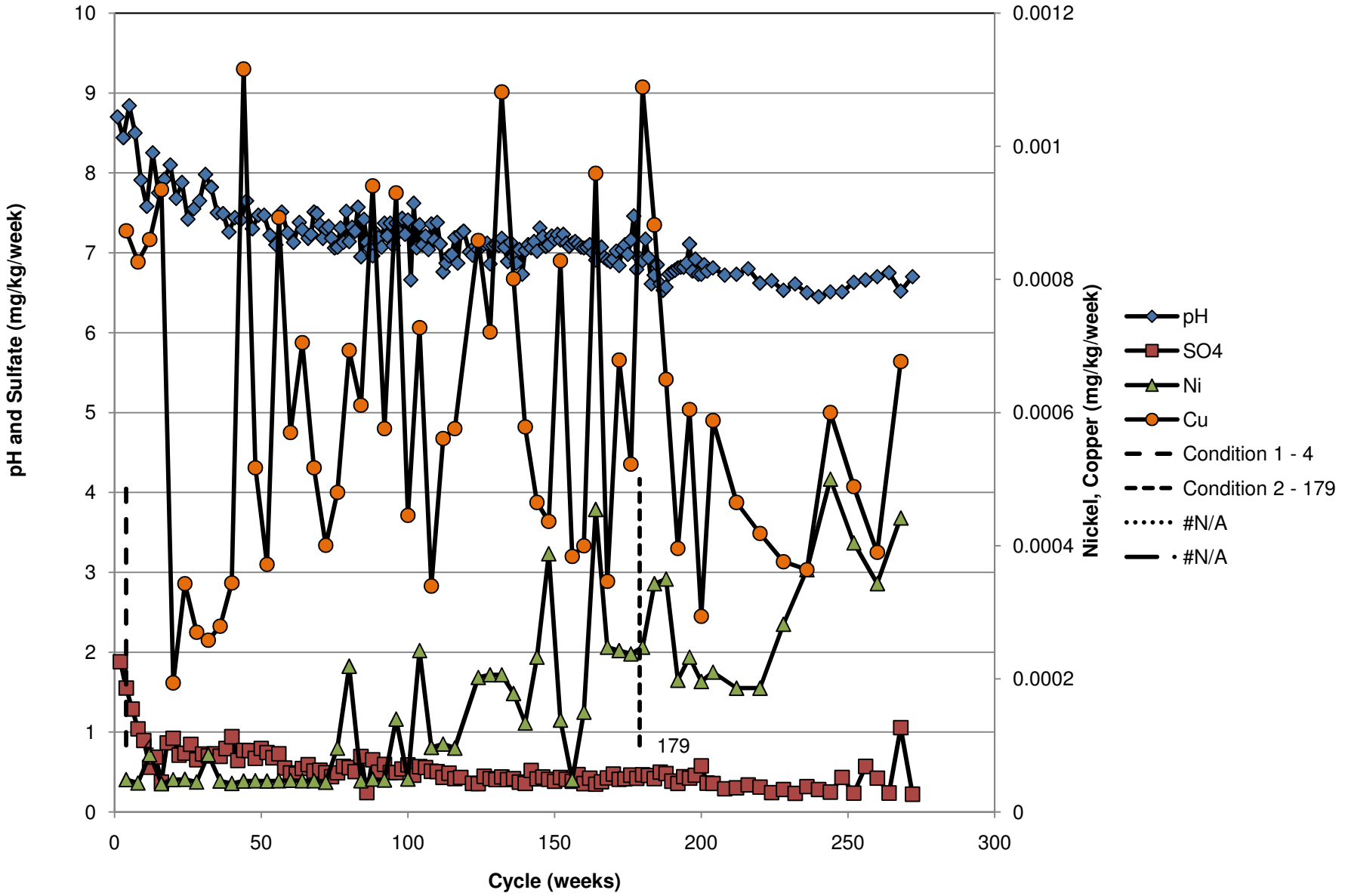
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.06%

00-334C(580-600)



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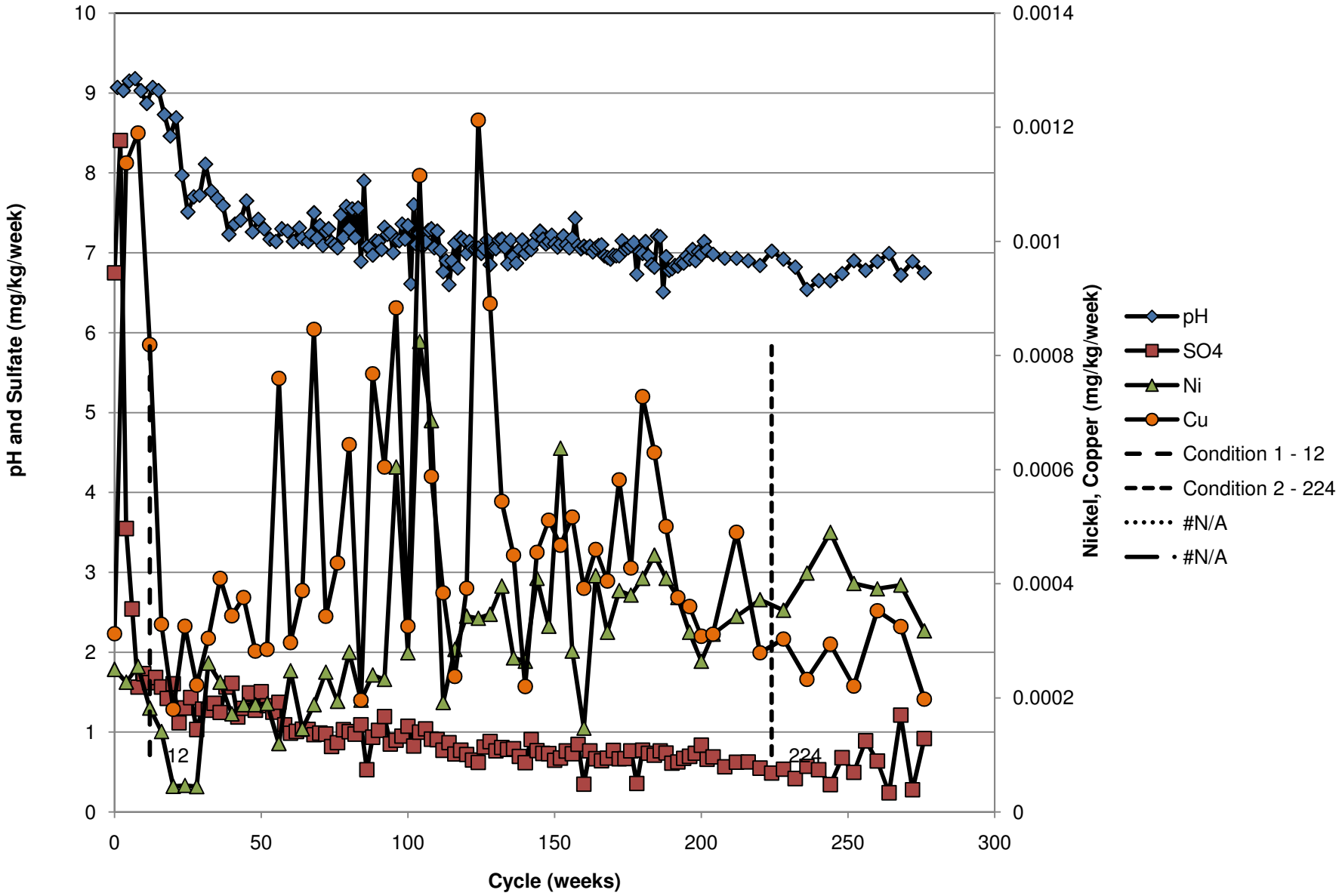
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.07%

00-334C(640-660)



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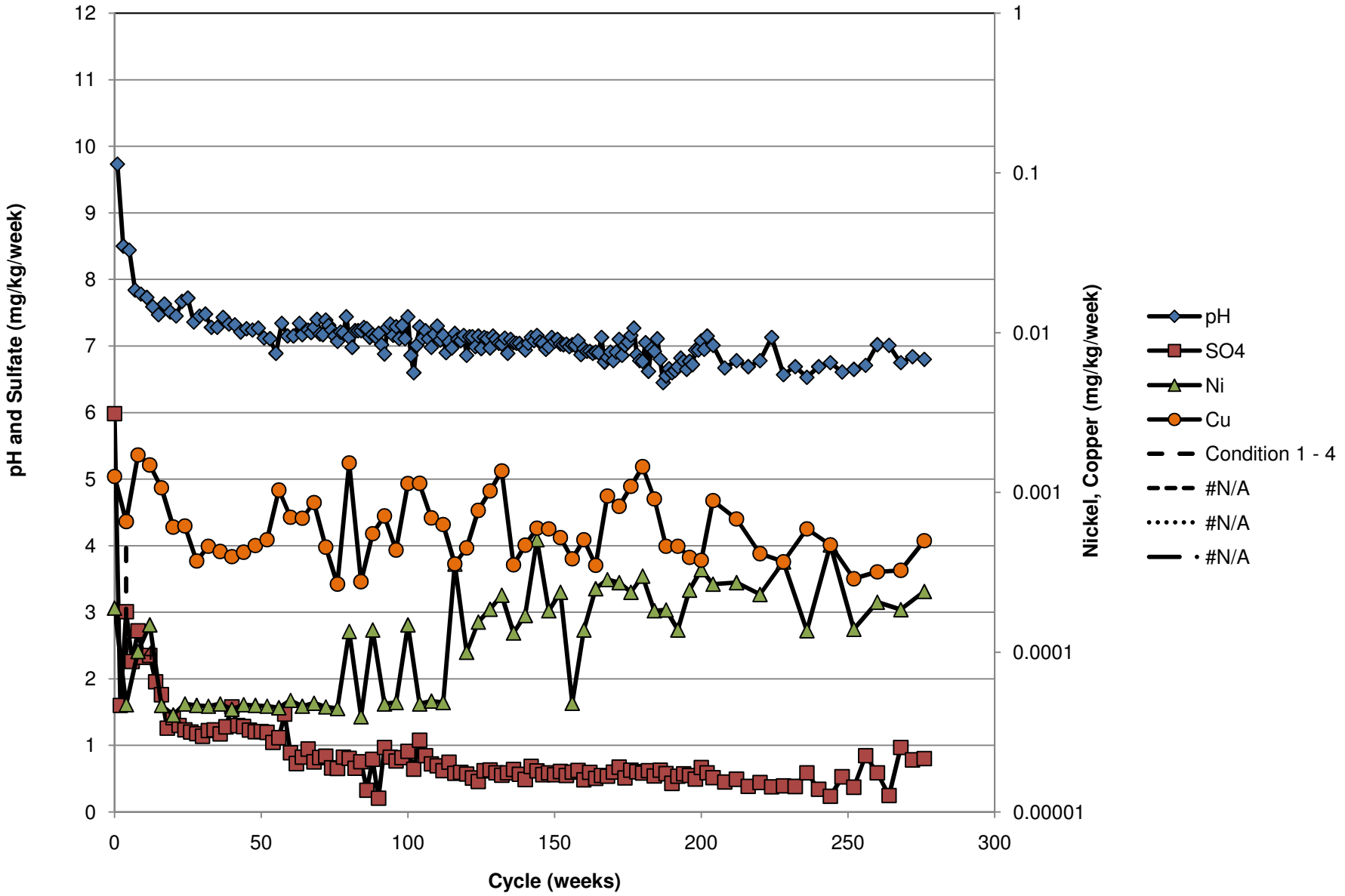
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.07%

99-320C(315-330)



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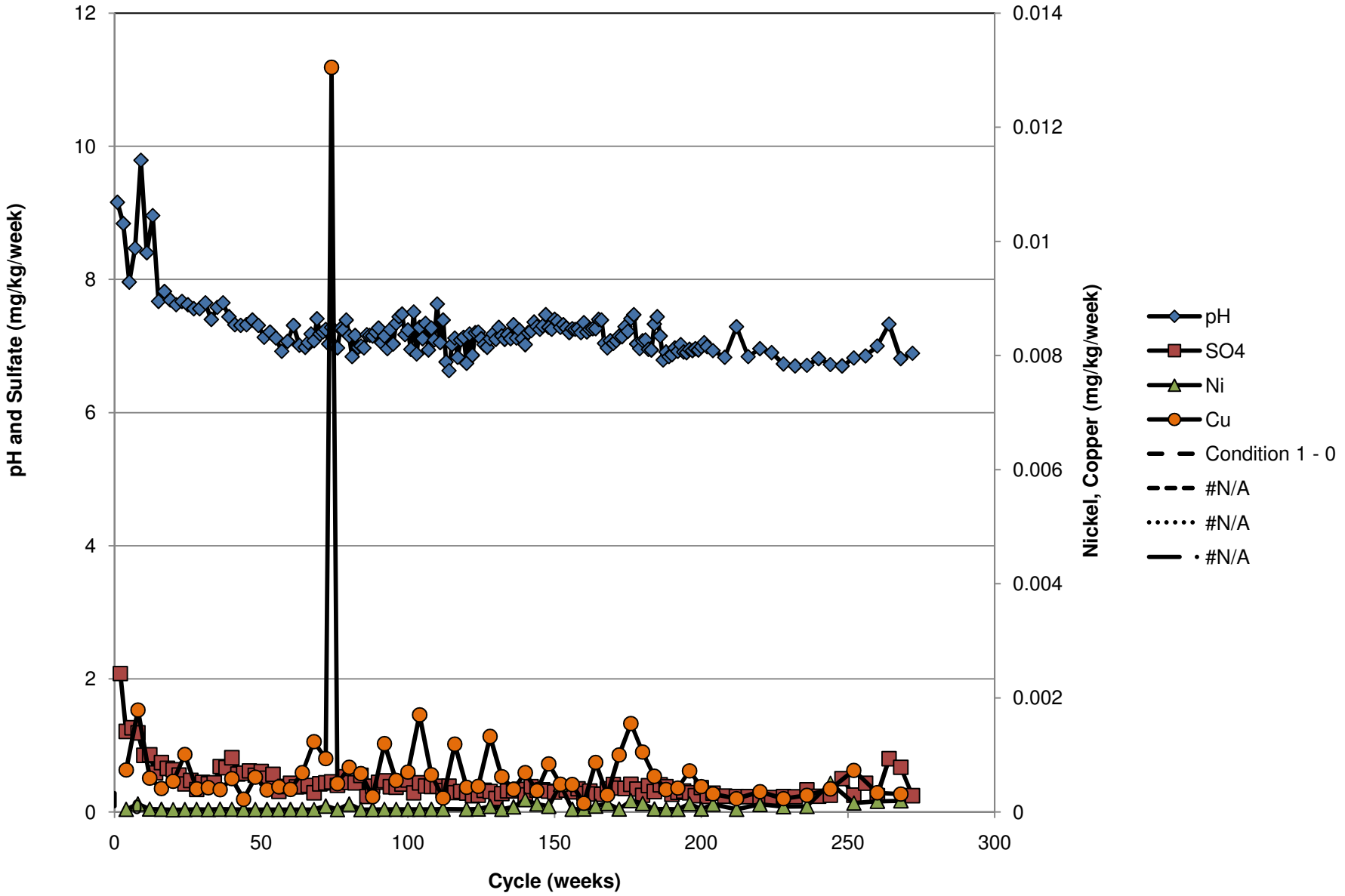
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.04%

00-367C(290-310)



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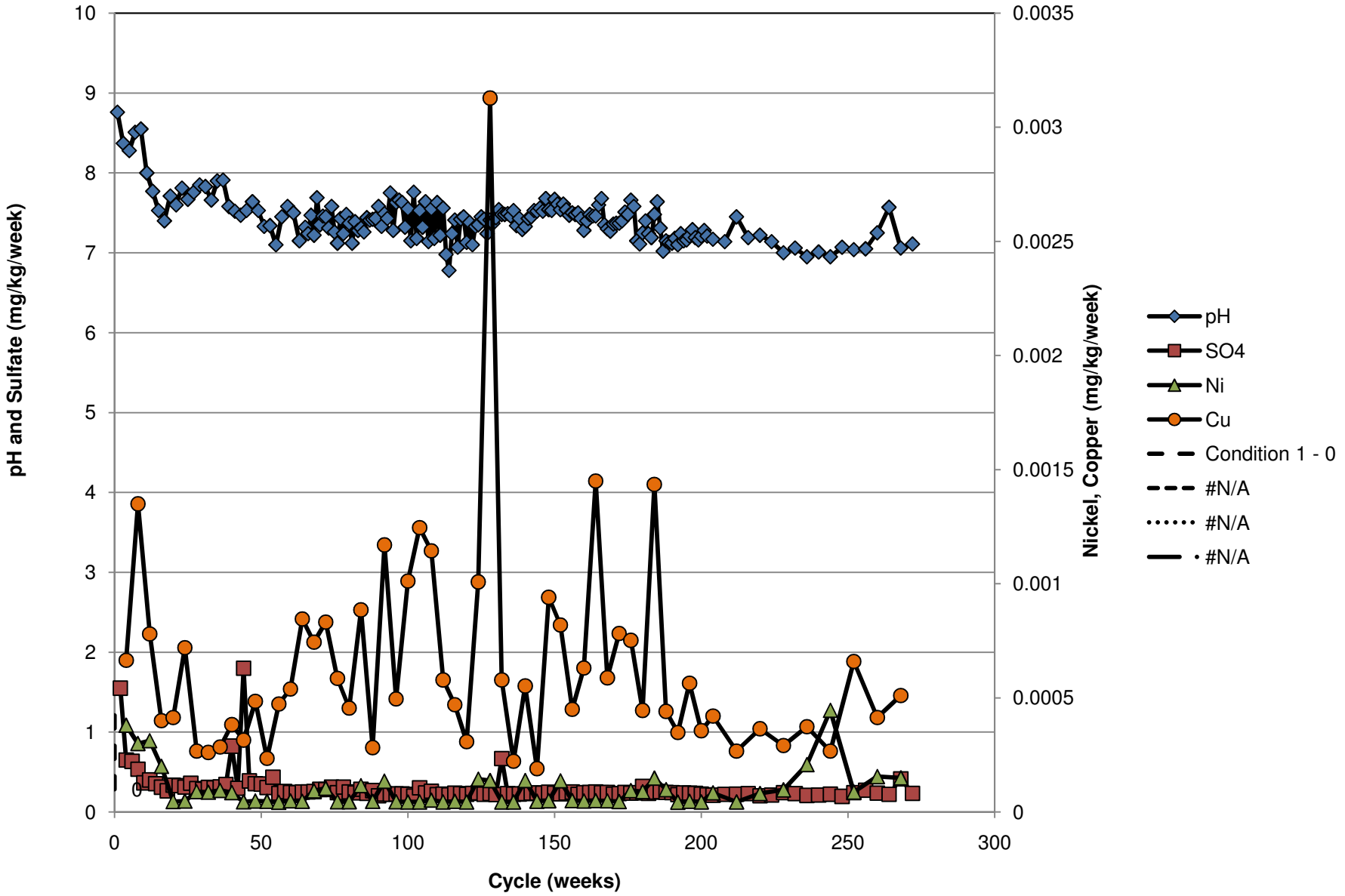
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.02%

26064(44-54)



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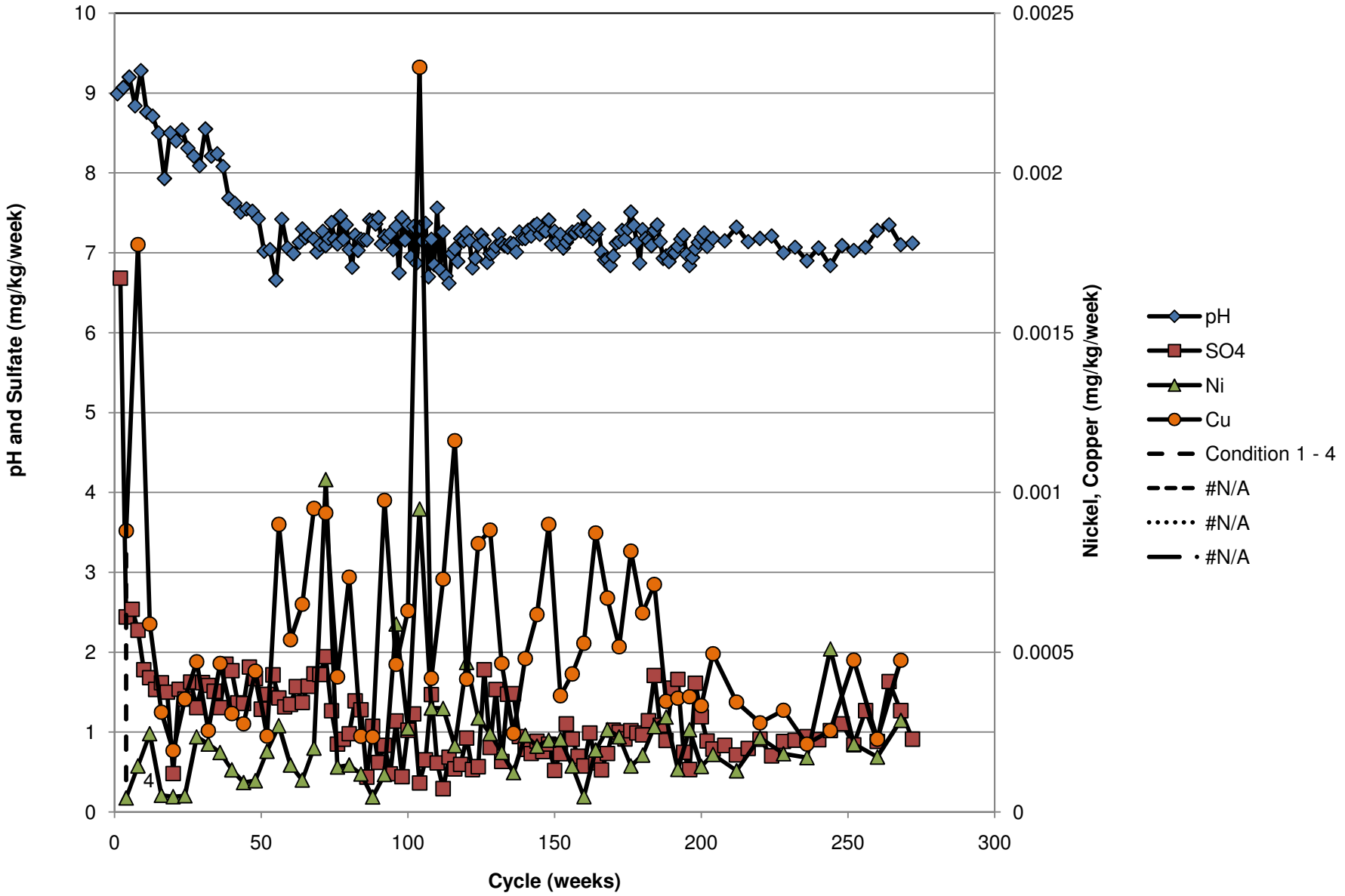
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.06%

26064(264+146-269+156)



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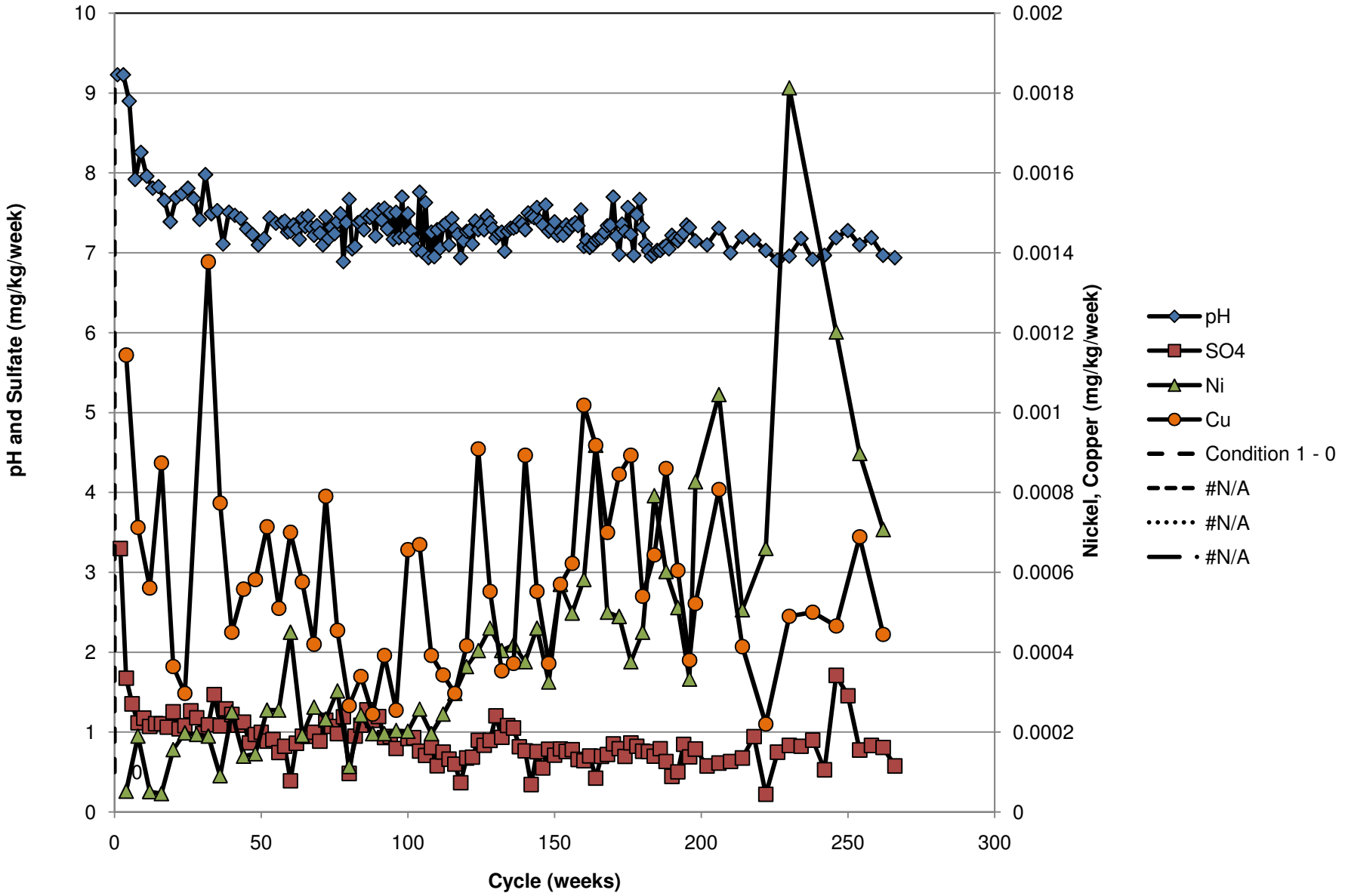
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.05%

26056(135-153)



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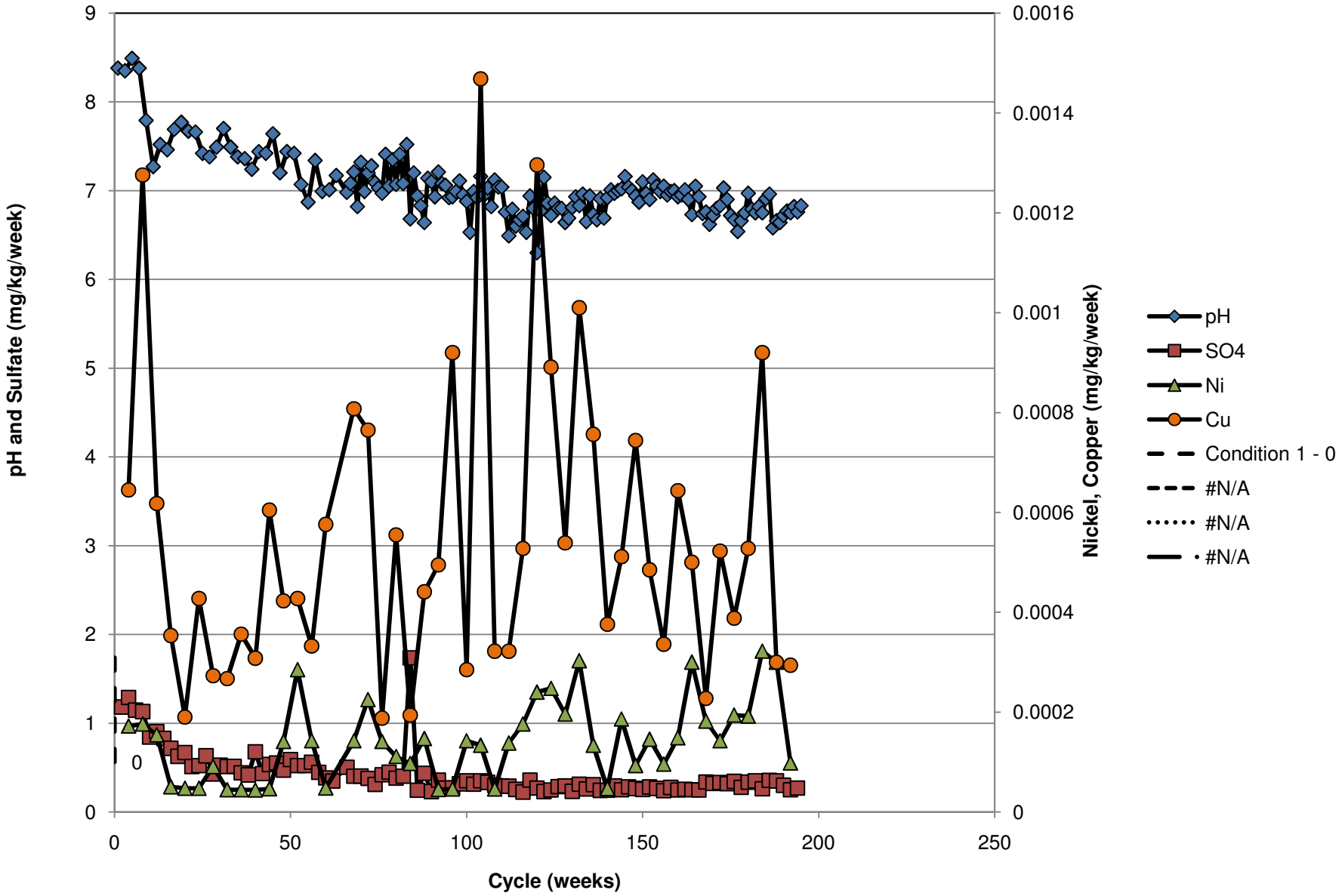
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.04%

00-340C(595-615)



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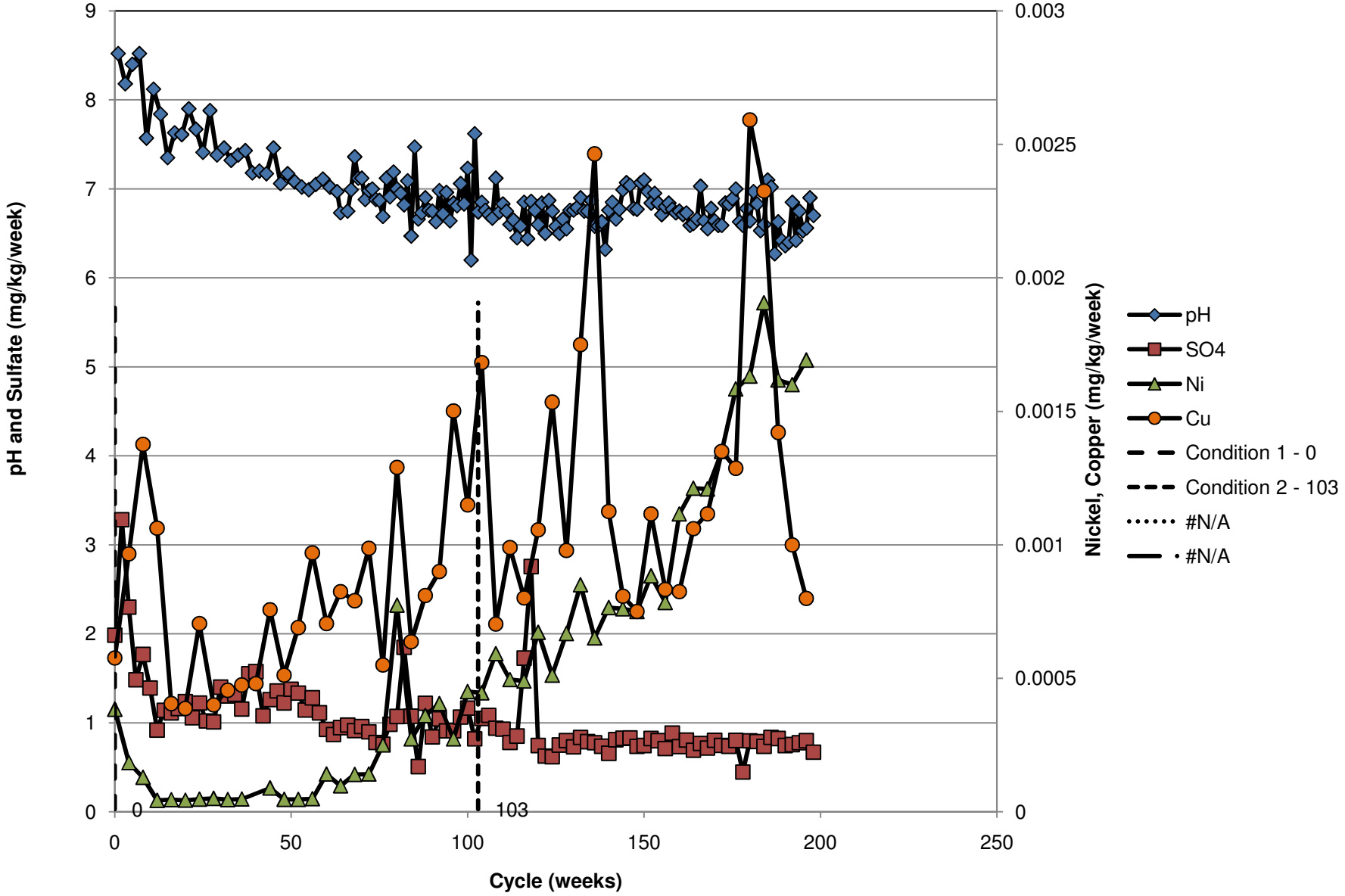
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.07%

00-347C(795-815)



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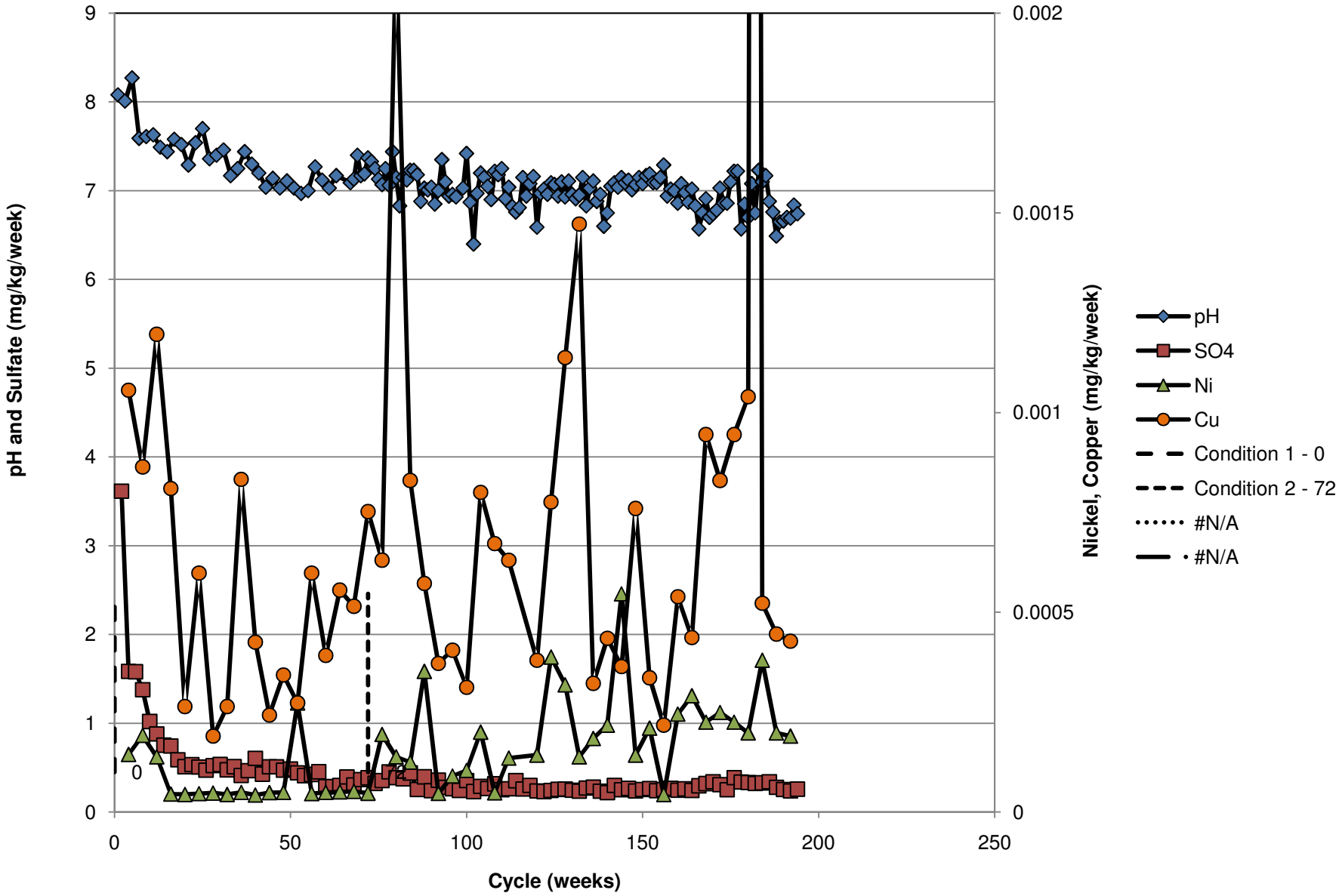
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.04%

99-318C(250-270)



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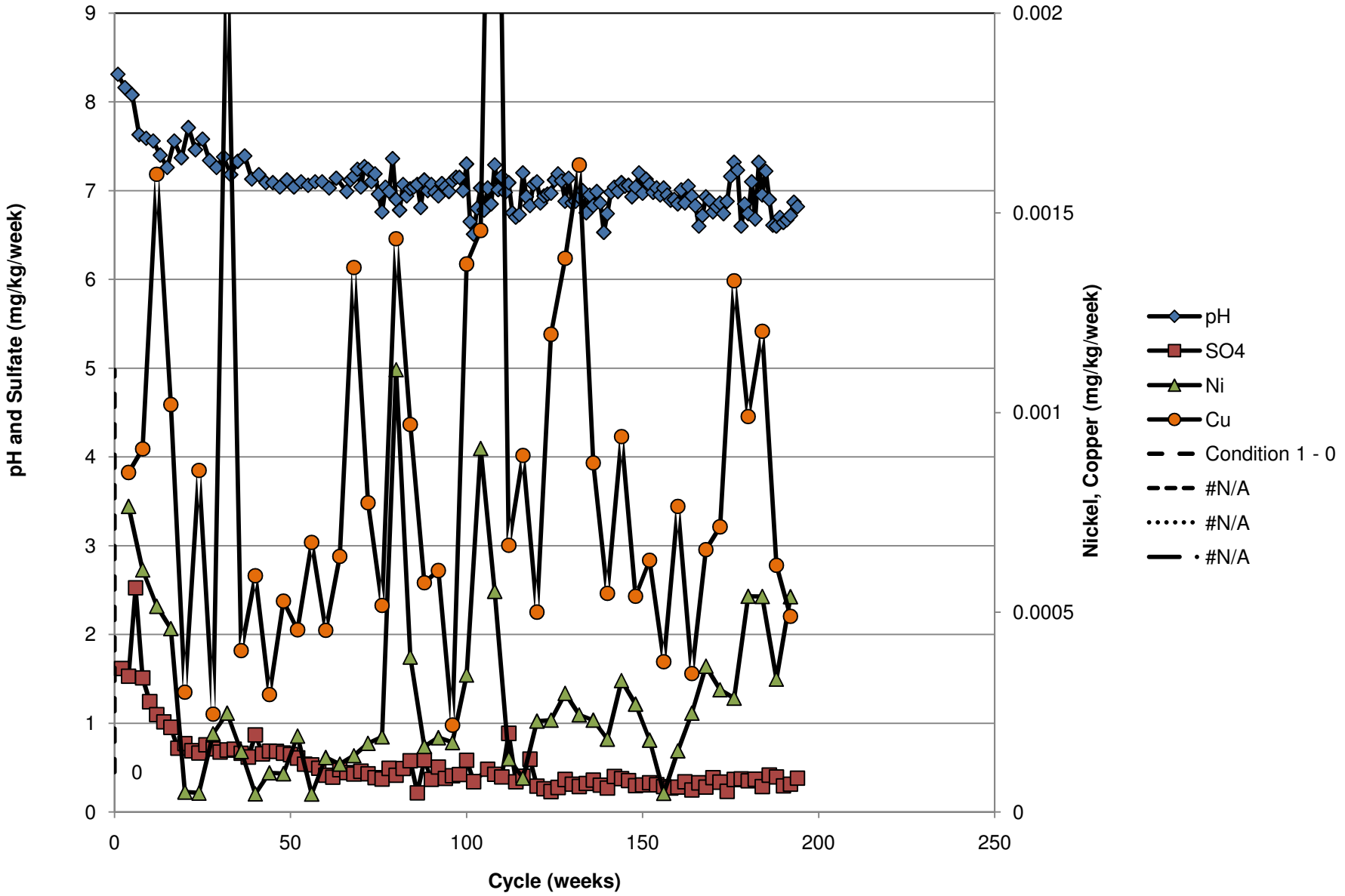
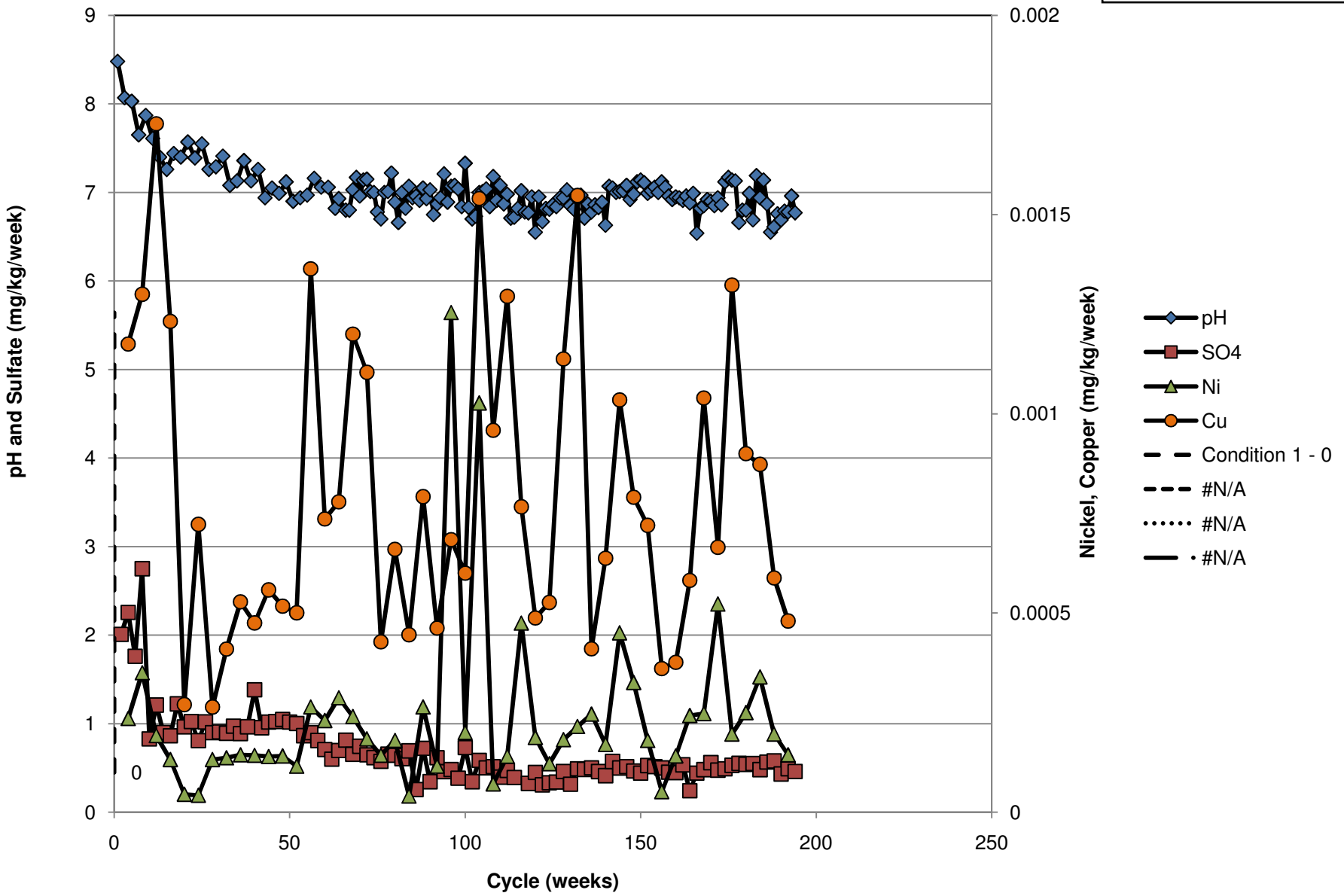
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.04%

00-373C(95-115)



Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.06%



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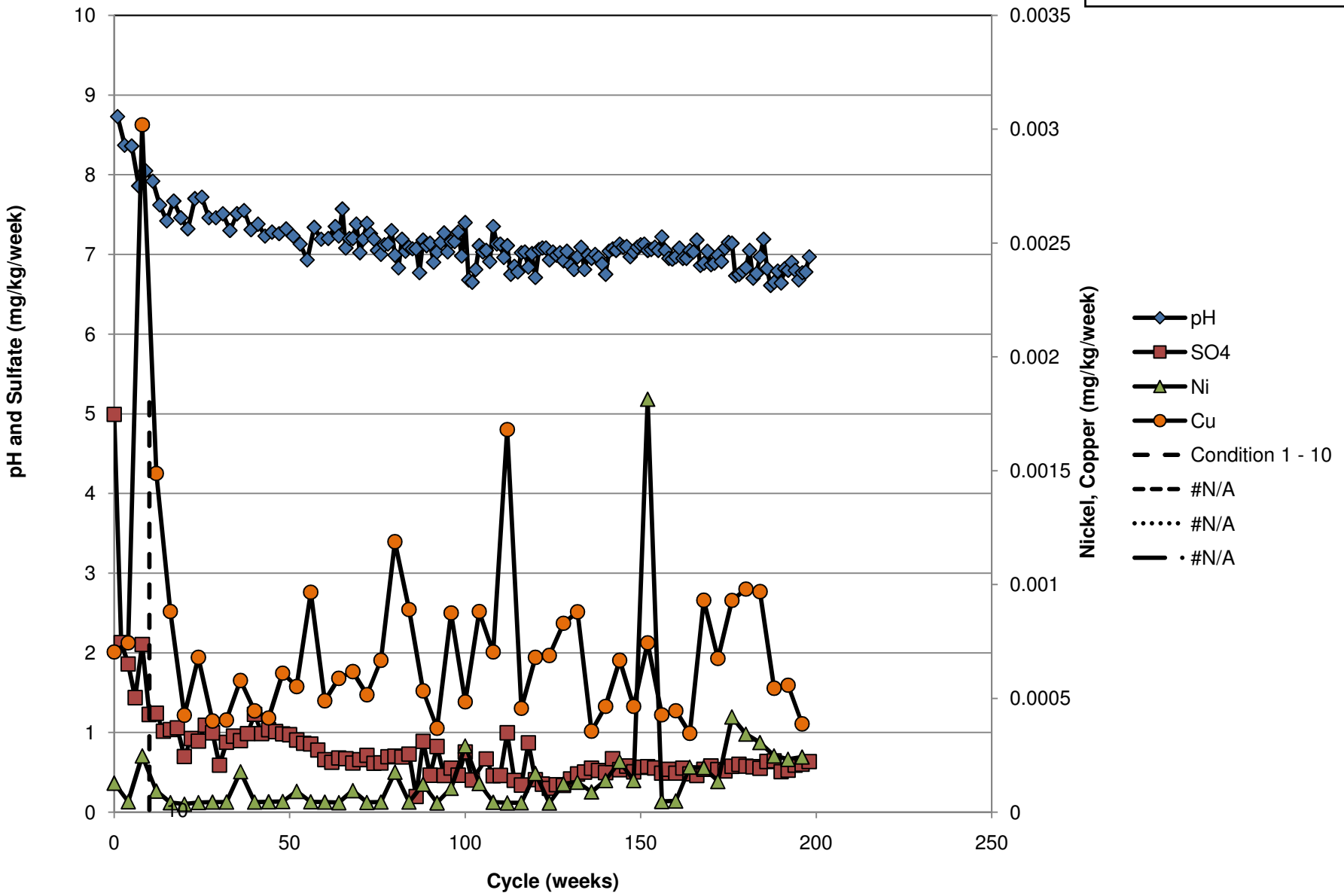
00-357C(110-130)

Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.08%



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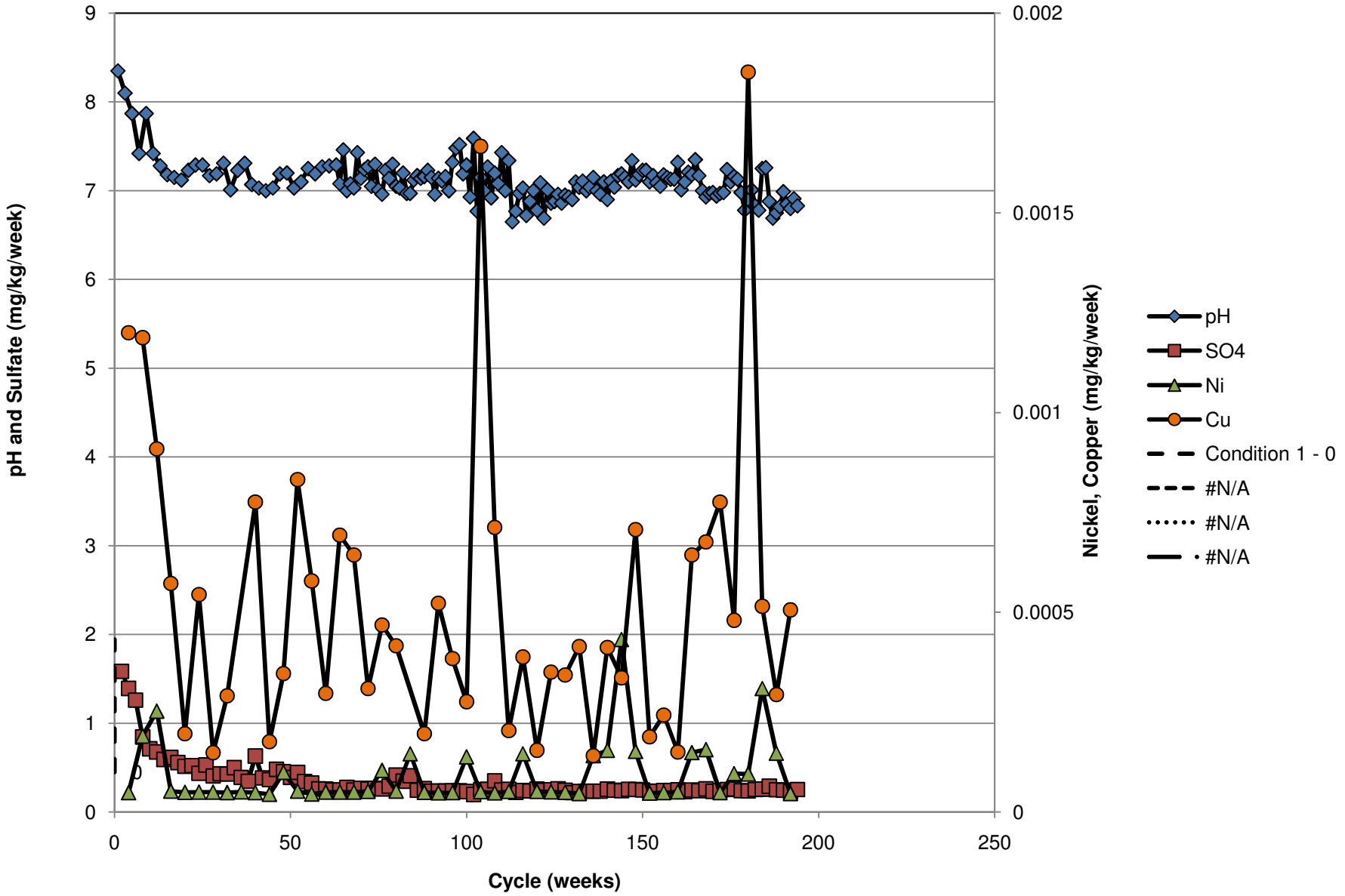
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.02%

00-366C(35-55)



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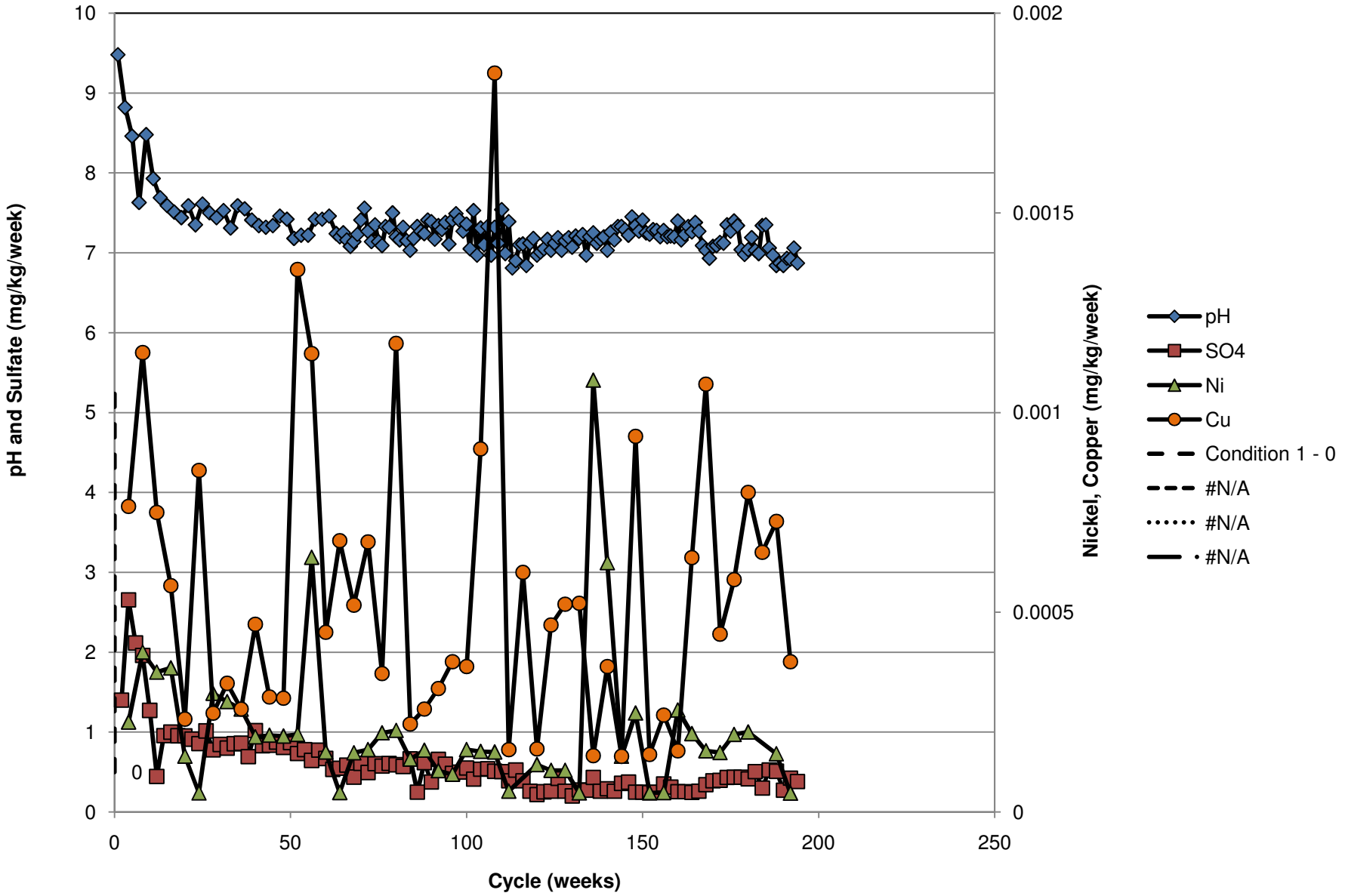
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.04%

00-334C(110-130)



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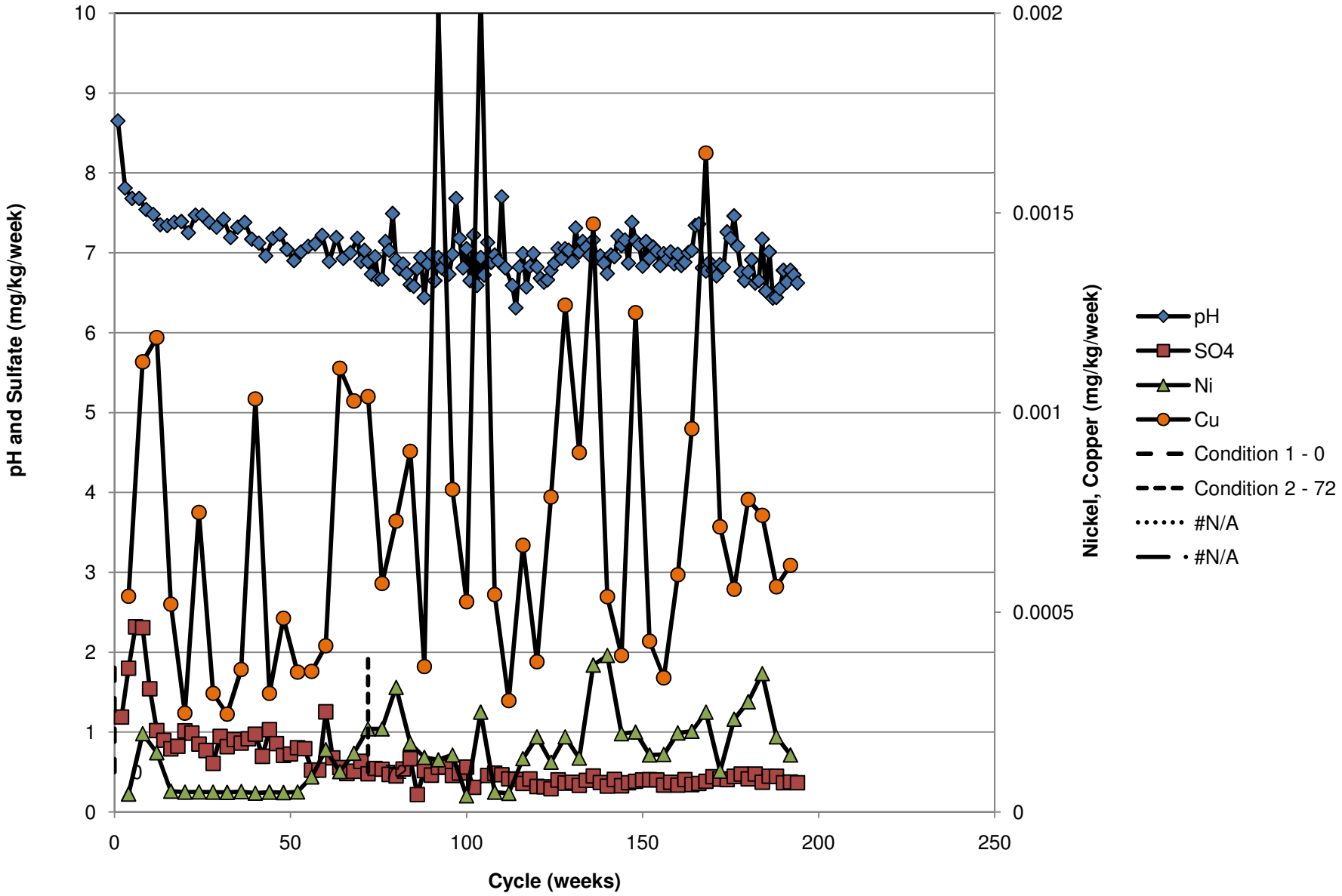
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.06%

00-347C(155-175)



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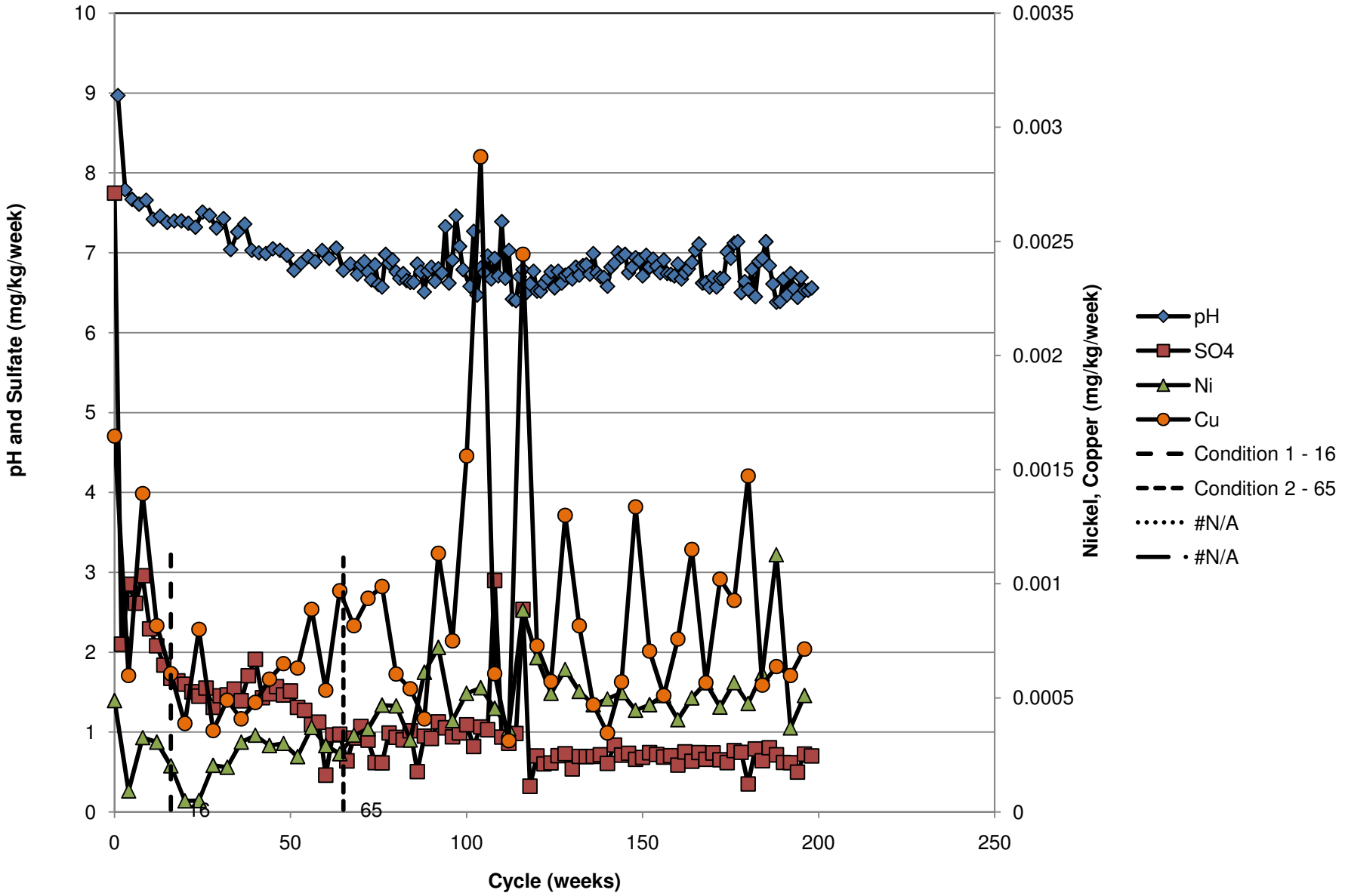
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.06%

00-347C(280-300)



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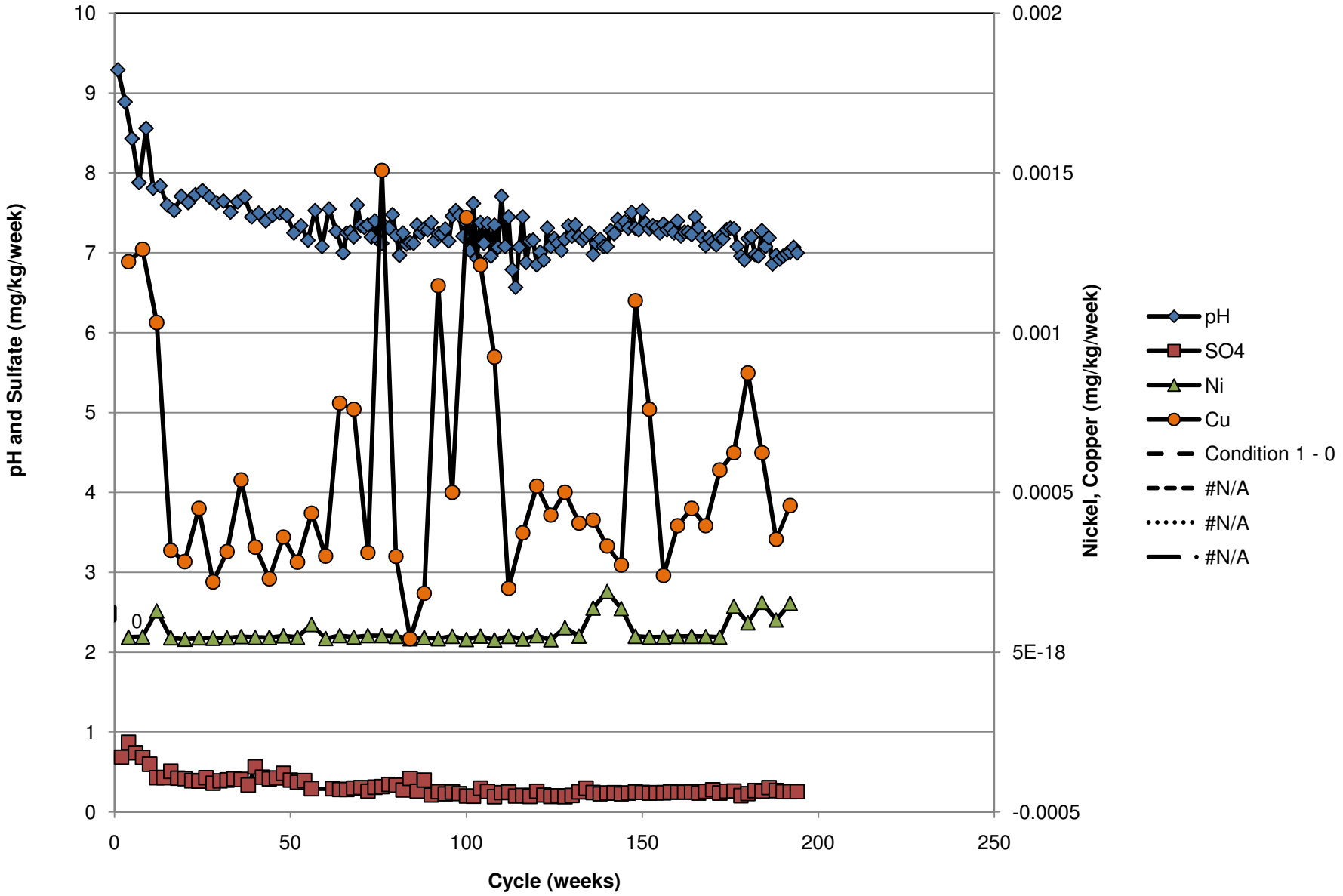
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.03%

00-367C(50-65)



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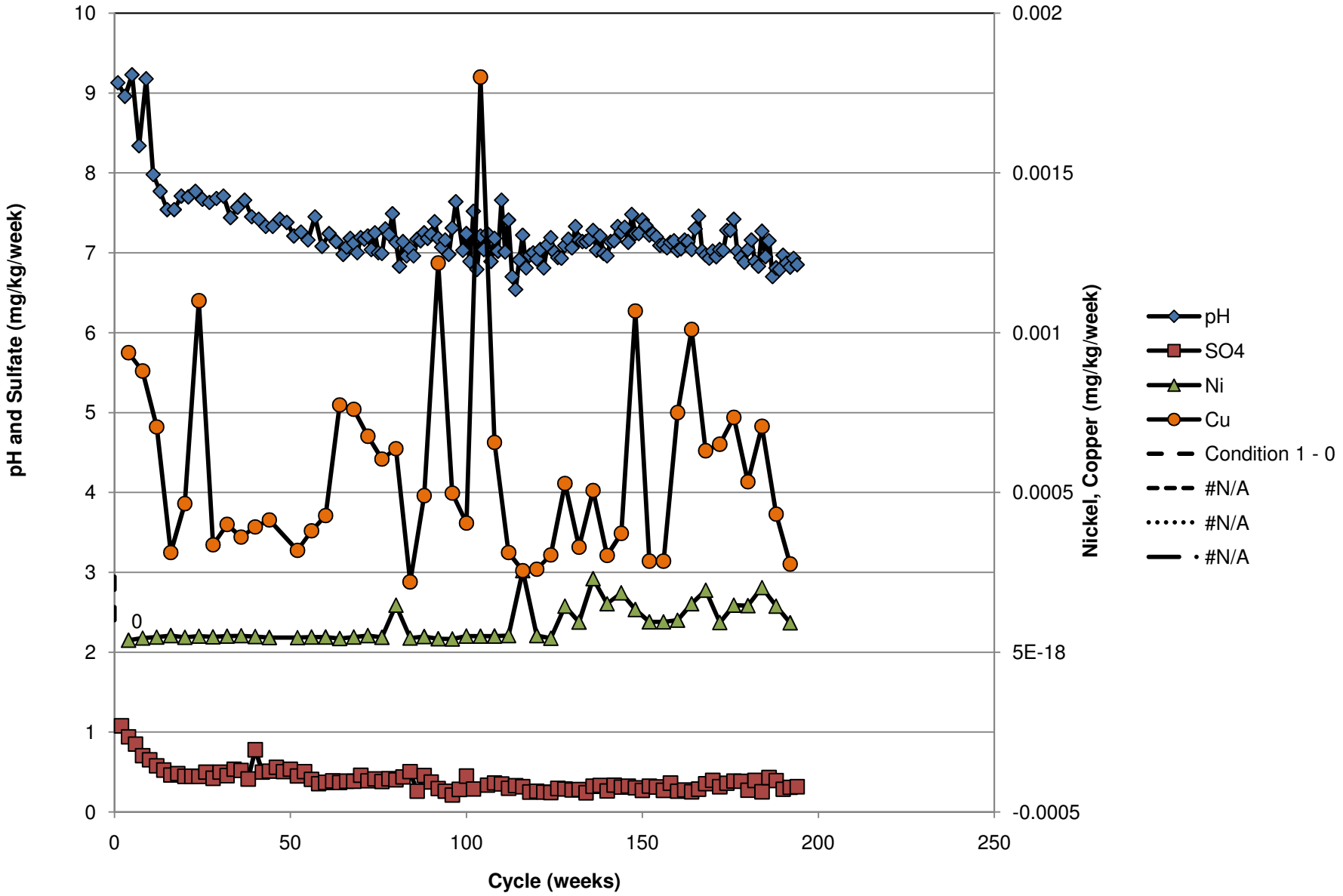
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.04%

00-367C(260-280)



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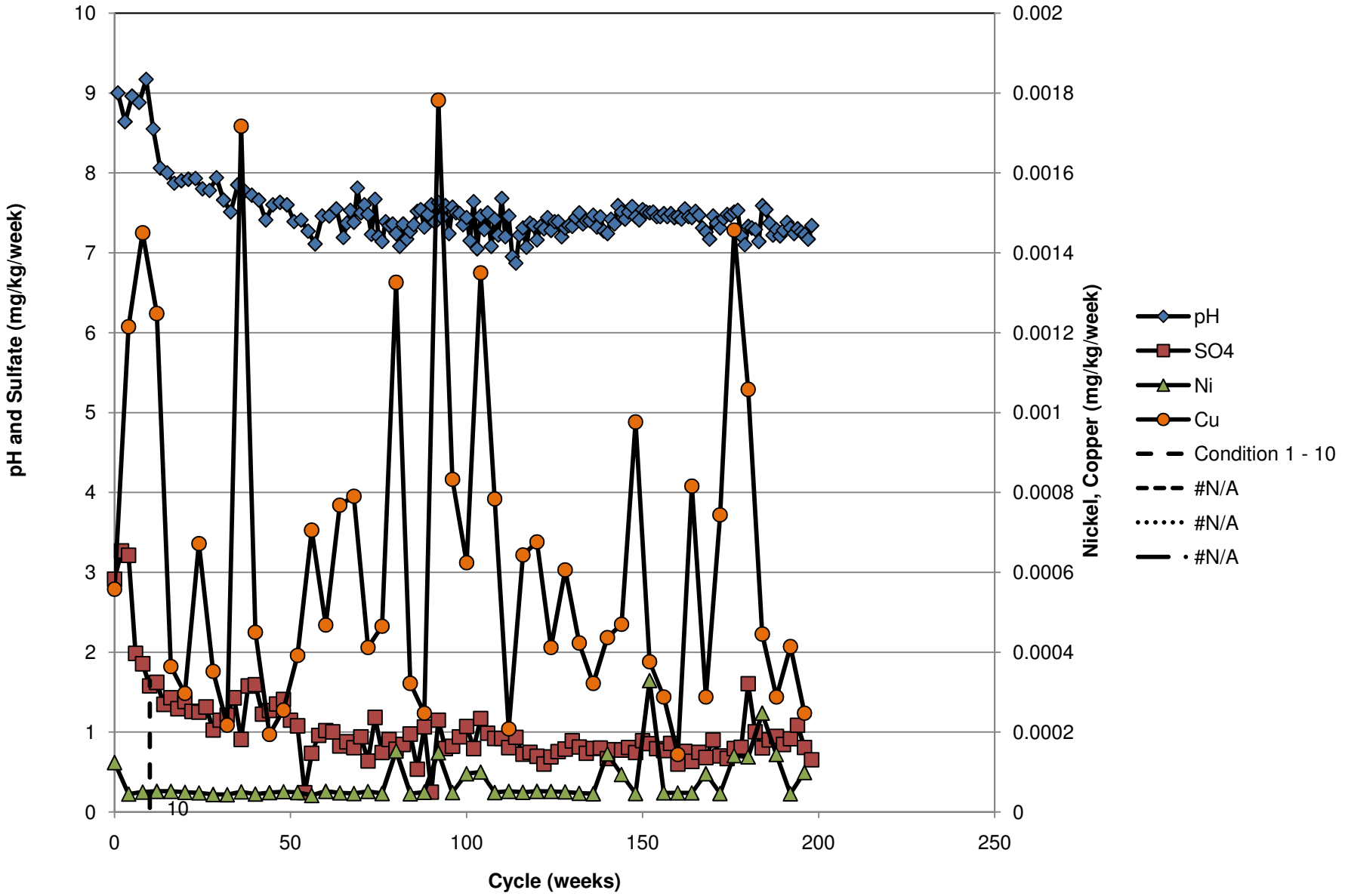
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.08%

00-370C(20-30)



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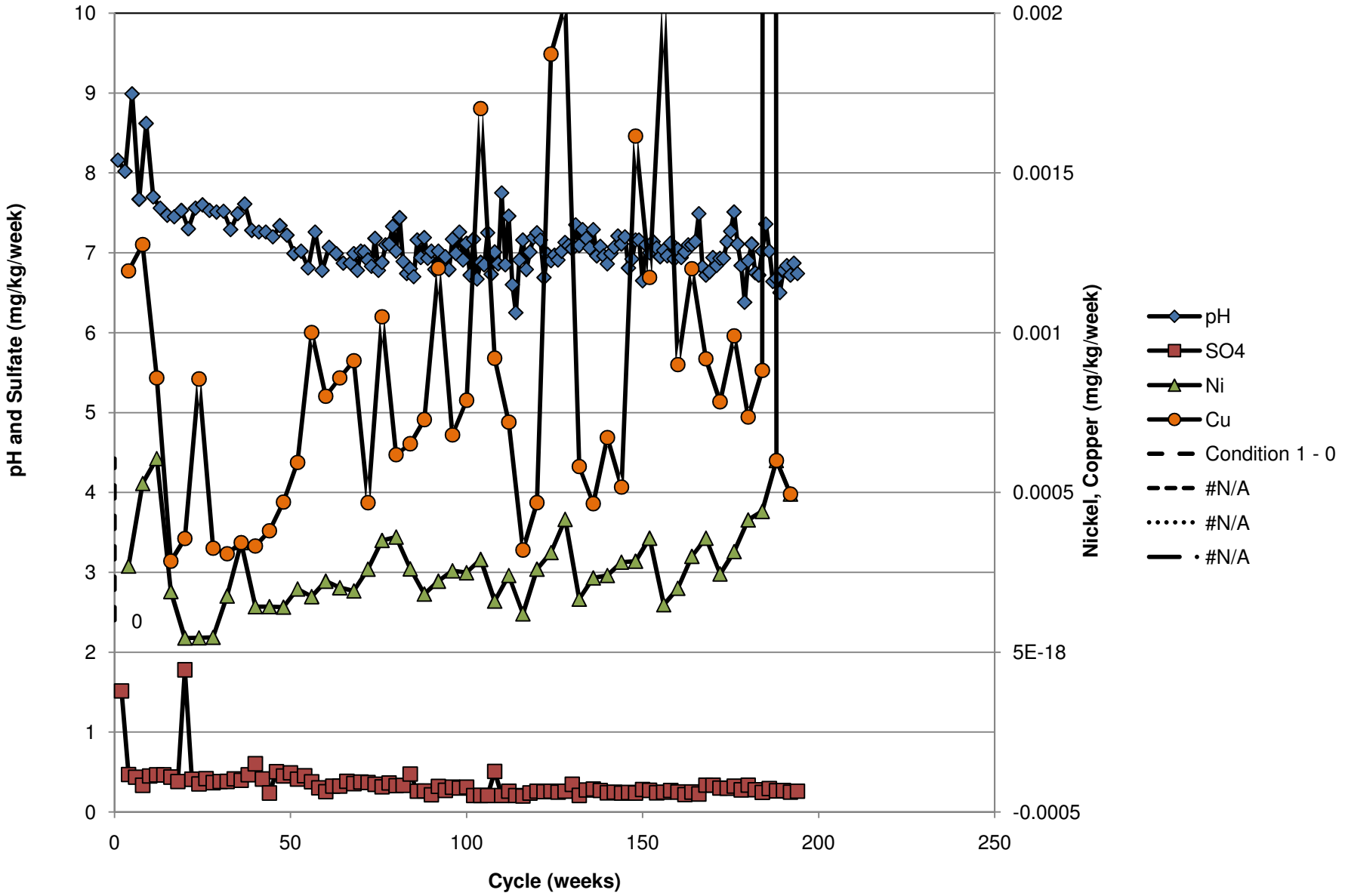
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.04%



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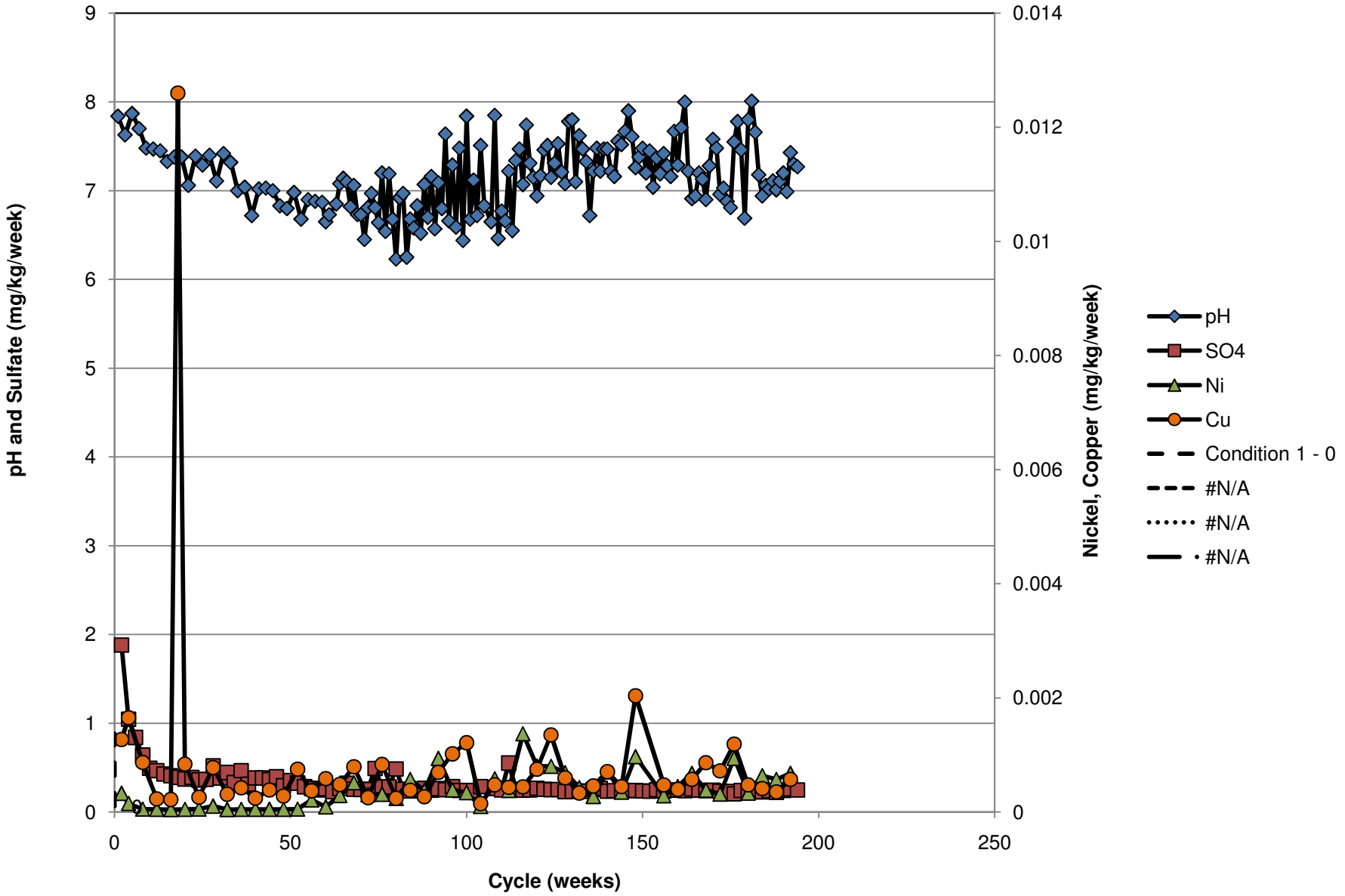
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.02%

26029(815-825)



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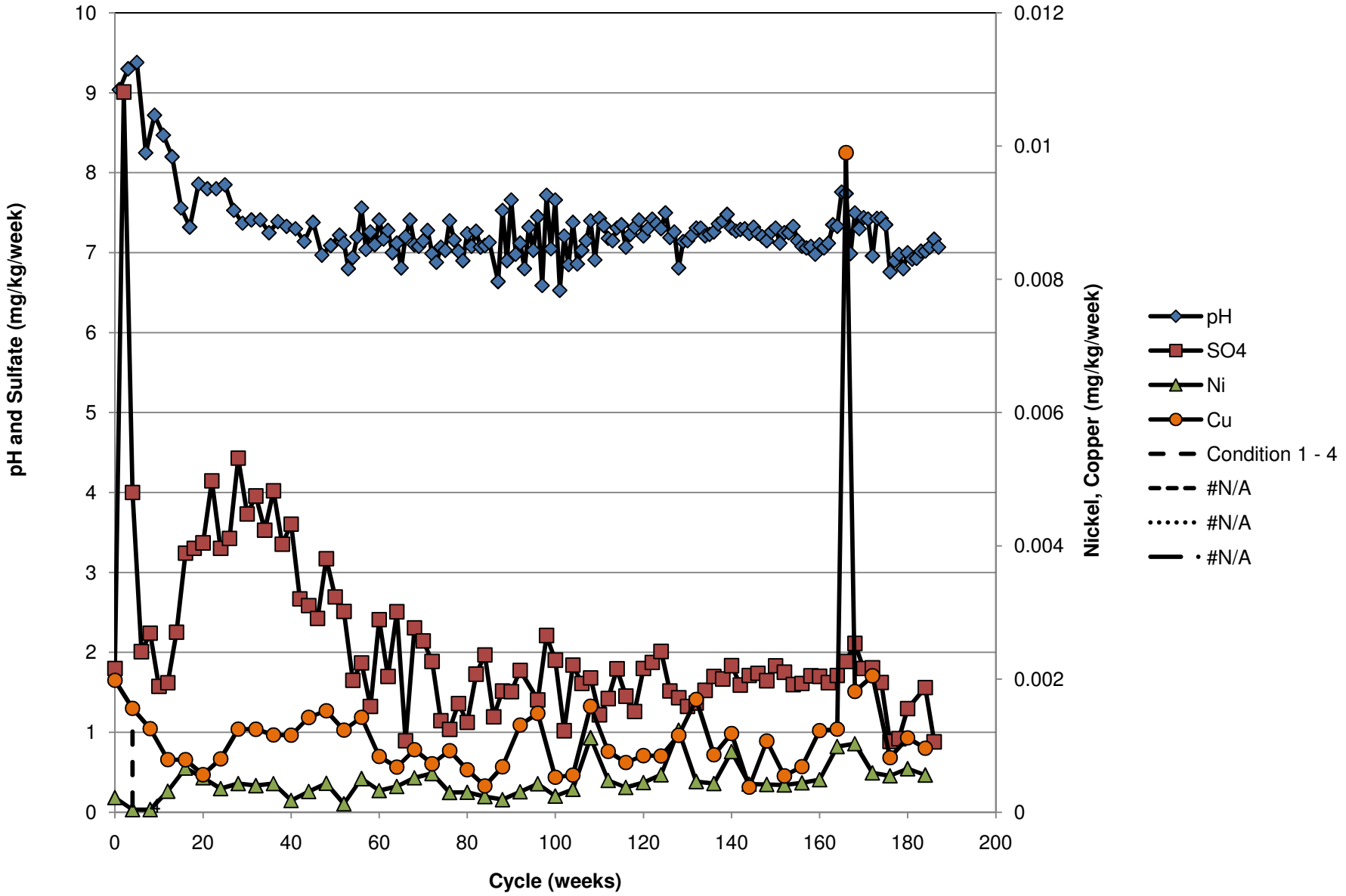
Waste Category:	1
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.08%



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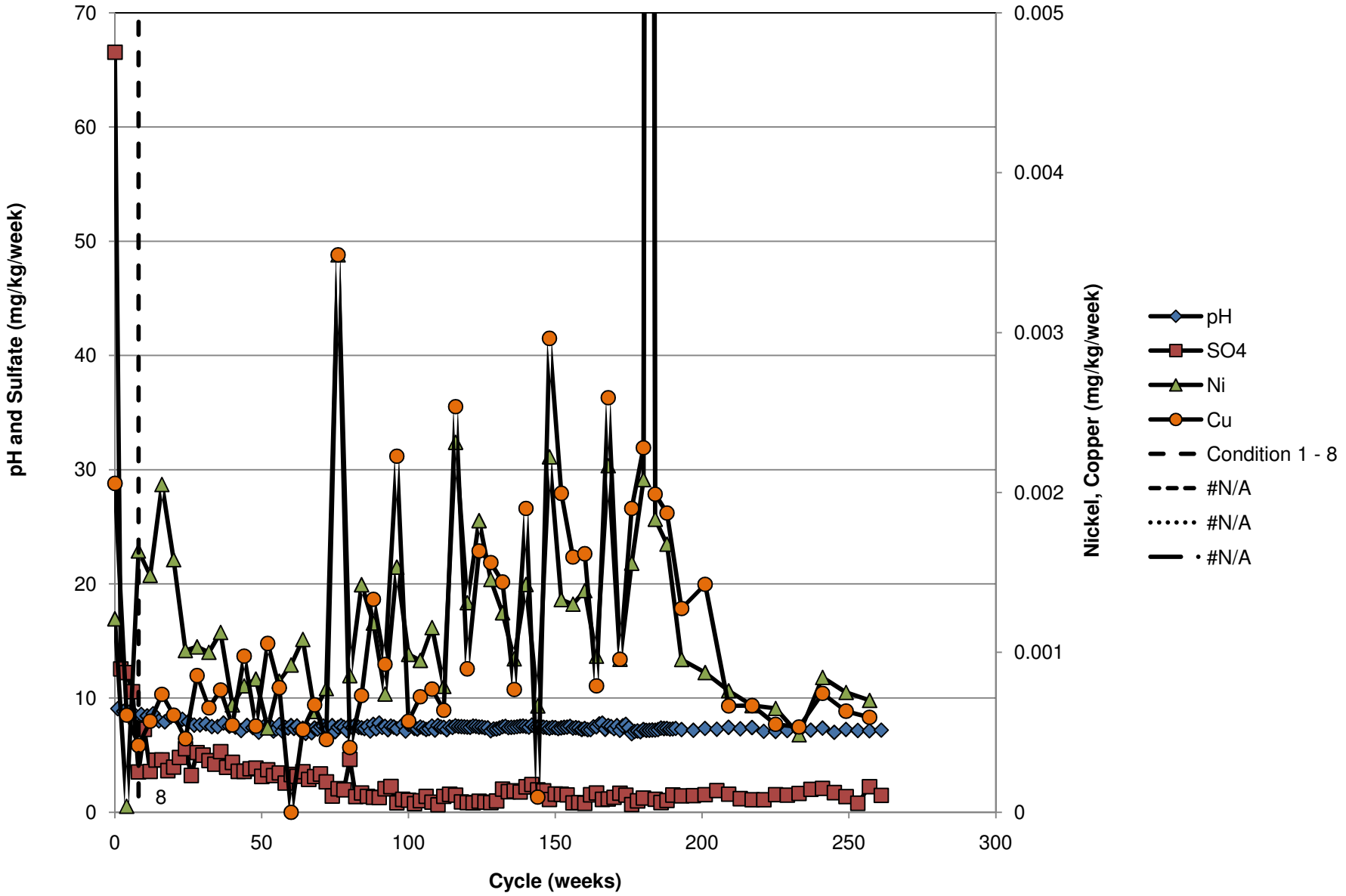
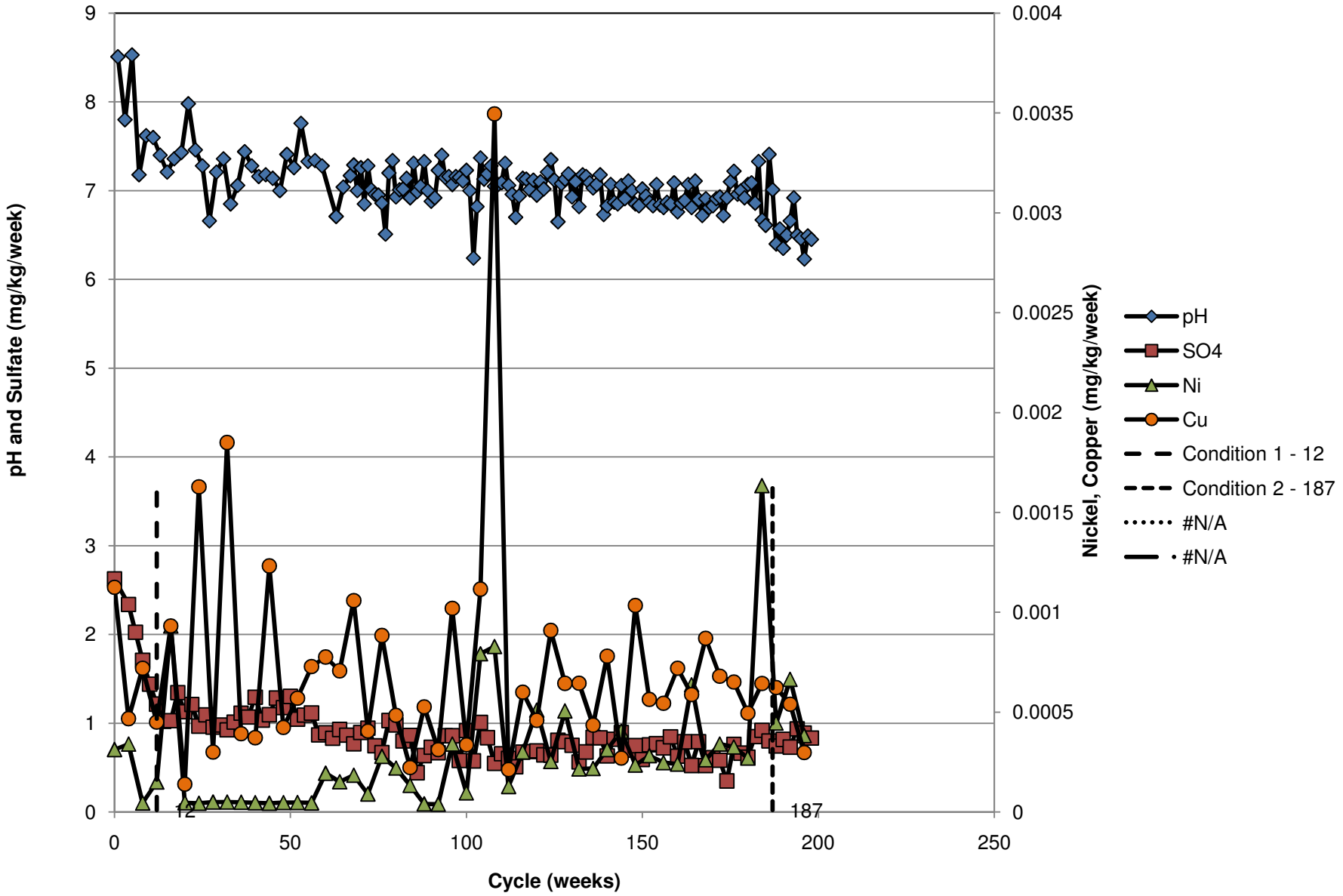
Waste Category:	1
Rock Type:	Ultramafic
Geology:	Duluth Complex
Sulfur:	0.12%

00-326C(225-235)



Waste Category:	1
Rock Type:	Ultramafic
Geology:	Duluth Complex
Sulfur:	0.08%

00-357C(335-340)



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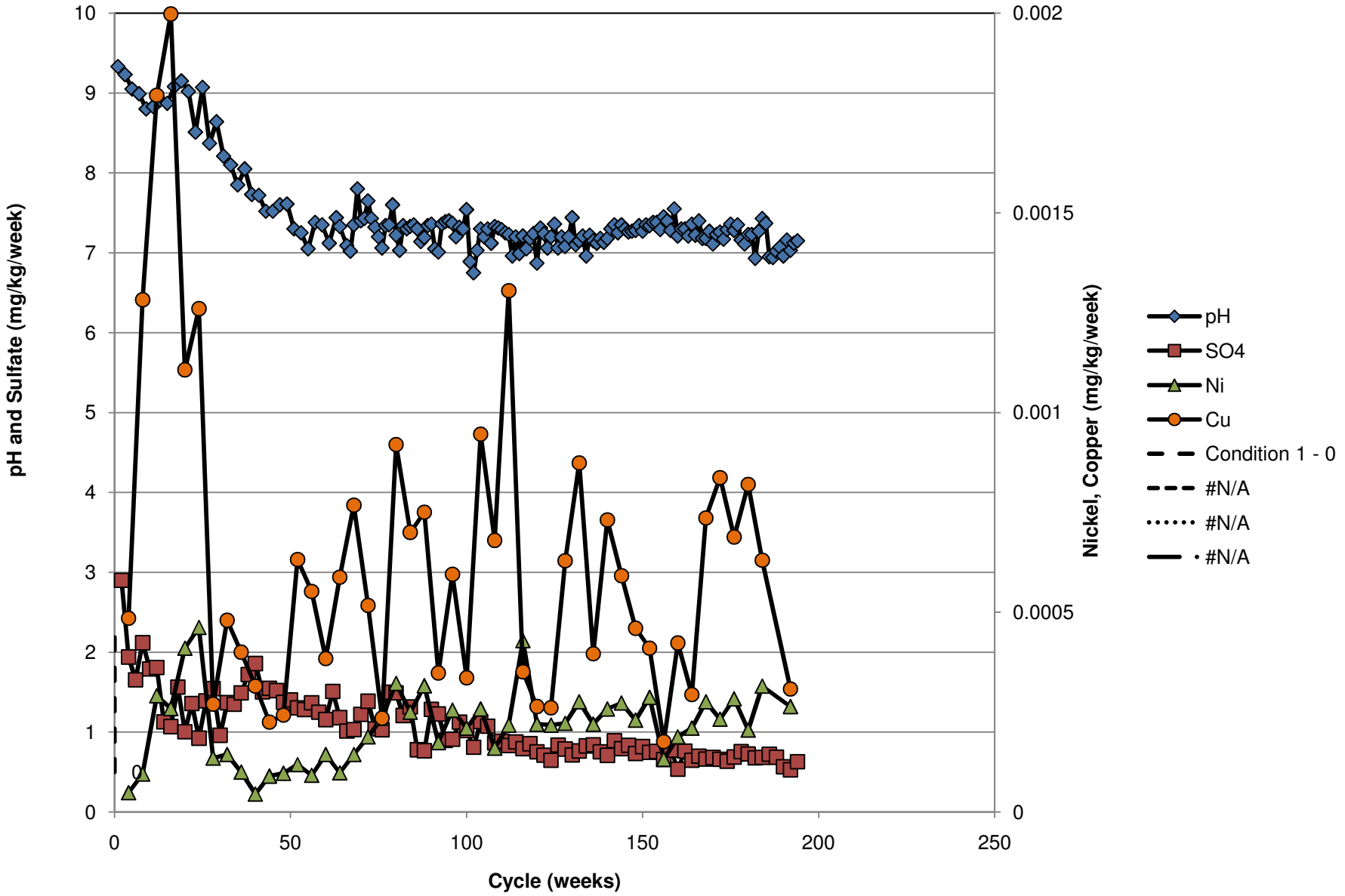
Waste Category:	1
Rock Type:	Ultramafic
Geology:	Duluth Complex
Sulfur:	0.06%

00-368C(460-465)



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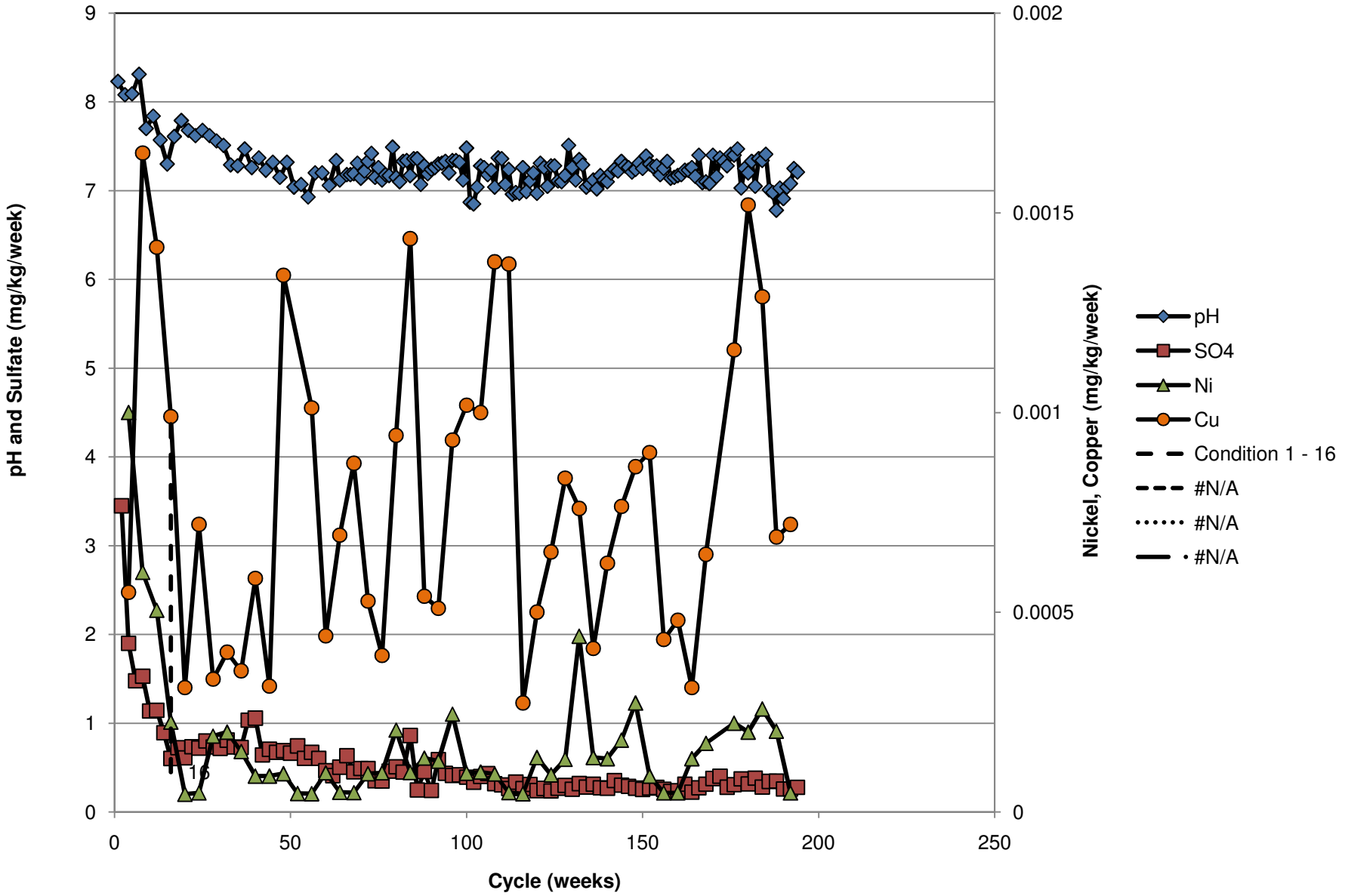
Waste Category:	1
Rock Type:	Ultramafic
Geology:	Duluth Complex
Sulfur:	0.06%

26055(940-945)



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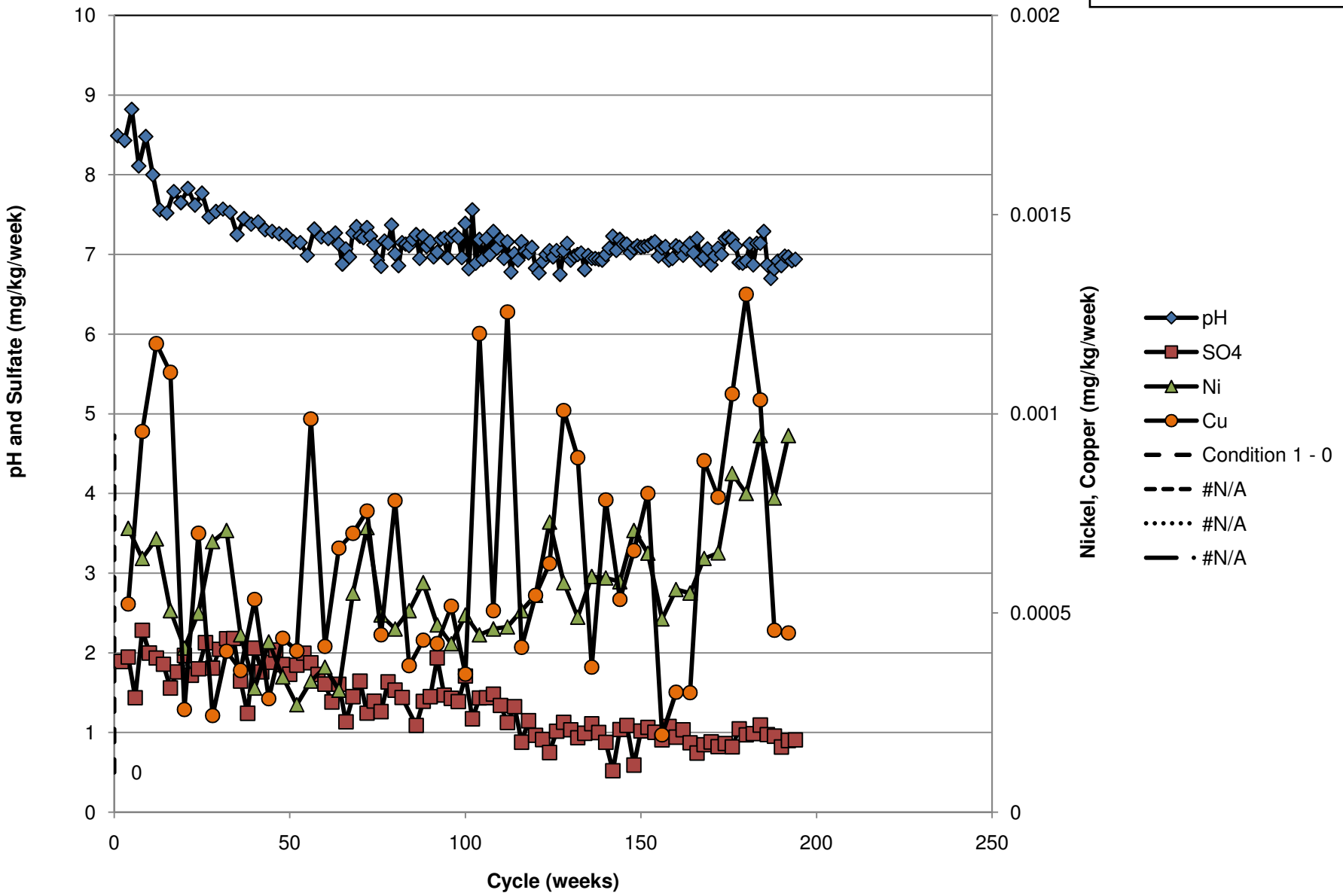
Waste Category:	1
Rock Type:	Ultramafic
Geology:	Duluth Complex
Sulfur:	0.10%

26098+00-337C



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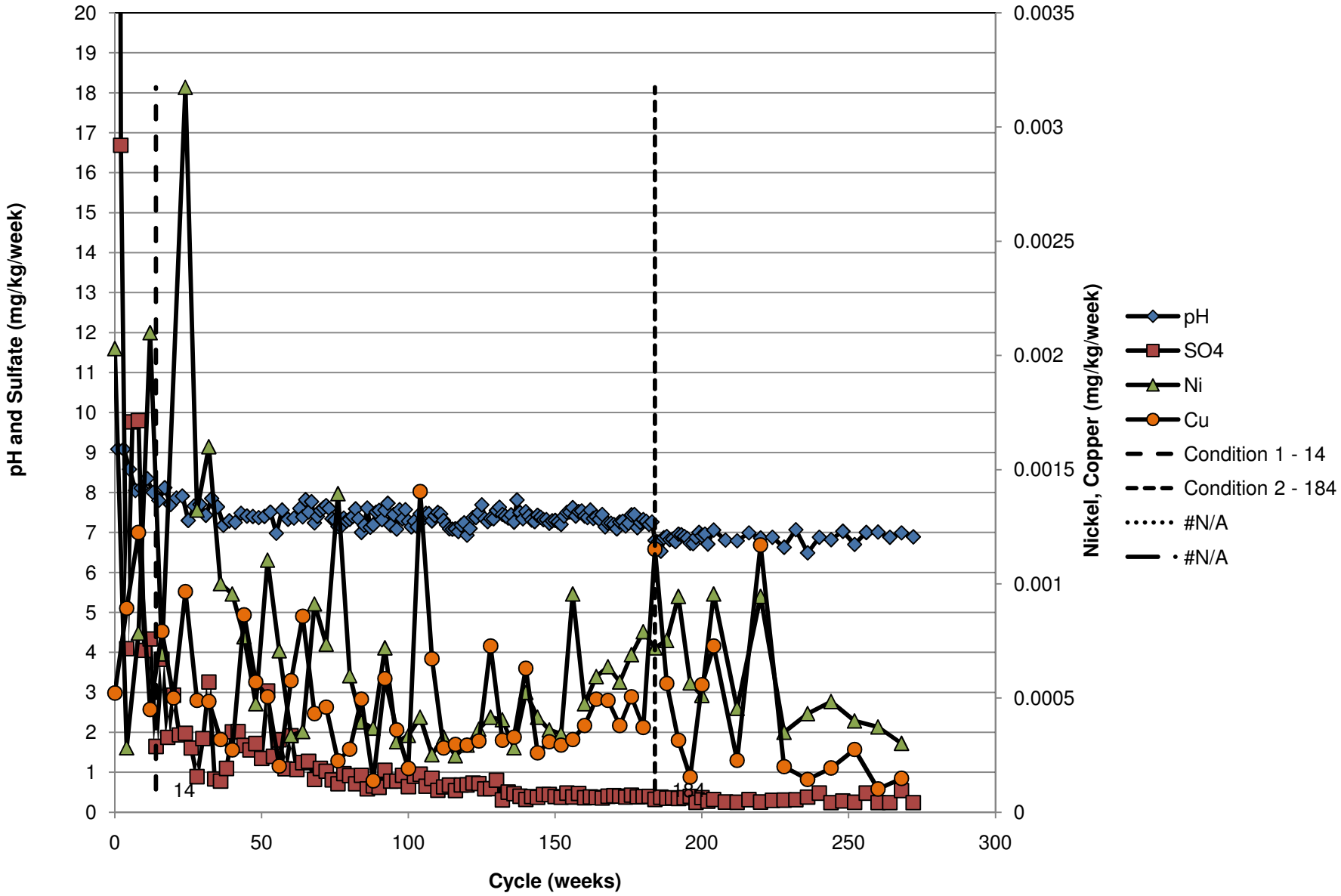
Waste Category:	1
Rock Type:	Ultramafic
Geology:	Duluth Complex
Sulfur:	0.06%

00-361C(240-245)



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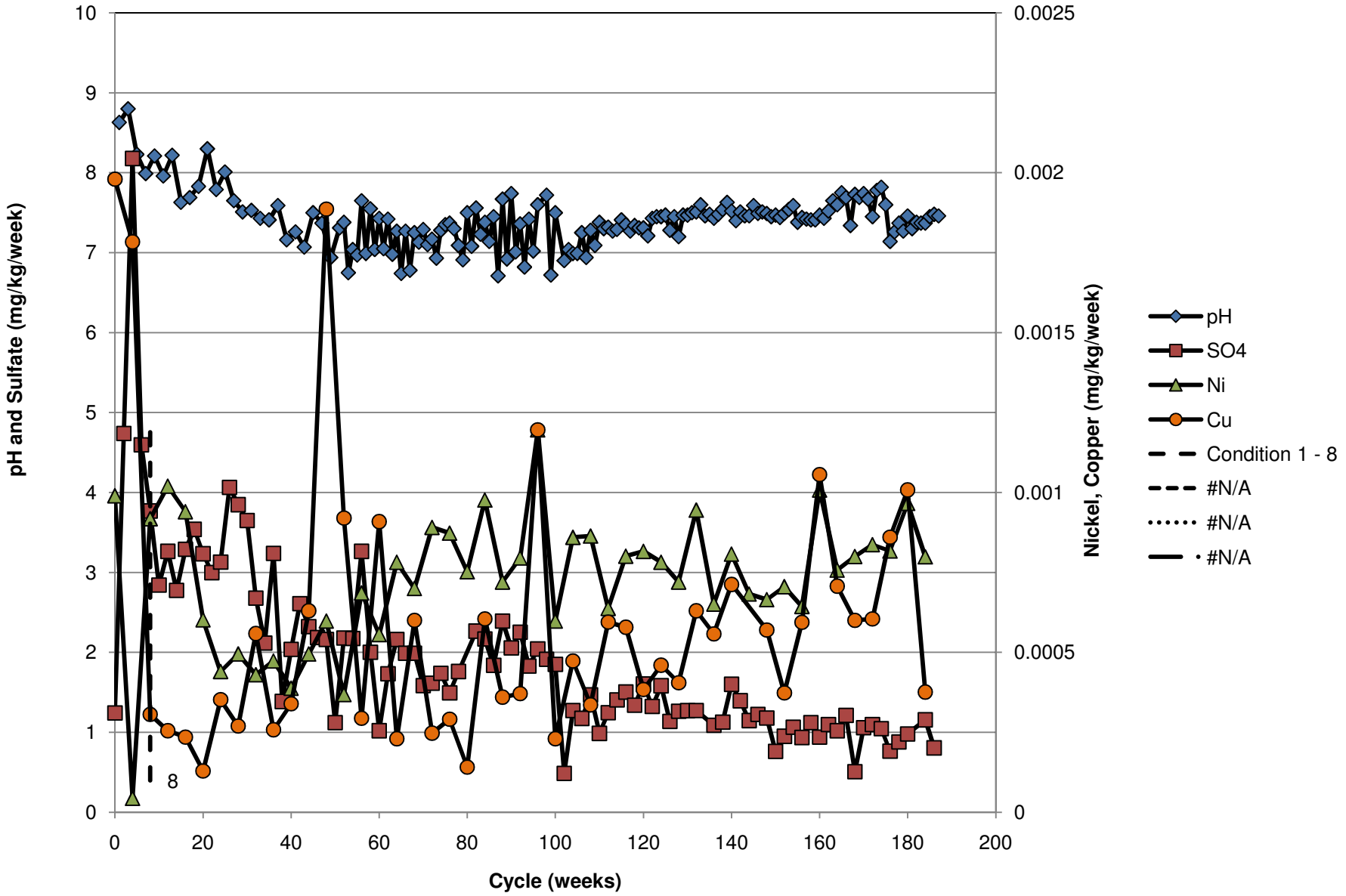
Waste Category:	1
Rock Type:	Ultramafic
Geology:	Duluth Complex
Sulfur:	0.06%



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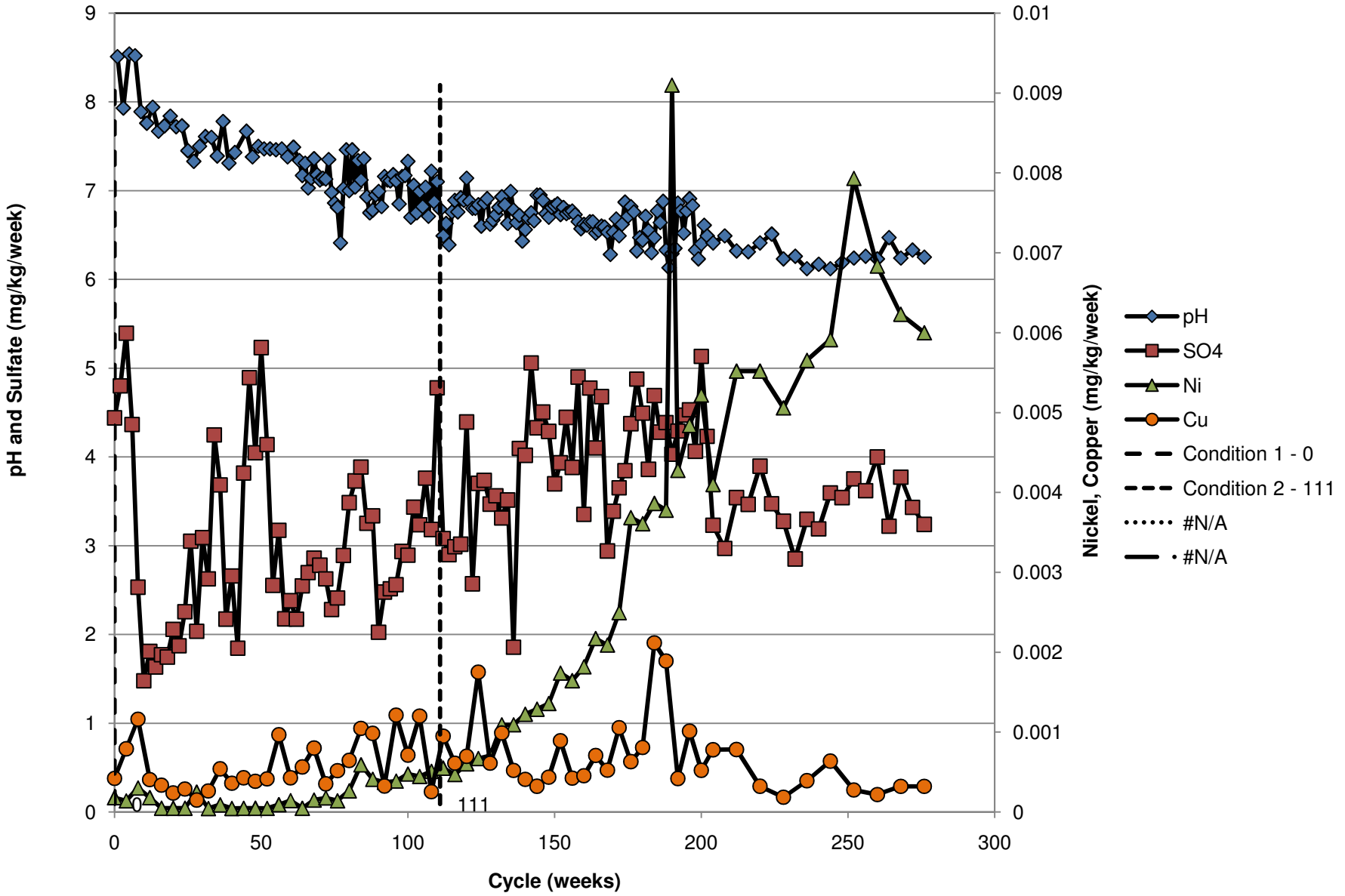
b. Category 2/3

Waste Category:	2/3
Rock Type:	Anorthositic
Geology:	Duluth Complex
Sulfur:	0.18%

00-361C(310-320)



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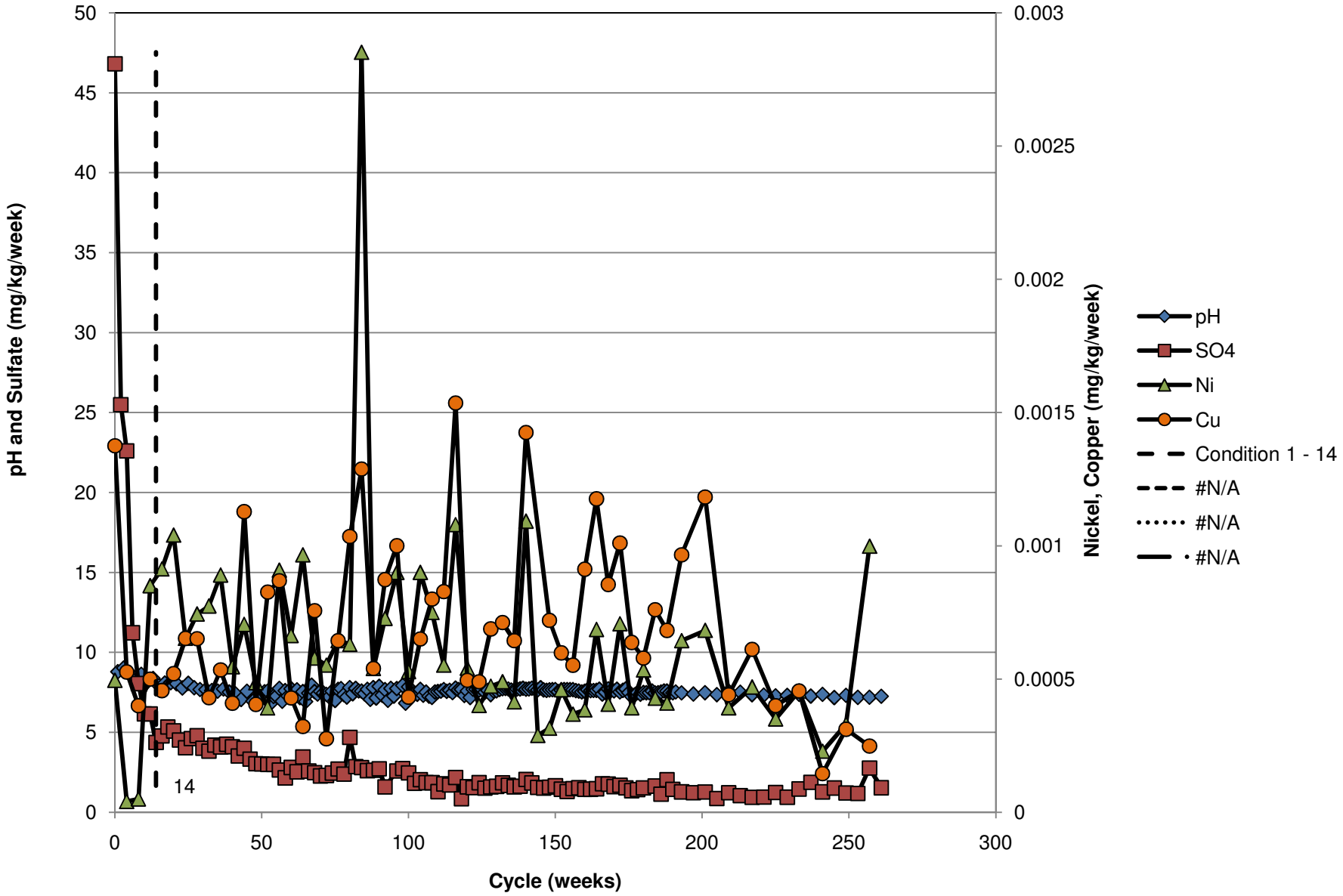
Waste Category:	2/3
Rock Type:	Anorthositic
Geology:	Duluth Complex
Sulfur:	0.18%

99-320C(400-405)



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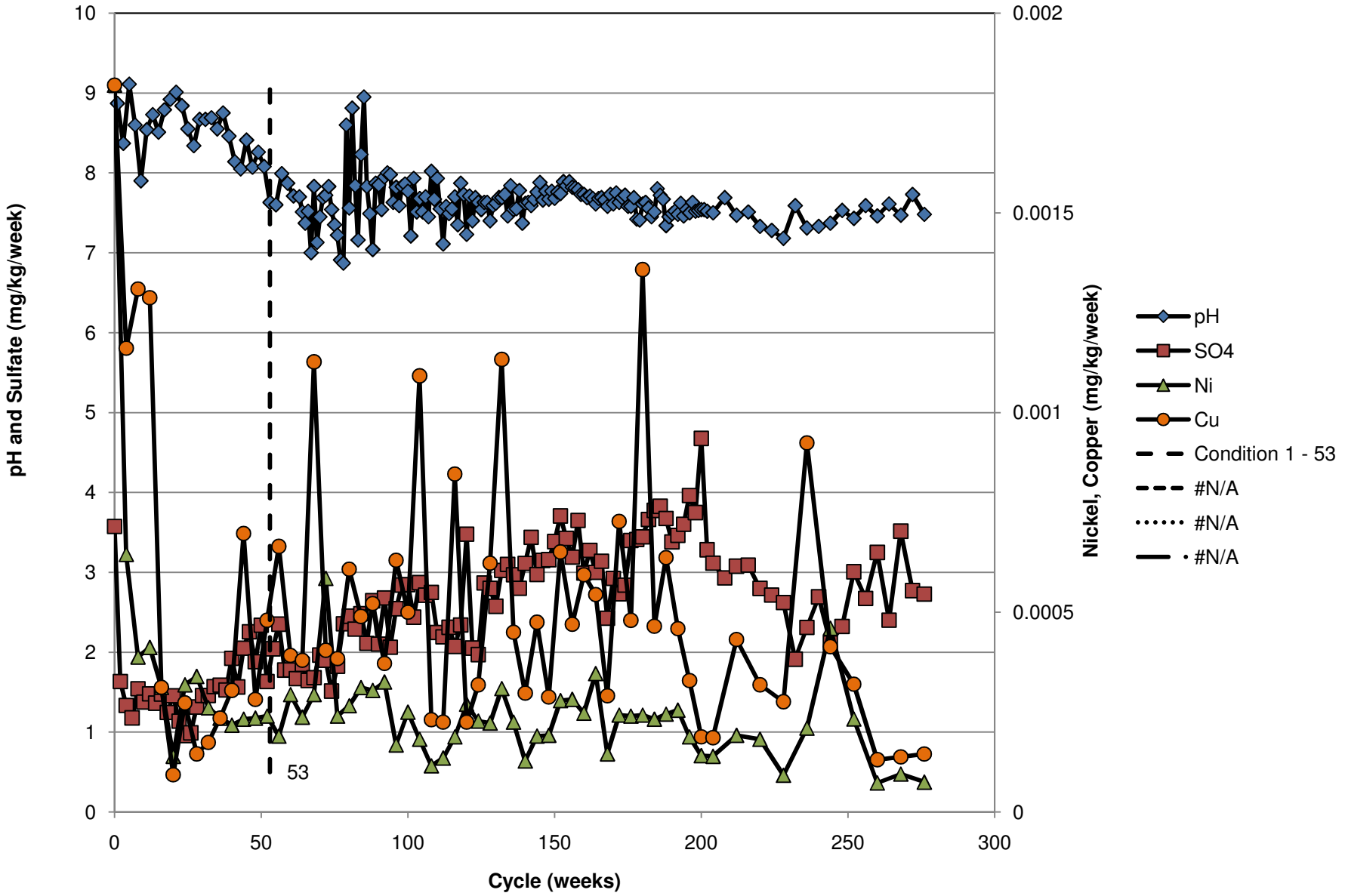
Waste Category:	2/3
Rock Type:	Sedimentary Hornfels
Geology:	Duluth Complex
Sulfur:	0.24%

26030(1047-1052)



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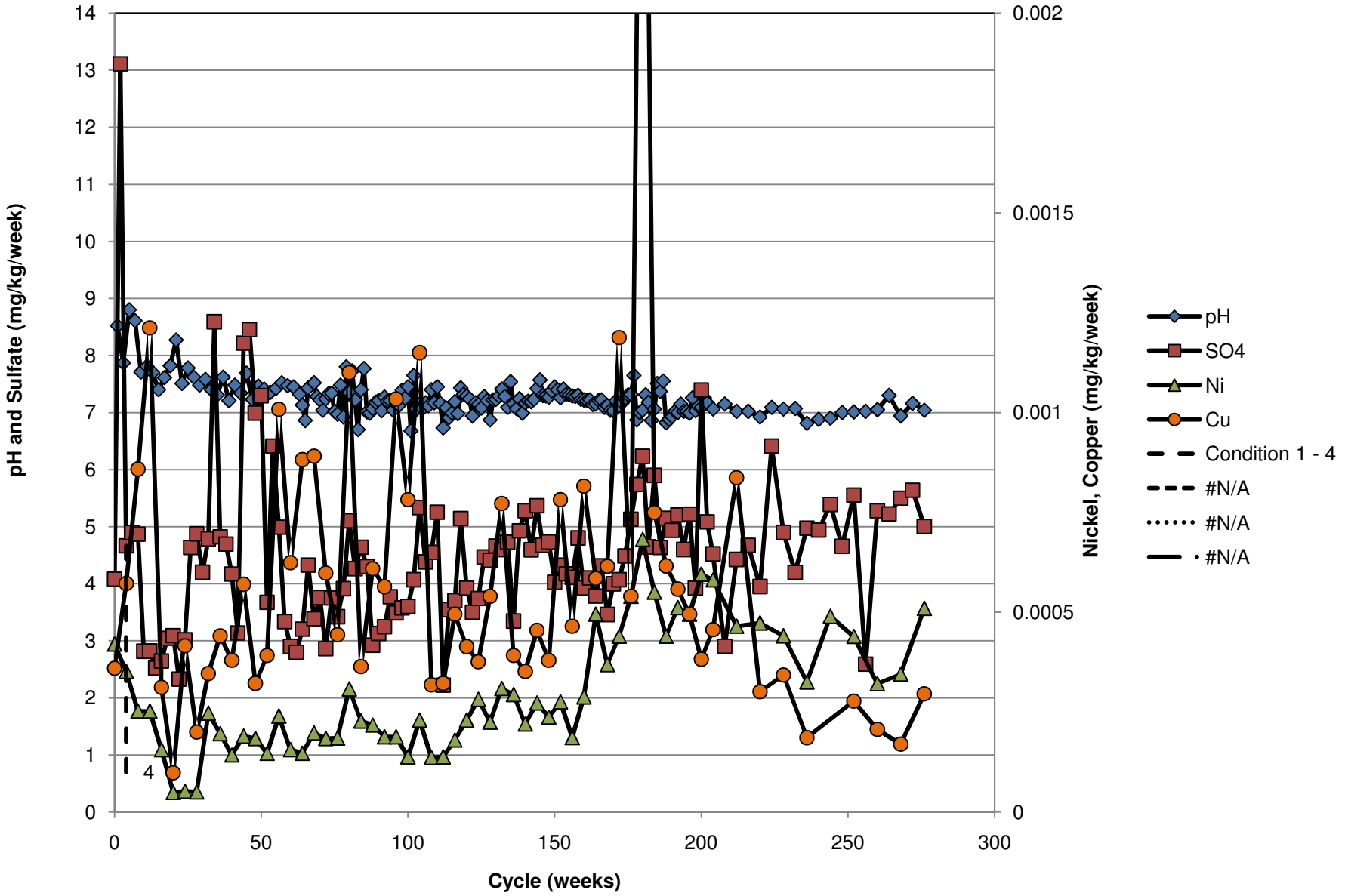
Waste Category:	2/3
Rock Type:	Sedimentary Hornfels
Geology:	Duluth Complex
Sulfur:	0.44%

26061(1218-1233)



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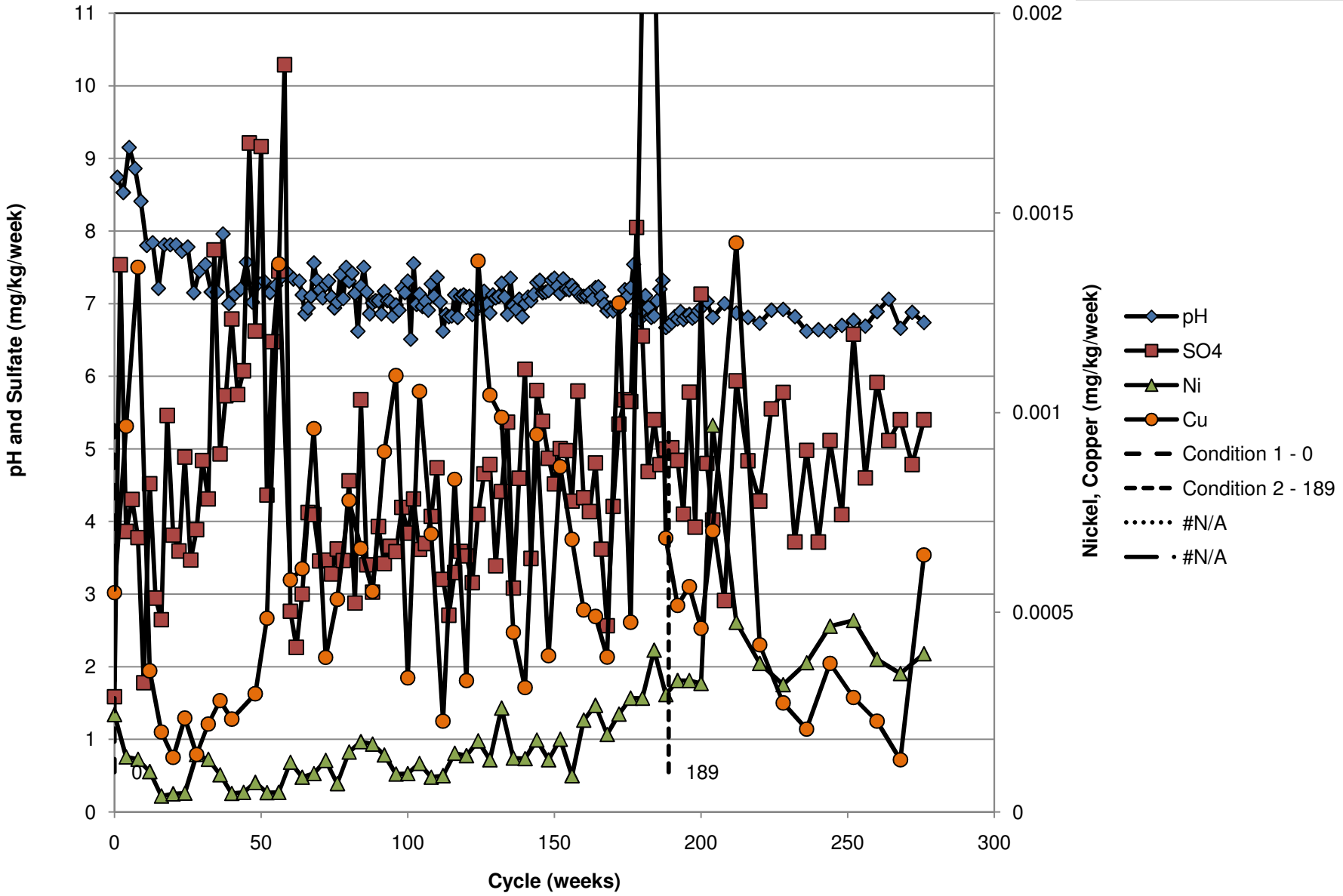
Waste Category:	2/3
Rock Type:	Sedimentary Hornfels
Geology:	Duluth Complex
Sulfur:	0.55%



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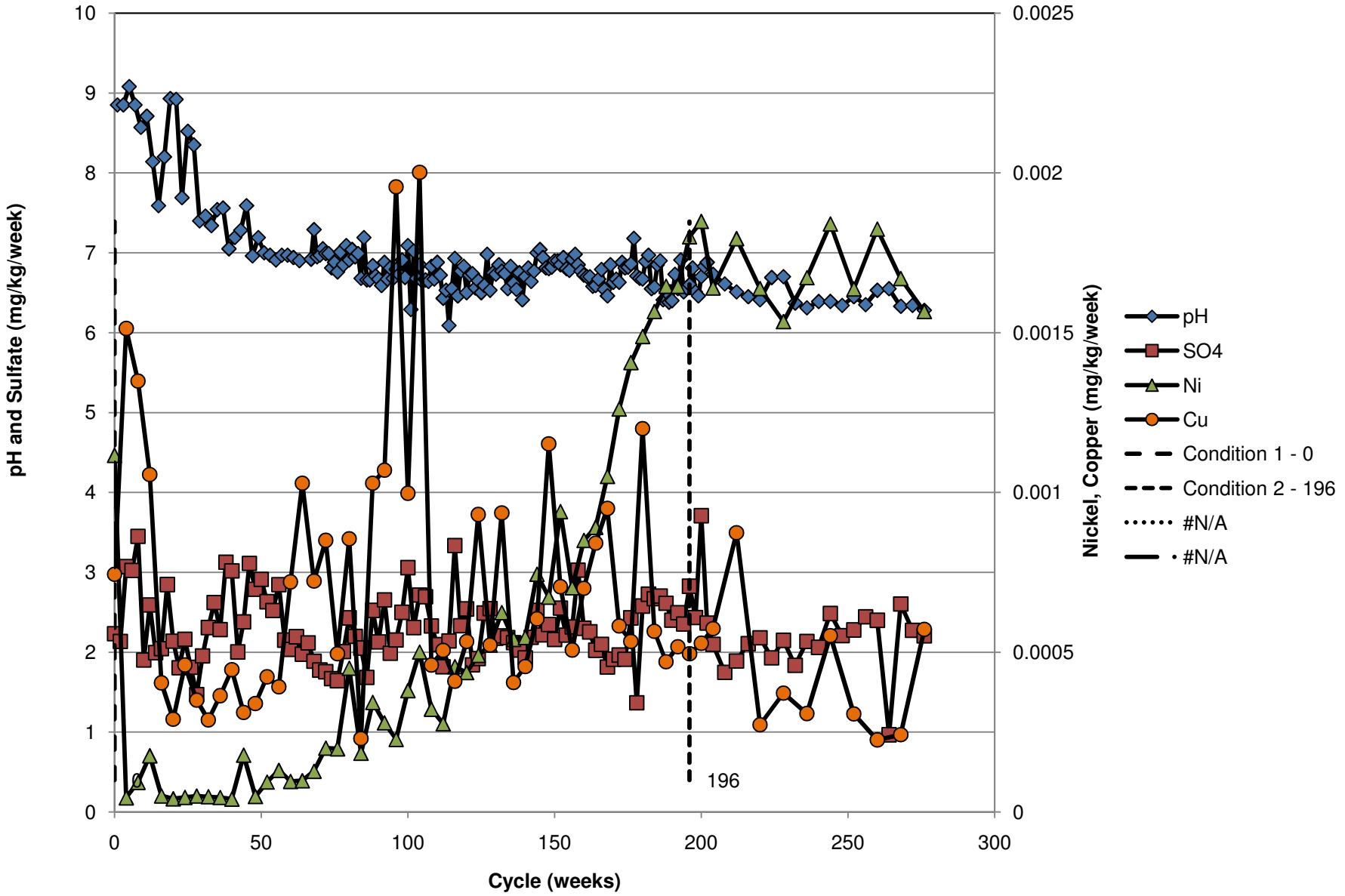
Waste Category:	2/3
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.19%

00-350C(580-600)



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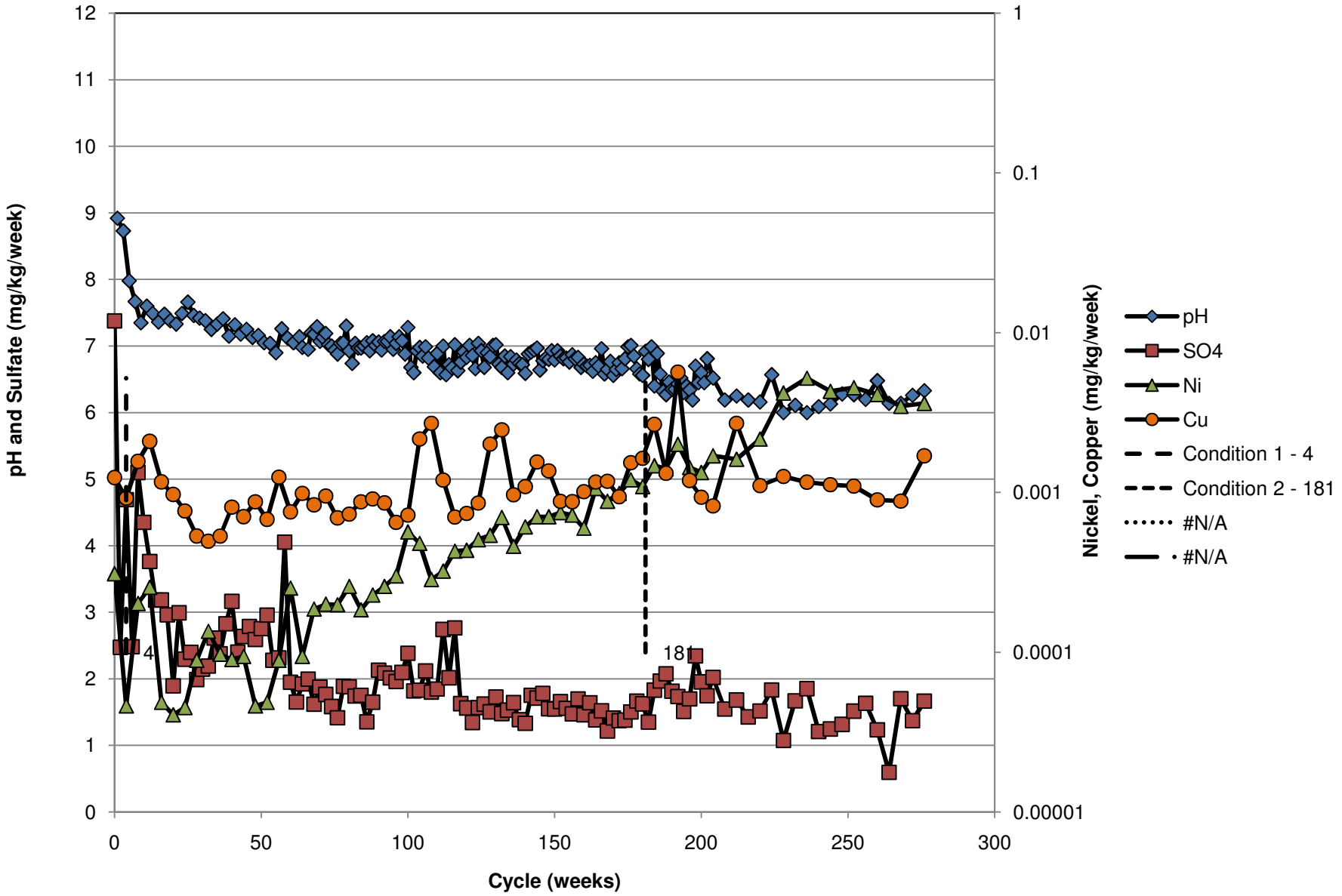
Waste Category:	2/3
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.18%

00-369C(335-345)



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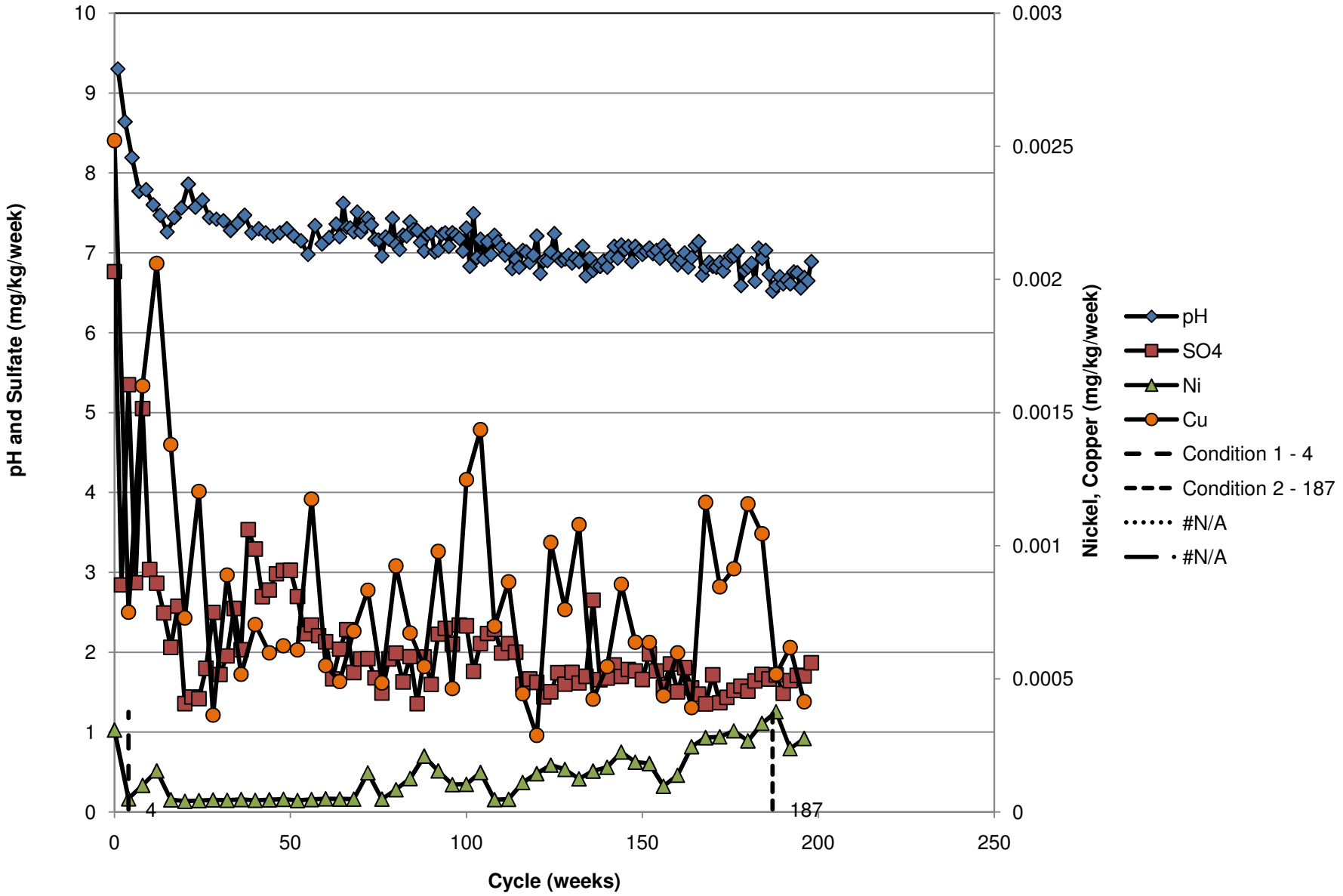
Waste Category:	2/3
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.25%

00-369C(305-325)



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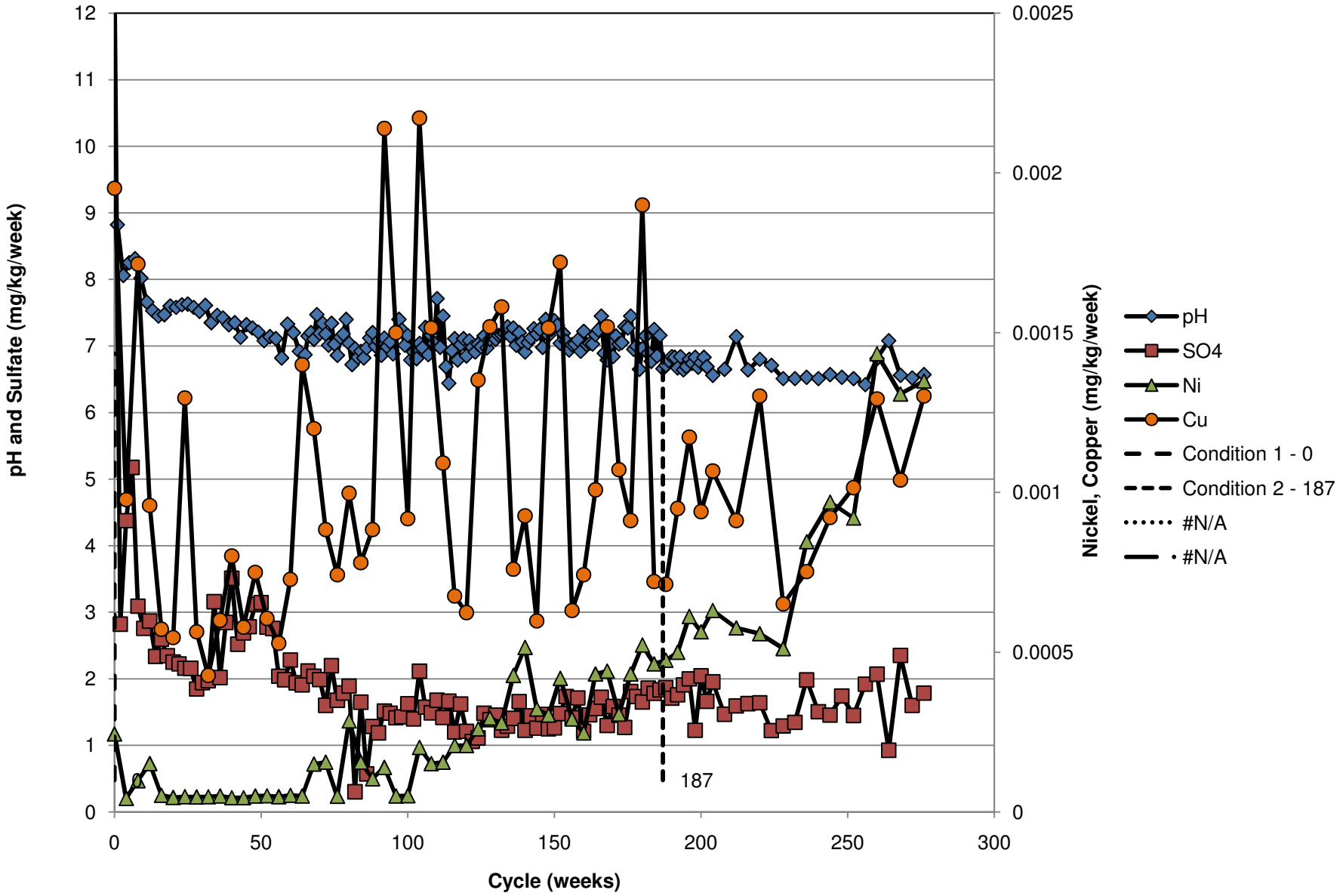
Waste Category:	2/3
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.21%

00-369C(20-30)



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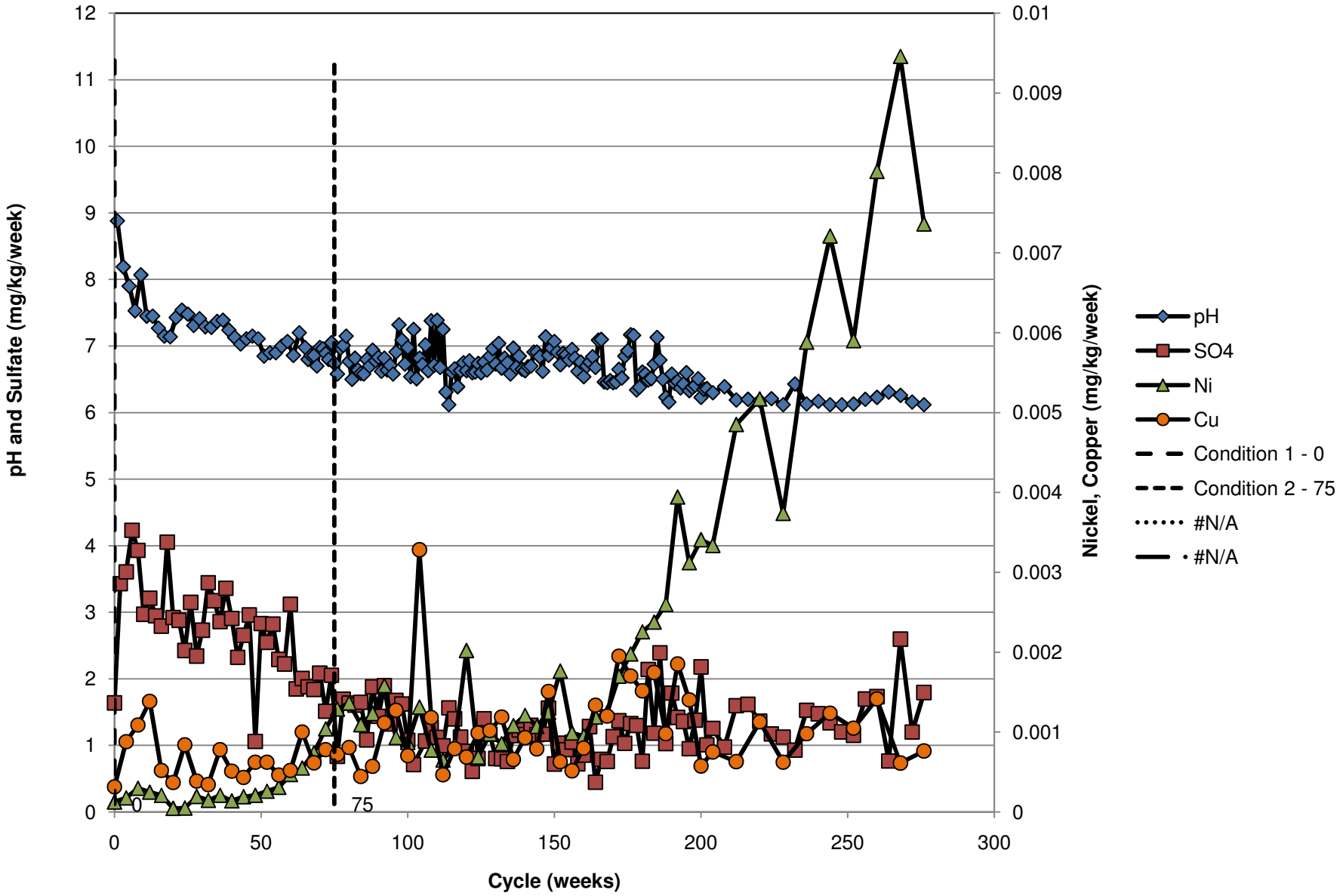
Waste Category:	2/3
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.14%

00-326C(60-70)



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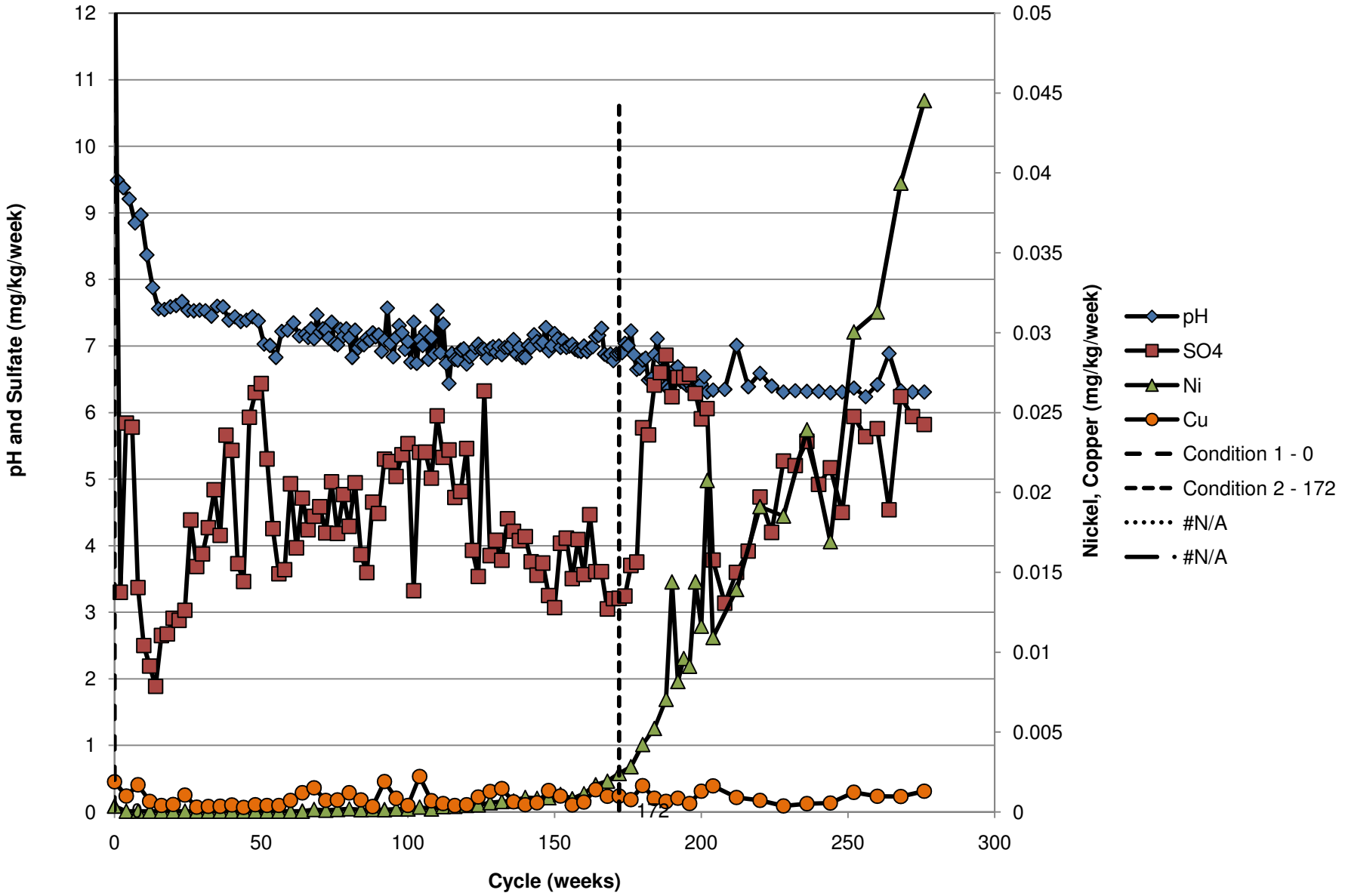
Waste Category:	2/3
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.51%

00-367C(170-175)



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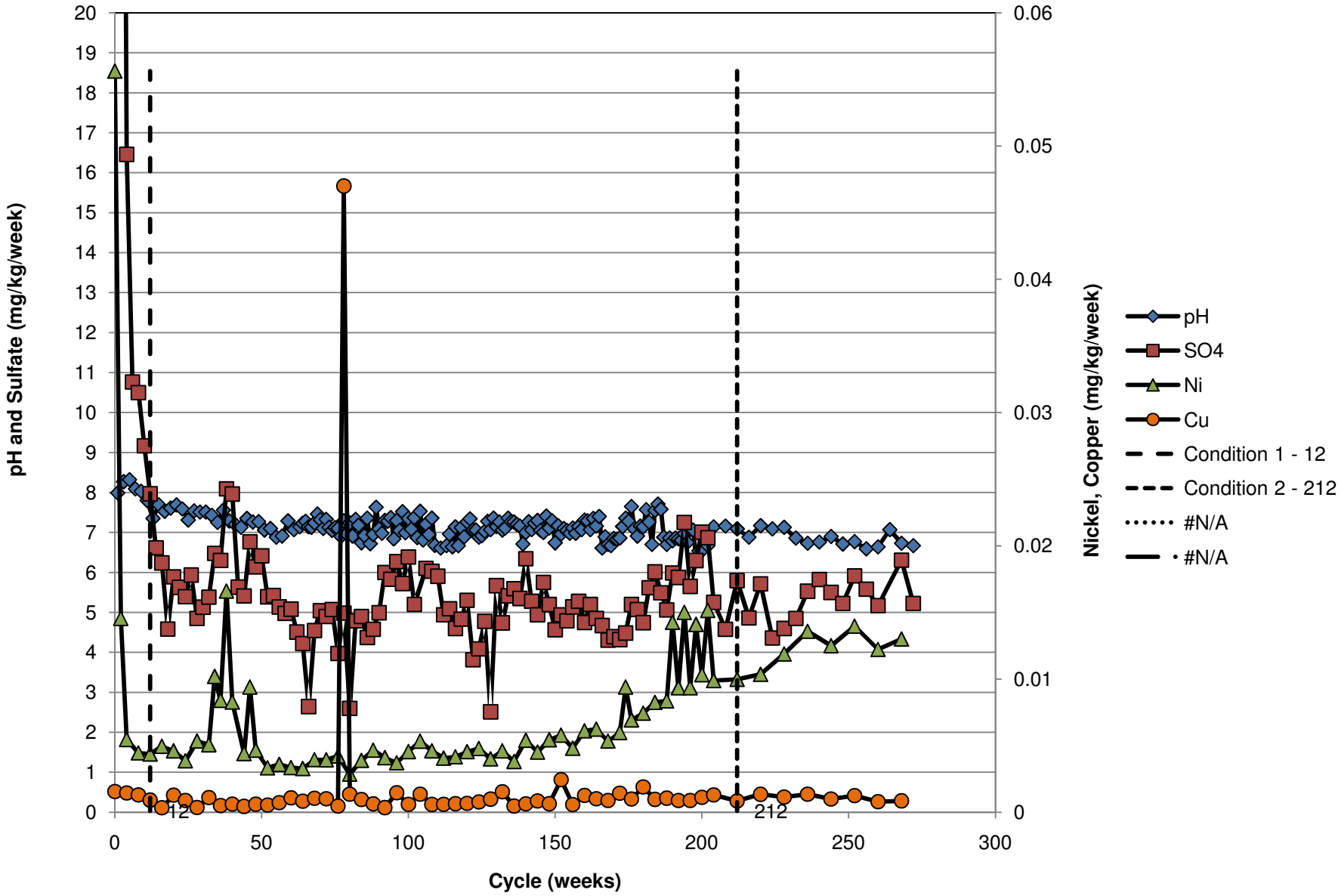
Waste Category:	2/3
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.23%

26056(302-312)



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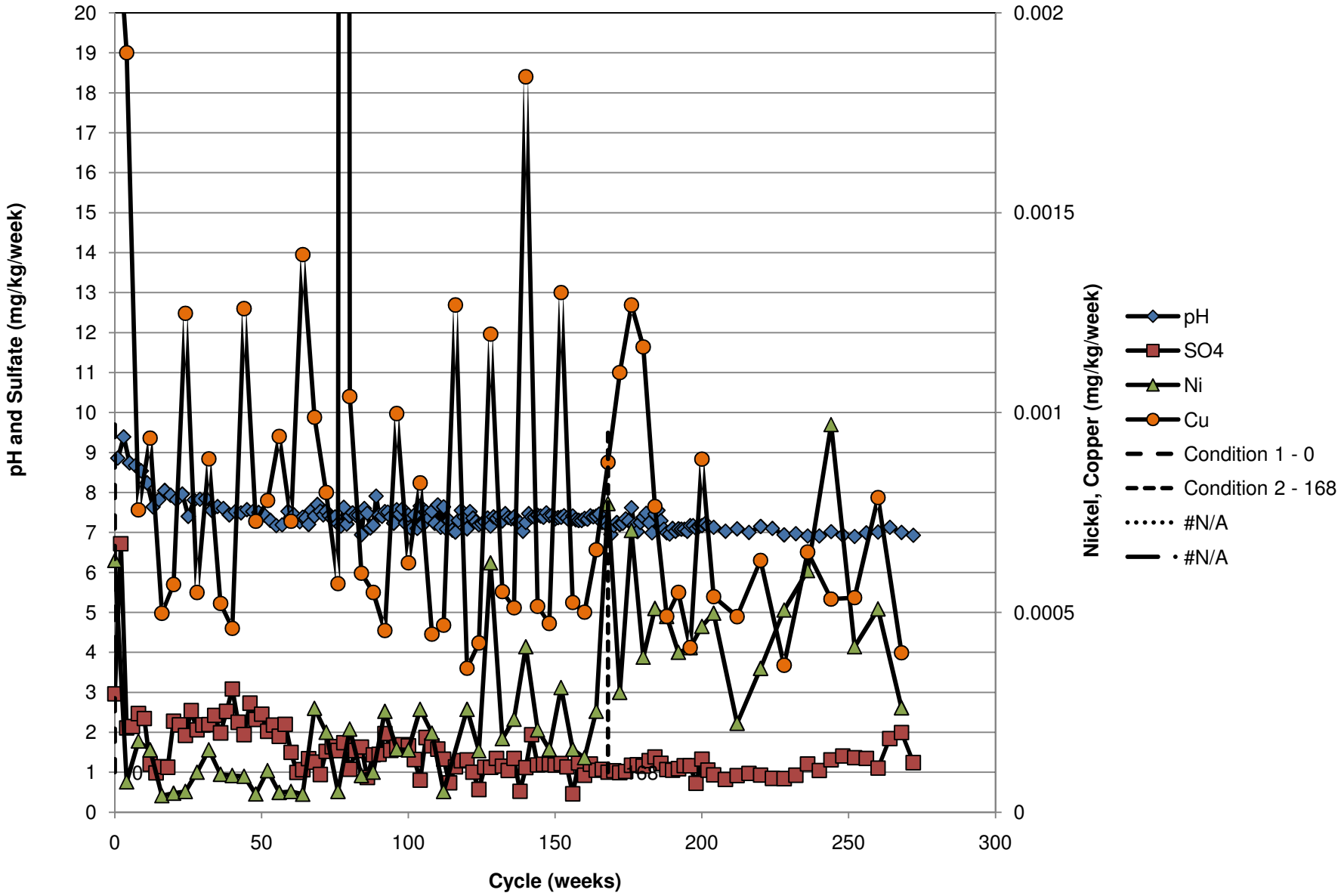
Waste Category:	2/3
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.18%

26142(360+345-365+350)



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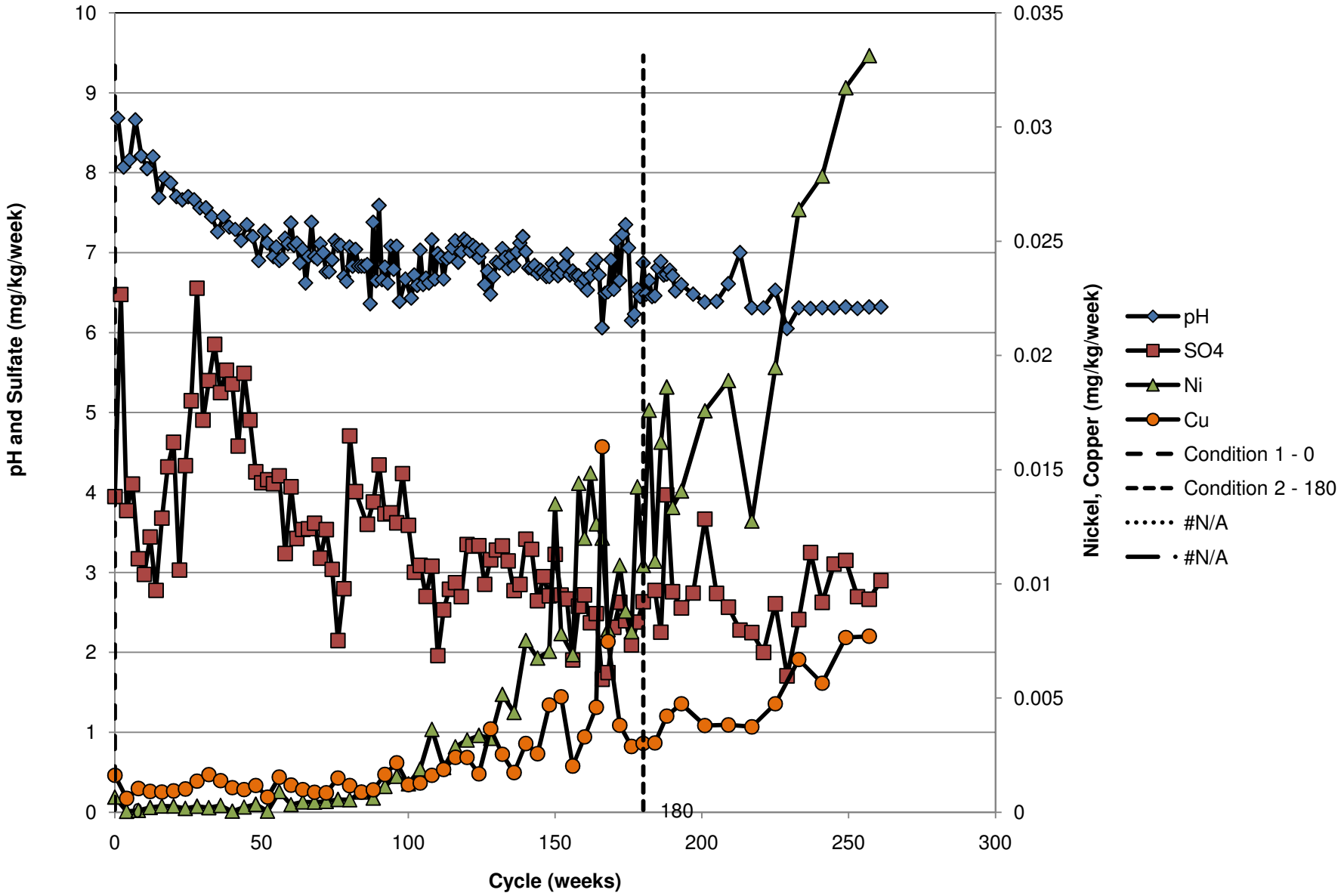
Waste Category:	2/3
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.17%

99-318C(325-330)



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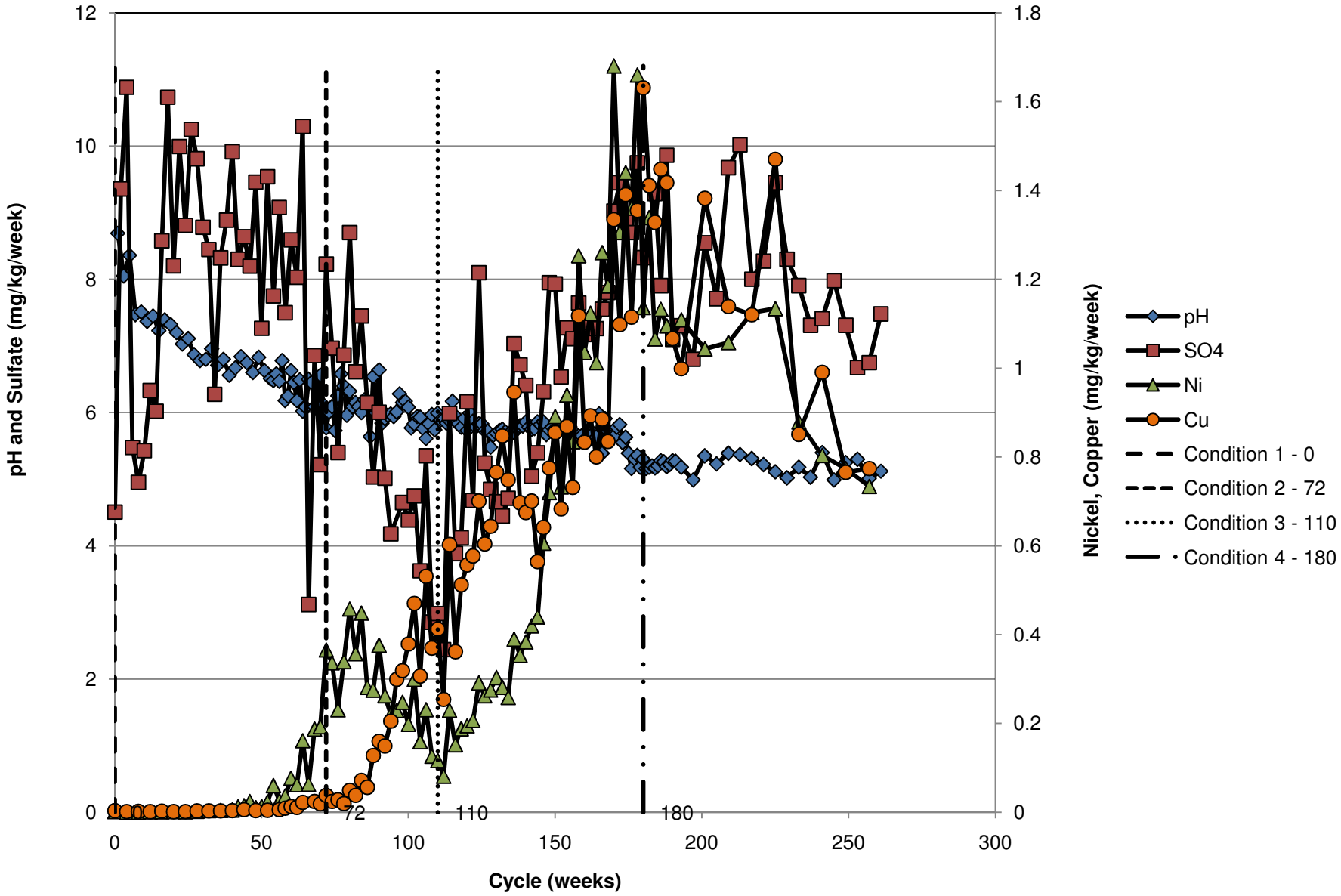
Waste Category:	2/3
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.36%

00-340C(910-925)



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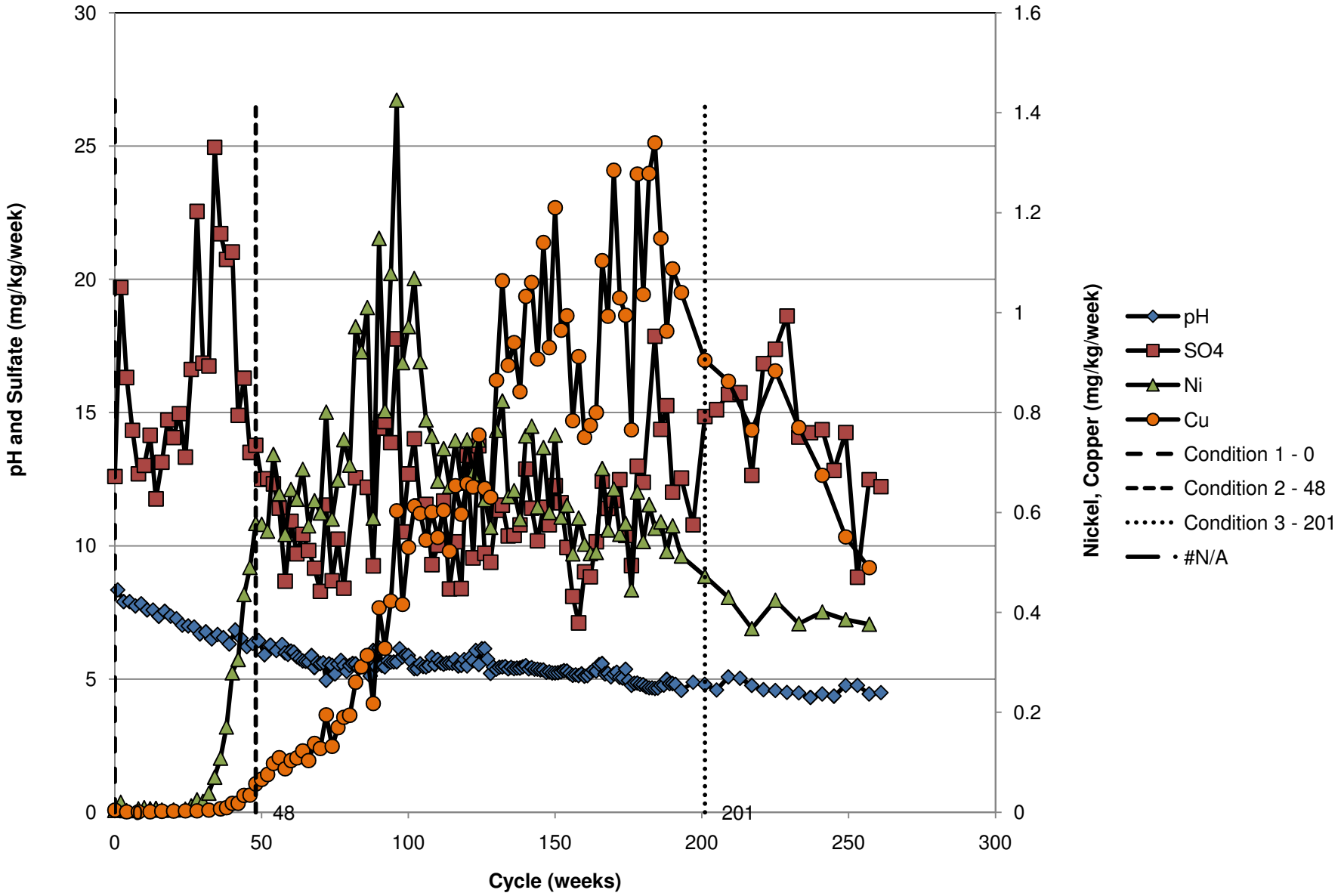
Waste Category:	2/3
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.42%

00-331C(190-210)



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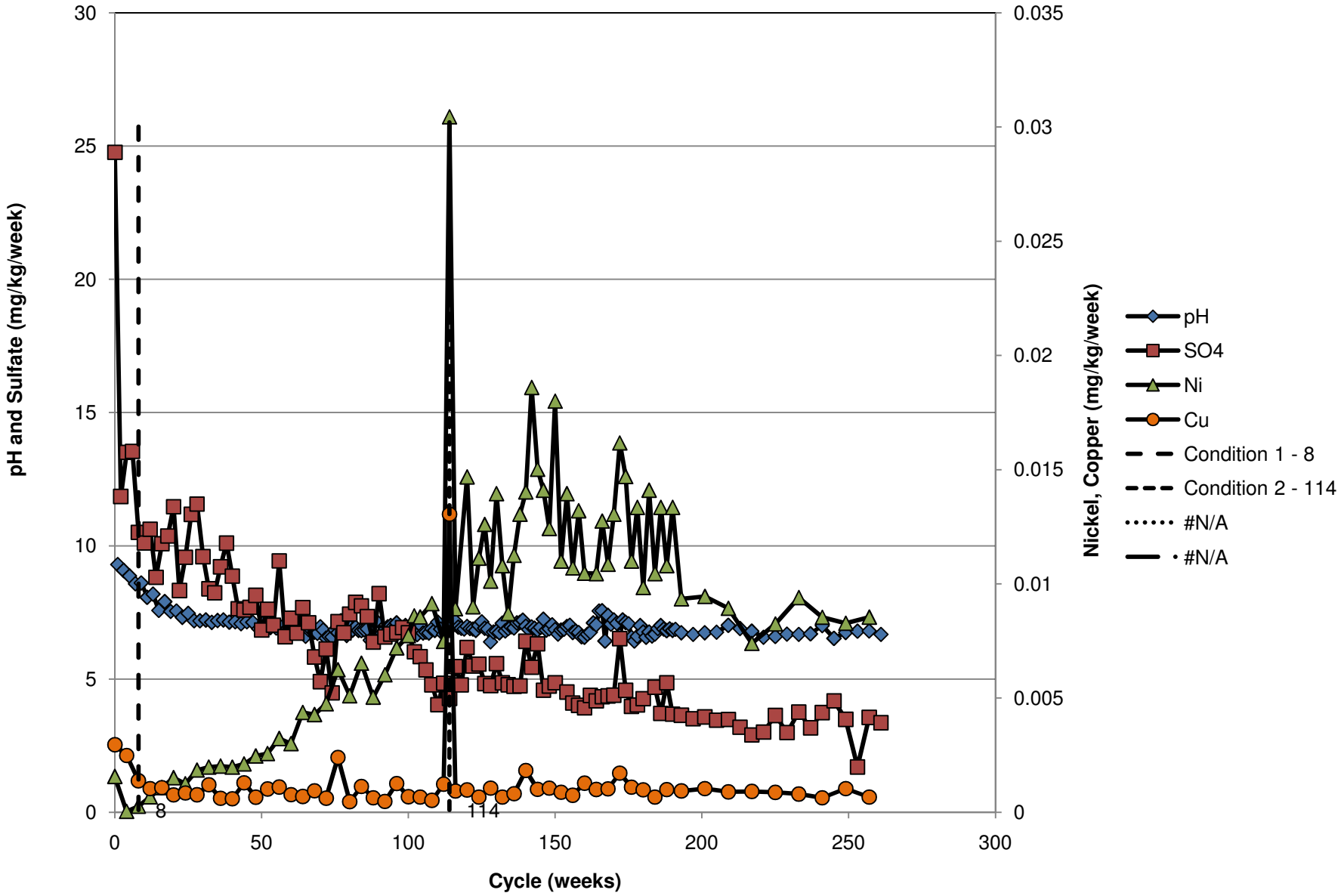
Waste Category:	2/3
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.28%

00-367C(495-500)



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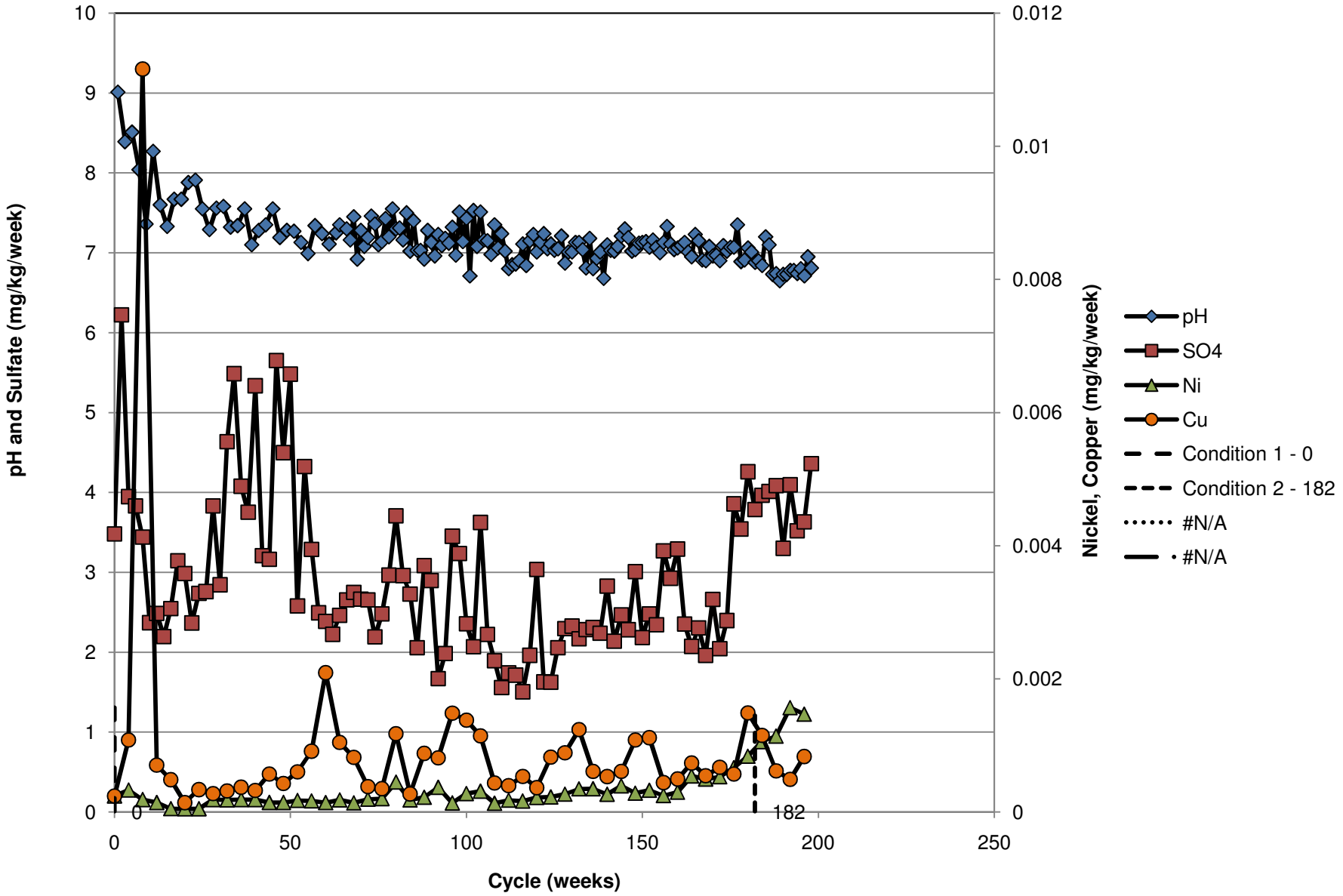
Waste Category:	2/3
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.44%

00-327C(225-245)



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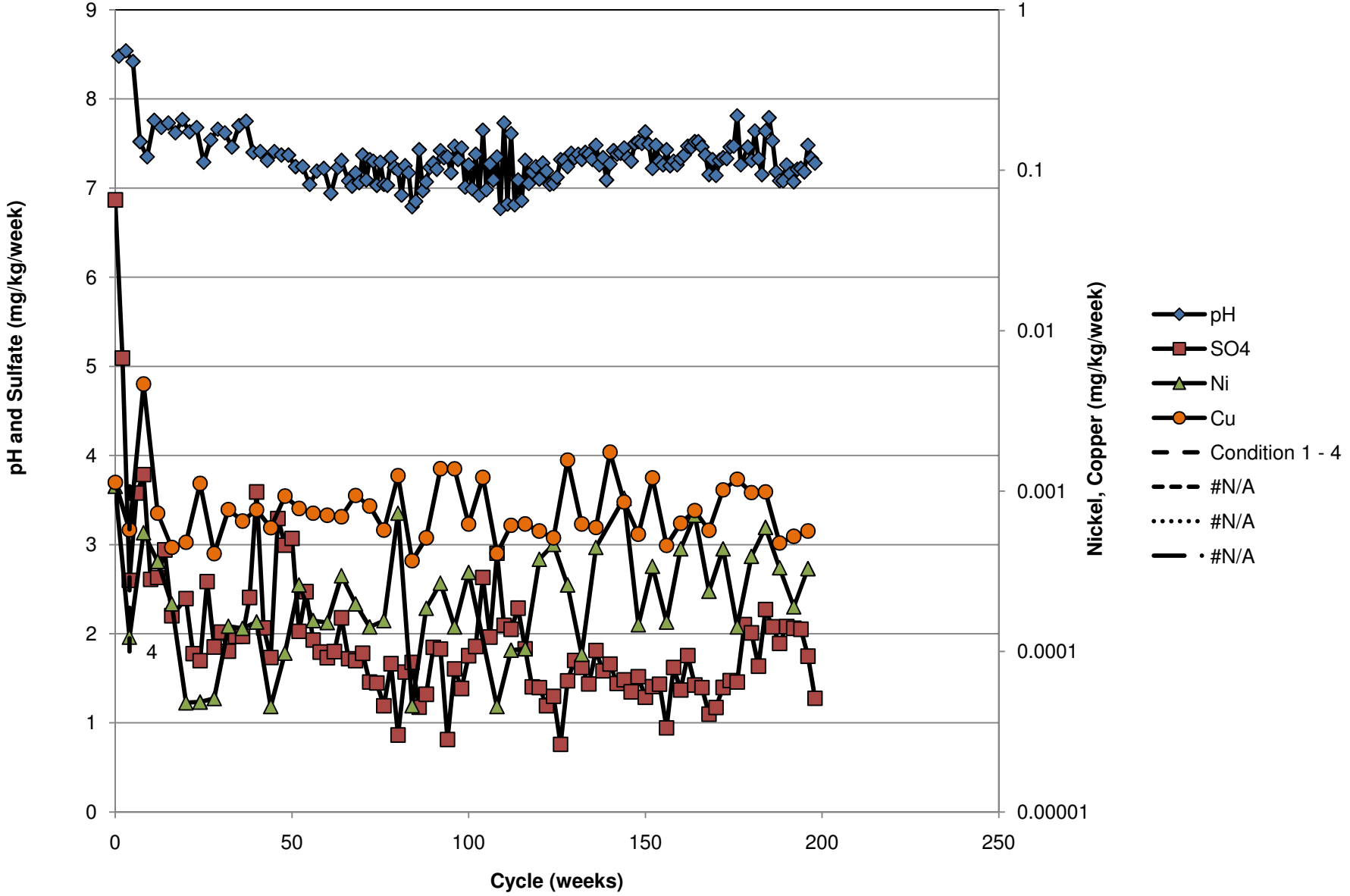
Waste Category:	2/3
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.15%

00-340C(380-390)



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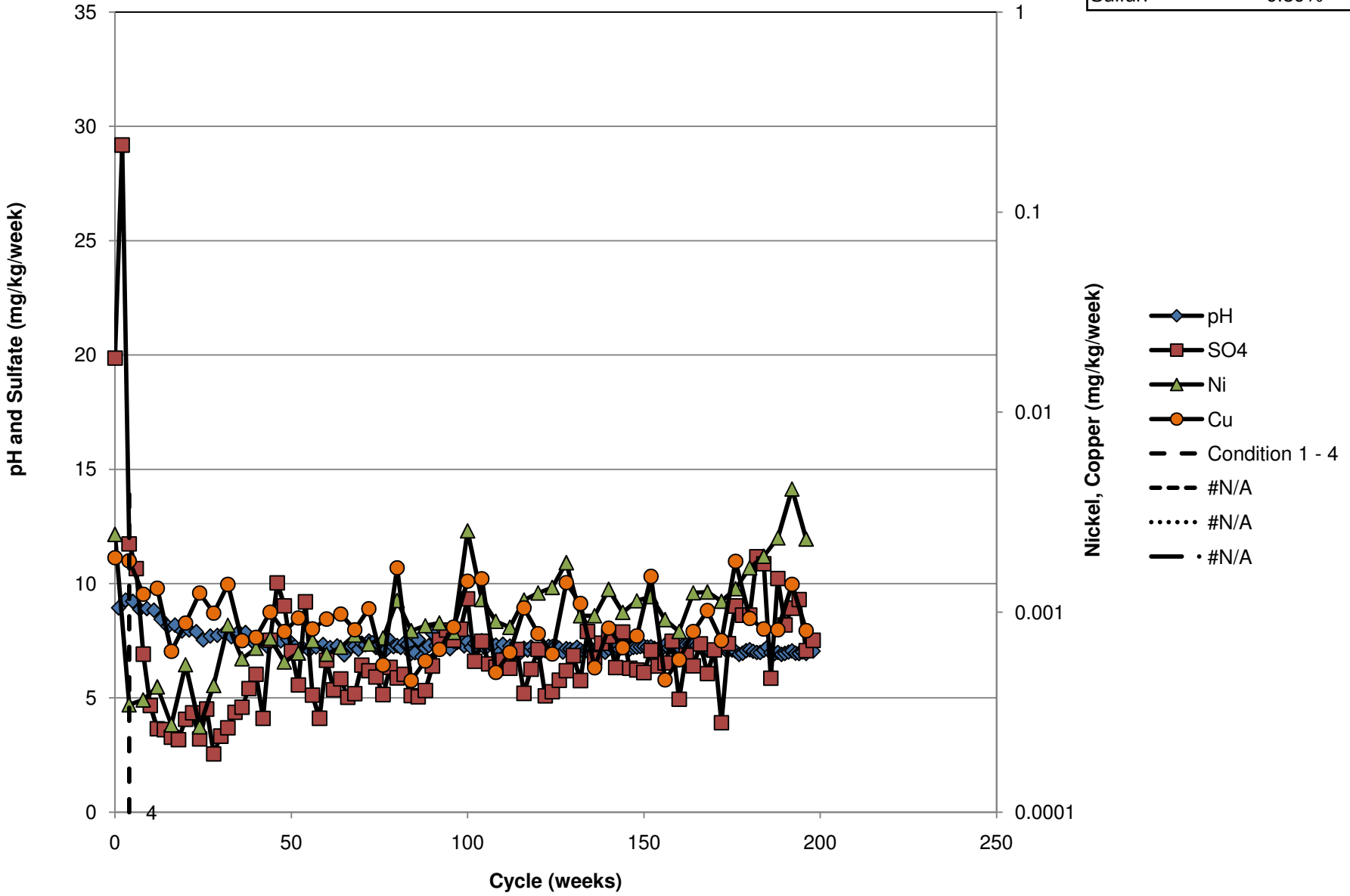
26049+26030

Waste Category:	2/3
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.59%



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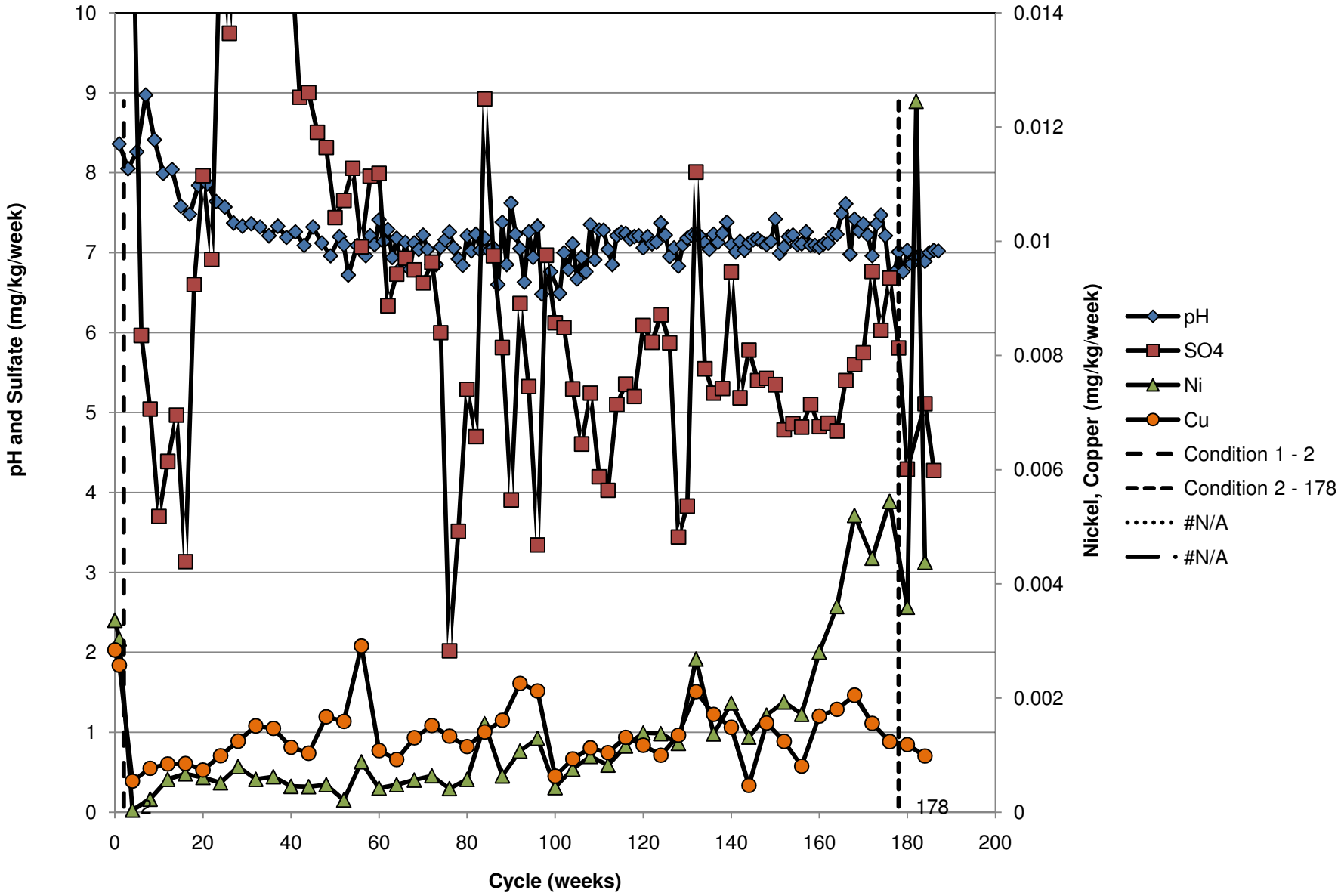
Waste Category:	2/3
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.32%

26056(282-292)



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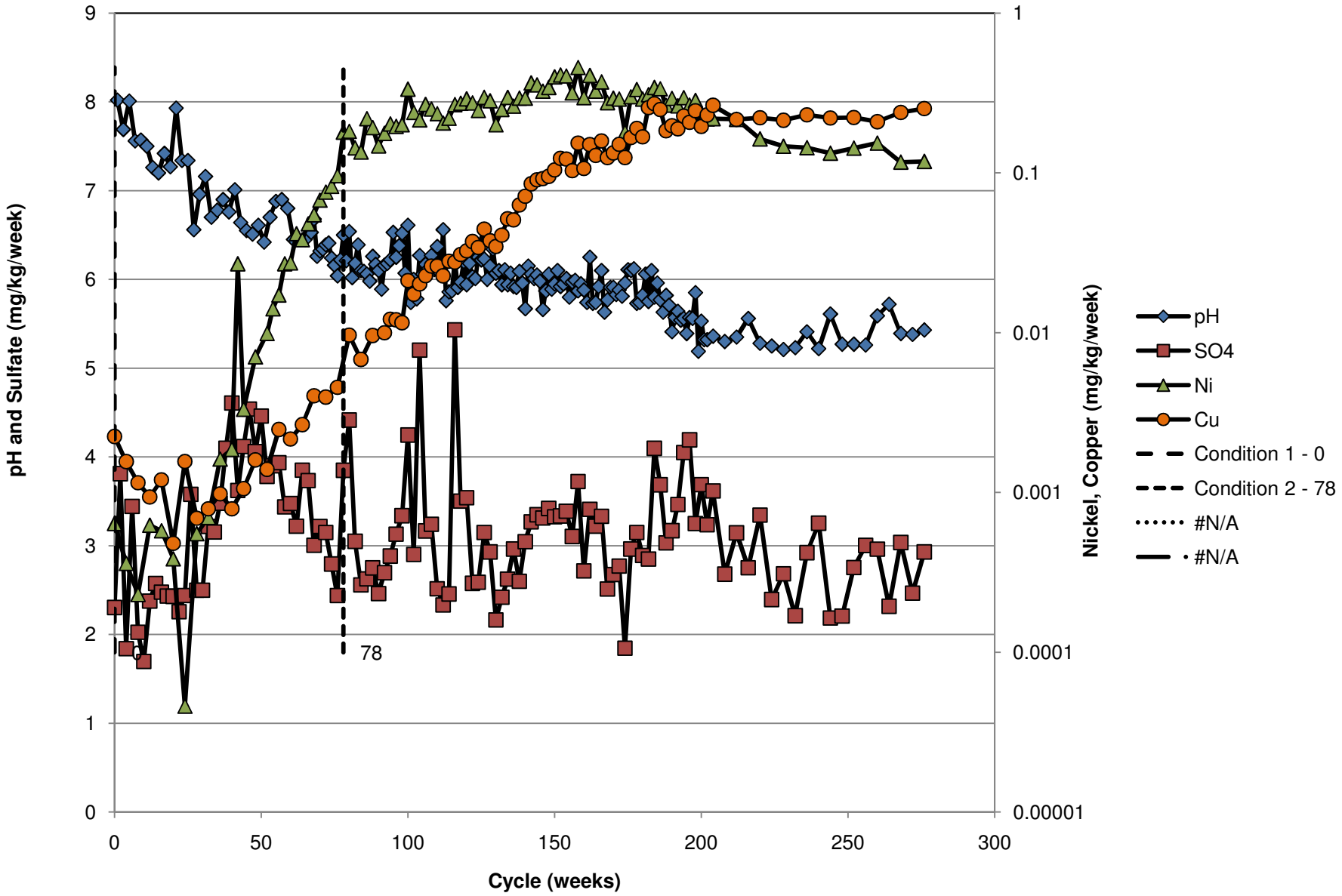
Waste Category:	2/3
Rock Type:	Ultramafic
Geology:	Duluth Complex
Sulfur:	0.20%

00-357C(535-540)



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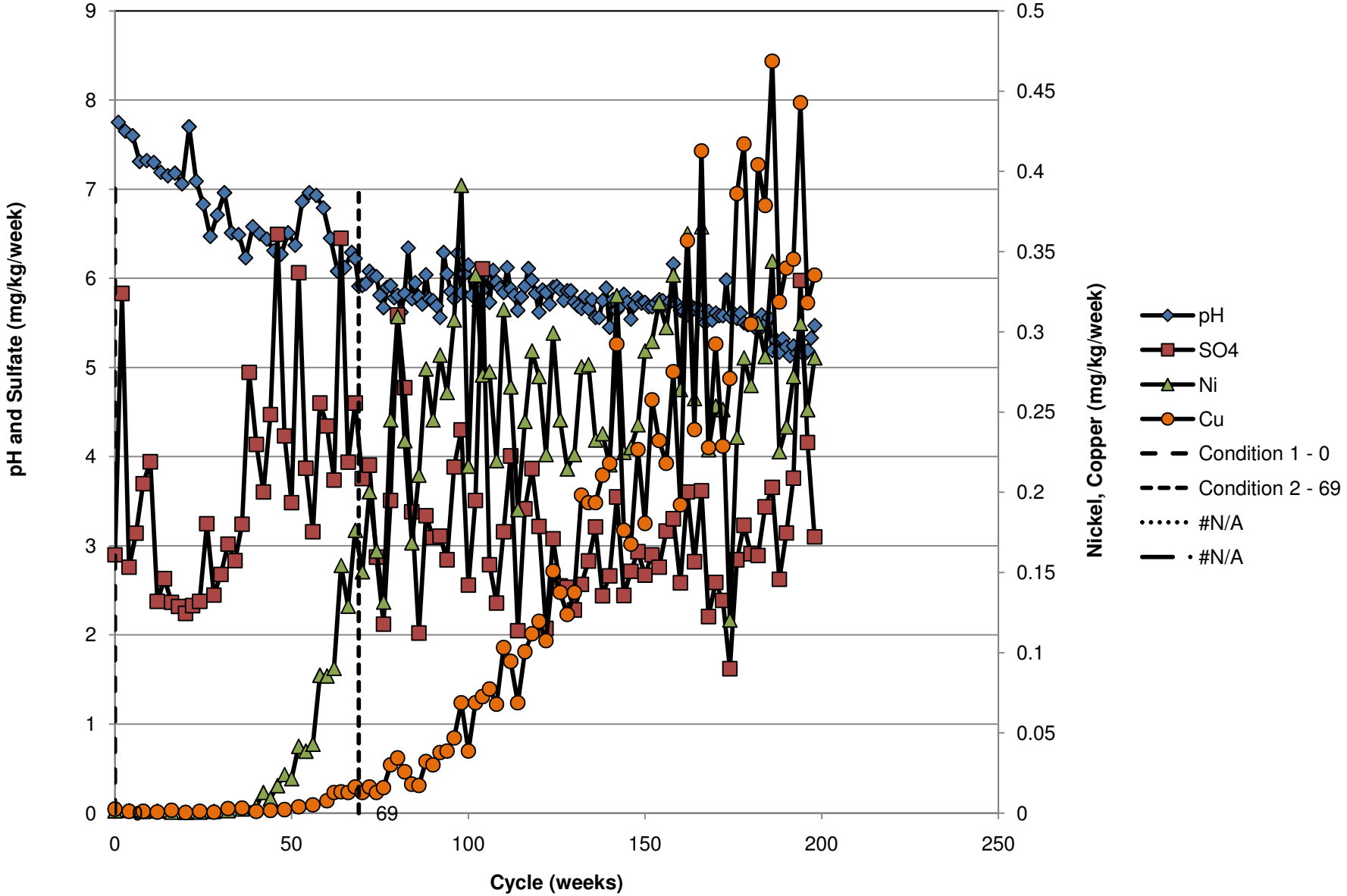
Waste Category:	2/3
Rock Type:	Ultramafic
Geology:	Duluth Complex
Sulfur:	0.30%

00-326C(680-685)



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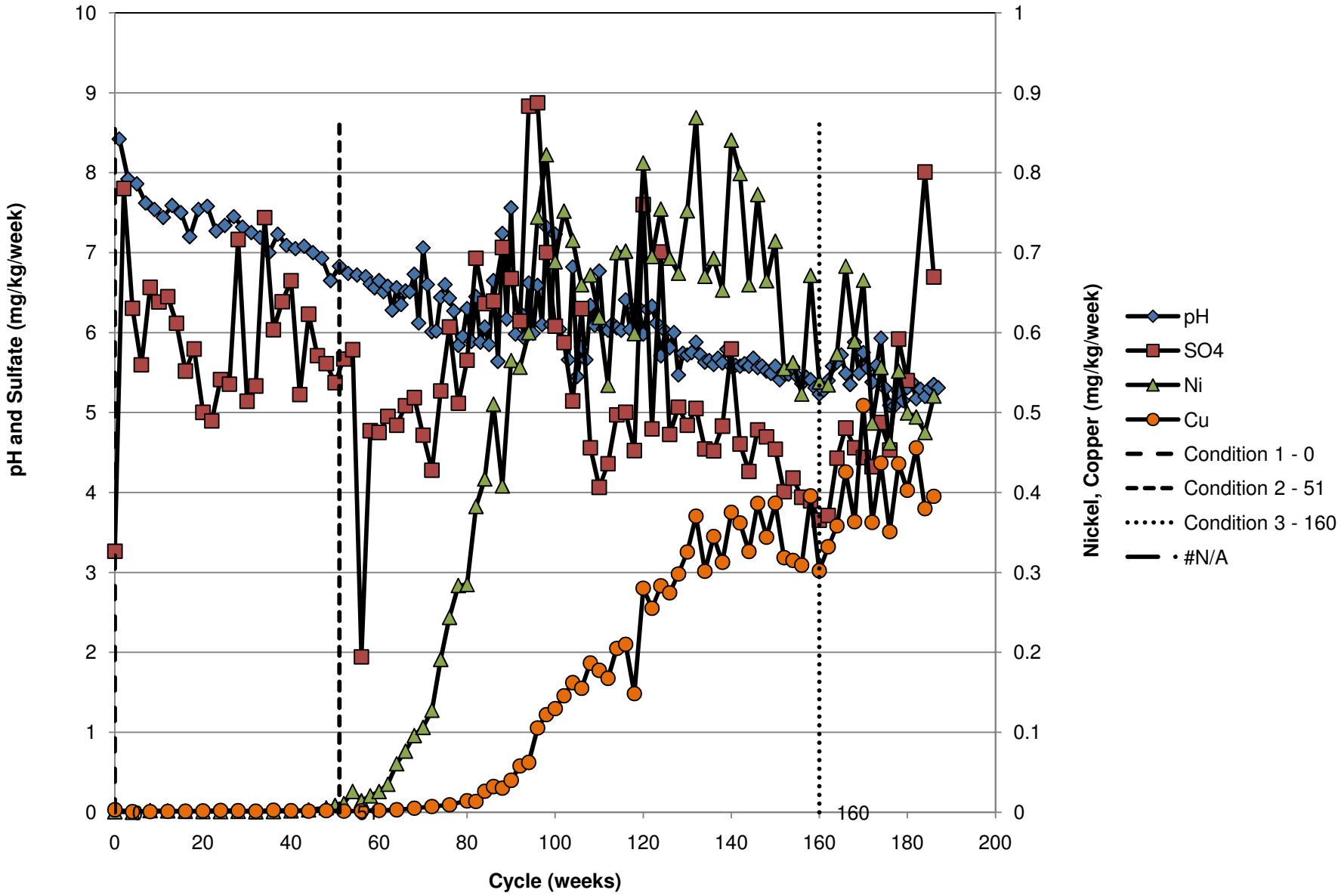
Waste Category:	2/3
Rock Type:	Ultramafic
Geology:	Duluth Complex
Sulfur:	0.34%

00-344C(630-635)



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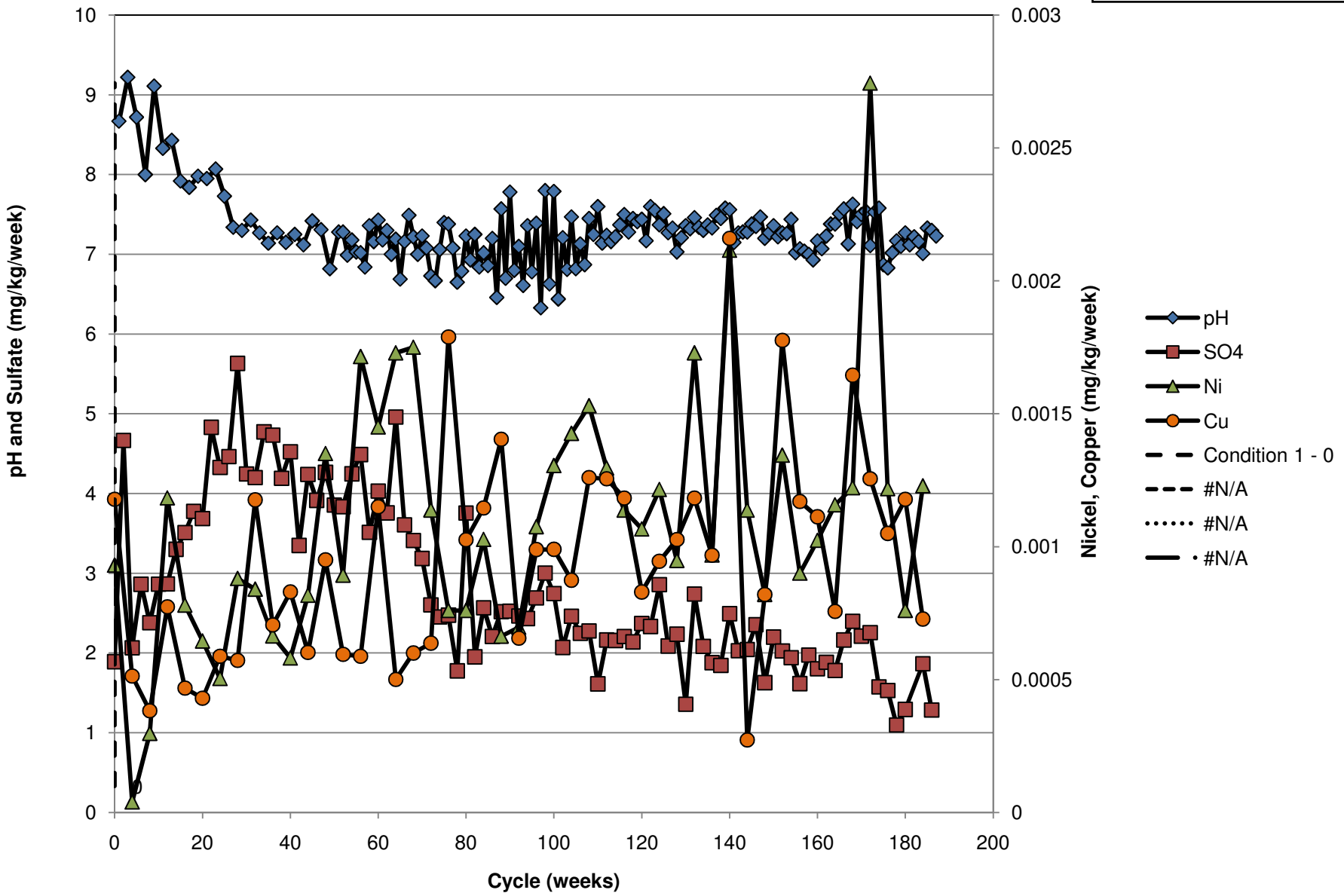
Waste Category:	2/3
Rock Type:	Ultramafic
Geology:	Duluth Complex
Sulfur:	0.16%

00-326C(495-505)



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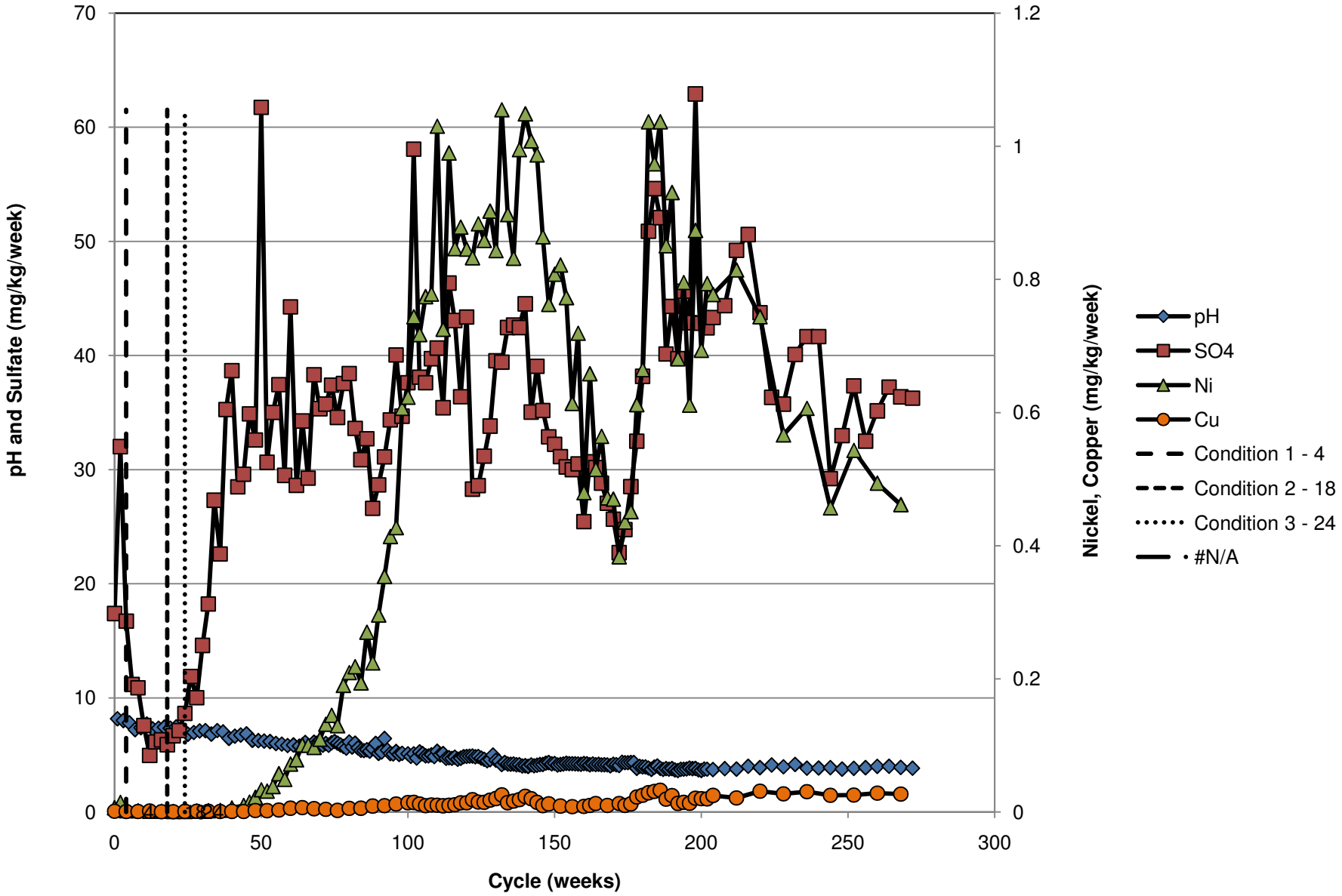
c. Category 4 (Duluth Complex)

Waste Category:	4
Rock Type:	Anorthositic
Geology:	Duluth Complex
Sulfur:	1.83%

26027(616-626)



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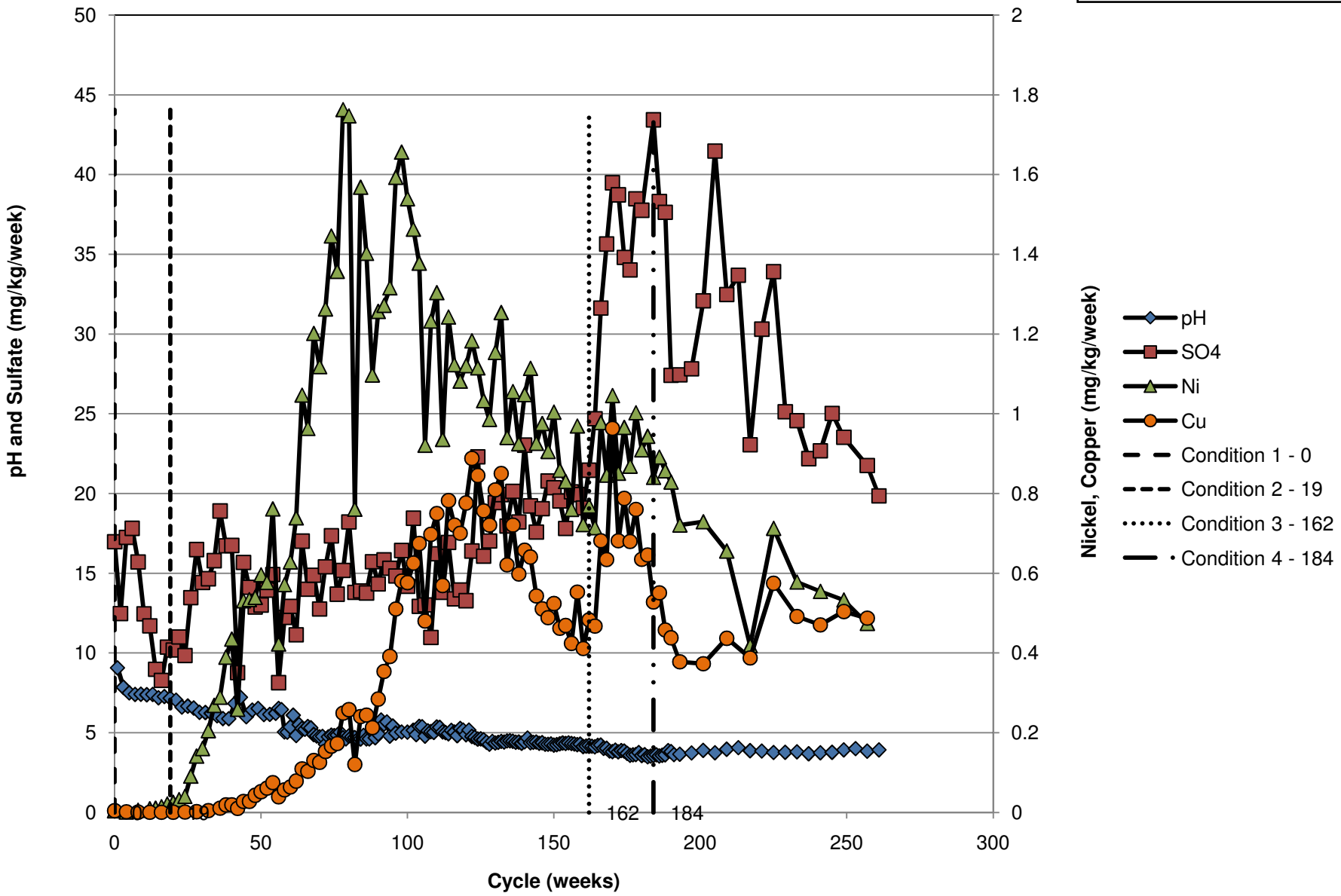
Waste Category:	4
Rock Type:	Anorthositic
Geology:	Duluth Complex
Sulfur:	0.86%



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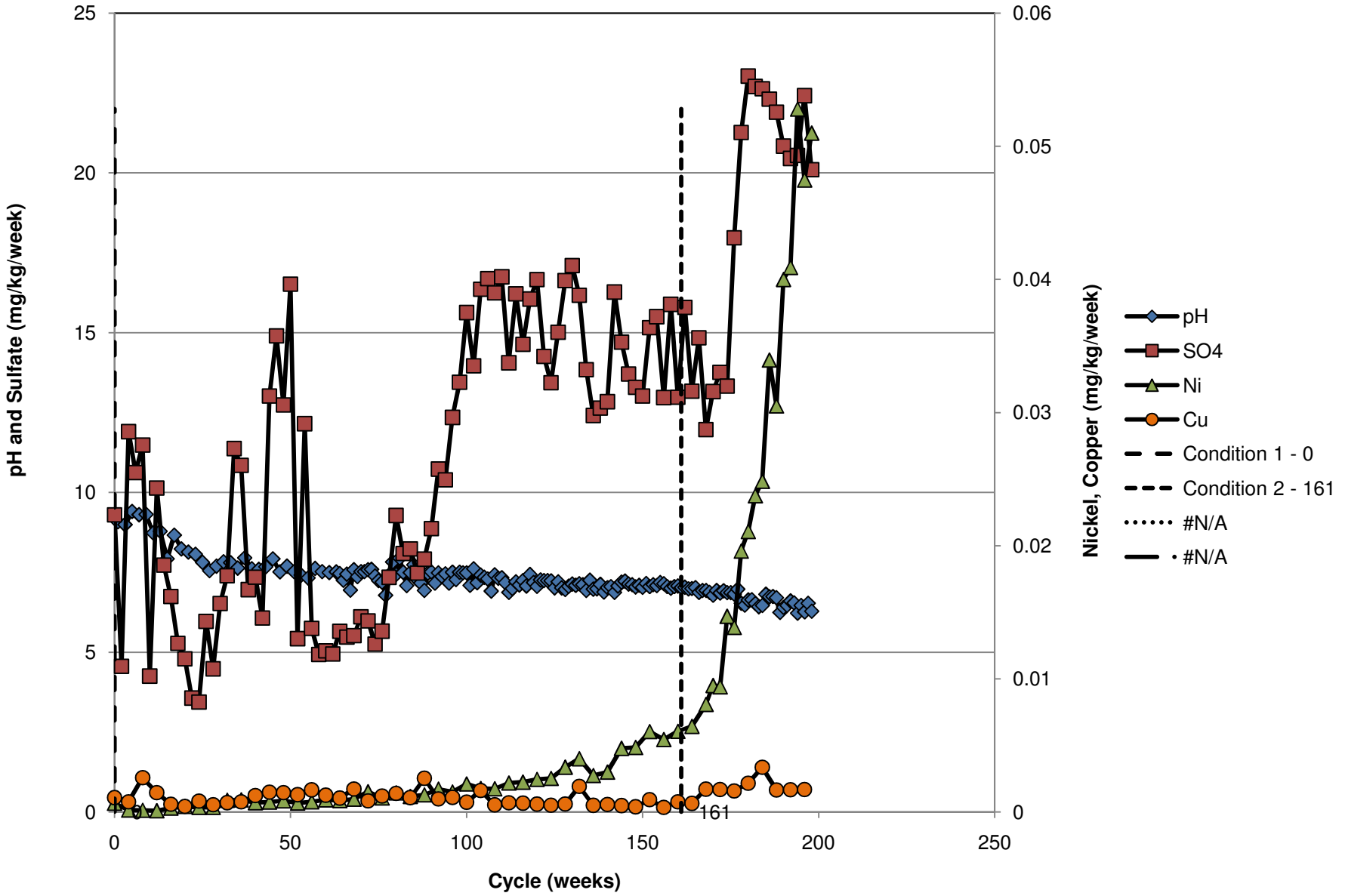
Waste Category:	4
Rock Type:	Anorthositic
Geology:	Duluth Complex
Sulfur:	0.68%

00-343C(240-250)



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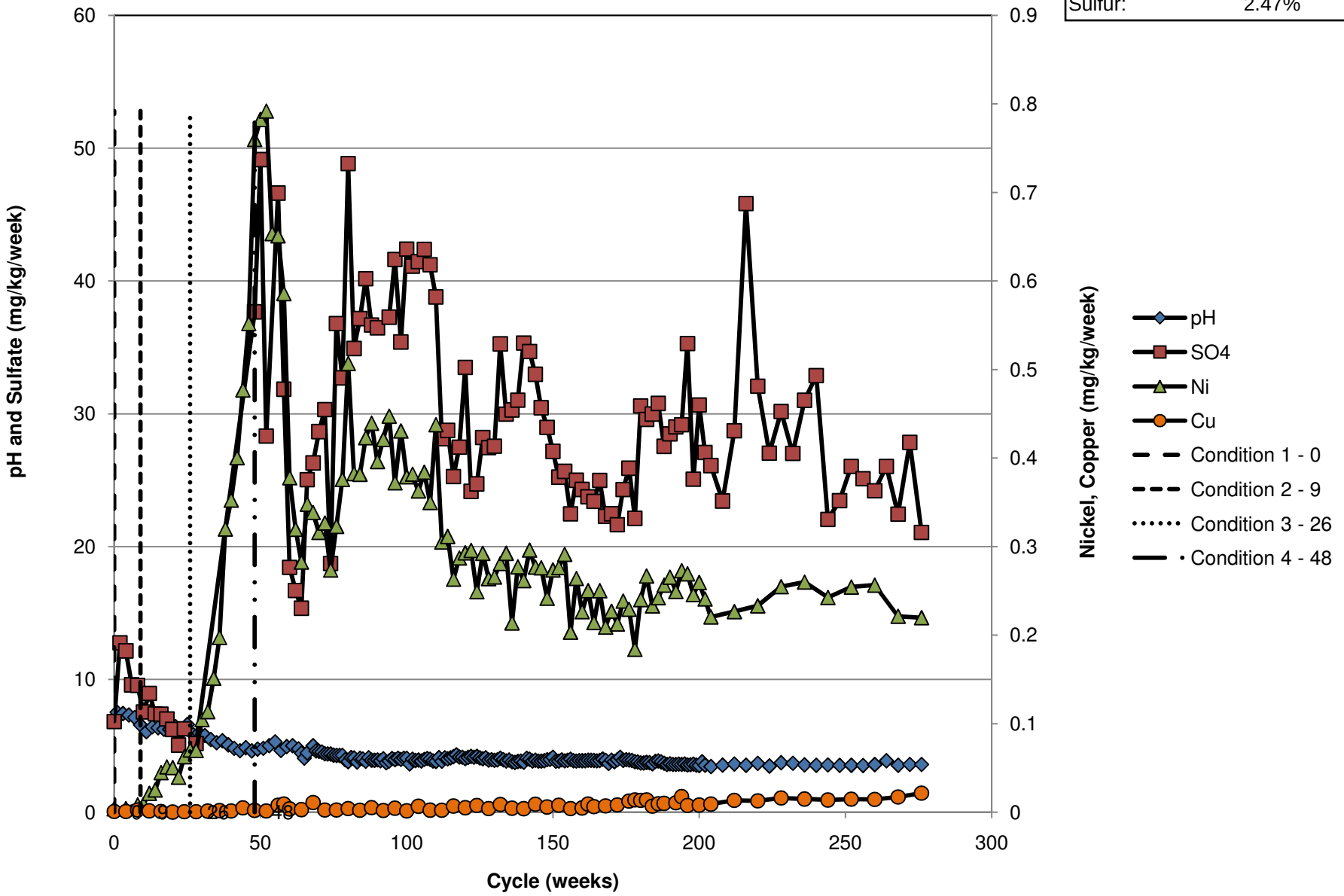
26043+26027

Waste Category:	4
Rock Type:	Sedimentary Hornfels
Geology:	Duluth Complex
Sulfur:	2.47%



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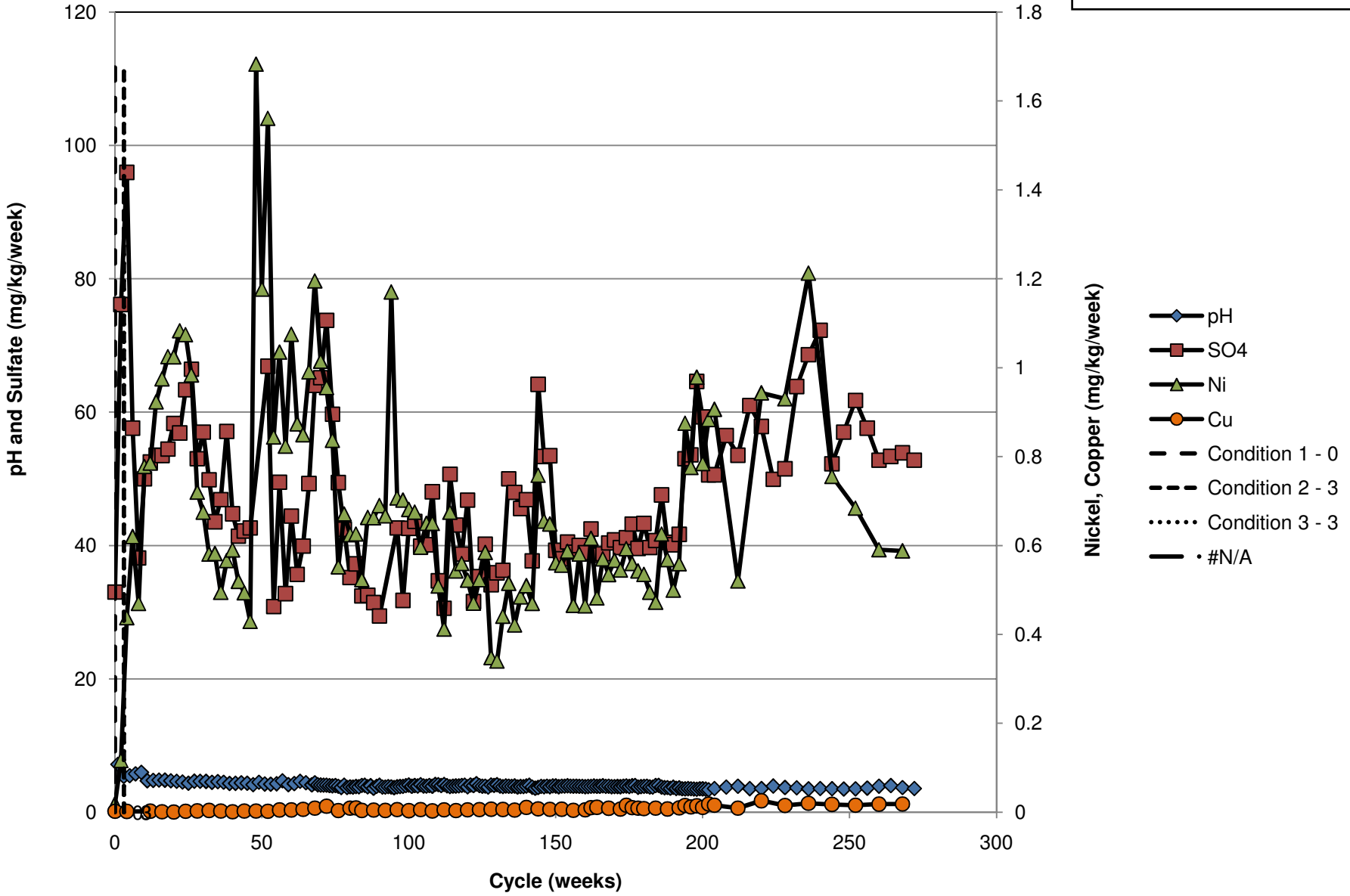
Waste Category:	4
Rock Type:	Sedimentary Hornfels
Geology:	Duluth Complex
Sulfur:	4.46%

26062+26026



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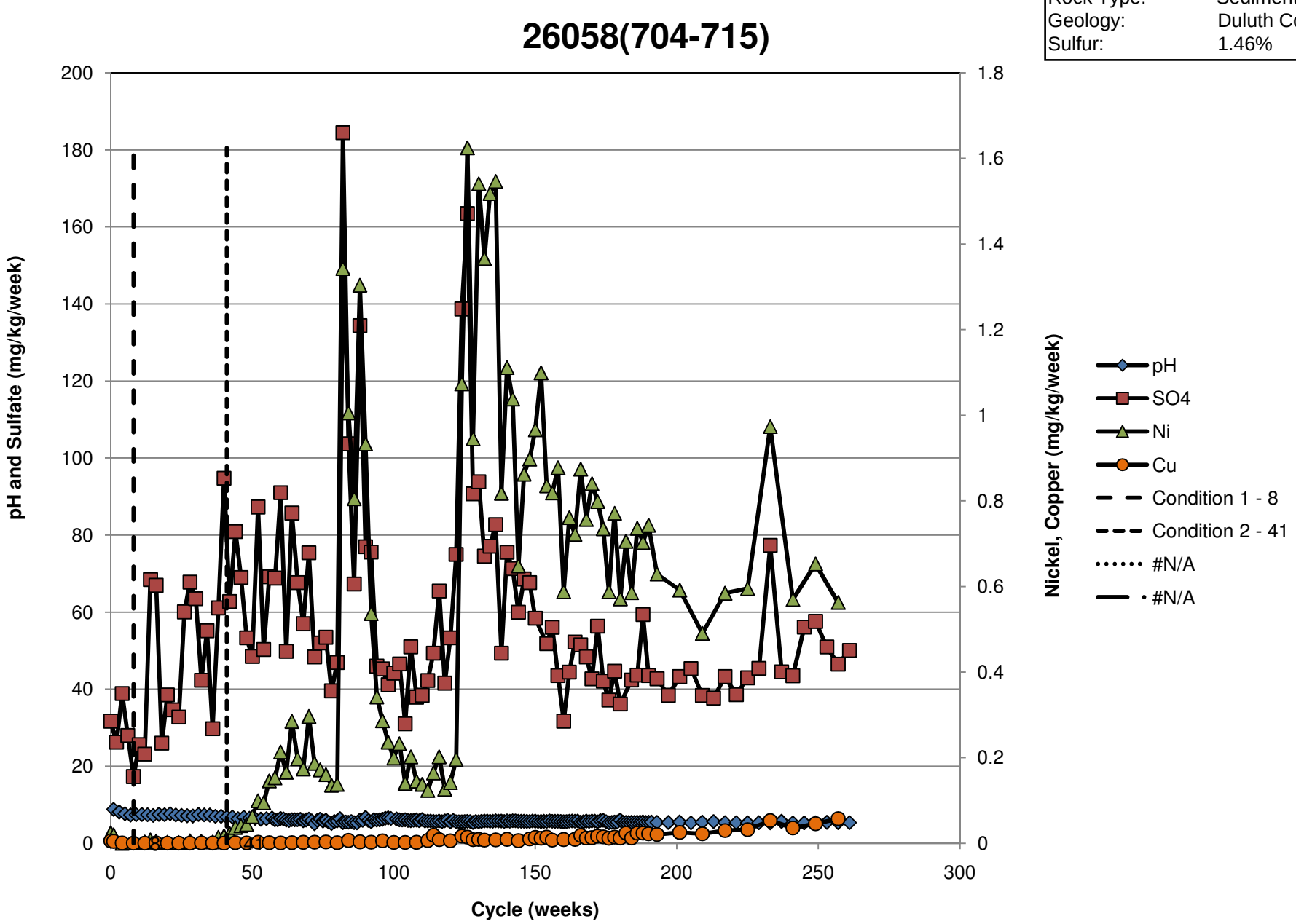
Waste Category:	4
Rock Type:	Sedimentary Hornfels
Geology:	Duluth Complex
Sulfur:	1.46%



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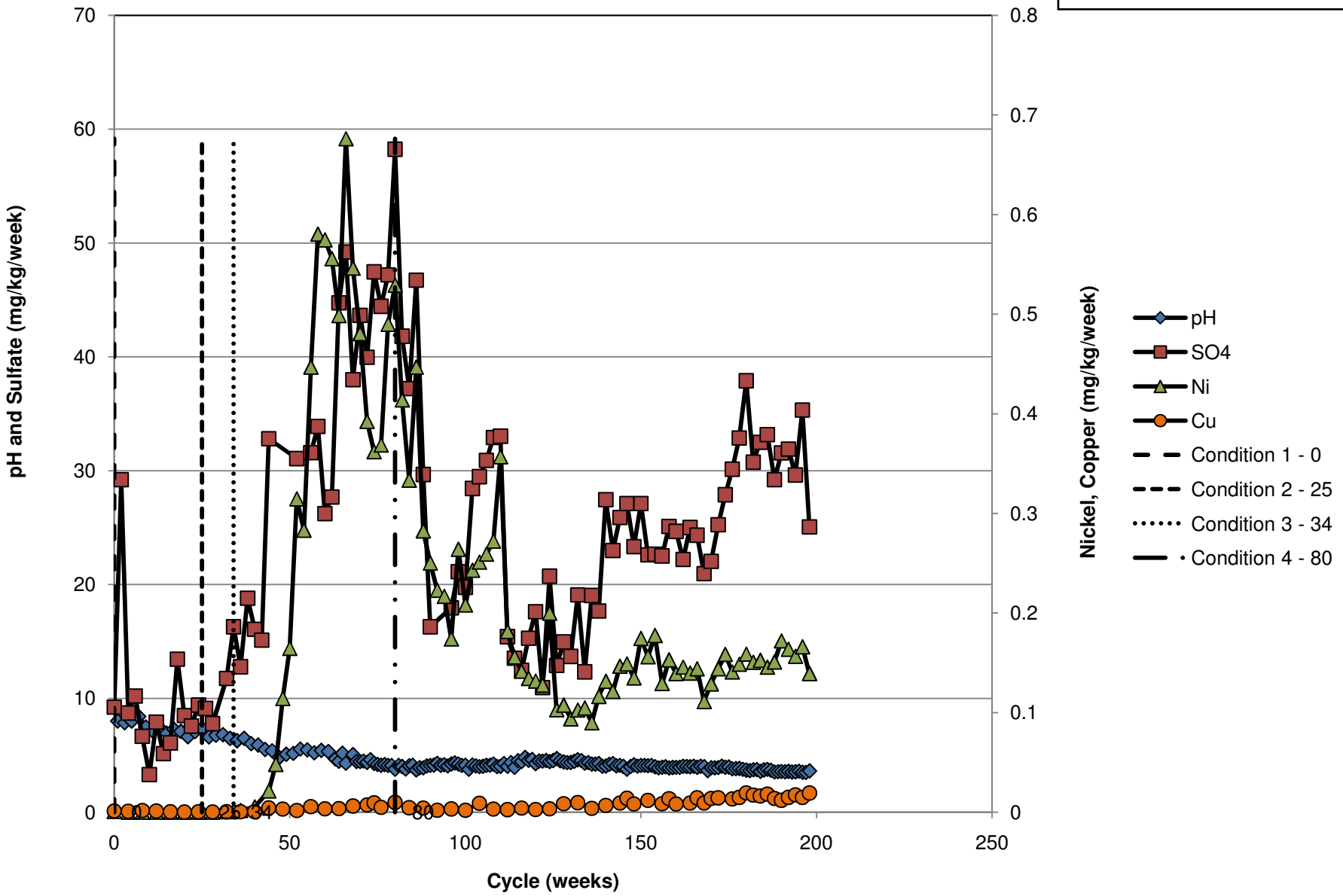
Waste Category:	4
Rock Type:	Sedimentary Hornfels
Geology:	Duluth Complex
Sulfur:	1.74%

00-340C(965-974.5)



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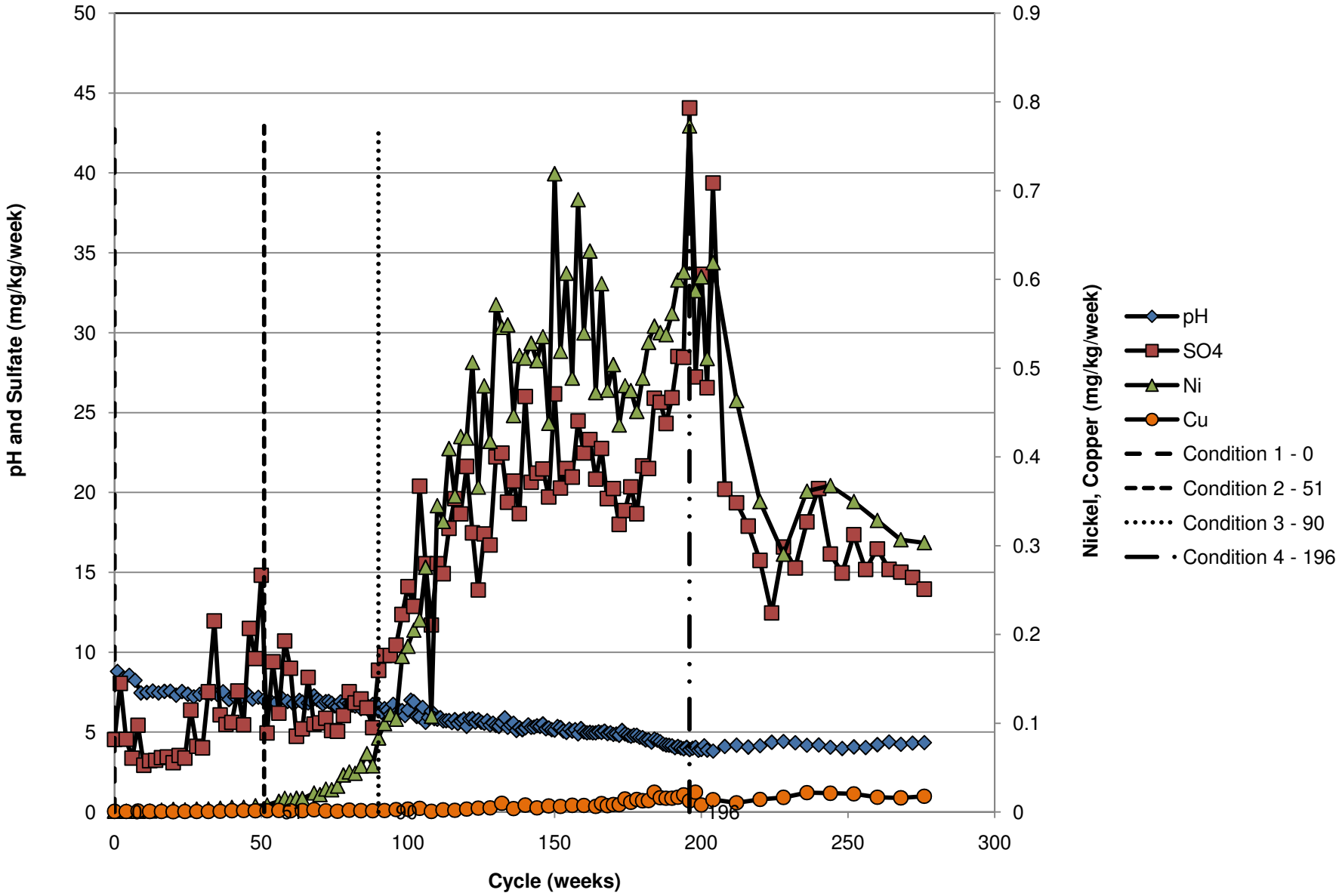
Waste Category:	4
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.88%

00-371C(435-440)



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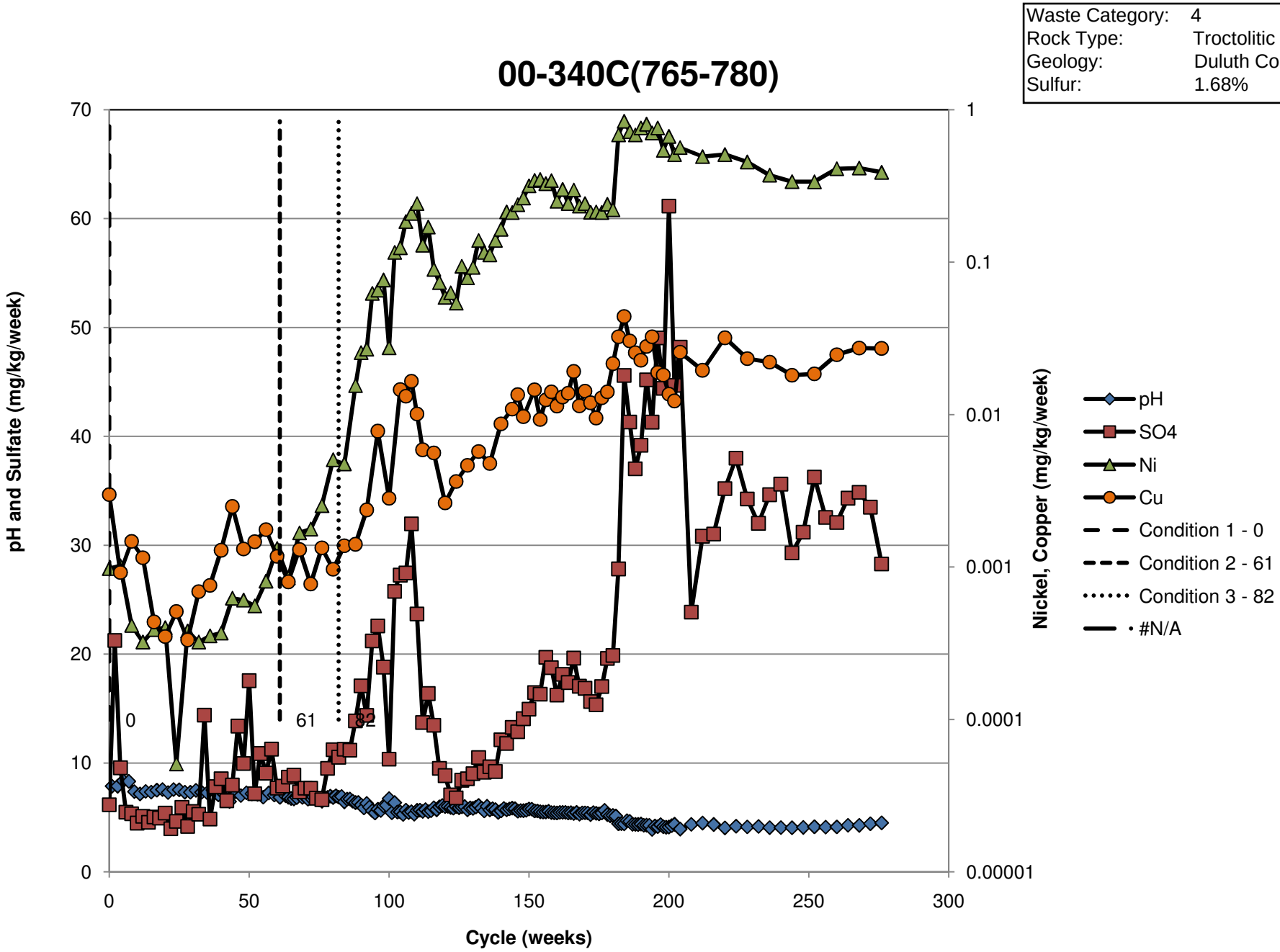

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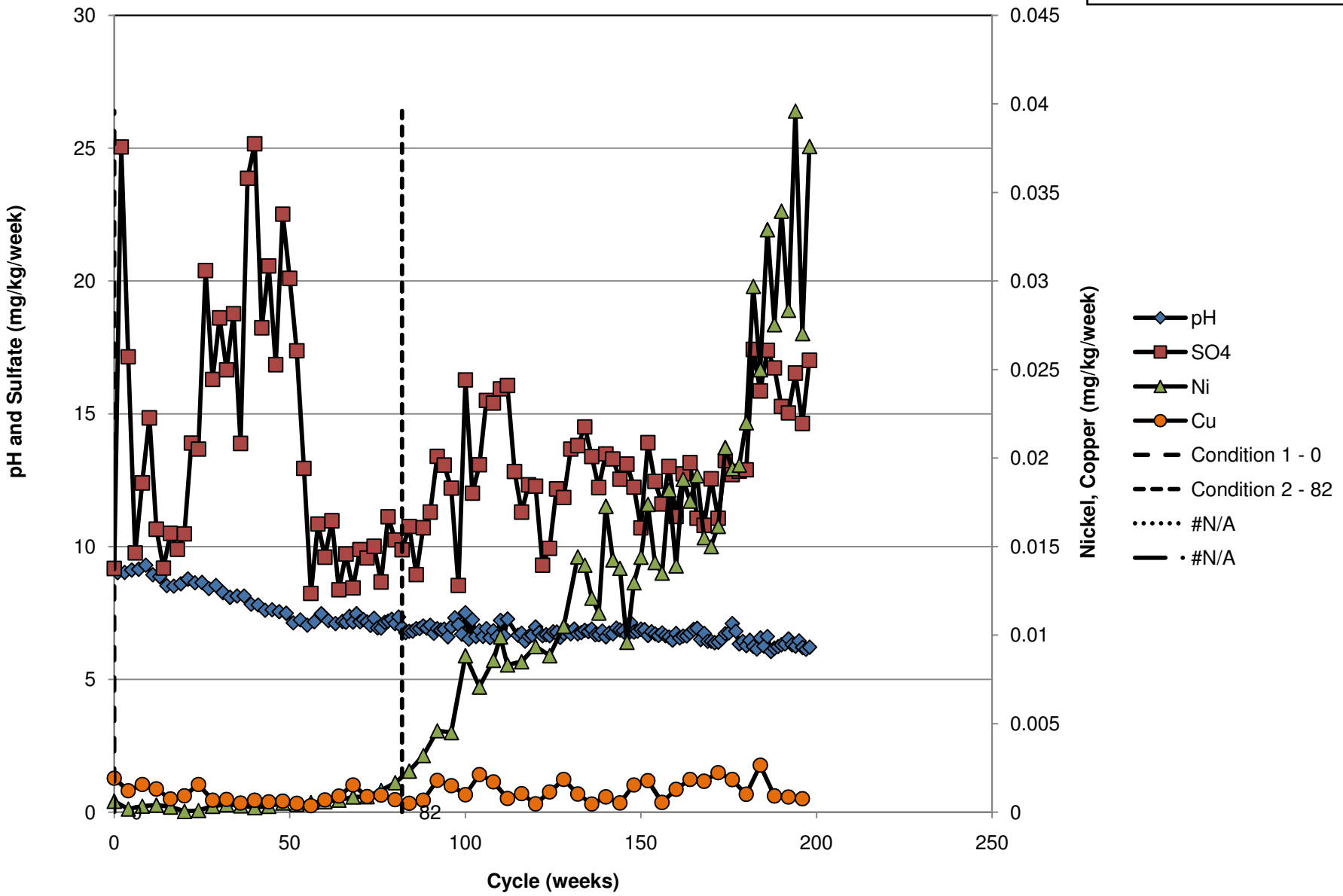
Waste Category:	4
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	0.77%

00-367C(395-400)



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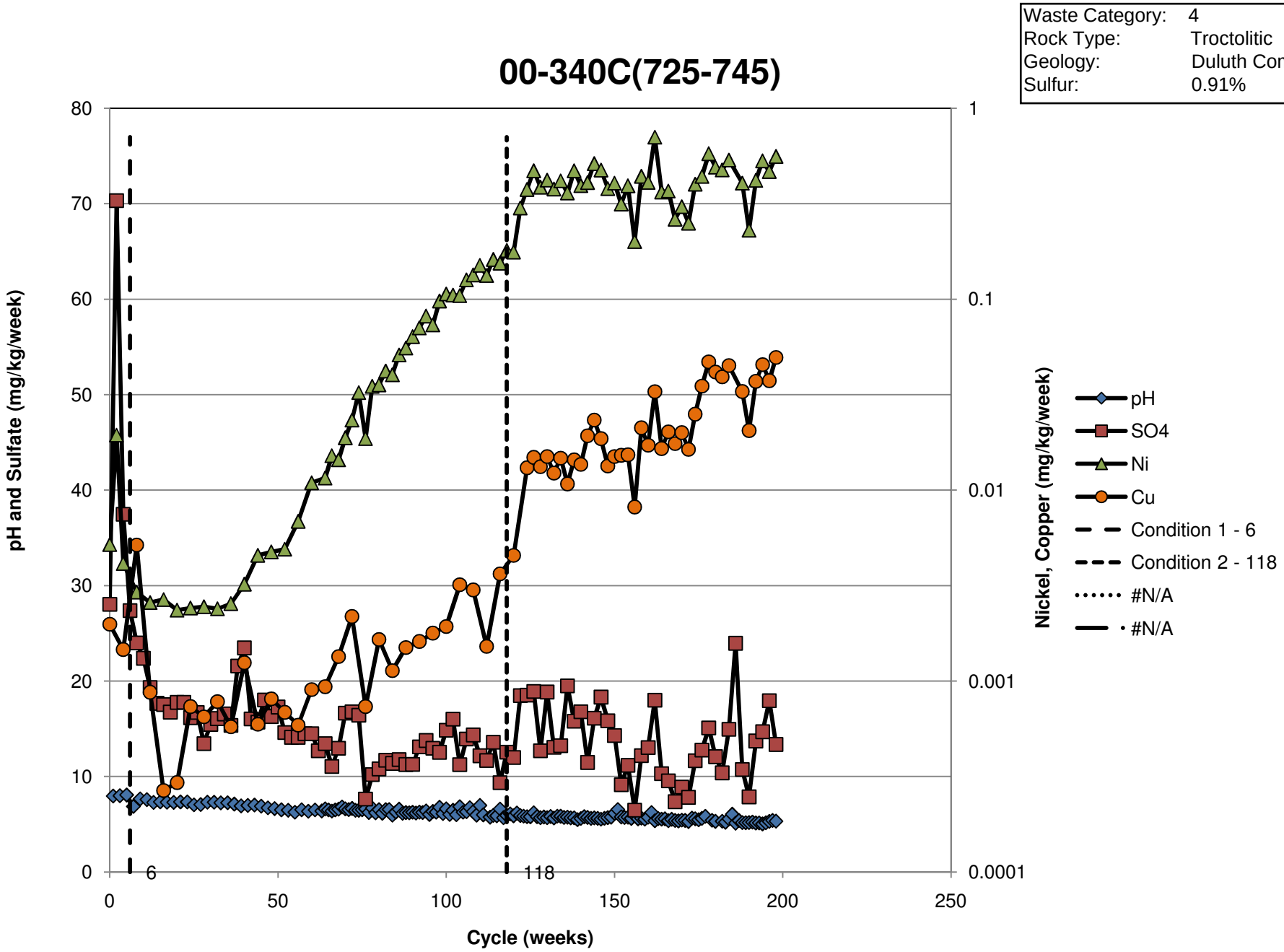

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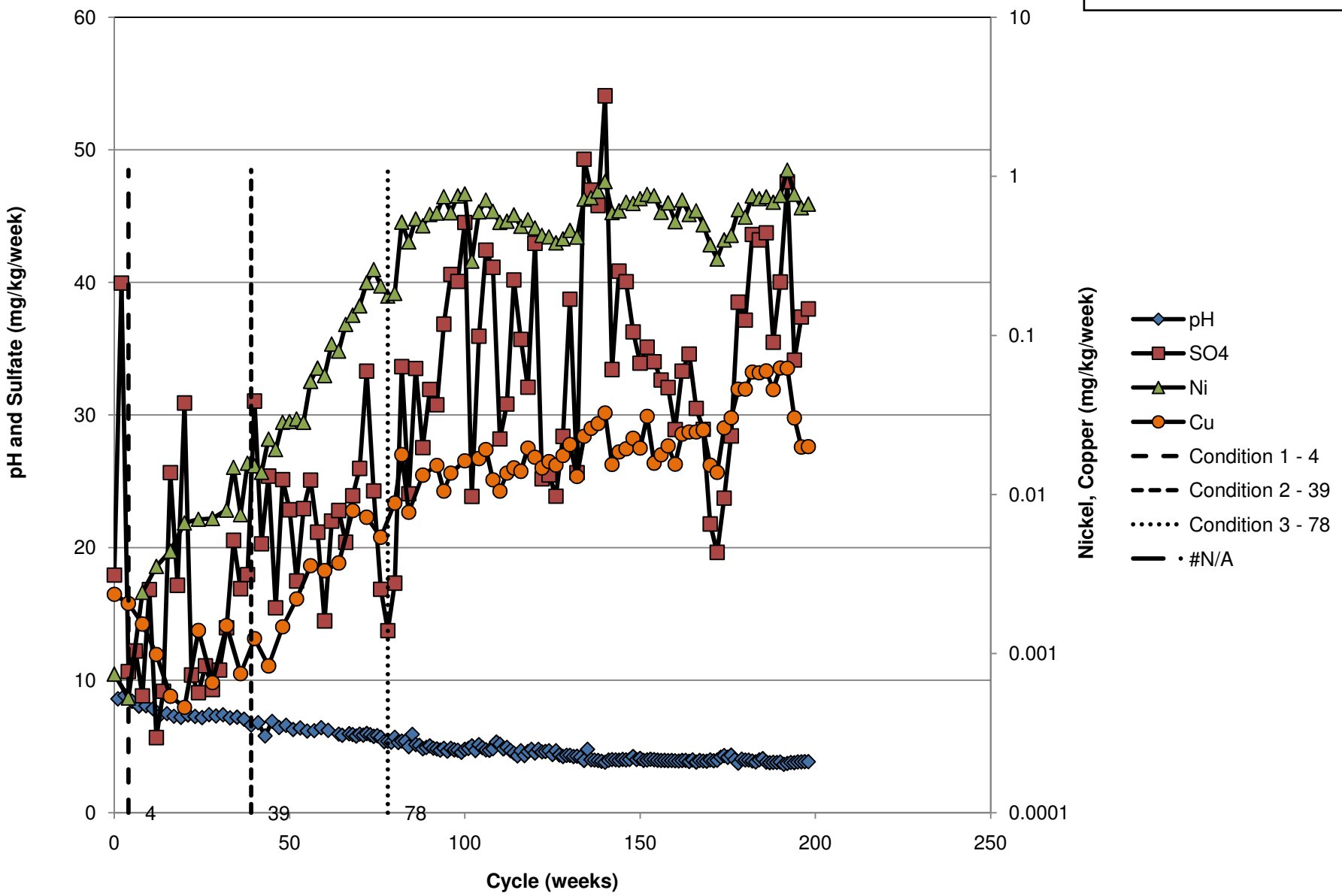
Waste Category:	4
Rock Type:	Troctolitic
Geology:	Duluth Complex
Sulfur:	1.37%



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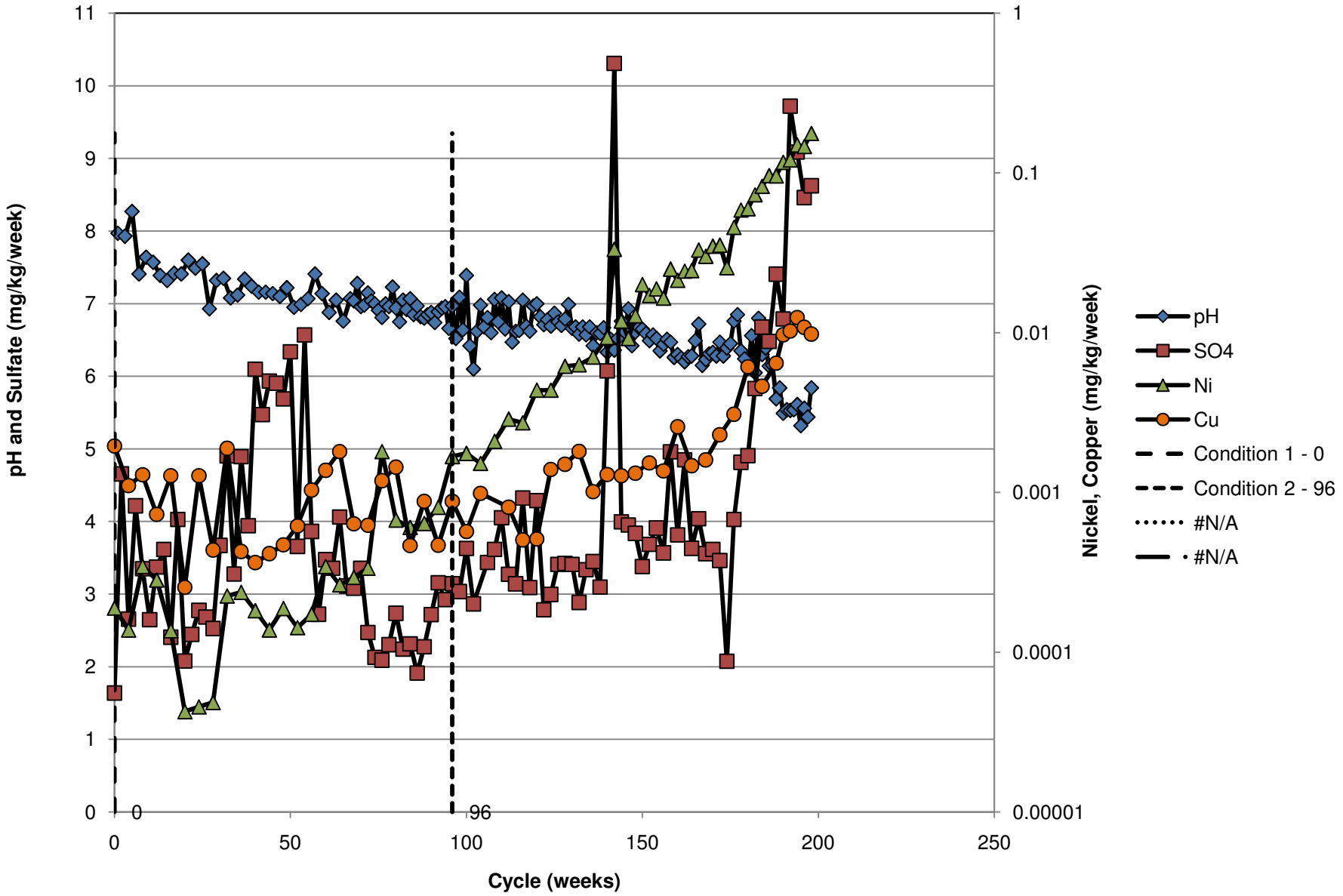
Waste Category:	4
Rock Type:	Ultramafic
Geology:	Duluth Complex
Sulfur:	0.72%

99-318C(725-735)



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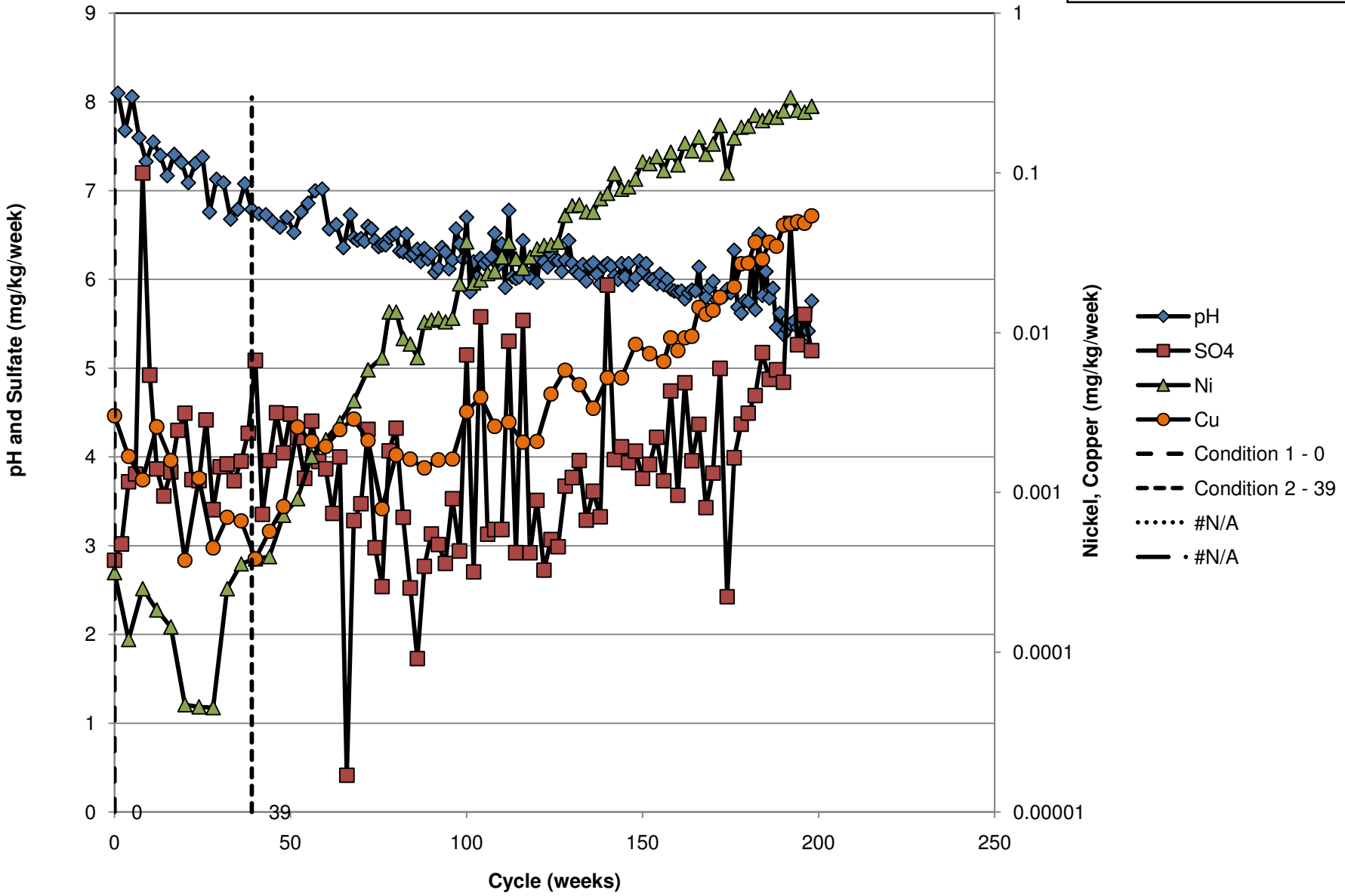
Waste Category:	4
Rock Type:	Ultramafic
Geology:	Duluth Complex
Sulfur:	1.24%

99-317C(460-470)



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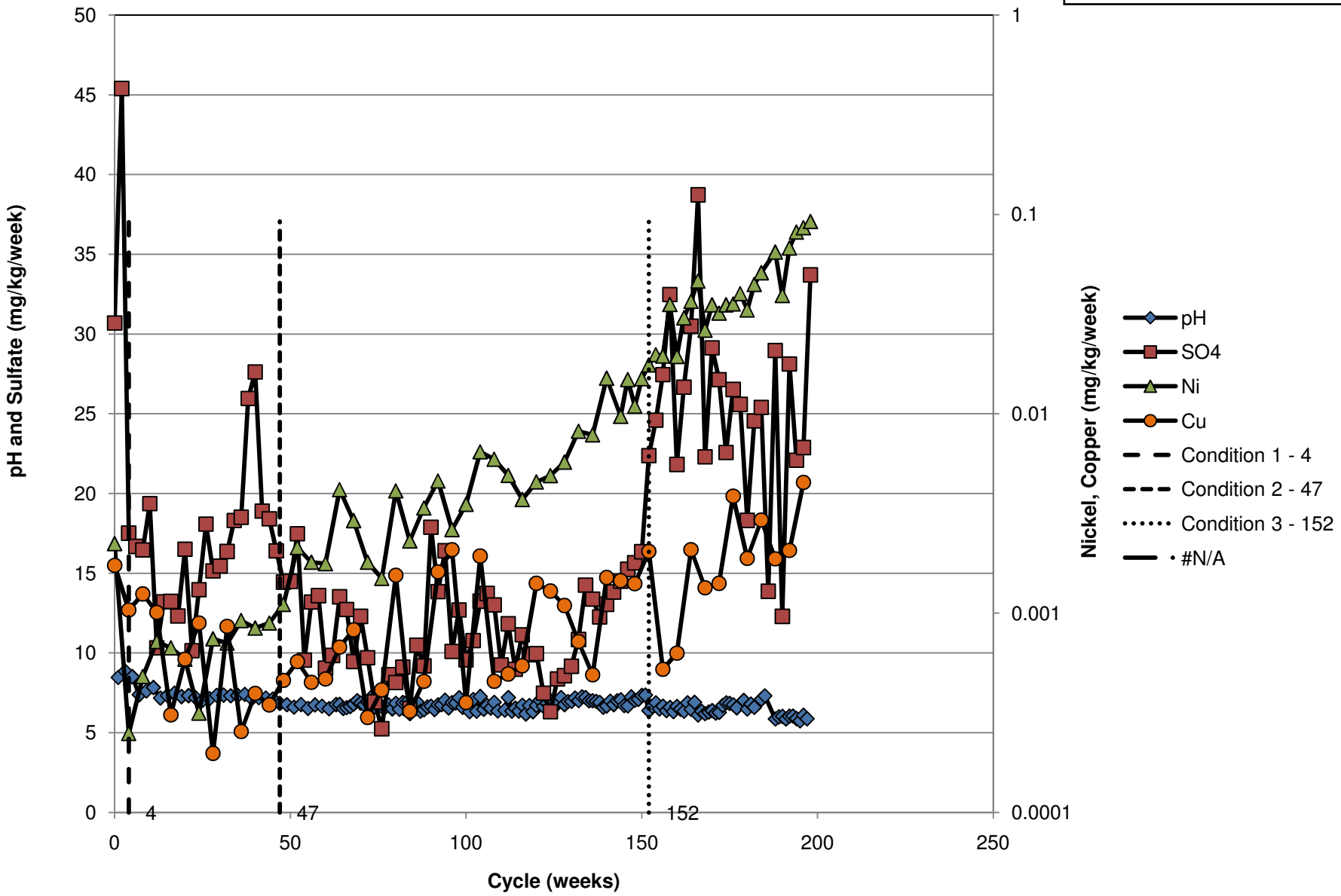
Waste Category:	4
Rock Type:	Ultramafic
Geology:	Duluth Complex
Sulfur:	1.20%

00-344C(515-520)



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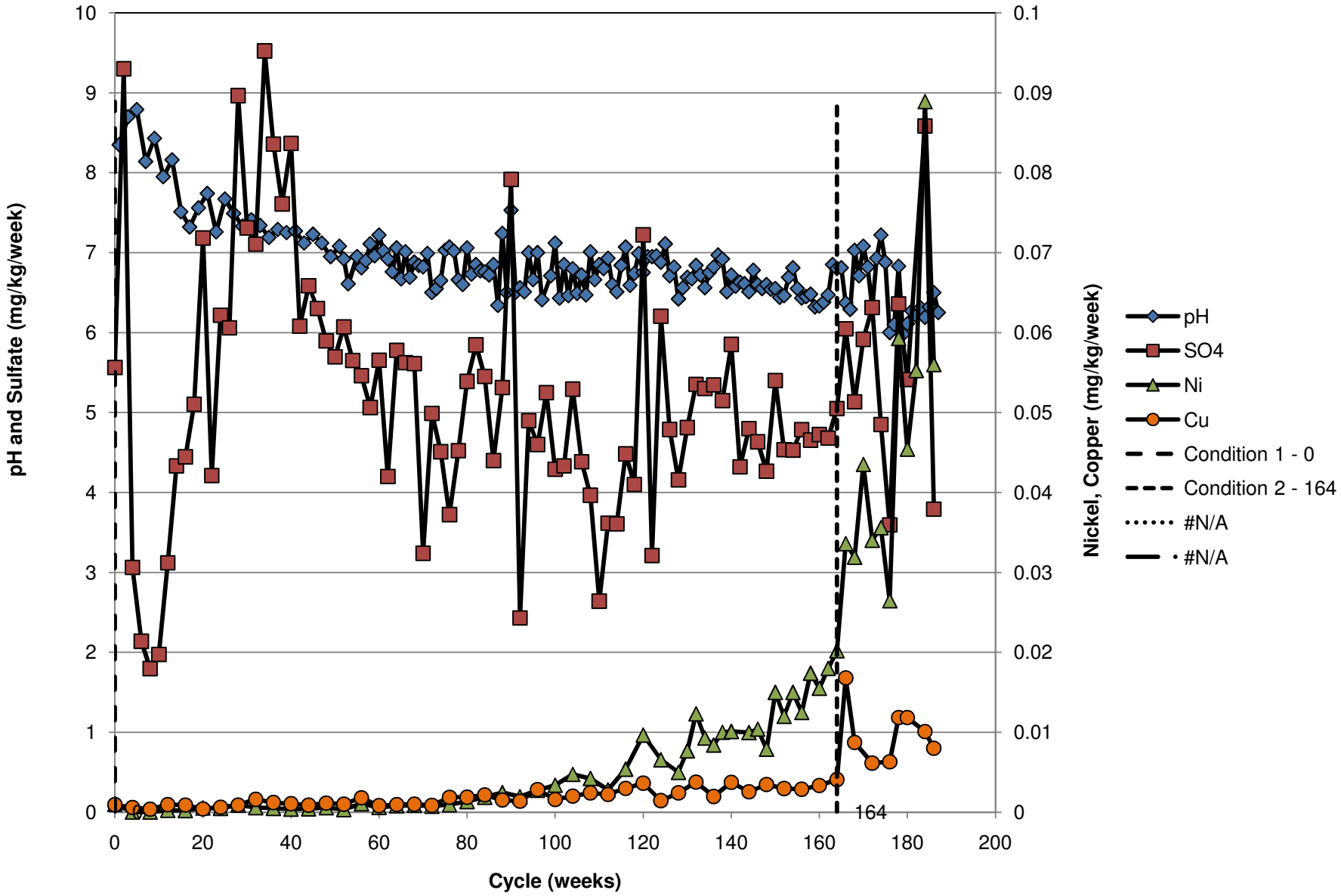
Waste Category:	4
Rock Type:	Ultramafic
Geology:	Duluth Complex
Sulfur:	0.75%

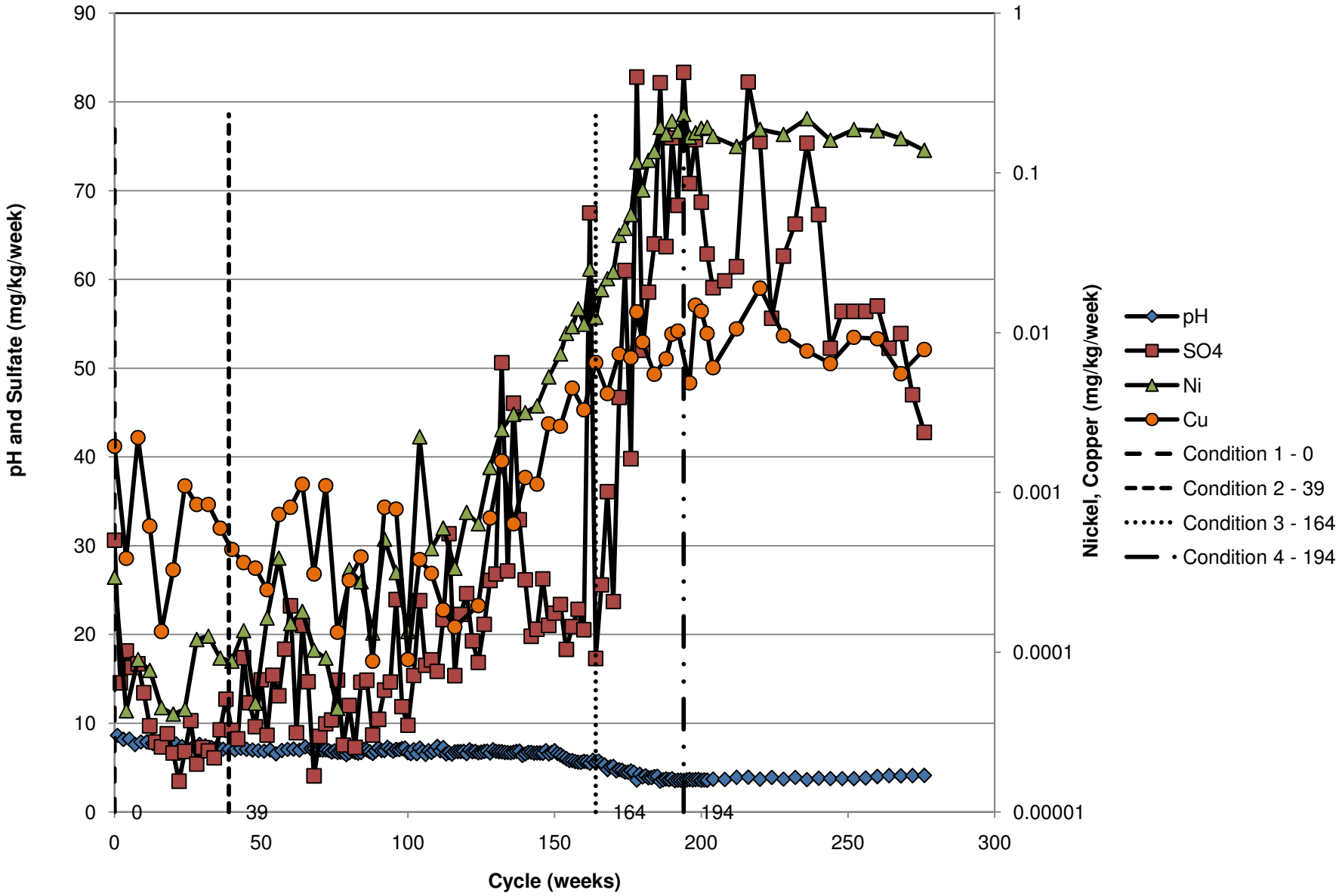


\\Van-svr0.van.na.srk.ad\ge_projects\PolyMet

d. Category 4 (Virginia Formation)

Waste Category:	4
Rock Type:	Virginia
Geology:	Virginia Formation
Sulfur:	2.00%

00-361C(737-749)



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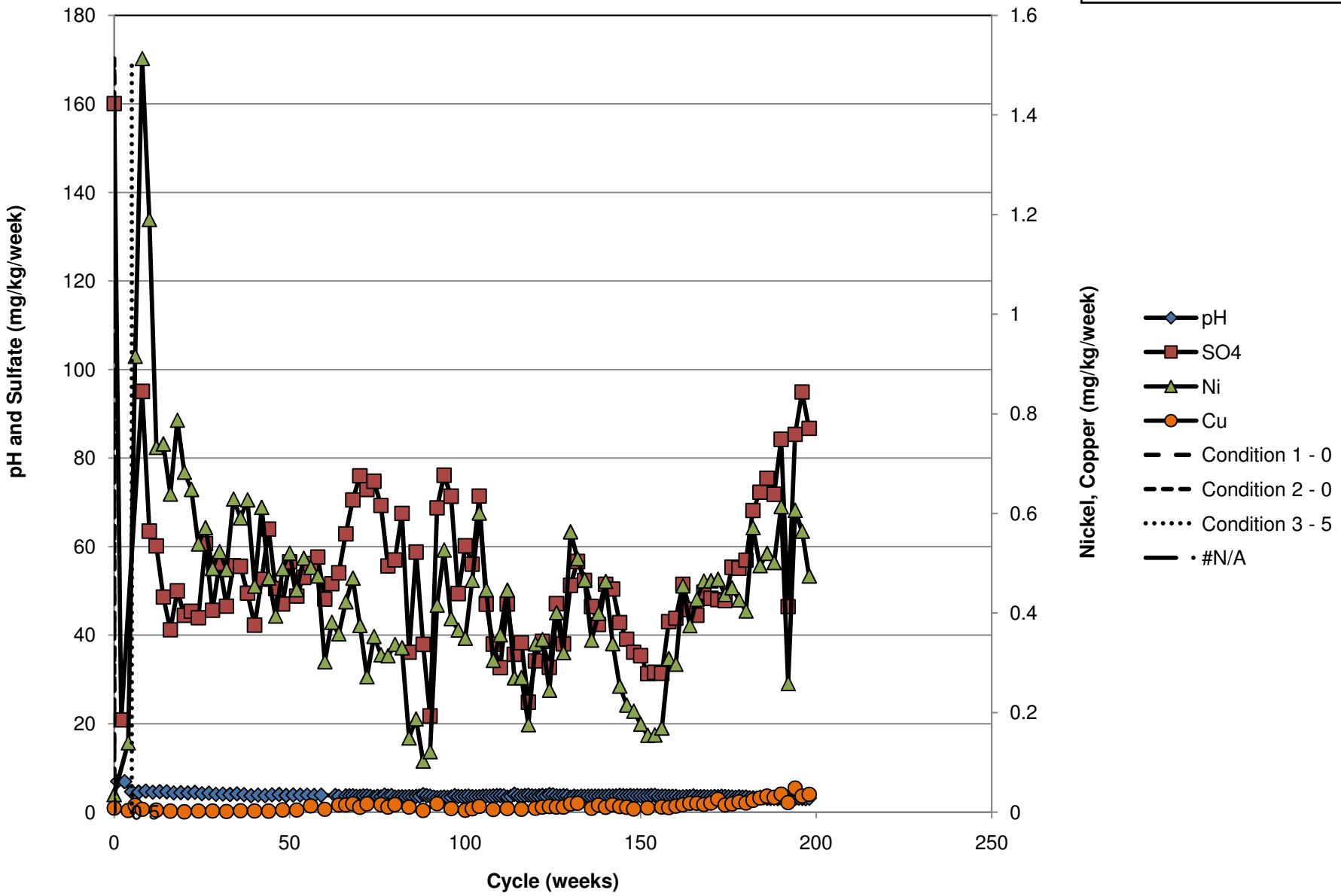
Waste Category:	4
Rock Type:	Virginia
Geology:	Virginia Formation
Sulfur:	3.79%

00-364C(210-229)



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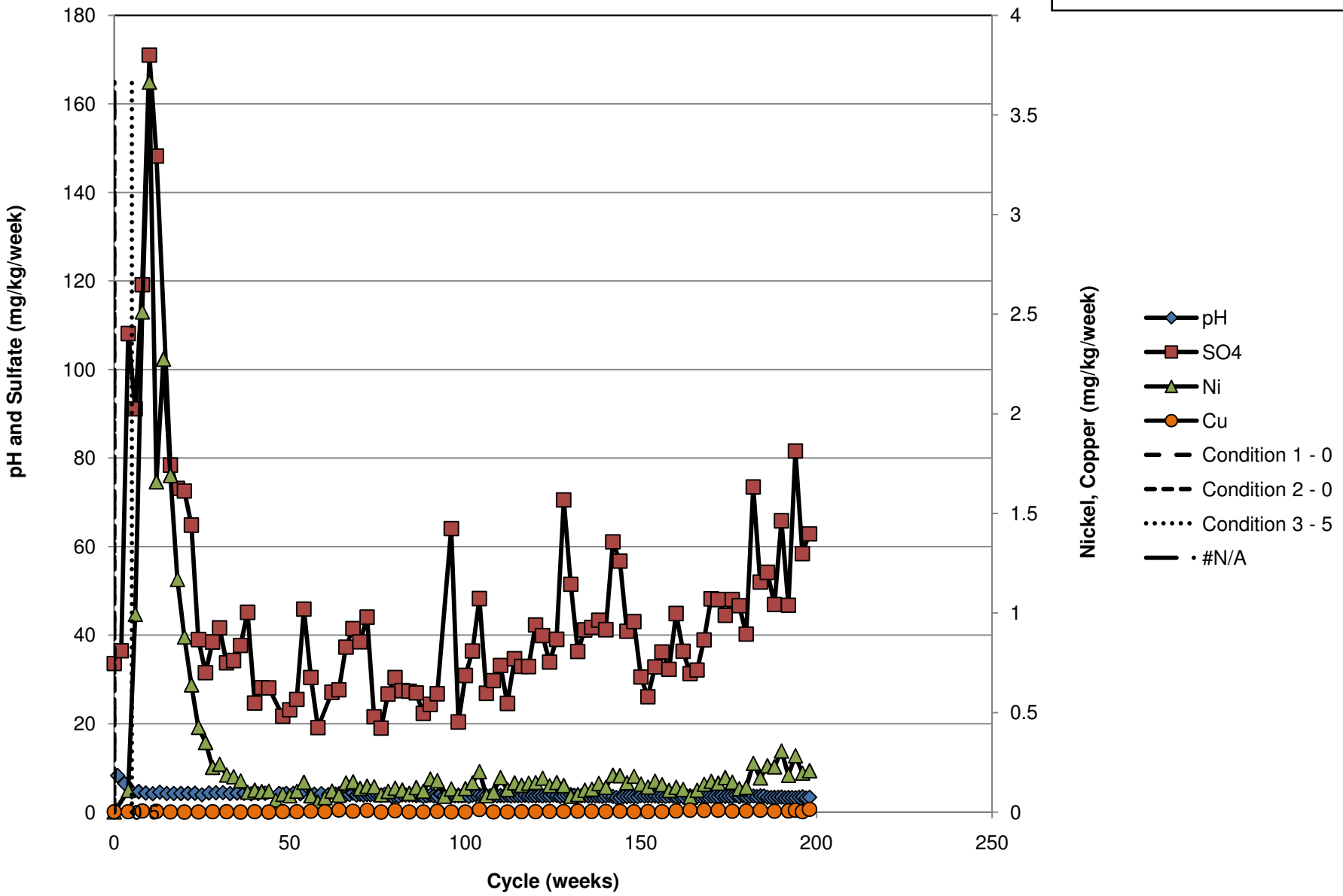
Waste Category:	4
Rock Type:	Virginia
Geology:	Virginia Formation
Sulfur:	5.68%

00-337C(510-520)



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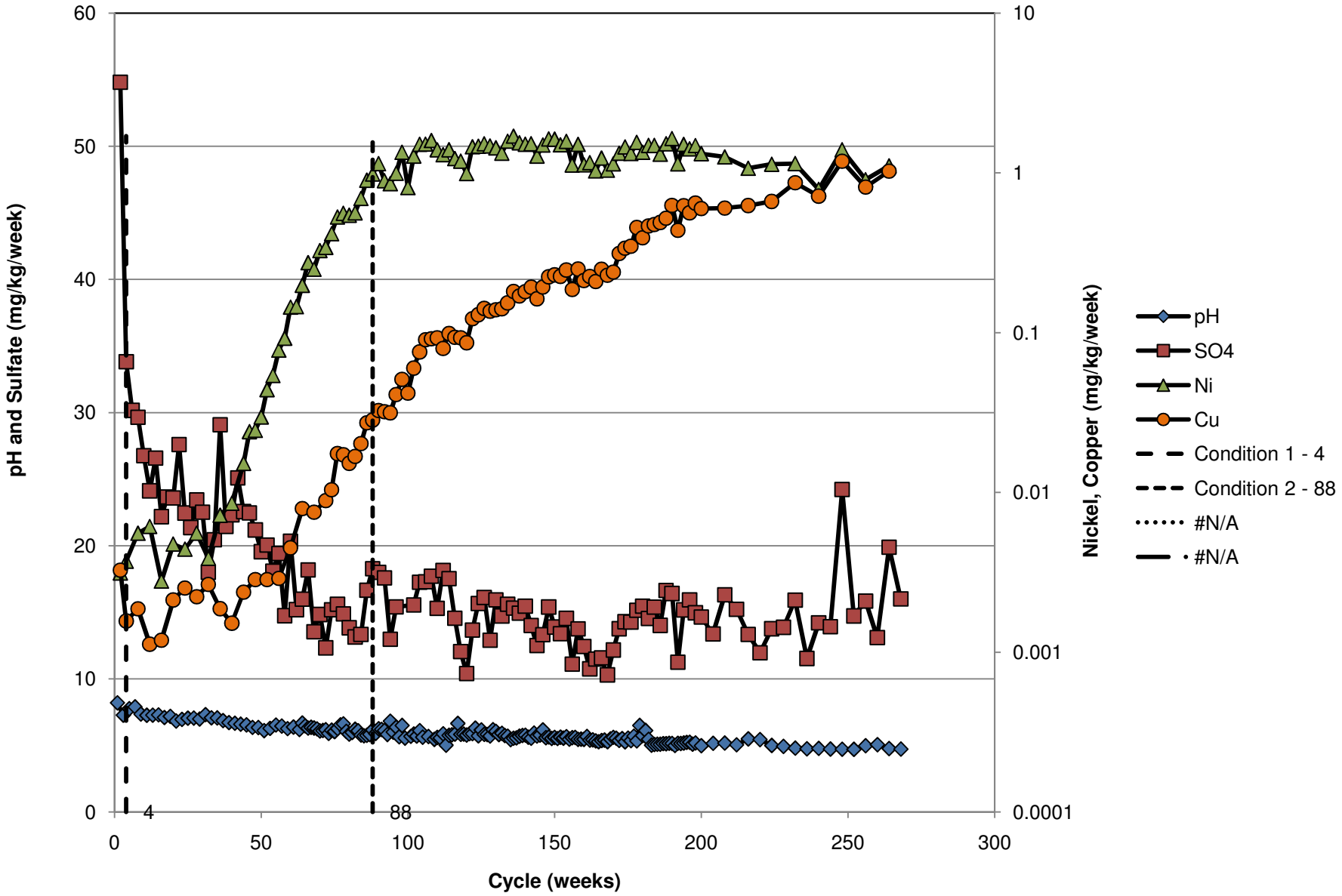
e. Ore Composites

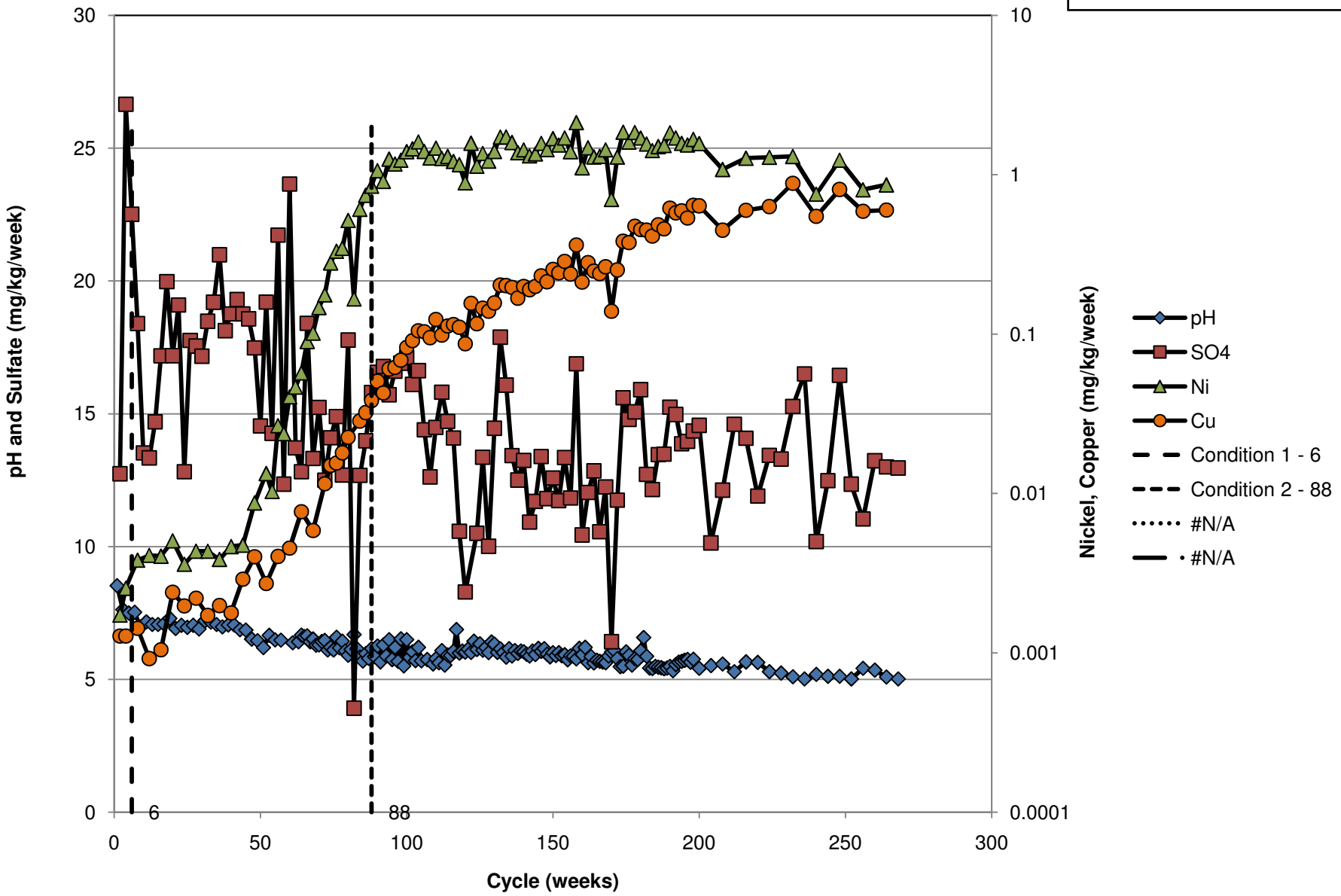
Waste Category:	Ore
Rock Type:	Ore Composite
Geology:	Duluth Complex
Sulfur:	0.86%



\\Van-svr0.van.na.srk.ad\ge_projects\PolyMet

Waste Category:	Ore
Rock Type:	Ore Composite
Geology:	Duluth Complex
Sulfur:	0.90%

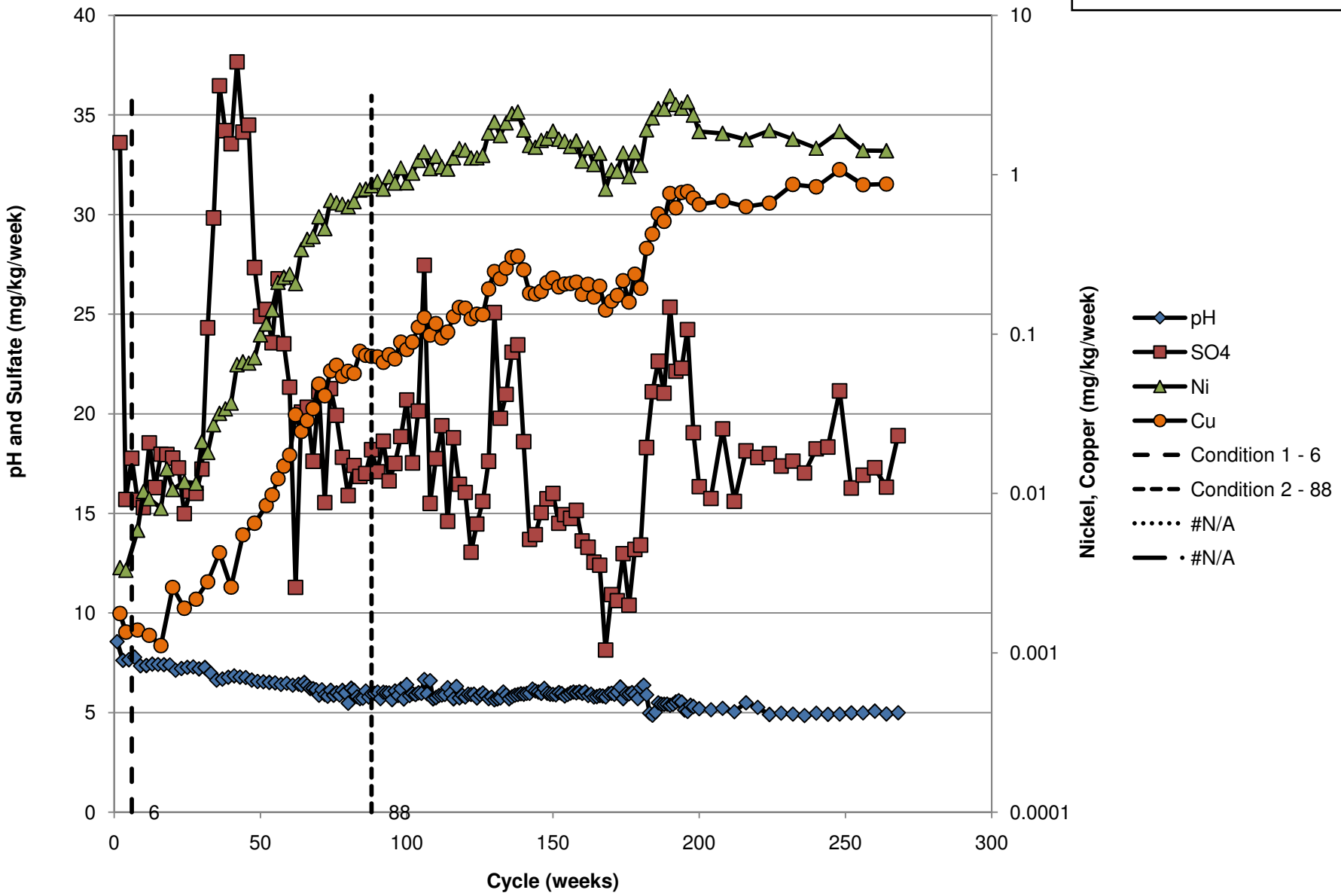
P20



\\Van-svr0.van.na.srk.ad\ge_projects\PolyMet

Waste Category:	Ore
Rock Type:	Ore Composite
Geology:	Duluth Complex
Sulfur:	0.86%

P30



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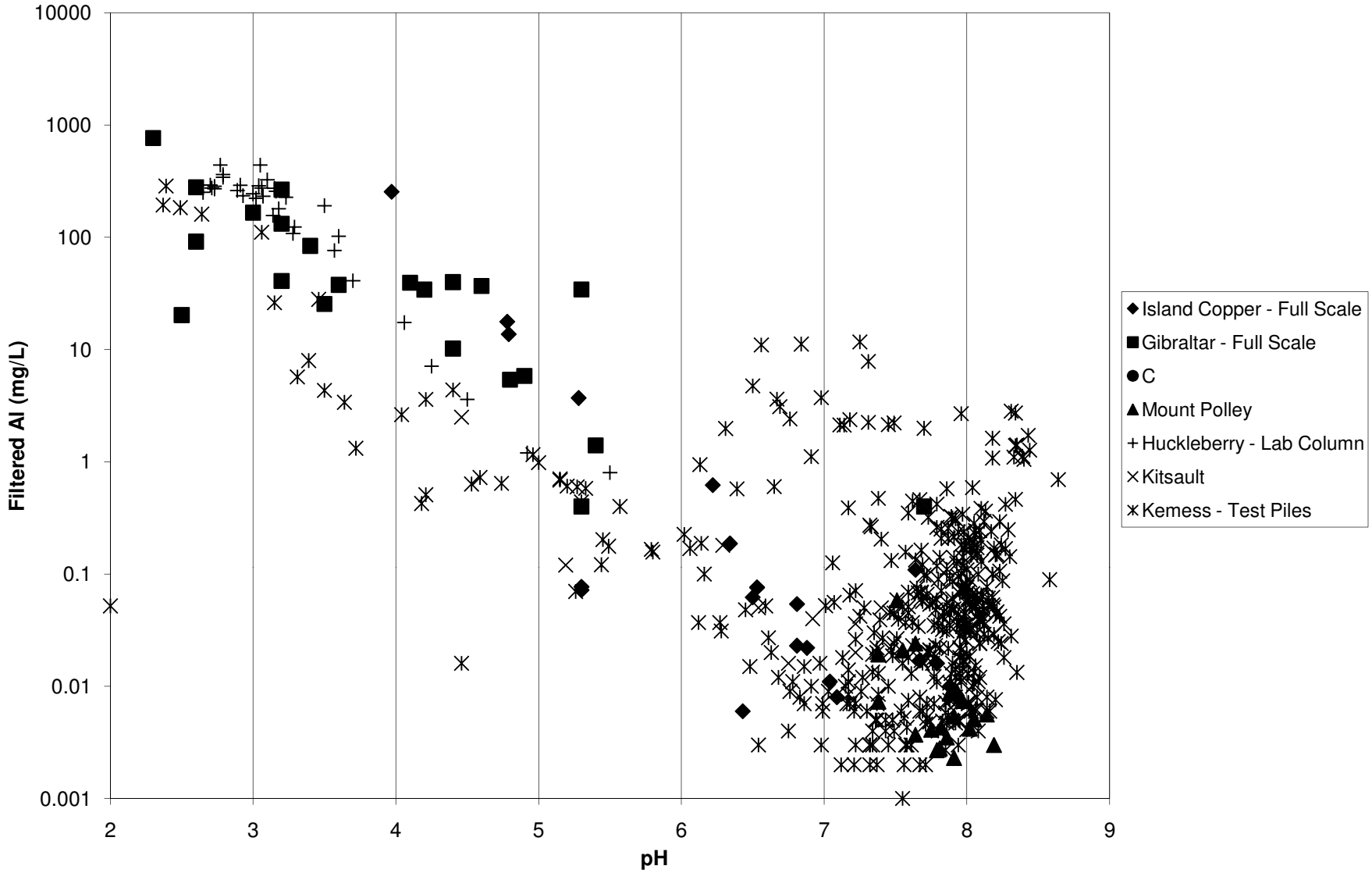
Attachment 3
Porphyry Database



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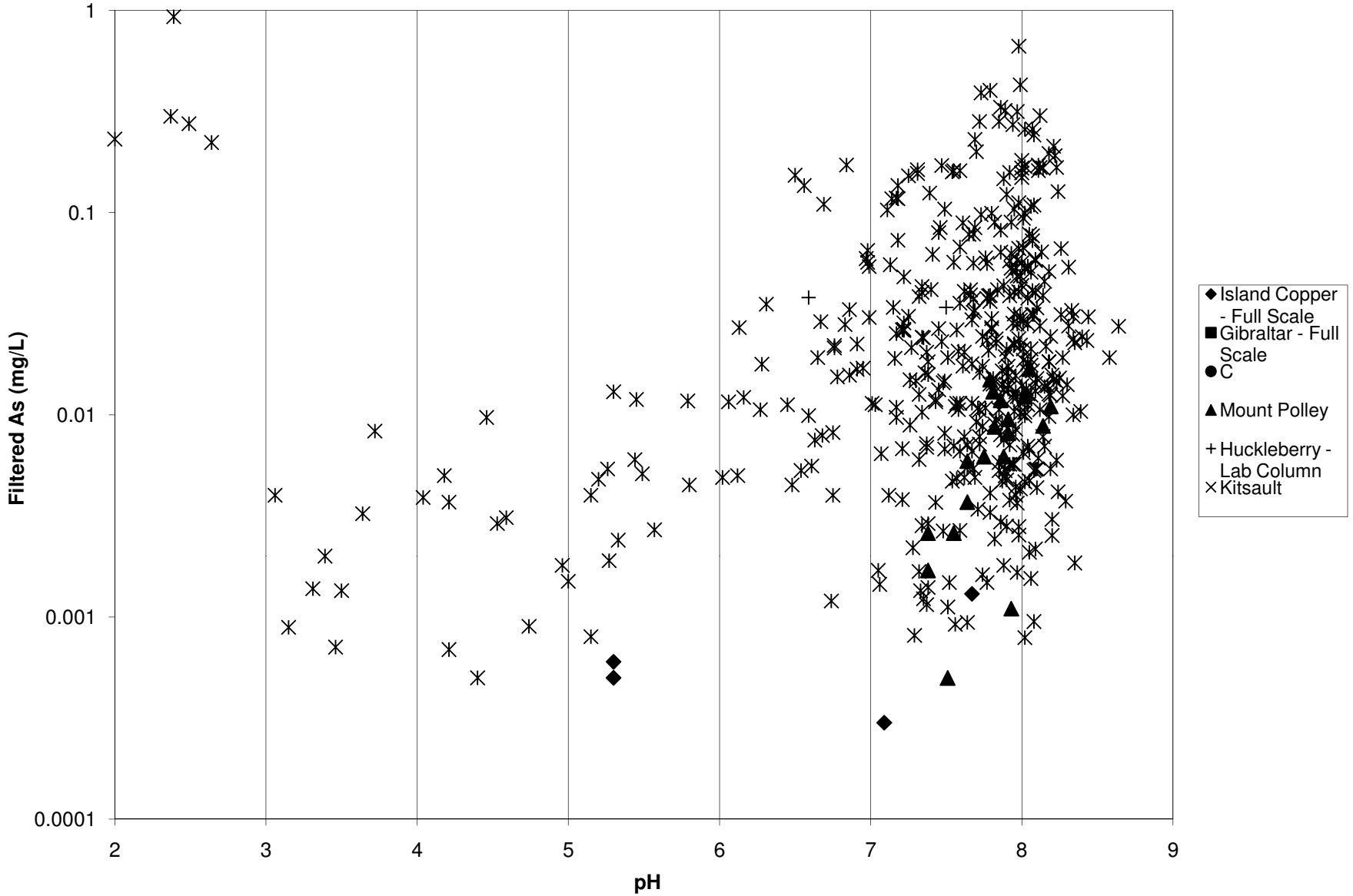
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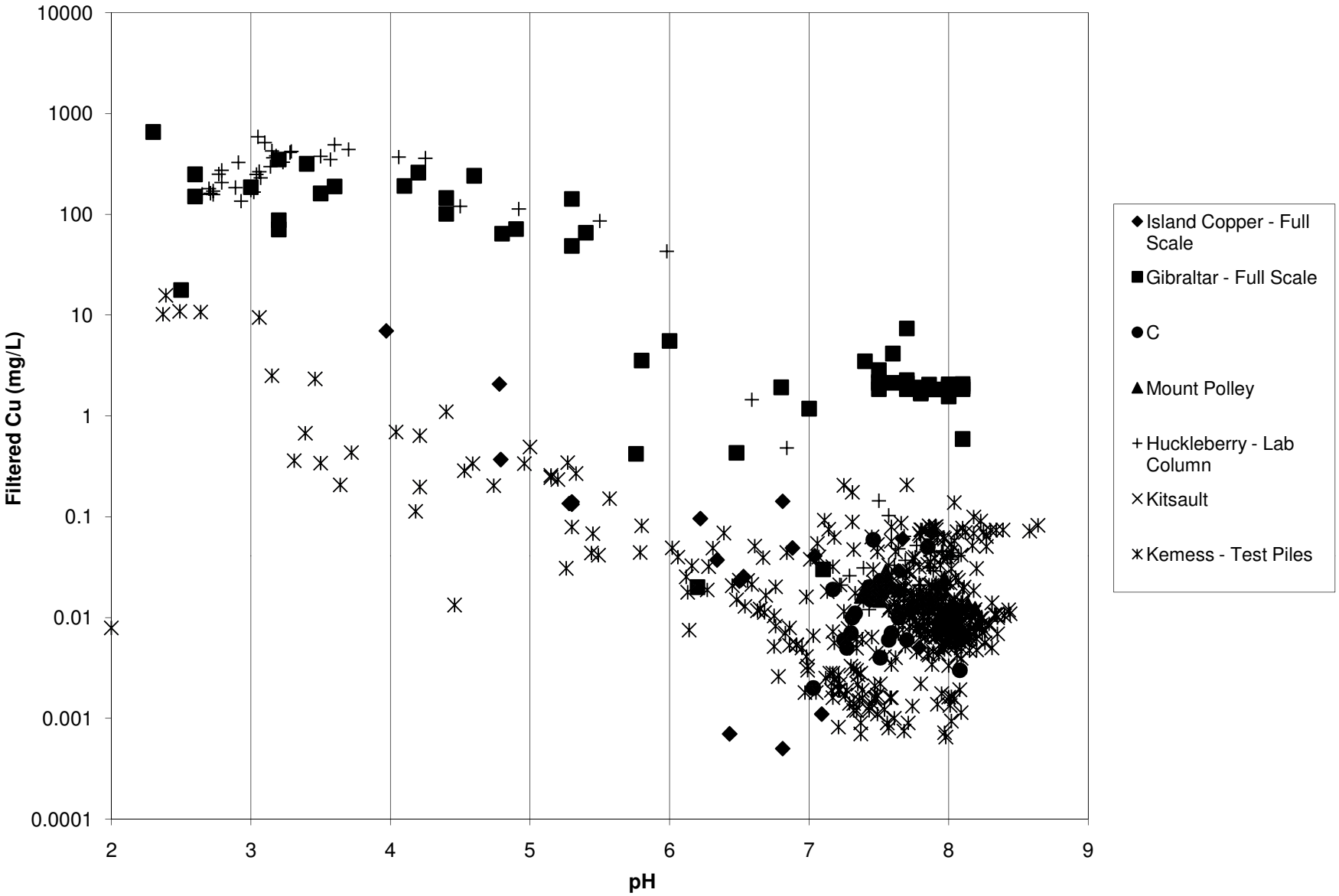
pH
Island Copper - Full Scale
Gibraltar - Full Scale
C
Mount Polley
Huckleberry - Lab Column
Kitsault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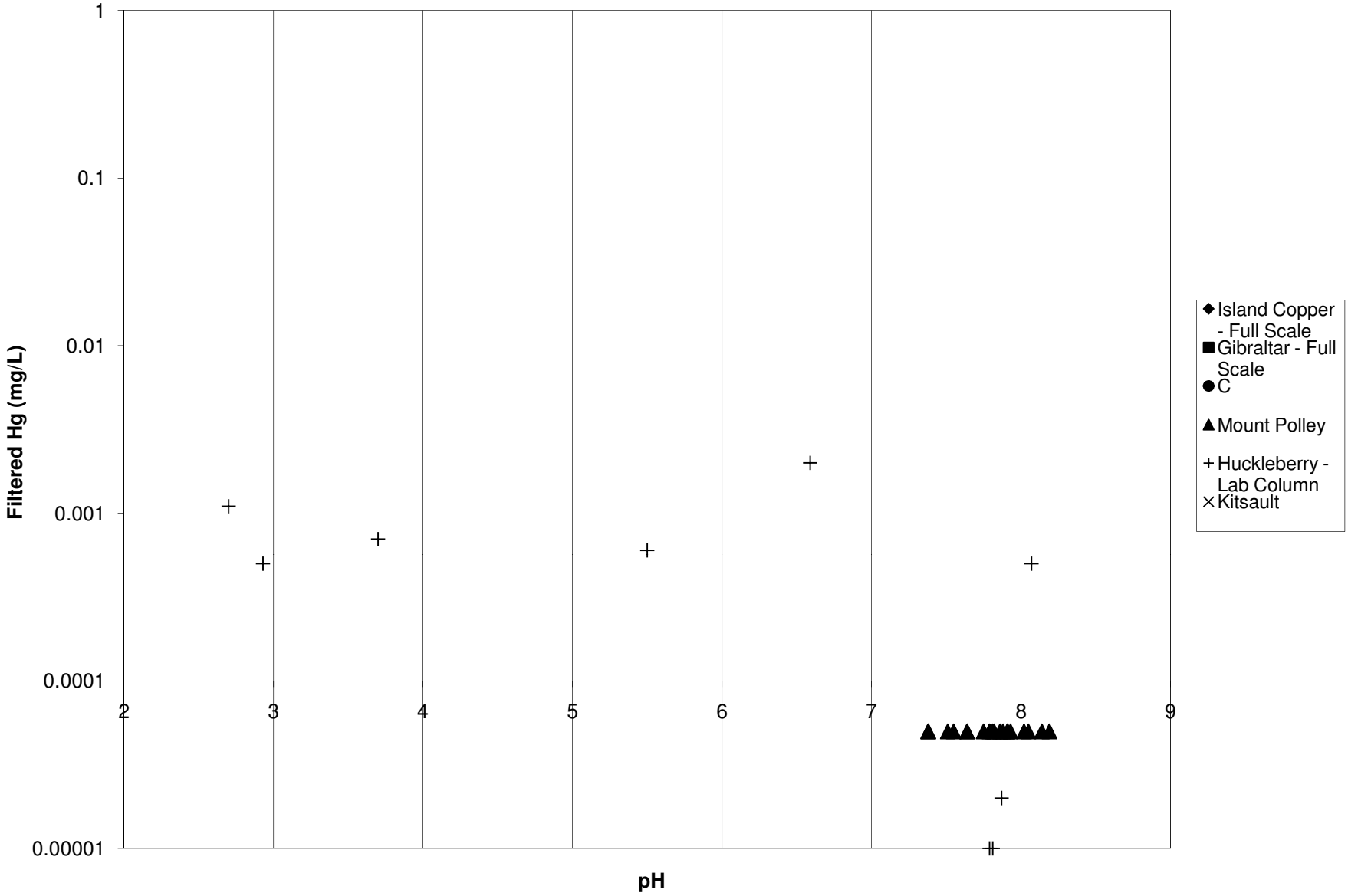
Mount Polley
Huckleberry - Lab Column
Kitsault
Kemess - Test Piles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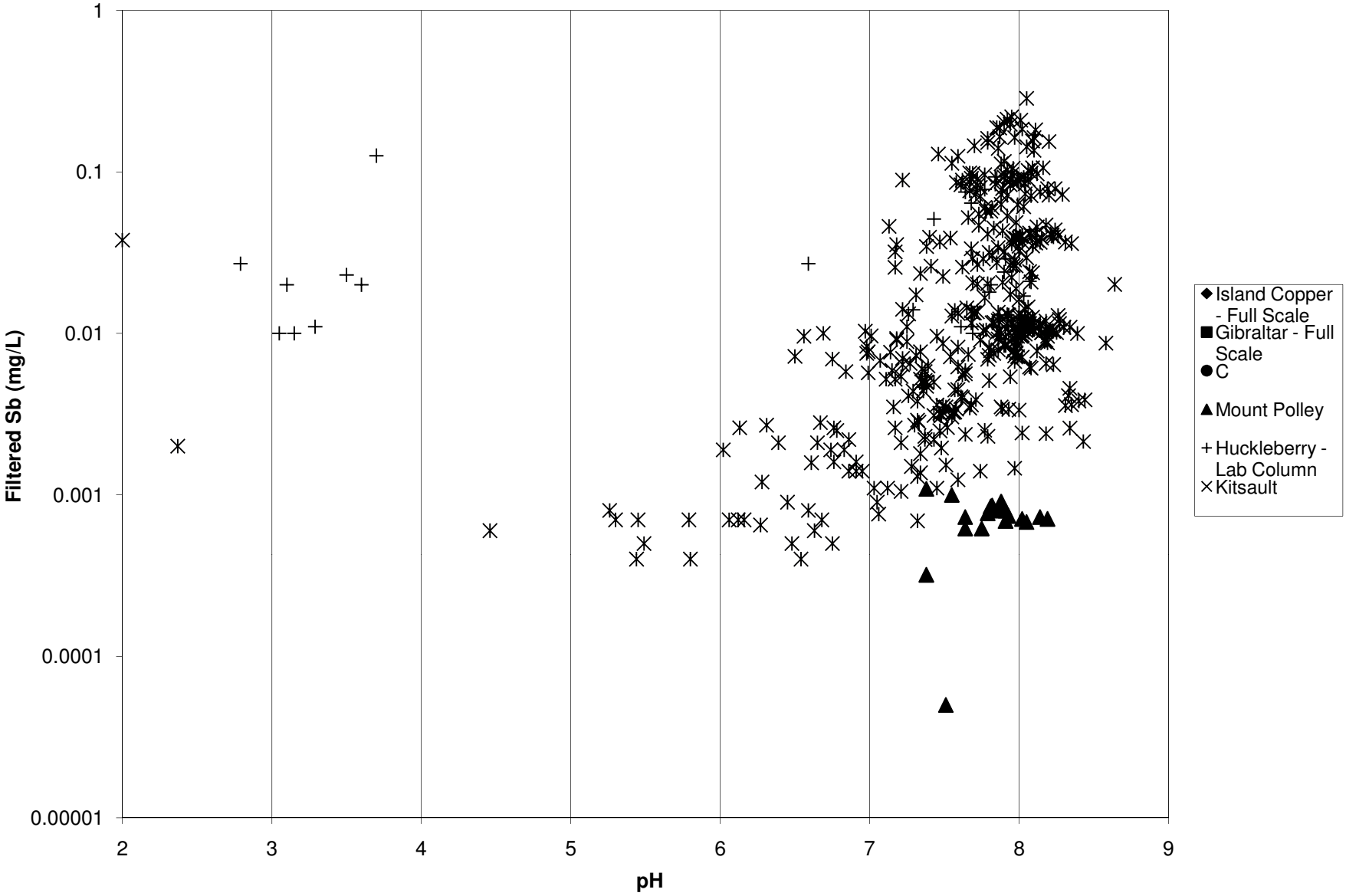
C
Mount Polley
Huckleberry - Lab Column
Kitsault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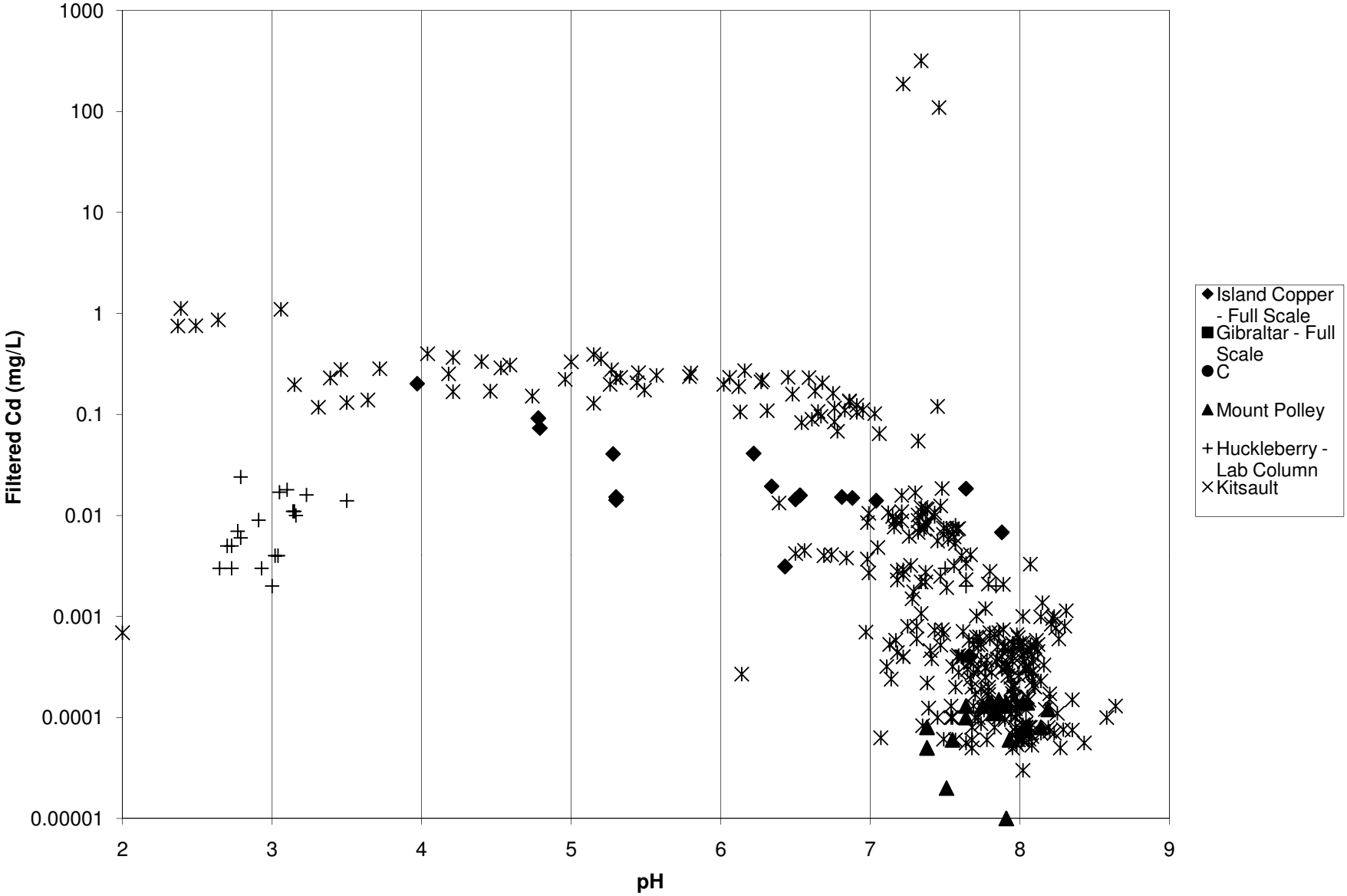
Huckleberry - Lab Column
Kitsault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Huckleberry - Lab Column
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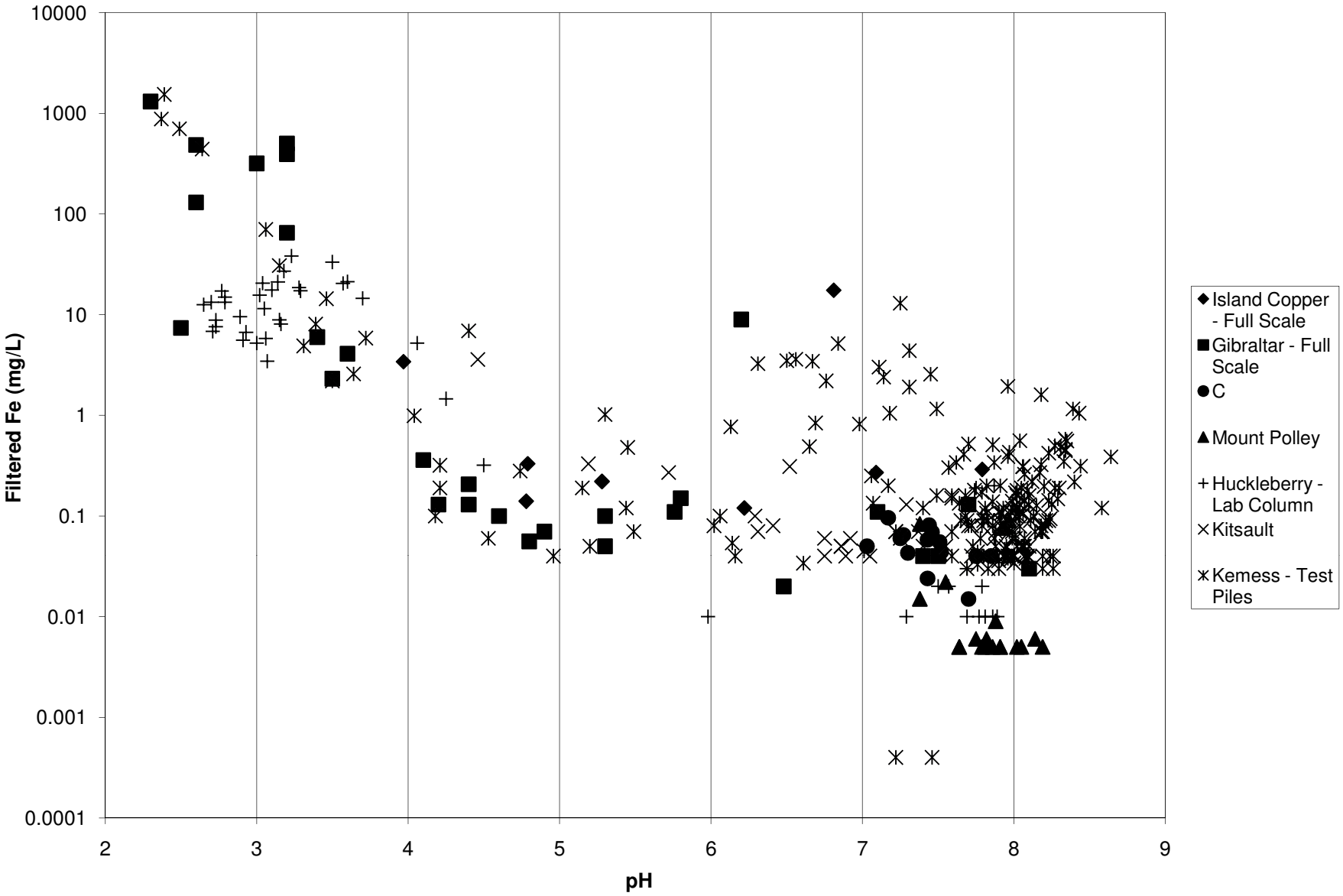
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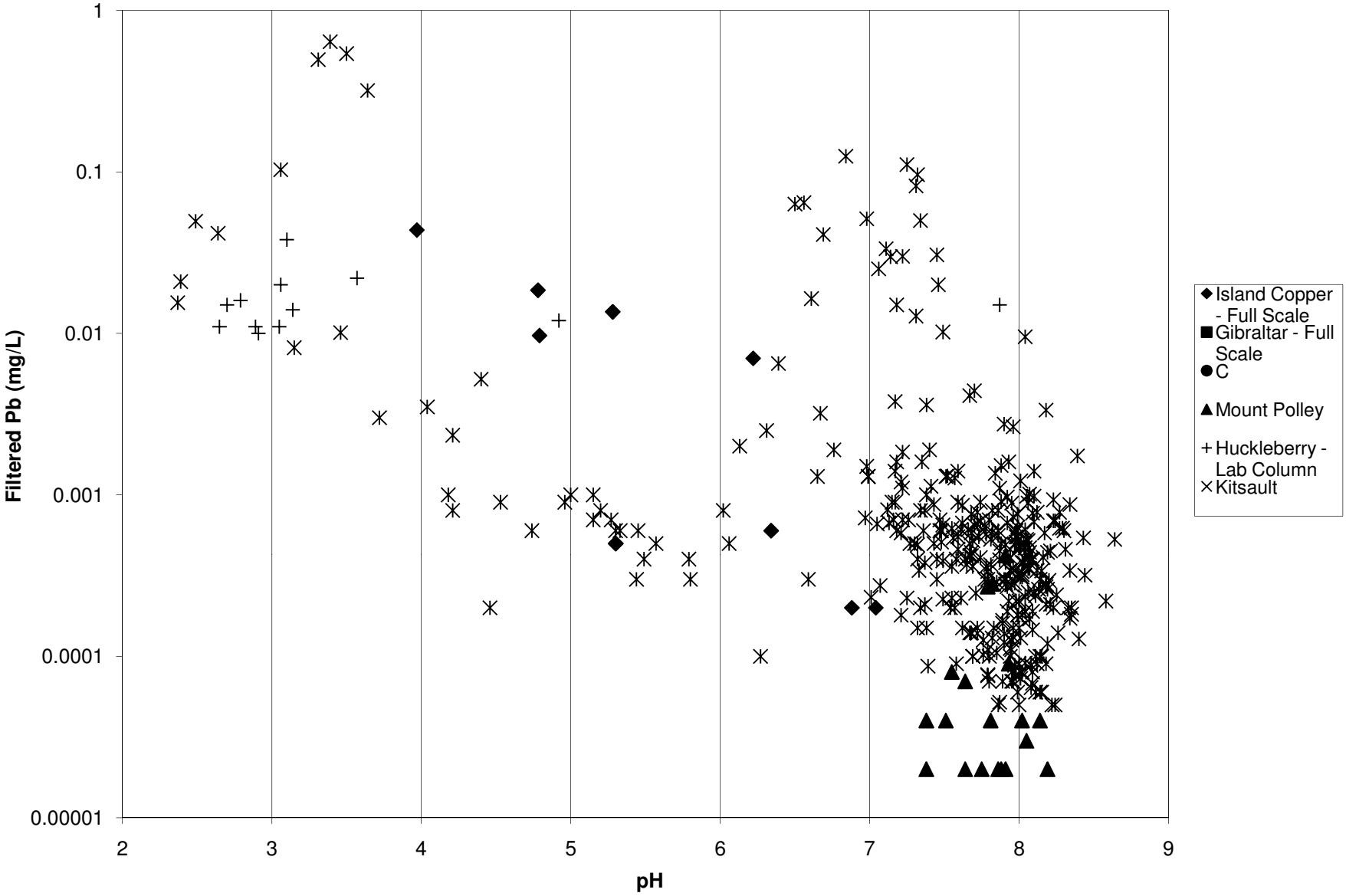
Huckleberry - Lab Column
Kitsault
Kemess - Test Piles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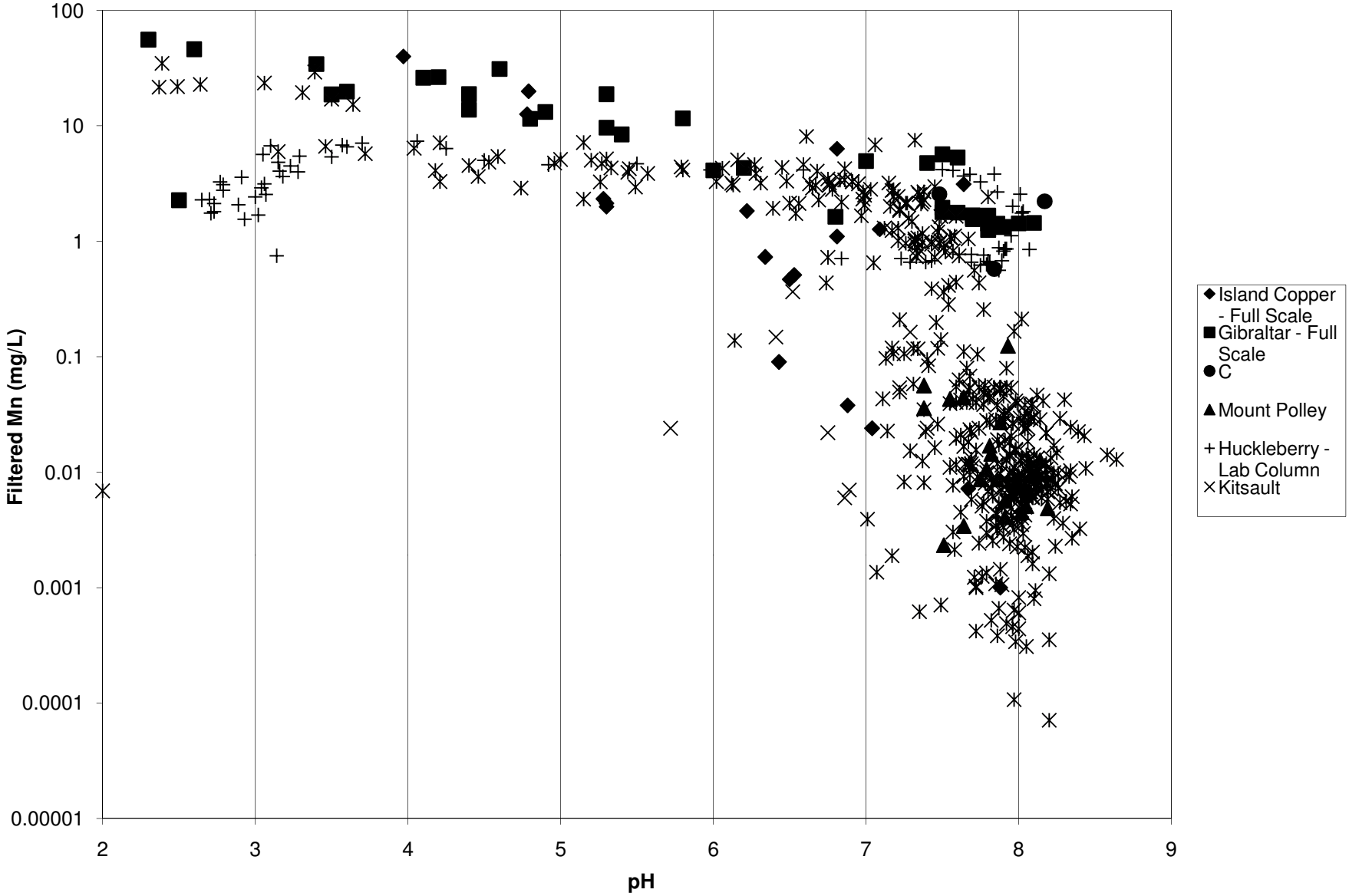
Gibraltar - Full Scale
C
Mount Polley
Huckleberry - Lab Column
Kitsault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Gibraltar - Full Scale
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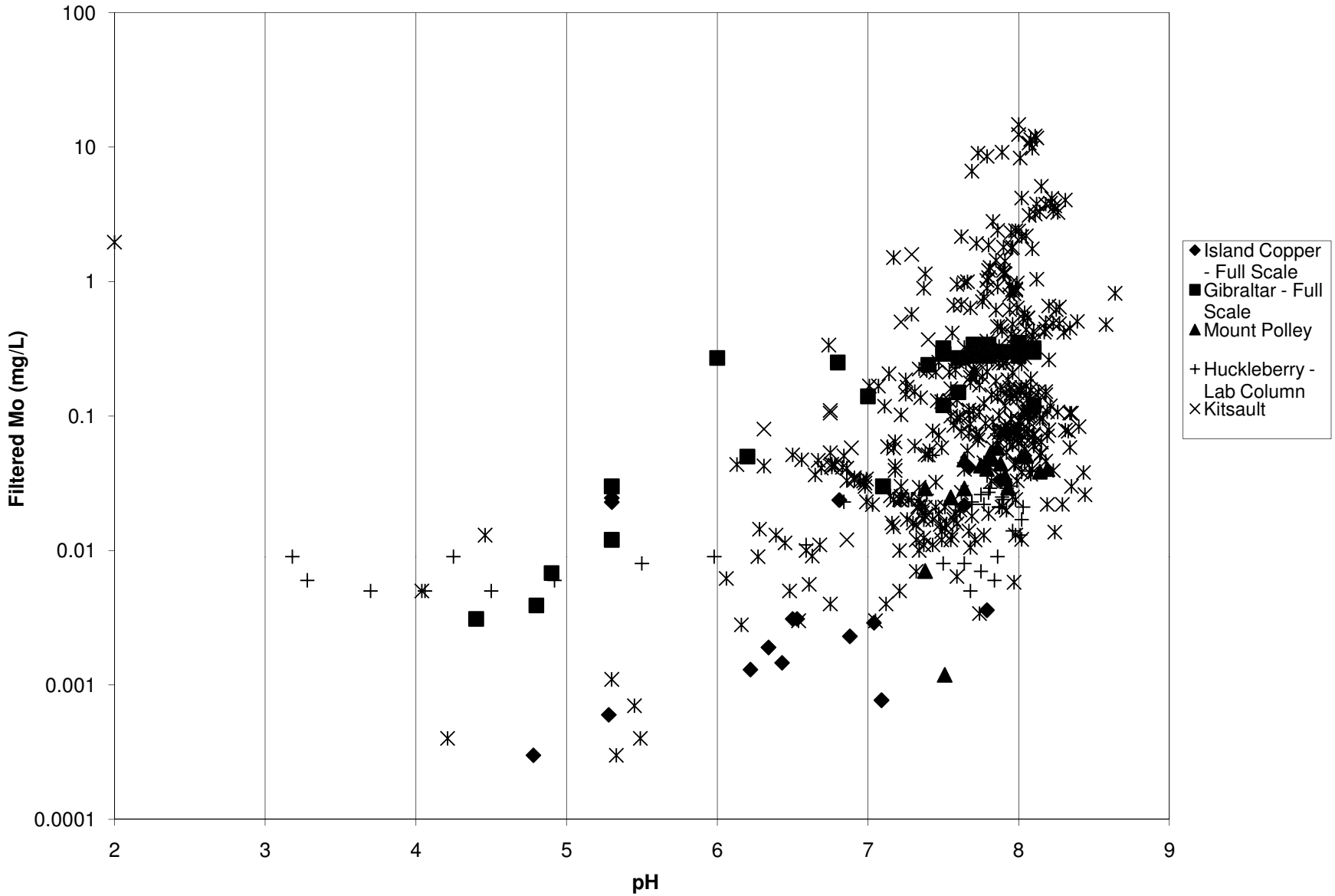
Mount Polley
Huckleberry - Lab Column
Kitsault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Mount Polley
Huckleberry - Lab Column
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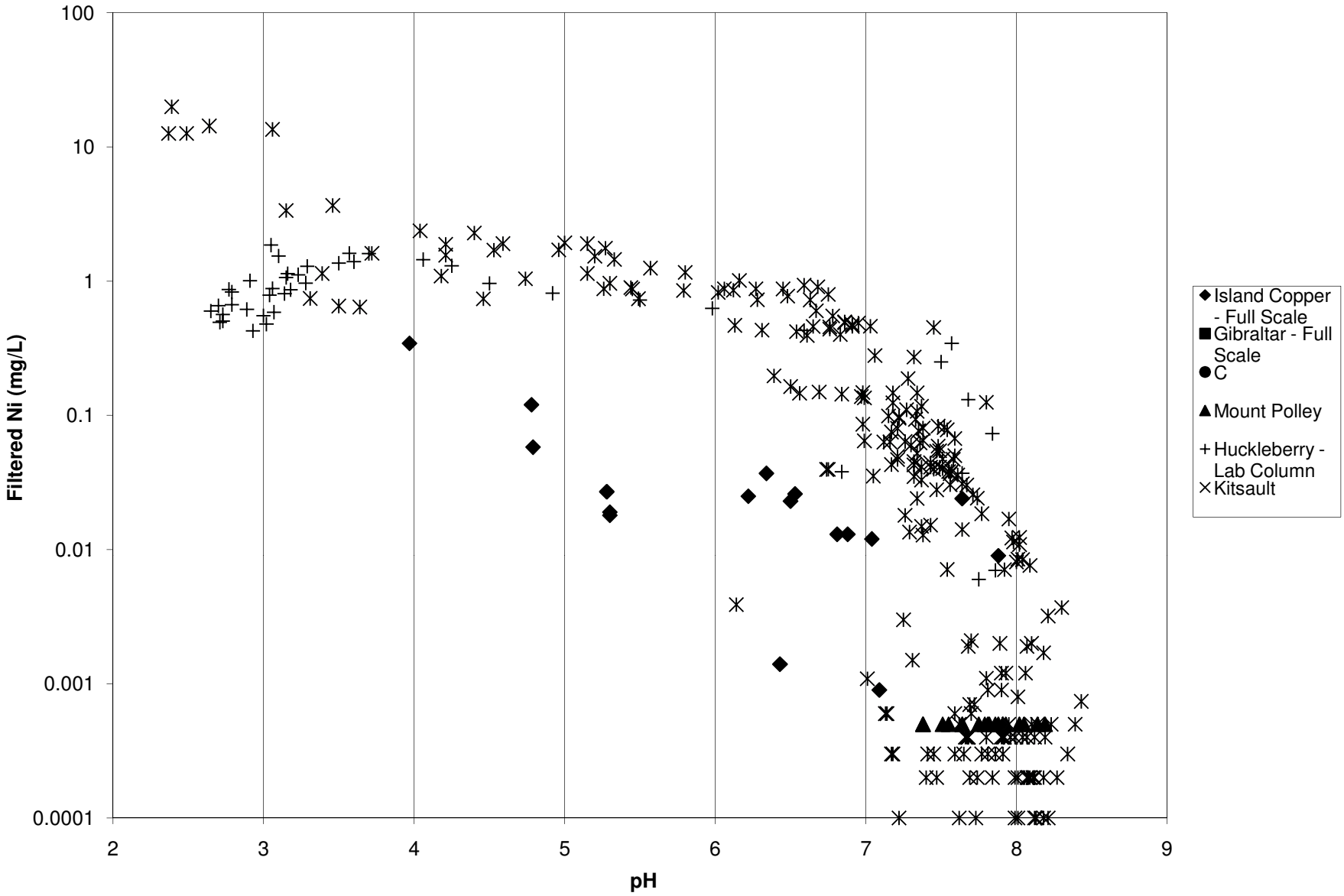
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Gibraltar - Full Scale
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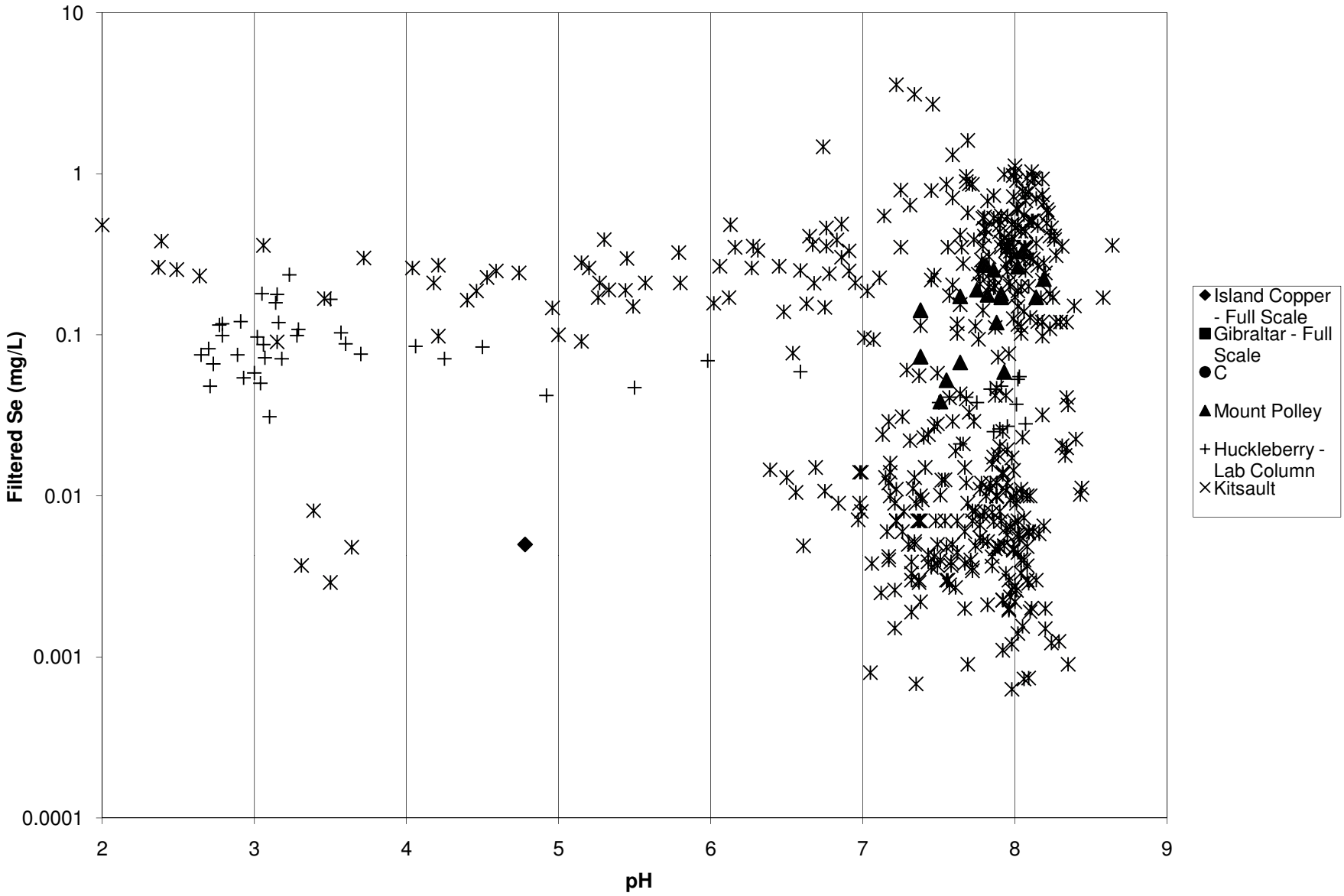
Mount Polley
Huckleberry - Lab Column
Kitsault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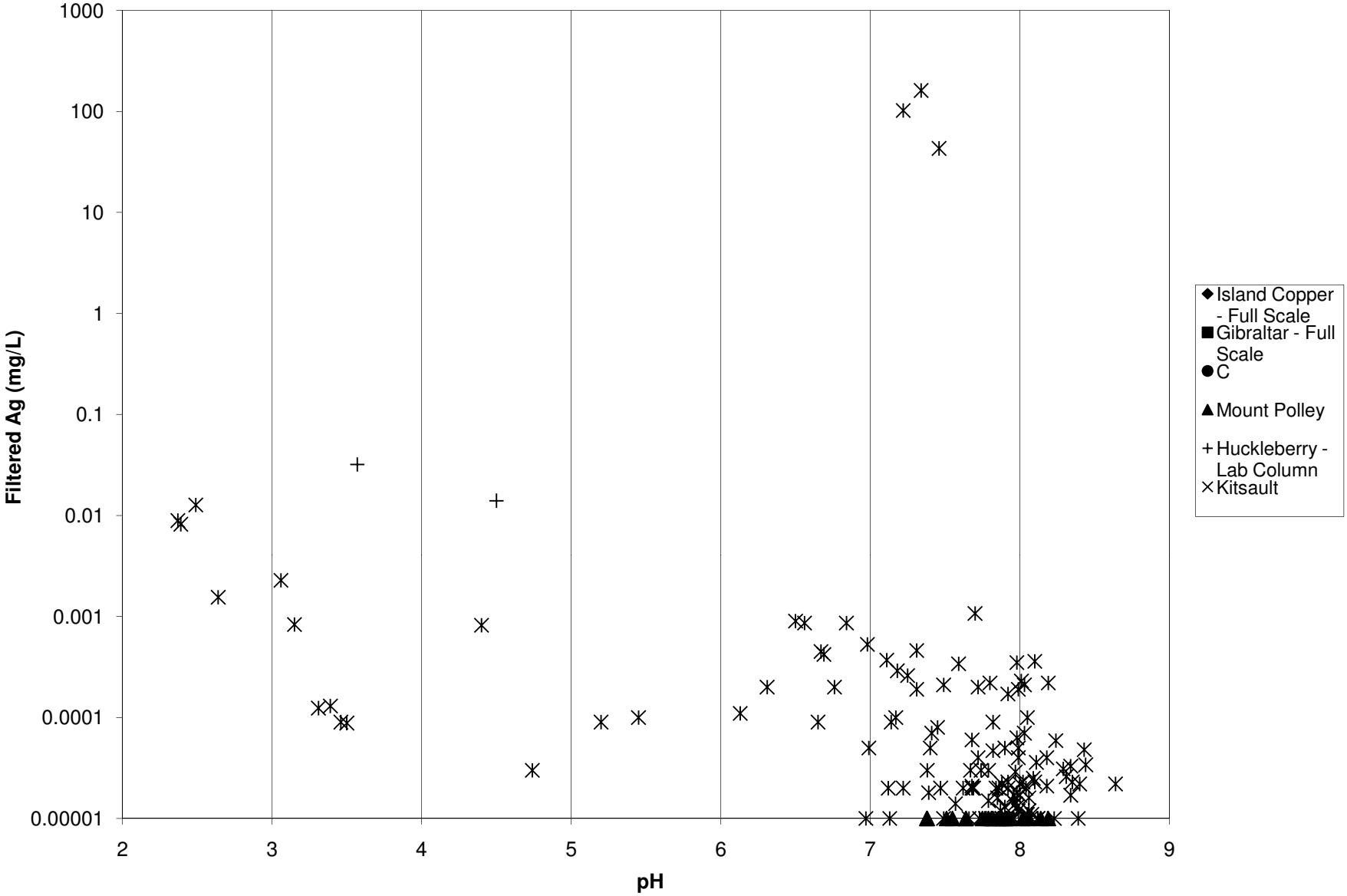
Gibraltar - Full Scale
C
Mount Polley
Huckleberry - Lab Column
Kitsault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C
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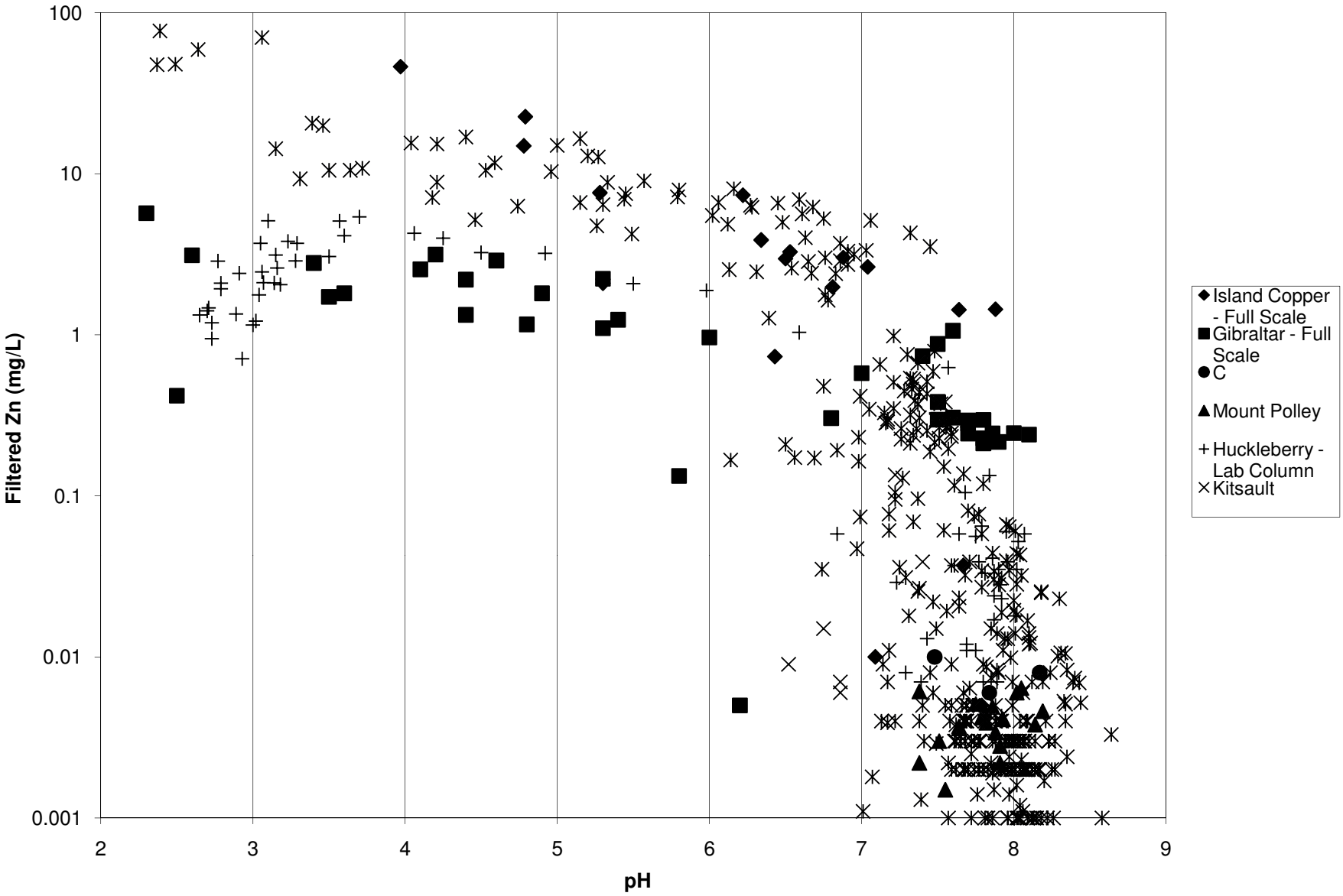
Huckleberry - Lab Column
Kitsault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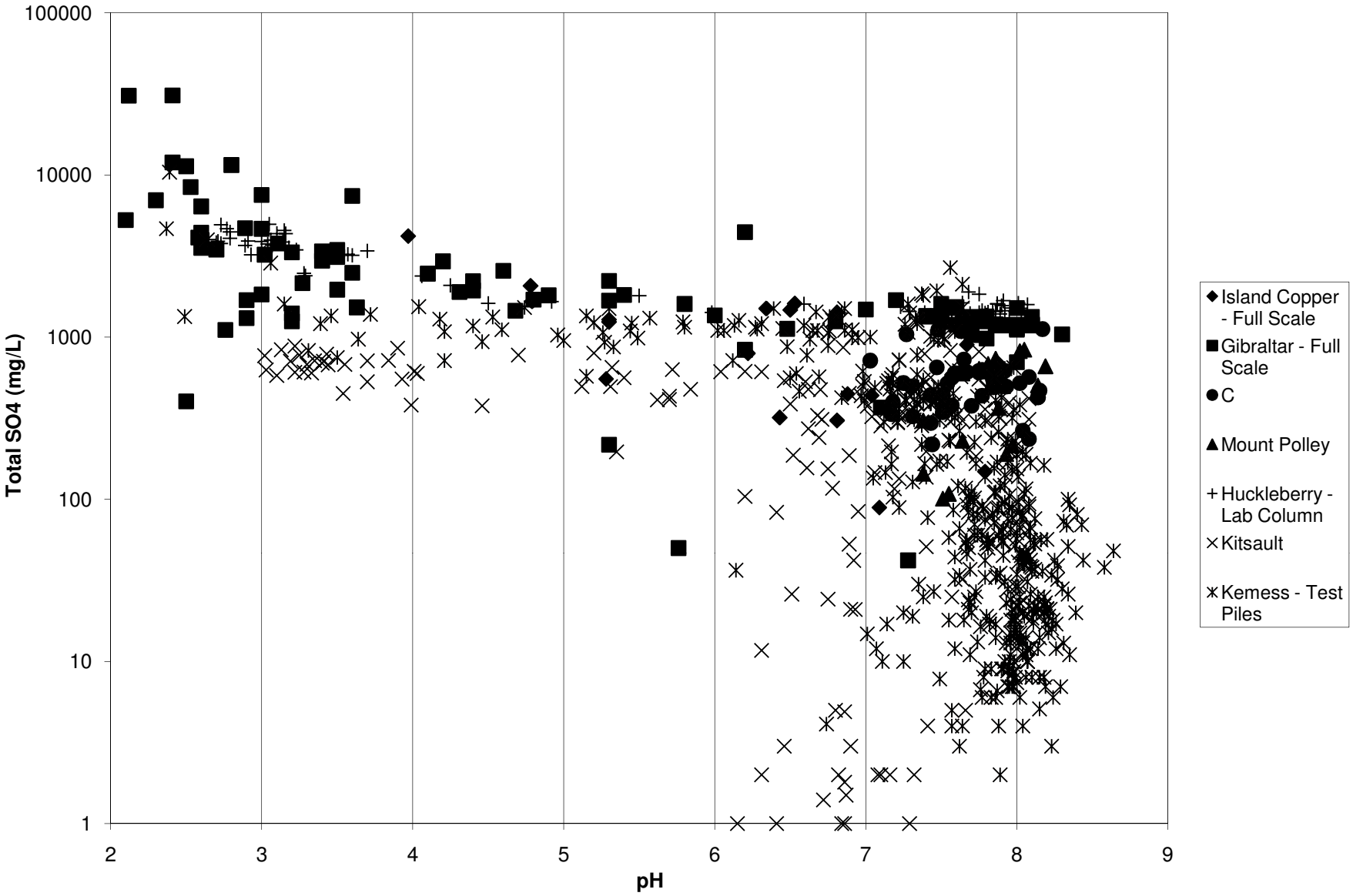
Huckleberry - Lab Column
Kitsault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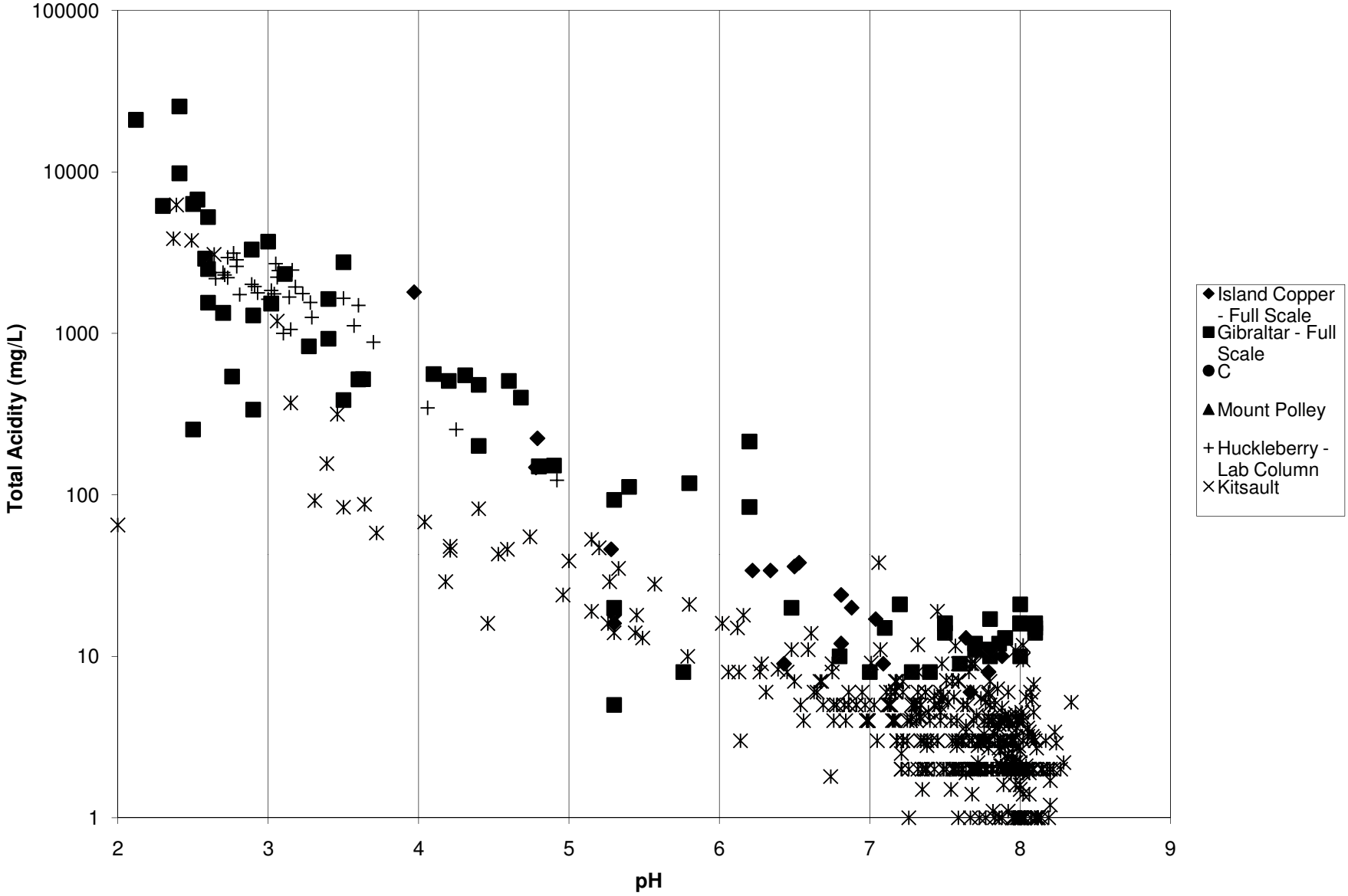
Huckleberry - Lab Column
Kitsault
Kemess - Test Piles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C
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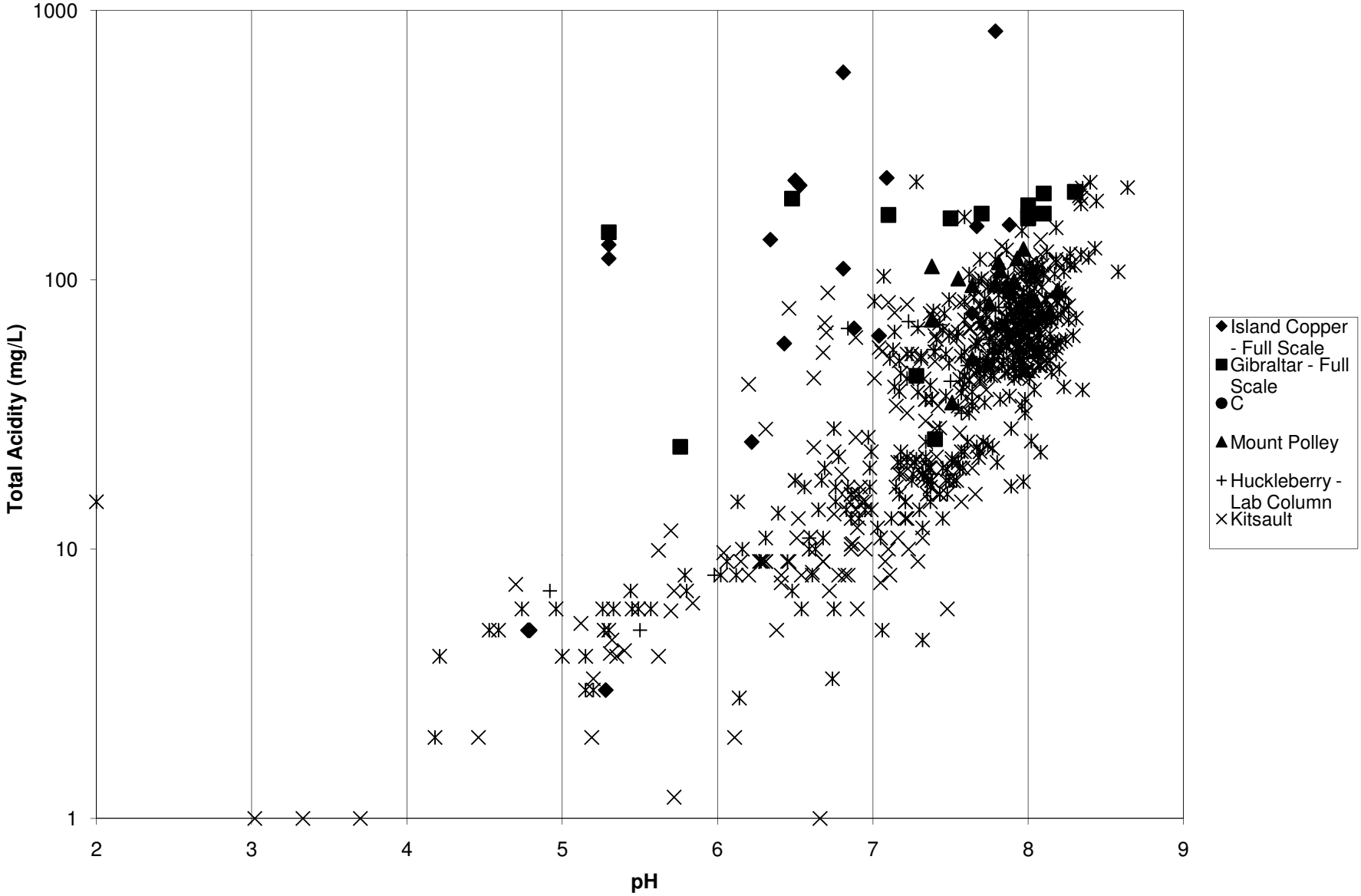
Huckleberry - Lab Column
Kitsault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pH
Island Copper - Full Scale
Gibraltar - Full Scale
C
Mount Polley
Huckleberry - Lab Column
Kitsault

Attachment 4
Whistle Mine Stockpile Water Chemistry Data

Sample ID	Sample Date	Lab	pH su	Eh mV	Conductivity uS	T oC	Alkalinity mgCaCO3/L	F mg/L	Cl mg/L	NO_2 mg/L	Br mg/L	NO_3 mg/L	PO_4^3 mg/L	SO_4 mg/L	Ag mg/L	Al mg/L	As mg/L	B mg/L	Ba mg/L	Be mg/L	Bi mg/L	Ca mg/L	Cd mg/L	Co mg/L	Cr mg/L	Cu mg/L	Fe mg/L	Ga mg/L	K mg/L	Li mg/L	Mg mg/L	Mn mg/L	Mo mg/L	Na mg/L	Ni mg/L	P mg/L	Pb mg/L	S mg/L	Sb mg/L	Se mg/L	Si mg/L	Sn mg/L	Sr mg/L	Ti mg/L	V mg/L	Y mg/L	Zn mg/L
951-1952 SUMP #1	16-Dec-96	Barringer Laboratories						<0.1	7	<0.2	<0.5	0.2	<1	1760	<0.005	42.4	<0.1	0.06	0.025	0.0009	<0.1	170	0.02	4.32	<0.01	3.51	0.73		15		224	11.4	<0.05	10.6	118	0.2	0.06	586	0.3	<0.1	9.82	<0.1	0.584	<0.005	<0.005		3.11
REPEAT 951-1952	16-Dec-96	Barringer Laboratories													<0.005	42.3	<0.1	0.06	0.025	0.0009	<0.1	717	0.019	4.33	<0.01	3.52	0.73		15		224	11.5	<0.05	10.6	115	<0.1	0.05	586	0.2	<0.1	9.82	<0.1	0.582	<0.005	<0.005		3.11
TP1	1-Nov-96	Barringer Laboratories	6.56	315	430	4	33							248	<0.05	<0.5	<1	<0.1	<0.05	<0.005	<1	67.7	<0.05	<0.1	<0.1	<0.1	<0.1		<10		15.2	0.2	<0.5	<1	2.5	<1	<0.5	69	<1	<1	3.4	<1	0.1	<0.05	<0.05		<0.1
TP2	1-Nov-96	Barringer Laboratories	4.87	303	1530	6	27							971	<0.05	4.8	<1	<0.1	<0.05	<0.005	<1	314	<0.05	0.4	<0.1	0.3	1.1		15		35.4	0.9	<0.5	2	9.9	1	<0.5	311	<1	<1	7.8	<1	0.57	<0.05	<0.05		0.3
TP3	1-Nov-96	Barringer Laboratories	6.56	167	8000	4	24							28	<0.05	<0.5	<1	<0.1	<0.05	<0.005	<1	16.9	<0.05	<0.1	<0.1	<0.1	<0.5		<10		2.6	<0.1	<0.5	<1	<0.1	<1	<0.5	10	<1	<1	0.6	<1	0.02	<0.05	<0.05		<0.1
BH1A DEEP	1-Nov-96	Barringer Laboratories	4.13	337	10100	7	105							13800	<0.05	778	<1	0.2	<0.05	0.019	<1	419	0.41	35.4	<0.1	28.8	4.4		16		1610	58.5	<0.5	29	828	2	<0.5	4210	3	<1	40.5	<1	0.96	<0.05	<0.05		56.5
BH2A DEEP	1-Nov-96	Barringer Laboratories	4.1	296	9800	7	144							12900	<0.05	690	<1	0.1	<0.05	0.023	<1	395	0.22	33.5	<0.1	46.6	67.5		24		1330	100	<0.5	37	778	2	<0.5	3690	3	<1	15	<1	1.83	<0.5	0.06		22.5
BH2B SHALLOW	1-Nov-96	Barringer Laboratories	4.21	263	8000	7	102							9190	<0.05	371	<1	<0.1	<0.05	0.011	<1	400	0.12	21.1	<0.1	9.8	151		23		1060	59.3	<0.5	46	513	<1	<0.5	2710	1	<1	18.3	<1	1.67	<0.05	<0.05		10.4
BH3A DEEP	1-Nov-96	Barringer Laboratories	4.41	238	8700	6	120							10200	<0.05	222	<1	<0.1	<0.05	<0.005	<1	381	0.06	35.2	<0.1	<0.1	206		38		1320	68.7	<0.5	86	733	<1	<0.5	2950	3	<1	5.6	<1	2.01	<0.05	<0.05		9
BH3B SHALLOW	1-Nov-96	Barringer Laboratories	4.48	259	7800	6	93							8750	<0.05	116	<1	0.2	<0.05	<0.005	<1	367	0.1	278	<0.1	2	138		41		1110	55.3	<0.5	71	587	<1	<0.5	2470	2	<1	6.7	<1	2.04	<0.05	<0.05		7.9
BH4A DEEP	1-Nov-96	Barringer Laboratories	4.07	394	10800	4	69							16500	<0.05	952	<1	0.3	<0.05	0.02	<1	391	0.17	33.9	<0.1	18.4	5.5		27		2110	39.4	<0.5	26	696	<1	<0.5	4980	2	<1	54.1	<1	0.54	<0.05	<0.05		7.6
BH5	1-Nov-96	Barringer Laboratories	7.42		250	9	192							246	<0.05	<0.5	<1	<0.1	0.09	<0.005	<1	40.4	<0.05	<0.1	<0.1	<0.1	0.3		<10		10.7	0.1	<0.5	2	<0.5	<1	<0.5	5	<1	<1	6	<1	0.1	<0.05	<0.05		<0.1
BH95 BEHIND FENCE W-MW1	1-Nov-96	Barringer Laboratories	6.72	164	680	10	132							204	<0.05	<0.5	<1	<0.1	<0.05	<0.005	<1	6	0.06	<0.1	<0.1	<0.1	0.2		<10		1.2	0.1	<0.5	154	0.5	<1	<0.5	52	<1	<1	6.5	<1	0.02	<0.05	<0.05		<0.1
REPEAT TP1	1-Nov-96	Barringer Laboratories												225	<0.05	<0.5	<1	<0.1	<0.05	<0.005	<1	65.2	<0.05	<0.1	<0.1	<0.1	<0.1		<10		14.6	0.2	<0.5	<1	2.4	<1	<0.5	66	<1	<1	3.3	<1	0.1	<0.05	<0.05		<0.1
BH95-2B SHALLOW SA#1	22-Aug-96	Barringer Laboratories	4.21	267.3	7500	16	18							7470	<0.05	564	<1	0.1	<0.05	0.017	<1	443	0.18	34	<0.1	23.1	88.6		41		1470	97.8	<0.5	52	784	<1	<0.5	3790	<1	<1	18.5	<1	2.05	<0.05	<0.05		20.3
BH95-2A DEEP SA#2	22-Aug-96	Barringer Laboratories	4.06	279	9400	16								10500	<0.05	275	<1	0.1	<0.05	0.01	<1	426	0.05	20.4	<0.1	5.5	159		31		1040	48.3	<0.5	71	487	1	<0.5	2620	<1	<1	20.6	<1	1.75	<0.05	<0.05		9
BH95-3B SHALLOW SA#3	22-Aug-96	Barringer Laboratories	4.31	234	8900	16	51							8960	<0.05	225	<1	0.1	<0.05	<0.005	<1	412	0.07	36.9	<0.1	0.5	212		46		1390	71.8	<0.5	91	765	<1	<0.5	3170	<1	<1	6.2	<1	2.16	<0.05	<0.05		9.6
BH95-3A DEEP SA#4	22-Aug-96	Barringer Laboratories	3.96	226.7	8800	15	30							9280	<0.05	235	<1	0.2	<0.05	<0.005	<1	412	<0.05	37.1	<0.1	<0.1	207		47		1390	71.8	<0.5	90	768	<1	<0.5	3190	<1	<1	6.1	<1	2.1	<0.05	0.05		9.9
TP#1 SA#5	22-Aug-96	Barringer Laboratories	5.17	307	1050	16	18							598	<0.05	0.7	<1	0.2	<0.05	<0.005	<1	232	<0.05	0.3	<0.1	<0.1	0.3		14		41.4	1	<0.5	5	8.6	<1	<0.5	204	<1	<1	6.5	<1	0.31	<0.05	<0.05		0.1
TP#3 SA#6	22-Aug-96	Barringer Laboratories	6.64	244	1710	16	24							1000	<0.05	<0.5	<1	0.1	<0.05	<0.005	<1	371	<0.05	<0.1	<0.1	<0.1	0.1		26		45	0.3	<0.5	3	3.7	<1	<0.5	353	<1	<1	3.5	<1	0.46	<0.05	<0.05		<0.1
BH95-4A SA#7	22-Aug-96	Barringer Laboratories	3.85	325	10700	15	8							13200	<0.05	878	<1	0.5	<0.05	0.02	<1	416	0.06	33.1	<0.1	23.1	30.4		37		1790	36.6	<0.5	30	663	<1	0.6	4580	2	<1	40.9	<1	0.69	<0.05	<0.05		7.4
BH954-C SA#8	22-Aug-96	Barringer Laboratories												13400	<0.05	875	<1	0.4	<0.05	0.019	<1	421	0.08	32.7	<0.1	22.9	30.4		33		1790	36.5	<0.5	31	658	<1	0.6	4540	2	<1	40.6	<1	0.69	<0.05	<0.05		7.4
BH95-1A (DEEP) SA#9	22-Aug-96	Barringer Laboratories	3.97	363	9200	15								10200	<0.05	504	<1	0.1	<0.05	0.014	<1	440	0.25	29.1	<0.1	21.2	6.5		20		1340	55.2	<0.5	118	705	<1	<0.5	3450	2	<1	31.1	<1	1.29	<0.05	<0.05		43.1
KDW-MW1 SA#10	22-Aug-96	Barringer Laboratories	6.86	260	540	15	63							90	<0.05	<0.5	<1	<0.1	<0.05	<0.005	<1	4.4	<0.05	<0.1	<0.1	<0.1	0.3		<10		0.9	<0.1	<0.5	118	<0.5	<1	<0.5	31	<1	<1	7	<1	0.01	<0.05	<0.05		<0.1
BH95-5 SA#11	22-Aug-96	Barringer Laboratories	7.67	236	250	15	159							14	<0.05	<0.5	<1	<0.1	0.09	<0.005	<1	48.7	<0.05	<0.1	<0.1	<0.1	0.2		<10		11.7	0.2	<0.5	3	<0.5	<1	<0.5	5	<1	<1	6	<1	0.1	<0.05	<0.05		<0.1
SW1 SA#12	22-Aug-96	Barringer Laboratories	3.65	370	4200	9	0							3890	0.05	125	<1	0.2	<0.05	<0.005	<1	339	<0.05	9	<0.1	11.7	6.9		25		488	19.4	<0.5	38	223	<1	<0.5	1300	<1	<1	18.3	<1	1.2	<0.05	<0.05		5.6
SW2 SA#13	22-Aug-96	Barringer Laboratories												1500	<0.05	11.5	<1	0.1	<0.05	<0.005	<1	211	<0.05	2	<0.1	0.7	0.2		16		184	6.3	<0.5	9	55	<1	<0.5	479	<1	<1	5.5	<1	0.77	<0.05	<0.05		1.3
REPEAT BH95-2B SHALLOW SA#1	22-Aug-96	Barringer Laboratories												7470	<0.05	549	<1	0.2	<0.05	0.016	<1	503	0.17	33.1	<0.1	22.6	86.4		37		1430	95.3	<0.5	50	759	<1	<0.5	3710	3	<1	18.1	<1	2.08	<0.05	<0.05		19.7
S-SA1	5-Oct-95	Barringer Laboratories	3.7	438	4400	4	0							5200						<0.0500	<10	417	<1	12.7	<1.00	13.4	5.12		<100		682	25.8	<5.00	38.1	297	<10.0	<5.00	1750	<10.0	<10.0	20.6	,10.0	1.77	<0.500	<0.500		7.57
BH95-2A	5-Oct-95	Barringer Laboratories	4.2	183	>2000	12								8720						<0.0500	<10	516	<1	28.2	<1.00	16.6	85		154		1100	73.7	<5.00	55	666	<10.0	<5.00	3010	<10.0	<10.0	17.7	<10.0	1.98	<0.500	<0.500		14.2
BH95-2B	5-Oct-95	Barringer Laboratories	4.3	230	<2000	8.5	30							7500						<0.0500	<10	484	<1	18.9	<1.00	4.28	141		<100		937	47	<5.00	178	446	<10.0	<5.00	2380	<10.0	<10.0	17.3	<10.0	1.84	<0.500	<0.500		6.67
BH95-4A	5-Oct-95	Barringer Laboratories	4.3	130	>2000	7.5								9660						<0.0500	<10	418	<1	20.8	<1.00	<1.00	271		<100		1100	35.5	<5.00	1020	438	<10.0	<53.00	3240	<10.0	<10.0	15.1	<10.0	2.25	<0.500	<0.500		<1
BH95-5A	5-Oct-95	Barringer Laboratories	6.3	124	260		127.5							25.3						<0.0005	<0	40	<0	<0.01	<0.01	<0.01	<0.01		1		10.1	0.08	<0.05	3.1	<0.05	<0.1	<0.05	6	<0.1	<0.1	4.7	<0.1	0.097	<0.005	<0.005		<1.00
BH95-5C	5-Oct-95	Barringer Laboratories																		<0.0005	<0	<0	<0	<0.01	<0.01	<0.01	<0.01		<1		<0.05	<0.01	<0.05	<0.1	<0.05	<0.1	<0.05	<0.1	<0.1	<0.1	<0.05	<0.1	<0.001	<0.005	<0.005		<0.01
BH95-3A	5-Oct-95	Barringer Laboratories	4.3	179	8300									9130						<0.0500	<10	439	<1	35.8	<1.00	<1.00	164		<100		1260	67.8	<5.00	99.8	760	<10.0	<5.00	3020	<10.0	<10.0	8.65	<10.0	2.21	<0.500	<0.500		10.1
BH95-3B	5-Oct-95	Barringer Laboratories	4.4	57	8500		0.5							8780						<0.0500	<10	436	<1	30.7	<1.00	1.08	159		<100		1140	60.9	<5.00	111	663	<10.0	<5.00	2710	<10.0	<10.0	7.62	<10.0	2.26	<0.500	<0.500		8.58
BH95-3C	5-Oct-95	Barringer Laboratories																		<0.0500	<10	447	<1	36.2	<1.00	<1.00	166		<100		1270	68.5	<5.00	101	777	<10.0	<5.00	3090	<10.0	<10.0	8.1	<10.0	2.25	<0.500	<0.500		10.2
BH95-1A	5-Oct-95	Barringer Laboratories	4.1	58	6100	7.5								7540						<0.0500	<10	470	<1	20.8	<1.00	8.54	79.8		<100		884	39.7	<5.00	65.7	483	<10.0	<5.00	2460	<10.0	<10.0	21.3	<10.0	1.19	<0.500	<0.500		29.3
REPEAT S-SA1	5-Oct-95	Barringer Laboratories																		<0.0500	<10	414	<1	12.4	<1.00	13.4	2.25		<100		688	25.7	<5.00	34.3	301	<10.0	<5.00	1800	<10.0	<10.0	20.1	<10.0	1.76	<0.500	<0.500		7.61
BH95-2A	5-Oct-95	Barringer Laboratories						<0.1	49.5	<0.2	<0.5	2.9	<10	8720																																	
REPEAT BH95-2A	5-Oct-95	Barringer Laboratories						<0.1	50.4	<0.2	<0.5	2.8	<10	8010																																	
96E0093* TP#3	8-May-96	Near North Laboratories	7.55	240	230		20							106	<0.02	0.154	<0.1		0.016	<0.005	<0.05	47.6	<0.02	<0.01	<0.01	0.014	0.135	<0.05	8.53	0.088	4.72	0.017	0.066	1.11	0.183	<0.1	<0.1						0.071	<0.01	<0.01	<0.005	<0.01
TP-2	8-May-96	Near North Laboratories	8.05	247	610		28																																						
96E0094 W11 SEEP	8-May-96	Near North Laboratories	4.2	361	4200	9	0							3600	<0.02	96.6	<0.1		0.019	<0.005	<0.05	249	<0.02	10.2	<0.01	9.45	1.61	0.25	22.8	0.224	501	14.6	0.222	6.35	175	0.164	<0.1						0.956	<0.01	<0.01	0.093	6.75
96E0095 BH 1	8-May-96	Near North Laboratories	4.05	400	9600									18100	<0.02	540	<0.1		0.013	<0.005	<0.05	374	<0.02	42.2	<0.01	23	2.76	<0.05	19.8	0.788	578	54.4	0.491	225	954	0.725	<0.1						1.29	<0.01	<0.01	0.482	43.2
96E0096 BH 3B	8-May-96	Near North Laboratories	4.5	247	8700	13	8							12400	<0.02	204	<0.1		0.016	<0.005	<0.05	352	<0.02	41.1	<0.01	1.98	167	<0.05	37.4	0.268	574	60	0.432	82	843	0.891	<0.1						1.96	<0.01	<0.01	0.244	9.03
96E0097 BH 2A	8-May-96	Near North Laboratories	4.38	266	8200	15								9600	<0.02	307	<0.1		0.012	<0.005	<0.05	363	<0.02	34.2	<0.01	5.89	136	0.135	29.8	0.264	572	65.4	0.395	68.7	734	0.628	<0.1						1.66	<0.01	<0.01	0.334	9.25
96E0098 BH SA 5A	8-May-96	Near North Laboratories	7.43	214	270	7	38.5							39	<0.02	0.154	<0.1		0.08	<0.005	<0.05	37.4	<0.02	<0.01	<0.01	<0.01	0.132	<0.05	4.73	0.066	8.94	0.136	0.047	3.31	0.033	<0.1	<0.1						0.094	<0.01	<0.01	<0.005	0.012
96E0099 W-MW1 OLD BACKGN	8-May-96	Near North Laboratories	6.96	261	300	9	103							39	<0.02	0.258	<0.1		<0.005	<0.005	<0.05	1.87	<0.02	<0.010	<0.01	0.014	0.39	<0.05	4.78	0.046	0.401	0.051	<0.02	69.8	0.071	<0.1	<0.1						0.007	<0.01	<0.01	<0.005	0.011
96E0100 BH96-7	8-May-96	Near North Laboratories												6965	<0.02	222	<0.1		0.015	<0.005	<0.05	360	<0.02	44.4	<0.01	<0.01	181	<0.05	37.5	0.306	575	64.5	0.432	88.4	839	0.78	<0.1						2.05	<0.01	<0.01	0.262	8.92
96E0101 BH 3A DEEP	8-May-96	Near North Laboratories	4.51	236	8800									13250	<0.02	229	<0.1		0.016	<0.005	<0.05	370	<0.02	45.7	<0.01	<0.01	186	<0.05	40.8	0.319	575	66.3	0.425	89.7	845	0.644	<0.1						2.1	<0.01	<0.01	0.268	9.15
96E0102 BH 95-YA DEEP 4A	8-May-96	Near North Laboratories	4.08	400	10100	9								5418	0.05	931	<0.1		0.019	0.007	<0.05	343	<0.02	36	<0.01	17	5.35	0.104	29.7	0.907	547	32.4	0.611	35.2	690	0.779	0.128						0.647	<0.01	<0.01	0.298	6.12
96E0132 BH 2B SHALLOW	8-May-96	Near North Laboratories	4.48	256	7500	9								11900	0.034	227	<0.1		0.017	<0.005	<0.05	390	<0.02	28.6	<0.01	2.87	151	0.115	28.3	0.265	570	51.8	0.377	64.5	595	0.688	<0.1	1.69						1.69	<0.01	<0.01	0.231	7.38
96B0002 BH-1A	29-Jan-96	Near North Laboratories	4.5	285	9200	4.5								11000	<0.020	518	<0.10		0.006	<0.005	<0.05	423	<0.020	38	0.017	19.1	74.5		6.16		583	51.5	0.18	110	739	<0.1	<0.100						1.12	<0.01	<0.01		46.4
96B0003 BH-2A	29-Jan-96	Near North Laboratories	4.32	247		8								13200	<0.020	510	<0.10		0.016	<0.005	<0.05	407	<0.020	40.6	0.015	13.4	136		16.6		583	83.4	0.2	717	803	<0.1	<0.100						1.92	<0.01	<0.01		16.9
96B0004 BH-2B	29-Jan-96	Near North Laboratories	4.28	251	7100	8								8200	<0.020	216	<0.10		0.008	<0.005	<0.05	405	<0.020	27.2	<0.010	4.42	128		19.6		527	49.4	0.13	195	548	0.3	<0.100						1.97	<0.01	<0.01		6.72
BH-3A	29-Jan-96	Near North Laboratories	4.24	233	9200	6																																							
96B0005 BH-4A	29-Jan-96	Near North Laboratories	3.92	288	12300	6.5								145000	<0.020	709	<0.10		0.011	<0.005	<0.05	360	<0.020	41.5	0.015	8.86	432		27.4		534	43.9	0.3	126	780	0.7	<0.100						1.22	<0.01	<0.01		8.75
TP-2	29-Jan-96	Near North Laboratories												885										0.567		1.29	299							10.3										0.574
95K0023 PILE SEEPAGE DISCHARGE	7-Nov-95	Near North Laboratories												4374	0.015	128		<0.010	0.007	<0.01		249	0.066	11.1	<0.010	12.5	7.01		37.8		741	26.4	<0.02	41.5	280	<0.1	0.004				21.7		1.6	<0.010	0.014		7.82
95K0024 TP1	7-Nov-95	Near North Laboratories												542	<0.005	4.62		<0.010	0.053	<0.01		123	<0.010	0.363	<0.010	0.111	8.75		18.4		57.7	2.08	<0.02	4.94	9.39	<0.1	0.0024				7.02		0.413	0.078	0.021		0.186
95K0025 TP3 CORRECTED	7-Nov-95	Near North Laboratories												IS	<0.01	1924		1.95	8.59	0.05		943	1.95	6.08	8.96	86.1	5533		221		2486	45.1	<0.02	39.7	178	45.1	11.8				2.99		3.72	69.3	4.15		23.9
95K0042 TP2	7-Nov-95	Near North Laboratories												885	0.009	55.3		<0.010	0.093	<0.01		292	<0.010	0.567	0.877	1.29	299		23.6		202	2.81	<0.02	5.74	10.3	<0.1	0.0091				28		1.13	1.13	0.091		0.574

Source: Vos, K.J., Pettit, C., Martin, J., Knapp, R.A. and Jansons, K.J. Whistle Mine Waste Rock Study. MEND Report 1.41.1. October, 1997.

Attachment 5
Vangorda Waste Rock Stockpile Acidic Water Chemistry Data

STATION	YEAR	DATE	AG-D mg/L	AG-T mg/L	AL-D mg/L	ALK-T mg/L	AL-T mg/L	AS-D mg/L	AS-T mg/L	BA-D mg/L	BA-T mg/L	B-D mg/L	BE-D mg/L	BE-T mg/L	BI-D mg/L	BI-T mg/L	B-T mg/L	CA-D mg/L	CA-T mg/L	CD-D mg/L	CD-T mg/L
V32	1988	6/15/1988	#N/A	#N/A	#N/A	-1	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/7/1988	#N/A	#N/A	#N/A	13.12	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/11/1988	#N/A	#N/A	#N/A	13.12	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/20/1988	#N/A	#N/A	#N/A	12.6	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/27/1988	#N/A	#N/A	#N/A	12.6	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	8/24/1988	#N/A	#N/A	#N/A	15.8	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	8/24/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	9/13/1988	#N/A	#N/A	#N/A	14.5	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	9/13/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	10/6/1988	#N/A	#N/A	#N/A	16	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	10/6/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	0.009	#N/A	#N/A	4	#N/A	-0.02	#N/A	-0.0006	#N/A	#N/A	-0.0002	#N/A	#N/A	#N/A	#N/A	513	#N/A	0.558
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	0	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	#N/A	0.015	#N/A	-5	0.66	#N/A	-0.02	#N/A	0.0127	#N/A	#N/A	-0.0002	#N/A	#N/A	#N/A	#N/A	453	#N/A	0.146
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1996	5/27/1996	0.04	#N/A	2.6	-5	#N/A	-0.2	#N/A	0.35	#N/A	#N/A	0.09	#N/A	-0.4	#N/A	#N/A	426	#N/A	0.39	#N/A
V32	1996	5/27/1996	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1996	9/11/1996	-0.03	#N/A	2.2	#N/A	#N/A	0.5	#N/A	0.46	#N/A	#N/A	0.02	#N/A	-0.4	#N/A	#N/A	412	#N/A	0.57	#N/A
V32	1996	9/11/1996	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1996	9/11/1996	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	5/12/1997	#N/A	0.034	#N/A	#N/A	1.68	#N/A	-0.02	#N/A	0.61	#N/A	#N/A	0.134	#N/A	0.37	4.98	#N/A	322.2	#N/A	0.556
V32	1997	5/12/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	5/12/1997	#N/A	#N/A	#N/A	0	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	6/30/1997	-0.003	-0.003	7.29	-5	7.42	0.12	0.14	0.021	0.018	-0.05	0.006	0.006	-0.04	-0.04	-0.05	411.2	433.7	1.305	1.295
V32	1997	6/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	7/22/1997	#N/A	#N/A	16	#N/A	#N/A	1.2	#N/A	0.1	#N/A	3.8	#N/A	#N/A	1.1	#N/A	#N/A	364.7	#N/A	2	#N/A
V32	1997	7/22/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	7/22/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	8/6/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	8/11/1997	#N/A	0.08	#N/A	#N/A	5.38	#N/A	-0.02	#N/A	0.019	#N/A	#N/A	0.006	#N/A	-0.04	-0.05	#N/A	368	#N/A	1.4
V32	1997	8/11/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	8/11/1997	0.071	#N/A	4.97	#N/A	#N/A	-0.02	#N/A	0.018	#N/A	-0.05	0.006	#N/A	-0.04	#N/A	#N/A	342.5	#N/A	1.265	#N/A
V32	1997	9/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	9/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	9/30/1997	#N/A	0.039	#N/A	#N/A	5.04	#N/A	0.62	#N/A	0.799	#N/A	#N/A	0.008	#N/A	0.68	-0.05	#N/A	398.9	#N/A	2.247
V32	1997	9/30/1997	0.023	#N/A	3.54	#N/A	#N/A	1.01	#N/A	0.55	#N/A	0.09	0.008	#N/A	0.78	#N/A	#N/A	405.4	#N/A	2.243	#N/A
V32	1997	10/20/1997	0.03	#N/A	2.4	#N/A	#N/A	-0.02	#N/A	1.169	#N/A	1.24	0.299	#N/A	-0.04	#N/A	#N/A	430.3	#N/A	2.113	#N/A
V32	1997	10/20/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	10/20/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	12/22/1997	0.007	#N/A	2.92	#N/A	#N/A	0.09	#N/A	0.553	#N/A	1.32	0.3	#N/A	-0.04	#N/A	#N/A	390.3	#N/A	1.91	#N/A
V32	1997	12/22/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	5/18/1998	0.059	#N/A	3.2	-5	#N/A	0.59	#N/A	0.101	#N/A	-0.05	0.005	#N/A	-0.04	#N/A	#N/A	335.4	#N/A	1.59	#N/A
V32	1998	5/18/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	6/29/1998	0.137	#N/A	4.56	#N/A	#N/A	2.11	#N/A	0.664	#N/A	0.07	0.012	#N/A	1.26	#N/A	#N/A	447	#N/A	3.86	#N/A
V32	1998	6/29/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	6/29/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	9/14/1998	0.174	#N/A	7.6	#N/A	#N/A	1.44	#N/A	0.294	#N/A	-0.05	0.001	#N/A	-0.04	#N/A	#N/A	407	#N/A	3.953	#N/A
V32	1998	9/14/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	9/14/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A

STATION	YEAR	DATE	AG-D mg/L	AG-T mg/L	AL-D mg/L	ALK-T mg/L	AL-T mg/L	AS-D mg/L	AS-T mg/L	BA-D mg/L	BA-T mg/L	B-D mg/L	BE-D mg/L	BE-T mg/L	BI-D mg/L	BI-T mg/L	B-T mg/L	CA-D mg/L	CA-T mg/L	CD-D mg/L	CD-T mg/L
V32	1999	5/18/1999	0.103	#N/A	1.38	#N/A	#N/A	0.048	#N/A	0.177	#N/A	-0.05	0.007	#N/A	-0.04	#N/A	#N/A	349.5	#N/A	2.025	#N/A
V32	1999	5/18/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	6/18/1999	-0.003	#N/A	2.22	#N/A	#N/A	-0.005	#N/A	0.523	#N/A	-0.05	0.009	#N/A	-0.04	#N/A	#N/A	400.2	#N/A	2.909	#N/A
V32	1999	6/18/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	8/12/1999	#N/A	#N/A	0.94	#N/A	#N/A	2.414	#N/A	#N/A	#N/A	2.44	0.038	#N/A	-0.04	#N/A	#N/A	455.4	#N/A	6.316	#N/A
V32	1999	8/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	8/12/1999	0.517	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	1.655	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	10/12/1999	-0.003	#N/A	6.03	-5	#N/A	-0.005	#N/A	0.382	#N/A	0.06	0.006	#N/A	-0.04	#N/A	#N/A	394.1	#N/A	5.496	#N/A
V32	1999	10/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	10/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	12/13/1999	-0.003	#N/A	6.2	#N/A	#N/A	0.419	#N/A	0.979	#N/A	2.23	0.135	#N/A	-0.04	#N/A	#N/A	406.6	#N/A	4.359	#N/A
V32	1999	12/13/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	12/13/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2000	5/31/2000	0.148	#N/A	7.89	#N/A	#N/A	-0.005	#N/A	0.423	#N/A	0.42	0.007	#N/A	-0.05	#N/A	#N/A	448.7	#N/A	4.763	#N/A
V32	2000	5/31/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2000	5/31/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2000	9/12/2000	0.015	#N/A	8.44	#N/A	#N/A	1.02	#N/A	0.083	#N/A	-0.05	0.009	#N/A	-0.05	#N/A	#N/A	354.6	#N/A	4.23	#N/A
V32	2000	9/12/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2001	6/13/2001	0.163	#N/A	15.73	#N/A	#N/A	0.014	#N/A	0.078	#N/A	1.2	-0.001	#N/A	0.08	#N/A	#N/A	335.2	#N/A	8.976	#N/A
V32	2001	6/13/2001	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2001	6/13/2001	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2003	6/17/2003	0.0007	#N/A	21.965	#N/A	22.089	0.007	#N/A	0.008	0.008	1.3	0.0057	0.0057	0.09	0.08	1.32	478.1	472.8	5.6536	5.5935
V32	2003	6/17/2003	#N/A	0.0008	#N/A	#N/A	#N/A	#N/A	0.007	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2003	9/15/2003	0.0294	#N/A	41.248	#N/A	#N/A	0.748	#N/A	0.009	#N/A	1.46	0.0075	#N/A	0.11	#N/A	#N/A	485.9	#N/A	6.7453	#N/A
V32	2003	9/15/2003	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	6/28/2004	-0.00025	#N/A	#N/A	#N/A	#N/A	0.032	#N/A	0.016	#N/A	-0.05	0.01	#N/A	-0.001	#N/A	#N/A	488	#N/A	8.26	#N/A
V32	2004	6/28/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	6/28/2004	#N/A	#N/A	220	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	9/20/2004	-0.00025	#N/A	193	#N/A	#N/A	0.018	#N/A	0.007	#N/A	-0.05	0.016	#N/A	-0.001	#N/A	#N/A	403	#N/A	#N/A	#N/A
V32	2004	9/20/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	9/20/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	18.3	#N/A
V32	2005	6/22/2005	-0.00025	#N/A	363	#N/A	#N/A	0.018	#N/A	0.007	#N/A	-0.05	0.023	#N/A	-0.001	#N/A	#N/A	499	#N/A	19.9	#N/A
V32	2005	6/22/2005	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2005	9/19/2005	-0.00025	#N/A	290	#N/A	#N/A	0.024	#N/A	0.008	#N/A	-0.05	0.016	#N/A	-0.001	#N/A	#N/A	469	#N/A	14.8	#N/A
V32	2005	9/19/2005	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2006	6/8/2006	-0.00025	#N/A	160	#N/A	#N/A	0.015	#N/A	0.005	#N/A	-0.05	0.011	#N/A	-0.001	#N/A	#N/A	334	#N/A	7.33	#N/A
V32	2006	9/18/2006	0.011	#N/A	131	#N/A	#N/A	0.016	#N/A	0.006	#N/A	-0.05	0.01	#N/A	-0.001	#N/A	#N/A	407	#N/A	5.77	#N/A
V32	2007	6/11/2007	0.017	#N/A	221	#N/A	#N/A	0.011	#N/A	0.004	#N/A	-0.05	0.017	#N/A	-0.001	#N/A	#N/A	366	#N/A	5.03	#N/A
V32	2007	10/1/2007	#N/A	#N/A	353	#N/A	#N/A	#N/A	#N/A	-0.1	#N/A	#N/A	-0.3	#N/A	#N/A	#N/A	#N/A	425	#N/A	10	#N/A
V32	2008	6/1/2008	0.027	#N/A	691	#N/A	#N/A	0.017	#N/A	0.005	#N/A	-0.05	0.02	#N/A	-0.001	#N/A	#N/A	437	#N/A	6.36	#N/A
V32	2008	9/29/2008	0.02	#N/A	482	#N/A	#N/A	0.013	#N/A	0.007	#N/A	-0.05	0.025	#N/A	-0.001	#N/A	#N/A	374	#N/A	9.88	#N/A

STATION	YEAR	DATE	CN-T mg/L	N-WAD mg	CO-D mg/L	COLOR CU	OND-F µS/c	OND-L µS/c	CO-T mg/L	CR-D mg/L	CR-T mg/L	CU-D mg/L	CU-T mg/L	DO mg/L	DO-% %	DOC mg/L	FE-D mg/L	FE-T mg/L	LOWLS L/s	WM3S m^3	HARD mg/L
V32	1988	6/15/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/7/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/11/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/20/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/27/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	8/24/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.002	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	8/24/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	9/13/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.002	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	9/13/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	10/6/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.002	#N/A	#N/A	#N/A	#N/A	0.095	#N/A	#N/A	#N/A
V32	1988	10/6/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	7200	#N/A	15	#N/A	-0.001	#N/A	-0.002	#N/A	#N/A	#N/A	#N/A	0.807	#N/A	#N/A	4240
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	6900	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	7090	#N/A	10.1	#N/A	0.036	#N/A	-0.002	#N/A	#N/A	#N/A	#N/A	112	#N/A	#N/A	4
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	#N/A	6600	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0	#N/A	#N/A
V32	1996	5/27/1996	#N/A	#N/A	6.62	#N/A	#N/A	3008	#N/A	-0.05	#N/A	0.06	#N/A	#N/A	#N/A	#N/A	17.7	#N/A	0.019	#N/A	2389
V32	1996	5/27/1996	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1996	9/11/1996	#N/A	#N/A	7.43	#N/A	#N/A	9220	#N/A	-0.05	#N/A	0.12	#N/A	#N/A	#N/A	#N/A	36.8	#N/A	#N/A	#N/A	#N/A
V32	1996	9/11/1996	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1996	9/11/1996	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	5/12/1997	#N/A	#N/A	#N/A	#N/A	#N/A	5150	6.392	#N/A	-0.005	#N/A	0.09	#N/A	#N/A	#N/A	#N/A	33.81	0.02	#N/A	#N/A
V32	1997	5/12/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	5/12/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	5/12/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	6/30/1997	#N/A	#N/A	10.175	#N/A	#N/A	9120	10.378	-0.005	-0.005	0.079	0.094	#N/A	#N/A	#N/A	2.29	18.23	0.05	#N/A	#N/A
V32	1997	6/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	7/22/1997	#N/A	#N/A	12.1	#N/A	#N/A	9550	#N/A	-0.005	#N/A	0.1	#N/A	#N/A	#N/A	#N/A	28.3	#N/A	0.02	#N/A	#N/A
V32	1997	7/22/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	7/22/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	8/6/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.02	#N/A	#N/A
V32	1997	8/11/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	10.503	#N/A	-0.005	#N/A	0.106	#N/A	#N/A	#N/A	#N/A	27.79	0.02	#N/A	#N/A
V32	1997	8/11/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	8/11/1997	#N/A	#N/A	9.917	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	0.079	#N/A	#N/A	#N/A	#N/A	15.02	#N/A	#N/A	#N/A	#N/A
V32	1997	9/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	9950	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.02	#N/A	#N/A
V32	1997	9/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	9/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	11.2	#N/A	-0.005	#N/A	0.138	#N/A	#N/A	#N/A	#N/A	22.21	#N/A	#N/A	#N/A
V32	1997	9/30/1997	#N/A	#N/A	11.215	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	0.077	#N/A	#N/A	#N/A	#N/A	#N/A	16.25	#N/A	#N/A	#N/A
V32	1997	10/20/1997	#N/A	#N/A	11.76	#N/A	#N/A	10250	#N/A	-0.005	#N/A	0.122	#N/A	#N/A	#N/A	#N/A	2.87	#N/A	0.008	#N/A	#N/A
V32	1997	10/20/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	10/20/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	12/22/1997	#N/A	#N/A	11.489	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	0.12	#N/A	#N/A	#N/A	#N/A	36.43	#N/A	0.004	#N/A	#N/A
V32	1997	12/22/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	5/18/1998	#N/A	#N/A	6.82	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	0.092	#N/A	#N/A	#N/A	#N/A	9.15	#N/A	0.008	#N/A	#N/A
V32	1998	5/18/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	6/29/1998	#N/A	#N/A	12.419	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.123	#N/A	#N/A	#N/A	#N/A	55.98	#N/A	0.03	#N/A	#N/A
V32	1998	6/29/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	6/29/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.87	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	9/14/1998	#N/A	#N/A	13.682	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.147	#N/A	#N/A	#N/A	#N/A	38.17	#N/A	0.02	#N/A	#N/A
V32	1998	9/14/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	9/14/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.816	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A

STATION	YEAR	DATE	CN-T mg/L	N-WAD mg	CO-D mg/L	COLOR CU	OND-F µS/c	OND-L µS/c	CO-T mg/L	CR-D mg/L	CR-T mg/L	CU-D mg/L	CU-T mg/L	DO mg/L	DO-% %	DOC mg/L	FE-D mg/L	FE-T mg/L	LOWLS L/s	WM3S m^3	HARD mg/L
V32	1999	5/18/1999	#N/A	#N/A	9.669	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	0.199	#N/A	#N/A	#N/A	#N/A	127.98	#N/A	0.01	#N/A	#N/A
V32	1999	5/18/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	6/18/1999	#N/A	#N/A	11.939	#N/A	#N/A	#N/A	#N/A	0.177	#N/A	0.11	#N/A	#N/A	#N/A	#N/A	224.27	#N/A	0.018	#N/A	#N/A
V32	1999	6/18/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	8/12/1999	#N/A	#N/A	15.582	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	0.087	#N/A	#N/A	#N/A	#N/A	167.3	#N/A	0.019	#N/A	#N/A
V32	1999	8/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	8/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	10/12/1999	#N/A	#N/A	9.901	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	0.114	#N/A	#N/A	#N/A	#N/A	123.7	#N/A	0.03	#N/A	#N/A
V32	1999	10/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	15200	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	10/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	12/13/1999	#N/A	#N/A	22.105	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	0.375	#N/A	#N/A	#N/A	#N/A	188.88	#N/A	#N/A	#N/A	#N/A
V32	1999	12/13/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	12/13/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2000	5/31/2000	#N/A	#N/A	12.425	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	0.112	#N/A	#N/A	#N/A	#N/A	315	#N/A	#N/A	#N/A	#N/A
V32	2000	5/31/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2000	5/31/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.1	#N/A	#N/A
V32	2000	9/12/2000	#N/A	#N/A	11.41	#N/A	#N/A	#N/A	#N/A	0.082	#N/A	0.204	#N/A	#N/A	#N/A	#N/A	273.3	#N/A	#N/A	#N/A	#N/A
V32	2000	9/12/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2001	6/13/2001	#N/A	#N/A	19.088	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	0.105	#N/A	#N/A	#N/A	#N/A	679.08	#N/A	#N/A	#N/A	#N/A
V32	2001	6/13/2001	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2001	6/13/2001	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2003	6/17/2003	#N/A	#N/A	14.027	#N/A	#N/A	#N/A	13.967	-0.001	-0.001	0.103	0.117	#N/A	#N/A	#N/A	714.108	890.457	#N/A	#N/A	#N/A
V32	2003	6/17/2003	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2003	9/15/2003	#N/A	#N/A	22.158	#N/A	#N/A	#N/A	#N/A	-0.001	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	1135.386	#N/A	#N/A	#N/A	#N/A
V32	2003	9/15/2003	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.483	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	6/28/2004	#N/A	#N/A	26.1	#N/A	#N/A	#N/A	#N/A	0.016	#N/A	0.11	#N/A	#N/A	#N/A	#N/A	1360	#N/A	#N/A	#N/A	#N/A
V32	2004	6/28/2004	#N/A	#N/A	#N/A	#N/A	#N/A	47900	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	6/28/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	9/20/2004	#N/A	#N/A	30.8	#N/A	#N/A	#N/A	#N/A	0.011	#N/A	0.048	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	9/20/2004	#N/A	#N/A	#N/A	#N/A	#N/A	102000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	9/20/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	2330	#N/A	#N/A	#N/A	#N/A
V32	2005	6/22/2005	#N/A	#N/A	21.8	#N/A	#N/A	#N/A	#N/A	0.01	#N/A	0.037	#N/A	#N/A	#N/A	#N/A	2970	#N/A	#N/A	#N/A	#N/A
V32	2005	6/22/2005	#N/A	#N/A	#N/A	#N/A	#N/A	72400	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2005	9/19/2005	#N/A	#N/A	29.7	#N/A	#N/A	#N/A	#N/A	0.011	#N/A	0.031	#N/A	#N/A	#N/A	#N/A	2930	#N/A	#N/A	#N/A	#N/A
V32	2005	9/19/2005	#N/A	#N/A	#N/A	#N/A	#N/A	85000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2006	6/8/2006	#N/A	#N/A	21.7	#N/A	#N/A	#N/A	#N/A	0.012	#N/A	0.039	#N/A	#N/A	#N/A	#N/A	3240	#N/A	#N/A	#N/A	#N/A
V32	2006	9/18/2006	#N/A	#N/A	24.4	#N/A	#N/A	#N/A	#N/A	0.009	#N/A	0.017	#N/A	#N/A	#N/A	#N/A	3260	#N/A	#N/A	#N/A	#N/A
V32	2007	6/11/2007	#N/A	#N/A	23.8	#N/A	#N/A	97000	#N/A	0.017	#N/A	0.044	#N/A	#N/A	#N/A	#N/A	3650	#N/A	#N/A	#N/A	#N/A
V32	2007	10/1/2007	#N/A	#N/A	26	#N/A	#N/A	98100	#N/A	#N/A		#N/A	#N/A	#N/A	#N/A	#N/A	3870	#N/A	#N/A	#N/A	#N/A
V32	2008	6/1/2008	#N/A	#N/A	26.6	#N/A	#N/A	129000	#N/A	0.018	#N/A	0.035	#N/A	#N/A	#N/A	#N/A	5080	#N/A	#N/A	#N/A	#N/A
V32	2008	9/29/2008	#N/A	#N/A	19.6	#N/A	#N/A	77500	#N/A	0.025	#N/A	0.32	#N/A	#N/A	#N/A	#N/A	2710	#N/A	#N/A	#N/A	#N/A

STATION	YEAR	DATE	ARD-C mg/L	HG-D mg/L	HG-T mg/L	K-D mg/L	K-T mg/L	LA-D mg/L	LA-T mg/L	LC50 %v/v	LI-D mg/L	LI-T mg/L	MG-D mg/L	MG-T mg/L	MN-D mg/L	MN-T mg/L	MO-D mg/L	MO-T mg/L	NA-D mg/L	NA-T mg/L	NH3 mg/L
V32	1988	6/15/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/7/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/11/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/20/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/27/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	8/24/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.006	#N/A	#N/A	#N/A	1.8	0.73
V32	1988	8/24/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	9/13/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.007	#N/A	#N/A	#N/A	1.8	0.46
V32	1988	9/13/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	10/6/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.002	#N/A	#N/A	#N/A	1.6	0.1
V32	1988	10/6/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	26	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	542	#N/A	387	#N/A	-0.005	#N/A	14.8	6
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	20	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	543	#N/A	380	#N/A	-0.005	#N/A	13.4	5
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1996	5/27/1996	#N/A	-0.2	#N/A	#N/A	#N/A	-0.05	#N/A	#N/A	#N/A	#N/A	378	#N/A	244.5	#N/A	0.02	#N/A	27	#N/A	13.52
V32	1996	5/27/1996	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1996	9/11/1996	#N/A	-0.2	#N/A	#N/A	#N/A	-0.05	#N/A	#N/A	#N/A	#N/A	578	#N/A	371.1	#N/A	0.04	#N/A	#N/A	#N/A	4.73
V32	1996	9/11/1996	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1996	9/11/1996	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	122	#N/A	#N/A
V32	1997	5/12/1997	#N/A	#N/A	#N/A	#N/A	-1	#N/A	0.417	#N/A	#N/A	#N/A	#N/A	426.3	#N/A	305.5	#N/A	-0.002	#N/A	19	4.93
V32	1997	5/12/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	5/12/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	5/12/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	6/30/1997	#N/A	#N/A	#N/A	20	20	0.172	0.158	#N/A	#N/A	#N/A	929.2	928.5	#N/A	#N/A	1.515	1.559	16	15	3.16
V32	1997	6/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	7/22/1997	#N/A	#N/A	#N/A	17	#N/A	0.2	#N/A	#N/A	#N/A	#N/A	862	#N/A	779.5	#N/A	2.4	#N/A	29	#N/A	6.7
V32	1997	7/22/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	7/22/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	8/6/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	8/11/1997	#N/A	#N/A	#N/A	#N/A	18	#N/A	0.144	#N/A	#N/A	#N/A	#N/A	885.6	#N/A	536.2	#N/A	1.5	#N/A	14	8.79
V32	1997	8/11/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	8/11/1997	#N/A	#N/A	#N/A	19	#N/A	0.164	#N/A	#N/A	#N/A	#N/A	835.1	#N/A	492.74	#N/A	1.425	#N/A	13	#N/A	#N/A
V32	1997	9/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	6.69
V32	1997	9/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	9/30/1997	#N/A	#N/A	#N/A	#N/A	16	#N/A	0.306	#N/A	#N/A	#N/A	#N/A	930.5	#N/A	591.3	#N/A	2.306	#N/A	37	#N/A
V32	1997	9/30/1997	#N/A	#N/A	#N/A	12	#N/A	0.375	#N/A	#N/A	#N/A	#N/A	953.8	#N/A	606.21	#N/A	2.224	#N/A	36	#N/A	#N/A
V32	1997	10/20/1997	#N/A	#N/A	#N/A	26	#N/A	0.202	#N/A	#N/A	#N/A	#N/A	1041	#N/A	750.48	#N/A	0.248	#N/A	39	#N/A	7.14
V32	1997	10/20/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	10/20/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	12/22/1997	#N/A	#N/A	#N/A	7	#N/A	0.023	#N/A	#N/A	#N/A	#N/A	972.7	#N/A	685.28	#N/A	-0.002	#N/A	28	#N/A	6.35
V32	1997	12/22/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	5/18/1998	#N/A	#N/A	#N/A	15	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	584.2	#N/A	316.6	#N/A	0.98	#N/A	12	#N/A	-0.05
V32	1998	5/18/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	6/29/1998	#N/A	#N/A	#N/A	17	#N/A	0.215	#N/A	#N/A	#N/A	#N/A	1000	#N/A	665.74	#N/A	2.479	#N/A	23	#N/A	-0.05
V32	1998	6/29/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	6/29/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	9/14/1998	#N/A	#N/A	#N/A	18	#N/A	0.336	#N/A	#N/A	#N/A	#N/A	1000	#N/A	940.35	#N/A	2.974	#N/A	22	#N/A	-0.05
V32	1998	9/14/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	9/14/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A

STATION	YEAR	DATE	ARD-C mg/L	HG-D mg/L	HG-T mg/L	K-D mg/L	K-T mg/L	LA-D mg/L	LA-T mg/L	LC50 %v/v	LI-D mg/L	LI-T mg/L	MG-D mg/L	MG-T mg/L	MN-D mg/L	MN-T mg/L	MO-D mg/L	MO-T mg/L	NA-D mg/L	NA-T mg/L	NH3 mg/L
V32	1999	5/18/1999	#N/A	#N/A	#N/A	15	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	767	#N/A	419.04	#N/A	0.839	#N/A	14	#N/A	1.36
V32	1999	5/18/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	6/18/1999	#N/A	#N/A	#N/A	28	#N/A	0.349	#N/A	#N/A	#N/A	#N/A	1038.1	#N/A	661.68	#N/A	3.064	#N/A	22	#N/A	0.29
V32	1999	6/18/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	8/12/1999	#N/A	#N/A	#N/A	-1	#N/A	0.294	#N/A	#N/A	#N/A	#N/A	1740	#N/A	1245.81	#N/A	1.792	#N/A	-1	#N/A	#N/A
V32	1999	8/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	8/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	10/12/1999	#N/A	#N/A	#N/A	14	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	1581.1	#N/A	979.15	#N/A	5.188	#N/A	12	#N/A	-0.05
V32	1999	10/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	10/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	12/13/1999	#N/A	#N/A	#N/A	8	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	1498.6	#N/A	-0.01	#N/A	#N/A	#N/A	17	#N/A	#N/A
V32	1999	12/13/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	12/13/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	98.693	#N/A	#N/A	#N/A	#N/A
V32	2000	5/31/2000	#N/A	#N/A	#N/A	13	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	1342	#N/A	1199	#N/A	1.941	#N/A	11	#N/A	#N/A
V32	2000	5/31/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2000	5/31/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2000	9/12/2000	#N/A	#N/A	#N/A	15	#N/A	0.54	#N/A	#N/A	#N/A	#N/A	1394.1	#N/A	643.9	#N/A	2.84	#N/A	13	#N/A	#N/A
V32	2000	9/12/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2001	6/13/2001	#N/A	#N/A	#N/A	11	#N/A	0.324	#N/A	#N/A	#N/A	#N/A	1737	#N/A	1231.8	#N/A	-0.002	#N/A	13	#N/A	#N/A
V32	2001	6/13/2001	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2001	6/13/2001	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2003	6/17/2003	#N/A	#N/A	#N/A	5.5	5.4	0.367	0.382	#N/A	#N/A	#N/A	3094.2	3024.6	1994.527	1975.2	0.012	0.012	13.5	13.5	#N/A
V32	2003	6/17/2003	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2003	9/15/2003	#N/A	#N/A	#N/A	4.1	#N/A	0.437	#N/A	#N/A	#N/A	#N/A	3313.1	#N/A	2145.981	#N/A	0.213	#N/A	14.9	#N/A	#N/A
V32	2003	9/15/2003	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	6/28/2004	#N/A	#N/A	#N/A	1.3	#N/A	#N/A	#N/A	#N/A	0.31	#N/A	3190	#N/A	2340	#N/A	-0.0005	#N/A	7.92	#N/A	7.53
V32	2004	6/28/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	6/28/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	6/28/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	9/20/2004	#N/A	#N/A	#N/A	2.3	#N/A	#N/A	#N/A	#N/A	0.44	#N/A	3960	#N/A	3340	#N/A	-0.0005	#N/A	7.44	#N/A	4.82
V32	2004	9/20/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	9/20/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2005	6/22/2005	#N/A	#N/A	#N/A	1.4	#N/A	#N/A	#N/A	#N/A	0.47	#N/A	3500	#N/A	3290	#N/A	-0.0005	#N/A	7.77	#N/A	3.53
V32	2005	6/22/2005	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2005	9/19/2005	#N/A	#N/A	#N/A	1.7	#N/A	#N/A	#N/A	#N/A	0.54	#N/A	4430	#N/A	3500	#N/A	-0.0005	#N/A	7.73	#N/A	3.34
V32	2005	9/19/2005	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2006	6/8/2006	#N/A	#N/A	#N/A	1.1	#N/A	#N/A	#N/A	#N/A	0.42	#N/A	3310	#N/A	3380	#N/A	0.0011	#N/A	8.53	#N/A	3.55
V32	2006	9/18/2006	#N/A	#N/A	#N/A	1.7	#N/A	#N/A	#N/A	#N/A	0.52	#N/A	4670	#N/A	3550	#N/A	0.208	#N/A	9.8	#N/A	7.1
V32	2007	6/11/2007	#N/A	#N/A	#N/A	2	#N/A	#N/A	#N/A	#N/A	0.54	#N/A	4710	#N/A	3690	#N/A	0.298	#N/A	11.3	#N/A	4.86
V32	2007	10/1/2007	#N/A	#N/A	#N/A	34	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	5530	#N/A	3860	#N/A	-2	#N/A	18	#N/A	6.61
V32	2008	6/1/2008	#N/A	#N/A	#N/A	1.3	#N/A	#N/A	#N/A	#N/A	0.62	#N/A	5770	#N/A	4310	#N/A	0.555	#N/A	12	#N/A	5.38
V32	2008	9/29/2008	#N/A	#N/A	#N/A	0.6	#N/A	#N/A	#N/A	#N/A	0.88	#N/A	3790	#N/A	2320	#N/A	0.281	#N/A	7	#N/A	2.73

STATION	YEAR	DATE	NI-D mg/L	NI-T mg/L	NO2 mg/L	O2NO3 mg	NO3 mg/L	N-T mg/L	PB-D mg/L	PB-T mg/L	P-D mg/L	PH-F pH un	PH-L pH un	PO4 mg/L	P-T mg/L	SB-D mg/L	SB-T mg/L	S-D mg/L	SE-D mg/L	SE-T mg/L	SI-D mg/L
V32	1988	6/15/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	7.05	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/7/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	7.67	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/11/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	7.25	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/20/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	6.5	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/27/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	7.5	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	8/24/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.005	#N/A	#N/A	8.16	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	8/24/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	9/13/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.007	#N/A	#N/A	7.8	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	9/13/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	10/6/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	8	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	10/6/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	10.8	-50	#N/A	-10	#N/A	#N/A	-0.01	#N/A	4.5	4.5	#N/A	-0.06	#N/A	-0.02	#N/A	#N/A	-0.02	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	#N/A	6.57	-50	#N/A	-5	#N/A	#N/A	0.23	#N/A	5.5	3.8	#N/A	-0.06	#N/A	-0.02	#N/A	#N/A	-0.02	#N/A
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1996	5/27/1996	3.59	#N/A	#N/A	#N/A	#N/A	#N/A	-0.2	#N/A	0.9	5	#N/A	#N/A	#N/A	0.8	#N/A	#N/A	#N/A	#N/A	4.4
V32	1996	5/27/1996	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1996	9/11/1996	4.42	#N/A	#N/A	#N/A	#N/A	#N/A	0.6	#N/A	0.6	5.34	#N/A	#N/A	#N/A	1.1	#N/A	#N/A	#N/A	#N/A	11.5
V32	1996	9/11/1996	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1996	9/11/1996	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	5/12/1997	#N/A	3.261	#N/A	#N/A	#N/A	#N/A	#N/A	0.32	#N/A	5.22	#N/A	#N/A	2.08	#N/A	1.13	#N/A	#N/A	-0.03	#N/A
V32	1997	5/12/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	5/12/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	5/12/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	6/30/1997	6.346	6.399	#N/A	#N/A	#N/A	#N/A	4.17	4.24	96.23	4.08	#N/A	#N/A	97.54	3.2	3.19	#N/A	-0.03	-0.03	13.4
V32	1997	6/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	7/22/1997	7.7	#N/A	#N/A	#N/A	#N/A	#N/A	8.9	#N/A	168.6	3.91	#N/A	#N/A	#N/A	4.8	#N/A	#N/A	-0.03	#N/A	21.2
V32	1997	7/22/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	7/22/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	8/6/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	8/11/1997	#N/A	6.411	#N/A	#N/A	#N/A	#N/A	#N/A	4.46	#N/A	4.14	#N/A	#N/A	97.02	#N/A	3.14	#N/A	#N/A	-0.03	#N/A
V32	1997	8/11/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	8/11/1997	6.053	#N/A	#N/A	#N/A	#N/A	#N/A	4.22	#N/A	93.1	#N/A	#N/A	#N/A	#N/A	2.82	#N/A	#N/A	-0.03	#N/A	11.5
V32	1997	9/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	3.7	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	9/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	9/30/1997	#N/A	6.931	#N/A	#N/A	#N/A	#N/A	#N/A	6.98	#N/A	#N/A	#N/A	#N/A	148.74	#N/A	7.8	#N/A	#N/A	-0.03	#N/A
V32	1997	9/30/1997	7.024	#N/A	#N/A	#N/A	#N/A	#N/A	7.37	#N/A	141.82	#N/A	#N/A	#N/A	#N/A	8.1	#N/A	#N/A	-0.03	#N/A	14.4
V32	1997	10/20/1997	7.673	#N/A	#N/A	#N/A	#N/A	#N/A	0.95	#N/A	3.57	4.21	#N/A	#N/A	#N/A	3.06	#N/A	#N/A	1.25	#N/A	19.5
V32	1997	10/20/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	10/20/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	12/22/1997	7.656	#N/A	#N/A	#N/A	#N/A	#N/A	0.34	#N/A	10.88	4.31	#N/A	#N/A	#N/A	1.66	#N/A	#N/A	0.095	#N/A	17.2
V32	1997	12/22/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	5/18/1998	4.182	#N/A	#N/A	#N/A	#N/A	#N/A	3.65	#N/A	82.5	4.7	#N/A	#N/A	#N/A	2.51	#N/A	#N/A	-0.03	#N/A	9.7
V32	1998	5/18/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	6/29/1998	8.551	#N/A	#N/A	#N/A	#N/A	#N/A	11.82	#N/A	291.97	3.37	#N/A	#N/A	#N/A	6.18	#N/A	#N/A	-0.03	#N/A	17
V32	1998	6/29/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	6/29/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	9/14/1998	10.116	#N/A	#N/A	#N/A	#N/A	#N/A	11.22	#N/A	240.05	3.51	#N/A	#N/A	#N/A	7.67	#N/A	#N/A	-0.03	#N/A	16.5
V32	1998	9/14/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	9/14/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A

STATION	YEAR	DATE	NI-D mg/L	NI-T mg/L	NO2 mg/L	O2NO3 mg	NO3 mg/L	N-T mg/L	PB-D mg/L	PB-T mg/L	P-D mg/L	PH-F pH un	PH-L pH un	PO4 mg/L	P-T mg/L	SB-D mg/L	SB-T mg/L	S-D mg/L	SE-D mg/L	SE-T mg/L	SI-D mg/L
V32	1999	5/18/1999	5.943	#N/A	#N/A	#N/A	#N/A	#N/A	4.99	#N/A	119.64	5.82	#N/A	#N/A	#N/A	1.49	#N/A	#N/A	-0.03	#N/A	9.3
V32	1999	5/18/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	6/18/1999	7.451	#N/A	#N/A	#N/A	#N/A	#N/A	9.6	#N/A	257.07	4.61	#N/A	#N/A	#N/A	9.06	#N/A	#N/A	-0.03	#N/A	11.5
V32	1999	6/18/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	8/12/1999	11.64	#N/A	#N/A	#N/A	#N/A	#N/A	6.8	#N/A	18.9	4.81	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	28.7
V32	1999	8/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	8/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	16.03	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	10/12/1999	9.756	#N/A	#N/A	#N/A	#N/A	#N/A	4.71	#N/A	19.38	3.04	#N/A	#N/A	#N/A	5.26	#N/A	#N/A	-0.005	#N/A	12.2
V32	1999	10/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	10/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	12/13/1999	9.578	#N/A	#N/A	#N/A	#N/A	#N/A	1.62	#N/A	9.11	3.98	#N/A	#N/A	#N/A	5.52	#N/A	#N/A	-0.005	#N/A	19.5
V32	1999	12/13/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	12/13/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2000	5/31/2000	8.023	#N/A	#N/A	#N/A	#N/A	#N/A	6.49	#N/A	18	3.55	#N/A	#N/A	#N/A	-0.03	#N/A	#N/A	-0.005	#N/A	16.7
V32	2000	5/31/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2000	5/31/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2000	9/12/2000	7.6	#N/A	#N/A	#N/A	#N/A	#N/A	6.14	#N/A	304	3.81	#N/A	#N/A	#N/A	6.29	#N/A	#N/A	-0.005	#N/A	13.7
V32	2000	9/12/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2001	6/13/2001	8.436	#N/A	#N/A	#N/A	#N/A	#N/A	1.49	#N/A	-1	3.6	#N/A	#N/A	#N/A	-0.03	#N/A	#N/A	0.93	#N/A	16.4
V32	2001	6/13/2001	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2001	6/13/2001	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2003	6/17/2003	13.049	12.834	#N/A	#N/A	#N/A	#N/A	0.272	0.27	-0.01	#N/A	#N/A	#N/A	-0.01	-0.002	-0.002	#N/A	#N/A	3.346	#N/A
V32	2003	6/17/2003	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	3.287	#N/A	#N/A
V32	2003	9/15/2003	13.094	#N/A	#N/A	#N/A	#N/A	#N/A	0.682	#N/A	-0.01	2.8	#N/A	#N/A	#N/A	-0.002	#N/A	#N/A	1.231	#N/A	#N/A
V32	2003	9/15/2003	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	6/28/2004	19.8	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.2	2.2	#N/A	#N/A	#N/A	-0.001	#N/A	#N/A	0.18	#N/A	#N/A
V32	2004	6/28/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	6/28/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.11	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	53.7
V32	2004	9/20/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.84	#N/A	-0.15	3.2	#N/A	#N/A	#N/A	0.002	#N/A	#N/A	0.13	#N/A	41.2
V32	2004	9/20/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	9/20/2004	23.2	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2005	6/22/2005	14.1	#N/A	#N/A	#N/A	#N/A	#N/A	0.54	#N/A	-0.15	3.2	#N/A	#N/A	#N/A	0.002	#N/A	#N/A	0.1	#N/A	36.9
V32	2005	6/22/2005	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2005	9/19/2005	22.3	#N/A	#N/A	#N/A	#N/A	#N/A	0.52	#N/A	-0.15	3.3	#N/A	#N/A	#N/A	0.001	#N/A	#N/A	0.12	#N/A	33.2
V32	2005	9/19/2005	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2006	6/8/2006	14.1	#N/A	#N/A	#N/A	#N/A	#N/A	0.69	#N/A	-0.15	3.5	#N/A	#N/A	#N/A	0.001	#N/A	#N/A	0.077	#N/A	20.3
V32	2006	9/18/2006	11.3	#N/A	#N/A	#N/A	#N/A	#N/A	0.55	#N/A	-0.15	3.1	#N/A	#N/A	#N/A	0.001	#N/A	#N/A	0.094	#N/A	15.9
V32	2007	6/11/2007	12.8	#N/A	#N/A	#N/A	#N/A	#N/A	1.2	#N/A	-0.15	2.9	#N/A	#N/A	#N/A	0.001	#N/A	#N/A	0.072	#N/A	16.9
V32	2007	10/1/2007	19	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-15	3.1	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	22
V32	2008	6/1/2008	15	#N/A	#N/A	#N/A	#N/A	#N/A	0.35	#N/A	-0.15	2.7	#N/A	#N/A	#N/A	0.001	#N/A	#N/A	0.11	#N/A	16
V32	2008	9/29/2008	16.7	#N/A	#N/A	#N/A	#N/A	#N/A	1	#N/A	-0.15	3.7	#N/A	#N/A	#N/A	0.002	#N/A	#N/A	0.074	#N/A	13

STATION	YEAR	DATE	SI-T mg/L	SN-D mg/L	SN-T mg/L	SO4 mg/L	SR-D mg/L	SR-T mg/L	S-T mg/L	SWEL m	TDS mg/L	TE-D mg/L	TEMP-F °C	TEMP-L °C	TE-T mg/L	TH-D mg/L	TH-T mg/L	TI-D mg/L	TI-T mg/L	TKN mg/L	TL-D mg/L
V32	1988	6/15/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	14	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/7/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	9	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/11/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/20/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/27/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	8	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	8/24/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	5	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	8/24/1988	#N/A	#N/A	#N/A	-1	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	9/13/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	3	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	9/13/1988	#N/A	#N/A	#N/A	-1	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	10/6/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	8	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	10/6/1988	#N/A	#N/A	#N/A	-1	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	7.39	#N/A	-0.01	#N/A	#N/A	0.78	2060	#N/A	8080	#N/A	#N/A	21.6	#N/A	#N/A	#N/A	#N/A	-0.001	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	6390	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	9.94	#N/A	0.02	#N/A	#N/A	0.64	1890	#N/A	10100	#N/A	#N/A	13.6	#N/A	#N/A	#N/A	#N/A	0.01	#N/A	#N/A
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	#N/A	#N/A	#N/A	7110	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1996	5/27/1996	#N/A	#N/A	#N/A	#N/A	0.69	#N/A	#N/A	#N/A	1470	#N/A	#N/A	9	#N/A	#N/A	#N/A	-0.05	#N/A	#N/A	#N/A
V32	1996	5/27/1996	#N/A	#N/A	#N/A	3775	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1996	9/11/1996	#N/A	#N/A	#N/A	#N/A	1.06	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	9	#N/A	#N/A	#N/A	-0.05	#N/A	#N/A	#N/A
V32	1996	9/11/1996	#N/A	#N/A	#N/A	5459	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1996	9/11/1996	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	5/12/1997	10.4	#N/A	0.14	#N/A	#N/A	0.664	1250	#N/A	#N/A	#N/A	#N/A	4.5	#N/A	#N/A	#N/A	#N/A	0.034	#N/A	#N/A
V32	1997	5/12/1997	#N/A	#N/A	#N/A	3689	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	5/12/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	6/30/1997	13.8	-0.01	-0.01	#N/A	1.006	1.048	1074	#N/A	#N/A	#N/A	#N/A	19.6	#N/A	#N/A	#N/A	0.03	0.033	#N/A	#N/A
V32	1997	6/30/1997	#N/A	#N/A	#N/A	3319	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	7/22/1997	#N/A	-0.01	#N/A	#N/A	1.2	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	12.1	#N/A	#N/A	#N/A	0.3	#N/A	#N/A	#N/A
V32	1997	7/22/1997	#N/A	#N/A	#N/A	3868	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	7/22/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	8/6/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	8/11/1997	13.4	#N/A	-0.01	#N/A	#N/A	0.952	1315	#N/A	#N/A	#N/A	#N/A	15.8	#N/A	#N/A	#N/A	#N/A	0.029	#N/A	#N/A
V32	1997	8/11/1997	#N/A	#N/A	#N/A	3944	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	8/11/1997	#N/A	-0.01	#N/A	#N/A	0.876	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.026	#N/A	#N/A	#N/A
V32	1997	9/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	5.2	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	9/30/1997	#N/A	#N/A	#N/A	10493	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	9/30/1997	14.8	#N/A	-0.01	#N/A	#N/A	0.811	3498	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.059	#N/A	#N/A
V32	1997	9/30/1997	#N/A	-0.01	#N/A	#N/A	0.84	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.053	#N/A	#N/A	#N/A
V32	1997	10/20/1997	#N/A	#N/A	#N/A	#N/A	1.094	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	2.1	#N/A	#N/A	#N/A	0.025	#N/A	#N/A	#N/A
V32	1997	10/20/1997	#N/A	#N/A	#N/A	9267	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	10/20/1997	#N/A	0.1	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	12/22/1997	#N/A	0.06	#N/A	#N/A	0.941	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A
V32	1997	12/22/1997	#N/A	#N/A	#N/A	8558	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	5/18/1998	#N/A	-0.01	#N/A	#N/A	0.607	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.081	#N/A	#N/A	#N/A
V32	1998	5/18/1998	#N/A	#N/A	#N/A	3003	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	6/29/1998	#N/A	-0.01	#N/A	#N/A	1.012	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.112	#N/A	#N/A	#N/A
V32	1998	6/29/1998	#N/A	#N/A	#N/A	3961	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	6/29/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	9/14/1998	#N/A	-0.01	#N/A	#N/A	0.907	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	2	#N/A	#N/A	#N/A	0.018	#N/A	#N/A	#N/A
V32	1998	9/14/1998	#N/A	#N/A	#N/A	10947	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	9/14/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A

STATION	YEAR	DATE	SI-T mg/L	SN-D mg/L	SN-T mg/L	SO4 mg/L	SR-D mg/L	SR-T mg/L	S-T mg/L	SWEL m	TDS mg/L	TE-D mg/L	TEMP-F °C	TEMP-L °C	TE-T mg/L	TH-D mg/L	TH-T mg/L	TI-D mg/L	TI-T mg/L	TKN mg/L	TL-D mg/L
V32	1999	5/18/1999	#N/A	-0.01	#N/A	#N/A	0.688	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.022	#N/A	#N/A	#N/A
V32	1999	5/18/1999	#N/A	#N/A	#N/A	9365	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	6/18/1999	#N/A	-0.01	#N/A	#N/A	1.029	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	12	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A
V32	1999	6/18/1999	#N/A	#N/A	#N/A	9550	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	8/12/1999	#N/A	-0.01	#N/A	#N/A	0.711	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	19	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A
V32	1999	8/12/1999	#N/A	#N/A	#N/A	17010	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	8/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	10/12/1999	#N/A	-0.01	#N/A	#N/A	0.677	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	1	#N/A	#N/A	#N/A	0.068	#N/A	#N/A	#N/A
V32	1999	10/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	10/12/1999	#N/A	#N/A	#N/A	16970	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	12/13/1999	#N/A	-0.01	#N/A	#N/A	0.763	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A
V32	1999	12/13/1999	#N/A	#N/A	#N/A	14645	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	12/13/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2000	5/31/2000	#N/A	-0.01	#N/A	#N/A	0.679	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	11	#N/A	#N/A	#N/A	0.064	#N/A	#N/A	#N/A
V32	2000	5/31/2000	#N/A	#N/A	#N/A	15389	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2000	5/31/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2000	9/12/2000	#N/A	-0.01	#N/A	#N/A	0.662	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	5.7	#N/A	#N/A	#N/A	0.107	#N/A	#N/A	#N/A
V32	2000	9/12/2000	#N/A	#N/A	#N/A	17040	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2001	6/13/2001	#N/A	-0.01	#N/A	#N/A	0.51	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	8.7	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2001	6/13/2001	#N/A	#N/A	#N/A	14666	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2001	6/13/2001	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	2.179	#N/A	#N/A	#N/A
V32	2003	6/17/2003	#N/A	-0.002	-0.002	25050	1.062	1.075	8576.6	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.003	#N/A	-0.002
V32	2003	6/17/2003	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.002	#N/A	#N/A	#N/A
V32	2003	9/15/2003	#N/A	0.022	#N/A	6431	1.089	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	5.2	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.002
V32	2003	9/15/2003	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.004	#N/A	#N/A	#N/A
V32	2004	6/28/2004	#N/A	-0.001	#N/A	41100	0.15	#N/A	#N/A	#N/A	#N/A	-0.001	#N/A	17.3	#N/A	0.019	#N/A	0.005	#N/A	#N/A	0.0096
V32	2004	6/28/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	6/28/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	6/28/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	9/20/2004	#N/A	-0.001	#N/A	93000	0.68	#N/A	#N/A	#N/A	#N/A	-0.001	1.8	#N/A	#N/A	0.016	#N/A	#N/A	#N/A	#N/A	0.012
V32	2004	9/20/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	9/20/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.004	#N/A	#N/A	#N/A
V32	2005	6/22/2005	#N/A	-0.001	#N/A	59900	0.044	#N/A	#N/A	#N/A	#N/A	-0.001	10	#N/A	#N/A	0.0098	#N/A	0.002	#N/A	#N/A	0.0081
V32	2005	6/22/2005	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2005	9/19/2005	#N/A	-0.001	#N/A	59300	0.064	#N/A	#N/A	#N/A	#N/A	-0.001	6.7	#N/A	#N/A	0.012	#N/A	0.004	#N/A	#N/A	0.012
V32	2005	9/19/2005	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2006	6/8/2006	#N/A	-0.001	#N/A	53900	0.99	#N/A	#N/A	#N/A	#N/A	-0.001	11.5	#N/A	#N/A	0.012	#N/A	0.002	#N/A	#N/A	0.012
V32	2006	9/18/2006	#N/A	-0.001	#N/A	65900	1.16	#N/A	#N/A	#N/A	#N/A	-0.001	3.8	#N/A	#N/A	0.0087	#N/A	0.004	#N/A	#N/A	0.014
V32	2007	6/11/2007	#N/A	-0.001	#N/A	54500	1	#N/A	#N/A	#N/A	#N/A	-0.001	12.1	#N/A	#N/A	0.023	#N/A	0.005	#N/A	#N/A	0.021
V32	2007	10/1/2007	#N/A	-3	#N/A	67400	0.4	#N/A	#N/A	#N/A	#N/A	#N/A	0.4	#N/A	#N/A	#N/A	#N/A	-0.5	#N/A	#N/A	#N/A
V32	2008	6/1/2008	#N/A	-0.001	#N/A	78200	1.04	#N/A	#N/A	#N/A	#N/A	-0.001	8.6	#N/A	#N/A	0.009	#N/A	0.006	#N/A	#N/A	0.011
V32	2008	9/29/2008	#N/A	-0.001	#N/A	53700	0.75	#N/A	#N/A	#N/A	#N/A	-0.001	4	#N/A	#N/A	0.013	#N/A	0.005	#N/A	#N/A	0.0078

STATION	YEAR	DATE	TL-T mg/L	TOC mg/L	TSS mg/L	TURB NTU	U-D mg/L	U-T mg/L	V-D mg/L	V-T mg/L	WL ELEV m	WL-S m	W-T mg/L	ZN-D mg/L	ZN-IH mg/L	ZN-T mg/L	ZR-D mg/L	ZR-T mg/L
V32	1988	6/15/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	7/7/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.021	#N/A	#N/A
V32	1988	7/11/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.01	#N/A	#N/A
V32	1988	7/20/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.019	#N/A	#N/A
V32	1988	7/27/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	8/24/1988	#N/A	#N/A	-1	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.008	#N/A	#N/A
V32	1988	8/24/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	9/13/1988	#N/A	#N/A	-1	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.015	#N/A	#N/A
V32	1988	9/13/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1988	10/6/1988	#N/A	#N/A	-1	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.003	#N/A	#N/A
V32	1988	10/6/1988	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	116	2	#N/A	#N/A	#N/A	-0.002	#N/A	#N/A	#N/A	#N/A	#N/A	1680	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1994	8/9/1994	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	#N/A	#N/A	38	170	#N/A	#N/A	#N/A	-0.002	#N/A	#N/A	#N/A	#N/A	#N/A	1220	#N/A	#N/A
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1995	5/9/1995	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1996	5/27/1996	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.05	#N/A	#N/A	#N/A	#N/A	851	#N/A	#N/A	#N/A	#N/A
V32	1996	5/27/1996	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1996	9/11/1996	#N/A	#N/A	67	#N/A	#N/A	#N/A	-0.05	#N/A	#N/A	#N/A	#N/A	964.9	#N/A	#N/A	#N/A	#N/A
V32	1996	9/11/1996	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1996	9/11/1996	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	5/12/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.047	#N/A	#N/A	2.55	#N/A	#N/A	726.06	#N/A	#N/A
V32	1997	5/12/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	5/12/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	5/12/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	6/30/1997	#N/A	#N/A	17	#N/A	#N/A	#N/A	-0.005	-0.005	#N/A	#N/A	45.7	849.5	#N/A	843.64	#N/A	#N/A
V32	1997	6/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	7/22/1997	#N/A	#N/A	45	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	1986	#N/A	#N/A	#N/A	#N/A
V32	1997	7/22/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	7/22/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	8/6/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	8/11/1997	#N/A	#N/A	51	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	-0.03	#N/A	#N/A	865.48	#N/A	#N/A
V32	1997	8/11/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	8/11/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	832.97	#N/A	#N/A	#N/A	#N/A
V32	1997	9/30/1997	#N/A	#N/A	29	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	9/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	9/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	38.58	#N/A	#N/A	1652	#N/A	#N/A
V32	1997	9/30/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	1682	#N/A	#N/A	#N/A	#N/A
V32	1997	10/20/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	1957	#N/A	#N/A	#N/A	#N/A
V32	1997	10/20/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	10/20/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1997	12/22/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	1788.2	#N/A	#N/A	#N/A	#N/A
V32	1997	12/22/1997	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	5/18/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	706.1	#N/A	#N/A	#N/A	#N/A
V32	1998	5/18/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	6/29/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.184	#N/A	#N/A	#N/A	#N/A	1800.2	#N/A	#N/A	#N/A	#N/A
V32	1998	6/29/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	6/29/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	9/14/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	2179.79	#N/A	#N/A	#N/A	#N/A
V32	1998	9/14/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1998	9/14/1998	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A

STATION	YEAR	DATE	TL-T mg/L	TOC mg/L	TSS mg/L	TURB NTU	U-D mg/L	U-T mg/L	V-D mg/L	V-T mg/L	WL ELEV m	WL-S m	W-T mg/L	ZN-D mg/L	ZN-IH mg/L	ZN-T mg/L	ZR-D mg/L	ZR-T mg/L
V32	1999	5/18/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	1155.76	#N/A	#N/A	#N/A	#N/A
V32	1999	5/18/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	6/18/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	2243.28	#N/A	#N/A	#N/A	#N/A
V32	1999	6/18/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	8/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	4044	#N/A	#N/A	#N/A	#N/A
V32	1999	8/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	8/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.503	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	10/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	2886	#N/A	#N/A	#N/A	#N/A
V32	1999	10/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	10/12/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	12/13/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	3101.95	#N/A	#N/A	#N/A	#N/A
V32	1999	12/13/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	1999	12/13/1999	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2000	5/31/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	3410.8	#N/A	#N/A	#N/A	#N/A
V32	2000	5/31/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2000	5/31/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2000	9/12/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	1099.82	#N/A	#N/A	#N/A	#N/A
V32	2000	9/12/2000	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2001	6/13/2001	#N/A	#N/A	99	#N/A	#N/A	#N/A	-0.005	#N/A	#N/A	#N/A	#N/A	6700	#N/A	#N/A	#N/A	#N/A
V32	2001	6/13/2001	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2001	6/13/2001	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2003	6/17/2003	-0.002	#N/A	#N/A	#N/A	#N/A	#N/A	-0.001	-0.001	#N/A	#N/A	-0.03	2994.352	#N/A	5106.077	#N/A	#N/A
V32	2003	6/17/2003	#N/A	#N/A	292	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2003	9/15/2003	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	0.002	#N/A	#N/A	#N/A	#N/A	8231.812	#N/A	#N/A	#N/A	#N/A
V32	2003	9/15/2003	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	6/28/2004	#N/A	#N/A	-1	#N/A	0.184	#N/A	-0.001	#N/A	#N/A	#N/A	#N/A	7510	#N/A	#N/A	-0.01	#N/A
V32	2004	6/28/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	6/28/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	6/28/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	9/20/2004	#N/A	#N/A	43	#N/A	0.562	#N/A	0.004	#N/A	#N/A	#N/A	#N/A	10700	#N/A	#N/A	-0.01	#N/A
V32	2004	9/20/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2004	9/20/2004	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2005	6/22/2005	#N/A	#N/A	11	#N/A	0.502	#N/A	0.004	#N/A	#N/A	#N/A	#N/A	10200	#N/A	#N/A	-0.01	#N/A
V32	2005	6/22/2005	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2005	9/19/2005	#N/A	#N/A	28	#N/A	0.611	#N/A	0.005	#N/A	#N/A	#N/A	#N/A	10500	#N/A	#N/A	-0.01	#N/A
V32	2005	9/19/2005	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
V32	2006	6/8/2006	#N/A	#N/A	#N/A	#N/A	0.393	#N/A	0.004	#N/A	#N/A	#N/A	#N/A	9820	#N/A	#N/A	-0.01	#N/A
V32	2006	9/18/2006	#N/A	#N/A	#N/A	#N/A	0.409	#N/A	0.005	#N/A	#N/A	#N/A	#N/A	10600	#N/A	#N/A	-0.01	#N/A
V32	2007	6/11/2007	#N/A	#N/A	11	#N/A	0.946	#N/A	0.007	#N/A	#N/A	#N/A	#N/A	10300	#N/A	#N/A	-0.01	#N/A
V32	2007	10/1/2007	#N/A	#N/A	26	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	11300	#N/A	#N/A	-2	#N/A
V32	2008	6/1/2008	#N/A	#N/A	29	#N/A	0.75	#N/A	0.006	#N/A	#N/A	#N/A	#N/A	12500	#N/A	#N/A	-0.01	#N/A
V32	2008	9/29/2008	#N/A	#N/A	78	#N/A	1.06	#N/A	0.005	#N/A	#N/A	#N/A	#N/A	7380	#N/A	#N/A	-0.01	#N/A

Attachment B

Results of an Experiment to Evaluate Solubility Limits for Category 1 Waste Rock

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Memo

To:	Jim Scott, PolyMet	**Date:**	February 16, 2011
cc:	Peter Hinck, Barr	**From:**	Stephen Day
	Richard Patelke, PolyMet		
Subject:	Results of an Experiment to Evaluate Solubility Limits for Category 1 Waste Rock, NorthMet Project	**Project #:**	1UP005.001
	DRAFT		

1 Background

1.1 Introduction

The water quality prediction method for waste rock stockpiles and pit walls for the NorthMet Project relies on scale-up of laboratory tests (humidity cells) and consideration of constraining concentrations (solubility limits or concentration caps) resulting from the finite solubility of secondary minerals produced by weathering processes. As described by SRK (2010), solubility limits for some elements can be estimated using thermodynamic data for well-understood minerals (for example, sulfate constrained by gypsum solubility); however, the majority of elements occur as trace components in major secondary minerals (for example, cobalt co-precipitated with ferric oxyhydroxides). For this reason, solubility cannot be readily evaluated using thermodynamic data for the secondary minerals and other analogous data sources are needed.

SRK (2007)[1] previously used data from the AMAX field experiments directed by the MNDNR as analogs for the low sulfur Category 1/2 (now Category 1) waste rock stockpiles. For parameters not monitored by the MNDNR, laboratory-scale data obtained by PolyMet Inc. for the NorthMet Project were used as analogs. This approach was less than ideal because the field experiments were performed on rock classified as ore, whereas the laboratory experiments are affected by dilution resulting from the high liquid-to-solid leaching ratios. Concentrations from the field experiments may over-estimate solubility limits while laboratory experiments may under-estimate solubility limits.

This memorandum provides results of an experiment to directly evaluate solubility limits for Category 1 waste rock.

1.2 Method Design

The method used is referred to as a Sequential Meteoric Water Mobility Procedure (SMWMP). It is designed to simulate the accumulation of solutes that occur as a package of water contacts oxidizing waste along a flowpath. SRK (2000)[2] used the method as part of assessment of the Fort Knox Mine's True North Project for the Alaska Department of Environmental Conservation (AKDEC).

[1] SRK Consulting. 2007. RS53/RS42 – Waste Rock Characteristics/Waste Water Quality Modeling - Waste Rock and Lean Ore – NorthMet Project – DRAFT 01. March 9, 2007.
[2] SRK Consulting 2000. Review of geochemical issues True North Project Fort Knox Mine. Prepared for Alaska Department of Environmental Conservation. . December 2000. SRK Project 1UA004.01.

It was recognized that single pass leach procedures may not give a reliable indication of contact water chemistry because:

- the time frame of leaching was not sufficient to allow the solution chemistry to approach the theoretical limits indicated by the thermodynamic properties;
- the leachate produced by dissolution of readily soluble minerals did not react with less soluble minerals;
- the high liquid to solid ratio resulted in complete dissolution of a mineral component before the solubility limit could be reached.

Concerns have been raised by others about the limitations of single pass leach tests[3]. The method has subsequently been used to assess solubility limits for several projects in the Yukon Territory for a Canadian federal process[4], British Columbia for a provincial and federal process[5] and Chile.

The method is a modification of the Nevada Department of Environmental Protection's (NVDEP 1990[6]) Meteoric Water Mobility Procedure Method (MWMP) which results in a single pass of leachate at about a ratio of 1 L of water to 1 kg of waste. The SMWMP involves recovery of leachate from each leaching step and re-application of the leachate (less a split retained from analysis) to an unleached sample of the waste. The procedure is repeated until insufficient leachate is available for analysis. The leaching ratio (1 L/kg) is maintained for each step. As a result, the mass of sample leached in each step decreases due to loss of water for analysis and retention in the solids. The leaching ratio in the final step is therefore $^{1}/_{n}$ L/kg where n is the number of leaching steps.

To be effective, the method must be performed on well-oxidized waste so that soluble weathering products are available for leaching.

2 Methods

2.1 Composite Preparation

Coarse rejects from preparation of exploration drill core for analysis from 2005 onwards were available for testing. Six composites with sulfur concentrations of 0.02%, 0.04%, 0.06%, 0.08%, 0.10% and 0.12% were prepared by PolyMet to represent the expected range of sulfur concentrations in Category 1 waste rock. To obtain these concentrations and sufficient material (35 kg) for the procedure, it was necessary to composite materials from different locations throughout the project area.

Sample intervals used to prepare the composites are provided in Attachment 1.

Splits of each composite were analyzed for total metals and sulfur using PolyMet's exploration method. Metal analysis was performed using both an aqua regia and four-acid digestion.

2.2 Leach Procedure

2.2.1 Initial Leach

Initial sequential shake flask extraction procedures were performed to determine if the samples had oxidized in storage and were suitable for the SMWMP. This procedure involved in a 3:1 liquid to solid extraction in which the leachate was recovered and re-applied to an unleached split. Seven

[3] Schafer, W. 2008. Critical Review of Geochemical Prediction at Mines. Paper presented at Northwest Mining Association meeting, Reno, NV. December 3, 2008.
[4] SRK Consulting, 2004. Results of sequential leach tests on Blue Pit waste rock Brewery Creek Mine. Report prepared for Viceroy Minerals Corporation. SRK Project 1CV001.001. February 2004.
[5] SRK Consulting. 2006. Galore Creek Project ML/ARD Characterization Report. Report prepared for Novagold Resources Inc. SRK Project. 1CR003.002. May 2006.
[6] Nevada Division of Environmental Protection. 1990. Meteoric Water Mobility Procedure, Bureau of Mining Regulation and Reclamation, Nevada Division of Environmental Protection, 9/19/90. 6 pages. http://ndep.nv.gov/bmrr/mobilty1.pdf.

24-hour leaching steps were performed and each leachate was analyzed for pH and conductivity. The final leachate was also analyzed for sulfate. The procedure confirmed the composites contained oxidation products and were suitable for the SMWMP.

2.2.2 SMWMP

The leaching procedure consisted of a series of extractions performed in columns. The leachate from each column was used as the feed for leaching of a fresh sample.

Each leaching step was performed in a leach column as follows:

- The sample was placed in a 15.2 cm diameter column.
- The volume of feed water was determined as the amount needed to achieve a 1:1 leachate to sample ratio using the procedure described by NVDEP (1990). This volume was dripped through the column over a period of 96 hours.
- 200 mL of the resulting leachate is removed for analysis and the remainder is used for the next leach step.

Due to the decrease in leachate volume from retention of water in the solids, the mass of sample was calculated to provide sufficient leachate for seven leaching steps.

Leachates were analyzed for at least pH, Eh, electrical conductivity, ORP, dissolved major cations and anions and trace ions. The mass of sample used in each step, the volume of water applied and the volume of water collected from each stage were recorded.

2.3 Quality Control

A method blank using exactly the same procedure was used to assess the potential effects of the column construction materials. The resulting leachate had a pH 4.99 and conductivity of 3 µS/cm. Major ions were not detected in the leachates though several trace elements were detected at below parts per billion levels. The most significant detection was copper at 0.004 mg/L.

Leachate ion balances were normally within ±10% with exceptions occurring in the first and second relatively dilute leach cycles. Imbalances were generally positive (cation excess) with the exception of the two samples containing the higher sulfur concentrations which yielded some negative imbalances between 5 and 10%.

Due to the small volume of leachate available in the final leaching step, and the presence of highest ion strengths (therefore requiring leachate dilution), analytical precision was lowest for these analyses.

3 Results

3.1 Solid Characteristics

Solids characteristics are provided in Table 1.

Table 1: Composite Solids Characteristics

Composite	Total S	As	Cd	Co	Cr	Cu	Fe	Ni	Sb	Zn
	%	mg/kg	mg/kg	mg/kg	mg/kg	mg/kg	%	mg/kg	mg/kg	mg/kg
CAPCON 2	0.02	0.1	0.03	39.7	77	117.5	4.44	189	<0.05	49
CAPCON 4	0.04	0.3	0.09	44.3	94	214	5.61	185.5	0.1	70
CAPCON 6	0.05	0.6	0.07	45.8	72	284	5.48	217	0.06	65
CAPCON 8	0.07	0.4	0.09	44.1	91	354	5.58	206	0.07	72
CAPCON 10	0.1	2.8	0.14	48.2	90	457	6.05	226	0.12	76
CAPCON 12	0.11	2	0.09	51.7	54	560	6.57	278	0.14	77

1. Element concentrations determined by aqua regia.

Source:G:\PolyMet Mining\1UP005.01_Northmet_project_2004\Testwork\2010-07_SWMP\4.Interpretations\[SMWMP_Graphs_1UP005001_SJD.ver00.xlsx]

3.2 SMWMP Leachates

Attachment 2 shows charts for each element.

3.2.1 Major Ions

All leachates reached stable pHs after the first leach step. The range of stable pHs was from 7.9 to 8.4 with lowest pHs from the higher sulfur samples. Anion chemistry was dominated by sulfate and alkalinity (as bicarbonate), but chloride reached 490 mg/L for composite 10. Cation chemistry was dominated by sodium then calcium and magnesium. Sulfate increased as the experiment continued resulting in final concentrations ranging from 110 mg/L to 910 mg/L. Sulfate leaching was exactly correlated with sulfur content.

Alkalinity showed the reverse trend with lowest alkalinity correlated with higher sulfur content. Alkalinity increased mostly rapidly earliest then tended to stabilize except for the two lowest sulfur content samples which showed increasing concentrations up to 300 mg$CaCO_3$/L.

Chloride generally increased consistently with the exception of steps 3 to 4 for Sample 10.

Cations showed upward trends with absolute values and slopes of increases being greater for the higher sulfur content samples. A tendency toward flattening was apparent for the last step but as noted in Section 2.3; results for the last step were affected by the ionic strength and small sample available for analysis.

3.2.2 Trace Elements

Behavior of trace elements can be divided into the following broad groups:

- Concentrations increased in parallel with sulfate and did not reach a stable well-defined level – Ba, B, Cr (erratically), Cu, Mo, Ni, Se.
- Concentrations reached stable well-defined levels – Ba, F, Sb, As, Co, Cu, Mn, Ni, P, Si.
- Low concentrations showed neither increasing nor decreasing trends but were erratic – Ag, Cd, Fe, Hg, Pb, Tl, Zn.
- Concentrations decreased as the test progressed – Al, V.
- Concentrations were consistently undetected – Be

Barium, copper and nickel are shown in two of the groups because concentrations increased for samples with higher sulfur content but appeared to be stable for samples with lower sulfur content.

Many elements yielded strong correlations of leachable concentrations with bulk sulfur content, including arsenic, barium, chromium (weak), copper, nickel and selenium. Molybdenum leaching was negatively correlated with sulfur content. A few elements also yielded leachable concentrations correlated with their bulk content of the element, including arsenic, copper, molybdenum, nickel and selenium.

Elements for which correlations were not apparent included antimony, cadmium, cobalt, lead, manganese, silver, thallium, vanadium and zinc. These elements occurred at low concentrations in leachates and/or the solid phase.

4 Discussion

4.1 Sample Representation

The targeted sulfur range in the samples was achieved with coverage of the 0 to 90th percentile of the sulfur range. As a result of the tendency for copper concentrations to increase as sulfur content also increases, copper concentrations also covered a similar percentile range.

For other elements, the correlation with sulfide content is weaker and the process of compositing resulted in near median (50th percentile) metal content in most samples except the lower sulfur content samples.

4.2 Leachate Chemistry

4.2.1 Major Elements

Sulfate leaching was consistent with the oxidation of sulfides (pyrrhotite and chalcopyrite) in storage resulting in generation of sulfate and acidity. Sulfate leaching was strongly positively correlated with sulfur content, and the spread of pH and alkalinity showed that more acidity was generated at higher sulfide content.

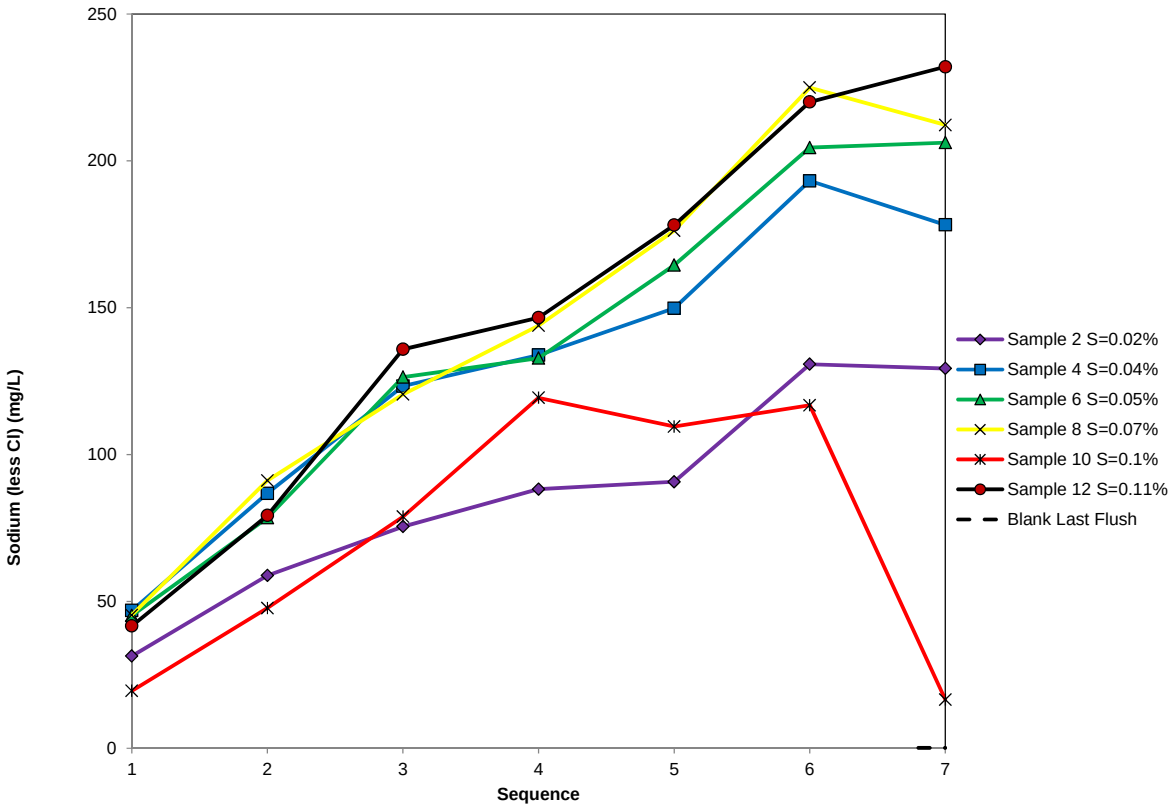
The source of alkalinity for neutralization of sulfuric acid was likely to be carbonate minerals. Saturation indices for calcite and dolomite show that both minerals approached then exceeded saturation for each test[7] (Attachment 3). Carbonate minerals may be present as hydrothermal alteration products of plagioclase and olivine, or as weathering products accumulated while in storage. While some sodium probably accumulated in SMWMP leachates due to leaching of trapped sodium chloride connate waters and drill fluids, the presence of elevated sodium concentrations indicates that sodium and carbonate minerals accumulated partly while in storage due to weathering of plagioclase. The relative enrichment of sodium in leachates is probably a result of retention of calcium as sparingly soluble calcium carbonate. In contrast, sodium carbonate and bicarbonate are both readily soluble, and are therefore flushed from the rock. These conclusions appear to be supported by the strong correlation of the calcium leaching trend with sulfate, but much weaker sodium correlation after allowing for sodium chloride (**Figure** 1).

Silicon leaching occurred at levels well below the concentration equivalent to sodium leaching from plagioclase. Silicon concentrations appeared to be constrained. Saturation indices were below saturation for amorphous silica but above the saturation level of kaolinite. It appears that silicon leaching was constrained by formation of an alumino-silicate weathering product.

The SMWMP data did not provide a direct indication of a solubility control for sulfate because concentrations were not constrained but it is likely that gypsum will control sulfate concentrations. The release of sodium and the implied presence of calcium carbonates showed that plagioclase will contribute both sodium and calcium. Olivine will contribute magnesium. The sulfate concentration supported by this solution chemistry is dependent on the $Ca/(Mg + Na)$ ratio because magnesium and sodium sulfates are highly soluble compared to gypsum. Humidity cell data indicate an average molar ratio of 1.2 which results in a modeled sulfate solubility limit of 2900 mg/L[8].

[7] Saturation indices were calculated using PHREEQC version 2.17.4137 and MINTEQ version 8 database. Charts showing SI trends are provided in Attachment 3.

[8] Calculated using Geochemist's Workbench assuming a temperature of 3^{o}C.



G:\PolyMet Mining\1UP005.01_Northmet_project_2004\Testwork\2010-07_SWMP\4.Interpretations\[SMWMP_Graphs_1UP005001_SJD.ver00.xlsx]

Figure 1: Trend in Sodium Leaching After Allowing for Sodium Associated with Chloride

4.2.2 Trace Elements

Unconstrained Elements

Several elements in this group (chromium, molybdenum, selenium) occur in solution as oxyanions at basic pH, or in the case of boron as uncharged $H_3BO_3^0$. Mobility of these ions may be high due to their weak tendency to adsorb. Molybdenum and selenium likely originate directly from oxidation of sulfide minerals. Chromium and boron are lithophile elements and therefore are probably leached from silicate rather than sulfide minerals.

Concentrations of these elements were well below saturation for secondary minerals they might form. The tests therefore did not provide a direct indication of solubility constraints for these elements. The last concentration measured indicates a minimum solubility constraint.

Both boron and chromium concentrations were close to those observed in the blank and therefore may reflect leaching of the column materials. Chromium co-precipitation with barite (see below) is a possible mechanism to constrain chromium concentrations (Prieto et al. 2002)[9].

Selenium concentrations were not constrained but selenium is sequestered by gypsum (Fernández-González et al. 2006[10]). The selenium to sulfate ratio indicated by the tests ($7x10^{-6}$, r=0.93) was used to estimate a selenium concentration constraint of 0.02 mg/L.

[9] Prieto, M., Fernández-González, A. and Martín-Díaz, R. 2002. Sorption of chromate ions diffusing through barite-hydrogel composites: Implications for the fate and transport of chromium in the environment. Geochimica et Cosmochimica Acta, Vol. 66, No. 5, pp. 783–795, 2002.

[10] Fernández-González, A., Andara, A., María Alía, J., and Prieto, M. 2006. Miscibility in the $CaSO_4 \cdot 2H_2O$–$CaSeO_4 \cdot 2H_2O$ system: Implications for the crystallisation and dehydration behaviour. Chemical Geology: 225 (3-4); 256-265.

Mobility of molybdenum may be controlled by formation of molybdate secondary minerals such as powellite ($CaMoO_4$) or ferrimolybdite ($Fe_2(MoO_4)_3.8H_2O$). Under basic conditions, the former is more likely though it is relatively soluble and supports molybdenum concentrations between 1 and 10 mg/L (e.g. Day et al. 2000[11]; Day et al. 2003[12]).

Barium likely originates from weathering of feldspars and probably occurs as a co-precipitated carbonate. It may be constrained by barium sulfate (barite) but the barite saturation indices for the samples yielding the higher sulfate concentrations exceeded 0 implying that barium leaching was not constrained by barite.

Copper concentrations for four of the tests (sulfur concentrations exceeding 0.04%) showed upward trends in copper concentrations; however, as a result of increasing alkalinity, saturation indices for tenorite showed weaker upward possibly stabilizing trends though the indices were well below 0. This implies that copper concentrations are possibly constrained by an oxide, and that tenorite would represent a maximum solubility for copper.

Two tests with more than 0.07% sulfur showed upward trending nickel concentrations. The other tests showed low nickel concentrations which appear to be constrained at parts per billion levels. Nickel concentrations were well below secondary minerals for which saturation indices could be calculated. However, the range of saturation indices for bunsenite (NiO) was much narrower than the range of nickel concentrations and showed flatter and similar trends for all tests implying that nickel solubility is controlled by co-precipitation with an oxide.

Well-Defined Constrained Elements

Elements in this group showed conventional behavior consistent in most cases with accumulation of solutes in sequence then stabilizing.

Antimony concentrations increased to maximum concentrations that were weakly related to sulfur content and unrelated to the low antimony content of the samples. It is likely that antimony is a trace component of sulfides, is released by oxidation then sorbed with iron oxhydroxides.

Arsenic trends in some cases showed decreases rather than increases before stabilizing. Due to the correlation with sulfur it is likely that arsenic also originates from oxidation of sulfides.

Cobalt leached at below parts per billion levels though only Sample 10 showed an increasing trend before stabilizing at 0.3 µg/L. The trend for this sample included a jump in concentrations in step 4. Manganese showed similar trends to cobalt though Sample 10 showed a peak in step 4 before following a decreasing trend.

Fluoride leaching typically showed a peak and then decreased or was stable. There was no relationship with other parameters. Fluorite saturation indices were well below 0 due to the low calcium concentrations but the indices were relatively stable implying that fluoride is constrained though not by fluorite.

Phosphorous leaching was stable with the exception of Sample 2 which contained lowest sulfur concentrations. Other samples either showed flat or slightly decreasing trends. Leachates were well over-saturated with respect to hydroxylapatite and well undersaturated with respect to a calcium phosphate phase in the thermodynamic database. However, the inverse relationship with calcium leaching implies a calcium phosphate control on leaching.

Silicon leaching was described in Section 3.2.1.

[11] Day, S., Sexsmith, K., Bowell, R. and Hockley, D. 2000. Geochemistry of Molybdenum Leaching at British Columbia Copper and Molybdenum Mines. Presented at BC Metal Leaching and Acid Rock Drainage Workshop. December 2000.

[12] Day, S., Sexsmith, K and Millard, J. 2003. Acidic Drainage From Calcareous Coarse Kimberlite Reject, Ekati Diamond Mine™, Northwest Territories, Canada. Presented at Sixth International Conference on Acid Rock Drainage, Cairns, Australia. July 12 to 18, 2003. Pages 587-600.

Erratic Non-Trending Elements

Silver, cadmium, lead, mercury, thallium and zinc all yielded concentrations usually within ten times of the low respective detection limits (5, 5, 30, 10, 10, 100 ng/L, respectively). Concentrations were erratic reflecting lower precision near the detection limits and no apparent increasing or decreasing trends were apparent. These elements are speciated as cations under these conditions and are therefore expected to be readily sorbed to ferric oxyhydroxides formed by oxidation of pyrrhotite. Since these elements are chalcophilic, they are probably released by oxidation of pyrrhotite which further favors sorption to ferric iron oxidation products.

Iron also falls into this group. Its concentrations were well above the saturation limits for ferrihydrite indicating that iron may have been released by the test as colloids which passed the 0.45 µm filter. This may in part contribute to the erratic concentrations indicated by the other elements though iron concentrations were not correlated with their concentrations.

Elements Showing Decreasing Concentrations

Aluminum concentrations decreased as the tests progressed reaching stable concentrations lower at the end of the test than at the beginning. The trend is unexpected because the dominant aluminum species is calculated to be $Al(OH)_4^-$ which is favored as leachates became more basic. Notably, greater concentrations were observed for the low sulfur tests (higher leachate pH) and vice versa which is consistent with the ionic form and control by precipitation of a basic aluminum mineral. The decrease in aluminum suggests initial release with subsequent precipitation as another ion is released. Vanadium showed very similar results and trends. Its dominant ionic forms were calculated to be anionic ($H_2VO_4^-$, HVO_4^{2-}).

Non-Detected Elements

Only beryllium was not detected consistently at its detection limit of 10 ng/L. The absence of a leaching trend does not eliminate a trend existing because it may be below the detection limit. However, concentrations were very low in the solids (close to the detection limit of 0.5 mg/kg), and beryllium most likely occurs with silicates. Also, beryllium is expected to occur in solution mainly as a cation which would favor adsorption. Low levels of beryllium leaching are therefore expected.

Summary of Solubility Constraints

Table 2 compares solubility constraints indicated by SMWMPs and other methods for Category 1 waste rock with those compiled previously to broadly cover non-acidic conditions for all waste rock categories in RS42. The SMWMP values are the highest concentrations for stable conditions. The qualifier indicates the highest value is shown but the trend was not stable.

For the majority of parameters, the SMWMP provided a useful indicator of solubility constraints and are shown as not qualified in Table 2. In a few cases, the constraints increased (Sb, Hg, Tl) from these presented in RS42. For Sb, leachates from the small MDNR reactor tests were used previously due to quality control problems for humidity cells. The difference for mercury reflects the higher detection limit for the SMWMP. Other tests have more reliably shown that mercury is readily sorbed to the rock (SRK 2005). The increase for thallium was minor.

Many parameters (As, Cd, Co, Pb, Mn, Ag, Zn) yielded lower constraints reflecting that Category 1 rock is much less mineralized than the data sources used previously in RS42.

Some parameters were not constrained by the SMWMPs. For sulfate, a calculated value based on the $Ca/(Mg + Na)$ ratio is shown. This resulted in a higher value than used previously in RS42. Copper was not constrained but the tests imply an oxide constraint. The concentration shown is for tenorite solubility at pH 7.9 which is the low end of the observed pH range.

Boron, chromium, molybdenum and nickel were not constrained by the SMWMP. For nickel, the concentration indicated in RS42 (0.86 mg/L) remains the recommended solubility limit for basic pH conditions. Boron and chromium are not expected to be readily soluble. Leaching in the SMWMP may reflect leaching from column materials. Chromium may co-precipitate with barite.

Table 2: Comparison of Solubility Constraints Indicated by SMWMP for Category 1 Waste Rock with RS42

Parameter	Units	RS42	SMWMP	SWMP Qualifier	Other
pH		8.0	7.9		
F	mg/L	-	0.57		
SO_4	mg/L	2150	910	>	2884 (gypsum model)
Al	mg/L	1.68	0.073		
Sb	mg/L	0.003	0.012		
As	mg/L	0.71	0.053		
Ba	mg/L	0.19	0.035	>	
Be	mg/L	0.0002	0.00001	>	
B	mg/L	0.76	0.46	>	
Cd	mg/L	0.00018	0.000051		
Cr	mg/L	0.0015	0.0032	>	
Co	mg/L	0.052	0.00031		
Cu	mg/L	0.092	0.0065	>	0.04 (tenorite model)
Fe	mg/L	0.81	0.012		
Pb	mg/L	0.0528	0.0011		
Mn	mg/L	0.75	0.017		
Hg	µg/L	0.006	0.03		
Mo	mg/L	0.0051	0.021	>	
Ni	mg/L	0.86	0.021	>	
Se	mg/L	0.0029	0.0077	>	0.02 (gypsum sequestration)
Ag	mg/L	0.0007	0.00005		
Tl	mg/L	0.00002	0.00005		
V	mg/L	-	0.003		
Zn	mg/L	0.09	0.0089		

Source:G:\PolyMet Mining\1UP005.01_Northmet_project_2004\Testwork\2010-07_SWMP\4.Interpretations\[SMWMP_Graphs_1UP005001_SJD.ver00.xlsx]

5 Conclusions

The Sequential Meteoric Water Mobility Procedure (SMWMP) performed to evaluate solubility constraints for Category 1 waste rock indicated the following:

- Leachates from low sulfur waste rock are expected to be basic with pH near 8.
- Leaching of Ag, Al, As, Be, Cd, Co, F, Fe, Hg, Mn, P, Pb, Sb, Si, Tl, V and Zn are better defined.
- Solubility of Ba, Cu, Se and SO_4 were not constrained in the procedure but concentrations are probably limited by secondary minerals.
- Leaching of B, Mo, Ni and Cr were not constrained by the procedure.

Attachments

Attachment 1

Composite Preparation

CAPPING CONCENTRATION SAMPLES

Richard Patelke-PolyMet

Revised, January, 2011

A) SAMPLE SELECTION PROCESS

Concepts for the use of the "capping concentration" samples are covered in memos by SRK.

The goal was obtain at least 80 pounds for each composite. Sample selection followed this sequence:

- Using block model data, the minimum, average, and maximum for copper and nickel were calculated for six subsets of Waste Rock Category 1 blocks, those with sulfur % values (not ranges or averages, but actual values) of 0.02, 0.04, 0.06, 0.08, 0.10, and 0.12.

- The assay data base was sorted down to low sulfur samples with coarse reject material that was available and accessible. Material was all selected from the 2005 drilling, which is the oldest available, and therefore has a better chance of being slightly oxidized.

- Samples were identified based first on sulfur, then copper and nickel. Weighted averages were checked based on expected sample weights.

- Samples were pulled from storage, if available, weighed, and the weighted average was recalculated for sulfur, copper, nickel.

- This was repeated until sufficient sample was found (89 to 127 lbs).

- No sample splitting was done at PolyMet.

- Samples were shipped in buckets and plastic bags via UPS to SGS / CEMI in Vancouver for compositing, and a split from each was sent to ALS-Chemex in Vancouver for analysis.

- Once samples were proved to be of "correct" chemistry, testing at SGS / CEMI proceeded, as detailed by SRK.

B) TABLES ATTACHED

Sample listing, with DDH, from, to, length, rocktype, unit, and weights

Calculated values vs. assayed values for each composite-elements above detection limits and mentioned in Steve Day memo to PolyMet on January 2, 2011

Values, where available, for assayed elements mentioned in Steve Day memo to PolyMet on January 2, 2011, where values are at or near detection limits. Boron, Chlorine, Fluorine, Selenium, and Silicon are not tested with this assay method (ALS-Chemex ME-ICP61). Thallium and Uranium were previously (2005) not tested by this method, but are now (2010).

C) MAP ATTACHED

Map showing collar locations of drill holes supplying sample to this project, with APA mine pit outlines.

SAMPLE NAME	HOLE_ID	FROM	TO	LENGTH	SAMPLE NUMBER	UNIT	ROCKTYPE	ACTUAL SAMPLE WEIGHT-POUNDS
CAPCON2	05-406C	68.00	78.00	10.00	106007	3	TROCT	10.0
CAPCON2	05-411M	325.00	329.00	4.00	111098	3	ANORTH	2.4
CAPCON2	05-430G	49.00	59.00	10.00	130002	7	ANORTH	15.2
CAPCON2	05-440G	439.00	449.00	10.00	140056	5	ANORTH	7.5
CAPCON2	05-449C	278.00	288.00	10.00	149032	4	ANORTH	10.8
CAPCON2	05-450C	168.00	178.00	10.00	150020	3	TROCT	18.1
CAPCON2	05-452C	398.00	408.00	10.00	152045	3	TROCT	19.1
CAPCON2	05-455C	238.00	248.00	10.00	155027	5	ANORTH	21.1
CAPCON2	05-459C	218.00	228.00	10.00	159029	3	ANORTH	21.3
								125.6
CAPCON4	05-416M	88.50	92.50	4.00	116026	2	TROCT	2.2
CAPCON4	05-421C	123.00	127.50	4.50	121027	6	ANORTH	2.0
CAPCON4	05-421C	132.50	137.50	5.00	121029	6	ANORTH	1.8
CAPCON4	05-425M	59.30	63.00	3.70	125015	2	TROCT	2.0
CAPCON4	05-431C	212.00	217.00	5.00	131048	1	TROCT	2.5
CAPCON4	05-435C	23.00	27.50	4.50	135001	3	ANORTH	2.6
CAPCON4	05-435C	32.50	37.00	4.50	135003	3	ANORTH	2.1
CAPCON4	05-435C	64.00	68.50	4.50	135010	3	ANORTH	2.2
CAPCON4	05-439G	8.00	18.00	10.00	139002	7	TROCT	9.5
CAPCON4	05-439G	108.00	118.00	10.00	139013	7	TROCT	10.1
CAPCON4	05-443G	199.00	209.00	10.00	143024	3	ANORTH	10.2
CAPCON4	05-450C	478.00	488.00	10.00	150054	2	TROCT	19.9
CAPCON4	05-457C	348.00	358.00	10.00	157044	3	ANORTH	20.8
CAPCON4	05-459C	518.00	528.00	10.00	159064	3	TROCT	21.6
CAPCON4	05-463G	148.00	158.00	10.00	163016	5	ANORTH	17.2
								126.8
CAPCON6	05-412M	663.50	667.00	3.50	112203	1	TROCT	3.3
CAPCON6	05-420C	1163.00	1168.00	5.00	120209	1	TROCT	11.2
CAPCON6	05-427C	538.00	548.00	10.00	127061	3	TROCT	10.4
CAPCON6	05-433M	418.50	421.50	3.00	133127	2	TROCT	2.7
CAPCON6	05-438C	532.70	537.30	4.60	138127	1	TROCT	2.2
CAPCON6	05-447G	289.00	299.00	10.00	147037	3	ANORTH	15.5
CAPCON6	05-452C	98.00	108.00	10.00	152011	4	TROCT	21.2
CAPCON6	05-455C	638.00	648.00	10.00	155072	3	ANORTH	22.3
CAPCON6	05-457C	658.00	668.00	10.00	157085	2	TROCT	21.8
CAPCON6	05-466C	128.00	138.00	10.00	166016	2	TROCT	19.6
								130.3

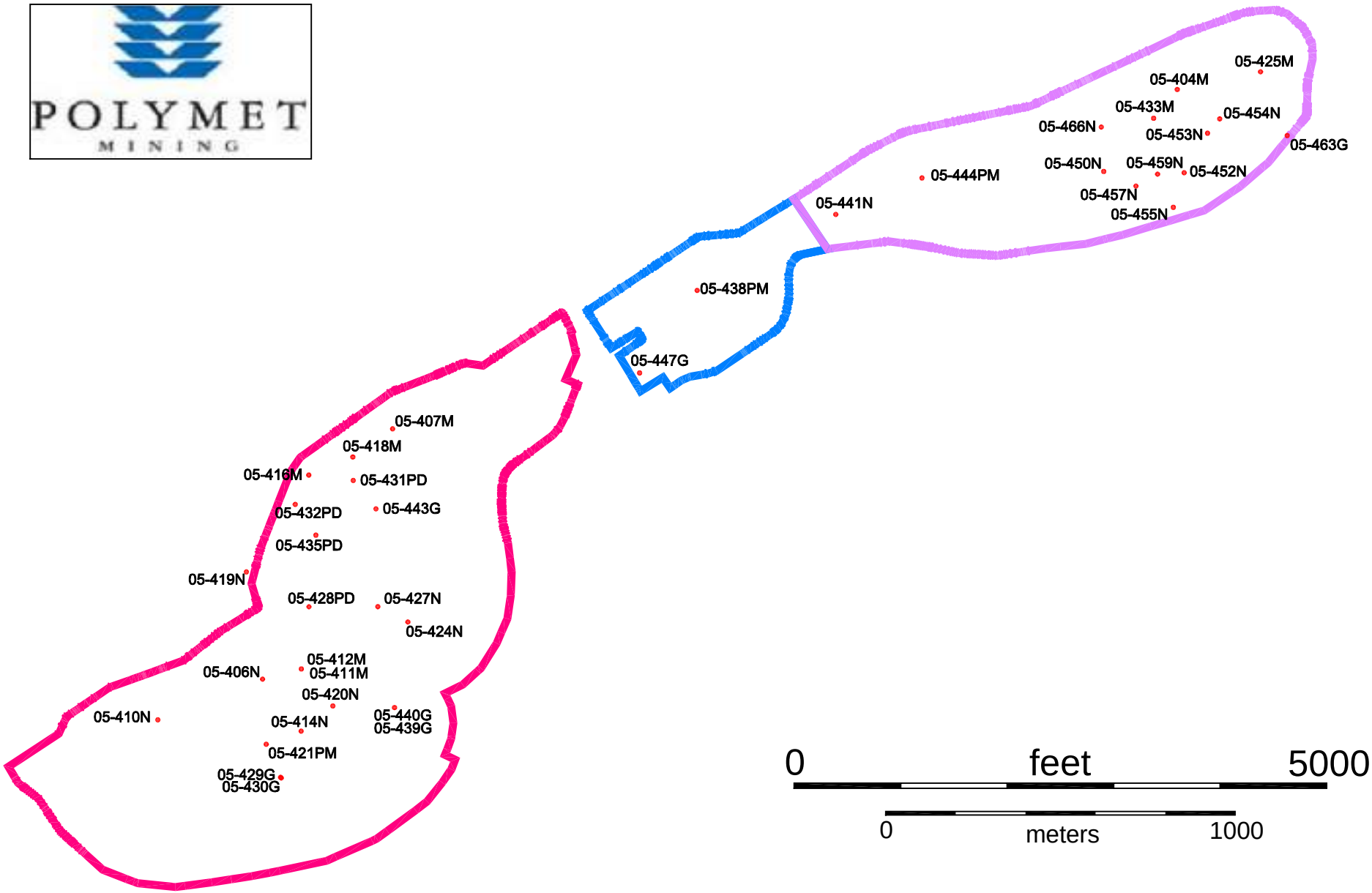
SAMPLE NAME	HOLE_ID	FROM	TO	LENGTH	SAMPLE NUMBER	UNIT	ROCKTYPE	ACTUAL SAMPLE WEIGHT-POUNDS
CAPCON8	05-407M	143.00	146.20	3.20	107043	1	TROCT	1.5
CAPCON8	05-414C	338.00	348.00	10.00	114057	5	ANORTH	11.8
CAPCON8	05-418M	231.50	235.00	3.50	118071	1	TROCT	2.9
CAPCON8	05-419C	423.00	428.00	5.00	119088	1	TROCT	4.3
CAPCON8	05-420C	1288.00	1293.00	5.00	120236	1	TROCT	8.3
CAPCON8	05-429G	974.00	979.00	5.00	129144	1	ANORTH	2.7
CAPCON8	05-439G	28.00	38.00	10.00	139004	7	TROCT	10.7
CAPCON8	05-443G	99.00	109.00	10.00	143013	3	ANORTH	8.6
CAPCON8	05-454C	488.00	493.00	5.00	154062	1	TROCT	9.2
CAPCON8	05-457C	168.00	178.00	10.00	157020	4	TROCT	8.4
CAPCON8	05-459C	158.00	168.00	10.00	159020	4	TROCT	20.7
								89.0
CAPCON10	05-404M	340.00	343.50	3.50	104090	1	TROCT	2.2
CAPCON10	05-406C	628.00	633.00	5.00	106114	1	TROCT	10.9
CAPCON10	05-407M	326.50	329.50	3.00	107100	1	TROCT	1.1
CAPCON10	05-414C	333.00	338.00	5.00	114056	5	ANORTH	2.6
CAPCON10	05-414C	348.00	358.00	10.00	114058	5	ANORTH	12.3
CAPCON10	05-414C	1203.00	1208.00	5.00	114226	1	TROCT	6.8
CAPCON10	05-419C	418.00	423.00	5.00	119087	1	TROCT	6.4
CAPCON10	05-420C	1173.00	1178.00	5.00	120211	1	TROCT	9.6
CAPCON10	05-424C	995.00	1000.00	5.00	124144	1	TROCT	5.5
CAPCON10	05-432C	383.50	387.00	3.50	132088	1	TROCT	2.3
CAPCON10	05-439G	538.00	548.00	10.00	139073	6	ANORTH	9.8
CAPCON10	05-441C	333.00	338.00	5.00	141069	1	TROCT	10.0
CAPCON10	05-443G	119.00	129.00	10.00	143015	3	TROCT	8.9
CAPCON10	05-453C	613.00	618.00	5.00	153075	1	TROCT	10.1
CAPCON10	05-463G	78.00	88.00	10.00	163008	5	ANORTH	17.3
								115.9

SAMPLE NAME	HOLE_ID	FROM	TO	LENGTH	SAMPLE NUMBER	UNIT	ROCKTYPE	ACTUAL SAMPLE WEIGHT-POUNDS
CAPCON12	05-407M	24.00	27.50	3.50	107006	1	TROCT	3.6
CAPCON12	05-410C	653.00	658.00	5.00	110125	1	TROCT	5.4
CAPCON12	05-414C	633.00	638.00	5.00	114102	1	UMAFIC	3.0
CAPCON12	05-419C	33.00	38.00	5.00	119004	1	ANORTH	10.4
CAPCON12	05-419C	88.00	93.00	5.00	119016	1	TROCT	9.2
CAPCON12	05-419C	288.00	293.00	5.00	119059	1	TROCT	5.4
CAPCON12	05-420C	643.00	648.00	5.00	120094	3	TROCT	4.6
CAPCON12	05-428C	536.50	540.50	4.00	128122	1	TROCT	1.8
CAPCON12	05-431C	104.60	109.40	4.80	131022	1	TROCT	2.3
CAPCON12	05-441C	216.00	223.00	7.00	141044	1	TROCT	6.5
CAPCON12	05-444C	368.00	373.00	5.00	144090	1	ANORTH	1.9
CAPCON12	05-449C	438.00	448.00	10.00	149049	4	TROCT	9.2
CAPCON12	05-454C	428.00	438.00	10.00	154053	2	TROCT	23.4
CAPCON12	05-457C	298.00	308.00	10.00	157039	3	TROCT	21.5
								108.3

POLYMET / SRK CAPPING CONCENTRATION SAMPLES-CALCULATED VALUES VS. ASSAYED VALUES																						
		TARGET COPPER	TARGET NICKEL	TARGET SULFUR	S% ME-ICP61	S% LECO	CU%	NI%	CO_PPM	ZN_PPM	BA_PPM	CR_PPM	V_PPM	AL%	CA%	FE%	MG%	MN_PPM	K%	NA%	P_PPM
CAPCON 2	WEIGHTED AVERAGE	0.012	0.029	0.02	0.02		0.013	0.024	42	64	92	159	71	9.29	6.96	5.54	3.69	731	0.19	1.92	292
CAPCON 2	ASSAY ON COMPOSITE				0.02	0.02	0.013	0.025	45	75	100	206	76	9.32	7.31	5.67	3.47	800	0.22	1.99	300
CAPCON 4	WEIGHTED AVERAGE	0.021	0.026	0.04	0.04		0.021	0.023	50	89	130	152	119	9.22	6.40	7.56	4.19	988	0.36	1.98	538
CAPCON 4	ASSAY ON COMPOSITE				0.04	0.04	0.022	0.024	53	105	150	190	132	10.45	6.79	8.16	4.55	1095	0.41	2.06	580
CAPCON 6	WEIGHTED AVERAGE	0.028	0.026	0.06	0.06		0.028	0.026	50	87	135	148	109	9.65	6.50	7.23	4.25	928	0.37	1.98	592
CAPCON 6	ASSAY ON COMPOSITE				0.06	0.05	0.030	0.028	53	95	150	169	112	9.44	6.81	7.39	4.01	999	0.37	2.06	620
CAPCON 8	WEIGHTED AVERAGE	0.035	0.027	0.08	0.08		0.035	0.025	52	99	136	168	143	9.16	6.25	7.70	4.09	994	0.37	1.97	658
CAPCON 8	ASSAY ON COMPOSITE				0.09	0.07	0.037	0.026	54	103	160	195	147	10.05	6.92	8.18	4.18	1095	0.41	2.13	710
CAPCON 10	WEIGHTED AVERAGE	0.04	0.027	0.1	0.10		0.041	0.026	53	102	151	185	140	9.16	6.18	8.26	4.26	1055	0.43	1.93	712
CAPCON 10	ASSAY ON COMPOSITE				0.12	0.1	0.046	0.028	59	110	150	216	145	9.44	6.51	8.48	4.12	1125	0.44	1.98	720
CAPCON 12	WEIGHTED AVERAGE	0.05	0.028	0.12	0.12		0.049	0.031	56	106	146	139	147	9.00	6.01	8.81	4.56	1108	0.40	1.89	702
CAPCON 12	ASSAY ON COMPOSITE				0.13	0.11	0.057	0.033	59	112	150	164	152	9.18	6.22	9.04	4.48	1180	0.41	1.91	670
METHOD CODE-CHEMEX					ME-ICP61	S-IR08	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61	ME-ICP61
DETECTION LIMIT					0.01	0.01	1	1	1	2	10	1	1	0.01	0.01	0.01	0.01	5	0.01	0.01	10
ME-ICP61 IS FOUR ACID DIGESTION FOLLOWED BY ICP-AES																					
S-IR08 IS TOTAL SULFUR BY LECO FURNACE																					
"WEIGHTED AVERAGE" IS VALUE CALCULATED FOR COMPOSITE FROM INDIVIDUAL WEIGHED SAMPLES																					

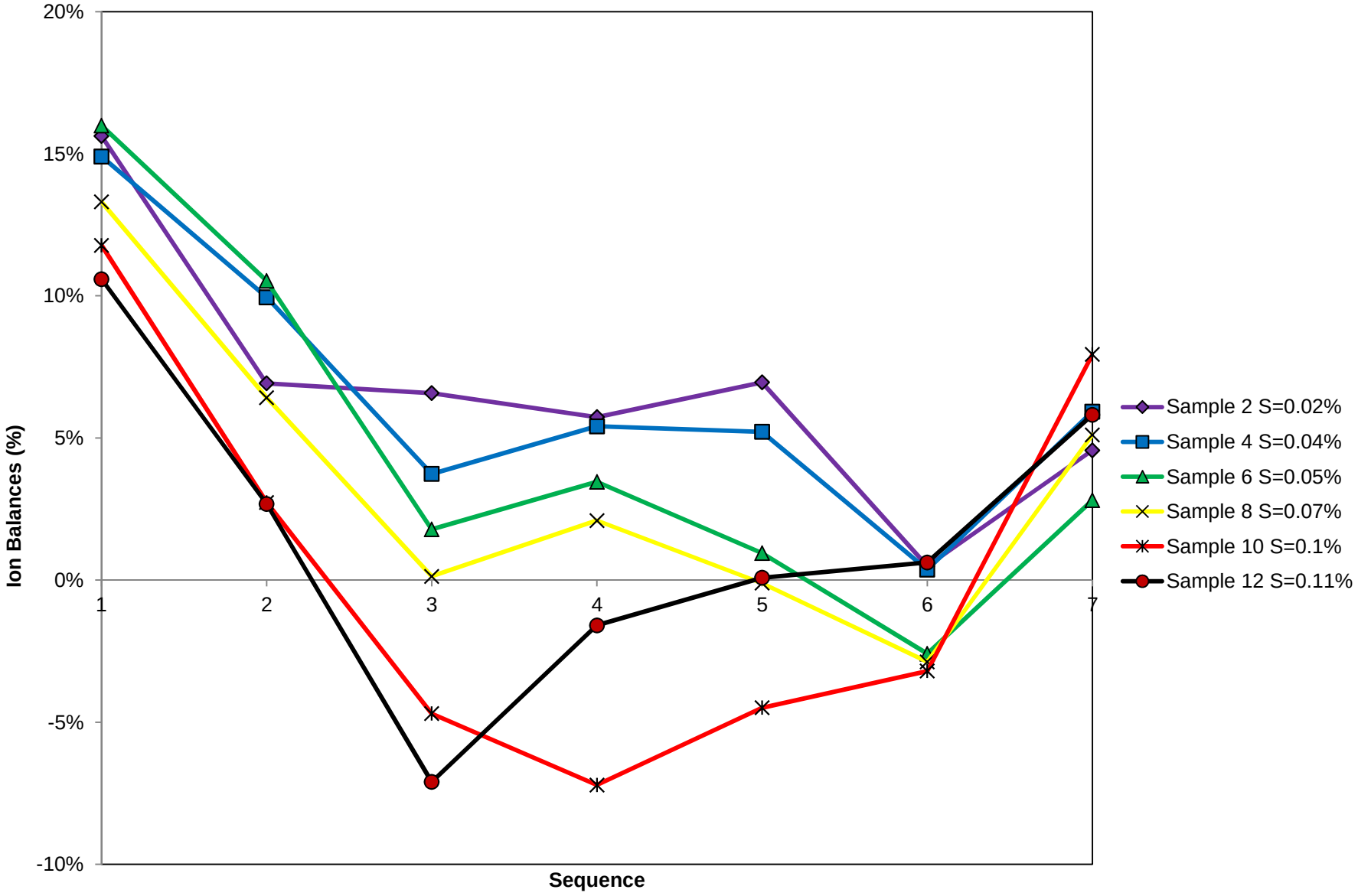
SAMPLES REFERENCED IN STEVE DAY MEMO THAT ARE LARGELY BELOW ASSAY DETECTION LIMITS						
	ELEMENT	DETECTION LIMIT	TOTAL NUMBER SAMPLES IN COMPOSITE	TOTAL BELOW DETECTION LIMT	UNWEIGHTED AVERAGE OF VALUES ABOVE DETECTION LIMIT	SINGLE ASSAY VALUE FROM COMPOSITE USED IN TEST
CAPCON 2	Arsenic	5 ppm	9	6	9	<5
	Antimony	5 ppm	9	9	NA	<5
	Silver	0.5 ppm	9	8	0.6	<0.5
	Beryllium	0.5 ppm	9	9	NA	<0.5
	Cadmium	0.5 ppm	9	9	NA	<0.5
	Lead	2 ppm	9	2	7	<2
	Molybdenum	1 ppm	9	8	3	<1
	Thallium	10 ppm	9	NS	NS	10
	Uranium	10 ppm	9	NS	NS	<10
CAPCON 4	Arsenic	5 ppm	15	10	6	<5
	Antimony	5 ppm	15	14	5	6
	Silver	0.5 ppm	15	14	0.5	<0.5
	Beryllium	0.5 ppm	15	6	0.6	0.5
	Cadmium	0.5 ppm	15	14	0.6	<0.5
	Lead	2 ppm	15	4	6	7
	Molybdenum	1 ppm	15	10	1	1
	Thallium	10 ppm	15	NS	NS	10
	Uranium	10 ppm	15	NS	NS	<10
CAPCON 6	Arsenic	5 ppm	10	5	6	<5
	Antimony	5 ppm	10	9	8	<5
	Silver	0.5 ppm	10	10	NA	<0.5
	Beryllium	0.5 ppm	10	4	0.7	0.5
	Cadmium	0.5 ppm	10	10	NA	<0.5
	Lead	2 ppm	10	2	5	2
	Molybdenum	1 ppm	10	7	1	<1
	Thallium	10 ppm	10	NS	NS	10
	Uranium	10 ppm	10	NS	NS	<10
CAPCON 8	Arsenic	5 ppm	11	7	11	10
	Antimony	5 ppm	11	11	NA	<5
	Silver	0.5 ppm	11	11	NA	<0.5
	Beryllium	0.5 ppm	11	3	0.7	0.6
	Cadmium	0.5 ppm	11	10	0.9	<0.5
	Lead	2 ppm	11	1	7	8
	Molybdenum	1 ppm	11	8	1	1
	Thallium	10 ppm	11	NS	NS	10
	Uranium	10 ppm	11	NS	NS	<10

	ELEMENT	DETECTION LIMIT	TOTAL NUMBER SAMPLES IN COMPOSITE	TOTAL BELOW DETECTION LIMT	UNWEIGHTED AVERAGE OF VALUES ABOVE DETECTION LIMIT	SINGLE ASSAY VALUE FROM COMPOSITE USED IN TEST
CAPCON 10	Arsenic	5 ppm	15	11	13	13
	Antimony	5 ppm	15	14	7	<5
	Silver	0.5 ppm	15	14	0.5	0.6
	Beryllium	0.5 ppm	15	6	1	0.6
	Cadmium	0.5 ppm	15	15	NA	<0.5
	Lead	2 ppm	15	2	7	10
	Molybdenum	1 ppm	15	9	1	<1
	Thallium	10 ppm	15	NS	NS	<10
	Uranium	10 ppm	15	NS	NS	<10
CAPCON 12	Arsenic	5 ppm	14	7	9	<5
	Antimony	5 ppm	14	13	5	<5
	Silver	0.5 ppm	14	14	NA	<0.5
	Beryllium	0.5 ppm	14	4	0.7	0.6
	Cadmium	0.5 ppm	14	14	NA	<0.5
	Lead	2 ppm	14	10	5	5
	Molybdenum	1 ppm	14	10	1	<1
	Thallium	10 ppm	14	NS	NS	<10
	Uranium	10 ppm	14	NS	NS	<10
NS = NOT SAMPLED						
NA = NOT AVAILABLE						


POLYMET
MINING
05-425M
05-404M
05-433M
05-454N
05-466N
05-453N
05-463G
05-450N
05-459N
05-452N
05-444PM
05-457N
05-441N
05-455N
05-438PM
05-447G
05-407M
05-418M
05-416M
05-431PD
05-432PD
05-443G
05-435PD
05-419N
05-428PD
05-427N
05-424N
05-412M
05-406N
05-411M
05-420N
05-410N
05-414N
05-440G
05-439G
05-421PM
05-429G
05-430G
0
feet
5000
0
meters
1000
DRILL HOLES FOR "CAPCON" SAMPLES
COLLAR LOCATIONS
RLP-JANUARY 2011

Attachment 2

Graphs of SMWMP Results



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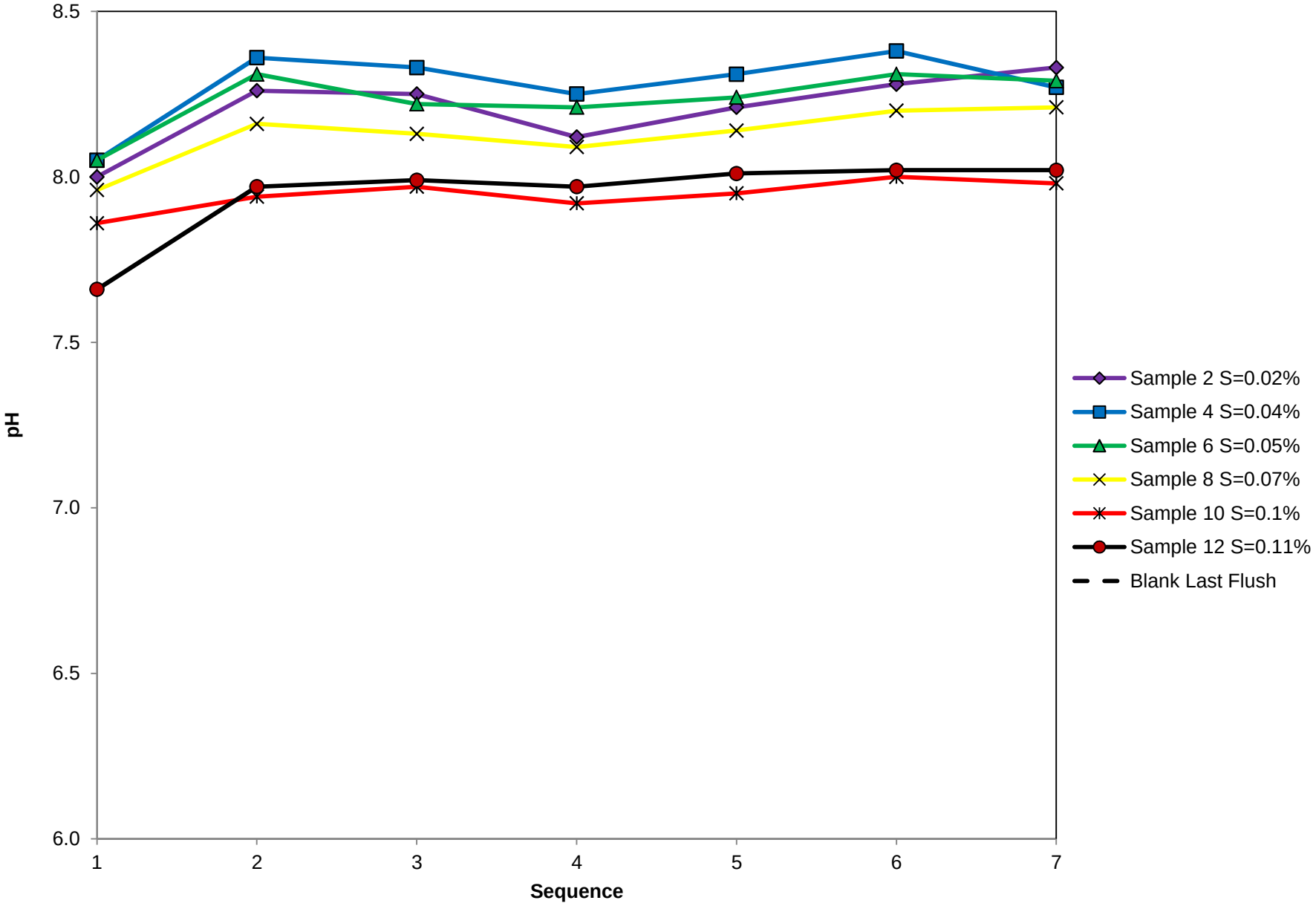

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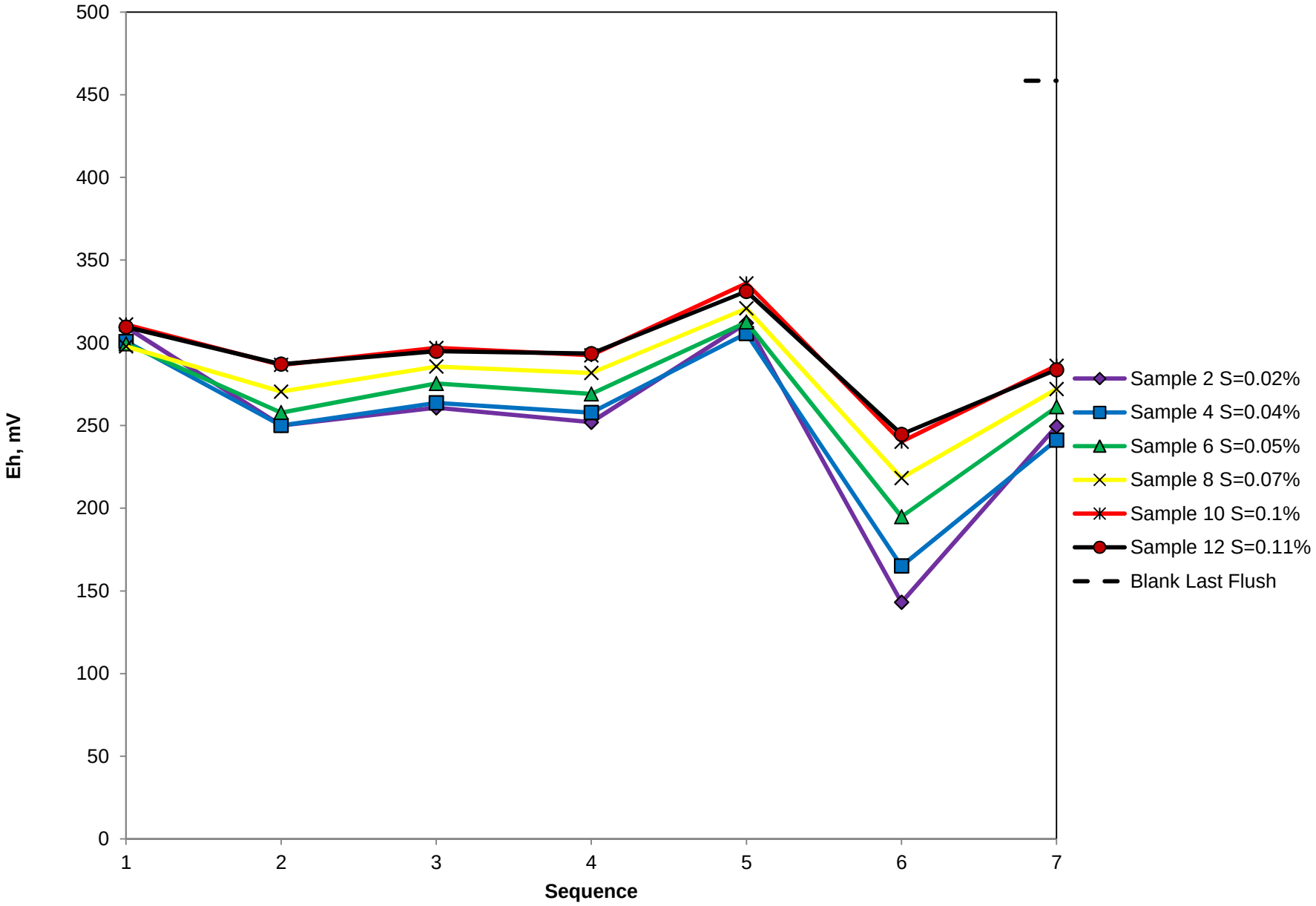

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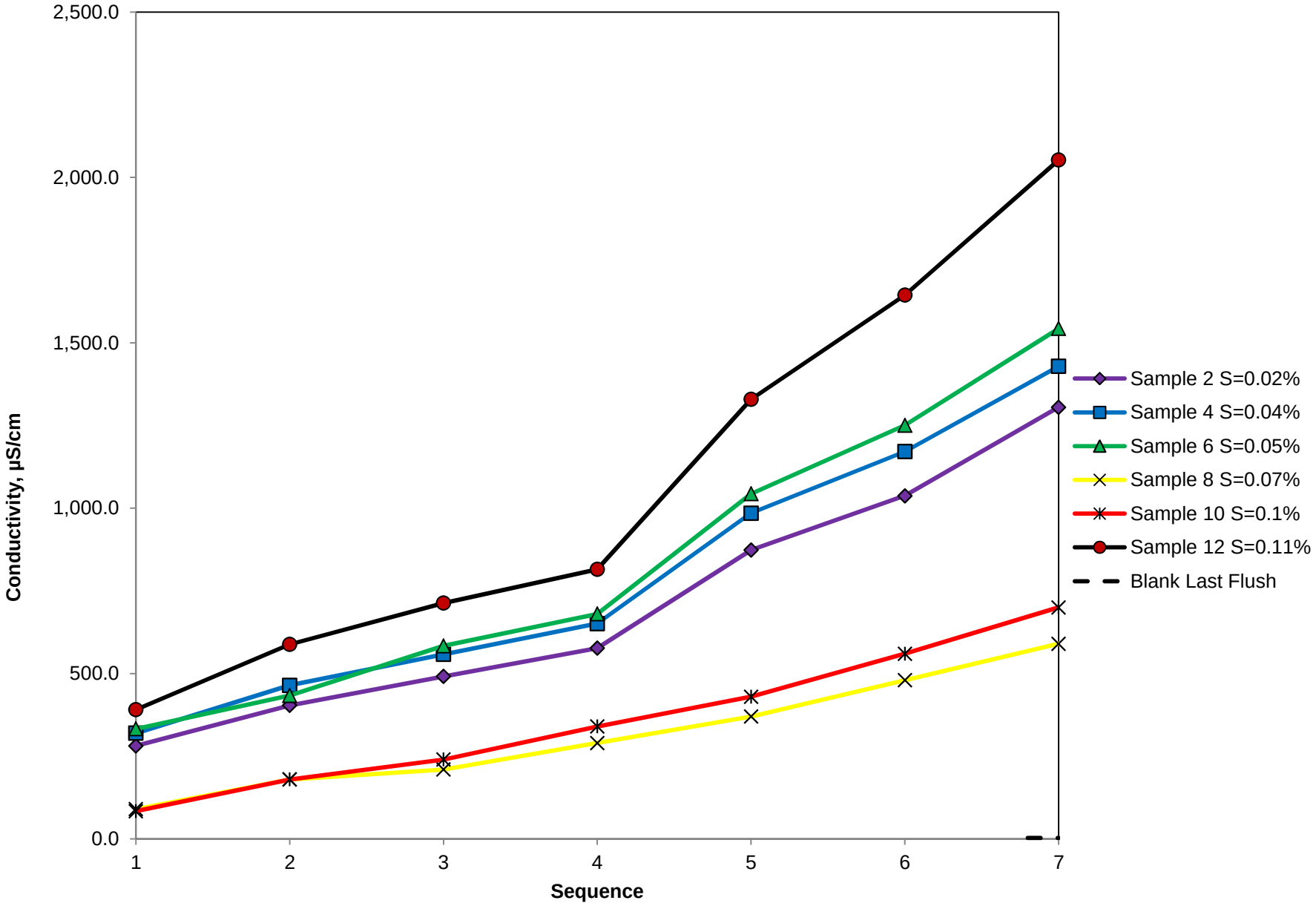

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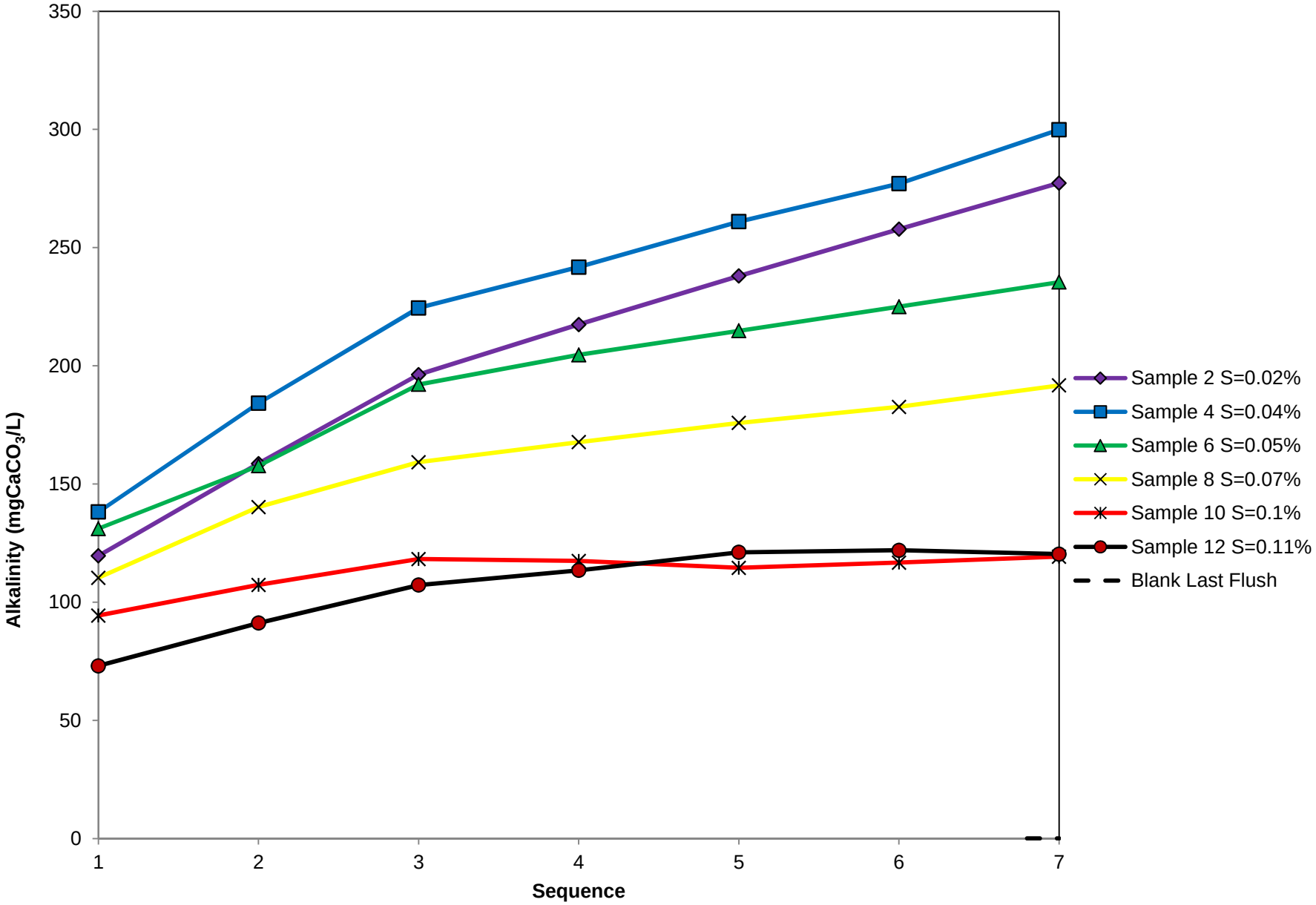

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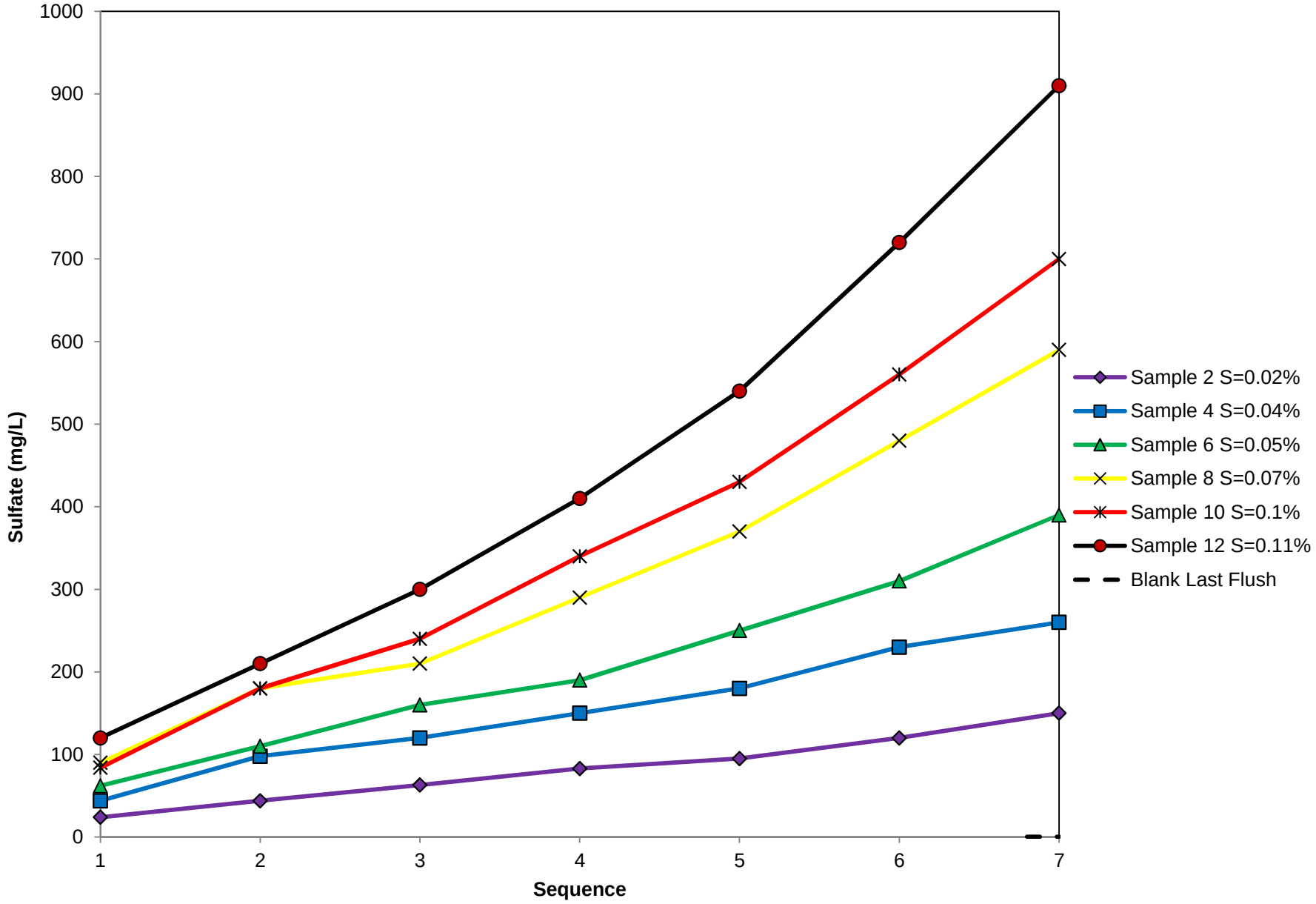

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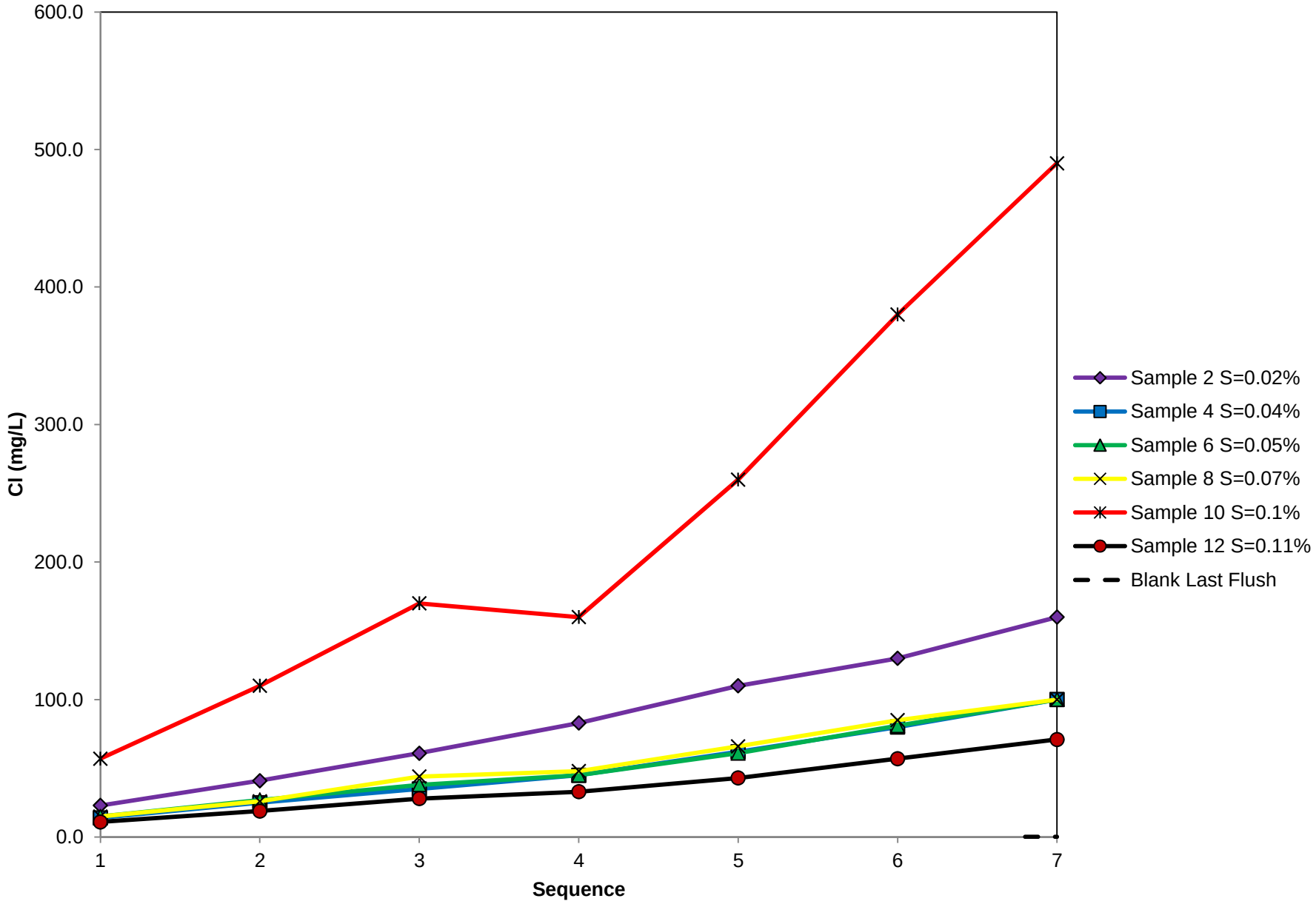

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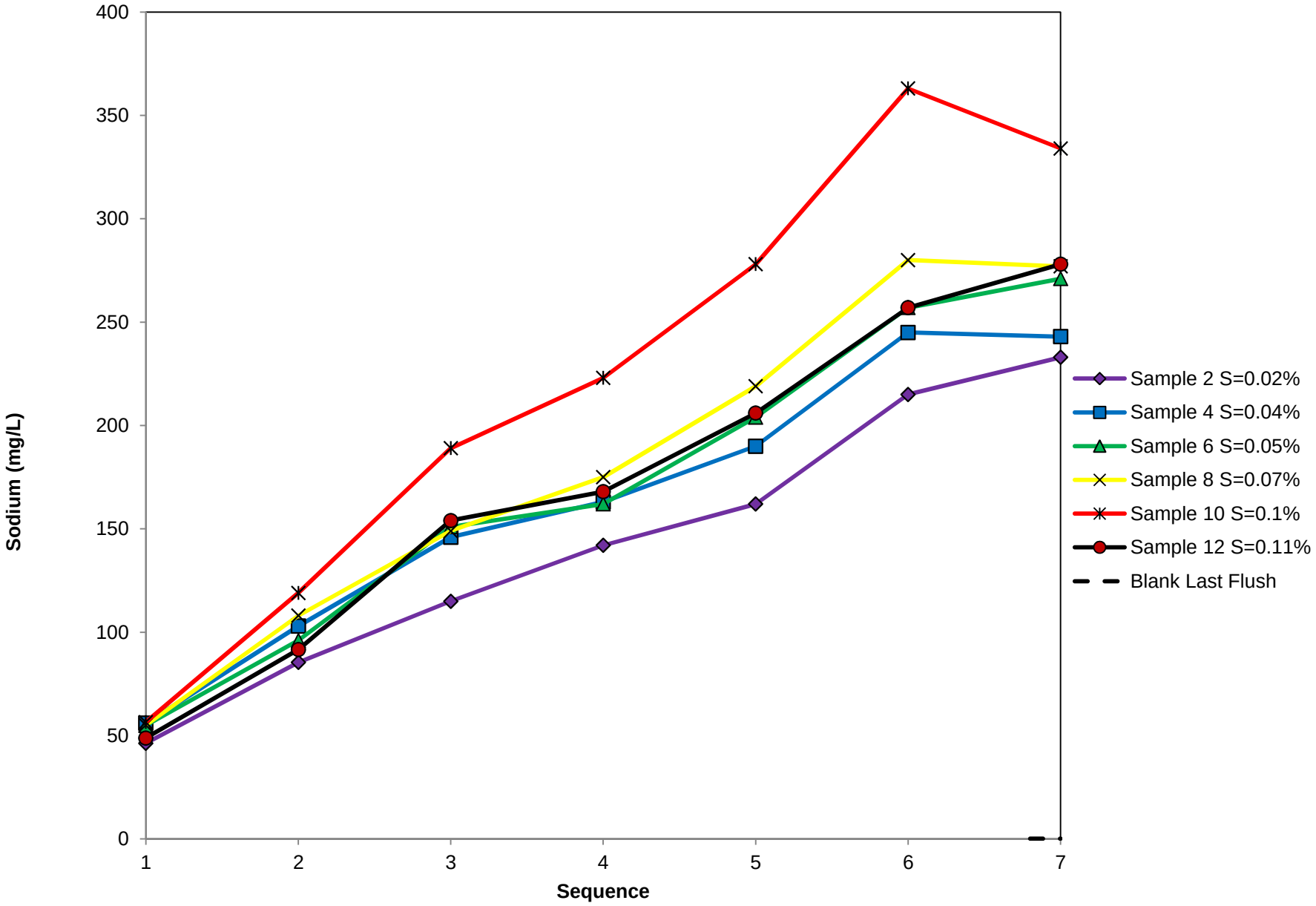

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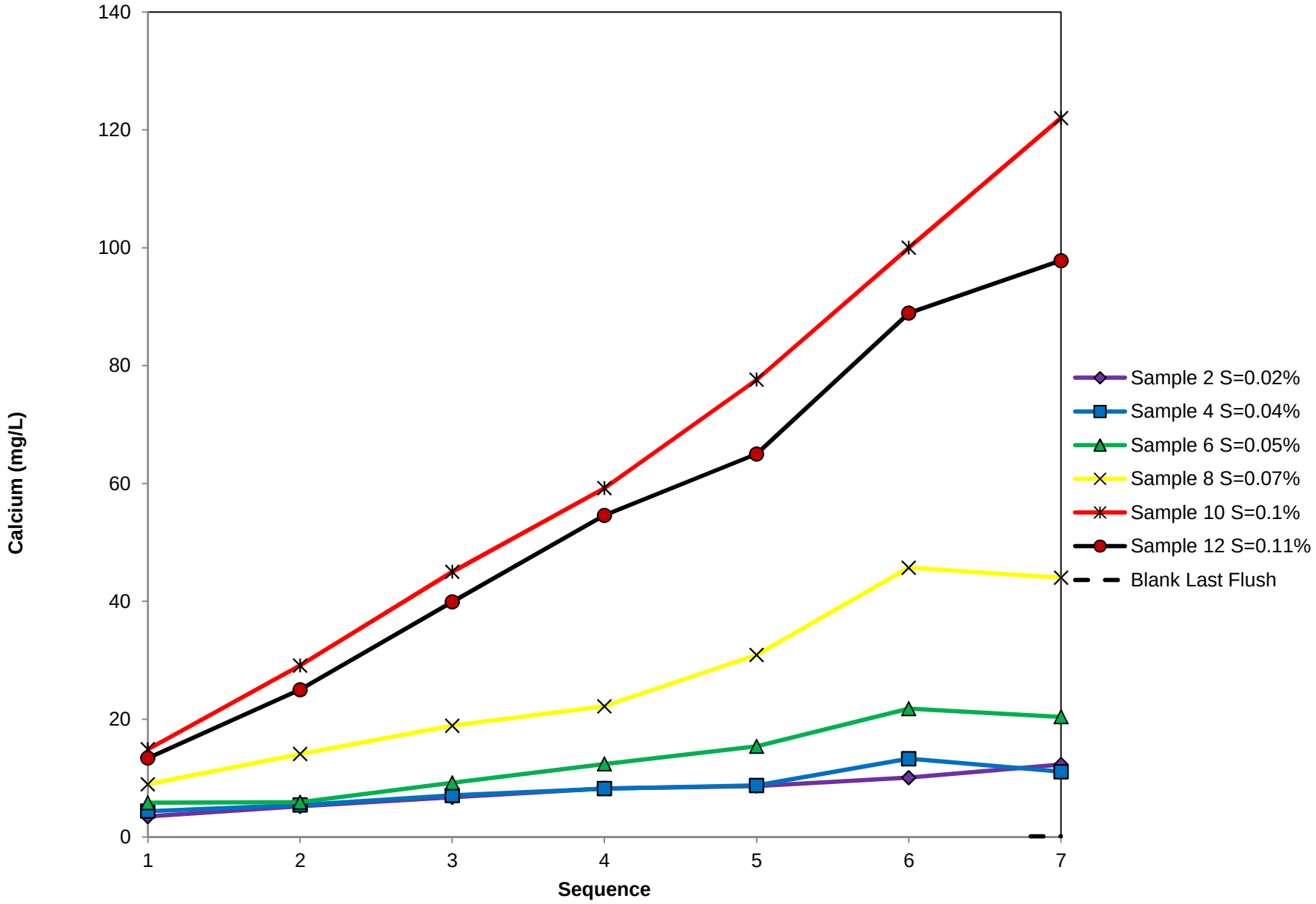

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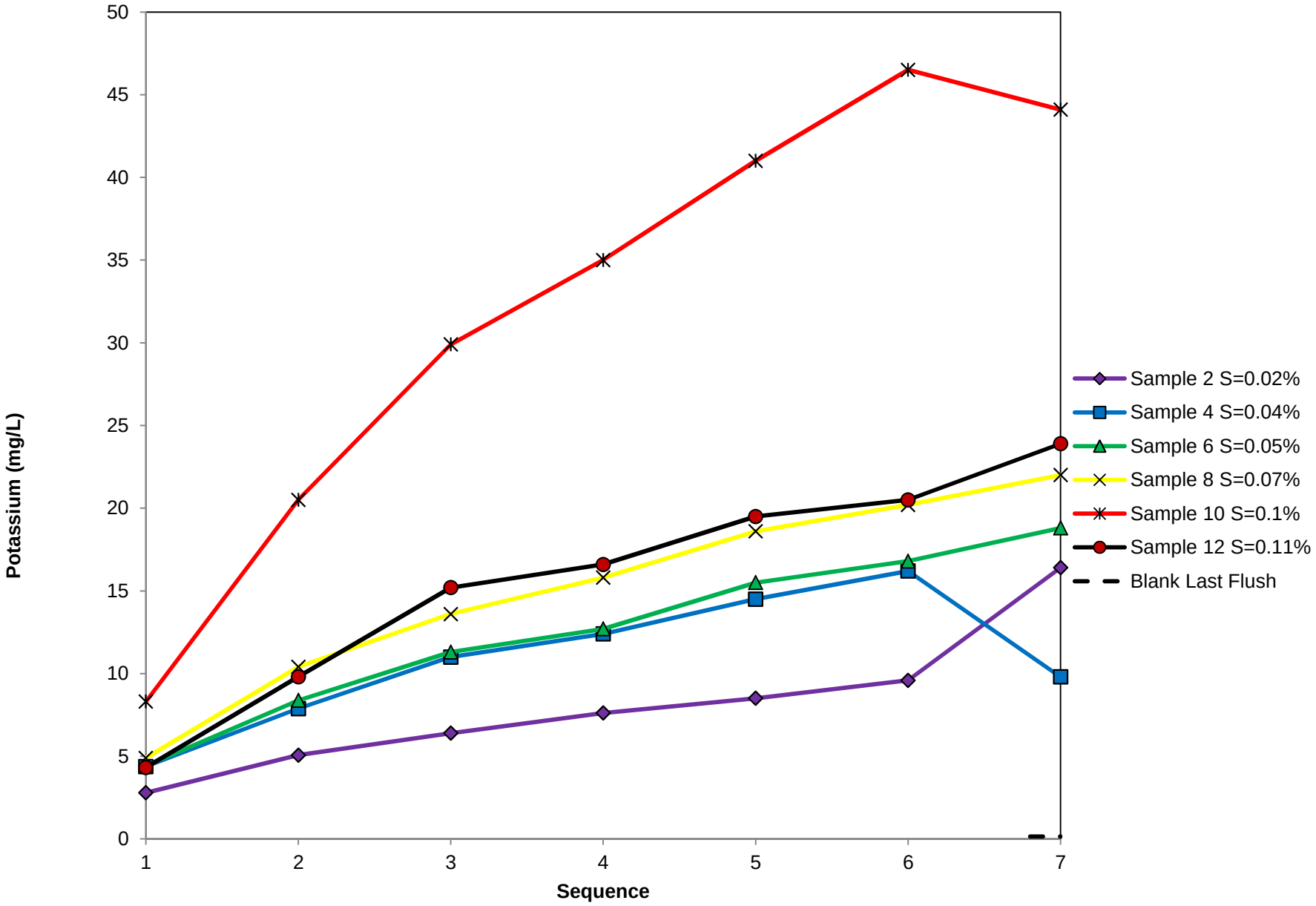

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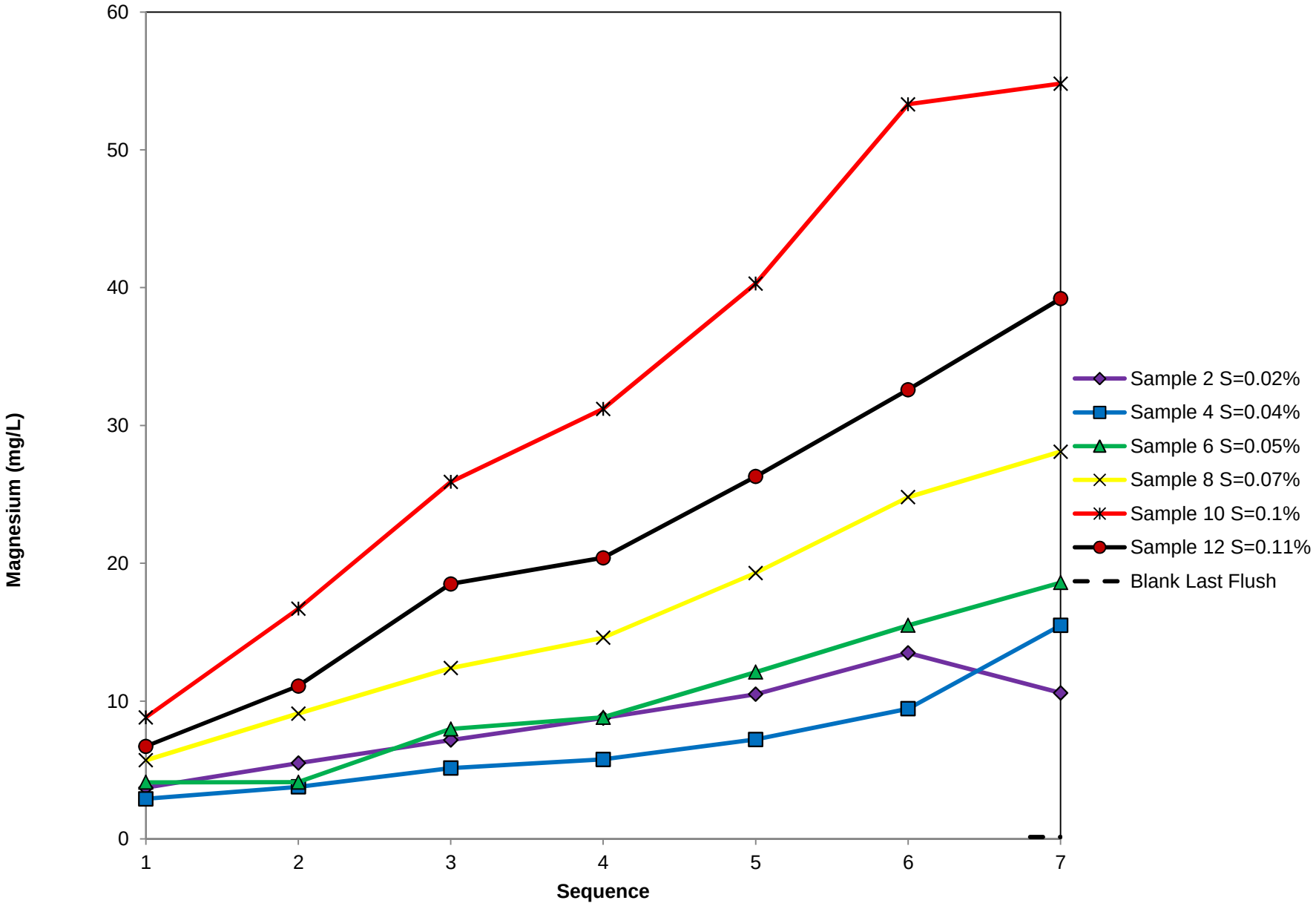

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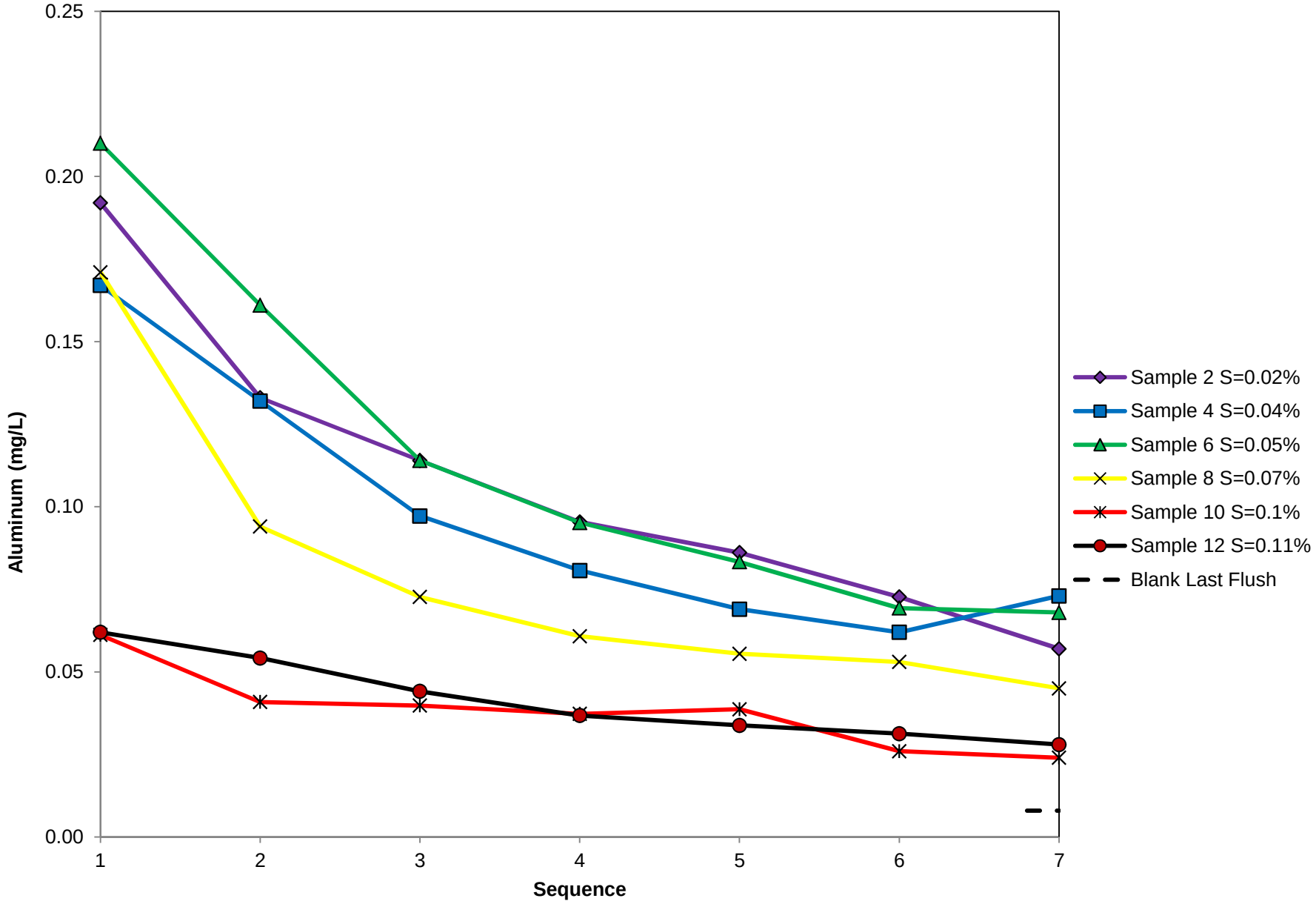

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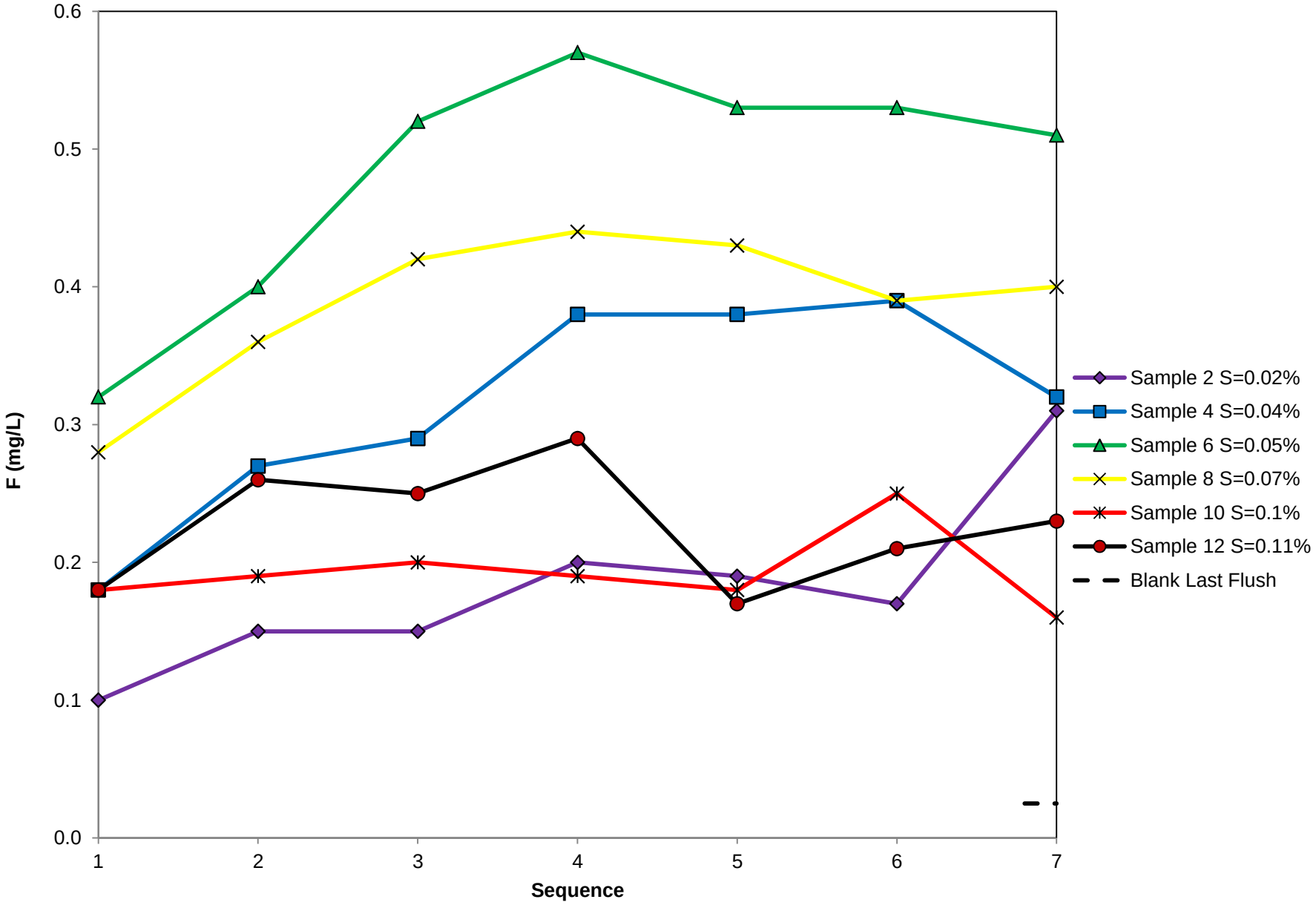

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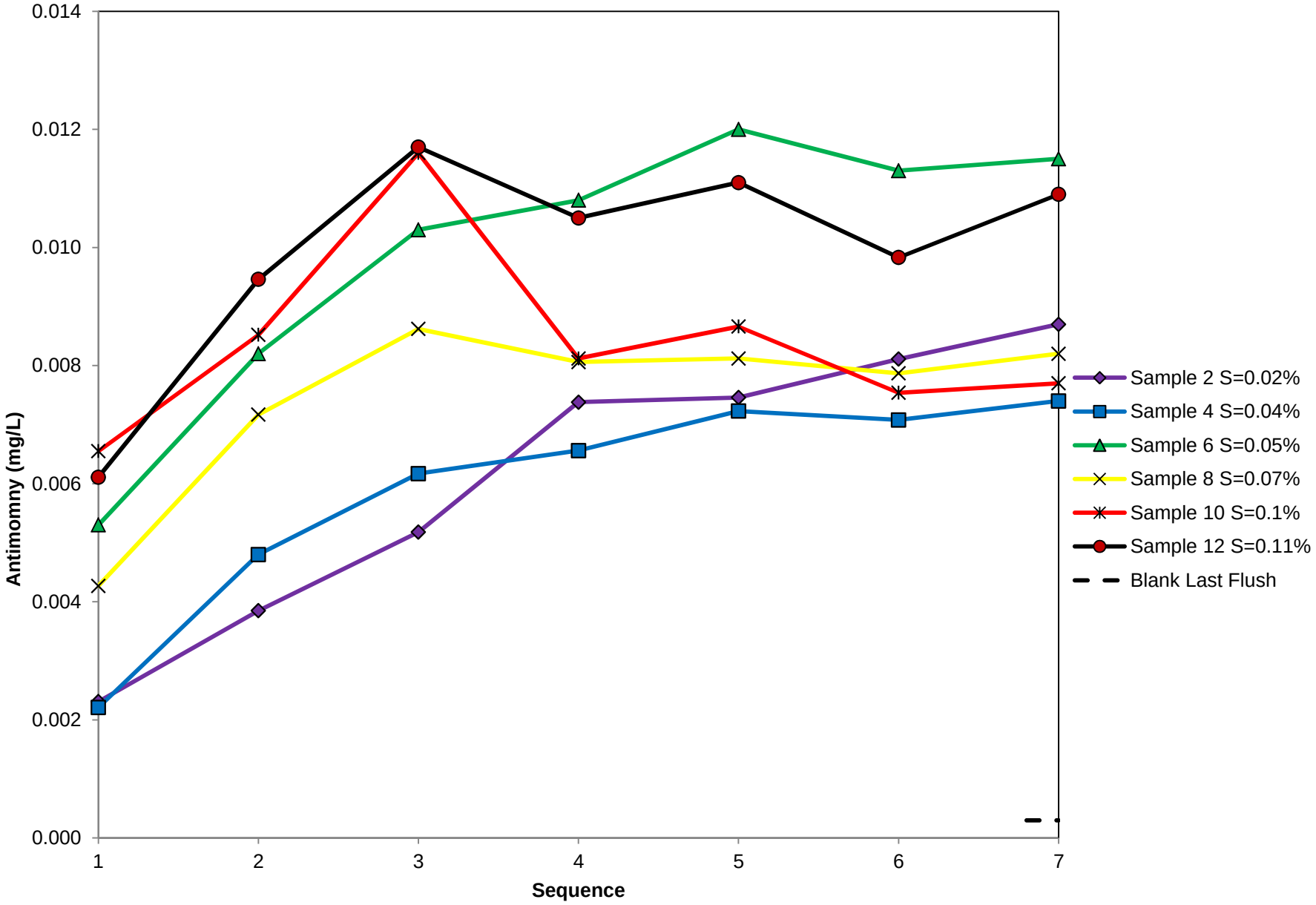

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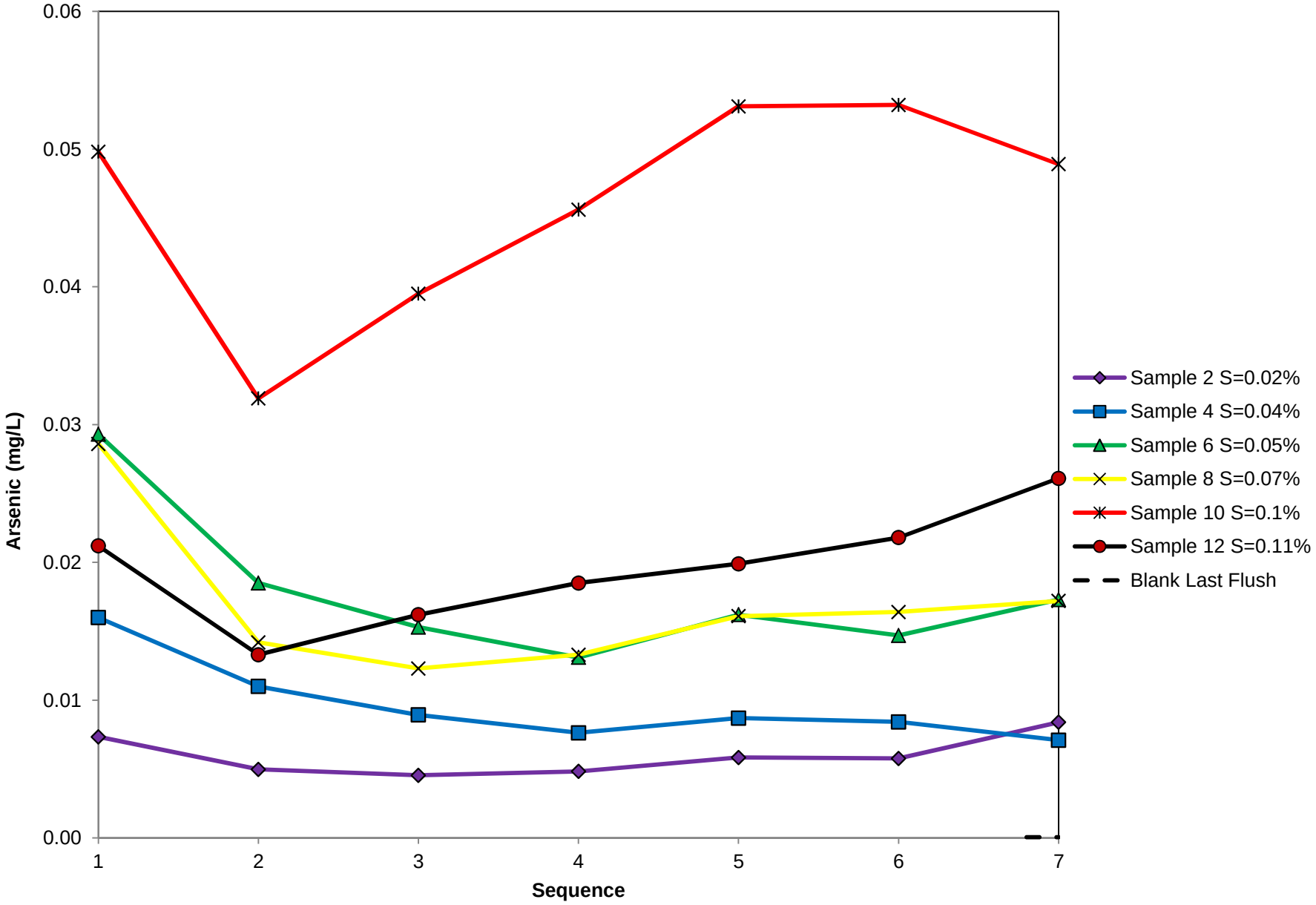

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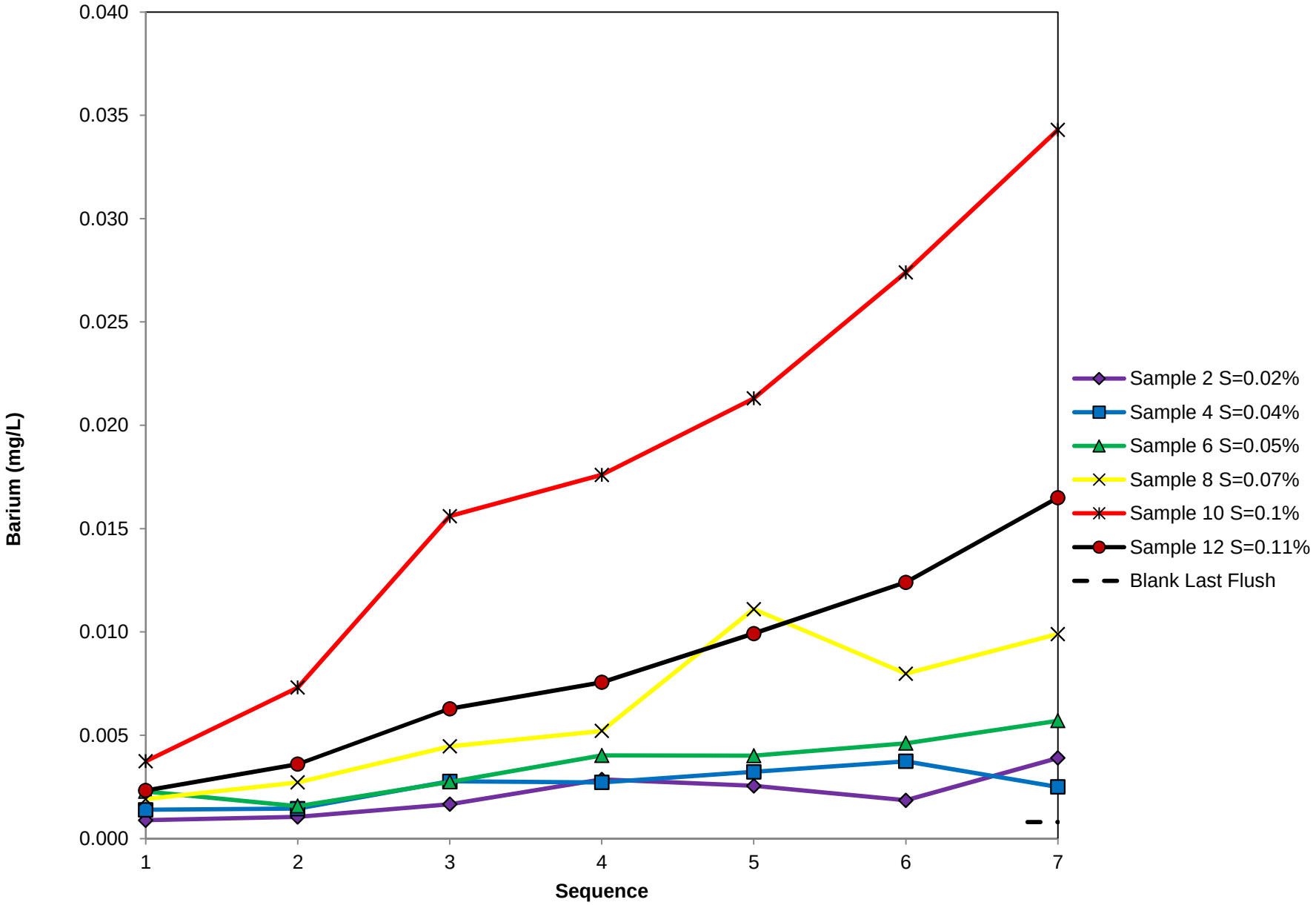

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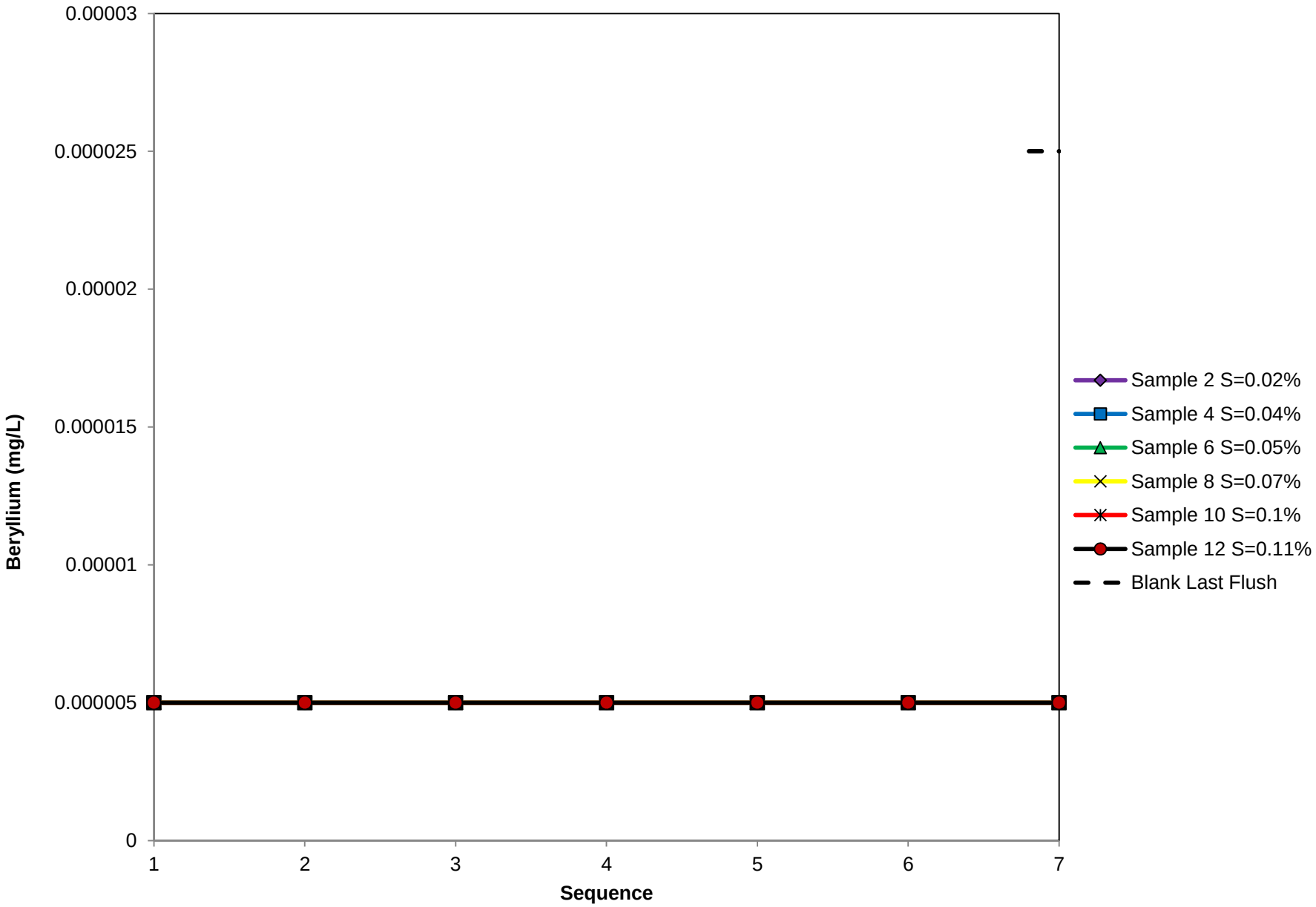

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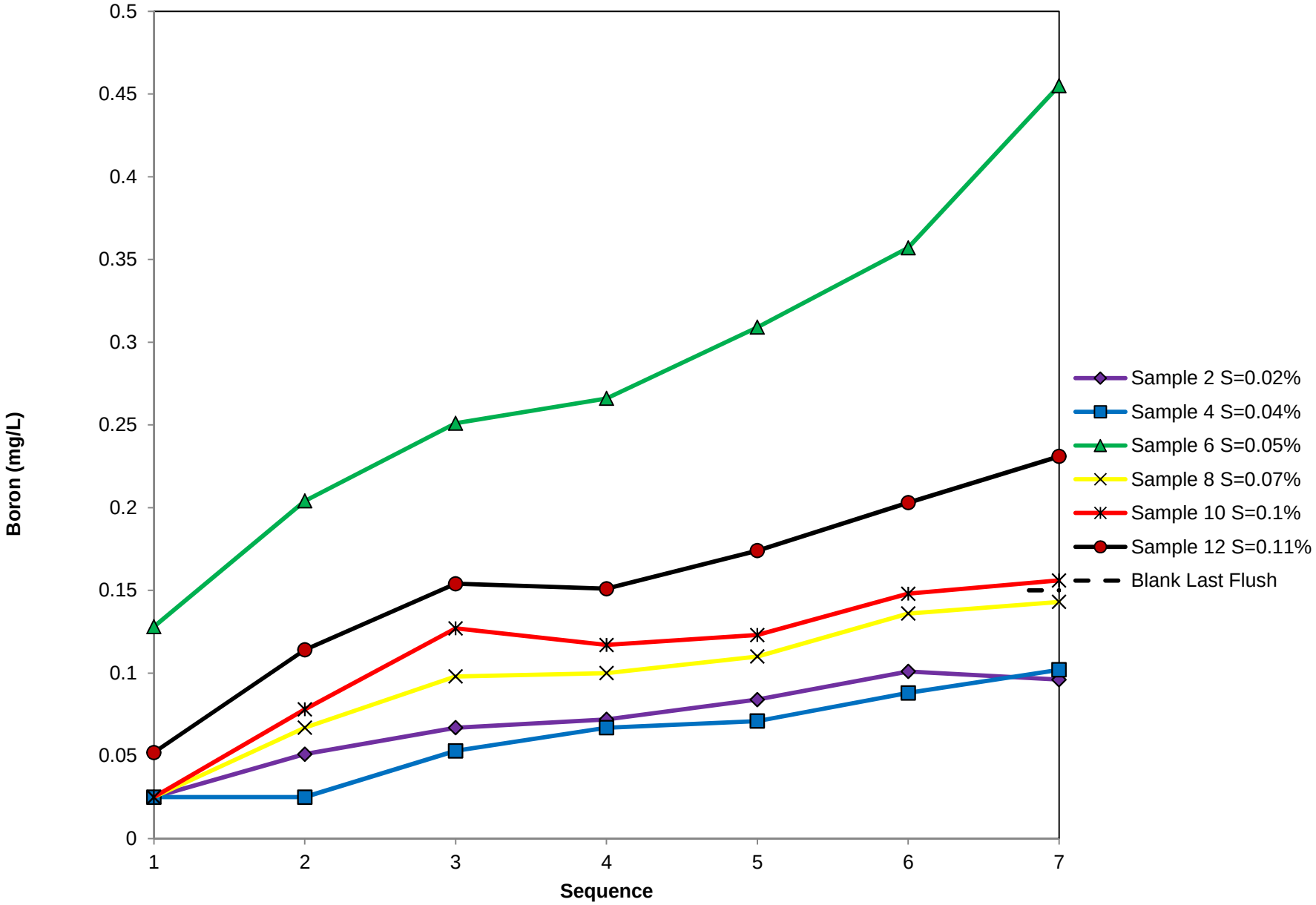

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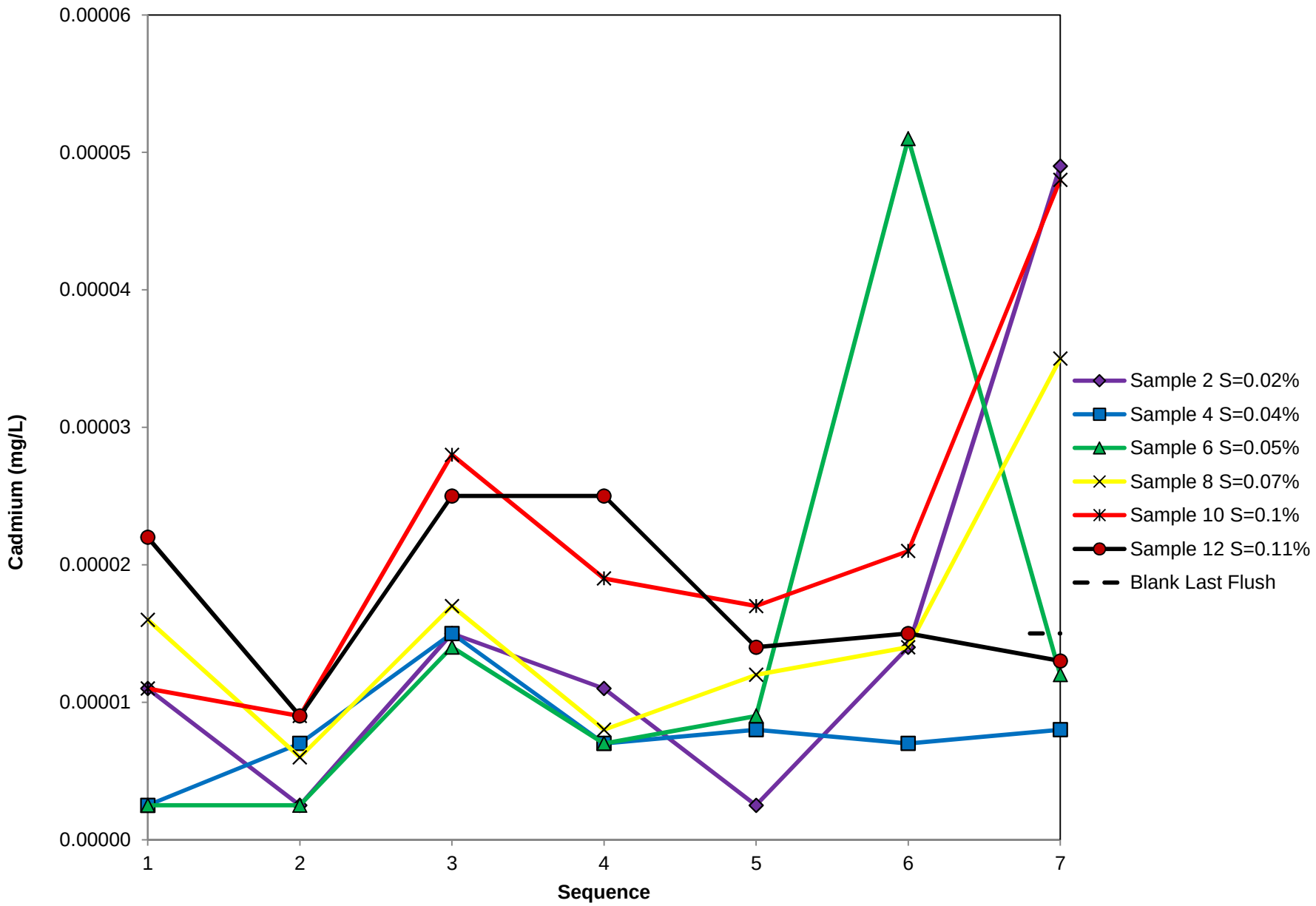

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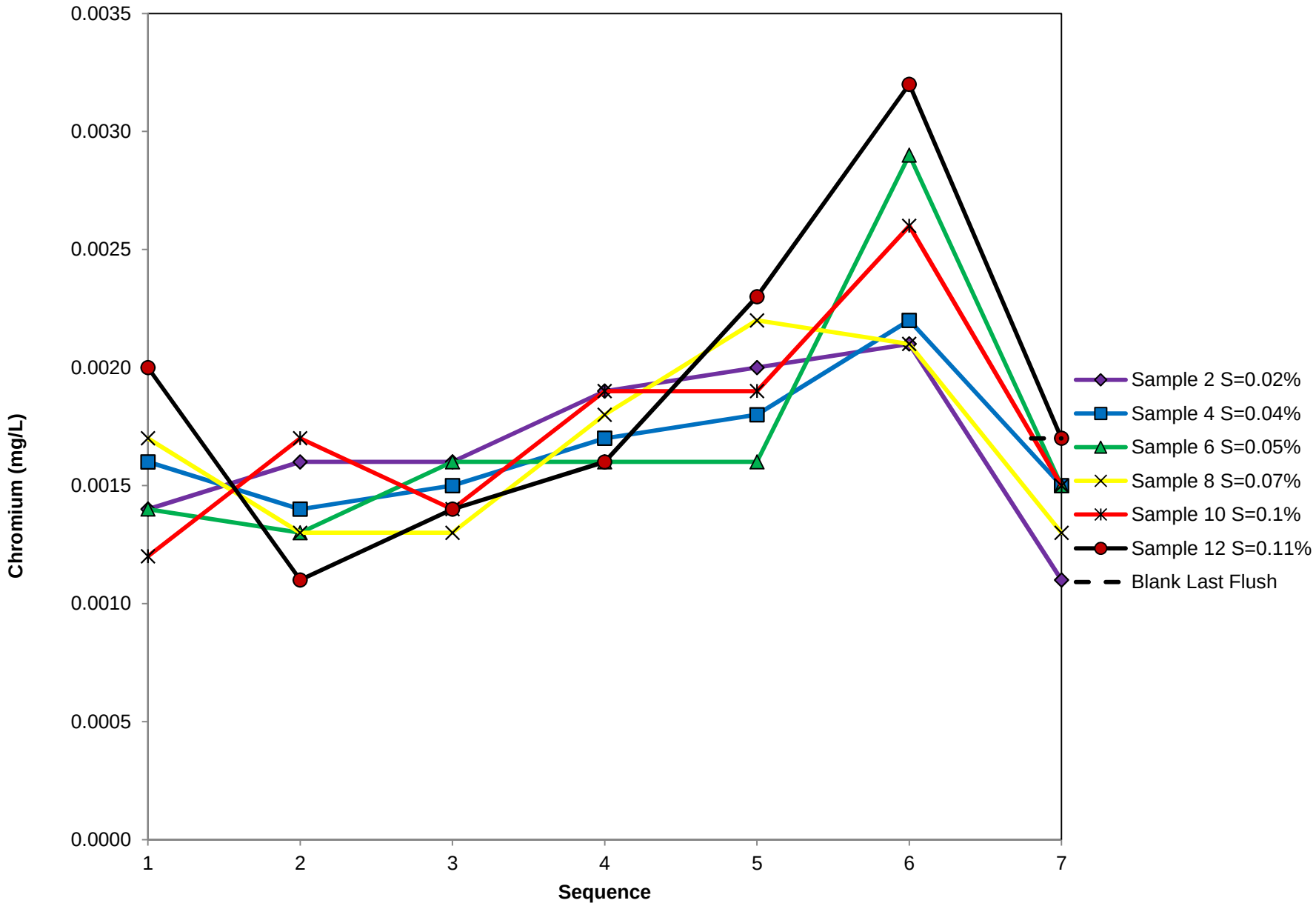

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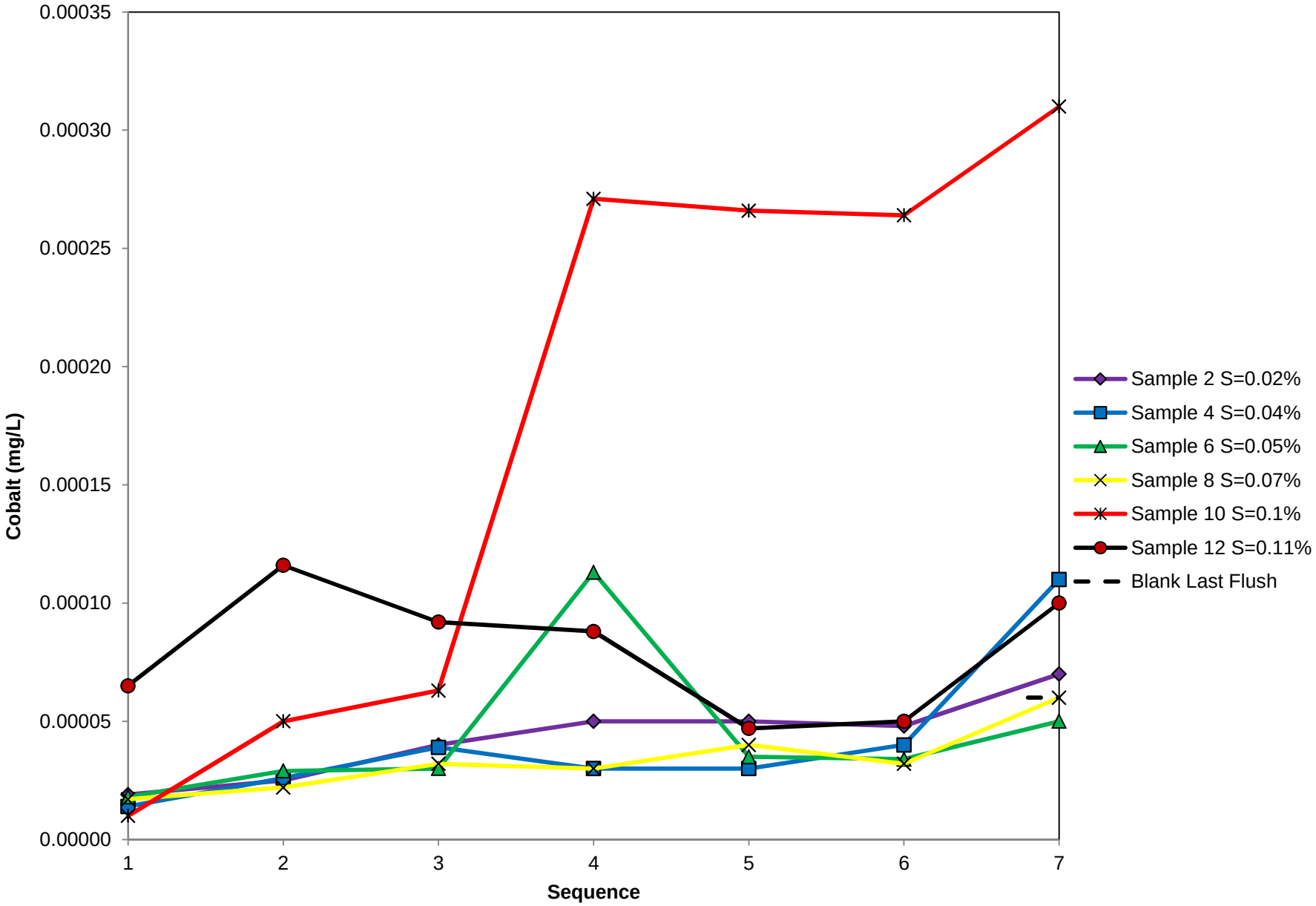

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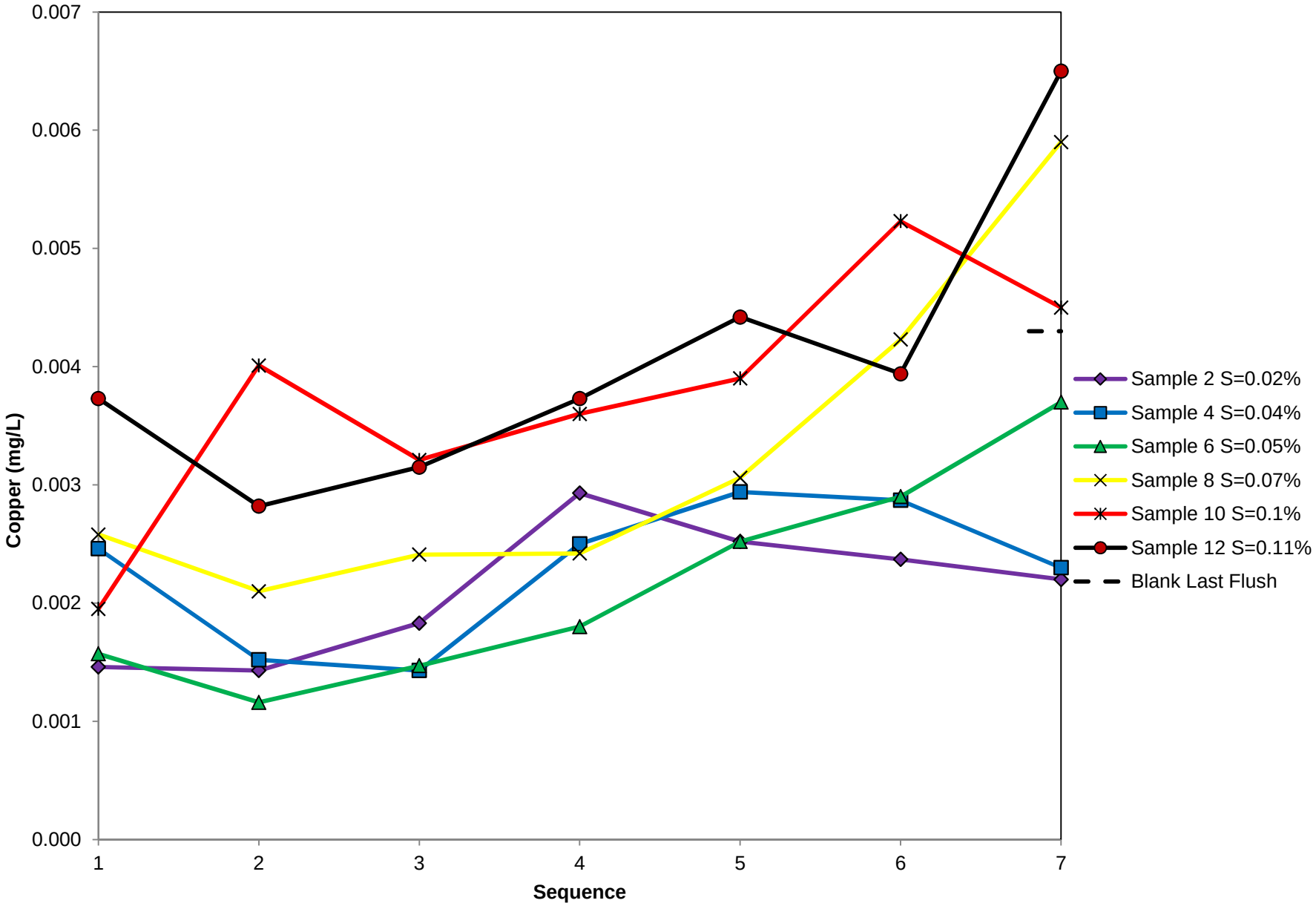

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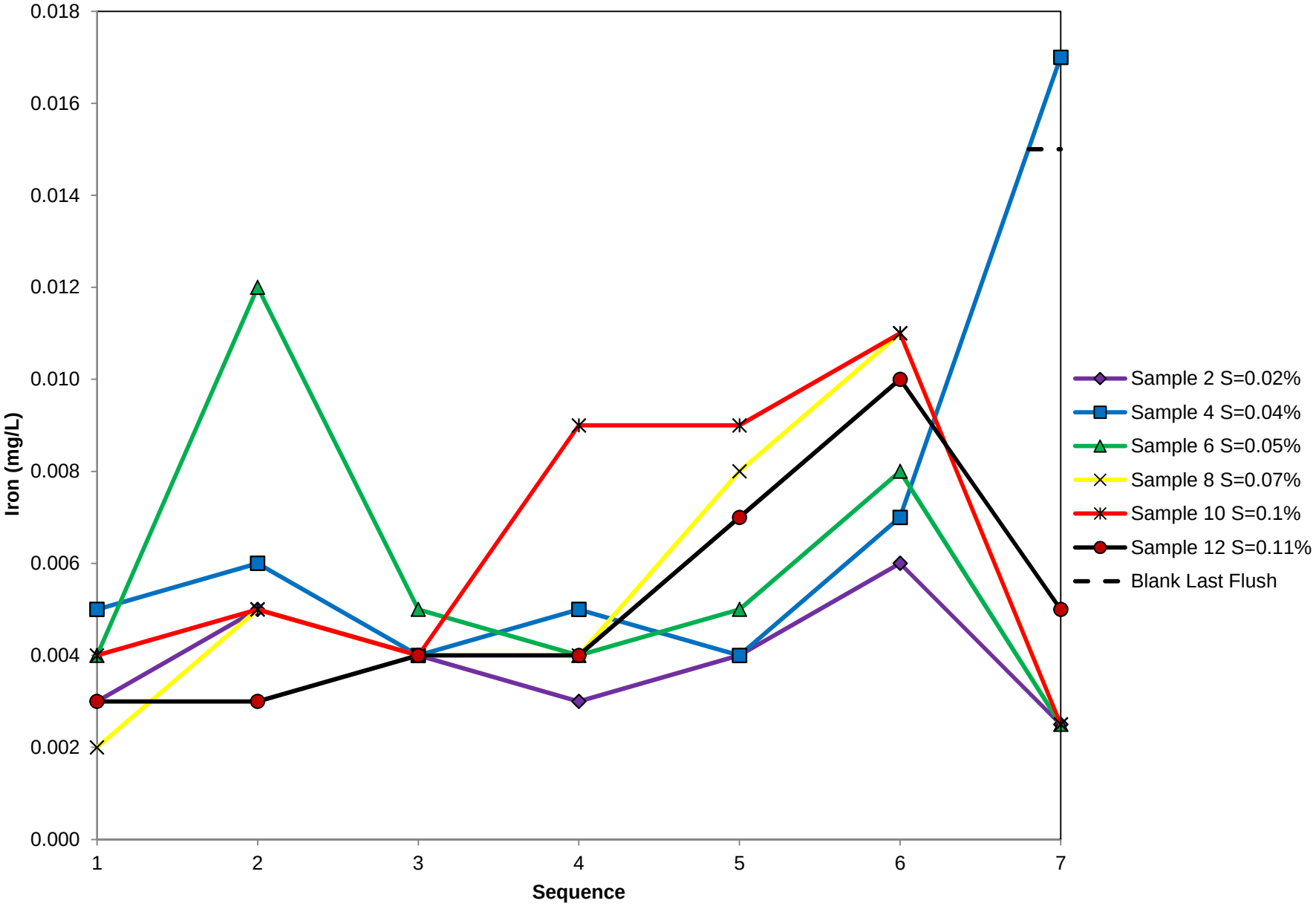

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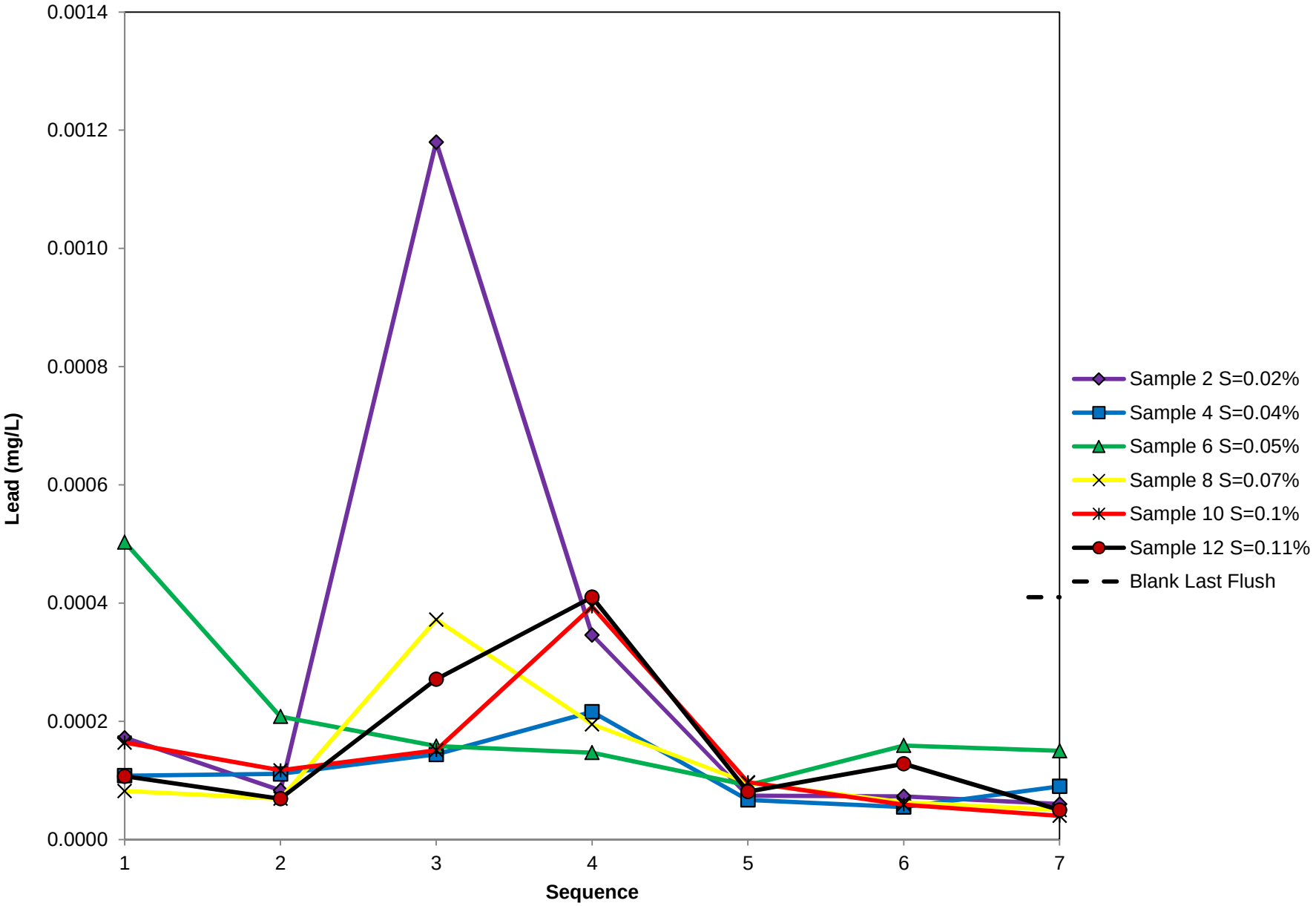

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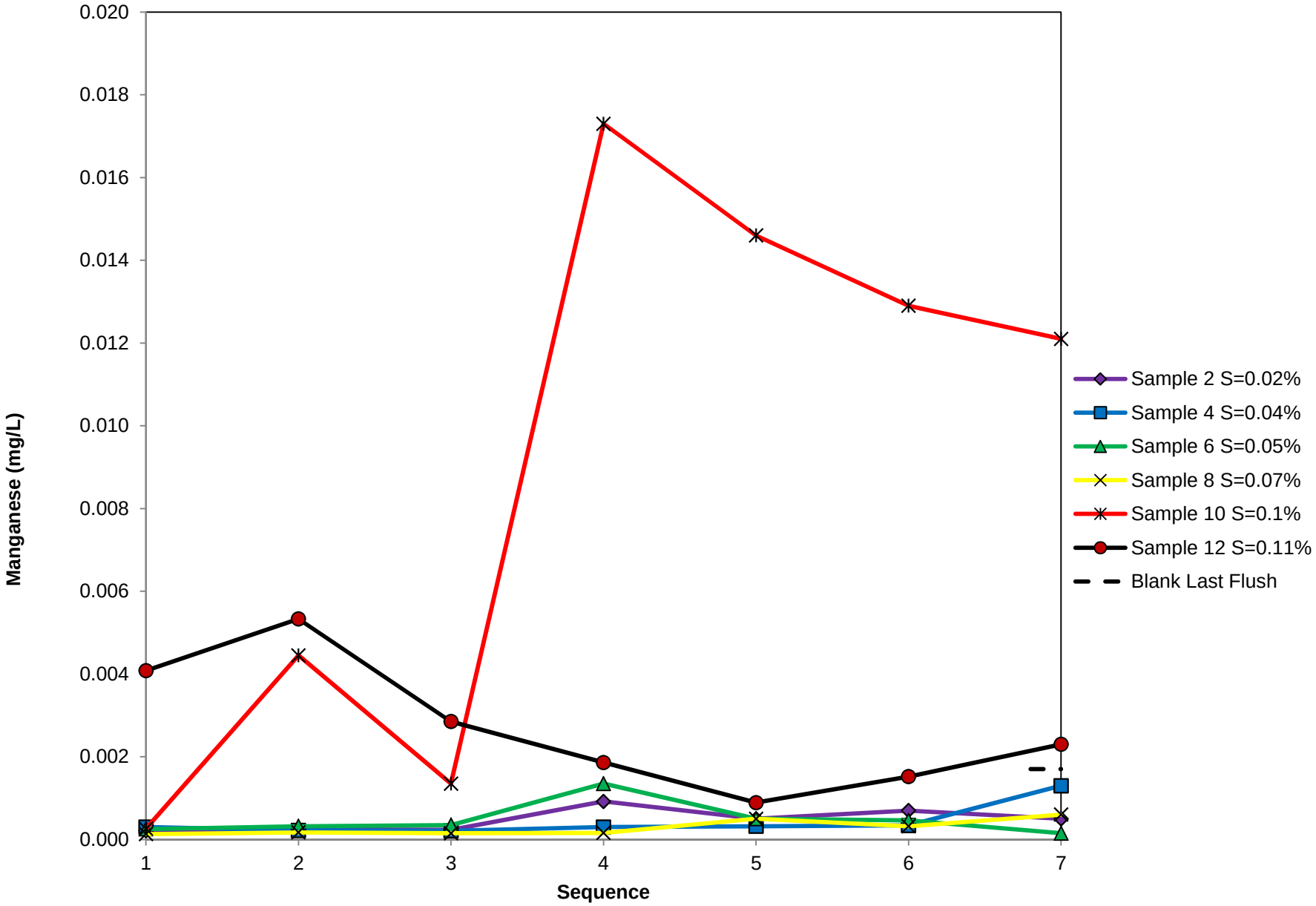

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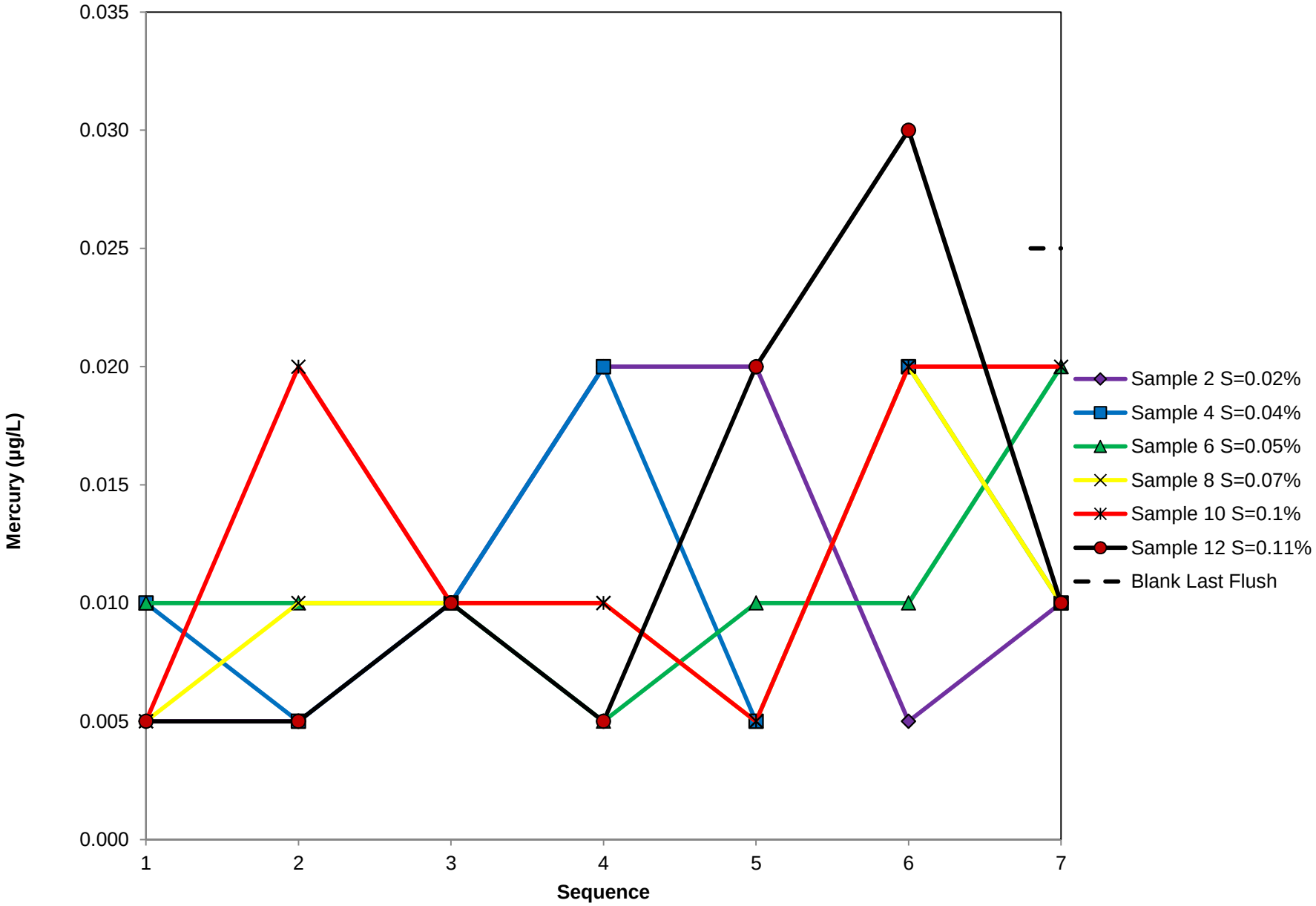

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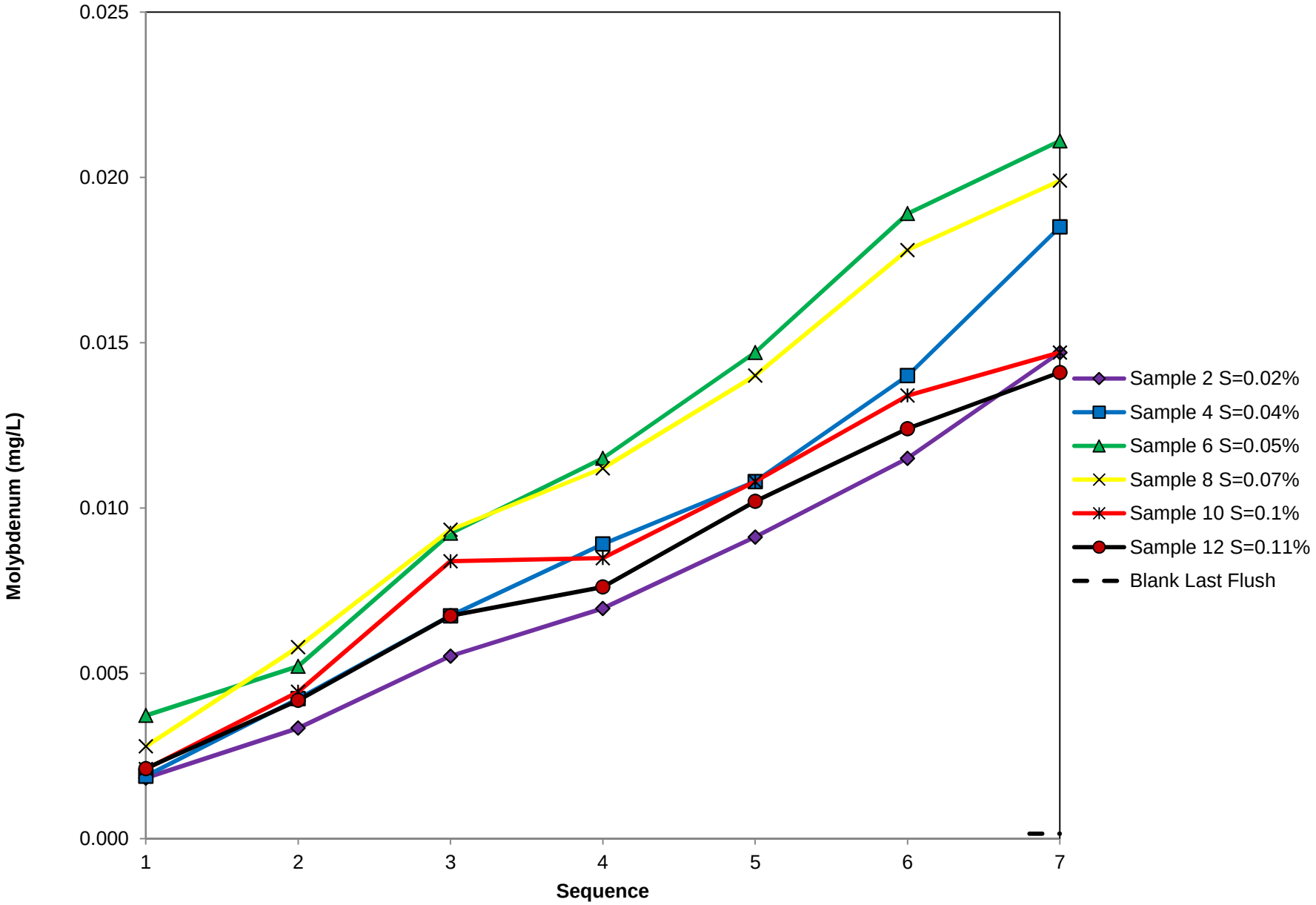

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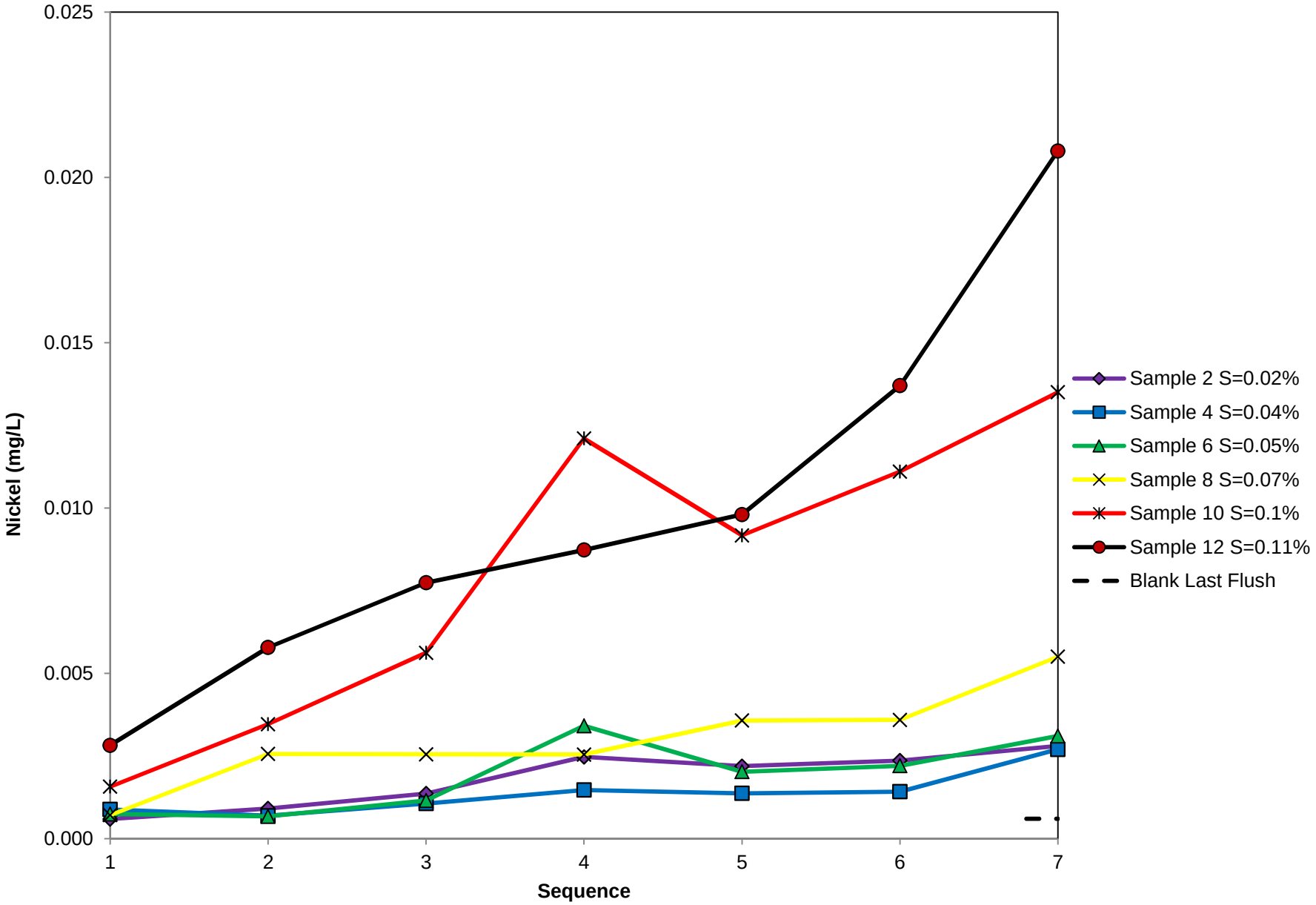

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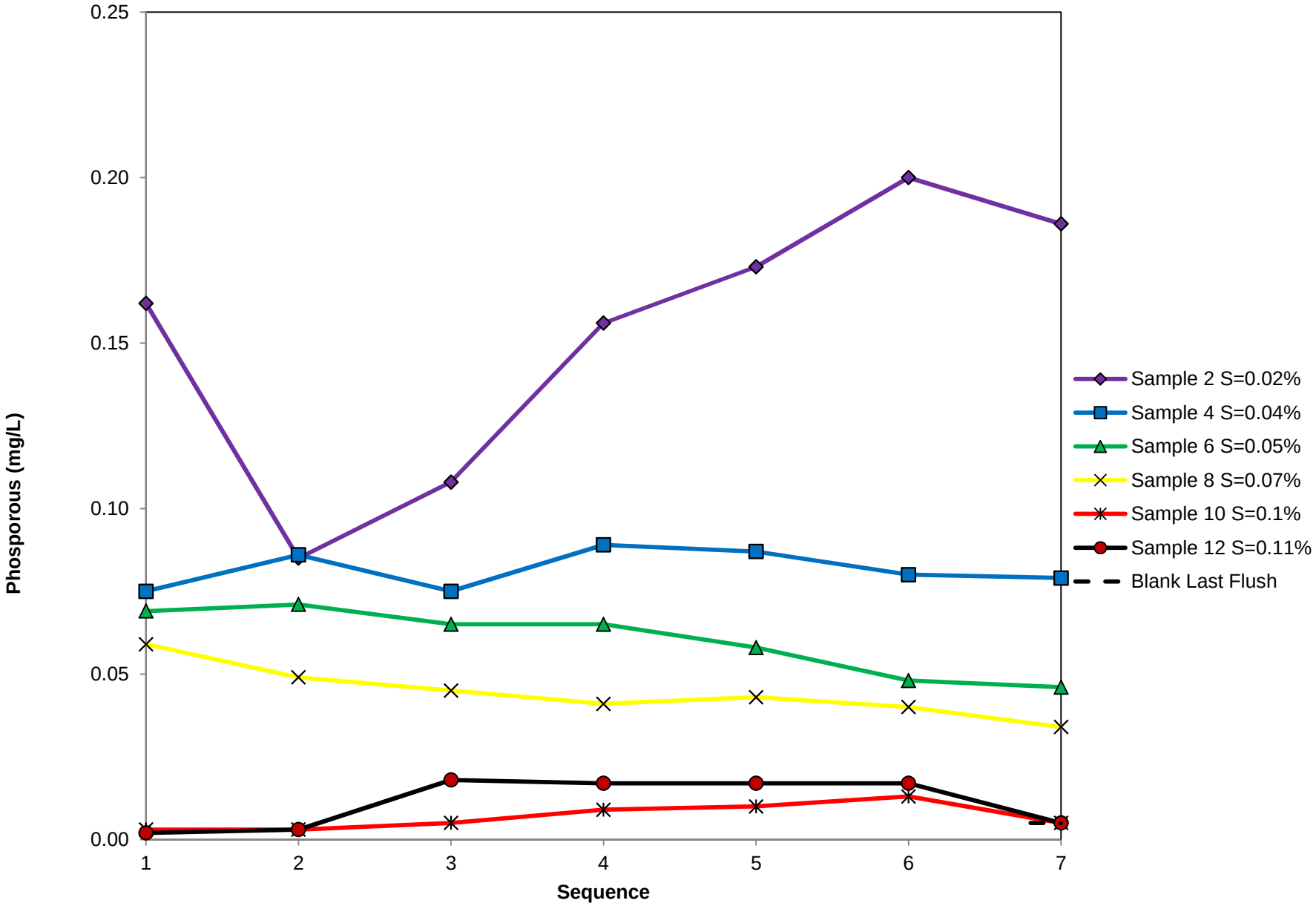

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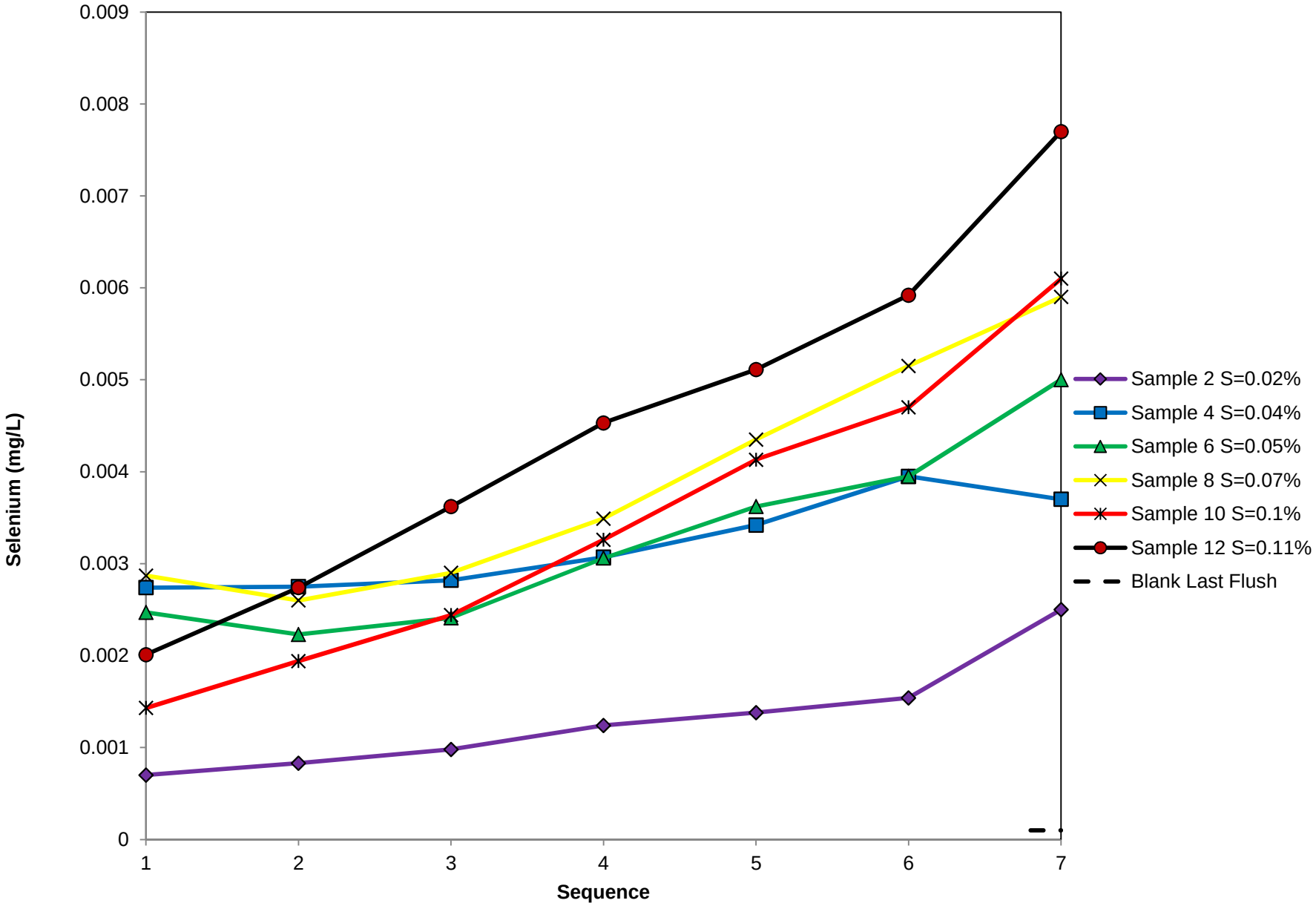

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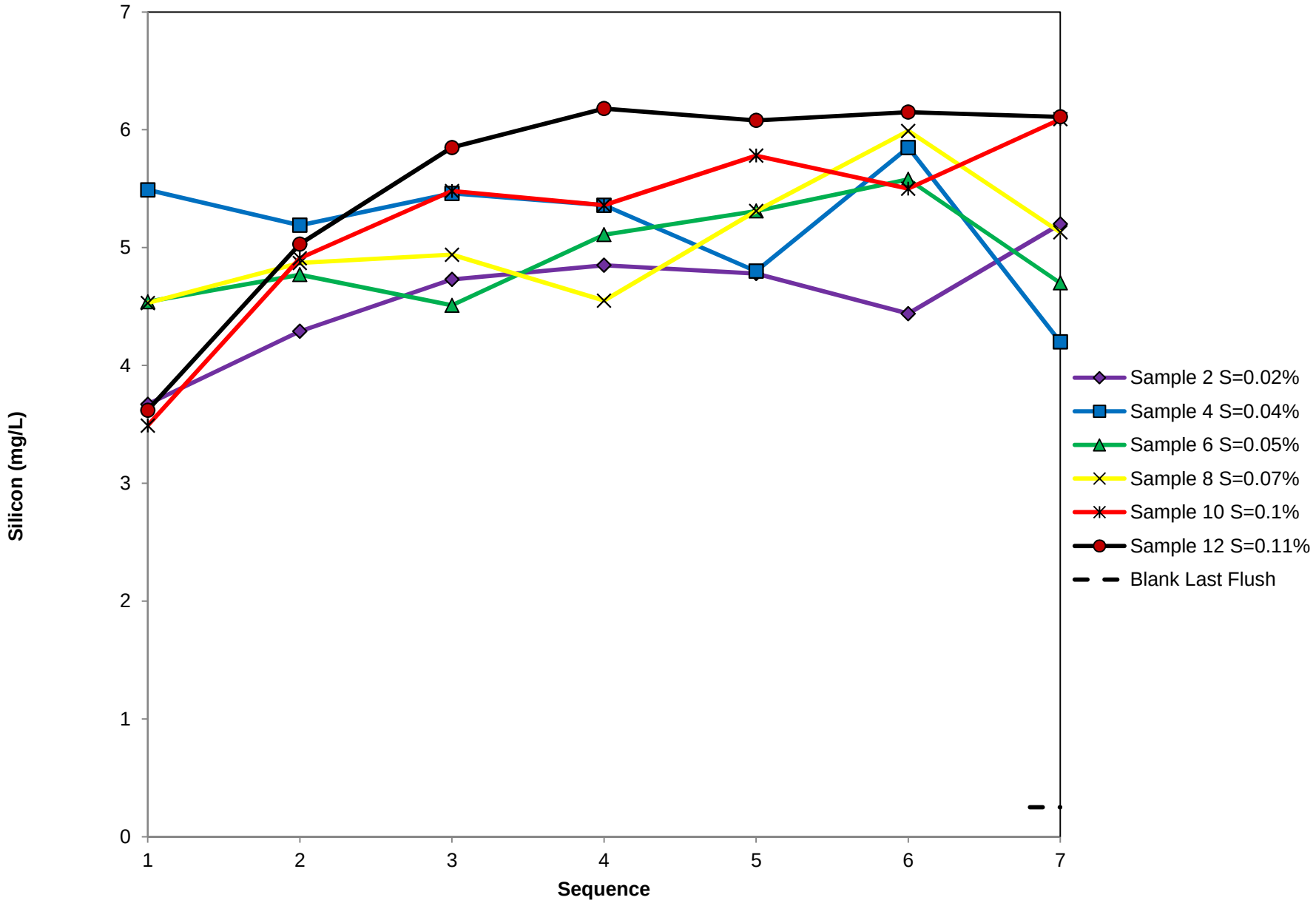

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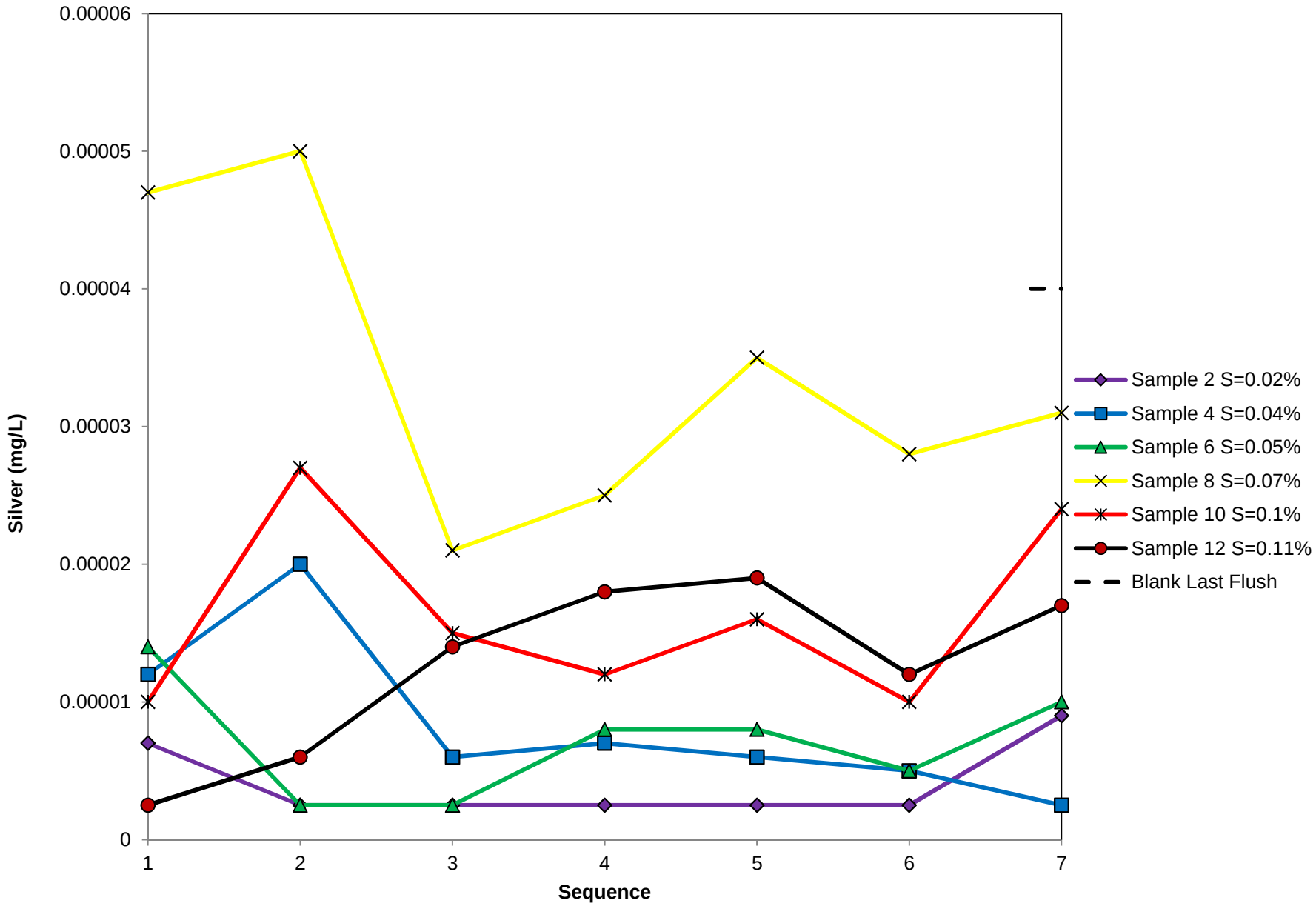

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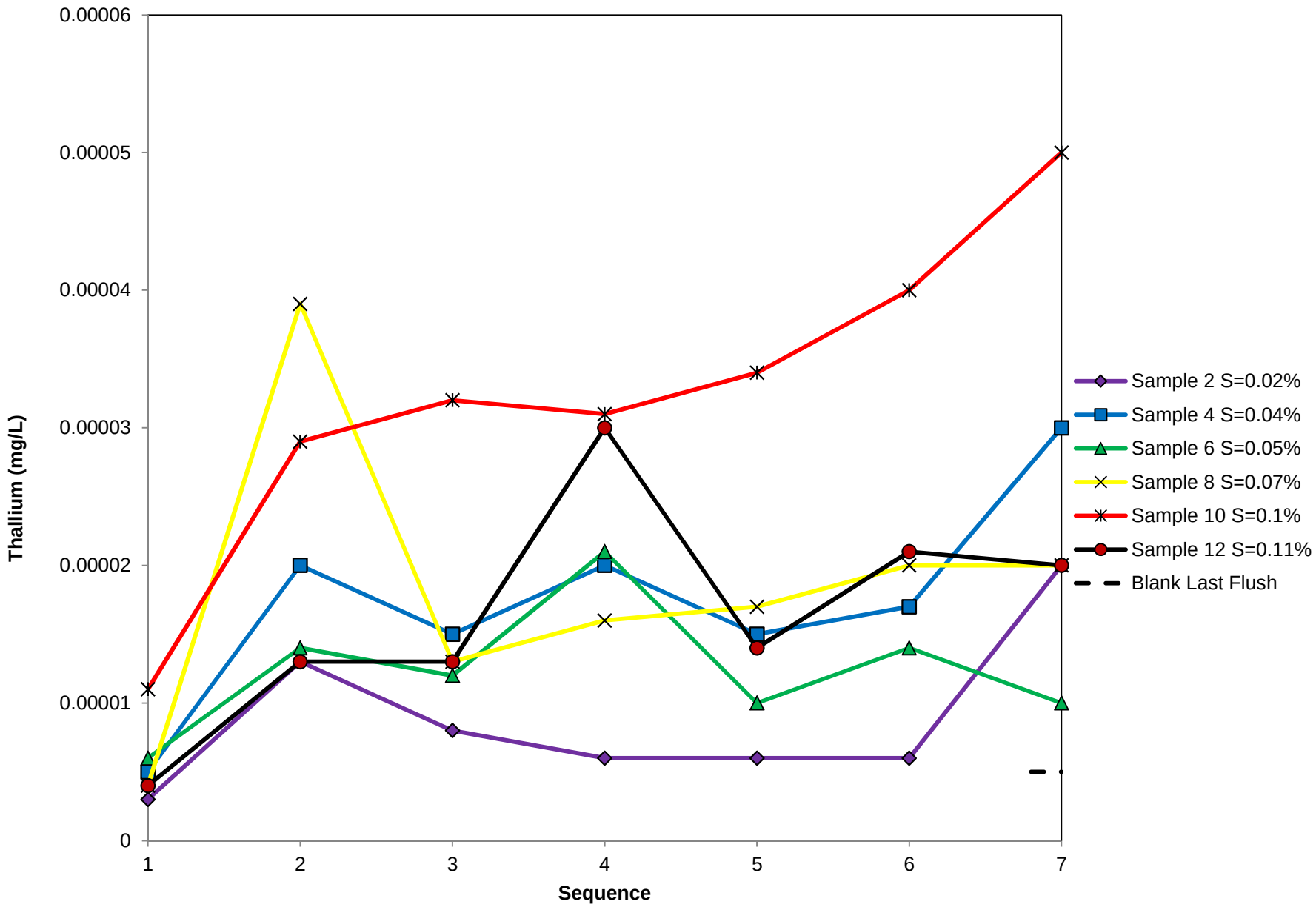

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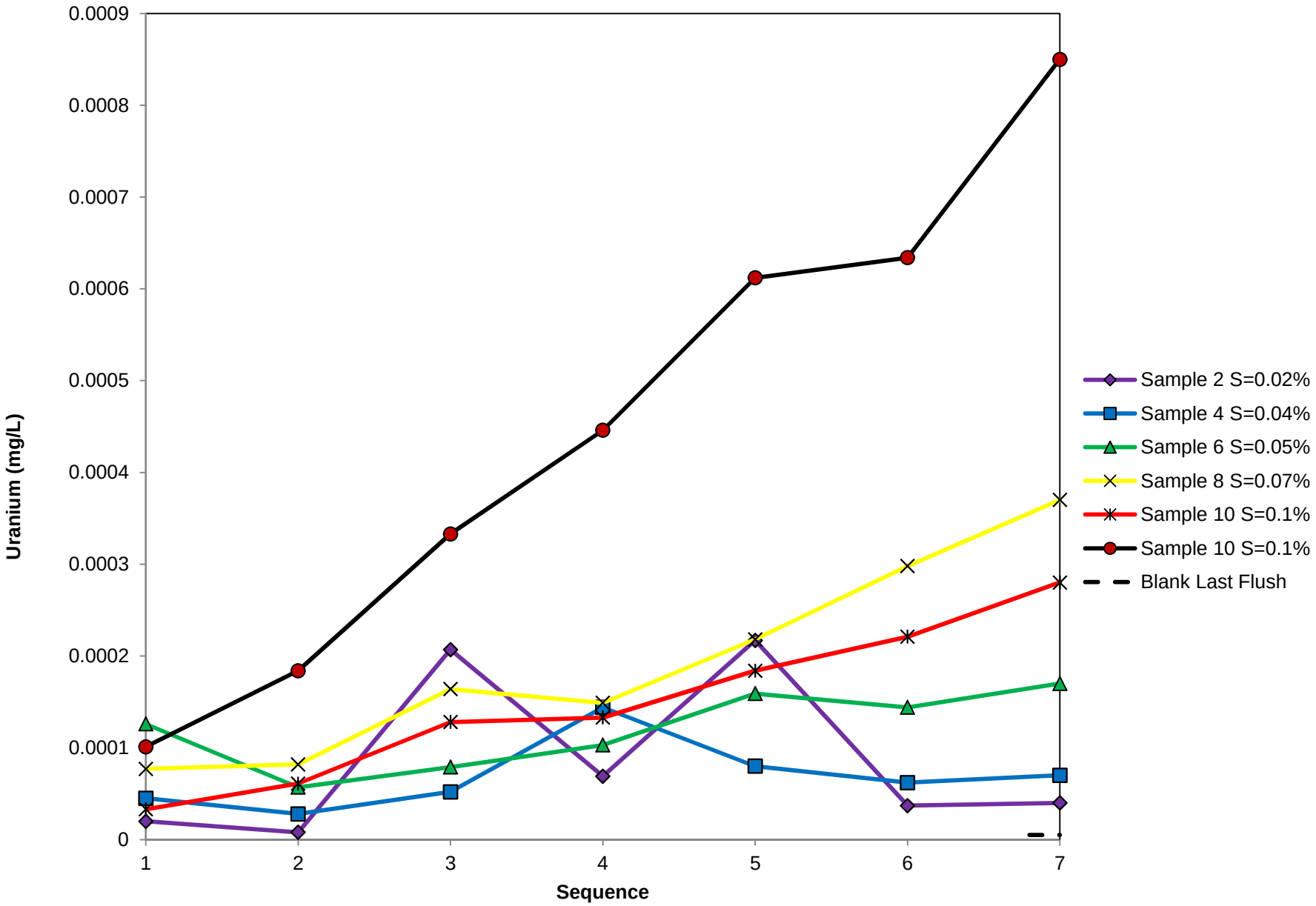

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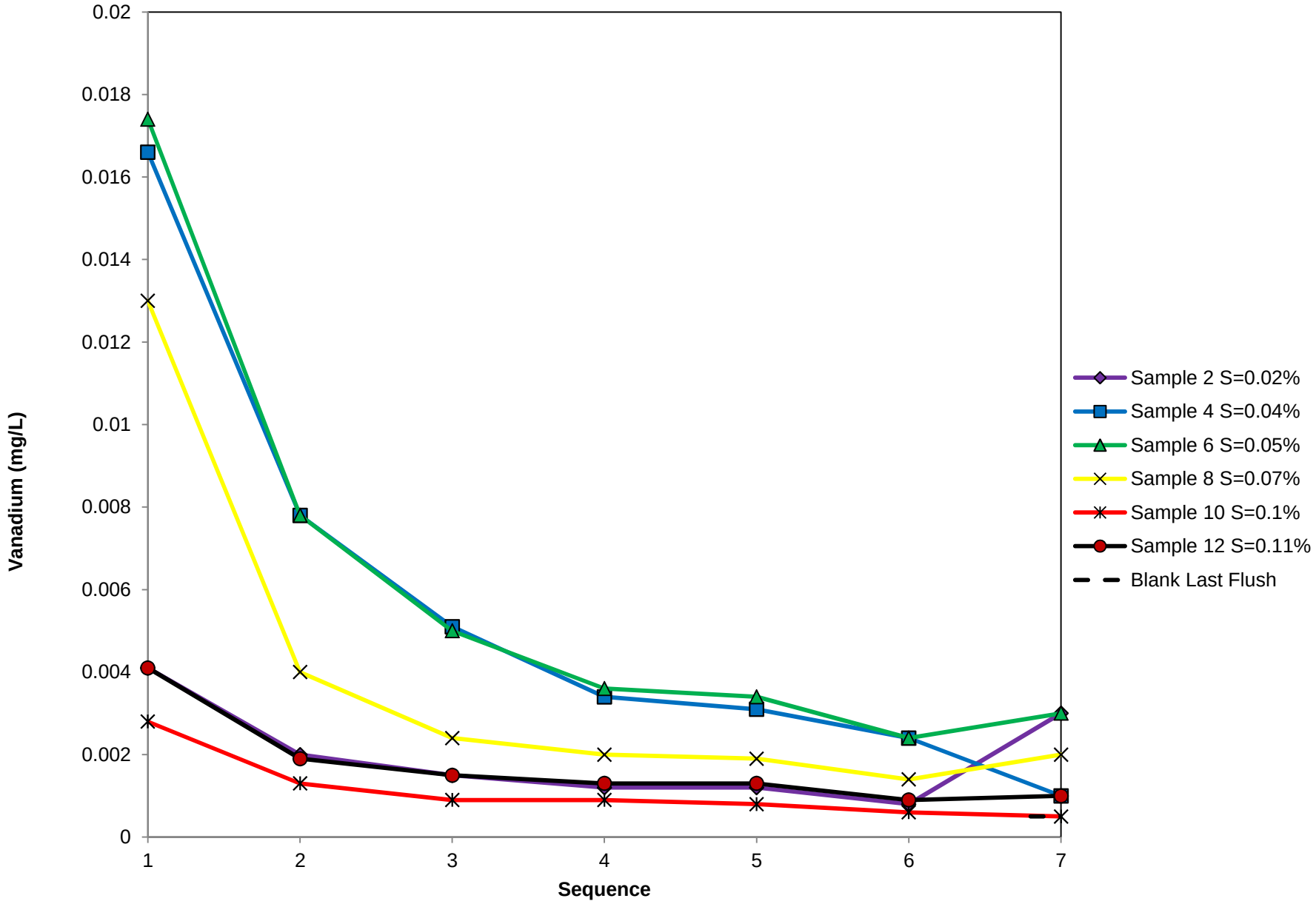

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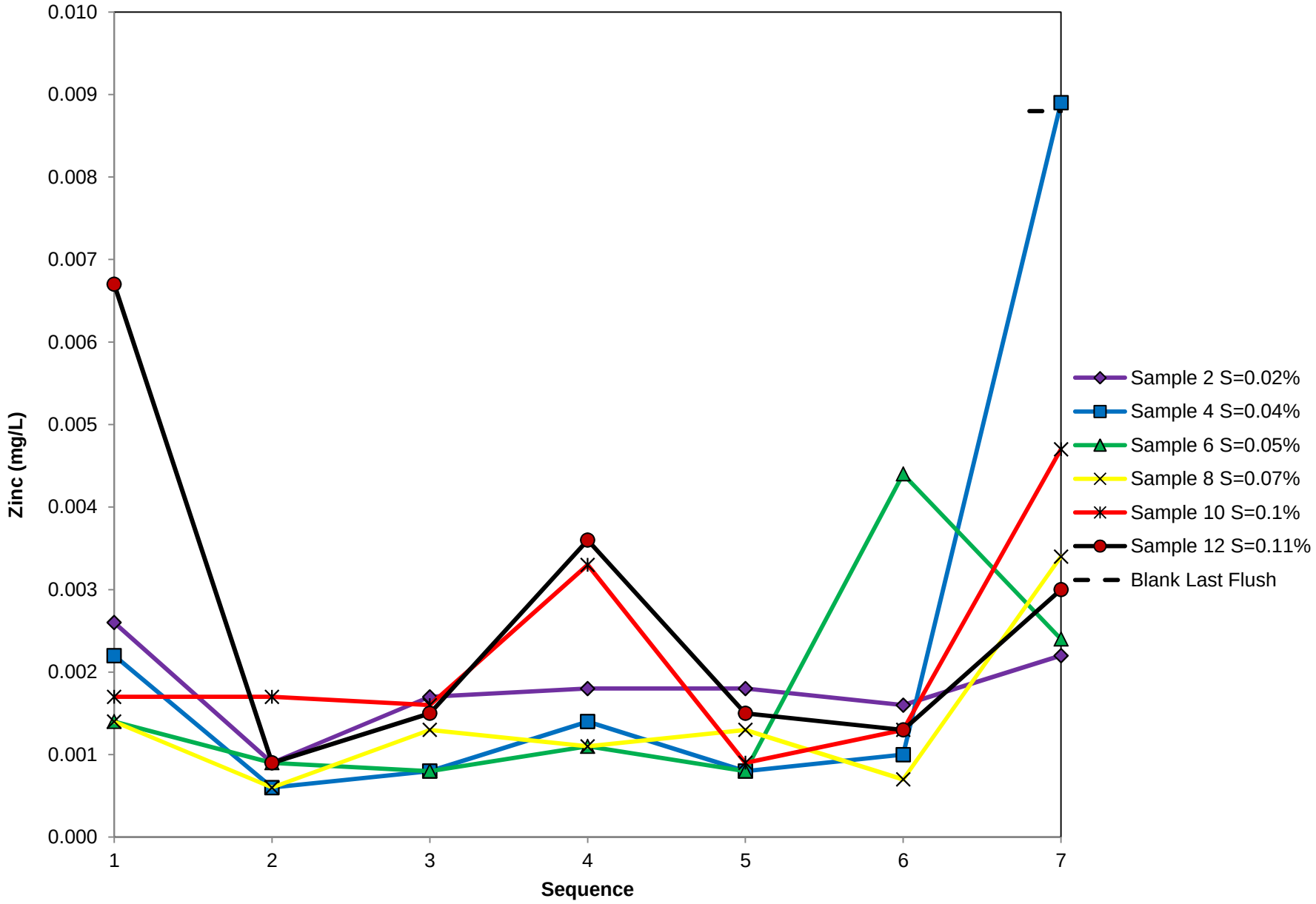

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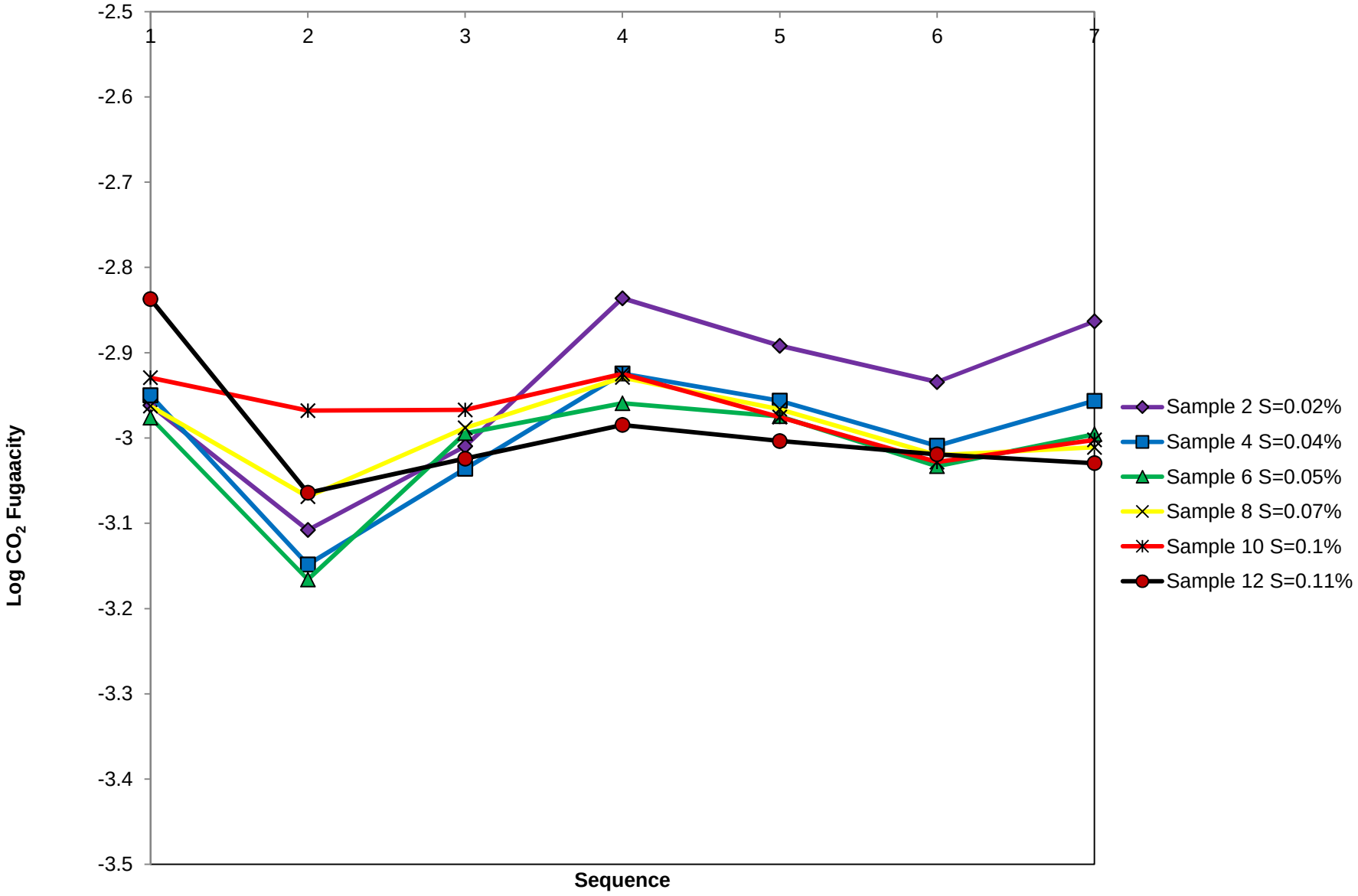
Attachment 3

Graphs of Saturation Indices



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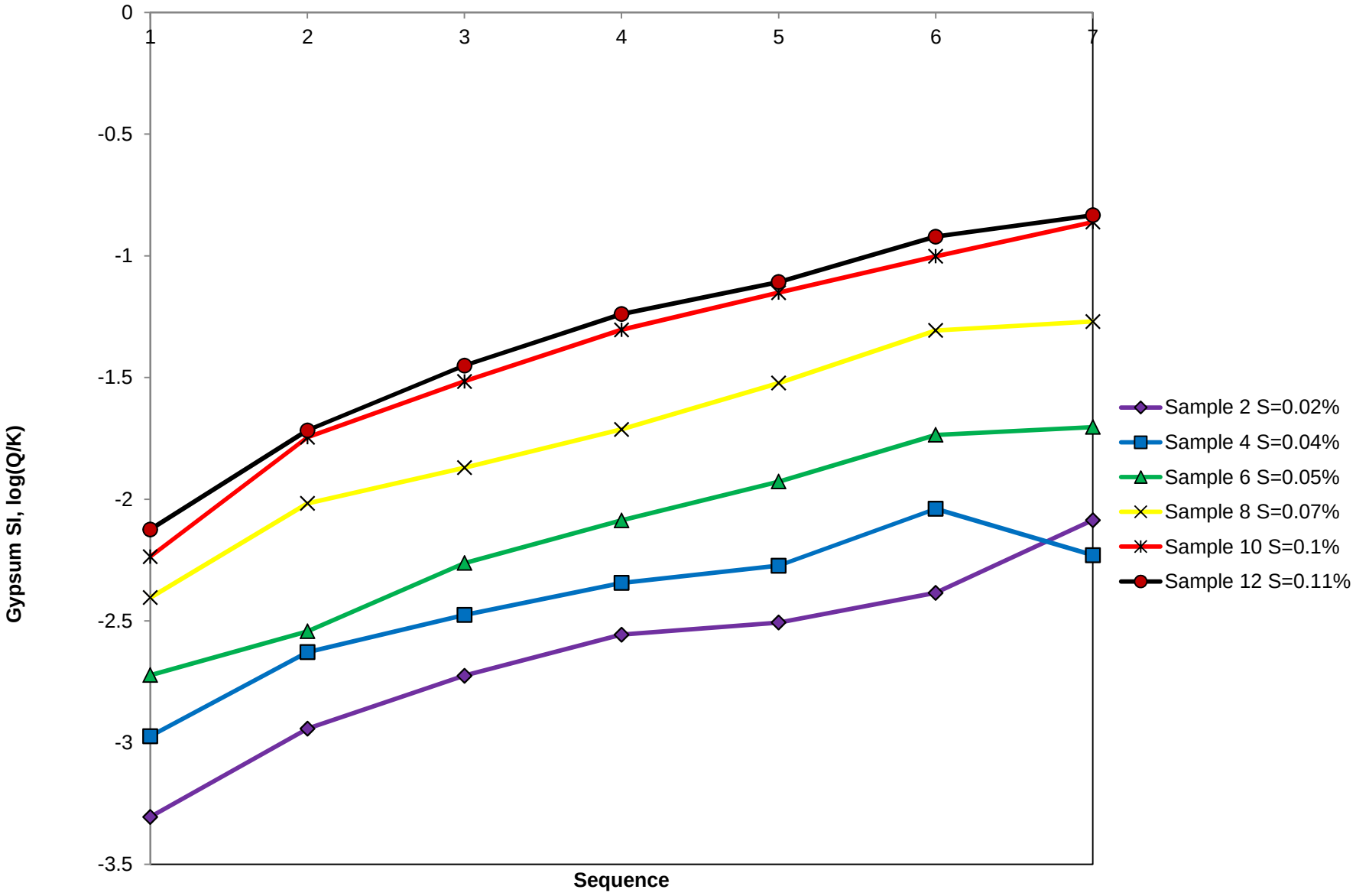

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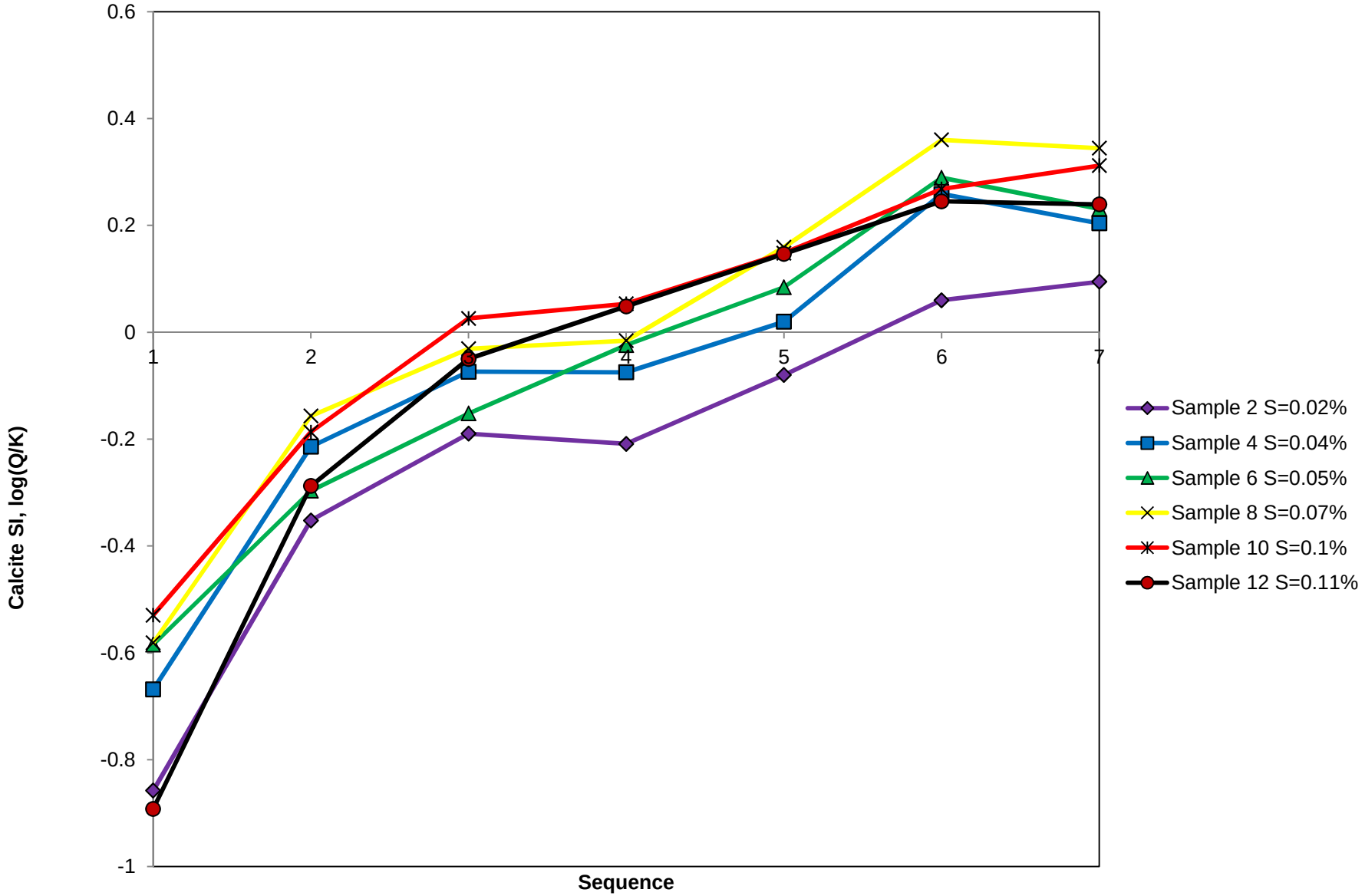

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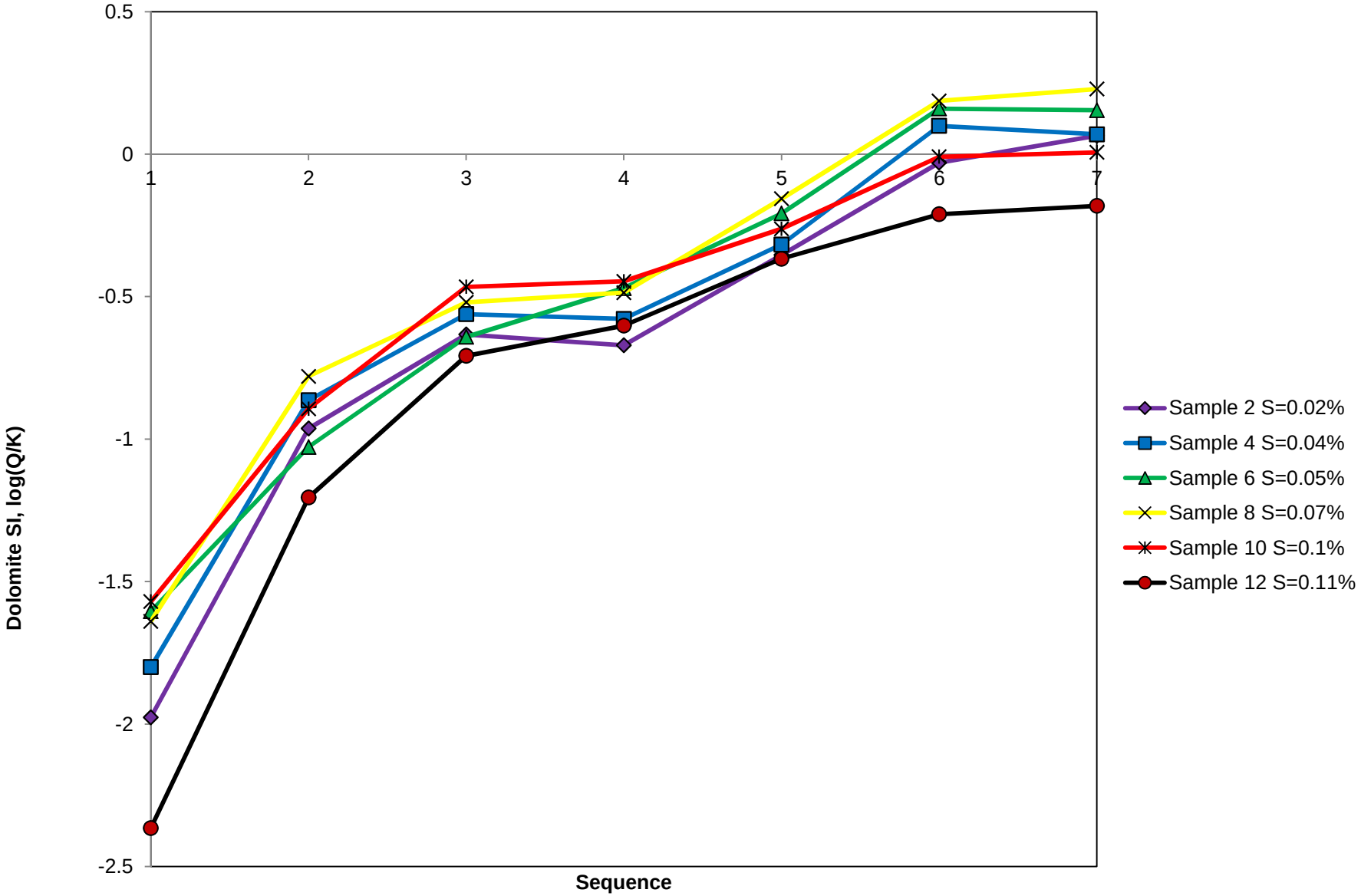

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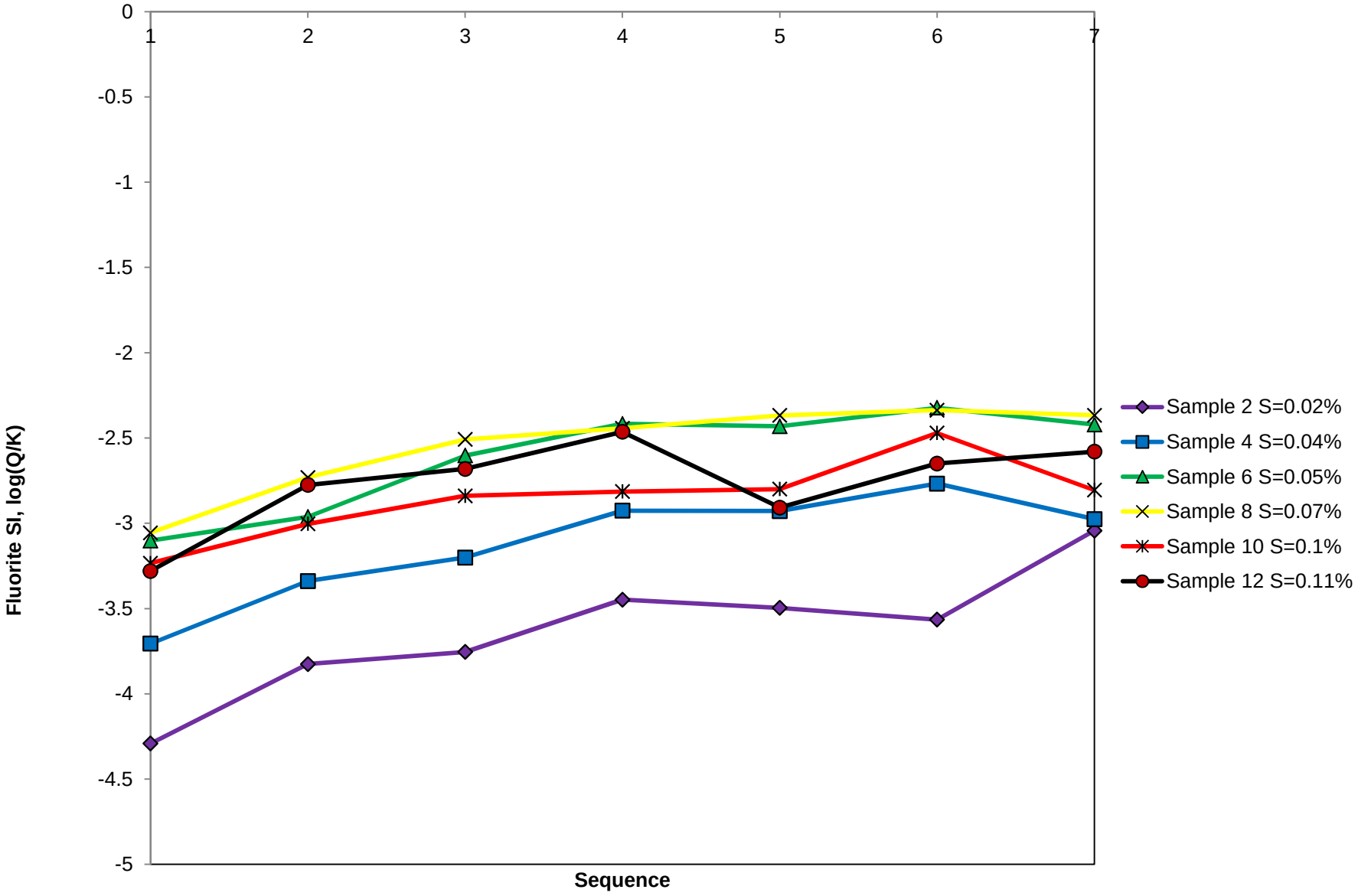

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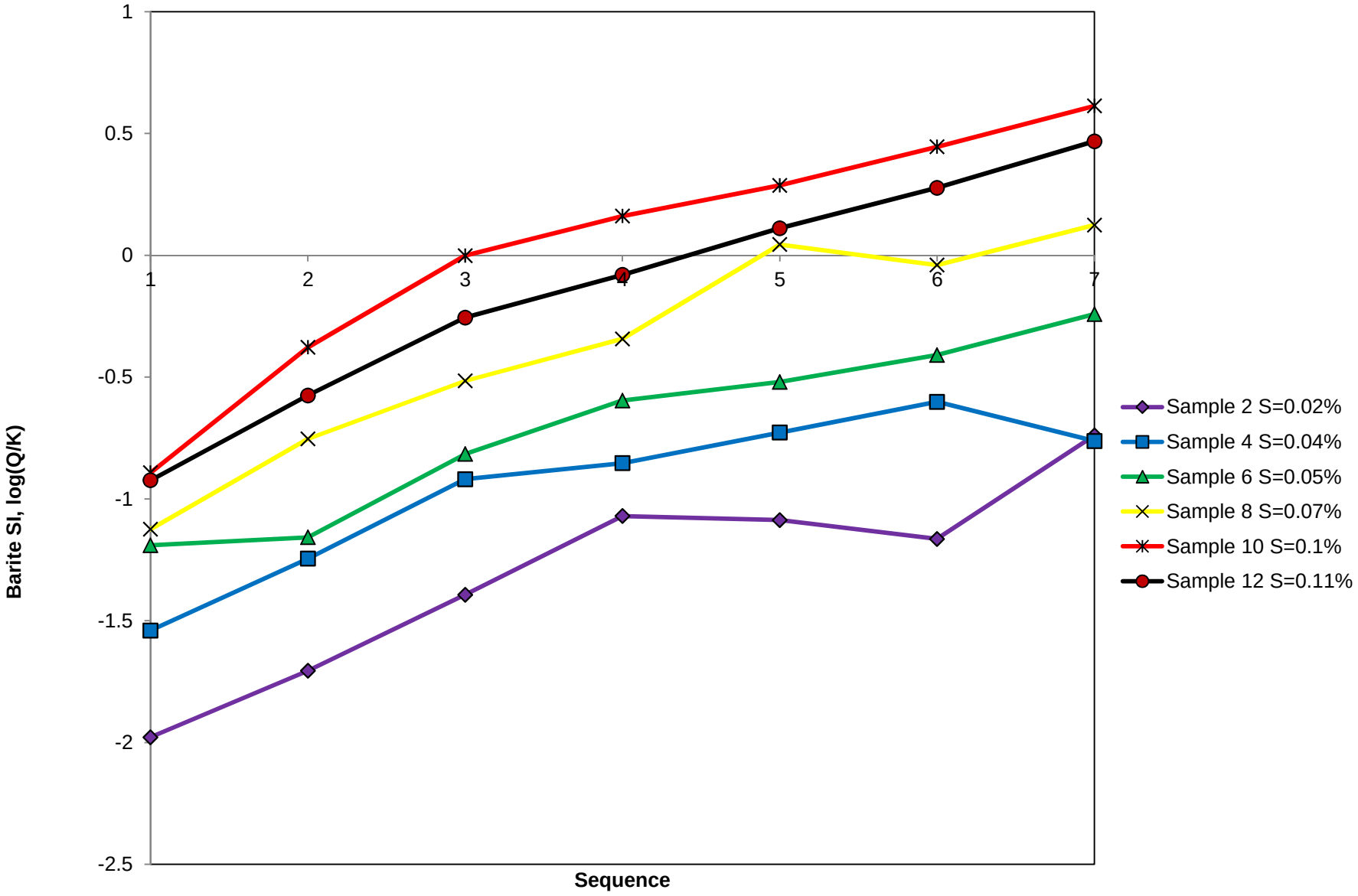

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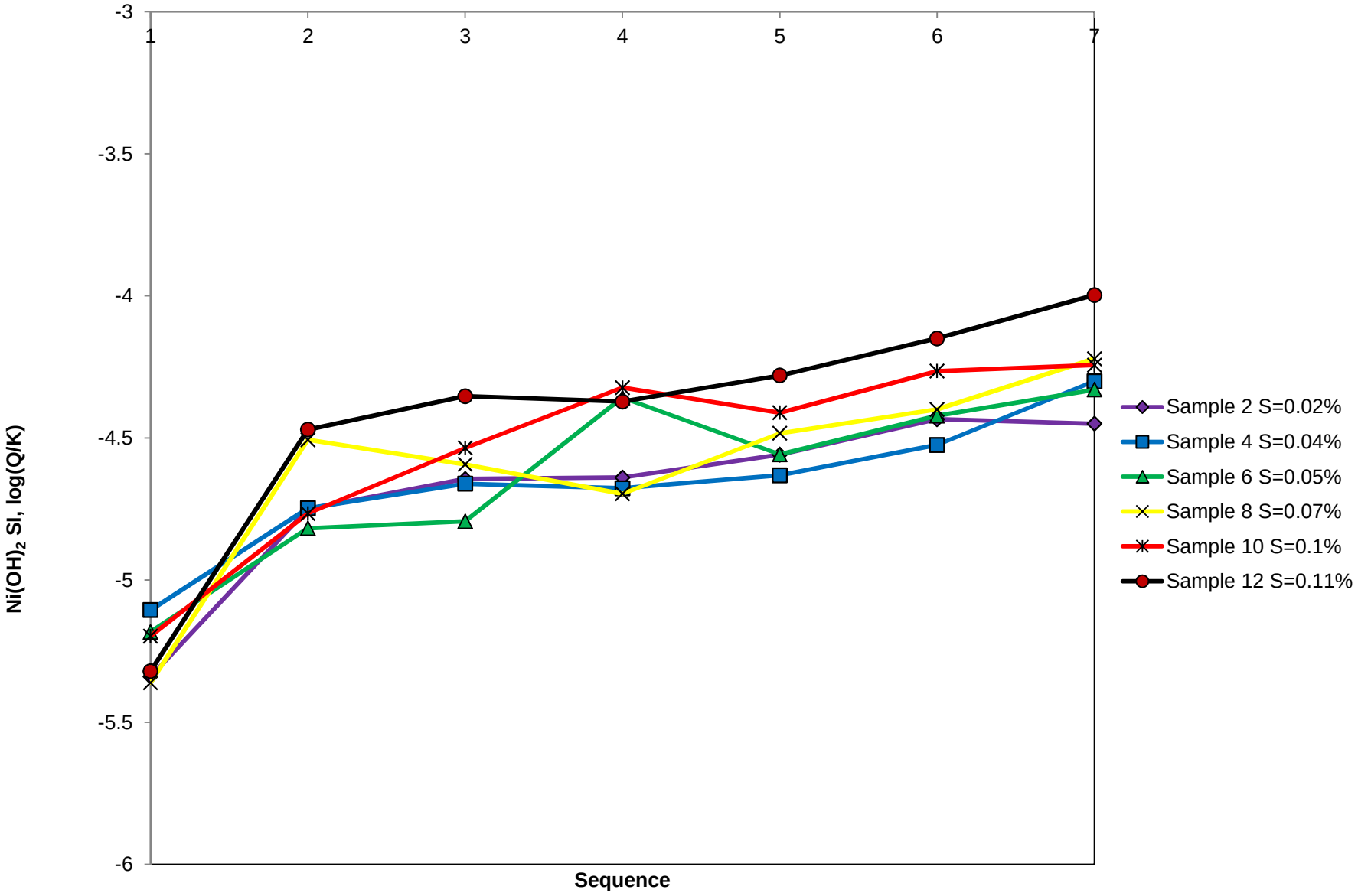

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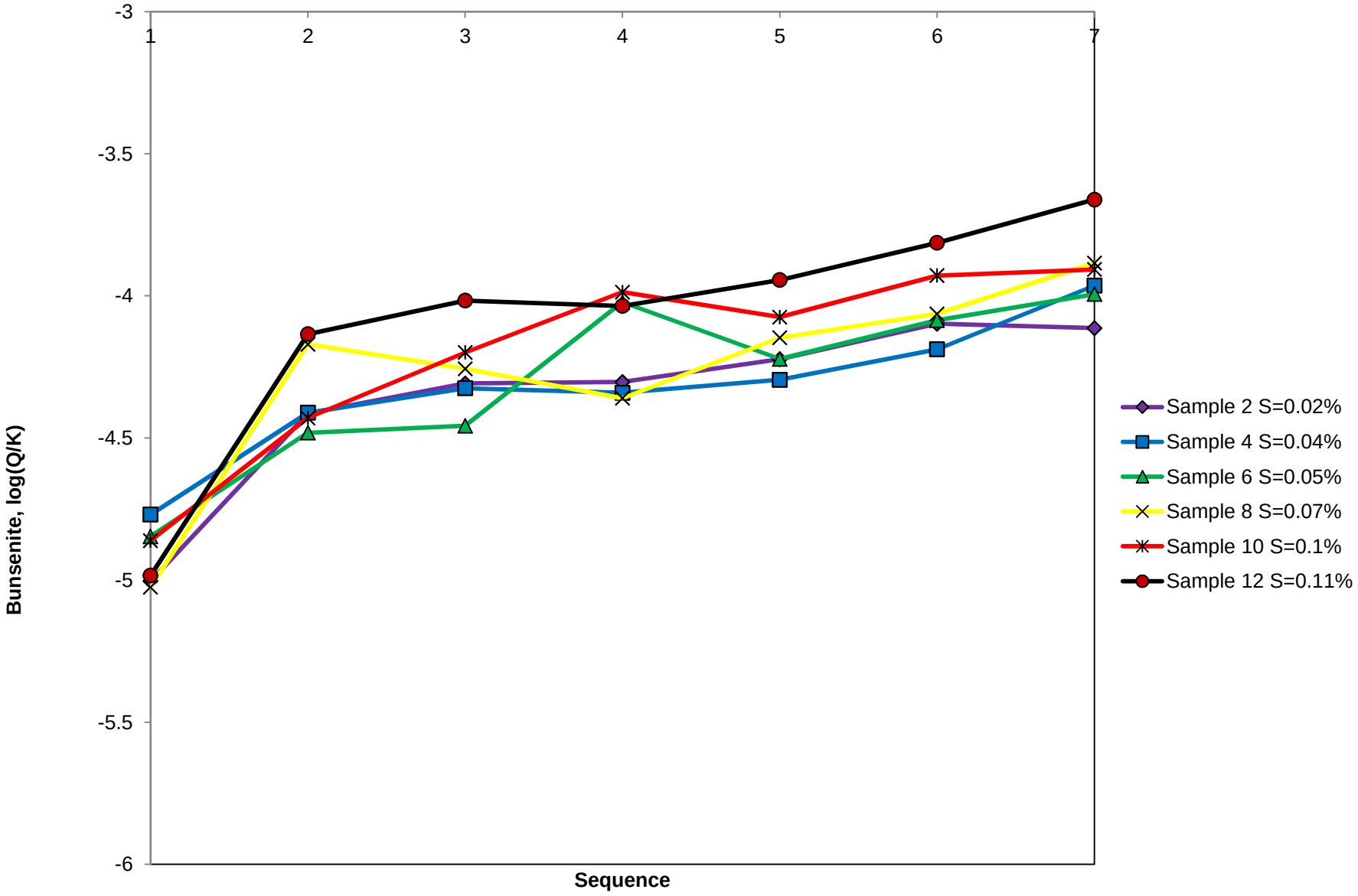

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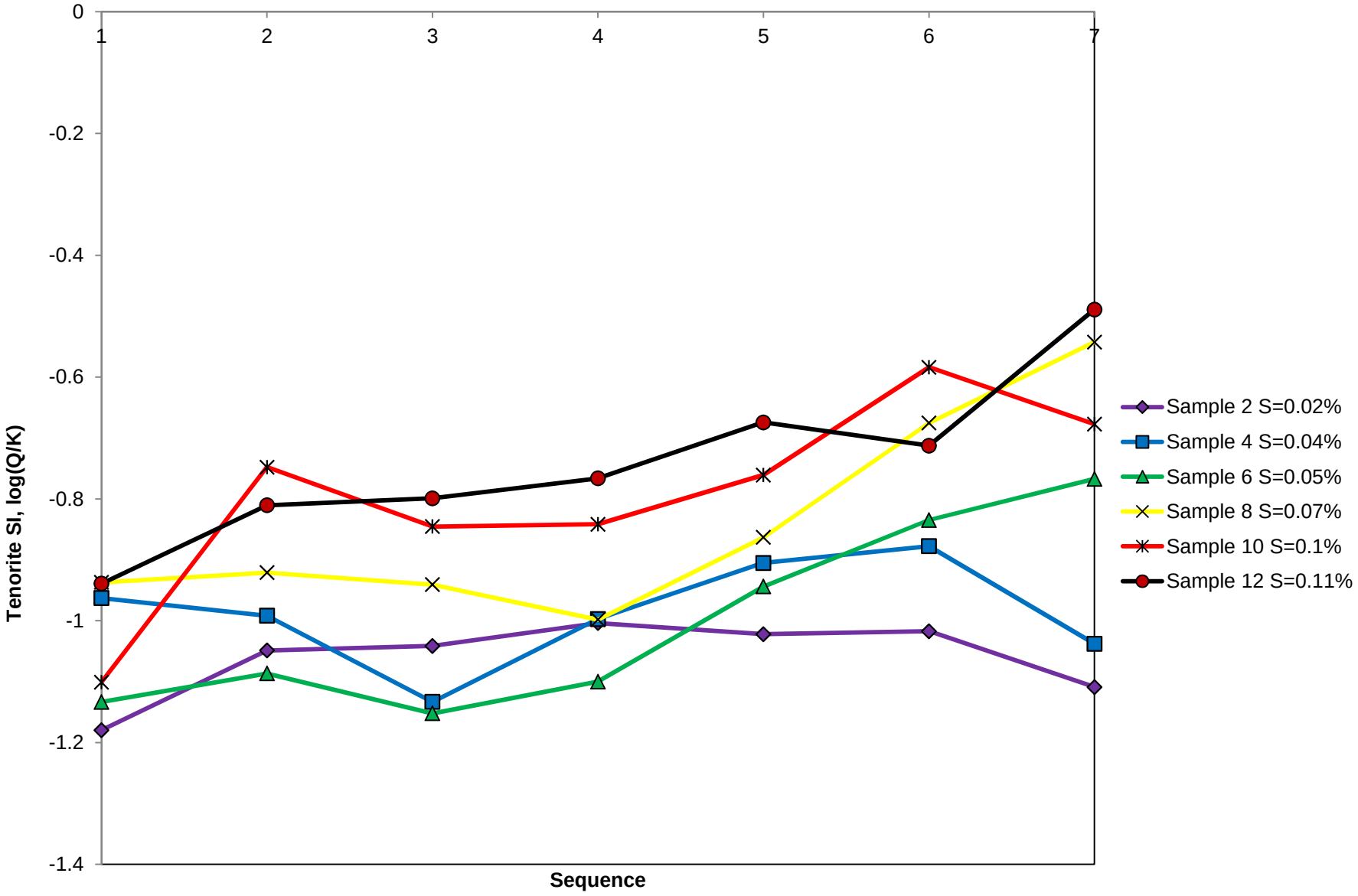

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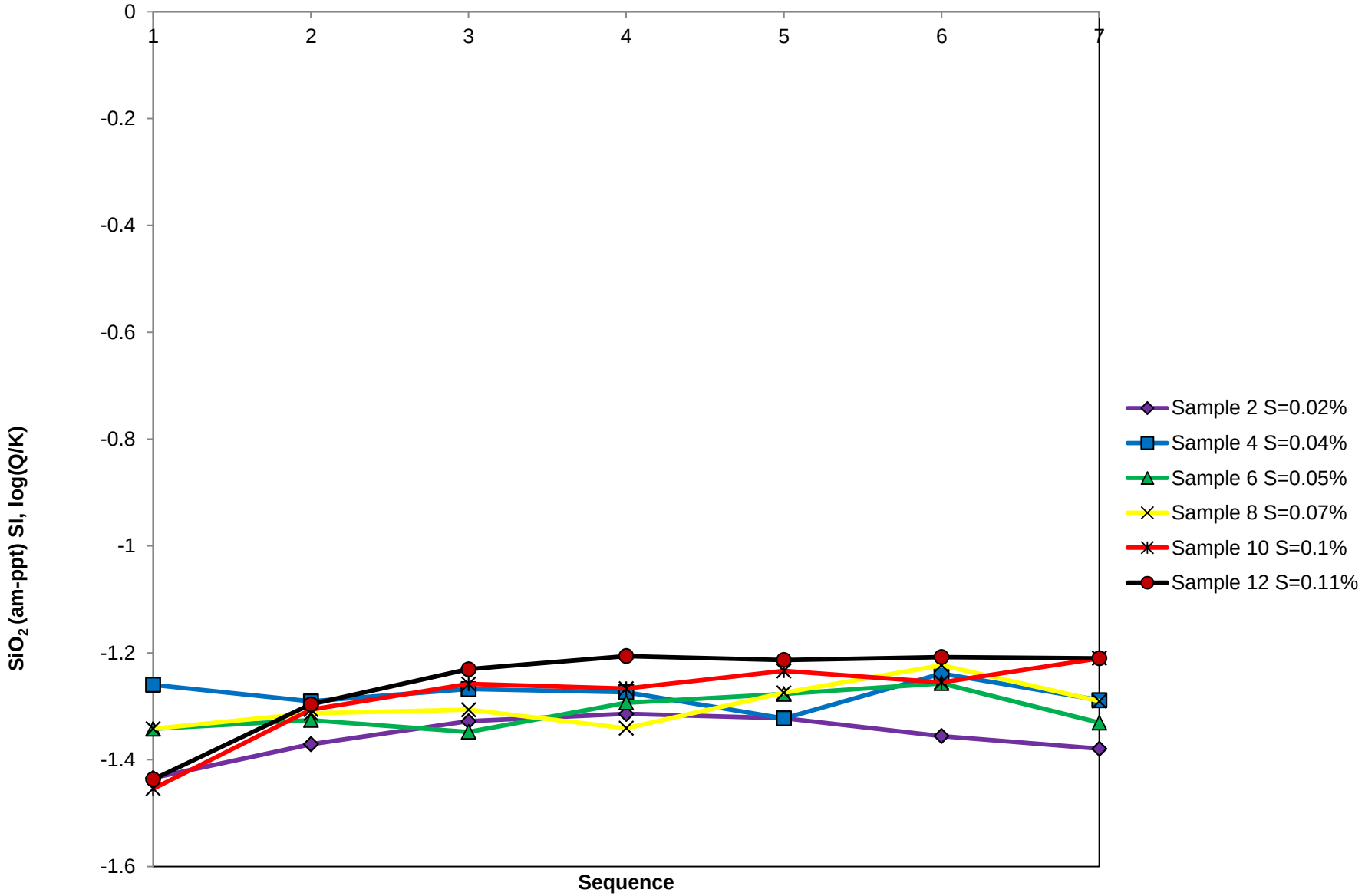

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Attachment C

2015 Update on Kinetic Test Data

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Memo

To:	Jim Scott	**Date:**	January 21, 2015[1]
Company:	PolyMet Mining	**From:**	Stephen Day
Copy to:		**Project #:**	1UP005.001
Subject:	Update on Kinetic Test Data, NorthMet Project – DRAFT		

1 Introduction

Kinetic tests (mainly humidity cells) were initiated for the NorthMet Project in 2004 on samples of rock, simulated tailings and hydrometallurgical process residues according to overall program designs prepared following discussion between PolyMet Mining Inc. (PolyMet), Minnesota Department of Natural Resources (MDNR) and SRK Consulting (Canada) Inc. (SRK). Results from these tests have been used at various junctures of the project to develop waste management plans and evaluate project environmental effects. In 2009, the program was modified in consultation with MDNR to stop some tests and modify the frequency of analysis of leachates based on trend interpretation (SRK 2009).

The purpose of this memorandum is to provide an update on the progress of the testwork and transmit release rates as it relates to water quality source term predictions in the Environmental Impact Statement (EIS).

2 Data Presentation

Electronic updates to the data graphing and calculations are available on Barr Engineering's NorthMet Project website. The data period covered by the testwork now includes up to 6.5 years because some waste rock humidity cells were initiated in August 2005. The cutoff for interpreting data for this update was the end of January to early February 2012. Due to reduced analytical frequency for most tests (SRK 2009), complete QA review and update of the database occurs quarterly. Data collected subsequent to February 2012 completed QA after this update was prepared and are not included in this review.

3 Waste Rock, Lean Ore and Ore Kinetic Tests

3.1 Status of Program

The original waste rock humidity cell program consisted of 92 tests distributed according to the four initially defined classifications shown in Table 1. Waste rock was subsequently classified into three categories. The current program consists of 43 tests with most ongoing tasks in the mid-sulfur range for waste rock and lean ore (19 tests). Table 2 shows all samples and the testwork duration used to calculate rates for use in water quality predictions.

The primary uses of the data in the waste characterization program are to:

- Understand long term performance of wastes and inform definition of waste management categories based on sulfur content; and
- Provide leaching rates for use in source term geochemical predictions.

[1] This version replaces a memo of the same title dated October 19, 2012 to correct graphs shown in Figure 1.

Table 1. Sample Categories

Initial Classification	Waste Category	Totals	Stopped	Continuing
S≤0.05%	1	27	20	7
S>0.05%	1	10	6	4
	2/3	14	3	11
	4	13	8	5
Lean Ore	1	4	3	1
	2/3	13	5	8
	4	8	4	4
Ore	Not applicable	3	0	3
Totals	All	92	49	43

Source: G:\PolyMet Mining\1UP005.01_Northmet_project_2004\Testwork\WR_KT\Characteristics\Humidity_Cell_Table\[1UP005.001_WR_Outcomes_ver01.xlsx]Testwork Tallies

Results are described below in the context of these two objectives. Each heading describes results according to the waste categories (1, 2/3 and 4). Charts for indicator parameters are shown to indicate major features (Figures 1 to 4). Overall tendency toward acidic conditions is shown by pH. Sulfate indicates sulfide mineral oxidation rates. Nickel is shown because as observed in this test program and testwork performed previously by DNR it (along with cobalt) tends to respond first to declining pH. Copper is shown because it leaches more rapidly as pH declines further. Arsenic provides an indication of how a heavy oxyanion is affected by changes in pH.

In order to obtain average leaching rates, typical trends have been recognized and average rates calculated for common qualitative leaching features or “conditions” as summarized in Table 3. In these definitions, “stable” means typically neither clearly increasing or decreasing. Due to variability in the testwork results, the definitions are not strict and require some flexibility in interpretation. For example, the transition from Condition 1 to 2 may not occur exactly at pH 7.

Condition 2 is typically accompanied by increasing release of cobalt and nickel which may continue into Condition 3. Lower pH (less than 6) typically occurs during Condition 3 and 4. In some cases, pH recovery is observed during Condition 4 as oxidation rates decrease.

Table 2. List of Rock Humidity Cells

HCT ID	Comment	Original Waste Type	Geological Unit	Rock Type	Category	Sample ID	S %	Initial Date	Data Record Length weeks	Duration Used in Current Modeling[2] weeks
1	9 Dup	RWR	1	Anorthositic	1	99-320C(830-850)	0.09	8/8/2005	337	284
3		RWR	1	Anorthositic	1	00-361C(345-350)	0.05	8/8/2005	337	284
26		NRWR	2	Anorthositic	1	00-366C(185-205)	0.02	8/9/2005	198	198
27		NRWR	2	Anorthositic	1	00-366C(230-240)	0.02	8/9/2005	198	198
28		NRWR	2	Anorthositic	1	99-320C(165-175)	0.03	8/9/2005	198	198
40		NRWR	3	Anorthositic	1	00-334C(30-50)	0.02	8/9/2005	337	284
41	37 Dup	NRWR	3	Anorthositic	1	00-368C(125-145)	0.04	8/10/2005	337	284
42		NRWR	3	Anorthositic	1	00-368C(20-40)	0.04	8/10/2005	198	198
13		NRWR	1	Troctolitic	1	00-340C(595-615)	0.04	8/10/2005	198	198
14		NRWR	1	Troctolitic	1	00-334C(580-600)	0.06	8/15/2005	336	284
15		RWR	1	Troctolitic	1	00-334C(640-660)	0.07	8/8/2005	337	284
16		RWR	1	Troctolitic	1	00-347C(795-815)	0.07	8/8/2005	198	198
29		NRWR	2	Troctolitic	1	99-318C(250-270)	0.04	8/9/2005	198	198
30		NRWR	2	Troctolitic	1	00-373C(95-115)	0.04	8/9/2005	198	198
31		NRWR	2	Troctolitic	1	00-373C(75-95)	0.06	8/9/2005	198	198
32		RWR	2	Troctolitic	1	00-357C(110-130)	0.08	8/9/2005	198	198
33		RWR	2	Troctolitic	1	99-320C(315-330)	0.07	8/9/2005	337	284
43		NRWR	3	Troctolitic	1	00-366C(35-55)	0.02	8/10/2005	198	198
44		NRWR	3	Troctolitic	1	00-334C(110-130)	0.04	8/10/2005	198	198
45		NRWR	3	Troctolitic	1	00-347C(155-175)	0.06	8/10/2005	198	198
46		RWR	3	Troctolitic	1	00-347C(280-300)	0.06	8/10/2005	198	198
49		NRWR	4	Troctolitic	1	00-367C(50-65)	0.03	8/10/2005	198	198
50		NRWR	4	Troctolitic	1	00-367C(260-280)	0.04	8/10/2005	198	198
51	57 Dup	NRWR	4	Troctolitic	1	00-367C(290-310)	0.04	8/10/2005	337	284
52		RWR	4	Troctolitic	1	00-370C(20-30)	0.08	8/10/2005	198	198
56		NRWR	5	Troctolitic	1	26064(44-54)	0.02	8/10/2005	337	284
59		NRWR	5	Troctolitic	1	26064(264+146-269+156)	0.06	8/10/2005	337	284
60		NRWR	6	Troctolitic	1	26056(110-125)	0.04	8/10/2005	198	198
74		NRWR	1	Troctolitic	1	26029(815-825)	0.02	9/8/2005	194	194
78		NRWR	6	Troctolitic	1	26056(135-153)	0.05	9/22/2005	331	278
99		LeanOre	1	Troctolitic	1	00-326C(250-265)	0.08	10/28/2005	187	186
21		RWR	1	Ultramafic	1	00-357C(335-340)	0.08	8/9/2005	198	198
35		NRWR	2	Ultramafic	1	00-368C(460-465)	0.06	8/9/2005	198	198
36		NRWR	2	Ultramafic	1	26055(940-945)	0.06	8/9/2005	198	198
39		NRWR	2	Ultramafic	1	26098+00-337C	0.1	8/9/2005	198	198
72	61 Dup	LeanOre	2	Ultramafic	1	00-361C(240-245)	0.06	8/11/2005	337	284
101		LeanOre	2	Ultramafic	1	26039(310-315)	0.06	10/28/2005	187	186
2		RWR	1	Anorthositic	2/3	00-361C(310-320)	0.18	8/8/2005	337	284
103		LeanOre	1	Anorthositic	2/3	99-320C(400-405)	0.18	10/28/2005	326	273
5		RWR	1	Sed Honfels	2/3	26030(1047-1052)	0.24	8/8/2005	337	284
6		RWR	1	Sed Honfels	2/3	26061(1218-1233)	0.44	8/8/2005	337	284
7		RWR	1	Sed Honfels	2/3	00-340C(990-995)	0.55	8/8/2005	337	284
17		RWR	1	Troctolitic	2/3	00-350C(580-600)	0.19	8/8/2005	337	284
18		RWR	1	Troctolitic	2/3	00-327C(225-245)	0.44	8/8/2005	198	198
34		RWR	2	Troctolitic	2/3	00-369C(335-345)	0.18	8/9/2005	337	284
47		RWR	3	Troctolitic	2/3	00-326C(60-70)	0.14	8/10/2005	337	284
48	38 Dup	RWR	3	Troctolitic	2/3	00-369C(305-325)	0.25	8/10/2005	198	198
53		RWR	4	Troctolitic	2/3	00-369C(20-30)	0.21	8/10/2005	337	284
54		RWR	4	Troctolitic	2/3	00-367C(170-175)	0.51	8/10/2005	337	284
71		LeanOre	2	Troctolitic	2/3	00-340C(380-390)	0.15	8/11/2005	198	198
75		LeanOre	3	Troctolitic	2/3	26049+26030	0.59	8/11/2005	198	198
77		LeanOre	5	Troctolitic	2/3	26056(302-312)	0.23	8/11/2005	337	284
80		LeanOre	6	Troctolitic	2/3	26142(360+345-365+350)	0.18	8/11/2005	337	284
96		LeanOre	2	Troctolitic	2/3	99-318C(325-330)	0.17	10/28/2005	326	273
98		LeanOre	5	Troctolitic	2/3	26056(282-292)	0.32	10/28/2005	187	186
100		LeanOre	1	Troctolitic	2/3	00-340C(910-925)	0.36	10/28/2005	326	273
102		LeanOre	1	Troctolitic	2/3	00-331C(190-210)	0.42	10/28/2005	326	273
105		LeanOre	3	Troctolitic	2/3	00-367C(495-500)	0.28	10/28/2005	326	273
22		RWR	1	Ultramafic	2/3	00-326C(680-685)	0.30	8/9/2005	198	198
23		RWR	1	Ultramafic	2/3	00-357C(535-540)	0.2	8/9/2005	337	284
94		LeanOre	1	Ultramafic	2/3	00-344C(630-635)	0.34	10/28/2005	187	186
95		LeanOre	1	Ultramafic	2/3	00-326C(495-505)	0.16	10/28/2005	187	186

[2] Waste Characterization Data Package Version 9 (July 3 2012).

HCT ID	Comment	Original Waste Type	Geological Unit	Rock Type	Category	Sample ID	S %	Initial Date	Data Record Length weeks	Duration Used in Current Modeling[2] weeks
104		LeanOre	2	Ultramafic	2/3	00-326C(225-235)	0.12	10/28/2005	326	273
4		RWR	1	Anorthositic	4	00-343C(240-250)	0.68	8/8/2005	198	198
65		LeanOre	1	Anorthositic	4	26027(616-626)	1.83	8/11/2005	337	284
93		LeanOre	1	Anorthositic	4	00-331C(255-260)	0.86	10/28/2005	326	273
8		RWR	1	Sed Honfels	4	00-340C(965-974.5)	1.74	8/8/2005	198	198
11		RWR	1	Sed Honfels	4	26043+26027	2.47	8/8/2005	337	284
68		LeanOre	1	Sed Honfels	4	26062+26026	4.46	8/11/2005	337	284
106		LeanOre	1	Sed Honfels	4	26058(704-715)	1.46	10/28/2005	326	273
19		RWR	1	Troctolitic	4	00-371C(435-440)	0.88	8/8/2005	337	284
20	10 Dup	RWR	1	Troctolitic	4	00-340C(765-780)	1.68	8/8/2005	337	284
55		RWR	4	Troctolitic	4	00-367C(395-400)	0.77	8/10/2005	198	198
69		LeanOre	1	Troctolitic	4	00-340C(725-745)	0.91	8/11/2005	198	198
76		LeanOre	4	Troctolitic	4	00-367(400-405)	1.37	8/11/2005	198	198
24		RWR	1	Ultramafic	4	99-318C(725-735)	0.72	8/9/2005	198	198
25		RWR	1	Ultramafic	4	99-317C(460-470)	1.24	8/9/2005	198	198
70		LeanOre	1	Ultramafic	4	00-344C(515-520)	1.2	8/11/2005	198	198
97		LeanOre	1	Ultramafic	4	00-330C(275-280)	0.75	10/28/2005	187	186
62		RWR	20	Virginia	4	00-361C(737-749)	2	8/11/2005	337	284
63	58 Dup	RWR	20	Virginia	4	00-364C(210-229)	3.79	8/11/2005	198	198
64		RWR	20	Virginia	4	00-337C(510-520)	5.68	8/11/2005	198	198
66		Ore			-	P10	0.86	9/8/2005	333	268
67		Ore			-	P20	0.9	9/8/2005	333	268
73		Ore			-	P30	0.86	9/8/2005	333	268

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Table 3. Qualitative Definition of Leaching Conditions

Condition	Trend Features
1	pH>7, stable SO_4
2	pH<7, typically stable SO_4
3	SO_4 sharply increasing and unstable
4	SO_4 peaked and decreasing

3.2 Description of Results by Waste-Rock Category

3.2.1 Category 1

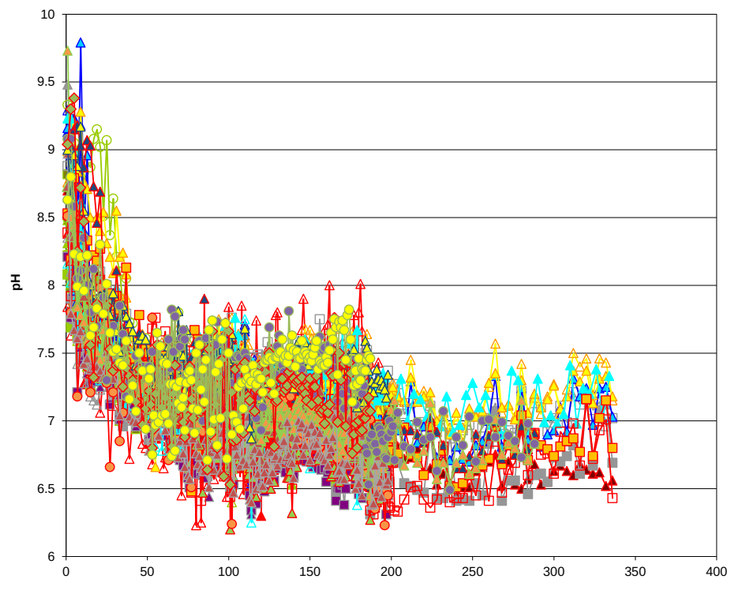
Results for pH, sulfate, nickel, copper and arsenic are shown in Figure 1.

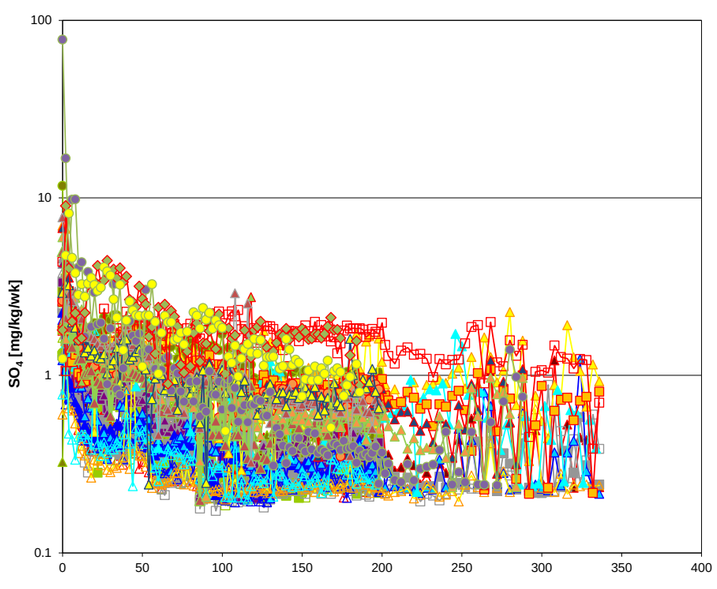
All Category 1 humidity cells have yielded pH above 6 throughout the program. Generally, pHs declined as the tests proceeded. Initial pHs in most cases were above 8 but declined rapidly and have typically fluctuated between 6.5 and 7.5. There is no indication for the 12 continuing tests that pHs are on a declining trend. In a few cases, pHs have recovered slightly. At the same time, the alkalinity trend is stable.

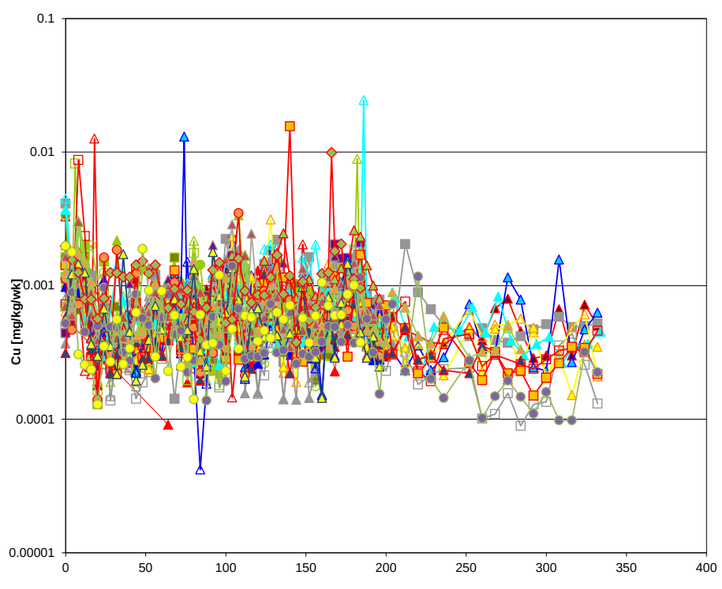
Sulfate leaching rates have been low throughout the program with most continuing tests showing rates below 1 mg/kg/week. At these low rates, variability in trends is apparent but no clear upward trends have been apparent.

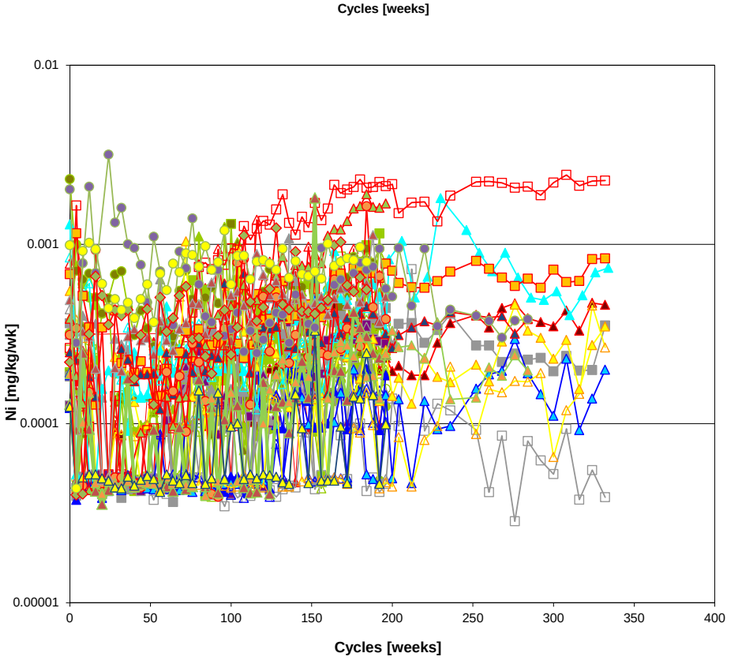
Nickel showed slight increases in leaching rates relative to other tests in the few cells in which pH decreases and peak nickel leaching rates were observed in cell 99-320C(830-850). This sample also showed the highest sulfate and cobalt leaching rates. Copper generally showed stable leaching rates.

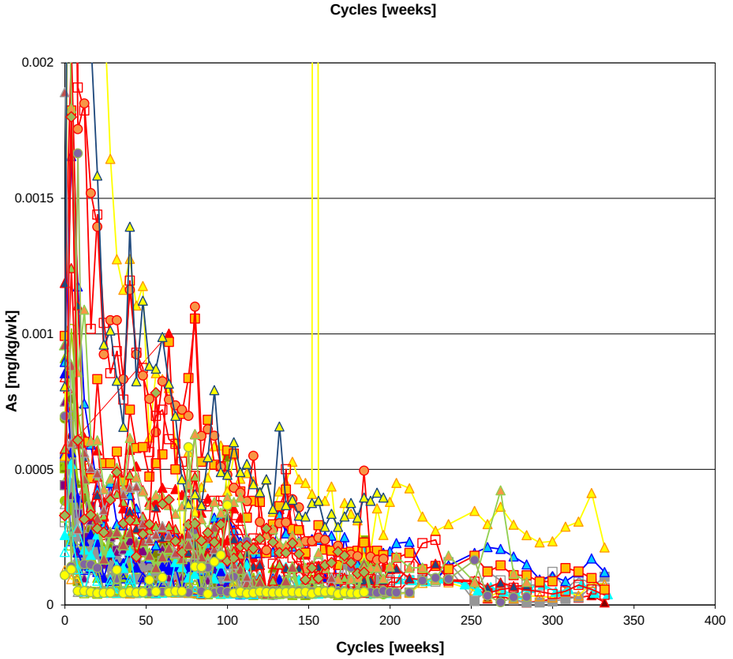
Arsenic showed no increase in leaching rates due to pH changes. Arsenic leaching has either stabilized or decreased steadily.











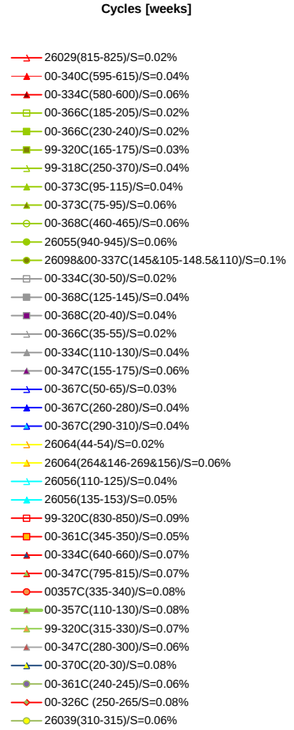
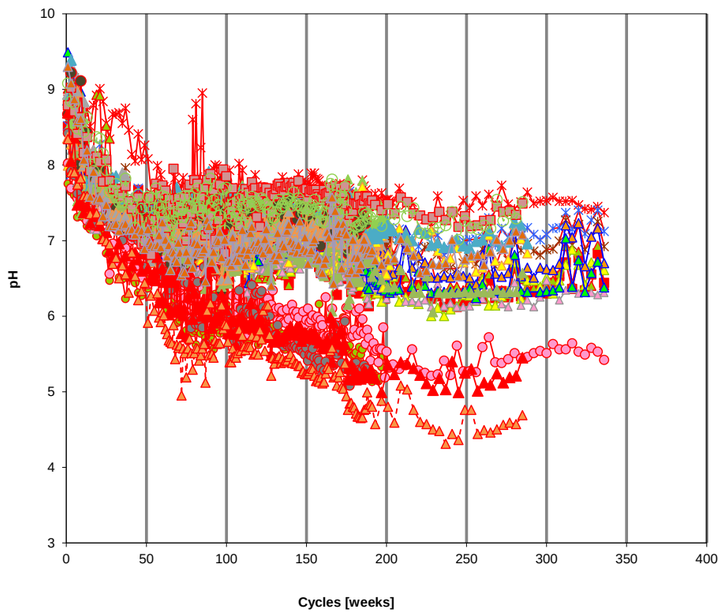
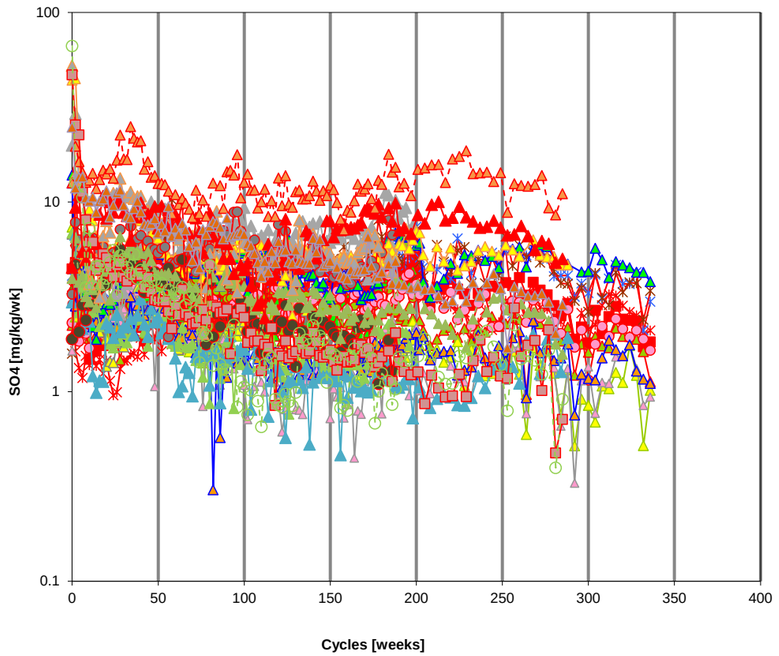
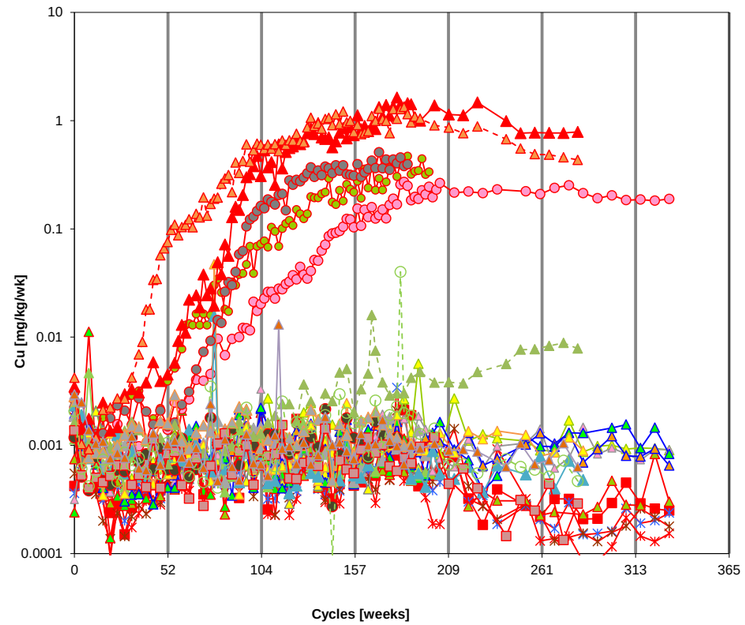


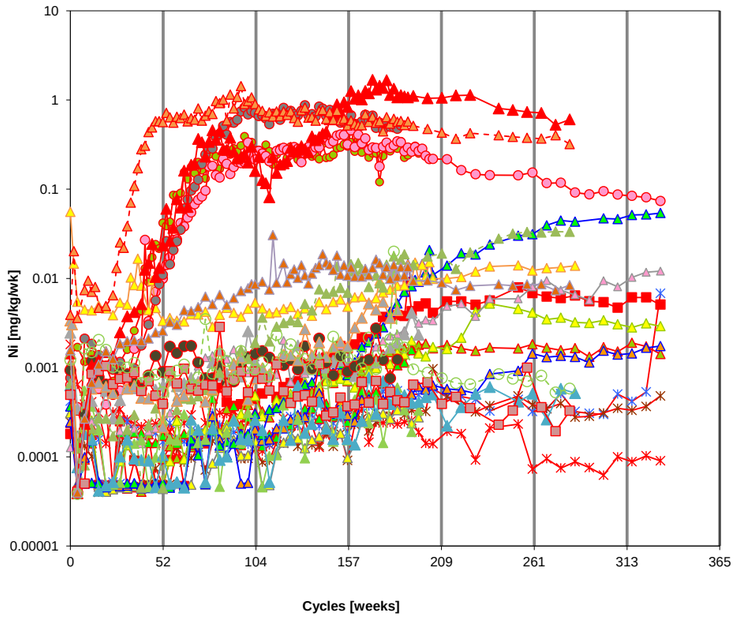
Figure 1. Leaching Trends for Category 1 Samples

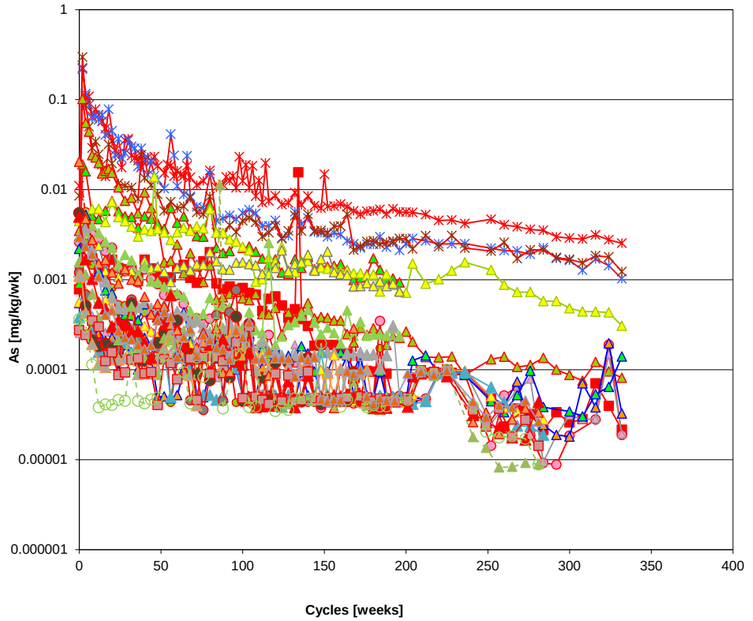
J:\PolyMet Mining\1UP005.01_Northmet_project_2004\Testwork\WR_KT\Results\Calculations\[Loadings_WR_Type1.xlsx]











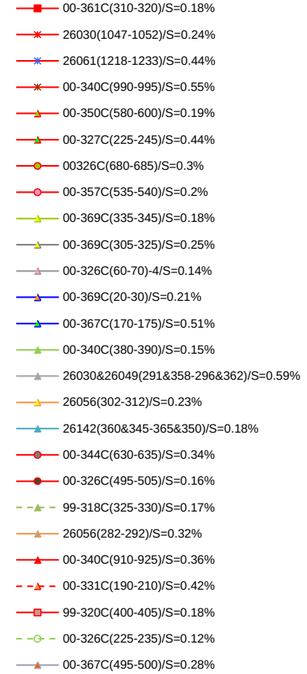


Figure 2. Leaching Trends for Category 2/3 Samples

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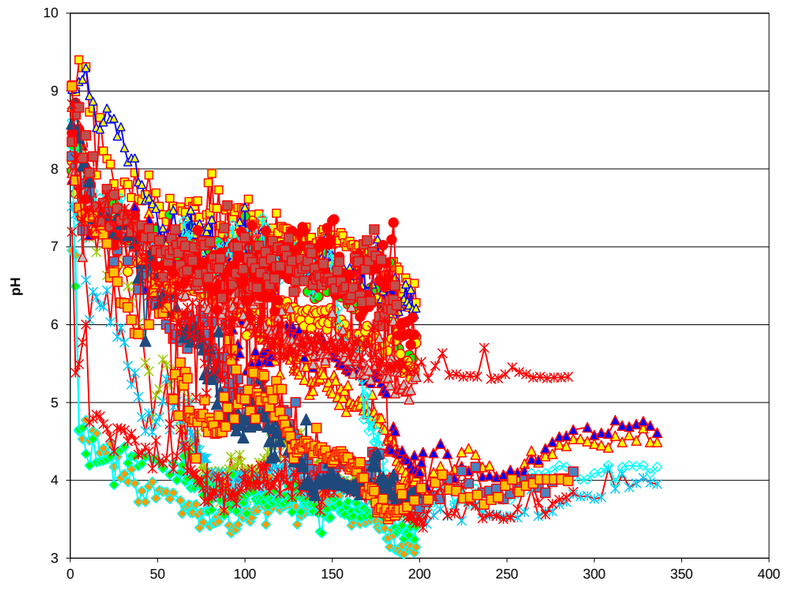

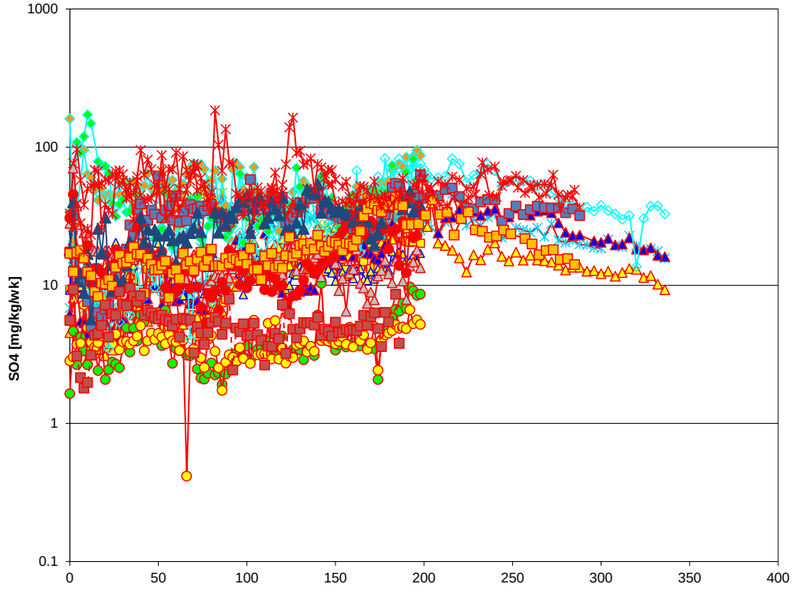

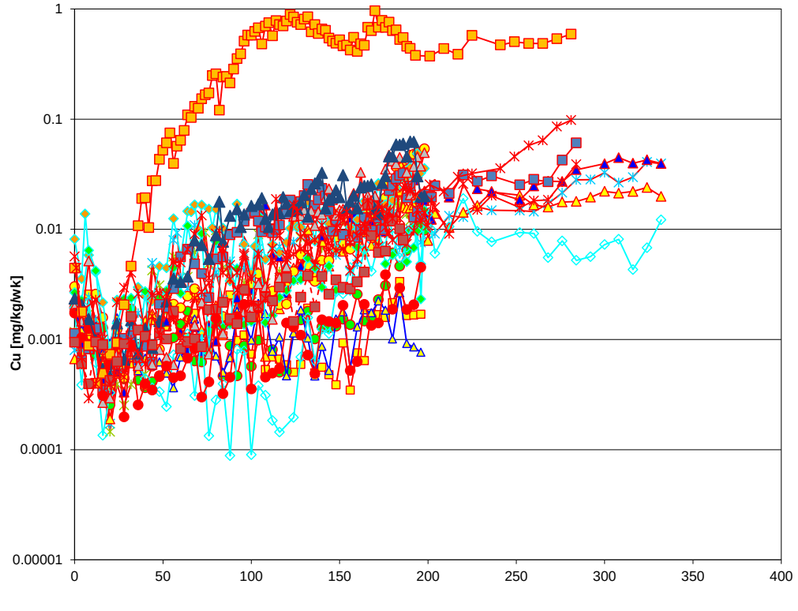

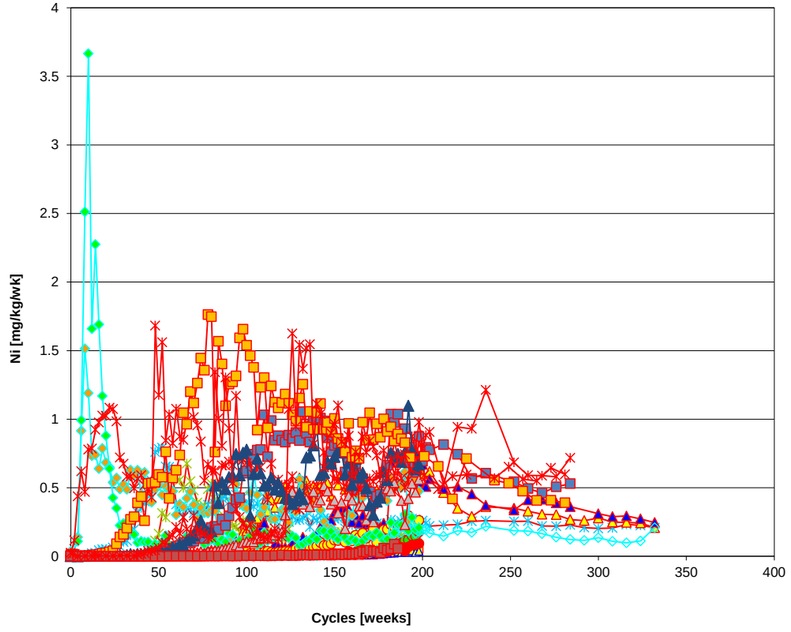

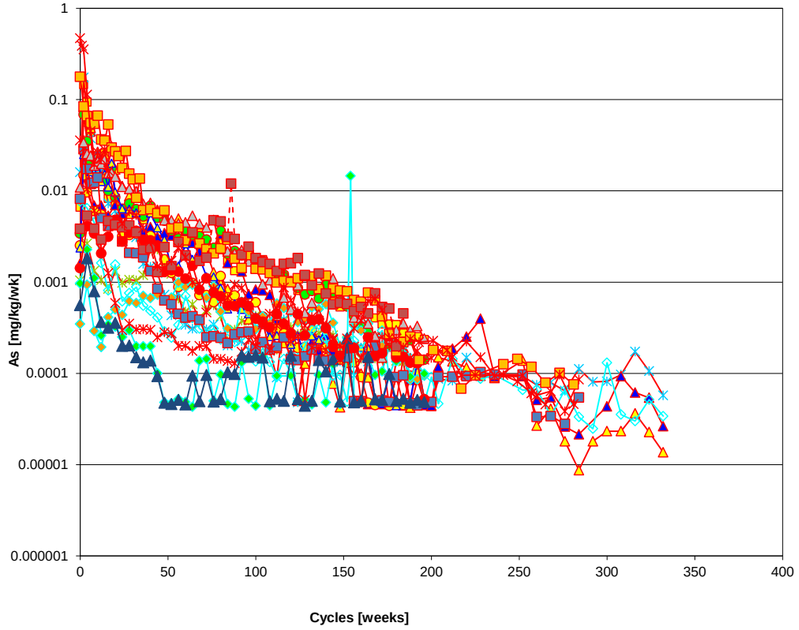

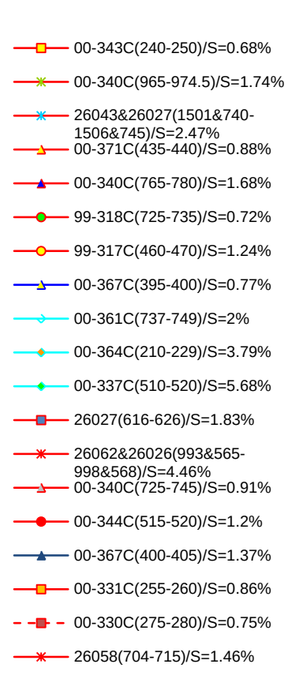

Figure 3. Leaching Trends for Category 4 Samples

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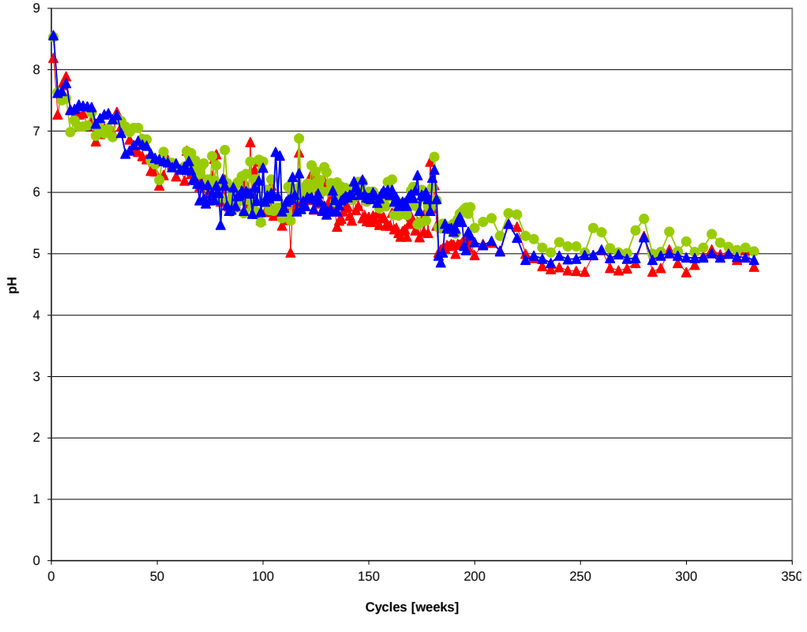

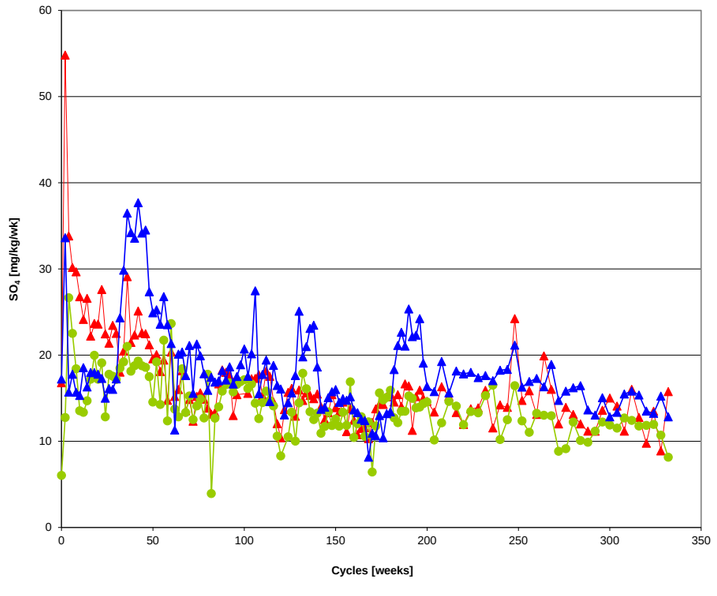

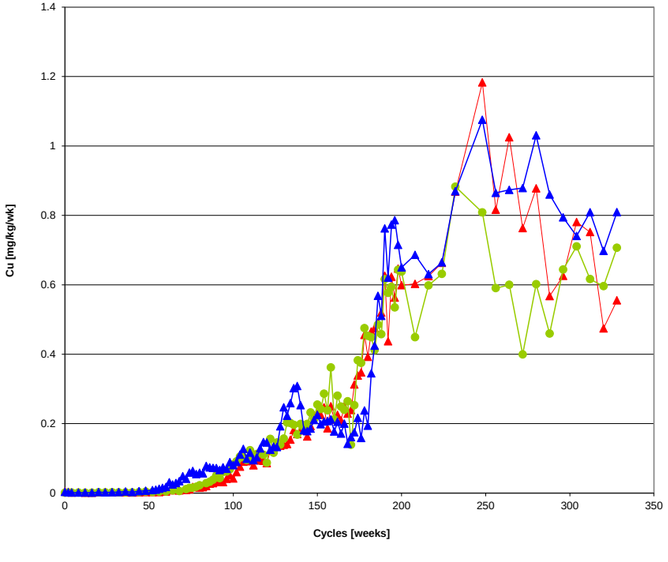

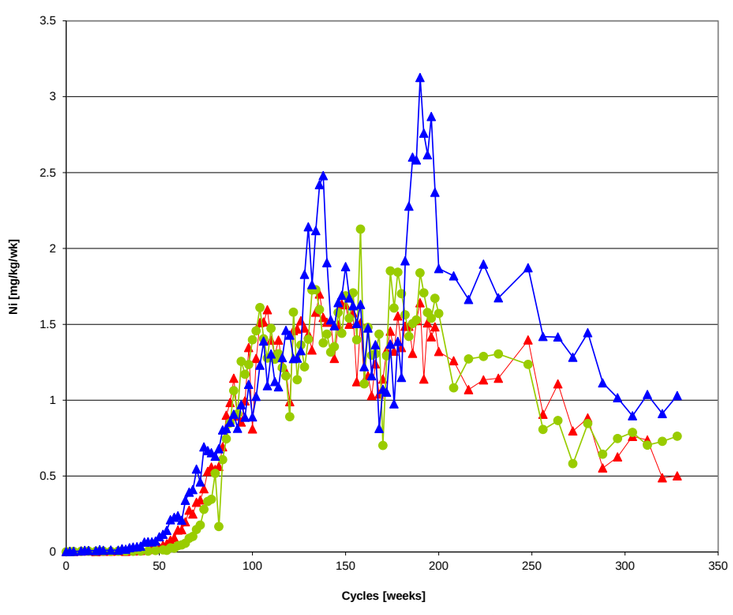

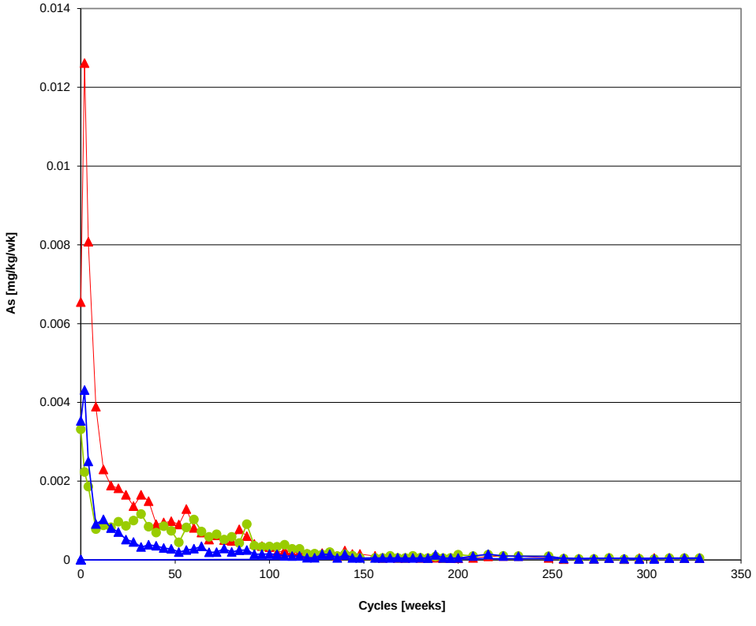

Figure 4. Leaching Trends for Ore Composite Samples

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Testing on Category 1 samples has confirmed to date that alkalinity generated by weathering of silicate minerals is able to result in pH above the pH of de-ionized water. This indicates that the alkalinity is consuming acidity introduced by both the dissolved CO_2, introduced by the deionized water (pH 5 to 6), and acidity produced by oxidation of sulfide minerals (locally pH<4). The steady maintenance of pHs above 6 shows that this is a steady long term mechanism for preventing pH depression below 6 in humidity cells.

3.2.2 Category 2/3

Results for pH, sulfate, nickel, copper and arsenic are shown in Figure 2.

Five samples (including three of the continuing 19 samples) showed pH depression below 6 while all other samples have shown stable pHs above 6 with similar trends to the Category 1 samples. Only one of the six samples has shown pH below 5. The sulfur content of the five samples varied from 0.2% to 0.42%. The sample with 0.2% sulfur showed the highest pH in this set at near 5.5. The sulfur contents of the other four samples were 0.3% and higher.

Sulfate release for Category 2/3 samples varied between 1 and 10 mg/kg/week with the highest rates being observed for samples containing higher sulfur contents. While sulfate release has varied, there was a lack of consistent upward trends.

A number of samples showed upwards trends in nickel and cobalt were related to pH decrease below 7. For the five samples showing distinctive pH depression, nickel release increased steeply during the first year of testing and subsequently showed gentle decline. Other samples showed slower nickel increase later in the program due to slower decline in pH. The same five samples also showed increase in copper leaching though the increase was less rapid than nickel and decline occurred later or not at all. One other sample (99-318C-325-330) showed accelerated copper leaching. Leachate pH was at the low end of those samples not showing substantial pH depression. It is a lean ore sample containing 0.1% copper.

Arsenic leaching trends were similar to Category 1. Rates have trended downwards. Higher rates are associated with samples yielding higher leachate pHs.

3.2.3 Category 4

Results for pH, sulfate, nickel, copper and arsenic are shown in Figure 3. Eight tests are continuing, two of which are Duluth Complex troctolite samples (00-371C(435-440) and 00-340C(765-780)), two are Duluth Complex Anorthositic Troctolites (00-331C (255-260) and 26027(616-626)), three are sedimentary hornfels samples (26058 (704-715), 26043&26027(740&1501-745&1506)) and DDH-26062&26026(993&565-998&568)-68) and one is Virginia Formation (00-361C(737-749)).

All tests have shown pH depression to some degree with pHs below 4. Of the continuing tests, one sedimentary hornfels sample (26058-704-715) showed the least depression with stable pHs near 5.3. The other samples showed leachate pHs near or below 4 and have recently shown pH recovery from minimum values. The two troctolite samples have shown pHs above 4.5 following a steep decline to pH near 4 earlier in the program. The other two Duluth Complex samples, Virginia Formation sample and other sedimentary hornfels samples have shown weaker pH recovery. Stabilization at pHs between 4 and 5 are indicative of buffering by reaction of acidity with alumino-silicate minerals resulting in formation of secondary aluminum minerals.

Some increases in sulfate release have been observed as pH decreased. Peak sulfate release rates have typically been less than a factor of five times lowest levels earlier in the test and peaks are not sharp but erratic. Decline in sulfate release has been observed following the peak. These declines are rapid at first then lessen as shown by straight lines on the log axis plot.

Due to the consistent pH depression below 7, all samples showed accelerated nickel leaching followed by declining leaching after reaching a subdued peak. All continuing tests have shown this decline parallels the sulfate release trend implying that nickel release is derived from sulfide minerals. Cobalt showed similar trends.

Upward trends in copper release were also apparent for all tests but as observed for acidifying Category 2/3 samples, the increase in copper release occurred later than nickel. Also, peaks followed by downward trends in copper release were less apparent for nickel and most ongoing tests have shown increase in copper release rather than a decrease paralleling nickel and sulfate release.

As with all other tests, arsenic release followed a declining trend.

3.2.4 Ore Composites

Results for pH, sulfate, nickel, copper and arsenic are shown in Figure 4.

Leachate pH trends for the three samples are nearly identical and show pH has been stable near 5 for over 2 years. Likewise, sulfate release has been similar for all three tests and has not shown consistent upward or downward trends. As pH decreased, nickel release accelerated, reaching peak rates after about three years then declining. Composite P30 showed a marked separate pH after 4 years that was not apparent in the other tests. A sulfate release peak was observed about the same time though was less pronounced compared to the other two tests.

Copper release also increased as the tests progressed but the upward trend was delayed compared to nickel and reached peak values after more than 4 years of testing. Copper release has subsequently declined.

3.3 Comparison Tests

Five samples tested as four size fractions continue to be tested in parallel with the humidity cell program. These tests are comparing differences in the weathering characteristics of the samples as well as differences in testing configuration (ASTM Humidity Cell and DNR Reactor) Samples are classified as Category 1 (two samples), Category 2/3 (two samples) and Category 4 (1 samples).

Both Category 1 samples have yielded leachate pHs near 7 in all size fractions and flat sulfate release trends. Sulfate release continues to be a function of particle size with lowest release (when detected) coming from the coarser size fractions. Similar findings apply to metal release. Nickel release is greater for the finer size fractions.

Consistent with the conventional humidity cells for Category 2/3 samples, lower pHs were observed but due to the sulfur content near 0.2% for both samples, pHs have remained above 6 and the samples continue to generate alkalinity particularly from the finer size fractions. Sulfate release was variable for both samples and no trends were apparent. Difference between sulfate releases in size fractions was most apparent for one sample (00-361C-310-320) with the fine fractions releasing higher sulfate than the coarse fraction. Both samples showed accelerated nickel leaching from all fractions as pH decreased below 7. Differences between size fractions were strongest for 00-361C-310-320. Temporal trends in copper release were not apparent but copper release rates were greater for the finer size fractions.

Virginia Formation rock classified as Category 4 showed sharp decline in pH to near and below 3.5 in all size fractions. Peak sulfate release was observed around 4 years for the two fine fractions. Peak sulfate release rates for the coarsest fractions have not observed because rates began increasing after six years. These differences in timing were not linked to greater pH depression for the fine fractions, in fact, the finer fractions yielded higher pHs.

These tests have shown that:

- Testing of different size fractions to date has shown that samples generating acidic leachate did so regardless of size fraction tested and protocol used.
- Rates of sulfate and metal release are generally higher for the finer size fractions presumably reflecting differences in surface area.
- For one sample that became strongly acidic, the finer fractions showed more rapid sulfate release earlier than the coarse fractions presumably reflecting faster oxidation and greater liberation of sulfide minerals.

3.4 Duplicate Testwork

Following observation of good reproducibility of trends previously (SRK 2009), all seven standard humidity cell tests being run as duplicates were stopped as part of reduction of the program in consultation with the DNR. Duplicate DNR reactor tests on two samples, and duplicate humidity cells containing size fractions for two samples have continued for a total of four duplicates (eight tests).

The two humidity cell-style duplicates are continuing to show strong reproducibility for the major ions and trace elements of interest (arsenic cobalt, copper, nickel). Weak reproducibility was apparent for barium (one sample).

The DNR-style reactors showed good reproducibility for one sample and poor reproducibility of pH for the other sample. The difference in pH (near 6 compared to 6.5) appeared to be due to differences in oxidation rate (faster for lower pH) which also resulted in greater leaching rates for cobalt and nickel.

3.5 Interpretation

3.5.1 Effect on Sulfur Management Criteria

Figure 5 shows minimum leachate pH as a function of total sulfur content for all humidity cell tests. Lowest pHs are observed for samples containing highest sulfur concentrations (Virginia Formation and Sedimentary Hornfels). The lowest sulfur concentration showing pH depression below 5 (i.e. below the range of pH of deionized water used in the test) was 0.36%. Samples with sulfur content below the Category 1 threshold of 0.12% have shown leachate pH above 6 indicating the threshold is appropriate based on current data.

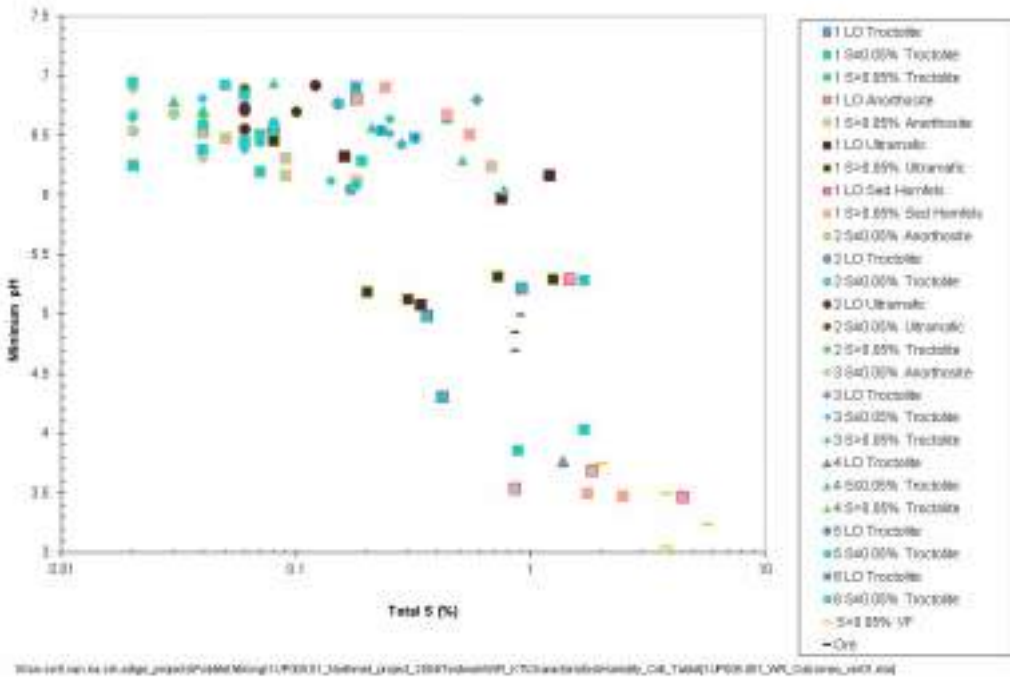

Figure 5. Minimum Leachate pH as a Function of Total Sulfur Content

3.5.2 Effect on Dissolution Rates Used to Estimate Water Quality

The condition definitions shown in Table 3 were used to assess trends for each test.

An example for one test showing all four conditions is provided in Figure 6. Condition 1 lasted for 39 weeks and was marked higher pHs and relatively stable SO_4 following an initial flush in the first few weeks. As pHs declined below 7, sulfate in this case become more variable and leaching of nickel accelerated. Condition 3 began at week 164 and was marked by steep decline in pH to 3.5 and rapidly increasing sulfate release paralleled by nickel. Condition 4 began at week 194 when sulfate release peaked and then began to declining.

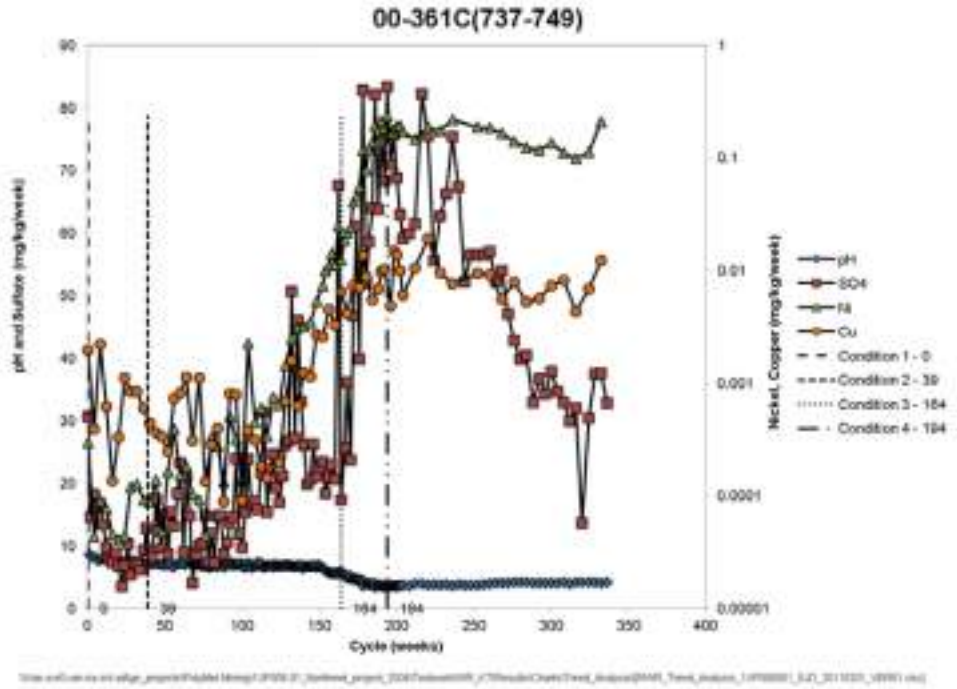

Figure 6. Example of Condition Assignment for a Waste Rock Humidity Cell Test

Resulting rates for each condition have been provided to Barr Engineering and include data collected to late January and early February 2012. These may be used as the basis for various inputs to source term predictions for the water quality model. However, the updated rate table primarily adds a few rates for samples that transitioned to acidic conditions. The majority of samples leaching under non-acidic conditions continue to show stable rates that have not changed significantly. As a result of this finding, there is likely little incremental value in modifying the rates used as inputs to the water quality modeling.

4 Tailings Kinetic Tests

4.1 Status of Program

A review of the tailings program was provided recently (SRK 2011). All tests have continued since findings were reported resulting in about six months of additional data. As this represents less than 15% additional time for most tests (with the exception of tests started in 2009), this memorandum provides a limited review to indicate whether any significant changes have occurred since the previous review.

4.2 Description of Results

4.2.1 NorthMet Project Tailings

Table 3 lists all tailings humidity cells. The test program consists of conventional humidity cells with parallel tests using the MDNR reactor configuration on bulk tailings and tailings size fractions.

Tests started in 2005 and 2006 (Pilot Plant 1) have continued to the same trends reported by SRK (2011) including stable or increasing pH, decreasing sulfate and stable or decreasing metal concentrations.

Tests started in 2008 (Pilot Plants 2 and 3) have reached stable pH with no indication of trends below pH 7 as shown in some samples produced by Pilot Plant 1. Sulfate release has stabilized for all tests. Nickel and cobalt release have not shown the same trends as Pilot Plant 1 samples but have shown stable concentrations in leachates below 0.002 mg/L and 0.0001 mg/L, respectively. PP2 +100 mesh showed an arsenic spike to 0.08 mg/L at week 168 following repair of the cell to address rapid drainage of water during the leach cycle.

Tests started in 2009 (scavenger tailings samples) have shown downwards trends in pH with lowest pHs typically above 7. At the same time, sulfate release has also trended downward and for some tests has stabilized. Nickel and cobalt release trended downward initially for all tests. Upwards trends for both elements were most apparent for samples Oct 1/09 09:00 (-100+200 mesh) which also showed some pHs below 7.

An arsenic spike resembling those for the 2008 samples discussed above was also apparent for two cells. These spikes were correlated with a large number of spikes in other parameters including aluminum, iron and silicon. It appears the cell malfunctioned perhaps allowing solids to be present in the leachates and thereby causing an increase in the dissolved fraction due to the presence of colloidal matter.

Table 4: Tailings Humidity Cells Used as Basis for Update Report

HCT ID	Fraction	HCT Full ID	Total Sulfur (%)	Start Date	Total Duration weeks	Duration Used for Current Modeling[3] weeks
T1	Whole	P1 (CuSO4)	0.1	9/8/2005	333	--
T2	Whole	P1 (no CuSO4)	0.23	9/8/2005	333	--
T3	Whole	P2 (no CuSO4)	0.2	9/8/2005	333	--
T4	Whole	P3 (CuSO4)	0.15	9/8/2005	333	--
T5	100	Parcel 2 P2S +100 mesh	0.15	2/10/2006	311	--
T6	100	Parcel 2 P2S -100 +200 mesh	0.17	2/10/2006	311	--
T7	-200	Parcel 2 P2S -200 mesh	0.24	2/10/2006	311	--
T8	100	Parcel 1-2 PISCS +100 mesh	0.11	2/10/2006	311	271
T9	100	Parcel 1-2 PISCS -100 +200 mesh	0.1	2/10/2006	311	271
T10	-200	Parcel 1-2 PISCS -200 mesh	0.09	2/10/2006	311	271
T11	100	Parcel 3 P3S +100 mesh	0.11	2/10/2006	311	271
T12	100	Parcel 3 P3S -100 +200 mesh	0.14	2/10/2006	311	271
T13	-200	Parcel 3 P3S -200 mesh	0.14	2/10/2006	311	271
T52	Whole	Pilot Plant 2 (0.30 Cu feed) Composite Bulk Tailings (as rec'd)	0.07	7/8/2008	185	--
T53	100	Pilot Plant 2 (0.30 Cu feed) Composite (+100 mesh)	0.08	7/8/2008	185	146
T54	100	Pilot Plant 2 (0.30 Cu feed) Composite (-100 + 200 mesh)	0.06	7/8/2008	185	146
T55	-200	Pilot Plant 2 (0.30 Cu feed) Composite (-200 mesh)	0.09	7/8/2008	185	146
T56	Whole	Pilot Plant 3 (0.25 Cu feed) Composite Bulk Tailings (as rec'd)	0.08	7/8/2008	185	--
T57	100	Pilot Plant 3 (0.25 Cu feed) Composite (+100 mesh)	0.1	7/8/2008	185	146
T58	100	Pilot Plant 3 (0.25 Cu feed) Composite (-100 + 200 mesh)	0.08	7/8/2008	185	146
T59	-200	Pilot Plant 3 (0.25 Cu feed) Composite (-200 mesh)	0.08	7/8/2008	185	146
T60	Whole	SCAV TAILS 9/30/09 1600	0.09	11/24/2009	113	--
T61	100	SCAV TAILS 9/30/09 1600 +100	0.1	11/24/2009	113	74
T62	100	SCAV TAILS 9/30/09 1600 -100+200	0.09	11/24/2009	113	74
T63	-200	SCAV TAILS 9/30/09 1600 -200	0.11	11/24/2009	113	74
T64	Whole	SCAV TAILS 10/1/09 09.00	0.13	11/24/2009	113	--
T65	100	SCAV TAILS 10/1/09 09.00 +100	0.11	11/24/2009	113	74
T66	100	SCAV TAILS 10/1/09 09.00 -100+200	0.14	11/24/2009	113	74
T67	-200	SCAV TAILS 10/1/09 09.00 -200	0.14	11/24/2009	113	74
T68	Whole	SCAV TAILS 10/1/09 17.00	0.12	11/24/2009	113	--
T69	100	SCAV TAILS 10/1/09 17.00 +100	0.1	11/24/2009	113	74
T70	100	SCAV TAILS 10/1/09 17.00 -100+200	0.1	11/24/2009	113	74
T71	-200	SCAV TAILS 10/1/09 17.00 -200	0.13	11/24/2009	113	74

[3] Waste Characterization Data Package Version 9 (July 3 2012)

4.2.2 LTVSMC Tailings

Four samples of LTVSMC tailings are being tested in humidity cells. No significant changes in leachate chemistry have been observed since the previous update (SRK 2011). Leachate pHs have continued to vary in a narrow band between 7.4 and 8 with variable but non-trending alkalinity and sulfate.

Other parameters have continued to show no upward trend or downward trends. Isolated data spikes (for example, arsenic) are observed but do not represent significant trends.

4.3 Interpretation

Ongoing tailings testwork has shown no indication of development of pHs below that of the deionized water used to leach the samples (Figure 7) indicating that weathering of silicates minerals in the samples is continuing to generate alkalinity to offset acidity from sulfide oxidation and the deionized water even in the tests that have been operating for 7 years. The longer term tests yielded pH minima in their trends which are responsible for the lower pHs shown in Figure 7. Upward trends in pH are observed in most cases so that recent pHs are higher than the lowest values.

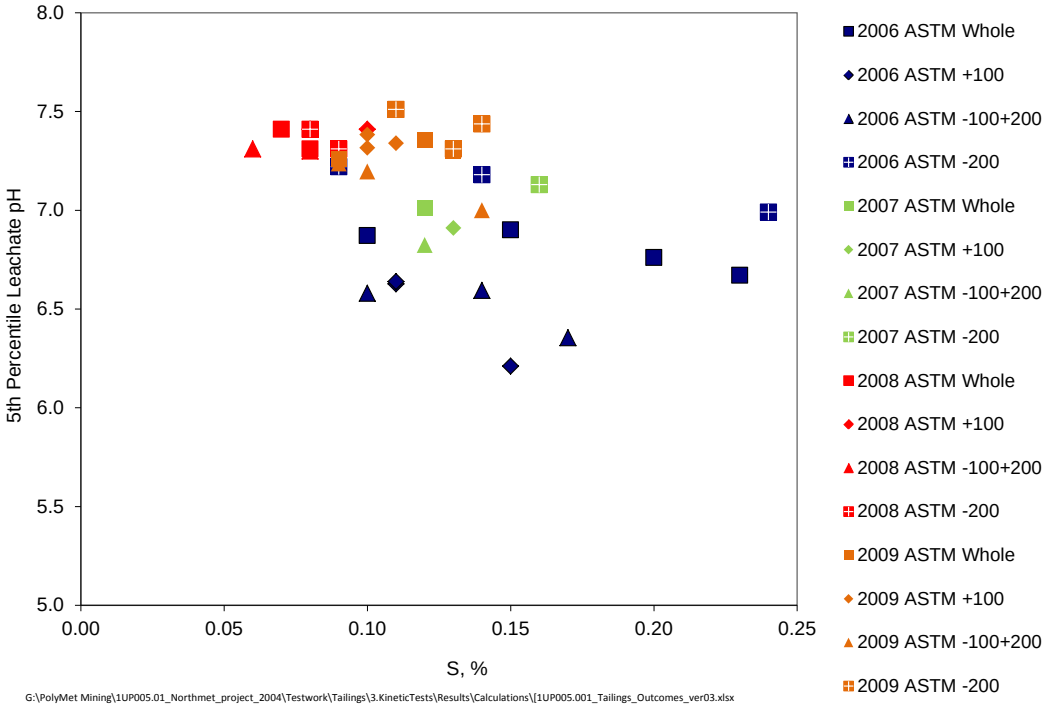

Figure 7. 5th Percentile pH as a Function of Total Sulfur Content for Tailings Humidity Cells

Correlations between initial total sulfur content and average release rates are apparent as have been observed previously (Figure 8). However, this figure suggests different relationships for the 2006 and 2007 datasets compared to the 2008 and 2009 datasets apparently implying that the earlier samples are less reactive. However, the differences are probably caused by significant depletion of sulfide as the tests have progressed with a correlated decrease in release rates, reducing the calculated average release rates. For example, between 30% and 78% of the initial sulfur has been depleted from the earlier tests compared to less than 30% for the more recent tests. For the oldest tests, remaining sulfur content varies from 0.04% to 0.12%. The general downward trend in sulfate release has resulted in progressively lower average sulfate release. The downward trend in sulfate release (and therefore acid generation rate) also provides a reasonable explanation for upward trends in pH.

The implication of results from the older tests is that the potential for long term pH depression below 5 is shown to be very low because silicate weathering was able to offset acid generation when sulfur content was higher than the current depleted levels.

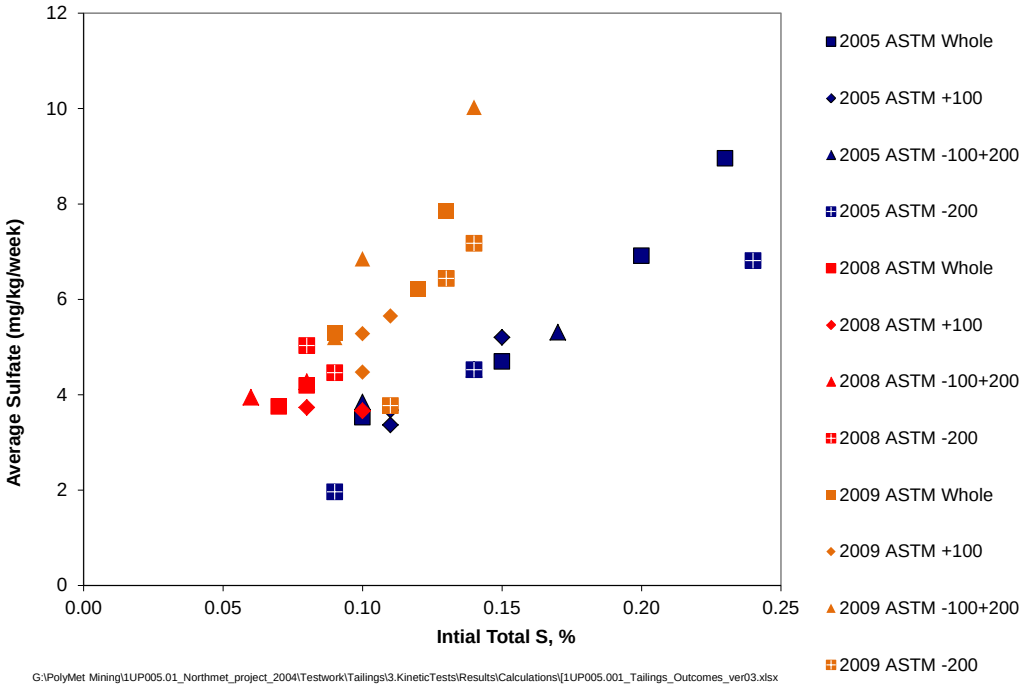

Figure 8. Average Sulfate Release as a Function of Initial Total Sulfur Content

The long term downward trend in sulfate release rates (Figure 9) indicates that average sulfate release rates for individual tests as used as input into the tailings water quality models are a function of time and will decrease as the test duration increases. To address this finding and provide sulfate release rates that are consistent with the assumed zero order reaction rate law for sulfide oxidation in the water quality model, relationships between sulfate release and sulfur depletion were examined to estimate sulfide oxidation rates at the initial sulfur content. The concept is to obtain the initial rates at t=0 by regressing depleted sulfur content against sulfate release rates to obtain equations of the form:

$$\frac{dM_{SO_4}}{dt} = a_1 M_S + a_0$$

Graphs of sulfate release rate as a function of sulfur remaining showed three different relationships. The most common relationship (15 tests) was for the slope of the equation to be steeper at the start than in the longer term. Other trends were (1) initially lower rates followed by a peak then long term decay (about 6 tests); and (2) consistent slope for the duration of the test (12 tests).

The first type of relationship is commonly observed in humidity cell results and is interpreted as initial flushing of stored sulfate oxidation products followed by sulfide oxidation. The initial part of the trend therefore does not represent sulfide oxidation and should not be included in the regression equations to obtain the oxidation rate at t=0. The preferred approach is to use the data for each test on its own merit to evaluate the break point when leaching of stored oxidation products is complete and the long term decay trend is well-established. LAM MDNR disagreed with this approach and required that the regression equations be developed excluding the first 5 weeks of data based on the following commentary in the ASTM humidity cell method D5744-96, (Section 11.2):

> "In the testing of mining wastes, cation and anion loadings are commonly high in the Week 0 leachate due to the dissolution of pre-existing soluble oxidation salts present in the sample prior to sample collection. The average number of weekly accelerated weathering cycles required to flush these pre-existing salts ranges from 3 to 5 weeks. Oxidation products observed during these 3 to 5 weeks are principally from pre-existing salts, while those

products observed after this period are considered to be solely a function of the accelerated weathering procedure."

SRK does not agree with the use of a fixed initial flushing period because it is unlikely that this flushing process can be generalized. In reality, the period required to fully flush stored oxidation products depends on the types, solubility and quantity of the oxidation products, and the physical and mineralogical characteristics of the samples. Tailings samples also contain entrained process water which is expected to be soaked into the finer particles leading to a protracted initial flushing trend. Nonetheless, to advance the process of acceptance, the LAM MDNR requested method was adopted to calculate oxidation rates at t=0.

Graphs for all tests showing regression relationships using the MDNR method are provided in Attachment A

Figure 9 shows initial sulfate release (determined from the regression equation using the initial sulfide content of the sample) as a function of initial total S content of the samples for comparison with Figure 8. Initial sulfate release calculated using the MDNR and SRK methods is shown. The difference in rates between tailings samples generated by different pilot plants implied by Figure 8 is reduced by both methods but the DNR method appears to result in lower rates for the 2005 samples and a weaker correlation with sulfur content than the SRK method which eliminates differences between the different datasets. The similarity of rates implies that the tailings generated at different times have similar reactivity. Correlation between initial sulfur content and initial sulfate release is apparent when considering the entire data range though weaker at lower initial sulfur contents.

For the purpose of water quality modeling, the distribution of sulfate release rates will be based on the initial sulfate rates (Figure 9) calculated using the MDNR method as agreed This method will result in the use of sulfide oxidation rates in the model that are nearly always higher than would actually be expected. Application of a single initial rate will result in predicted sulfate and trace element concentrations that are likely to be much higher than would really occur as sulfide content depletes over time. The approach is therefore conservative for predicting water quality.

For the purpose of modeling other parameters the following recommendations are made for rates:

- For parameters based on a solid ratio to sulfur content, rates should be calculated based on the ratios previously specified.
- For parameters based on average rate to average sulfate rate ratios (Ca, K, Mg, Na, Se), rates should be calculated based on the average rate ratio as previously specified. For the major ions, downward trends are apparent that parallel decrease in sulfate indicating that ratios of average rates are appropriate.

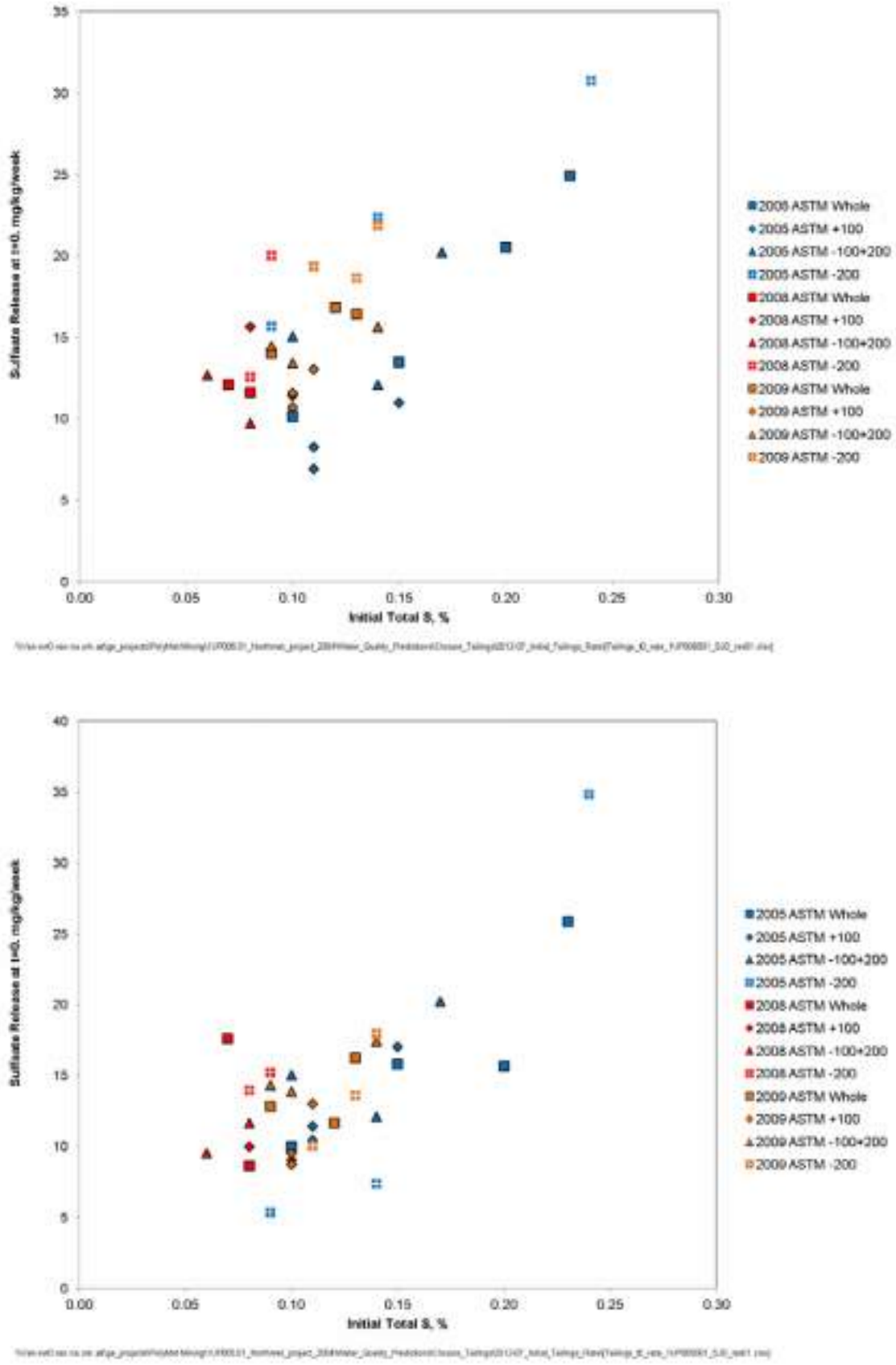

Figure 9. Calculated Initial Sulfate Release as a Function of Initial Total Sulfur Content Using MDNR (top) and SRK (bottom) methods.

5 Hydrometallurgical Residue Kinetic Tests

As agreed with MNDR, hydrometallurgical kinetic tests were stopped shortly after reported by SRK (2009).

6 Conclusions

Review of data generated by up to nearly eight years of testing indicates that:

- Waste rock classification into three categories using sulfur content to address potential for pH acidification appears to be robust. In particular, samples classified as Category 1 have not shown pH depression below that of the deionized water used to leach the samples.
- Sulfide oxidation and metal leaching trends for waste rock samples show well defined phases as pH depression occurs. These trends have been used to calculate release rates for use in development of source terms. Ongoing testwork has resulted in some additional rate information for later phases.
- Tailings humidity cells have not shown pH depression below that of the deionized water indicating that weathering of silicates provides alkalinity to offset acidity from sulfide oxidation.
- Long term trends in tailings humidity cells show declining oxidation rates which are presumably linked to depletion of the sulfide minerals and indicates that sulfide oxidation is following a non-zero order reaction rate law. However, to be consistent with the simplified modeling approach used in GoldSim®, which assumes a zero order reaction rate law, trends in sulfide oxidation rates have been used to back-calculate reaction rates at the initial sulfur content for use in modeling.
- Other than addressing this recommendation, updating the rates used as inputs to tailings water quality modeling is not recommended.

7 References

SRK Consulting. 2009. RS82 - Update on Use of Kinetic Test Data for Water Quality Predictions – DRAFT 02. Memorandum to Kim Lapakko, MDNR. February 7, 2009.

SRK Consulting. 2011. Update on Tailings Humidity Cell Test Data – NorthMet Project – Draft. Memorandum to Peter Hinck, Barr Engineering. December 23, 2011.

SRK Consulting (Canada) Inc.

Stephen Day, PGeo
Corporate Consultant (Geochemistry)

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The opinions expressed in this report have been based on the information available to SRK at the time of preparation. SRK has exercised all due care in reviewing information supplied by others for use on this project. Whilst SRK has compared key supplied data with expected values, the accuracy of the results and conclusions from the review are entirely reliant on the accuracy and completeness of the supplied data. SRK does not accept responsibility for any errors or omissions in the supplied information, except to the extent that SRK was hired to verify the data.

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Attachment A: Graphs

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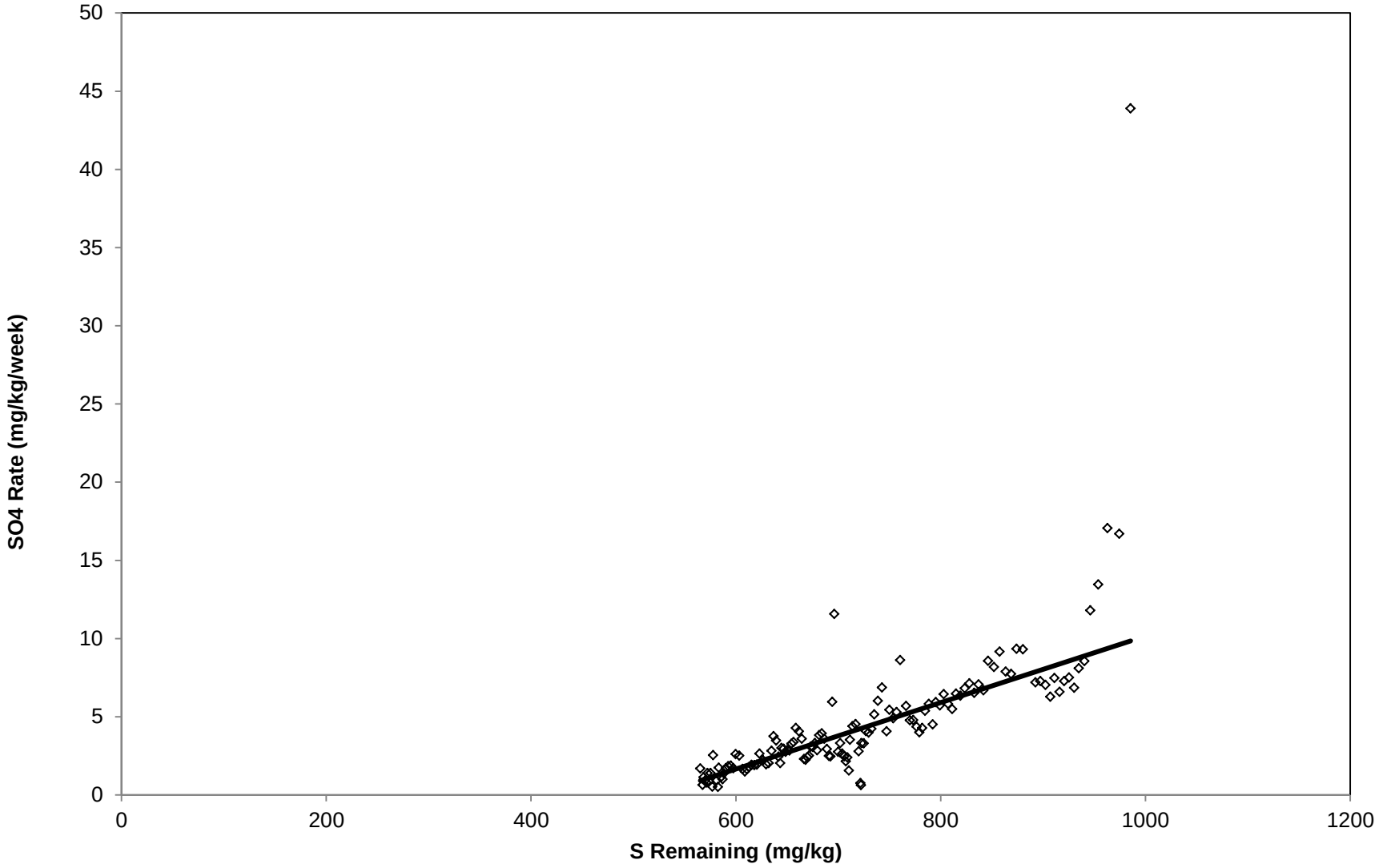

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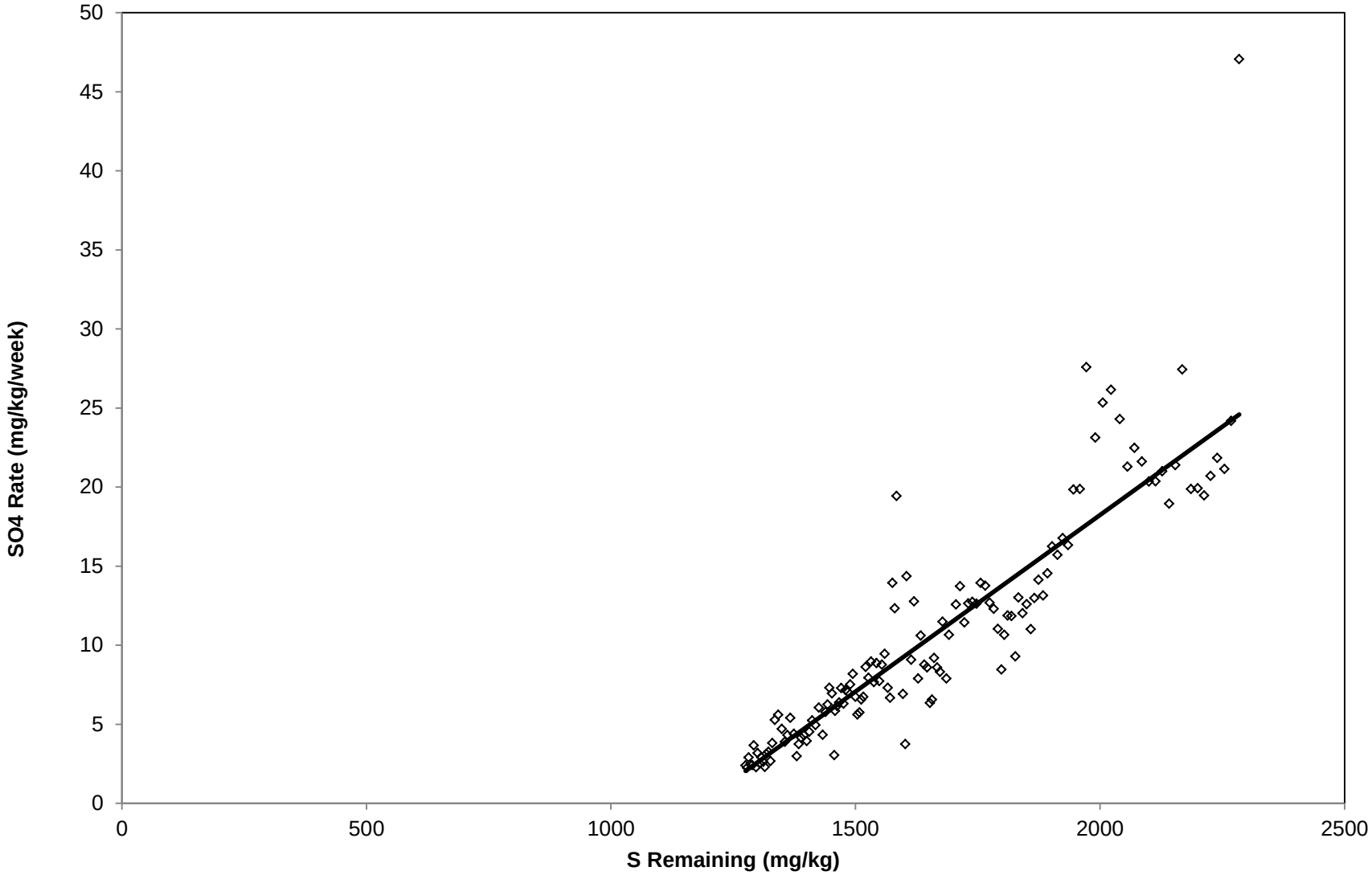

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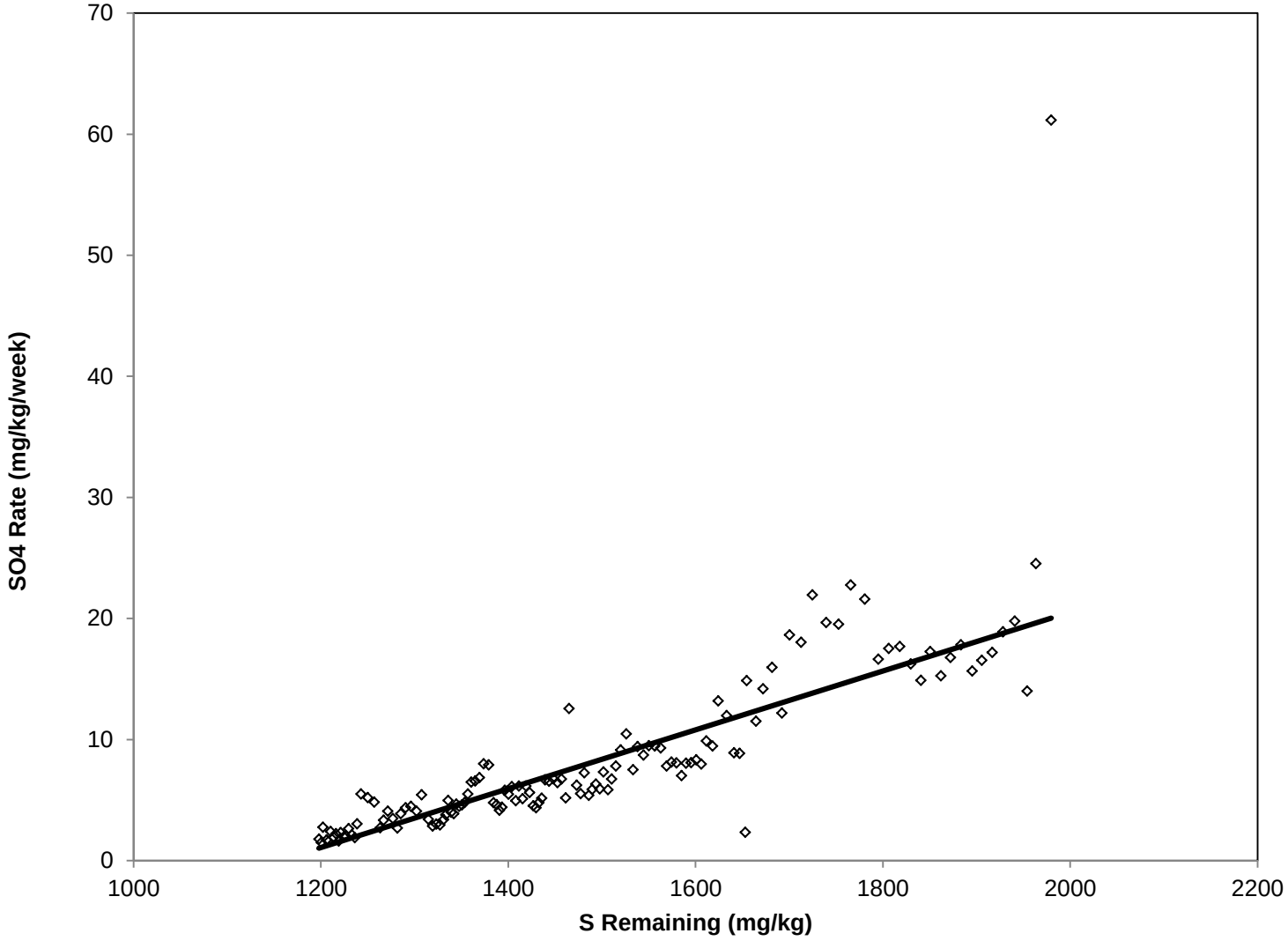

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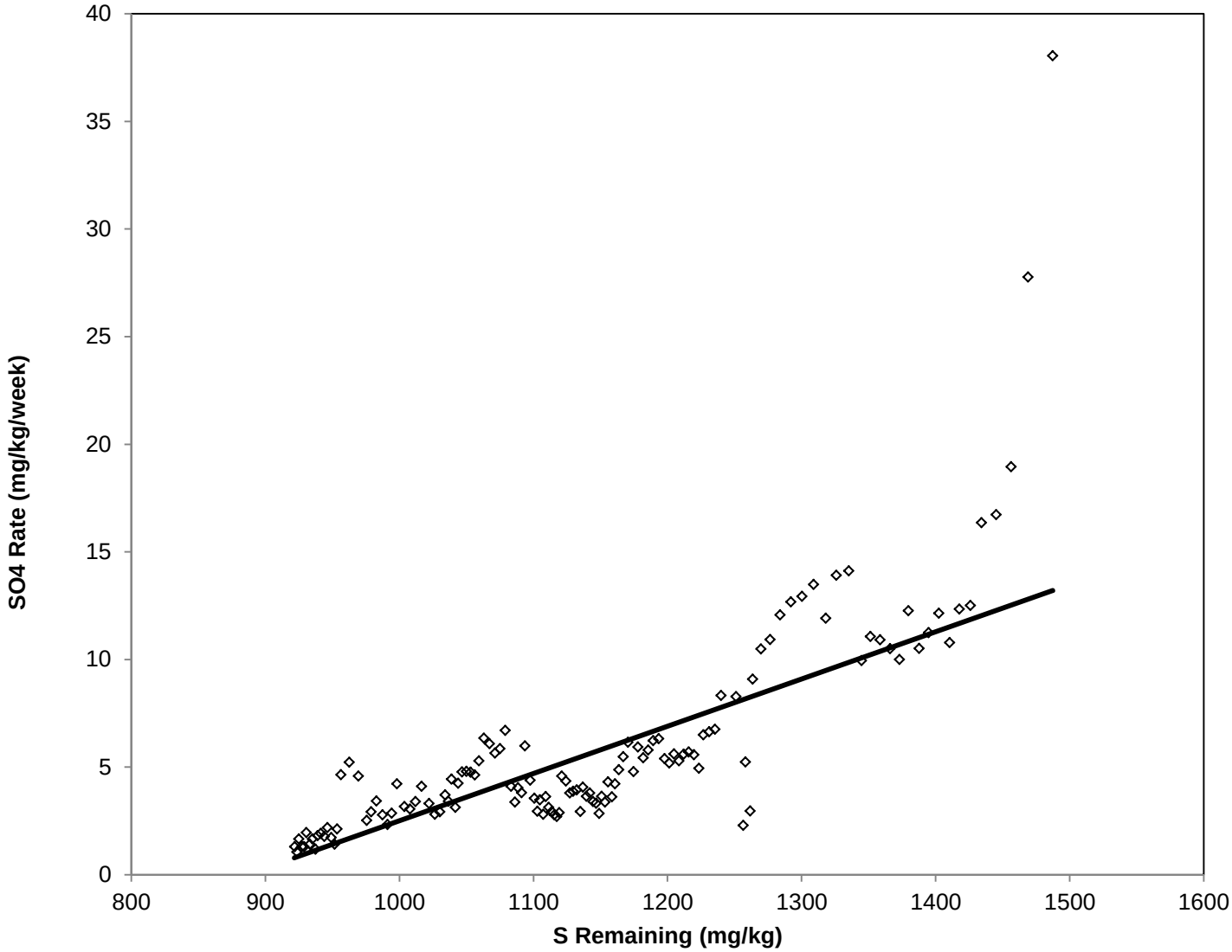

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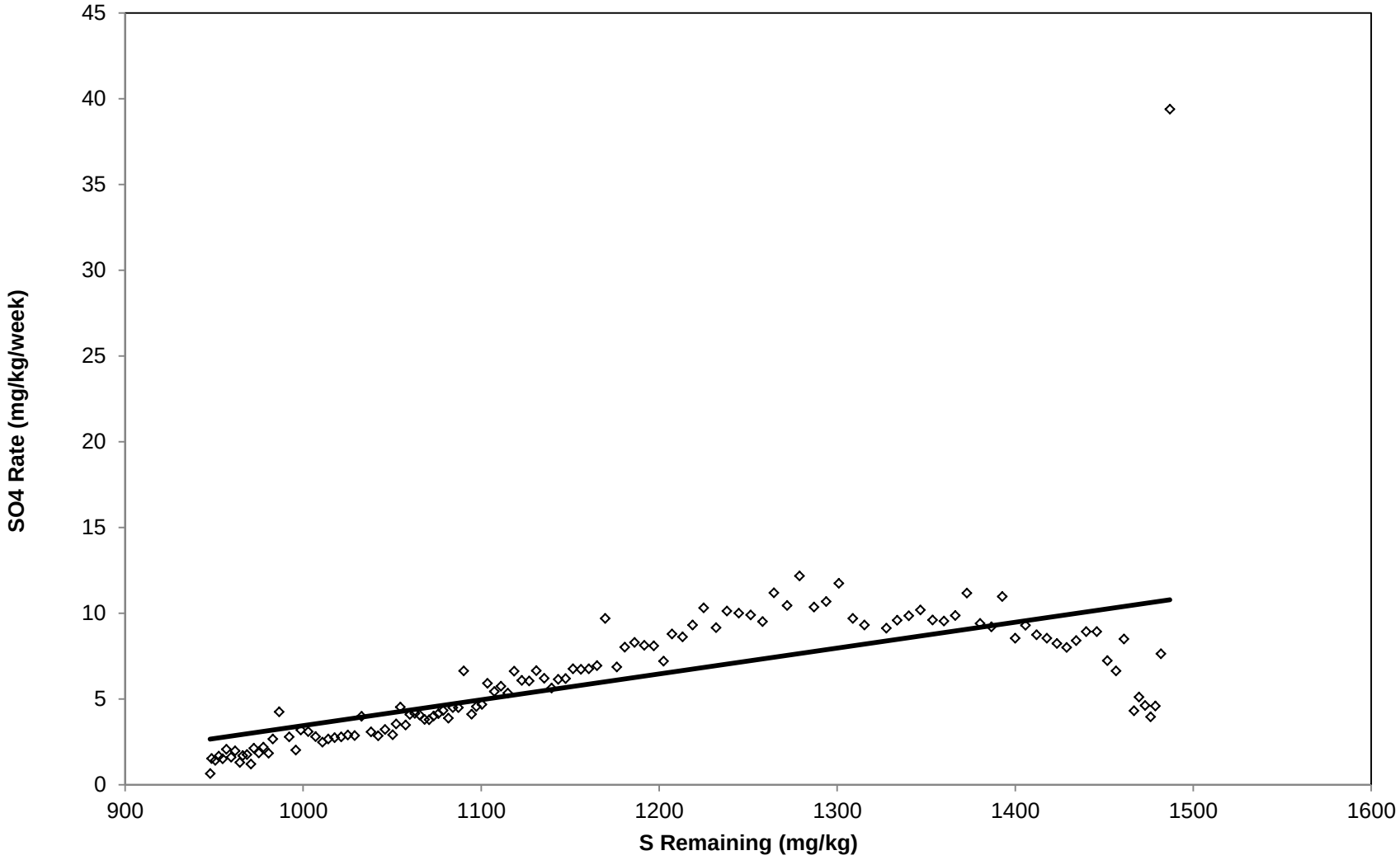

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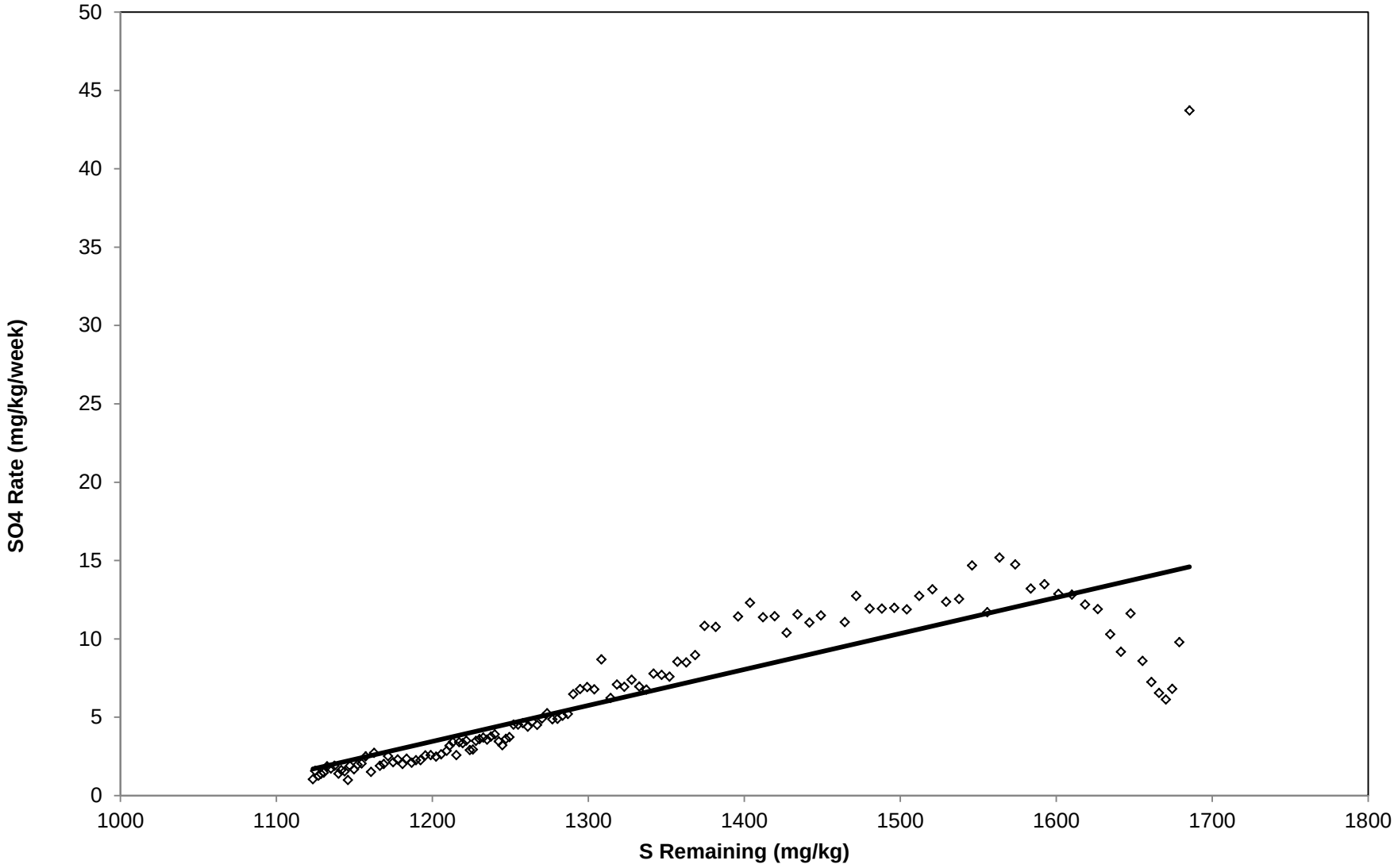

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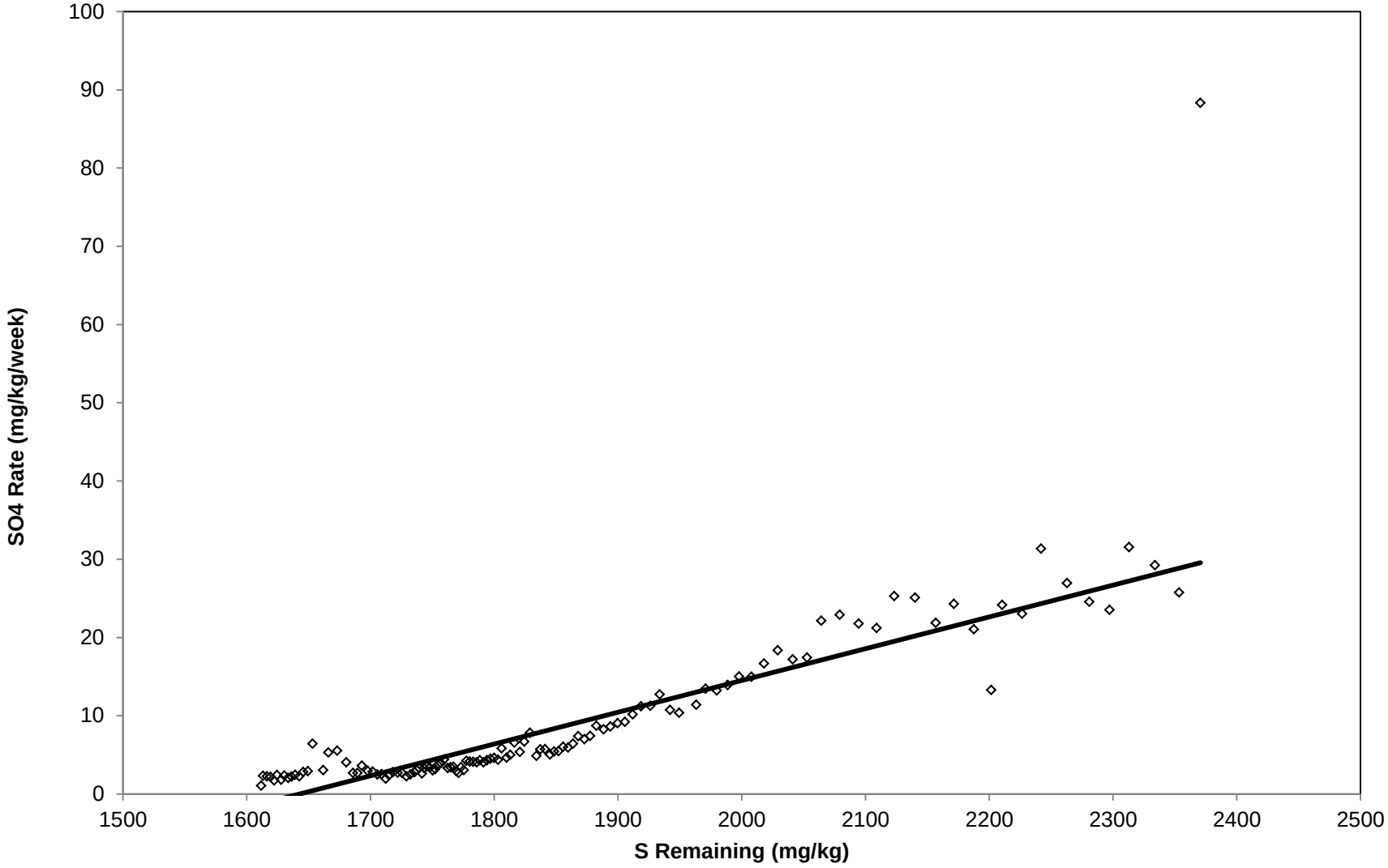

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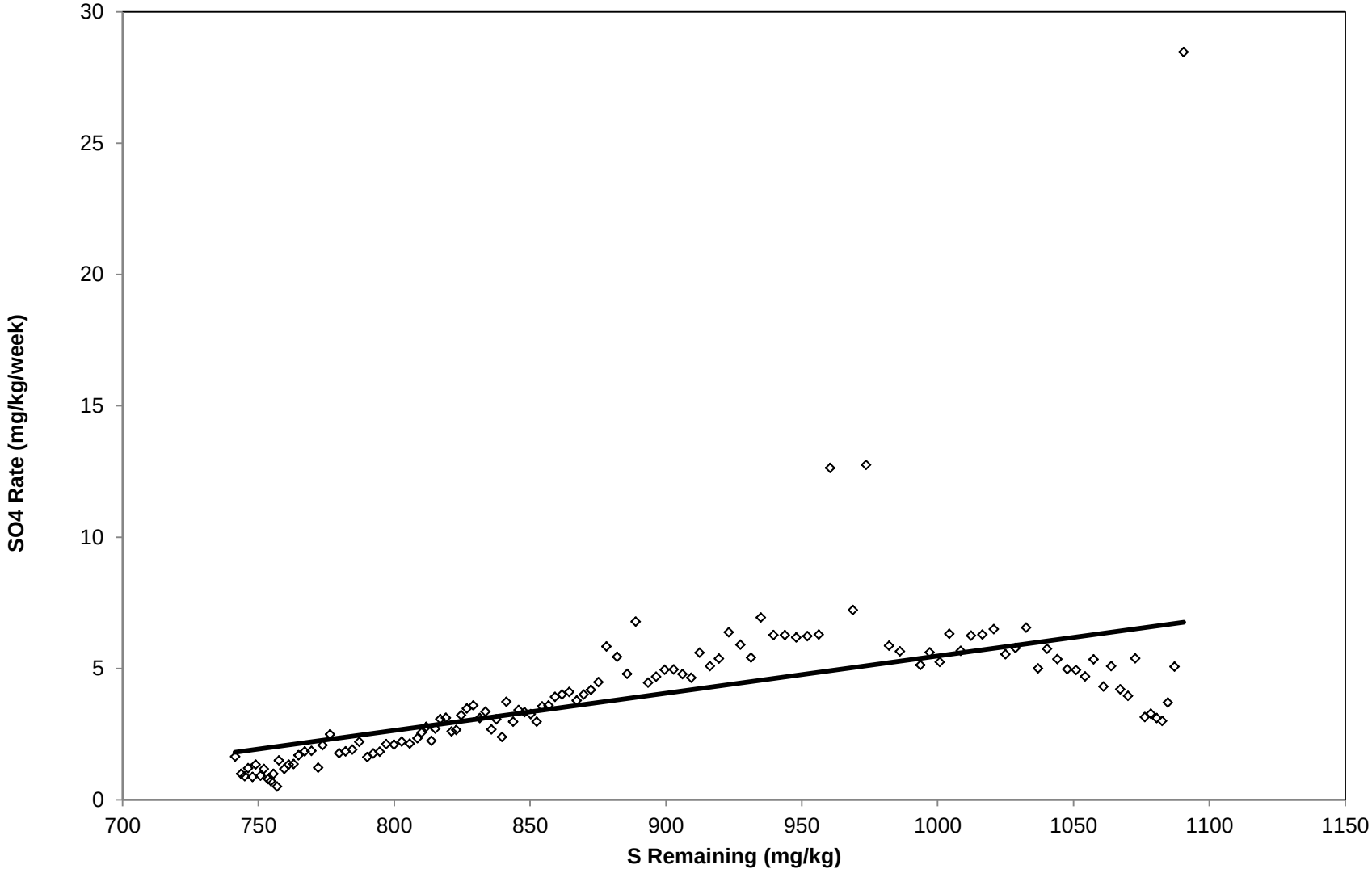

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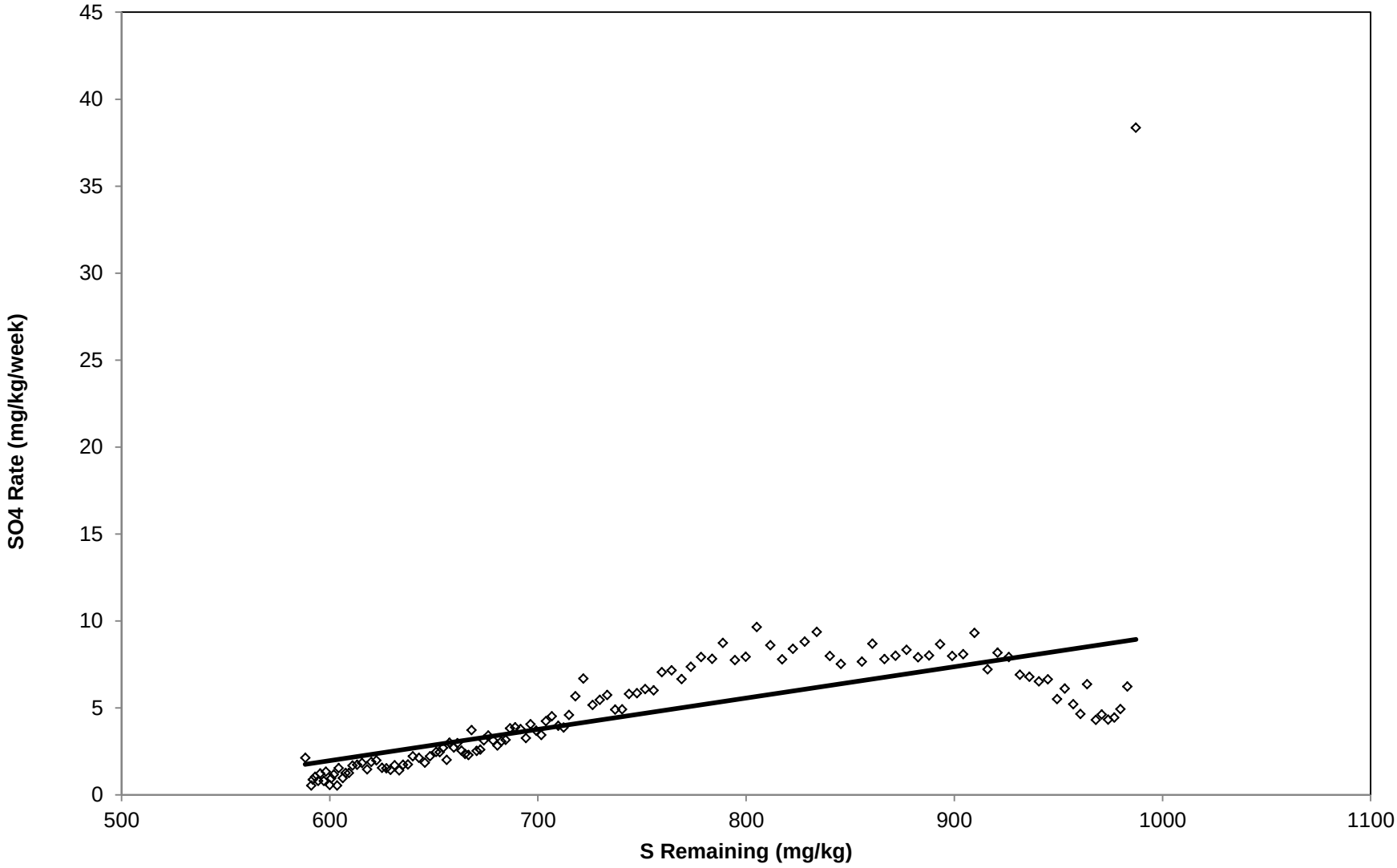

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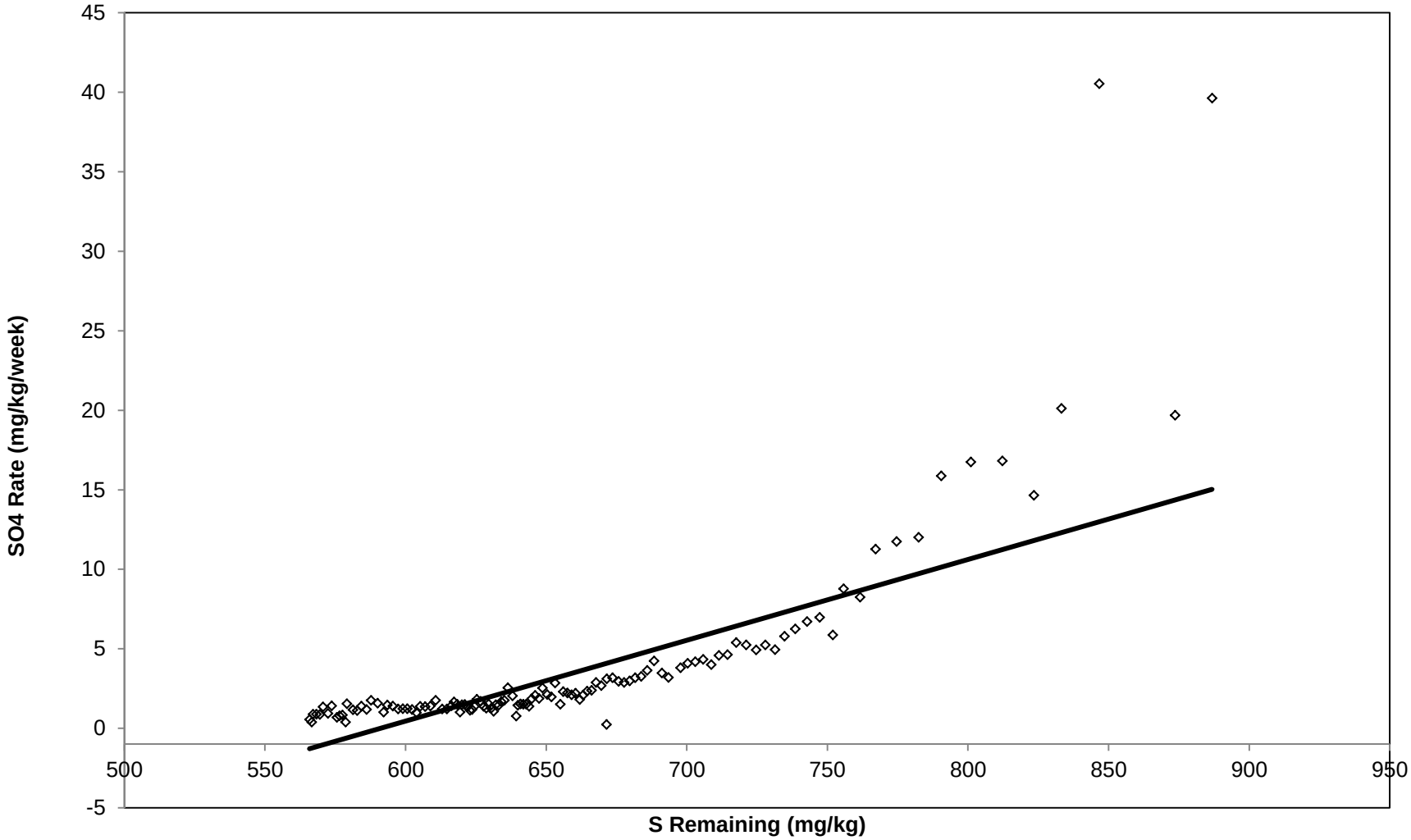

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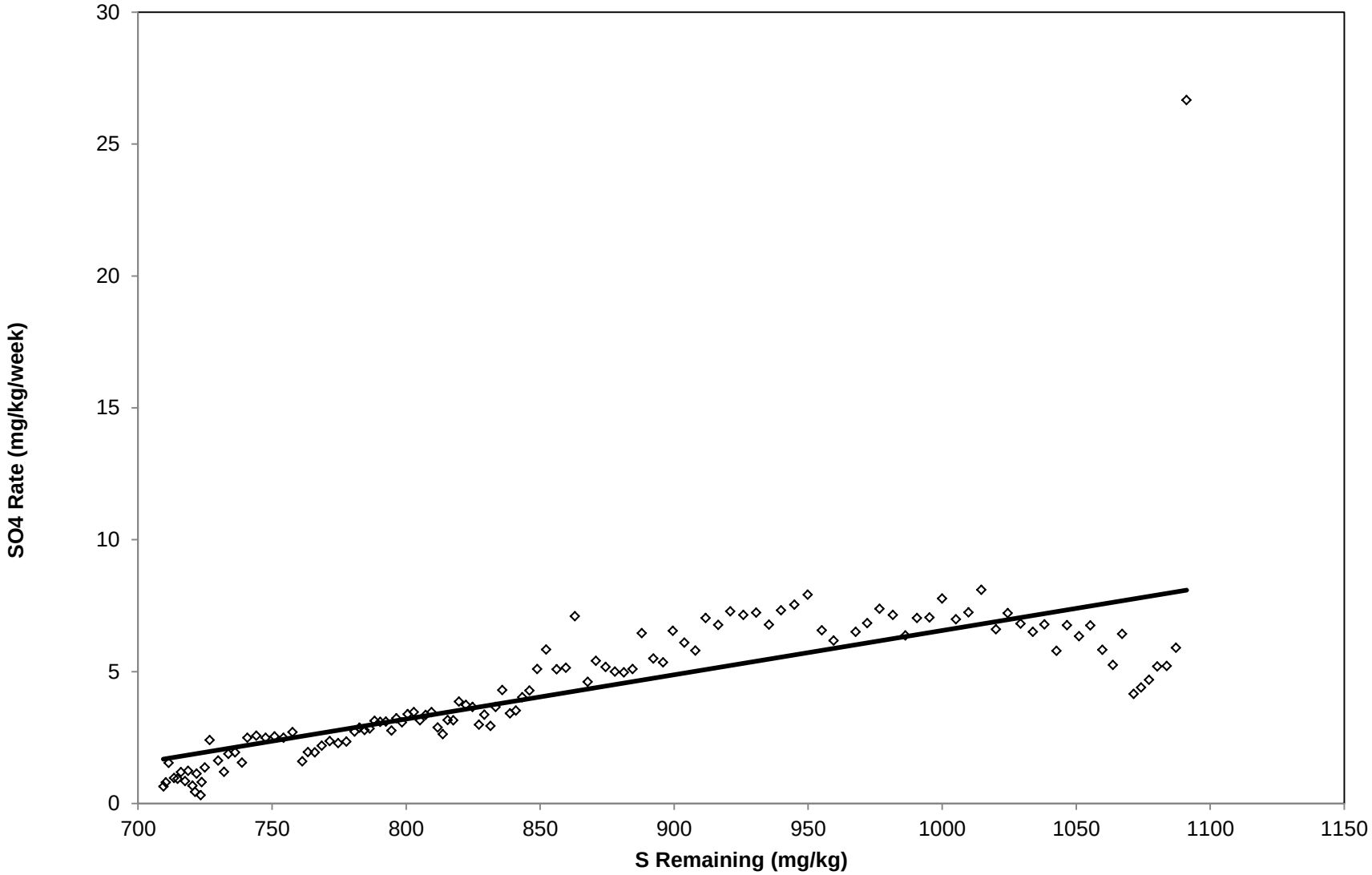

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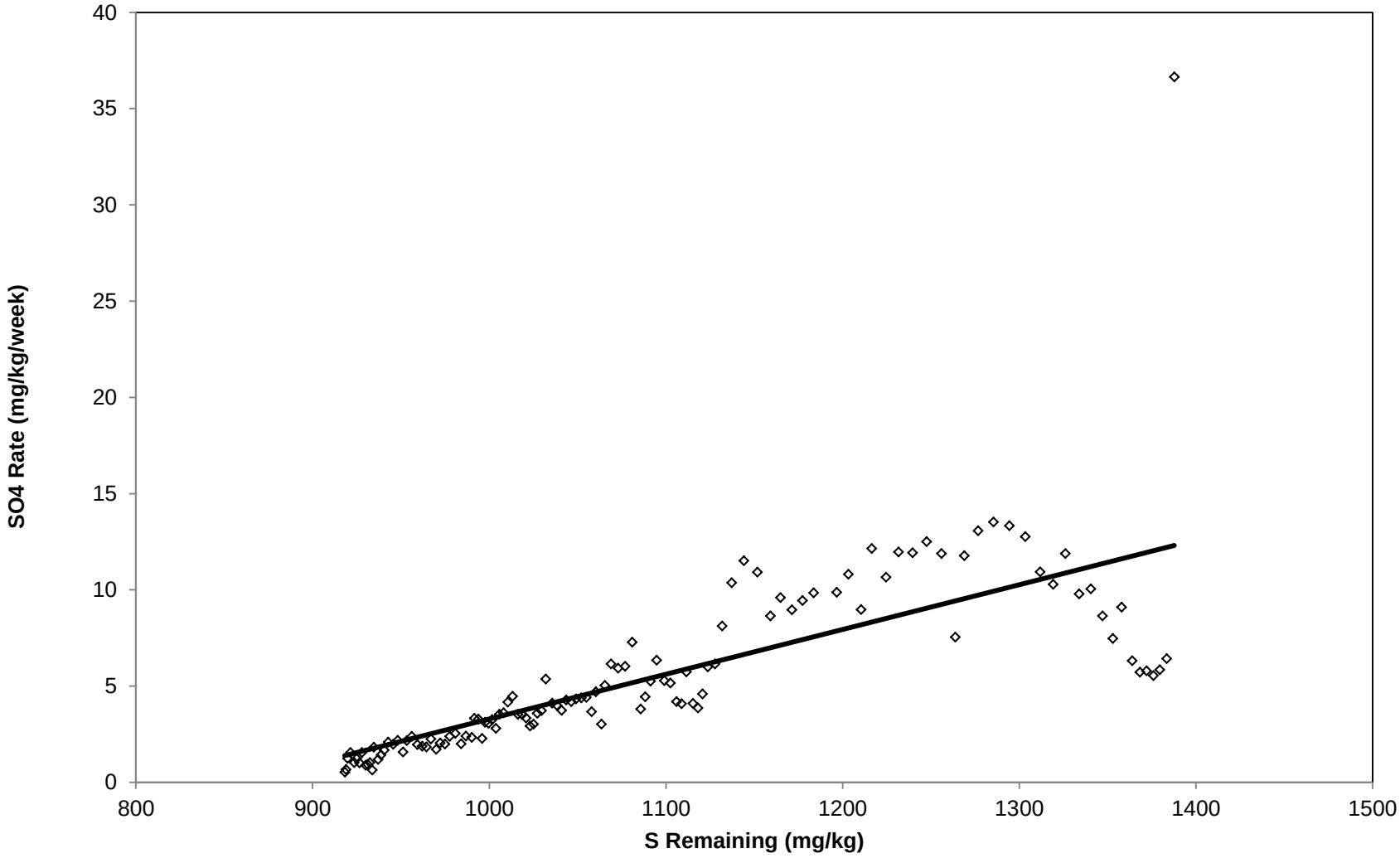

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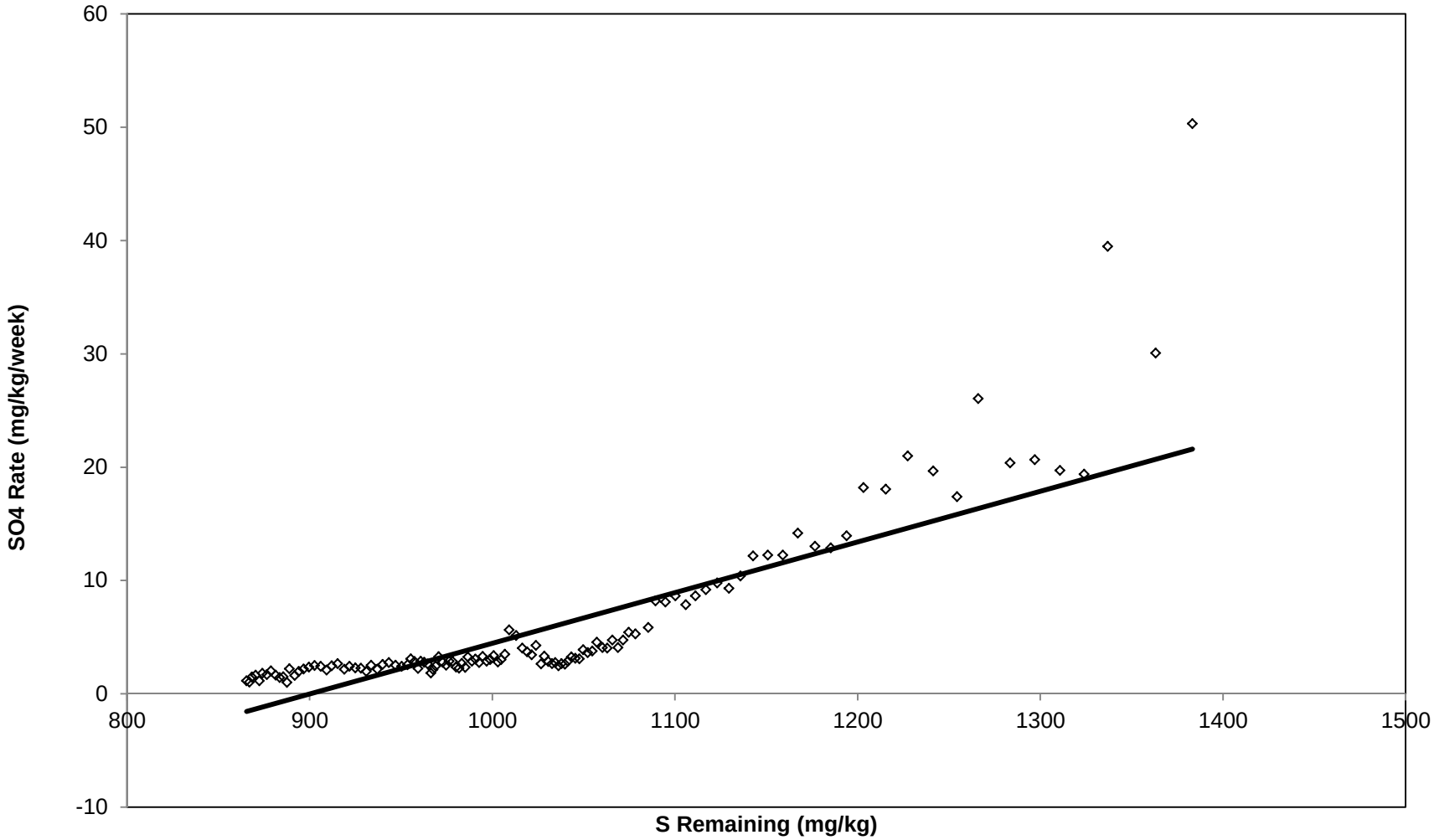

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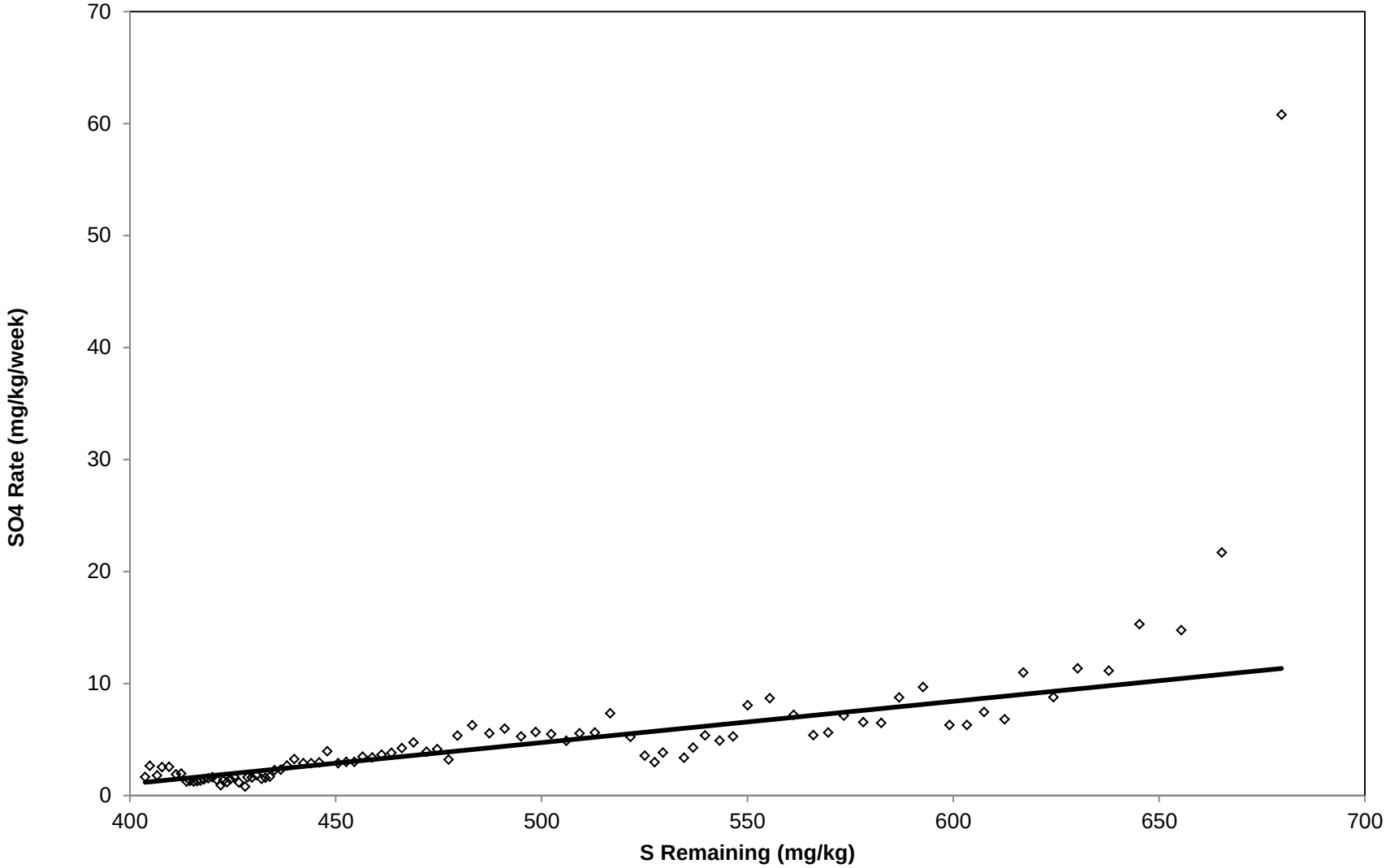

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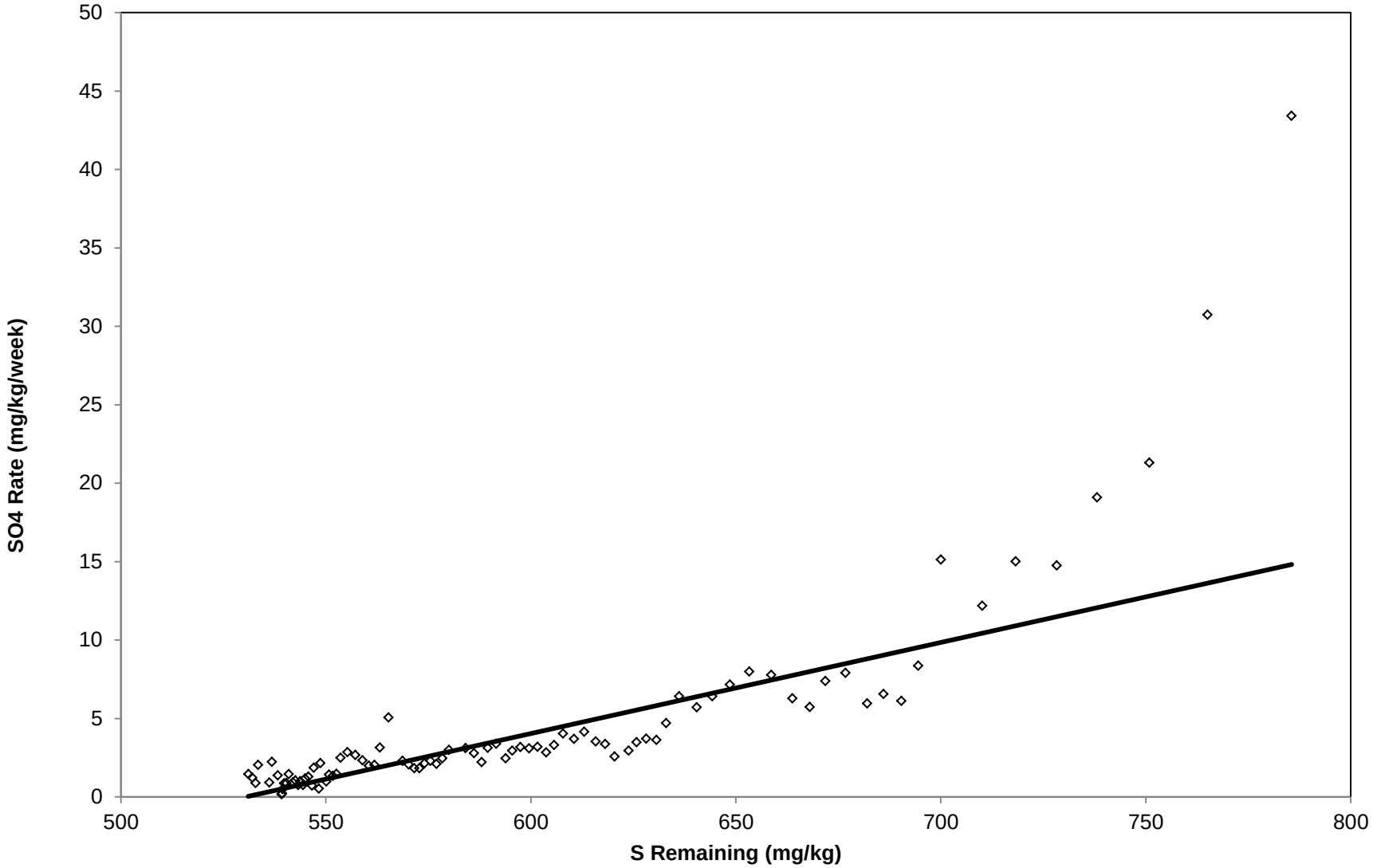

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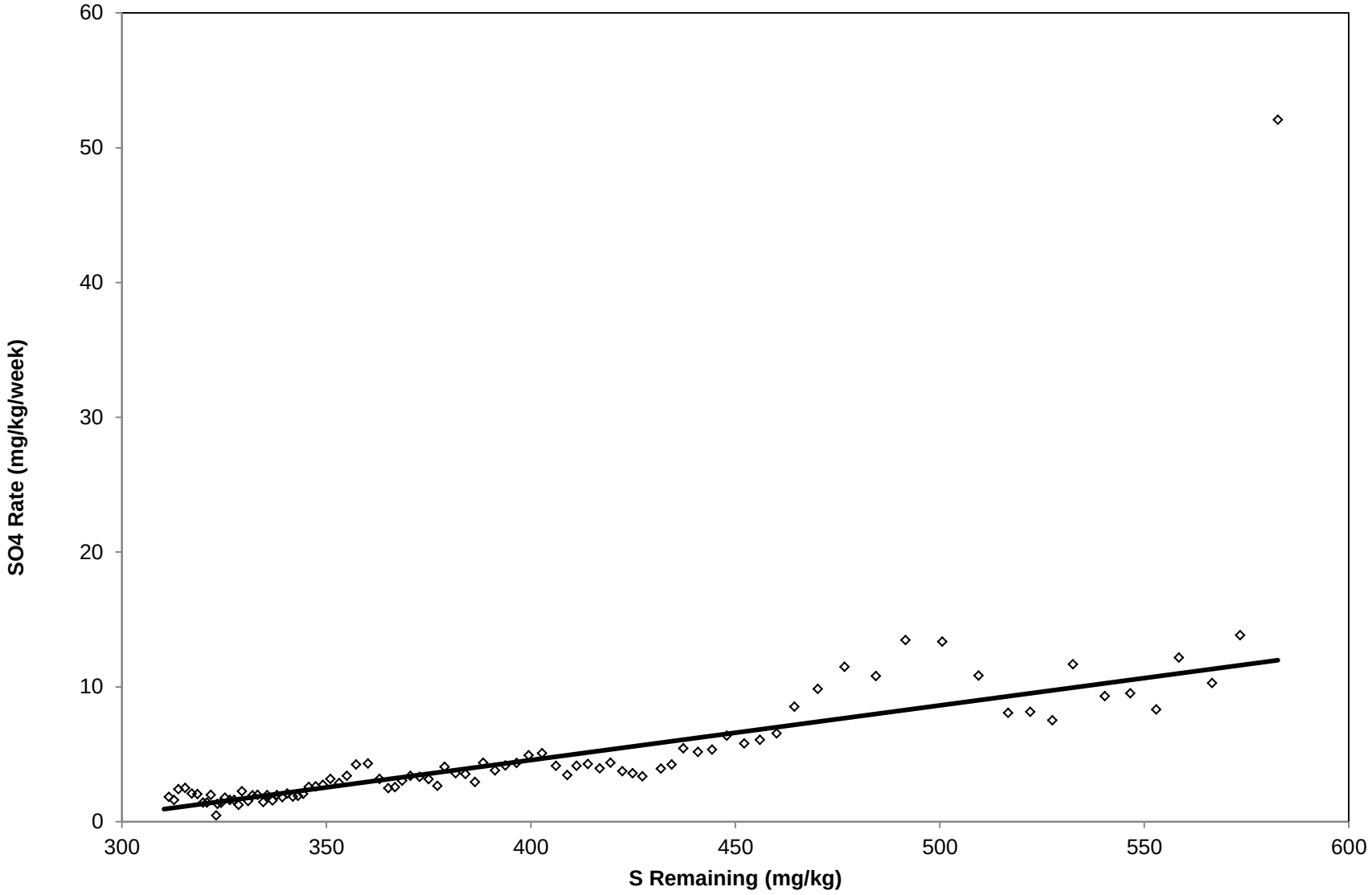

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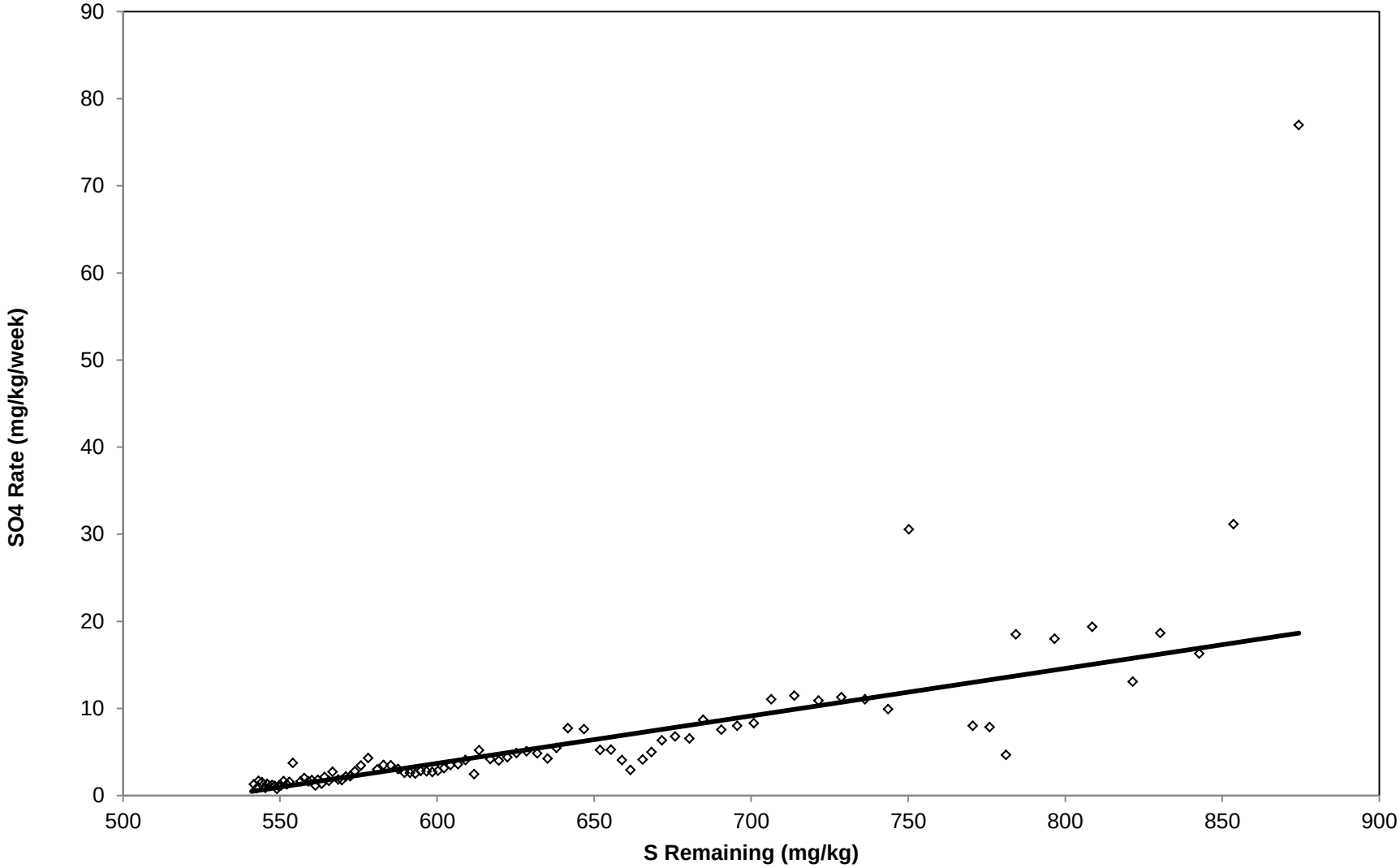

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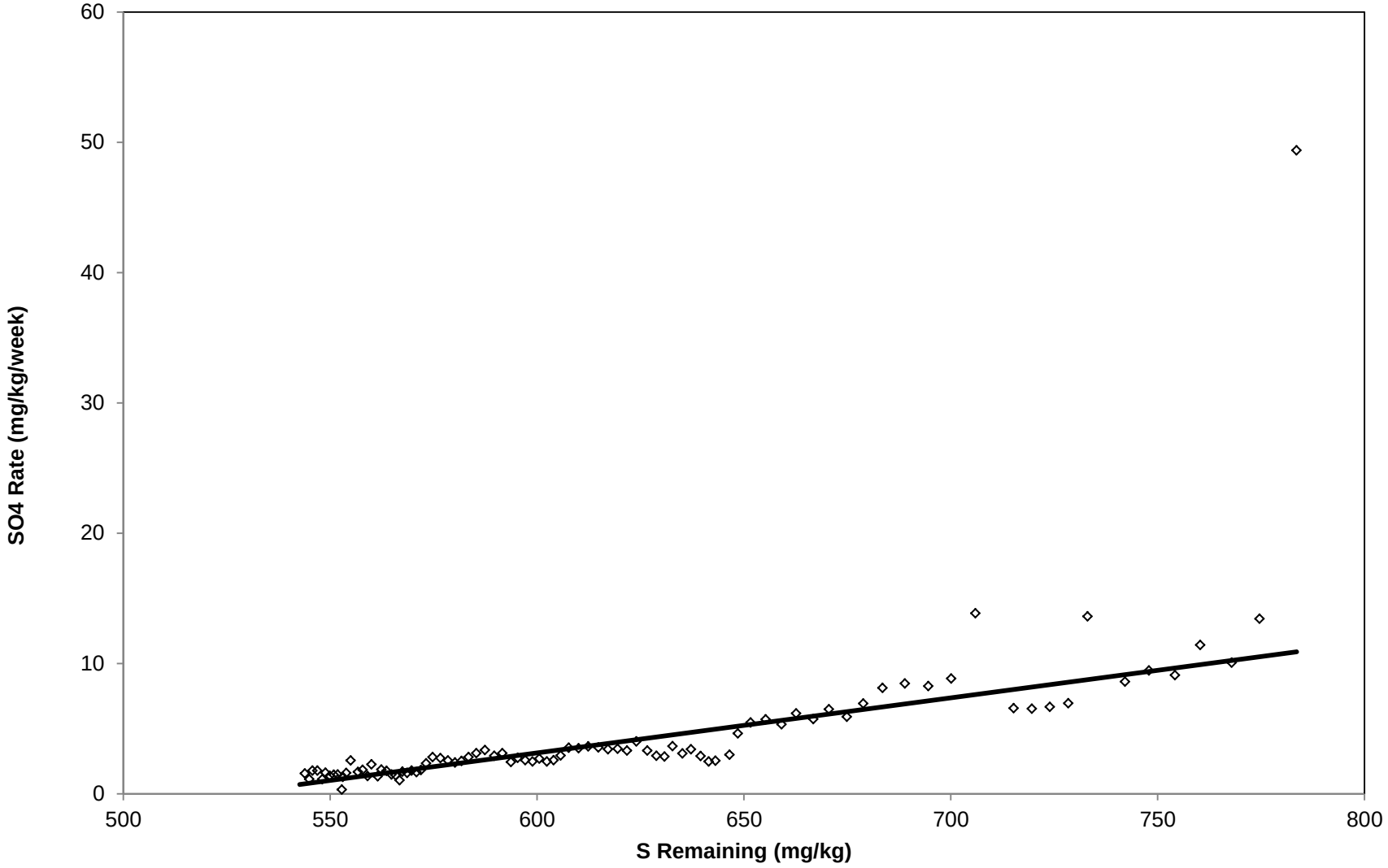

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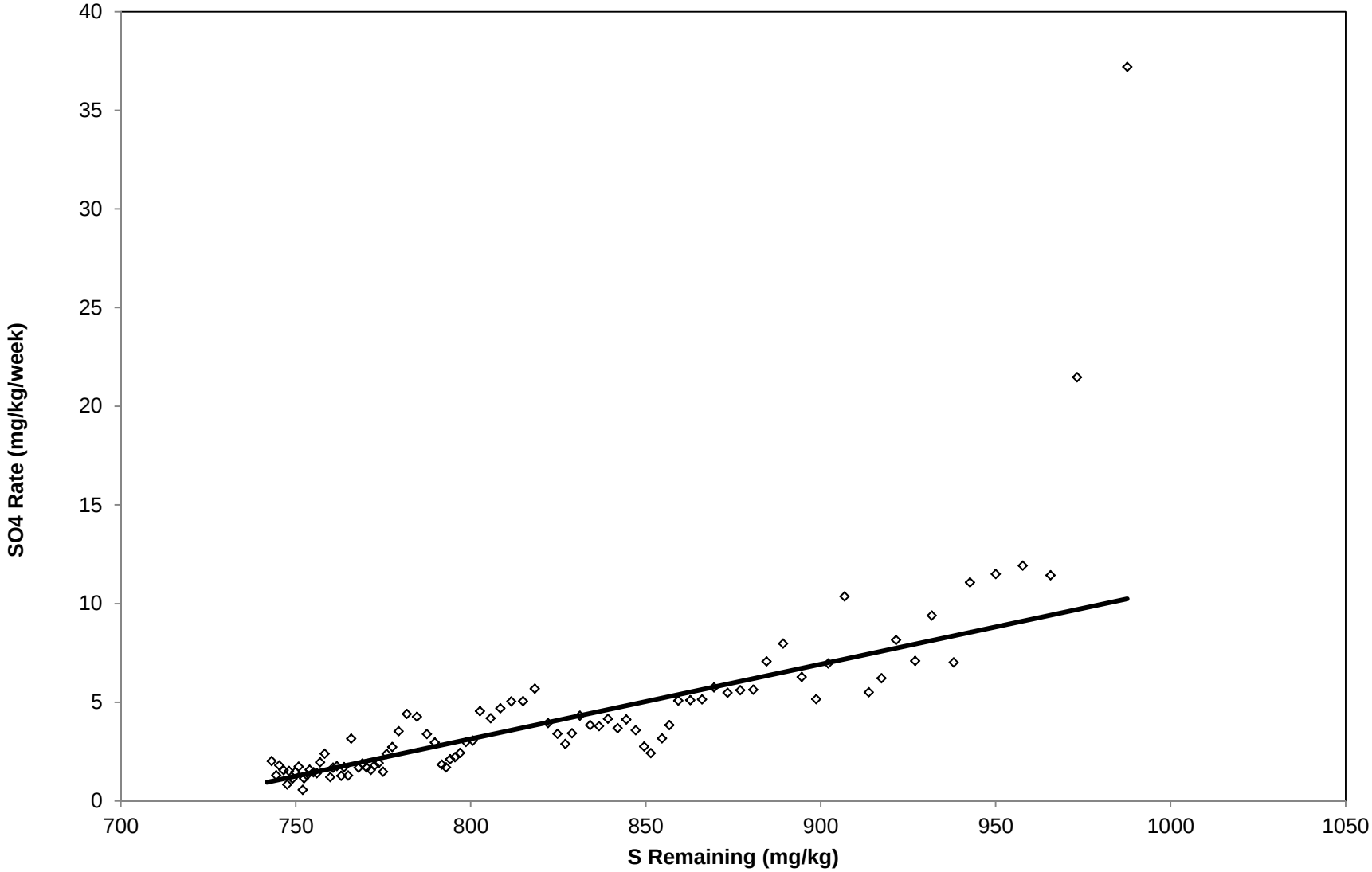

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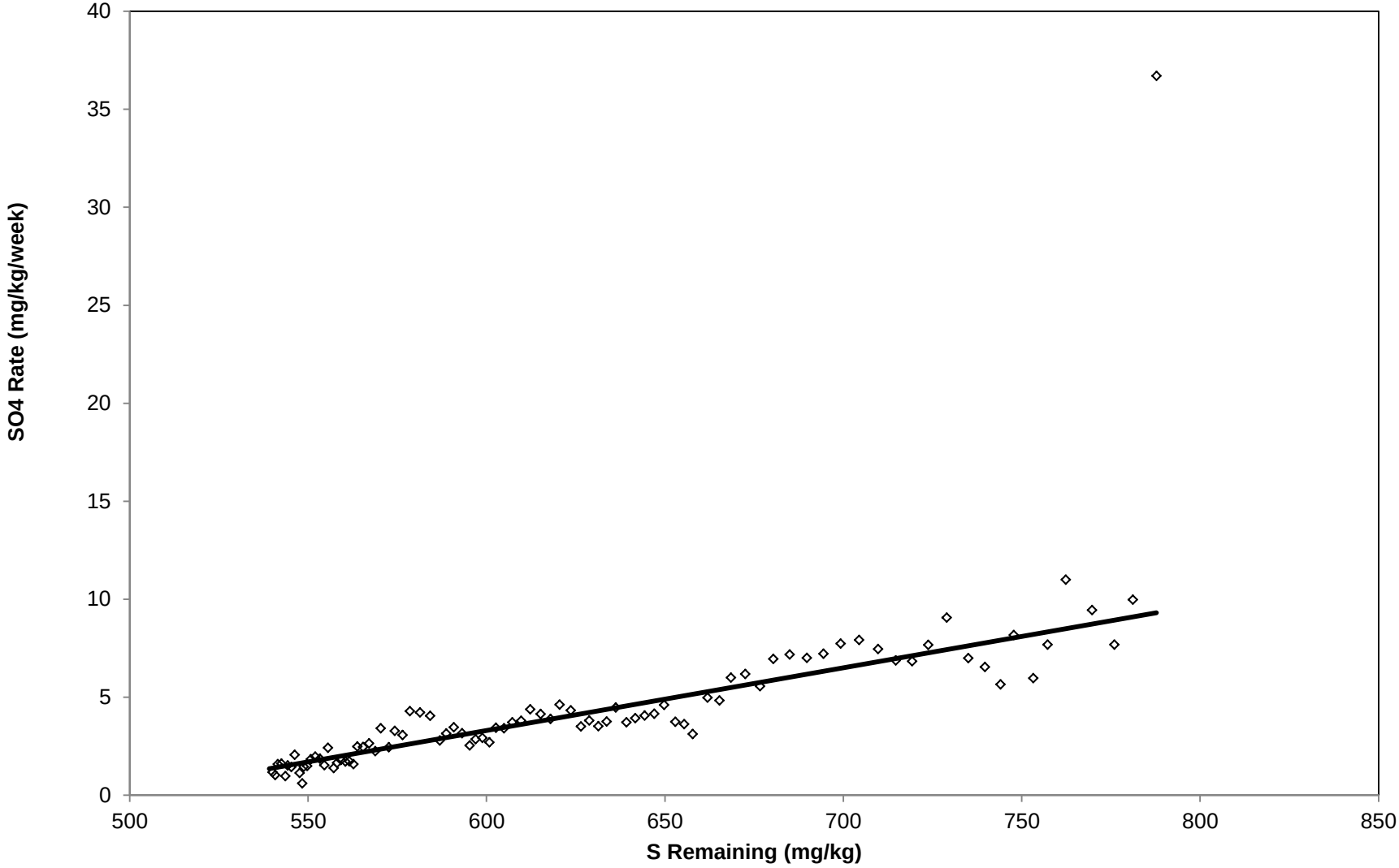

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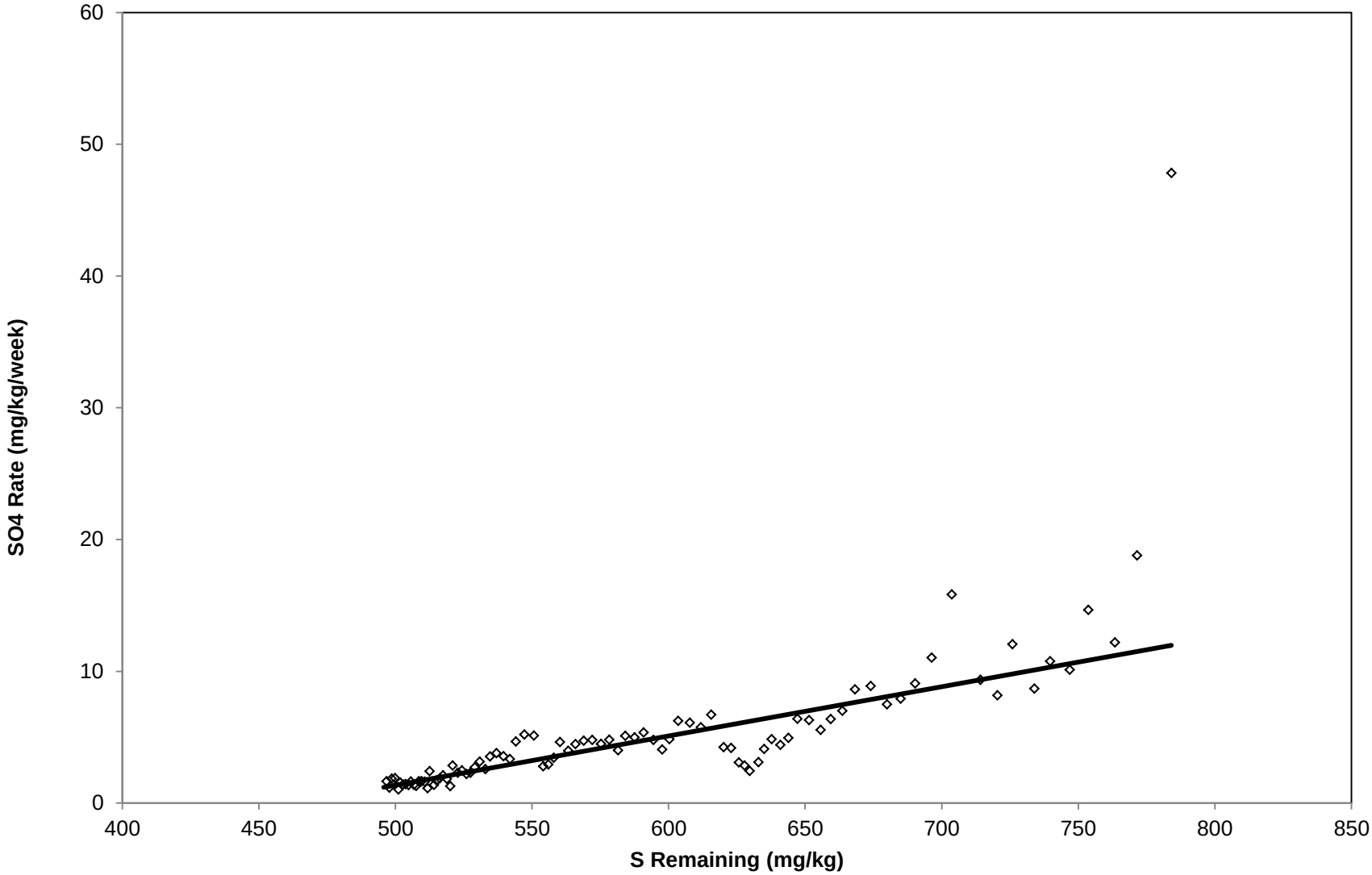

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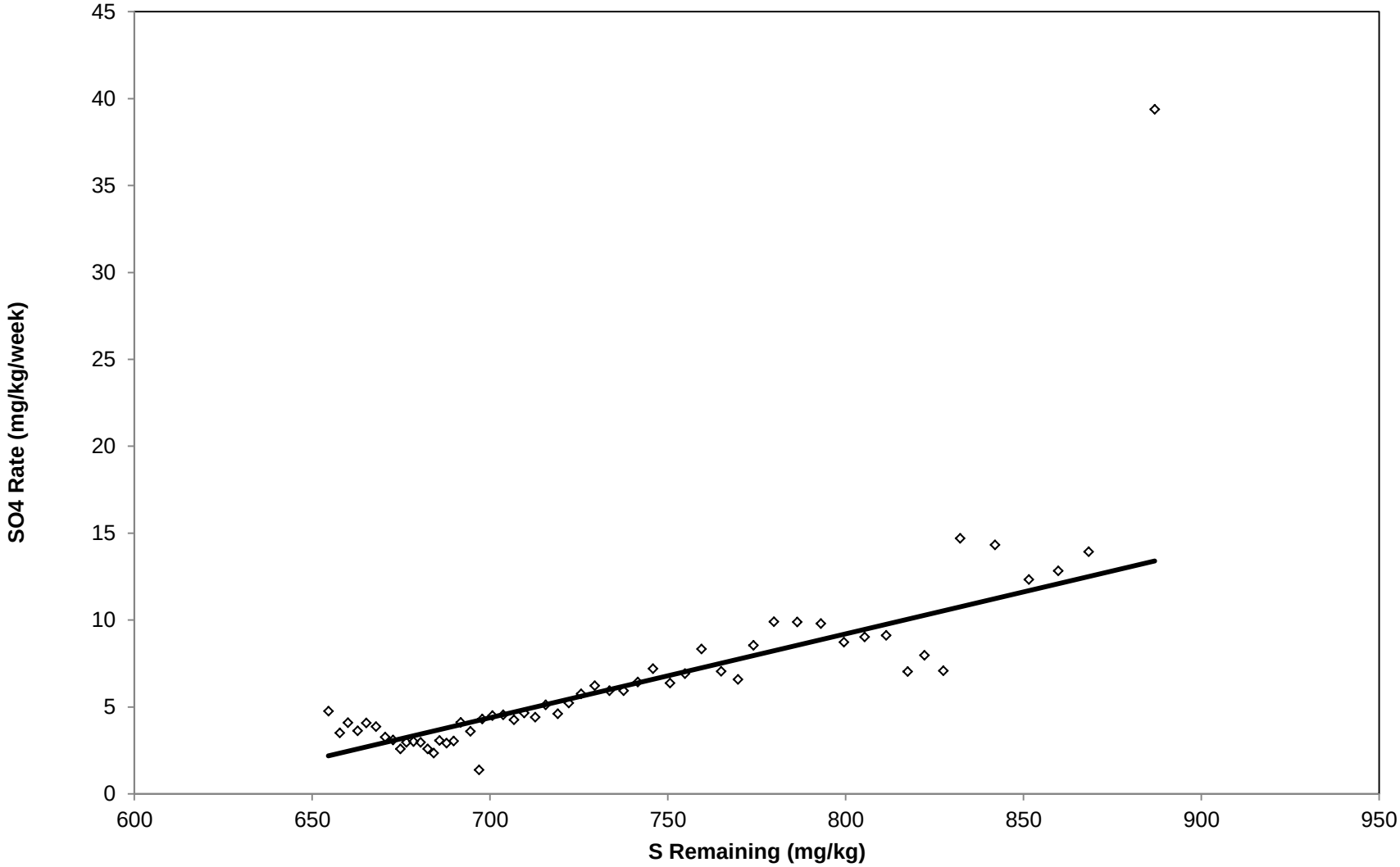

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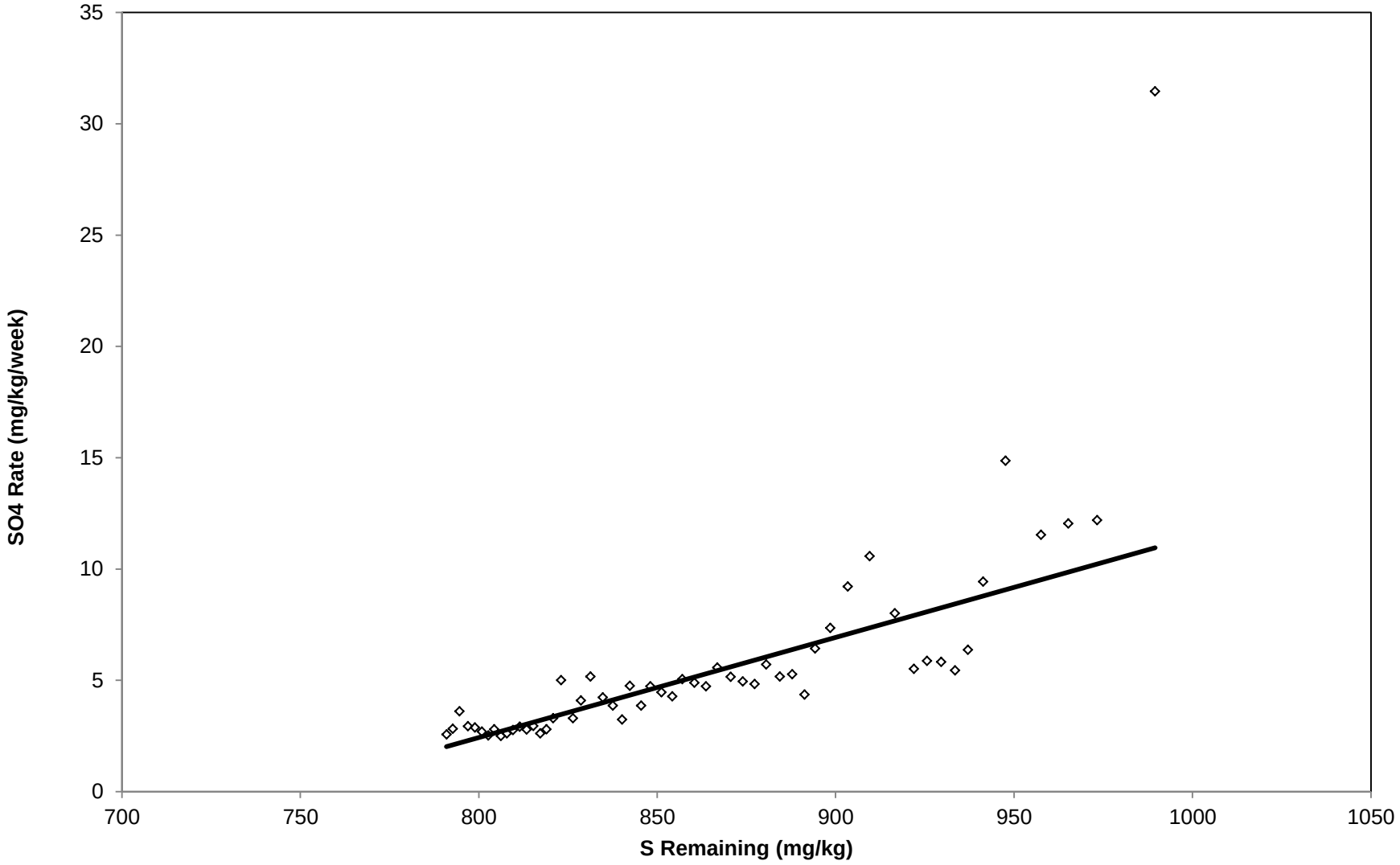

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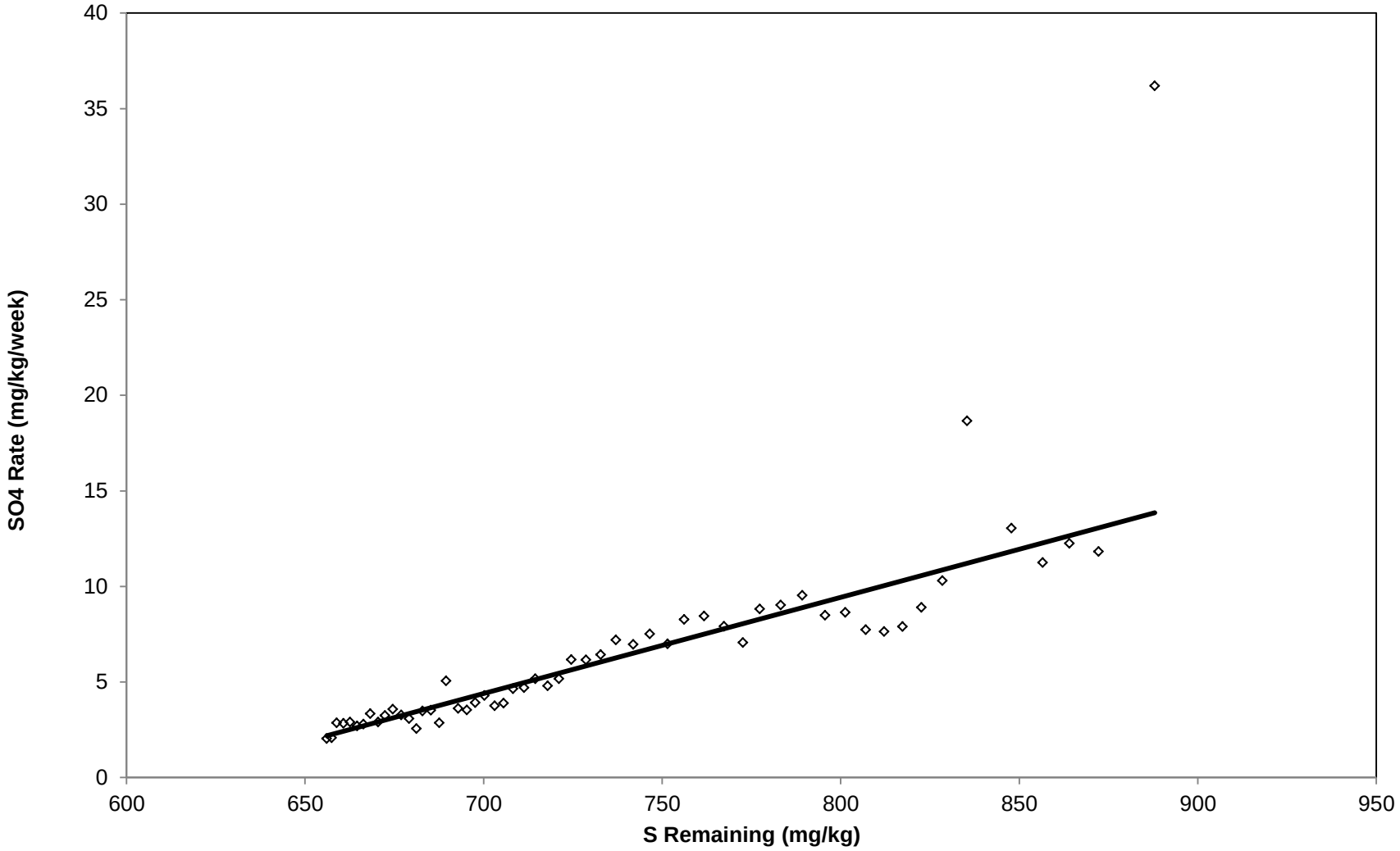

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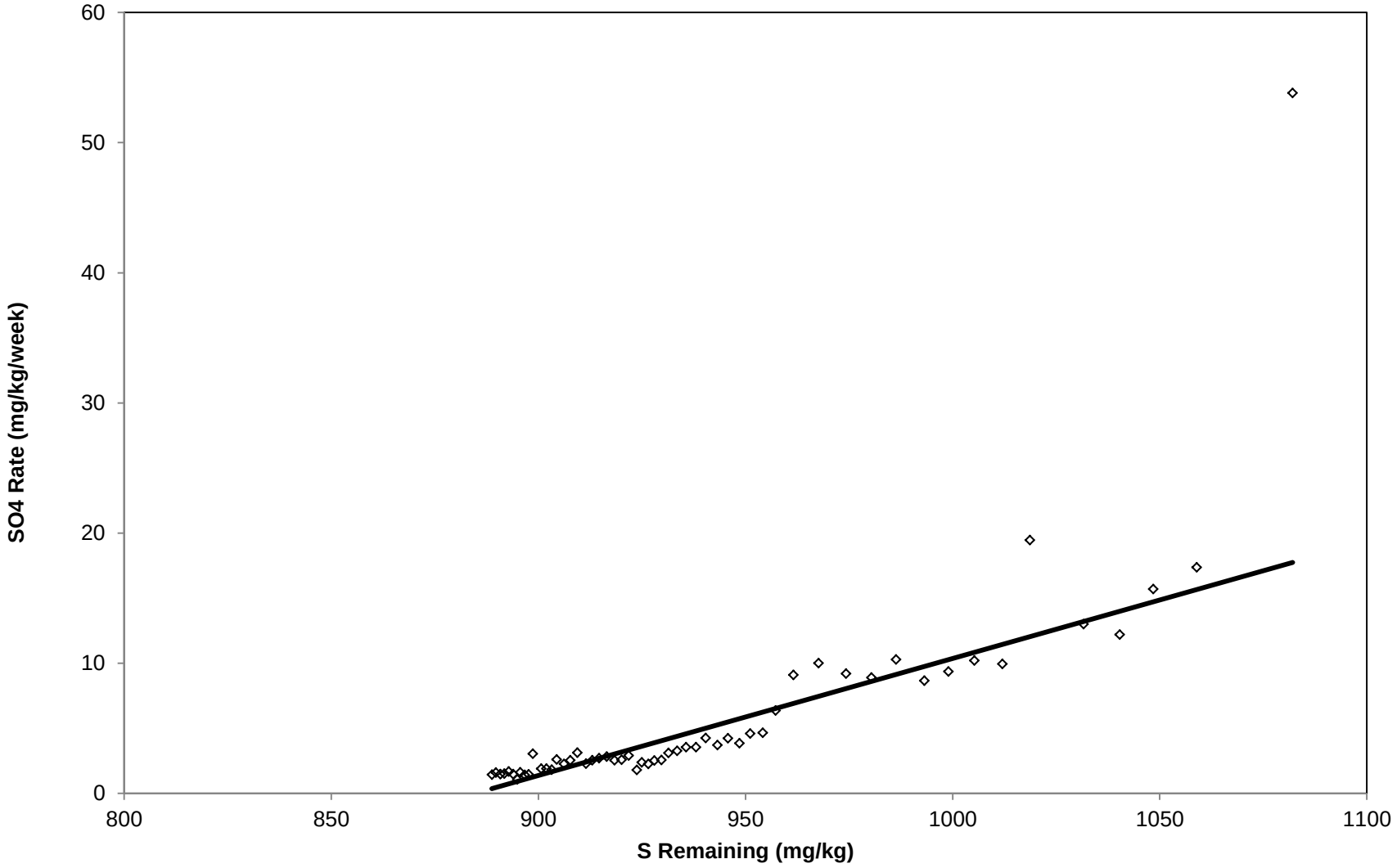

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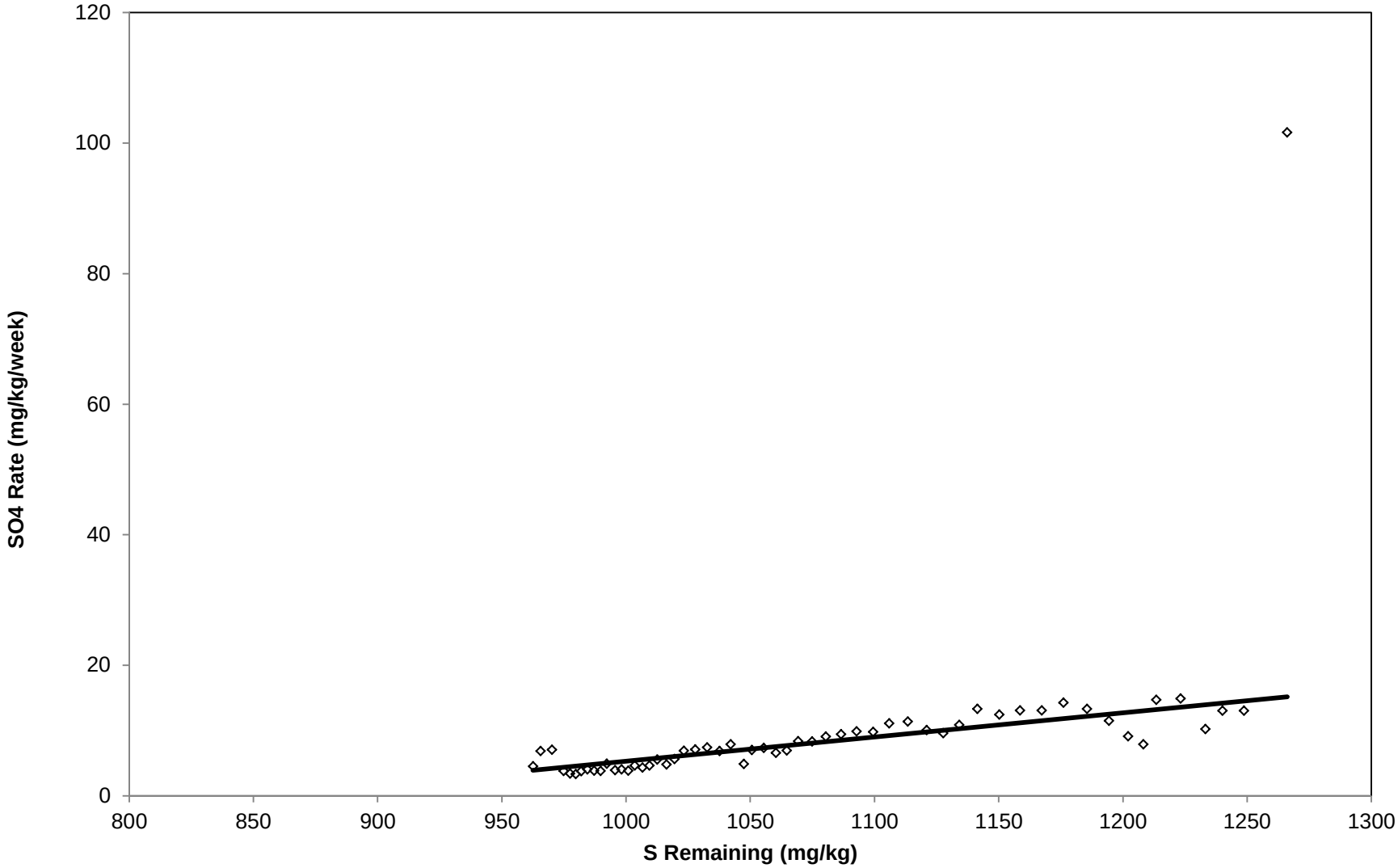

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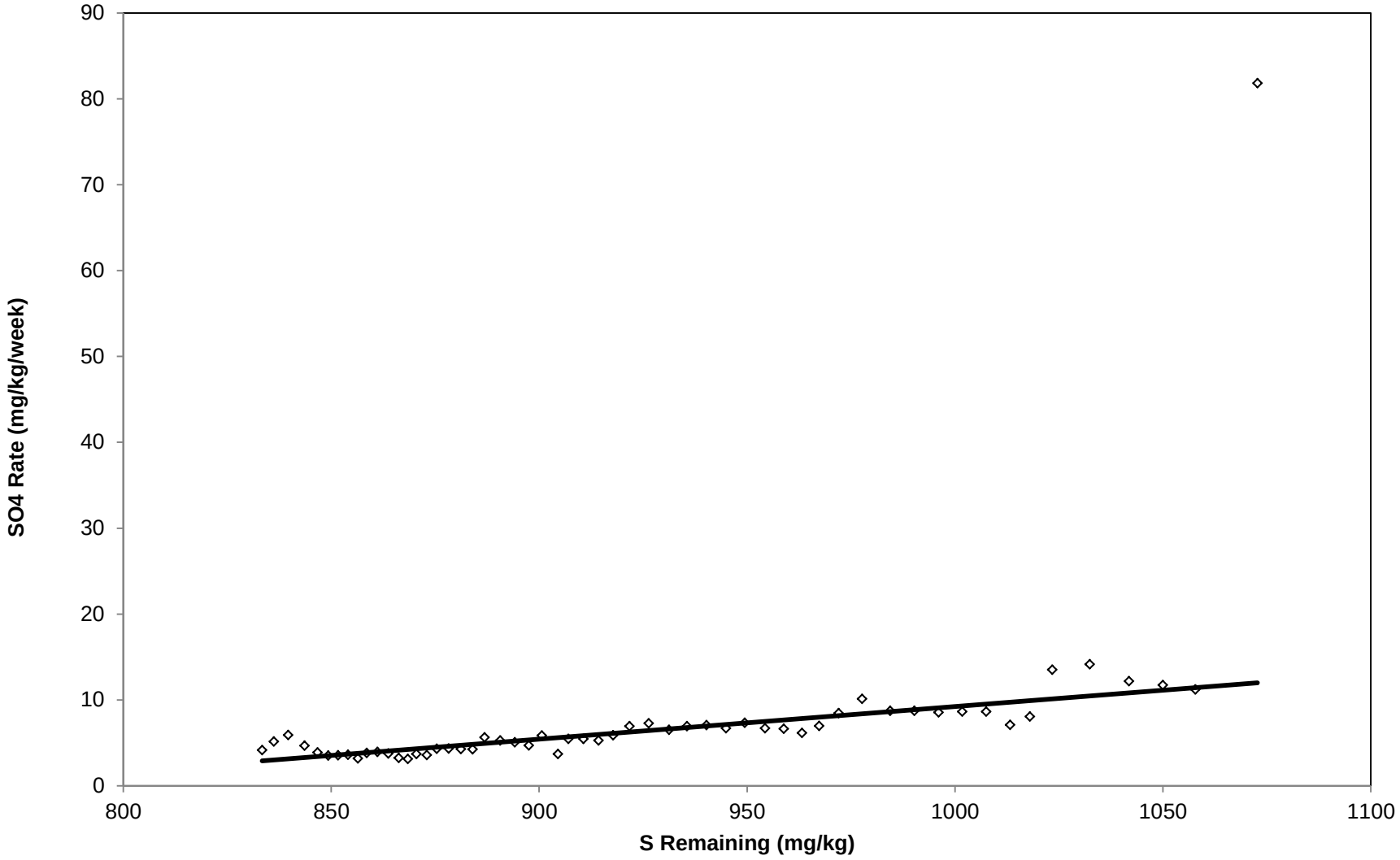

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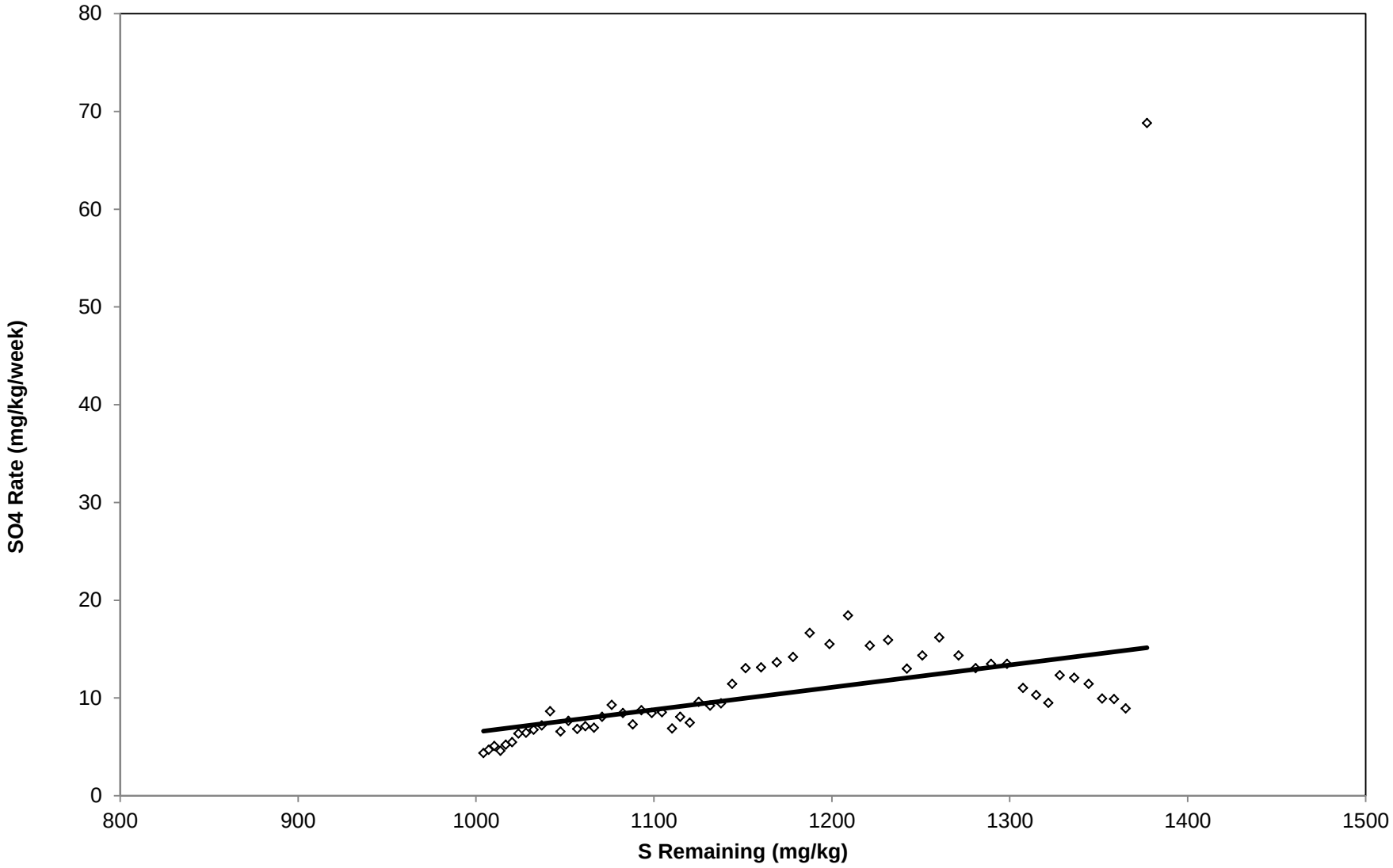

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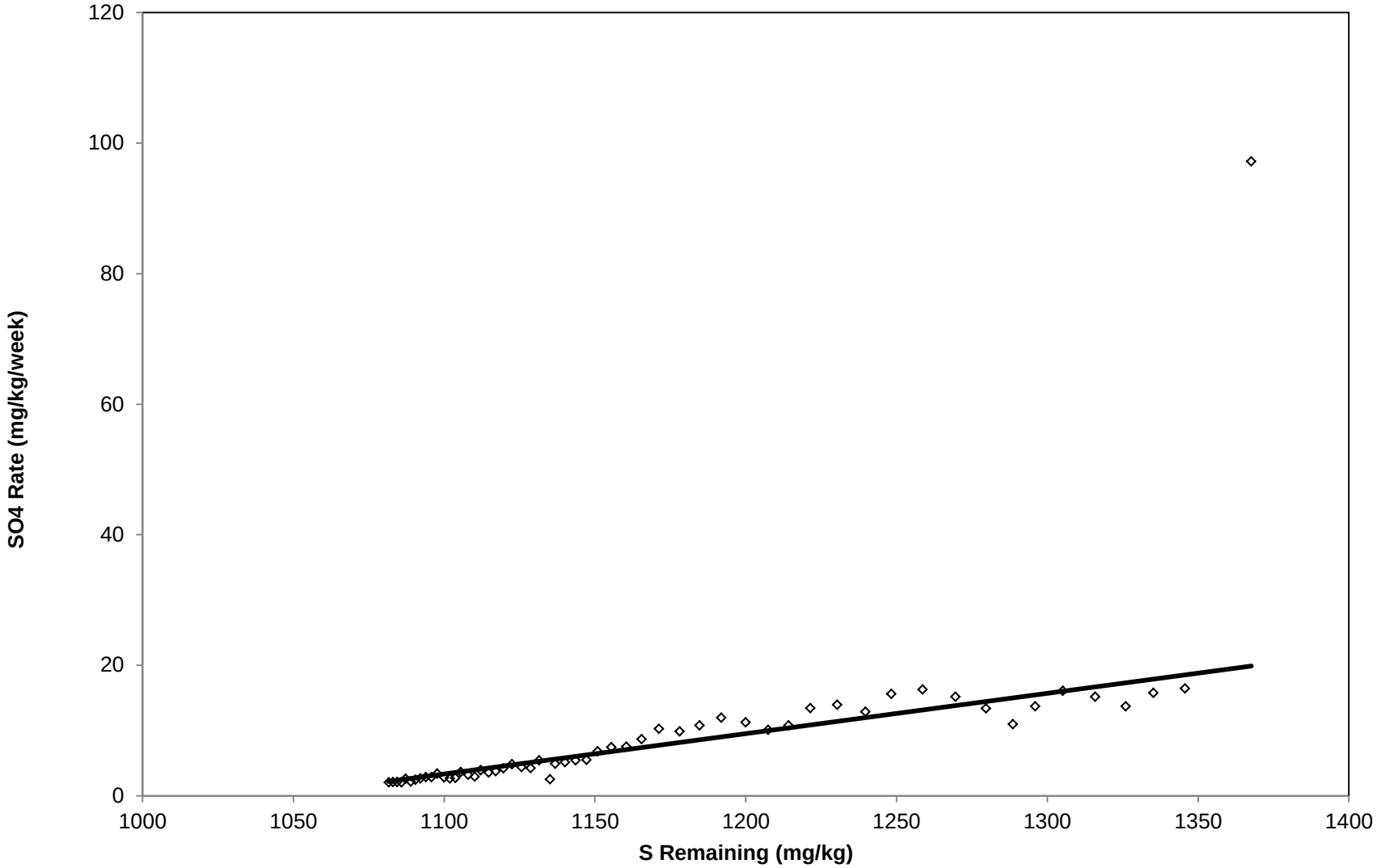

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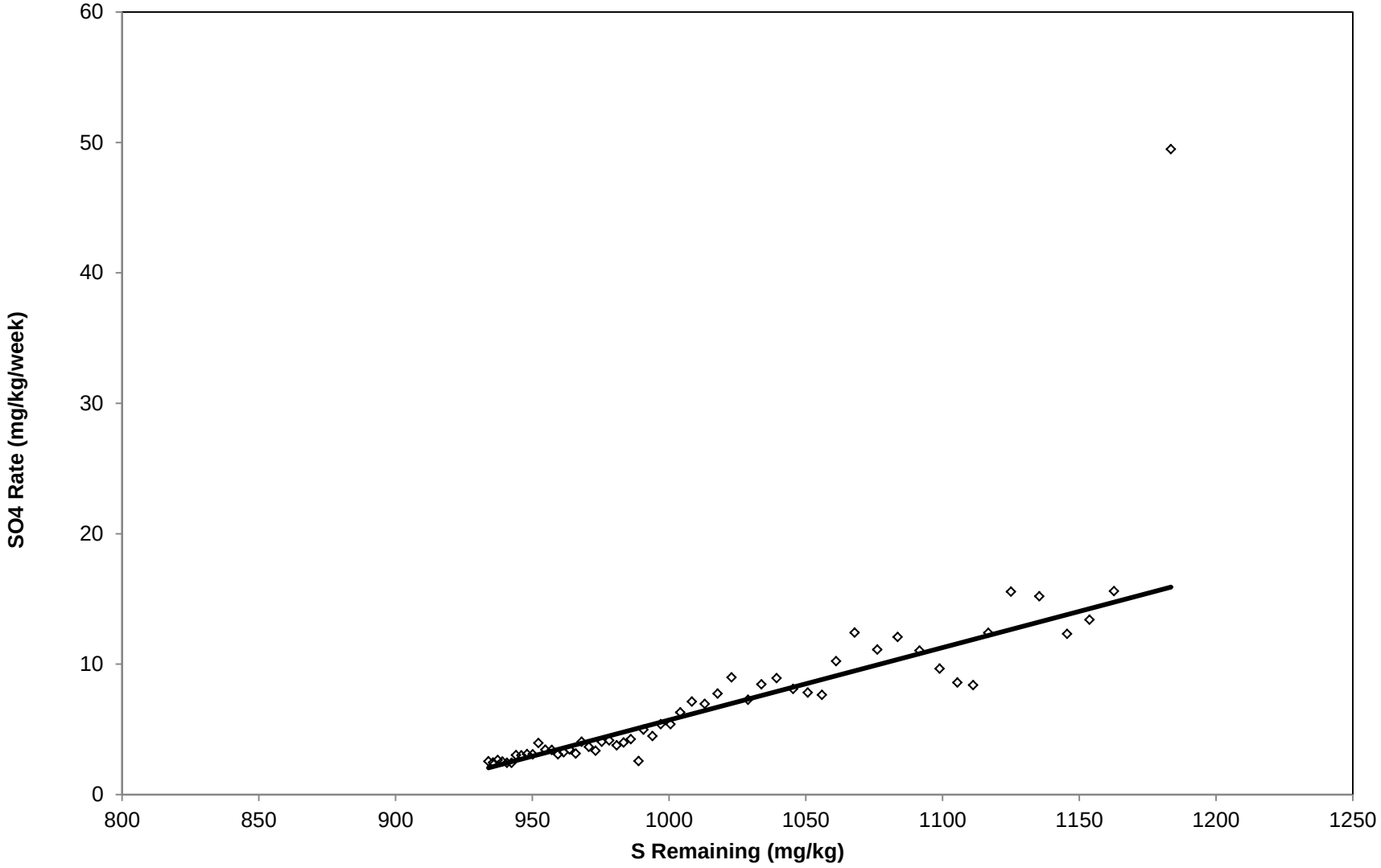

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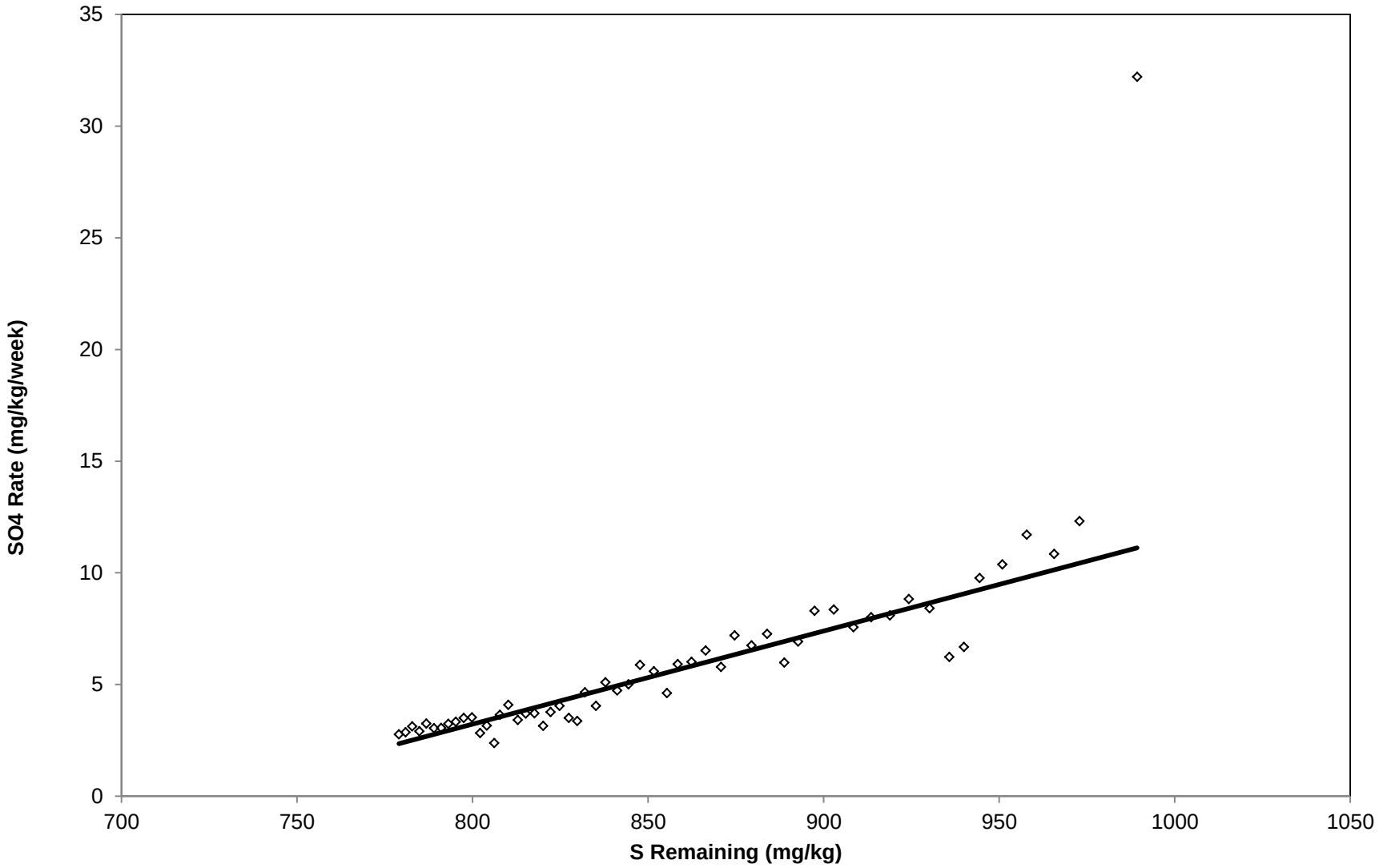

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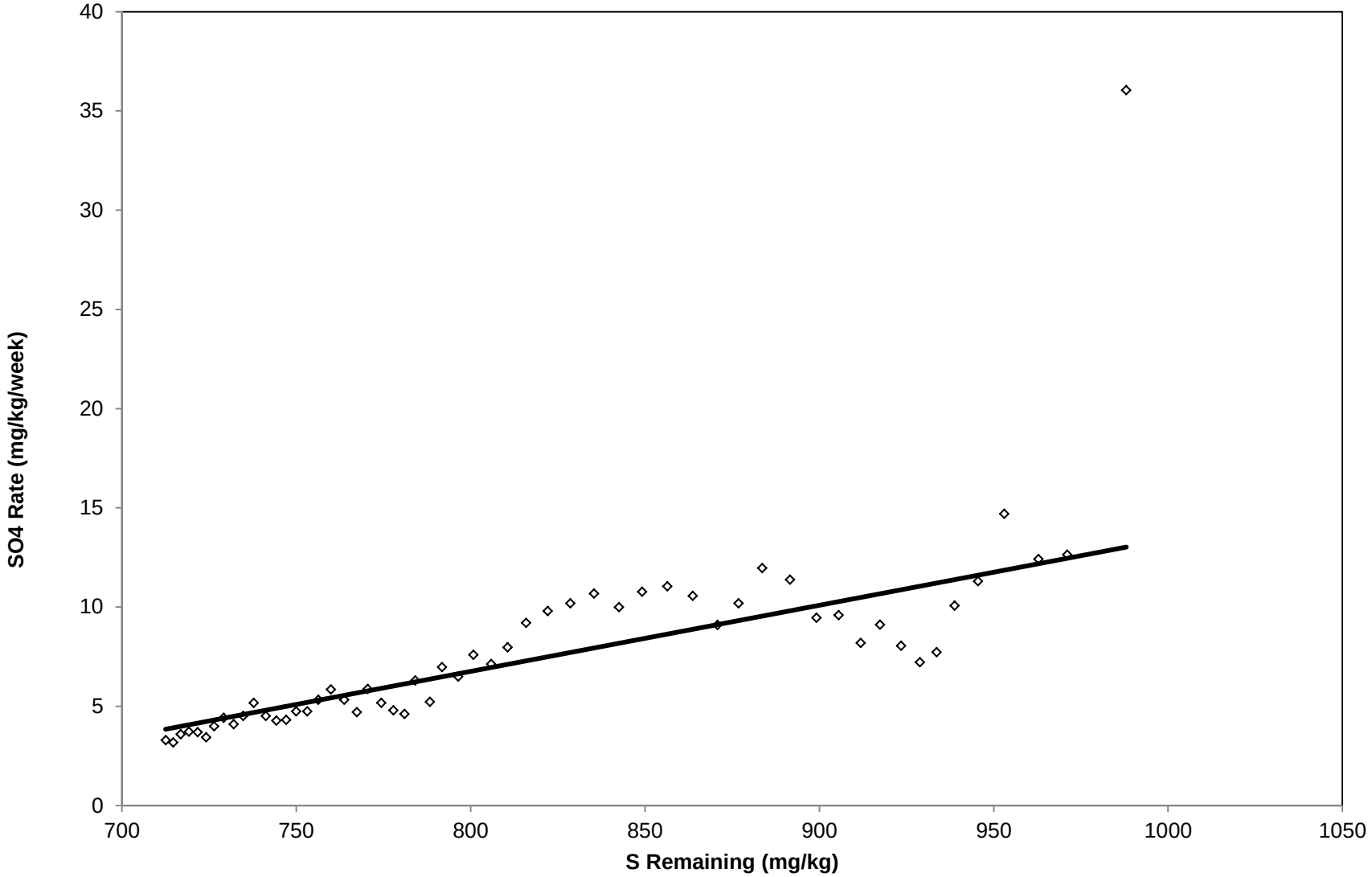

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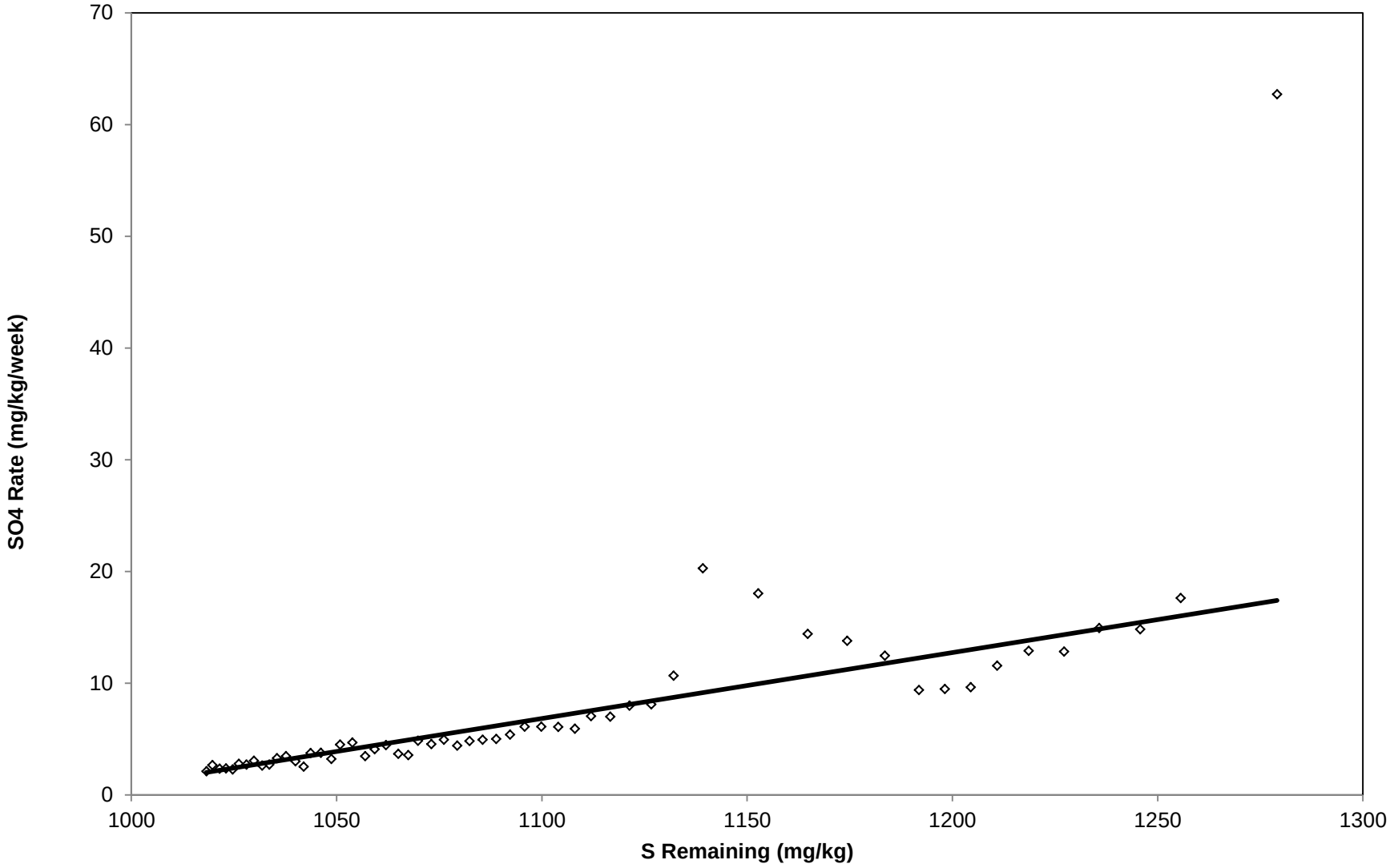

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Attachment D

2014 Update on Kinetic Test Data

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Memo

To:	Jim Scott Jennifer Saran Andrew Ware	**Client:**	PolyMet Mining Corp.
From:	Stephen Day	**Project No:**	1UP005.001
Cc:	Laura Donkervoort Rosemarie Cocuaco	**Date:**	March 17, 2014
Subject:	NorthMet Project Kinetic Test Program Status and Recommendations – DRAFT		

1 Background

Kinetic tests (mainly humidity cells) were initiated for the NorthMet Project beginning in 2004 on samples of rock, Flotation Tailings and Hydrometallurgical Residues produced in a pilot plant according to overall program designs prepared following discussion between PolyMet Mining Inc. (PolyMet), Minnesota Department of Natural Resources – Lands and Minerals (MDNR-LAM) and SRK Consulting (Canada) Inc. (SRK). Results from these tests have been used at various junctures of the project to develop waste management plans and evaluate project environmental effects. Additional tests on LTVSMC tailings planned for dam construction have been started to evaluate that aspect of the project. In 2009, the program was modified in consultation with MDNR-LAM to stop some tests and modify the frequency of analysis of leachates based on trend interpretation (SRK 2009).

The purpose of this memorandum is to provide an update on the progress of the testwork and provide recommendations for modification to the program.

2 Approach

Some components of the program have now yielded over eight years of data on dissolution trends and are showing strong consistency of results. Furthermore, due to slow weathering processes documented by this program and similar programs performed under the direction of MDNR-LAM, few significant changes have been observed in the testing program compared to those reported previously in the last status report (SRK 2012).

For this reason, the approach taken for the current review was to compile comparative observations to determine where modifications can be made to the program. Experience at other mining projects shows there is significant value in continuing some representative tests to verify conclusions drawn from the current dataset as the mine moves into production.

The following sections provide observations on trends observed for each of the major programs (rock, flotation tailings, LTVSMC tailings) and resulting recommendations (summarized in Table 1).

With agreement from MNDR-LAM, testing of residues from the hydromet process produced in 2005 was stopped in 2009 and is not reported in this memorandum. Testing of one sample of hydrometallurgical residue produced from a 2009 pilot plant was started in April 2010 and is continuing. It should be noted that the 2009 hydrometallurgical pilot plant was run with limited concentrate, so it was necessary to run part of the pilot plant using batch processing rather than continuous processing. The batch approach was appropriate to answer metallurgical questions, but produced residue that is not expected to be representative of the NorthMet hydrometallurgical residue during operations. Consequently, plans for environmental testing of the 2009 pilot plant residue (other than the autoclave leach residue, which was produced by a continuous process) were discontinued.

3 Analysis Recommendations

Leachate analysis for continuing tests is being performed at a frequency agreed with MDNR-LAM (SRK 2009a). As the testwork continues, it is clear that trends evolve over periods of years rather than weeks or months, and that trend definition can be achieved with testing on an infrequent basis. Table 2 provides analysis schedule recommendations for all recommended ongoing tests.

SRK (2009b) provided a protocol for adjustment of analytical frequency based on pH trends. Due to the tendency for trends to develop slowly, the proposed schedule shown in Table 2 will be able to detect changes with sufficient resolution that increased analytical frequency is not needed. SRK recommends that the frequency be fixed as proposed in Table 2.

4 Rock Characterization Program

4.1 Observations

Currently, 24 humidity cell tests and 15 “size fraction” tests are operating with the breakdown by waste category and type as indicated in Table 1. Ongoing testwork shows that pH of leachates are consistent with the waste categories:

- No changes in Category 1 tests have been observed since the previous report (SRK 2012). pH variations are apparent but remain within historical ranges.
- Some Category 2/3 tests generated acid as described in previous reports but a number of tests are continuing to yield leachate pHs that are above the pH of the drainage water used to leach the samples and therefore imply a delay to onset of acidic leachate exceeding eight years.
- All continuing Category 4 tests have generated acid as previously reported. The testwork has demonstrated that most Category 4 rock can be expected to generate acid and that the

timeframe to onset is several years in most cases. The main observation for the continuing tests is the upward trend in copper release (Figure 1).

- All three ore composite samples are showing the same trend with pH being comparable to Category 2/3 rock.
- The parallel tests on size fractions and using the DNR Reactor test configuration has shown that differences in rates can be explained by differences in reactive surface area and water application ratio but the tests do not fundamentally change how rock weathering is interpreted.

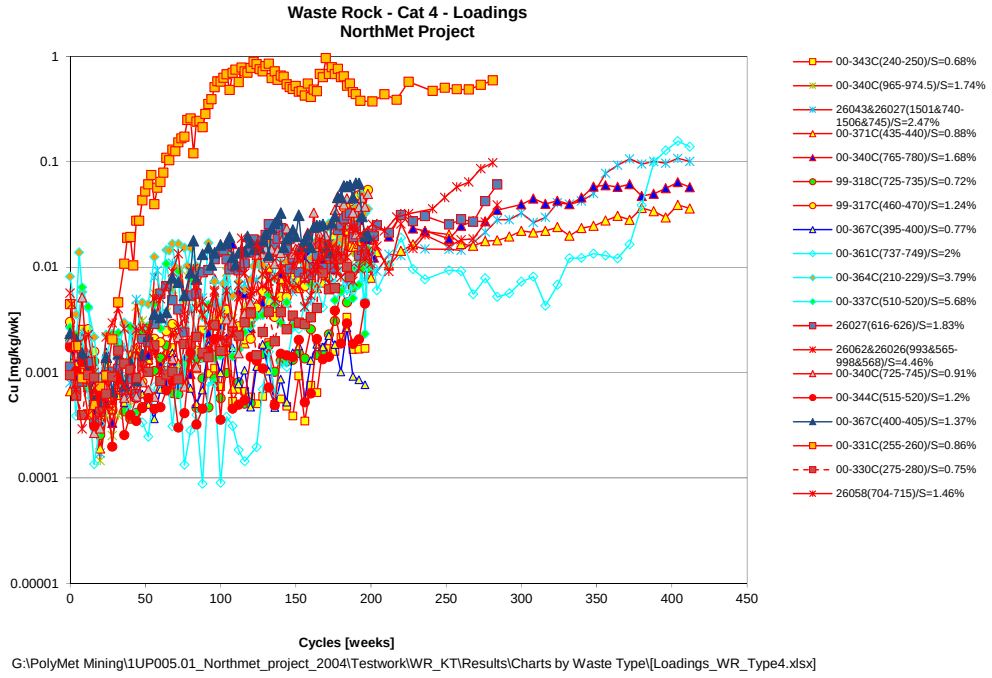

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Figure 1. Copper Release for Category 4 Samples

Table 1. Status of Testwork, Findings and Recommendations

Program	Sub-program	Type of Test	Original Total Tests	Tests Operating	Duration (weeks)	Status	Conclusions	Recommendation and Rationale	Tests to Continue
Rock	Category 1	HCT	37	7	420	Stable, no change since last report.	Tests are confirming pH near neutral and varying within historical range.	Continue three tests to monitor range of S content. Demonstrate that Cat 1 does not generate acid.	3
Rock	Category 2/3	HCT	26	10	420	Eight tests are stable pH above 6 with no change from last report. One test (UM, S=0.2% is showing lower pH with variations within historical range. Metal release peaked several years ago.	Tests have shown the majority of samples in Cat 2/3 do not show pH depression below 5 for at least eight years. The majority of tests are maintaining pH above 6 indicating that acid generation under site conditions would not occur for at least eight years.	Continue three tests with lower S contents to evaluate use of a higher criterion for segregation of Cat 1 from 2/3. Continue one test at highest S content to evaluate delay to onset of acidic conditions.	4
Rock	Category 4	HCT	22	4	420	Two tests with S>=2% have pH<4. Two tests with S<1.7 have 4<pH<5 and declining but pH is within historical range. SO_4 is declining for these tests. Cu leaching rate is increasing for all four tests.	Continuing tests show acidic pH but no indication of further declines after 4 years.	Continue two tests to evaluate copper leaching. One test from each current pH group.	2
Rock	Ore	HCT	3	3	420	Results are very similar for all tests. pH stable at 5.	Average ore type materials show consistent behavior without generating severe acidity after eight years. Variability of oxidation rates for ore grade sulphur content (Cat 4) is shown by Cat 4 HCTs.	All three samples so show similar behavior. Continue 1 test to evaluate long term leaching of ore type materials.	1
Rock	All categories	Parallel HCT, DNR reactors	15	5 samples (3 additional tests per sample)	420	Tests are demonstrating stable leachate chemistry. Results remain consistent with lower rates for coarser size fractions but also influenced by pH of deionized water resulting in higher metal solubility for coarser fractions.	Tests have shown reactivity differences for size fractions.	Terminate all tests.	0
FlotationTailings	Pilot Plant 1	HCT	13	13 (4 bulk, and 9 size fractions)	390 to 420	The majority of tests are yielding pHs above 7 showing pH recovery from as low as pH 6 earlier in test. Sulfide consumption is 50% in some cases and sulfate release continues to show declining trends. Samples with higher S content (from not using $CuSO_4$ during processing) initially showed higher metal release but all are showing steady decline. Metal release from lower S tailings is approaching historical lows.	Test work has shown that significant pH depression is not expected. The consumption of a significant fraction of the sulfide while continuing to generate alkalinity supports the conclusion that alkalinity generated by silicate weathering will permanently offset acid generated by sulfide oxidation.	Due to the consistency of trend after eight years of testing, a limited number of tests should continue to confirm the conclusion that acid generation will not occur. The P3 samples are proposed as they represent typical performance. The bulk, fines (-200 mesh) and coarse (+200 mesh) fractions have been selected.	3
FlotationTailings	Pilot Plant 1	DNR reactors	13	13 (4 bulk, and 9 size fractions)	390 to 420	The trend is similar to the HCTs but with pH shifted down to mostly below 7. Otherwise trends are the same as HCTs. pH is tending upwards from historical lows.	The main factor resulting in differences is the small sample volume and higher liquid to solid ratio.	Due to the similarity with HCTs on the same material and explainable differences, all tests should be discontinued.	0
Flotation Tailings	Pilot Plant 2 and 3	HCT	8	8 (2 bulk, 6 size fractions	270	These tests showed decreasing pH initially but pHs have not decreased below what appears to be a strong buffered pH of about 7.3 which has been maintained for four years. Long term decline in sulfate release is observed. Alkalinity has been leached at a higher level than the PP1 tests which may explain the higher and sustained pH. As a result, metal leaching has been observed at low levels.	The testwork has shown that tailings generated by Pilot Plants 2 and 3 have been able to sustain higher pHs and lower metal leaching. These tailings had lower S content than PP1 and the results are therefore consistent with PP1 - lower S results in less acid generation and higher alkalinity.	Since these tailings demonstrate the performance with lower S content, it is recommended that one set of tests (bulk, +200 and -200) be continued in parallel with PP1 to conform that higher pH can be sustained. PP2 is proposed as it contains the highest sulphur content.	3
Flotation Tailings	Pilot Plant 2 and 3	DNR reactors	8	8 (2 bulk, 6 size fractions	270	The trend is similar to the HCTs but with lower pH.. A dip in pH to below 7 was apparent earlier in the program which was not apparent in the HCTs. The distinctive "buffered" chemistry is apparent but at lower pH.	The main factor resulting in differences is the small sample volume and higher liquid to solid ratio.	Due to the similarity with HCTs on the same material and explainable differences, all tests should be discontinued.	0
Flotation Tailings	Scavenger Tailings	HCT	12	12 (3 bulk, 9 size fractions	200	The tests showed the same general trend as PP1 with pH declining then increasing, though the lowest pH was no lower than 6.8. One test showed an unusual "sawtooth" pH trend resulting in pHs to 6.5 which is being evaluated with the laboratory. The pH depression was accompanied by detectable increase in metal leaching but two orders of magnitude below that of PP1 samples.	Results from these tests were consistent with those of other programs and demonstrate that pH depression and metal leaching are linked to S content. None of the tailings samples tested in any program have generated leachate at a pH below that of the deionized water.	Due to the consistency with other long term testwork, it is recommended that testing of the Oct 1 9 am group be continued as it contains the highest sulphur content.	3
Flotation Tailings	Scavenger Tailings	DNR reactors	12	12 (3 bulk, 9 size fractions	200	The trends are similar to the HCTs but with a lower pH.	The main factor resulting in differences is the small sample volume and higher liquid to solid ratio.	Due to the similarity with HCTs on the same material and explainable differences, all tests should be discontinued.	0
LTVSMC Tailings	Borrow materials	HCT	5	5	160	All tests are showing stable leachate with pH above 7.3, stable alkalinity and sulfate release below 1 mg/kg/week. Very low, barely detectable metal leaching is occurring.	The results are consistent with the low sulfide content and excess of carbonate minerals. A trend toward lower pH is not expected.	Discontinue all tests.	0
Flotation Tailings on LTVSMC Tailings		Column	Two LTVSMC tailings controls, four layered tests	Two LTVSMC tailings controls, four layered tests	390	These tests were designed to evaluate geochemical effects from placing Flotation tailings on LTVSMC tailings. They are yielding stable trends that are consistent with the mineralogy of the samples and the parallel HCTs. Resultsare similar between tests despite testing different variables. The LTVSMC tailings are removing As and V leached from Flotation tailings. A weak Ni removal effect is apparent.	The testwork shows that placing Flotation tailings on LTVSMC tailings does not seem to contribute to leaching of LTVSMC tailings but LTVSMC tailings may be beneficial in removing some components leached from Flotation Tailings. The removal process is probably occurring via adsorption.	Due to the potential value of LTVSMC tailings to remove metals from Flotation leachate by adsorption, continuation of some testwork is recommended to evaluate "break through" effects. Recommended testing includes - Maintain one control due to similar performance of both (coarse selected). Monitor bottom port only. - Maintain the two P1 experiments because they are leaching highest As and are likely to breakthrough sooner. Monitor top and bottom ports only. - Perform testwork on decommissioned columns to look for precipitates.	One control, two column sets
Hydrometallurgical Residue		HCT, SFE, DNR Reactor	3	3	165	Leachate chemistry reflects dissolution of soluble components. pH has stabilized at near 5 as dissolution of the residue continues to occur.	Due to the high liquid to solid ratio, the testwork represents a very long duration under natural site conditions.	Discontinue both tests.	0

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Table 2. Analytical Schedule

Parameter Group	Original Schedule	Current Schedule	Recommended
Leachate recovery	Weekly	Weekly	Weekly
pH, conductivity	Weekly	4-weekly	Quarterly
Alkalinity, sulfate	Bi-weekly	4-weekly	Quarterly
ORP, acidity	Weekly, bi-weekly	4-weekly	None
Total inorganic carbon	Bi-weekly	None	None
Fluoride, chloride	Bi-weekly	None	None
ICP ES Elements	4-weekly (0, 4, 8, 12, etc.)	None	None
ICP MS Elements	4- weekly (2, 6, 10, etc.)	8-weekly (2, 10, 18, etc)	Twice annually

4.2 Recommendations

Ten samples are been recommended for continuation (Table 3) with the following objectives:

- Validate that Category 1 rock does not generate acidic leachate. Three samples were selected to cover the range of sulfur concentration.
- Continue three tests with lower S content in Category 2/3 as this may provide data to determine if the segregation criterion is appropriate.
- Continue one test at higher S content in Category 2/3 that currently has not generated acid to evaluate delay to onset of acidic conditions.
- Continue two Category 4 tests (one from each current pH group) to evaluate the copper leaching trend.
- Continue testing on one ore sample to monitor long term performance of typical ore type materials.

Implementation of these recommendations will result in 14 of the current tests being discontinued. For the residues from the tests, optical mineralogical characterization is recommended to evaluate oxidation features.

Table 3. Recommended Rock Humidity Cells for Continuation

Sample	Rock Type	Waste Category	S %	Reasoning
DDH-00-334C(580-600)-14	Troctolitic	1	0.06	Higher S
DDH-00-367C(290-310)-51	Troctolitic	1	0.04	Mid-Range S
DDH-26064(44-54)-56	Troctolitic	1	0.02	Lowest S
DDH-00-350C(580-600)-17	Troctolitic	3	0.19	S<0.2, evaluate segregation criterion
DDH-00-369C(335-345)-34	Troctolitic	3	0.18	S<0.2, evaluate segregation criterion
DDH-00-367C(170-175)-54	Troctolitic	3	0.51	High S, test onset, lower pH in this group
DDH-00-357C(535-540)-23	Ultramafic	3	0.2	S=0.2, lowest pH leachate
DDH-00-371C(435-440)-19	Troctolitic	4	0.88	Acidic but pH>4
DDH-00-361C(737-749)-62	Virginia	4	2	Acidic and pH<4
P2-0-67		4	0.9	Typical of average ore

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5 Northmet Project Flotation Tailings Characterization Program

5.1 Observations

The current tailings test program consists of 66 tests in HCT and DNR reactor configurations on Flotation Tailings samples obtained from four different pilot plant runs. The longer tests have run for more than eight years and nearly 50% of sulfur has been depleted in some cases. Consistent patterns have emerged from all these programs:

- No tests have generated pH lower than the deionized water used in the tests and detectable alkalinity is still being measured.
- Sulfide oxidation rates declined as sulfur was depleted indicating that the oxidation rate is not zero order. This was evaluated previously by SRK (2012).
  - Leachate pH declined initially, reached a low point after a few years than recovered. This pH depression is interpreted as representing slightly accelerated acid generation rates. This implies that in the long term there is a low likelihood of acidic conditions developing.
- Differences in sulfur content explain differences in oxidation rates and the degree to which pH is depressed. Lower pHs and higher oxidation rates are measured for samples containing higher sulfur content.
- Lower pHs result in accelerated metal (Ni, Co) leaching though only for the Pilot Plant 1 samples.
- The DNR reactors showed similar pH trends but at lower pHs. This is interpreted as reflecting the higher liquid-to-solid ratio used in the test compared to HCTs resulting in a stronger influence from the deionized water pH.

5.2 Recommendations

The following changes to the program are recommended:

- One set of tests from each of the major programs (PP1, PP2/PP3 and Scavenger tailings) should be continued to validate long term performance. For each set of tests, the bulk, +100 mesh and -200 mesh test would be continued and the -100+200 mesh sample stopped. The sample sets having the highest sulfur content in the bulk tailings and produced in a pilot run with copper sulfate activator added would be selected for continuation.
- All DNR Reactor tests should be discontinued because they are not providing additional information beyond that of the humidity cells.

The resulting number of continuing tests will be nine. For discontinued tests, the following analyses are recommended:

- Total sulfur analysis to provide mass balance confirmation with the original sample and leached sulfate.
- Optical mineralogy to evaluate dissolution features.

6 LTVSMC Tailings Borrow Sources Characterization Program

6.1 Observations

Five samples of weathered LTVSMC tailings are being tested in humidity cells. These samples have shown stable leachate with slightly basic pH consistent with low initial sulfur content and elevated carbonate content. No significant changes in leachate pH are anticipated.

6.2 Recommendation

It is recommended that these tests be terminated because leachate chemistry is stable and no long term changes are expected.

7 NorthMet Flotation Tailings on LTVSMC Tailings Column Tests

7.1 Observations

These tests were designed to evaluate geochemical effects from placing Flotation Tailings on LTVSMC tailings. The tests are constructed as sequential columns with oxygen control at intermediate sampling points. The tests have been operating for nearly eight years.

The following are major observations from this test program:

- Two control tests containing only LTVSMC tailings (coarse and fine fractions) yielded basic pH leachate containing high alkalinity concentrations which increased along the flow path. Some differences were observed between the two tests but leachate characteristics were generally similar and consistent with pH. Long term flushing trends were apparent for a number of parameters including sulfate, arsenic and molybdenum.
- All four column sequences testing the influence of Flotation Tailings on LTVSMC tailings showed similar results.
- Leachate from the Flotation Tailings had slightly lower pH when compared to LTVSMC tailings and showed declining sulfate release indicating depletion of sulfide minerals. Overall leachate chemistry was similar to humidity cells yielding leachate pH above 7. Metal concentrations were very low and accelerated leaching comparable to lower pH humidity cells was not apparent.
- Leachate from Flotation Tailings did not appear to influence leaching of the LTVSMC tailings but the LTVSMC tailings appeared to remove arsenic, vanadium and to a lesser extent nickel from Flotation Tailings leachates (examples shown in Figure 2). This is suspected to be a result of adsorption to LTVSMC tailings.

7.2 Recommendations

Because the LTVSMC tailings may be important in limiting metal leaching from Flotation Tailings, the following continuation of tests is recommended to evaluate breakthrough effects:

- Continue one set of controls due to similar performance of both coarse and fine LTVSMC tailings. The coarse fraction has been selected. Monitor bottom port only.
- Continue the two P1 sequence tests because they are leaching highest arsenic and are likely to breakthrough sooner. Monitor top and bottom ports only.

For decommissioned tests, testing of the column residues is recommended to determine where adsorption occurred in the column.

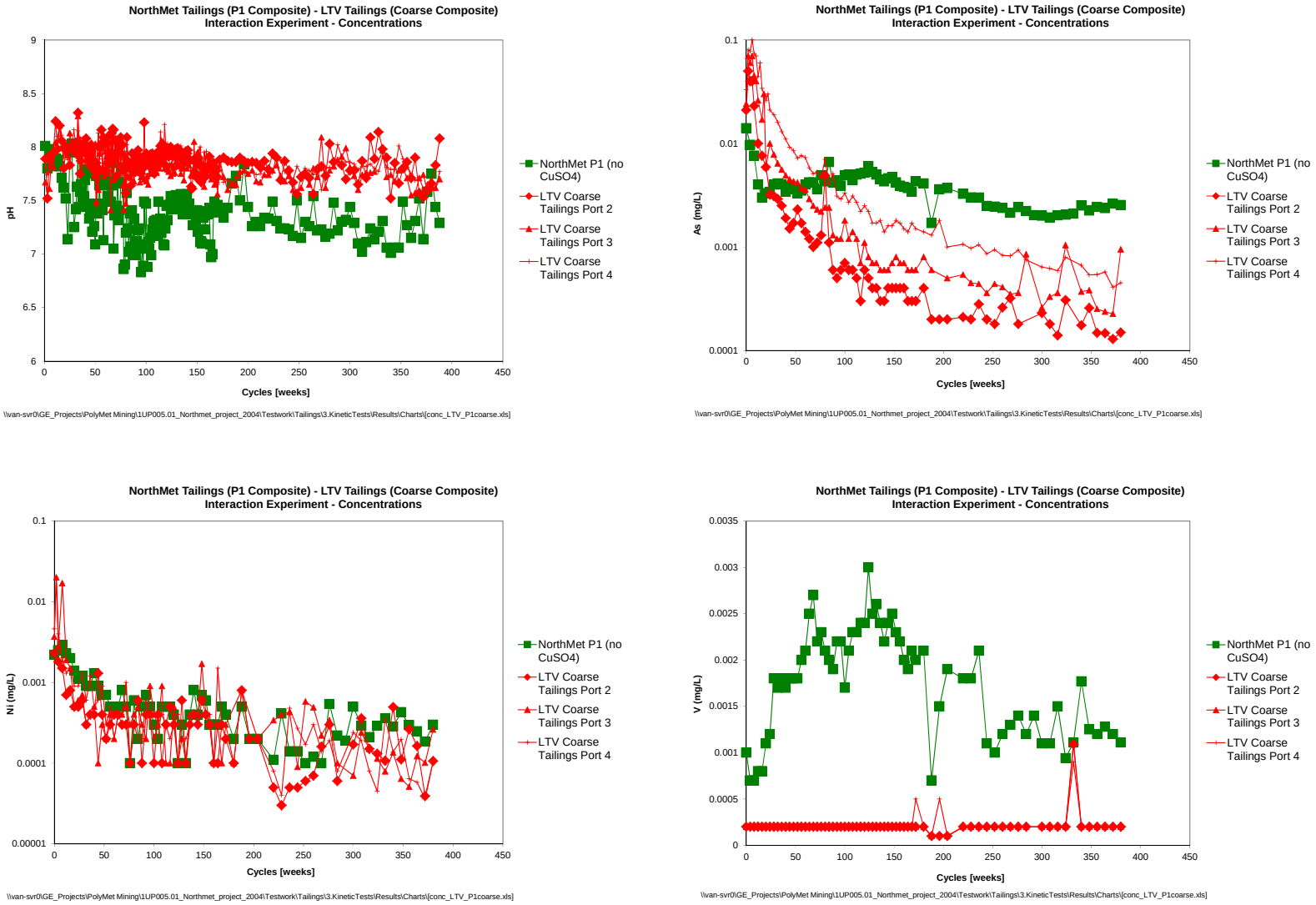

Figure 2. Examples of Leaching Trends for Flotation and LTVSMC Tailings Columns

8 Hydometallurgial Residue

8.1 Observations

One sample is being tested in humidity cell, shake flask and DNR Reactor configurations. All tests show trends consistent with flushing of soluble products. pH has stabilized at a consistent level.

8.2 Recommendation

Discontinuation of all three tests is recommended because the amount of flushing that has occurred far exceeds expected flushing under field storage conditions, and leachate chemistry is either stable or showing downward trends.

9 Conclusions

Long term (exceeding eight years in some cases) kinetic testwork for the NorthMet Project has yielded consistent results that support the interpretations made earlier in the program used to develop waste management criteria and water chemistry predictions.

As a result, SRK recommends that continuation of a limited number of tests is appropriate to validate predicted long term pH stability. In addition, based on the data obtained thus far, changes in leachate chemistry are expected to be detected with very low frequency analysis. It is recommended indicator parameters (pH, conductivity, sulfate and alkalinity) be measured quarterly and metals measured twice annually.

SRK Consulting (Canada) Inc.

"DRAFT"

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Corporate Consultant (Geochemistry)

Reviewed by:

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The opinions expressed in this report have been based on the information available to SRK at the time of preparation. SRK has exercised all due care in reviewing information supplied by others for use on this project. Whilst SRK has compared key supplied data with expected values, the accuracy of the results and conclusions from the review are entirely reliant on the accuracy and completeness of the supplied data. SRK does not accept responsibility for any errors or omissions in the supplied information, except to the extent that SRK was hired to verify the data.

10 References

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Attachment E

Physical Model Study of Fine-Grained Tailings Delta Formation under Various Flow Conditions: Project Report

Physical Model Study of Fine-Grained Tailings Delta Formation under Various Inflow Conditions: Project Report

Project Report Number 551

Prepared by

St. Anthony Falls Laboratory
University of Minnesota

For

Barr Engineering Company

March 2011

Project General Information

Title:
Physical Model Study of Fine-Grained Tailings Delta Formation under Various Inflow Conditions

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Executive Summary

The University of Minnesota's St. Anthony Falls Laboratory (SAFL) was contracted by Barr Engineering Co. to perform a series of physical laboratory models related to tailings delta formation as part of an Environmental Impact Study for the development of new copper/nickel mine in northern Minnesota. Phase I of the project involved 1D flume experiments, at field scales, to evaluate the potential for debris flow behavior and channelization as well as to develop an initial beach slope prediction matrix. In Phase II of the project, two laboratory scale deltas were grown in SAFL's delta basin. The delta experiments were designed such that the laboratory scale delta would have a similar degree of channelization as the field scale delta. The Phase II experiments also looked at the range of fines (<74 micron) retention in the deposit with focus on generated deposits that represent lower limit of fines concentration in the final surface of the deposit.

Prototype tailings were used in the Phase I study and were provided by Polymet. Phase I experiments indicated that debris flow-type behavior will not be the transport and depositional mechanism rather the delta be constructed from fluvial-braide processes and channelized. Phase I also indicated that, within the expected range of slurry discharges, the beach slope will likely range between 0.5% and 2%.

Phase II of the project focused on 2D basin experiments to evaluate grain size sorting, heterogeneity, and hydraulic conductivity in the deposit. An 5 m by 5 m by 0.4 m deep research basin located at SAFL-UMN was used for the tests. The degree of fines retention within the delta was measured from surface scrape samples taken from the beach at various times during basin operation. Scrape samples were analyzed for coarse/fine fraction by washing the samples through a 74 micron sieve. The degree of channel-lens formation in the deposit was investigated on two cross-sectional freeze slices taken from the Run 2 deposit. The Phase II results showed clear visual indication of sorting. Grain size sorting was present in the form of downstream fining (particle size segregation toward smaller grain size along the flow) and coarse/fine lenses (internal structures formed by filling abandoned flow channels with sediment). The results of Phase II suggest that the field scale delta will likely exhibit significant heterogeneity related to channelization. The field delta should have a minimum of 30% (by weight) fines retention throughout. Hydraulic conductivity measurements made on the laboratory deposits showed a decrease in conductivity with distance from the slurry source, suggesting that the groundwater transport through the delta will likely be greater at the upstream end than at the downstream end.

Prediction of the degree of water retention in the field scale deposit cannot be done conclusively. The estimated deposit thickness and the Soil-Water Characteristic Curve (SWCC) for the tailings suggests that suction (the tailings ability to wick water) will not be great enough to keep the deposit saturated; however, internal structures such as lenses, grainsize discontinuities, or micro-pore structure developed from natural deposition may increase the suction pressure of the material and decrease permeability.

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1.0 Introduction

The University of Minnesota's St. Anthony Falls Laboratory (SAFL) was contracted by Barr Engineering Co. (BARR) to perform a series of physical laboratory models related to tailings delta formation as part of an Environmental Impact Study for the development of a new copper/nickel mine in northern Minnesota. At the site, a tailings basin would be created to collect sediment from process water effluent. The tailings production would be approximately 32,000 tons/day and would discharge to the tailings basin as a 31.5% solids (by weight) slurry via a pressurized pipe. Multiple input points would be designed around the perimeter of the basin giving operators control on feed-points.

It is not feasible to reproduce, in the same experiment, both the local flow and sediment-transport conditions and a fully developed channel network. Accordingly, our strategy was to run two sets of experiments: one to study field-scale flow and sediment dynamics in a relatively narrow, long flume (Phase I); and the second to study channelization and its effect on deposit heterogeneity at a substantially reduced scale in an open basin (Phase II).

Phase I of the project involved 1D flume experiments, at field scales, to evaluate the potential for debris flow behavior and channelization as well as to develop an initial beach slope prediction matrix. These experiments were conducted in a 6-inch wide glass-walled flume with a metered slurry input at the proximal end and a pooled tailbox at the distal end.

Phase II of the project focused on 2D basin experiments to evaluate grain size sorting, heterogeneity, and hydraulic conductivity of the deposit. The goals of the 2D experiments were to 1) determine the expected lower limit of fines concentration in the deposit, 2) determine the degree of grain size segregation both vertically and horizontally within the deposit, and 3) evaluate the potential range of hydraulic conductivity throughout the model delta.

Phase II was conducted in a rectangular research basin located at SAFL. The research basin is designed for studying deltas and tailings ponds and is referred to as DeltaBasin2. Figure 1 is a schematic of the delta basin. The delta basin had inlet controls, pool level controls, overhead camera, and a topographic scanning system. The data collected for Phase II included grain coarse/fine fraction, hydraulic conductivity, topography, aerial images, freeze slices, and lens grain size distributions. The experimental facility, setup, and data collection is discussed in more detail in the sections below.

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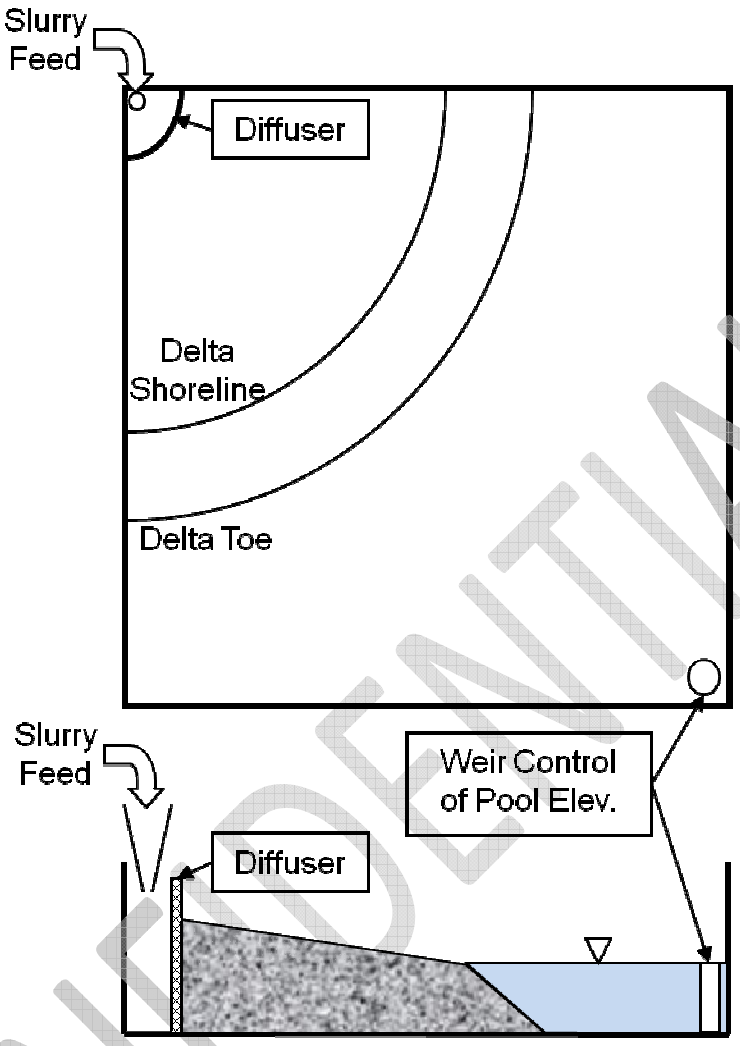

Figure 1 – Schematic of the delta basin. Top is plan view of the basin. Bottom is section view.

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2.0 Tailings Supply Information

At the field site, the tailings would be delivered to the basin in the form of a high concentration slurry. Under this design scenario, the slurry is expected to deposit sediment to create a sloping subaerial deposit termed the "beach". The planned slurry flow rate is 31.1 cfs with a solids fraction of 31.5% by weight (Table 1). The slurry would be delivered to the tailings basin at several locations, each location forming a radial fan with a spatial scale on the order of hundreds of feet. Laboratory testing focused on a single feed point.

The standard definition for the division between clay and silt size particles and between silt and sand size particles is 0.005 mm and 0.075 mm. The tailings material used in this study was prototype material provided by Polymet and was composed of fine sand to clay-sized particles, with a D_{50} (median grain size) of 60 microns (See Appendix A for Soil Engineering Testing, Inc. grain size distribution). By weight, the material averages around 41% (between 35 and 48%) of the tailings sediments were greater than 74 microns (also used as the division for "fine- " or "coarse-grained" material). Laser diffraction analysis of the tailings indicate that, although the material had many clay-sized particles, it did not contain much mineral clay.

It should be noted that PolyMet provided two grind types (grain size distributions) for their tailings however the difference between the two was minor. The grain sizes reported here are for the final grind. The preliminary grind was similar with 32% of the sediment greater than 74 microns by weight. Due to a limited supply of tailings from the pilot plant, the first run and 1D flume testing was conducted with the preliminary grind tailings supply, and the second run was conducted with the final grind tailings supply. It is believed that the two grinds are similar enough that they will produce similar delta characteristics.

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Table 1 – Slurry Source Information

Item	Qty	Unit
Solids Production*	1452	tons/hour
Tailings Production (wt)	34848	tons/day
Tailings Production (wt)	806.7	lb/sec
Liquor Flow*	3161	tons/hour
Liquor Flow	75864	tons/day
Liquor Flow	1756.1	lb/sec
Slurry Flow*	4614	tons/hour
Slurry Flow	110736	tons/day
Slurry Flow	2563.3	lb/sec
Solids Fraction by wt in slurry*	31.5	% wt
Specific Gravity of Solids*	3	
Specific Weight of Solids	187.2	lb/ft^3
Specific Gravity of Slurry*	1.322	
Specific Weight of Slurry	82.5	lb/ft^3
Tailings (Solids) Production (volume)	4.3	ft^3/sec
Water Flow Rate (volume)	28.1	ft^3/sec
Volumetric Flow Rate for Slurry	31.1	ft^3/sec
Volumetric Flow Rate for Slurry*	13947	gal/min
Solids Fraction by Volume	13.90%	% vol

* Value provided by Barr Engineering.

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3.0 Phase I – 1D Flume Experiments

3.1 Phase I Experimental Design

3.1.1 Design Theory

The 1D flume experiment was run approximately at field scale. The sediment and water flows in the flume were adjusted to model different locations on the tailings beach. When Phase I was set up, we had no way of knowing the general nature of the flow regime to be expected on the beach, i.e. channelized fluvial versus some form of mass or debris flow. One of the main goals of Phase I was to determine this. For design purposes, we assumed that the tailings beach would have sheet flow (i.e. no flow channelization or braiding) over a 180° fan. On a radial beach, the water flow spreads out as it flows away from the inlet. Although the total water discharge across the delta does not change with radial position, the unit discharge (flow per unit width) decreases as the flow spreads. The relationship between radial position and unit discharge is given by equations 1 and 2 (Figure 2). Once the unit discharge has been determined for any given radius, the required equivalent flume water discharge can be determined by multiplying by the flume width (6 inches). Results from these calculations are included in Table 2.

$$L(r) = \pi * r \qquad (1)$$

$$q(r) = \frac{Q_w}{L(r)} \qquad (2)$$

Where: Q_w = Water discharge at inlet

r = Radius

L = Arc length of the delta

q = Unit discharge of water

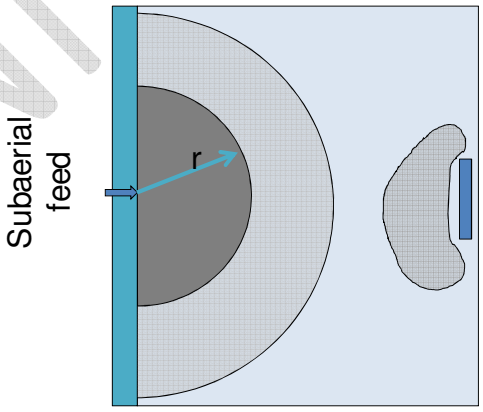

Figure 2 – Diagram of 180° delta fan.

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Table 2 – Radial Position and Model Discharge

r (ft)	L (ft)	qw (cfs/ft)	Model (cfs)	Model (GPM)
50	157	0.198	0.0990	44.40
100	314	0.099	0.0495	22.20
150	471	0.066	0.0330	14.80
200	628	0.049	0.0247	11.10
250	785	0.040	0.0198	8.90
300	942	0.033	0.0165	7.40
350	1100	0.028	0.0141	6.30
400	1257	0.025	0.0124	5.60
450	1414	0.022	0.0110	4.90
500	1571	0.020	0.0099	4.40
550	1728	0.018	0.0090	4.00
600	1885	0.016	0.0082	3.70
650	2042	0.015	0.0076	3.40
700	2199	0.014	0.0071	3.20
750	2356	0.013	0.0066	3.00
800	2513	0.012	0.0062	2.80

Like the water discharge, the sediment discharge also decreases with distance from the inlet. This is due not only to spreading of the flow across delta but more importantly to sediment being deposited over the length of the delta. In other words, the sediment unit discharge and concentration decrease with increasing radial position.

Phase I experiments focused on observing the character of the flow and quantifying slopes of the delta for various water discharge and sediment concentrations using prototype tailing provided by PolyMet. The experiments provided a clear picture of the flow regime to be expected and an understanding of the range of possible slopes and sediment concentrations as functions of radial position for the tailing beach being modeled.

3.1.2 Phase I Apparatus

The testing setup for the 1D study consisted of a mixing tank, a six-inch flume, and a tail box. The mixing tank was a 220-gallon stainless steel cylindrical tank with a conical bottom. A 0.25 hp Lightin© tank mixer with a seven-inch propeller was used to keep the solids fraction of the slurry in suspension. The outlet of the mixing tank was 8.5 feet above the inlet of the flume. Slurry was conveyed from the mixing tank to the flume inlet via 24 feet of 2-inch pvc pipe. There were ball valves at each end of the pipe and one gate valve at the downstream end to control the flow rate. The flow rate was measured using an inline Seametrics© EX-81 Electromagnetic Flow Sensor. The flume was 6 inches wide by 22 feet long with no slope and glass walls. Figure 3 is an image of the upstream half of the flume. After this picture was taken a diffuser grate was added two feet downstream of the inlet. The tail box was design to capture all of the effluent from the flume. The tail box had 400 gallons of storage below its outlet. Effluent

water was stored in the tail box until all fine material dropped out of suspension, at which point excess water was siphoned off and the remaining solids were removed.

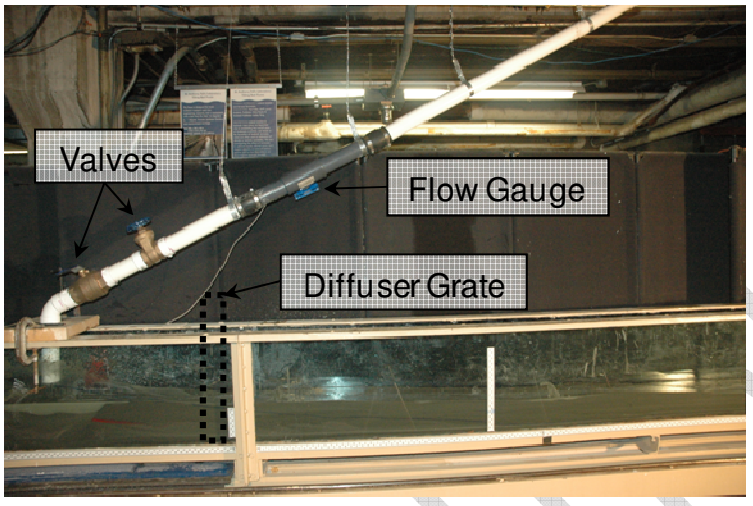

Figure 3 – Image of flume inlet.

The electromagnetic flow meter measurement was compared with that derived using a sharp crested weir placed at the downstream end of the empty flume. The flow sensor reported a discharge of 0.30 gallons/sec. The weir was 2 7/8 inches tall by 6.0 inches wide. The flow depth over the weir was 1.0 inches. Using a sharp crested weir equation, this yields a flow rate of 0.301 gallons/sec.

3.1.3 Phase I Experimental Procedure

Given the argument above that sediment concentration is expected to decrease down the beach, runs were conducted for a high and low solids fraction and slurry flow rates ranging from 5 to 60 GPM. Each run continued until the delta slope for that concentration and discharge was at equilibrium. One batch of slurry mix contained enough slurry for several runs. Runs were conducted in series starting with a high discharge and reducing for each consecutive run. The reducing discharge resulted in steeper slopes for each consecutive run, meaning that all runs were aggradational. This allowed the deltas from consecutive runs to be built on top of each other.

3.1.4 Phase I Sampling Plan

Sample locations and types are depicted in Figure 4. During the runs, grab samples were taken at the inlet and outlet of the flume. Siphon samples were also taken from the flow over the delta. Grab and siphon samples were measured for solids fraction. The upstream grab samples were used only to confirm influent solids fraction. After each run the deposit profile was measured using a point gauge. Shallow (0.4 in) scrape samples were taken from the bed. These samples were sieved to determine the grain size distribution and coarse fraction. Mini-core samples were also taken from the top 0.8 in of the bed. These samples were taken with a known volume and used to measure the porosity of the bed.

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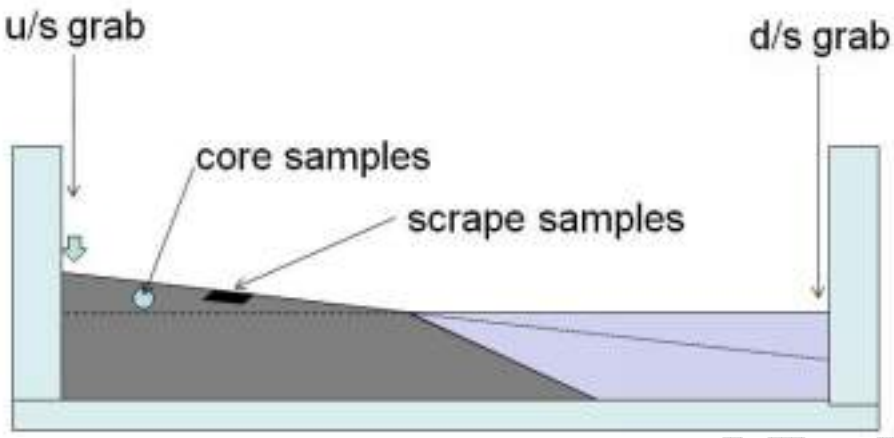

Figure 4 – Sample locations

3.2 Phase I Results

3.2.1 Qualitative Observations

Of the 16 flume runs conducted, none exhibited debris ("mud") flow behavior. Tailings were transported by the moving water as bedload and suspended load as is typical in fluvial systems. Surface flow showed a tendency to channelize and braid, even in the relatively narrow flume. Figure 5 shows cross-flow and asymmetric bedforms within the flume. These observations indicate strongly that the field scale tailings beach will also operate in a fluvial, braided and channelized regime and addresses the first two objectives of Phase I (evaluate the potential for debris flow behavior and channelization) by ruling out concerns of more complicated non-Newtonian rheologies (i.e. mud-like behaviors) and flow.

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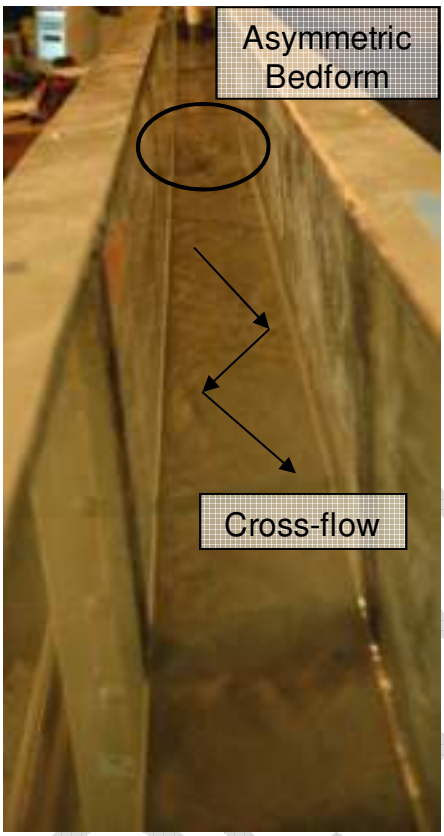

Figure 5 – Flume picture looking upstream

Nearly all runs also exhibited upstream migrating antidunes. Antidunes occur when the flow is near critical and are present in other tailings basins. They mix the upper layer of the deposit and have the potential to resuspend fine particles. A video of an upstream migrating antidune is included in Appendix D.

3.2.2 Slope Measurements

Final bed surface slopes were recorded after each test. Recall that each test provides a slope associated with a different position on the beach and is not by itself an actual beach profile. The run results have been binned into two categories - low and high solids fractions. Figures 6 and 7 show the bed profiles based on solids fraction. In both categories the slope decreased with increasing flow rates. The large spikes in the profiles are due to antidunes. The profiles for the higher discharge tests have more small spikes. These are due to the cross-flow (depicted in Figure 5) forming alternating dunes within the flume deposit. Figure 8 summarizes the slope results for Phase I showing deposit slopes for different discharges and solids fractions. Except very near the inlet, the beach slope should range between 0.5% and 2%.

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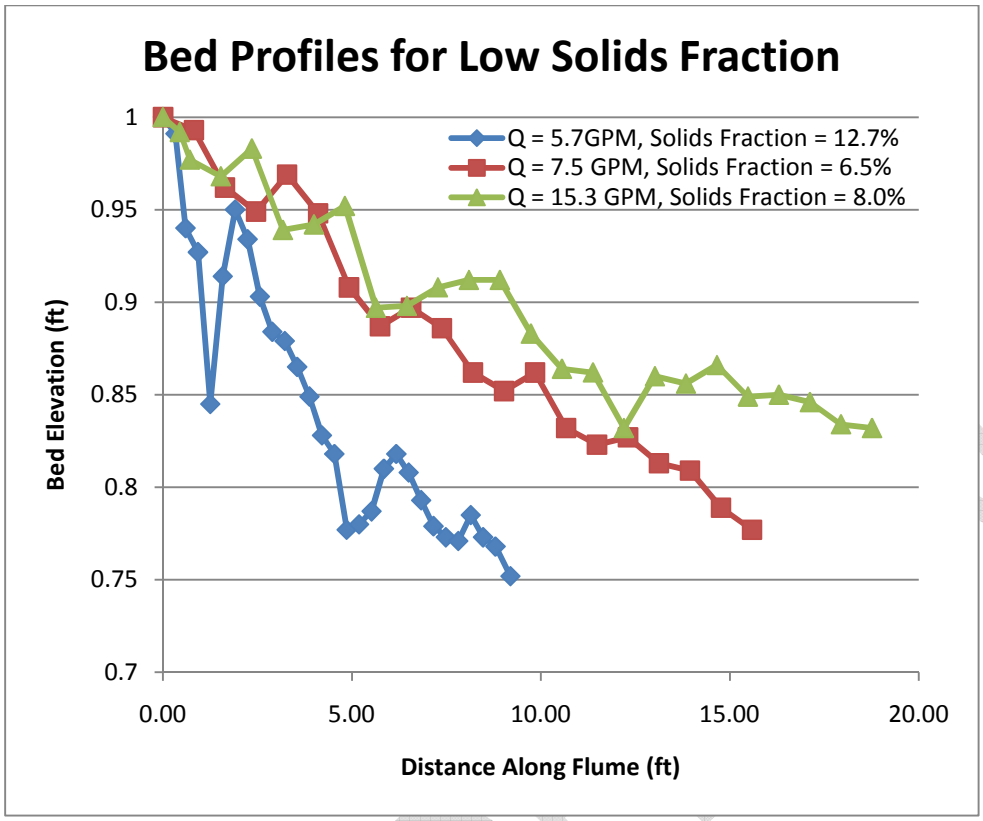

Figure 6 – Bed profiles for the low solids fraction runs

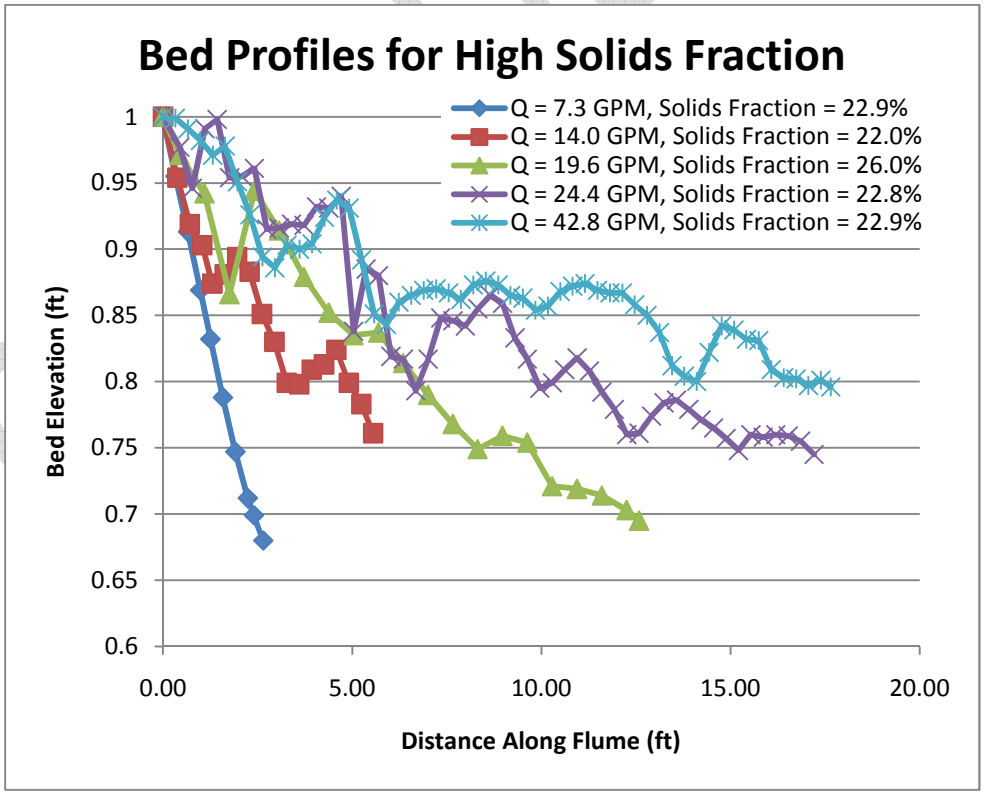

Figure 7 – Bed profiles for the high solids fraction runs

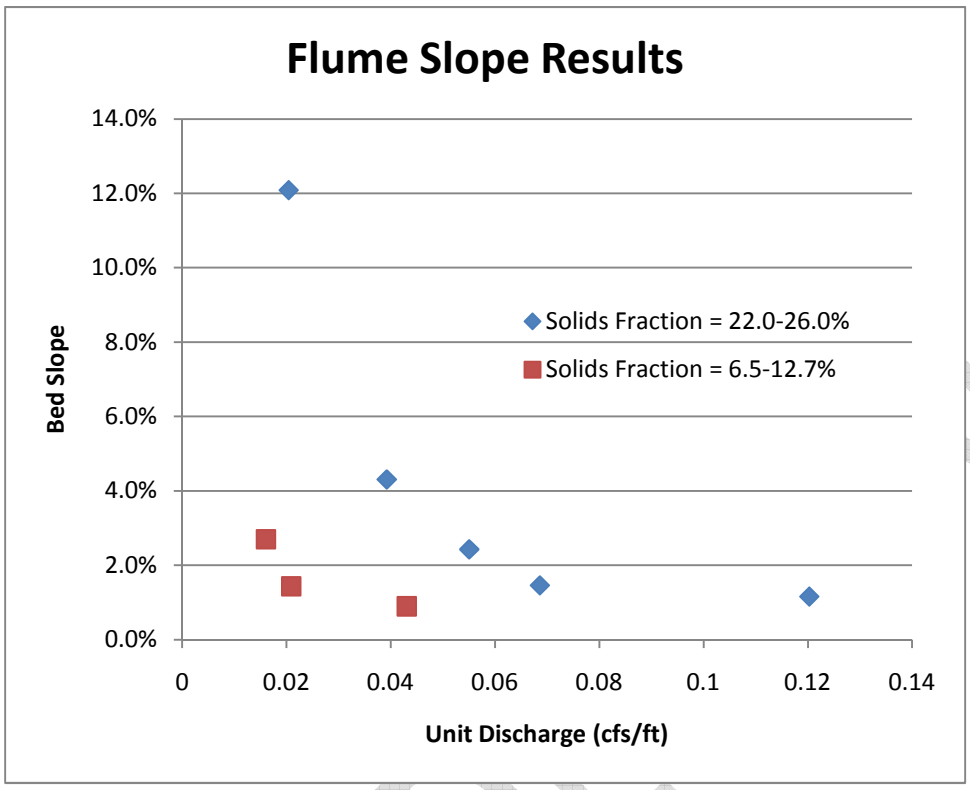

Figure 8 – Slope vs Discharge plots for high and low solids fractions.

3.2.3 Sediment Measurements

Downstream grab samples were taken from several of the runs. These samples were analyzed for solids fraction (Table 3). Samples taken at the downstream end of the flume consisted of suspended load only.

Table 3 – Downstream Solids Fraction

Q (GPM)	Feed Solids Fraction	Down Stream Solids Fraction
6.1	19.00%	6.30%
6.1	19.00%	7.00%
7.5	6.50%	2.70%
7.5	6.50%	3.50%
14.3	11.50%	2.90%
14.3	11.50%	3.00%
19.6	26.00%	4.30%
19.6	26.00%	6.10%

Porosity was measured at the end of four runs (Figure 9) using mini-core samples. A thin-walled, sharp-edged mini-corer was first inserted 0.8 in into the deposit. The deposit was then excavated away from around the mini-core and a blade was slid under the base of the mini-core to remove a

known volume of saturated deposit. The samples were then weighed saturated and dry to determine water weight. The water weight was then used to determine void volume and porosity. There does not appear to be a strong trend in porosity. The variability in porosity is likely driven by the localized flow phenomena occurring at the end of the test such as bedforms. The measured values for these samples were similar to the values measured for the 2D experiments (§5.3).

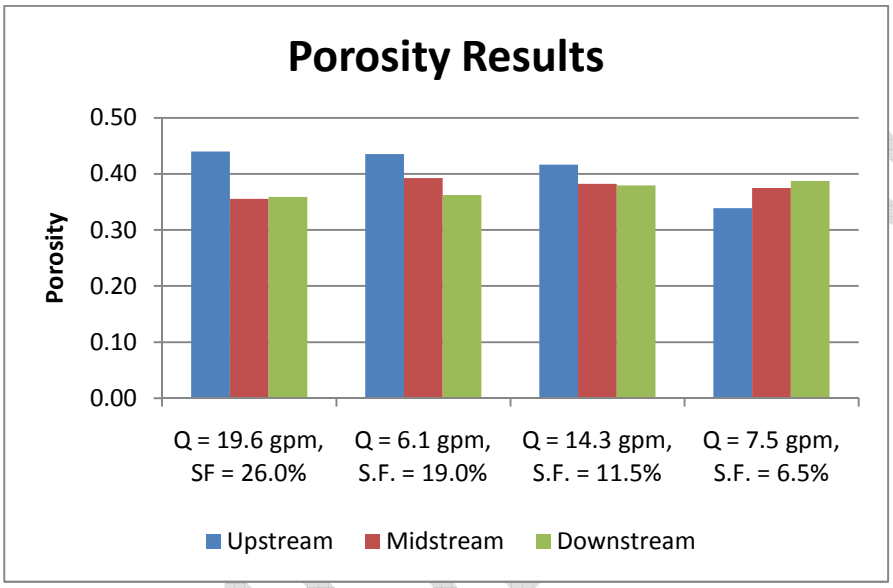

Figure 9 – 1D Experiment Porosity Measurements

Surface grain size was measured at various locations of the bed surface using surface scrape sampling method. The method involved sliding a blade 0.4 in below the surface to remove a 1.6 in square surface sample. Grain size distributions were measured for the 1D experiment scrape samples (Figure 10) by sieving. A comparison of the scrape samples to the fine tail of the bulk material showed that the deposit lost material in the 10 to 100 micron size range. This is the size range that remained in suspension and was transported out of the flume. Table 4 provides the coarse and fine fractions (above and below 74 microns) for the Figure 10 data. The coarse fraction is the total weight of sample retained on the 74 micron or larger sieve. The fine fraction is the total weight of sample passing the 74 micron sieve.

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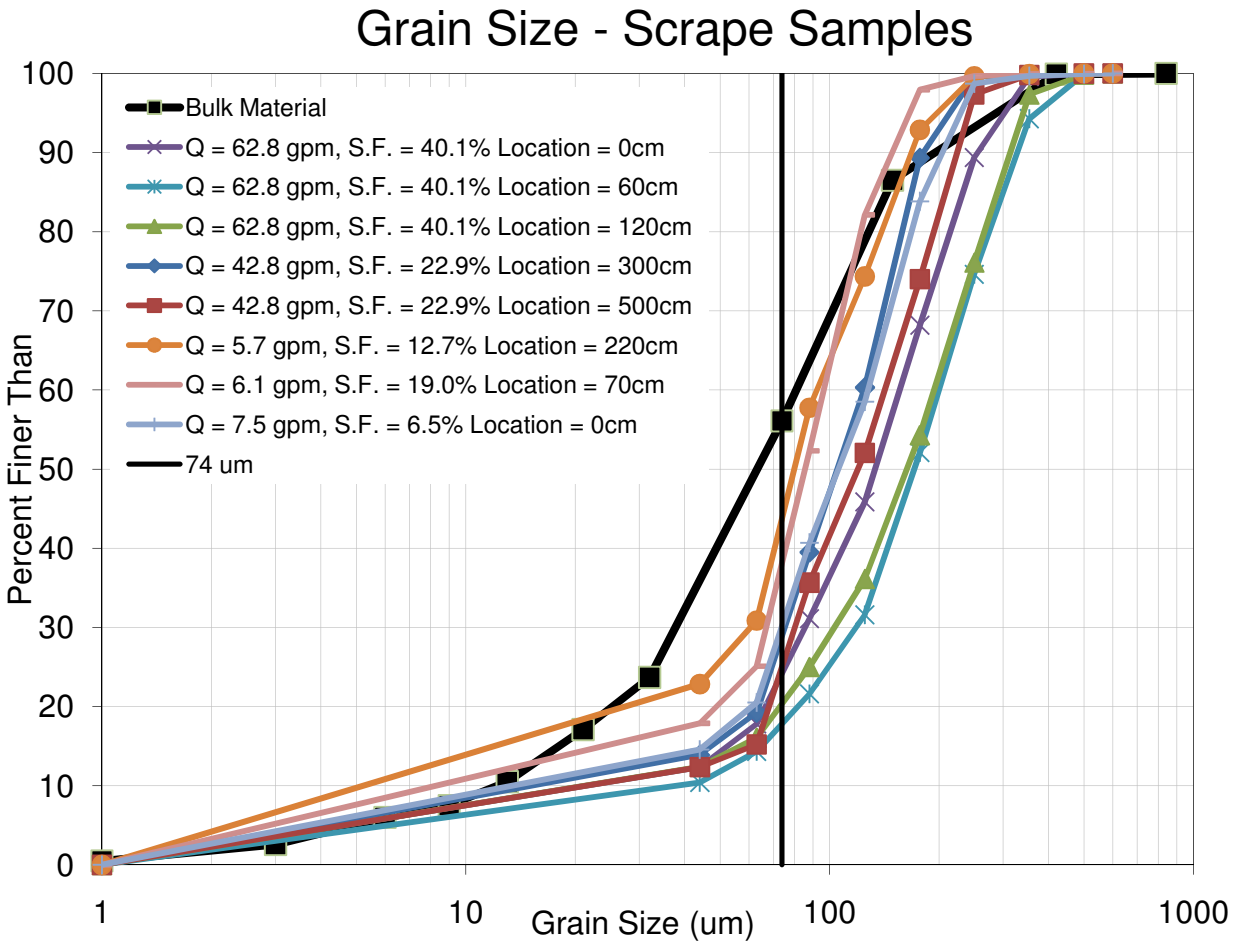

Figure 10 – 1D Experiment Scrape Sample Grain Size Distribution

Table 4 – 1D Experiment Scrape Sample Coarse and Fine Fractions (by Wt)

Discharge (GPM)	Solids Fraction	Distance from outlet	Coarse Fraction (%)	Fine Fraction (%)
62.8	40.1%	0 cm	76.3%	23.7%
62.8	40.1%	60 cm	82.5%	17.5%
62.8	40.1%	120 cm	80.0%	20.0%
42.8	22.9%	300 cm	71.8%	28.2%
42.8	22.9%	500 cm	75.8%	24.2%
5.7	12.7%	220 cm	57.3%	42.7%
6.1	19.0%	70 cm	62.9%	37.1%
7.5	6.5%	0 cm	70.6%	29.4%

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3.3 Phase I Experiment Conclusions

Phase I experiments provided the opportunity to observe the flow characteristics of the prototype material under conditions of sediment concentration and unit discharge similar to those expected in the field. The tests indicate that the beach will not exhibit mass-flow (mud like) behavior and that the full 2D beach will be channelized, with a braided network. Bedforms and channelization will likely play a large role in the variability of fines retention, porosity, and permeability. We expect this would be true over the length of a field-scale beach as well and that the channelization will create heterogeneity that will affect the bulk permeability and conductivity of the deposit. Within the expected range of discharges and solids fractions the delta beach will have a slope between 0.5% and 2%. At high concentration near the inlet to the basin, local slopes could reach 4-6%.

The tests show that a fraction of fine material is transported beyond and not retained in the deposit. In the region of higher unit discharge near the inlet, the deposit contains a lower fraction of fine material than the near the shoreline of fan where lower unit discharge make it easier for finer grains to be deposited; however, the test data suggest that even at high discharges 15-30% (by wt) of the deposit is comprised of material sizes <74 micron.

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4.0 Phase II – 2D Basin Experiments

Phase II of the project focused on a two dimensional physical model of a single delta using prototype tailing to observe processes and to evaluate grain size sorting and hydraulic conductivity of the deposit. The goals of the 2D experiments were to 1) determine the expected lower limit of fines concentration in the deposit, 2) determine the degree of grain size segregation both vertically and horizontally within the deposit and 3) evaluate the potential range of hydraulic conductivity throughout the model delta.

Phase II experiments were performed as "scaled" experiments using the approach described below. Experiments were conducted in a specially designed "Delta Basin" at SAFL, which provided access to several precision data acquisition tools.

4.1 Scaling Approach

Phase I experiments used field-scale unit discharges and actual tailings material to investigate flow and transport processes and deposit characteristics. Given the conclusion from Phase I that the flow would be channelized on the beach top, in Phase II the goal was to employ a reduced-scale modeling approach to investigate the formation and behavior of channels on the tailings beach without lateral constraints. Again, the prototype tailings material was used.

It is obviously not feasible to conduct full scale experiments on large deltas and therefore these experiments were done at substantially reduced scale. SAFL has over 15 years of experience in physical experiments of deltaic systems and this experience was applied in this project. The scaling methods used in projects of this type differ from traditional hydraulic physical models where near-exact geometric scaling and dynamic scaling of flow is possible. The scaling approach adopted for this project sought to provide similarity in Froude number (Froude scaling), general sediment-transport regime, and the ratio between normal flow depth to radial width of the delta. This ratio is defined as the aspect ratio, A. It has been shown that the aspect ratio is a predictor of channel morphology such as braiding, meandering or straight (Parker, 1976). This work also showed that for the low aspect ratio/high slope regime expected on the tailings beach, the degree of braiding is relatively insensitive to the exact value of A as long as A is sufficiently high (a few hundred or more). To determine input parameters for the laboratory delta, we made estimates of the aspect ratio for the field scale beach and used these aspect ratios to help set discharge and concentration parameters for the lab experiments.

For the experimental design, the delta is idealized with normal, sheet flow over a 90° opening angle. This assumption means that the flow width is equal to the arc length of the delta at any radial position, yielding for the aspect ratio.

$$A = \frac{H}{L(r)} \qquad (3)$$

Where: A = Aspect ratio

H = Flow depth

L = Arc length at a given radial position (r)

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r = radial position

The first step for determining the field aspect ratio is to determine the estimated flow depth in the field. Water unit discharge along the delta is described by:

$$q_w = \frac{2Q_w}{\pi r} \quad (4)$$

Where: q_w = Water unit discharge

Q_w = Total water discharge

Next the boundary shear stress can be derived from the normal flow assumption (Eq. 5) and by fluid drag (Eq. 6). A continuity equation relating unit discharge and flow velocity is also needed (Eq. 7).

$$\tau_b = \rho g H S \quad (5)$$

$$\tau_b = \rho U^2 C_f \quad (6)$$

$$q_w = UH \quad (7)$$

Where: τ_b = Boundary shear stress

ρ = Density of water

g = Acceleration of gravity

S = Bed slope

U = Average flow velocity

C_f = Coefficient of drag

Combining equations 5, 6 and 7 yields an expression for the flow depth (Eq. 8) which can be used to estimate the flow depth in the field. Equation 3 can then be used to determine the aspect ratio in the field.

$$H = \left(\frac{q_w^2 C_f}{gS}\right)^{\frac{1}{3}} \quad (8)$$

Table 5 shows the predicted flow depths and aspect ratios for the field. The same methods can be used to determine the aspect ratio for the experimental case. The aspect ratio of the experimental delta's shoreline is then used to estimate the equivalent radial position of the field scale beach. Run 1 had a water discharge of 4.90×10^{-4} m^3/s (1.73×10^{-2} cfs) and a sediment discharge of 7.35×10^{-5} m^3/s (2.60×10^{-3} cfs), which resulted in an aspect ratio of 1500 at the shoreline (r ~ 1.5 m, 5 ft). This translates to a radial position of about 40 m (130 ft) on the field delta. Run 2 had a water discharge of 5.18×10^{-4} m^3/s (1.83×10^{-2} cfs) and a sediment discharge of 7.77×10^{-5} m^3/s (2.74×10^{-3} cfs) which yields an aspect ratio of 3700 at the shoreline (r~2.5m, 8.2ft). This translates to a radial position of about 70 m (230ft) on the field delta.

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Table 5 – Aspect ratios for the field delta with a water discharge of 0.80 m^3/s (28.1 cfs) and a sediment discharge of 0.12 m^3/s (4.3 cfs).

r (m)	L (m)	H (m)	A
5	3.9	0.108	36
10	7.9	0.062	126
15	11.8	0.045	262
20	15.7	0.036	440
25	19.6	0.030	657
30	23.6	0.026	912
35	27.5	0.023	1204
40	31.4	0.021	1531
45	35.3	0.019	1892
50	39.3	0.017	2287
55	43.2	0.016	2715
60	47.1	0.015	3176
65	51.1	0.014	3668
70	55	0.013	4192
75	58.9	0.012	4746
80	62.8	0.012	5330
85	66.8	0.011	5945
90	70.7	0.011	6589
95	74.6	0.010	7263
100	78.5	0.0099	7965
105	82.5	0.0095	8696
110	86.4	0.0091	9456
115	90.3	0.0088	10244
120	94.2	0.0085	11059
125	98.2	0.0082	11902
130	102.1	0.0080	12773
135	106	0.0078	13671
140	110	0.0075	14596
145	113.9	0.0073	15547
150	117.8	0.0071	16526

4.2 Experimental Setup

The facility used in Phase II was an existing delta basin at SAFL (Figure 1). The delta basin is square, 5 m (16.4ft) on a side, and 40 cm (1.3ft) deep. Water and sediment were fed into one corner of the basin at a constant rate for each experiment. Prototype tailings were used as the sediment for the tests and were provided by the sponsor. Dry tailings material was fed using an auger-style sediment feeder and feeder discharge was calibrated using a capture and weigh technique.

City water was used for all experiments and the water feed rate was controlled by a gate valve and a rotameter flow meter. The water and sediment were allowed to mix in a funnel before discharging into the basin. The pool elevation in the model tailings basin was set by a computer-controlled siphon and weir that were adjusted at one-minute intervals and provided precise control of the water surface elevation throughout each experiment.

Prior to beginning the tests a drainage layer was placed in the basin that was composed of fine to medium sand. The depth of the drainage layer was 3 cm (1in) and it extended radially out from the source for 3m (10ft). This layer is typically installed in all SAFL experimental deltas to promote dewatering of the deposit post-run and to shorten drying time required before the deposit can be sectioned.

The basin was equipped with a data collection carriage including a laser scanner that can measure topography accurate to 0.5 mm vertically. The carriage was used to map surface topography throughout the testing. A digital SLR camera, mounted above the basin, was calibrated for optical distortion and used to collected time-lapse images of the delta formation and surface processes.

4.3 Experimental Procedure

Two deltas were constructed in Phase II (Run 1 and Run 2). Each delta was constructed in three phases. The first phase (growth phase) modeled the initial formation of a delta into a sediment free basin with a stationary pool elevation of 30 mm (1.2in) (Figure 11). During this phase the delta grew out to a radius of approximately 1.5 meters (5ft). The second phase (building phase) involved slowly raising the pool elevation such that the shoreline position was constant (Figure 12). This continued until the delta thickness increased by a total of 10 cm (4in). The third phase involved a slow decrease in pool elevation in order to promote delivery of tailing to the shoreline position and to minimize deposition on the fan surface except for the coarsest material (Figure 13).The third phase (falling phase) of each experiment was designed to generate the coarsest possible deposit, with the strongest lateral segregation of material possible. The goal was to understand how coarse and permeable the delta deposit could be for a "worst case" transport scenario. Analysis of the third phase deposit provides a reference case for permeability and other characteristics of a deposit created under the most extreme conditions of sorting and coarse-sediment retention.

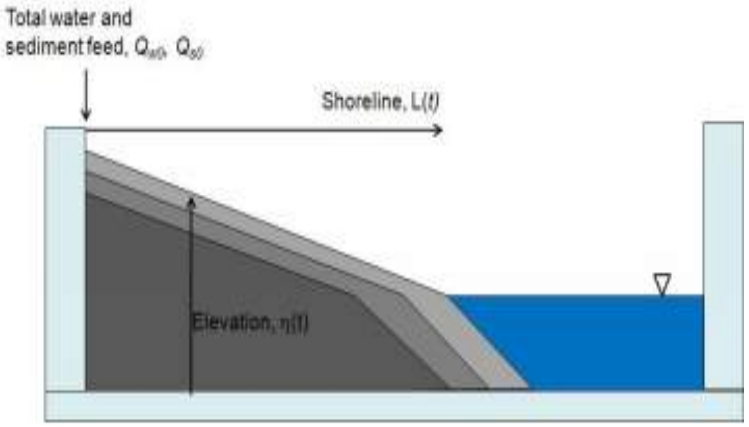

Figure 11- Delta growth with constant pool elevation

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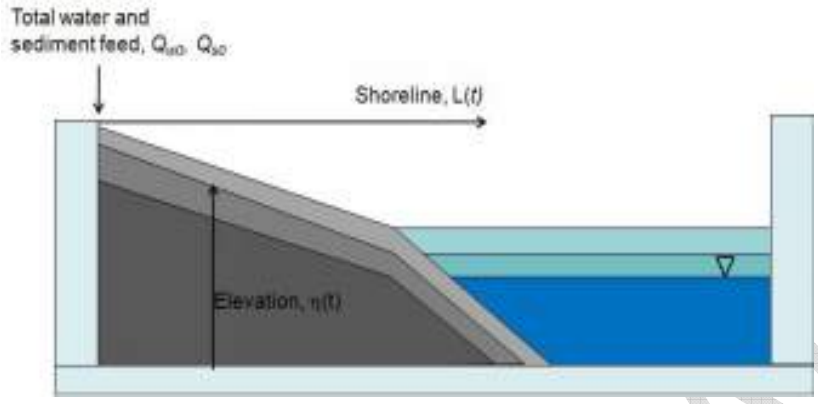

Figure 12 – Delta growth with fixed shoreline by balanced pool rise

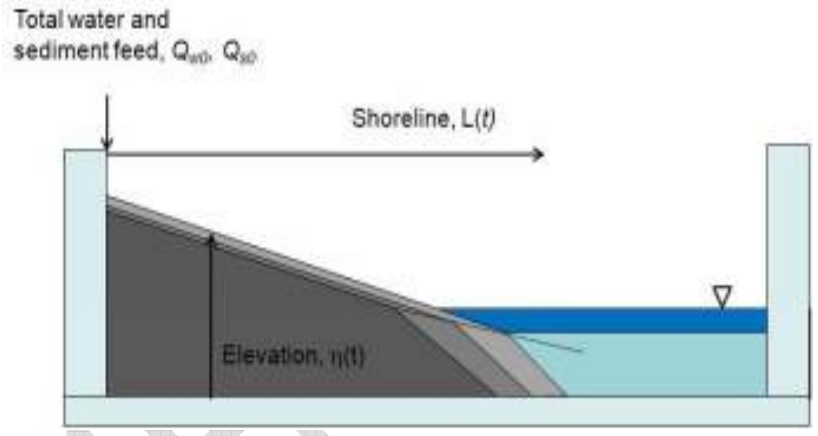

Figure 13 – Delta growth with falling pool elevation

Using the scaling approach described above, for Run 1 the water discharge was set at 0.49 liters/second (0.017 cfs). Sediment concentration was set equal to prototype design conditions giving a sediment supply rate of 0.074 liters/second (0.0026 cfs). For Run 2 the water discharge was set at 0.51 liters/second (0.018cfs) and sediment discharge at 0.078 liters/second (0.0028cfs).

4.4 Phase II Data Collection

Topographic Scans

A three axis data carriage was used to scan the surface of the subaerial deposit at the end of each pool control phase (growth, building and falling). Data are collected with a laser-based distance meter accurate to 0.5mm vertically. Scans were done on a 2mm x 2mm horizontal grid. Data were post processed and are presented later in the report.

Pool and Scrape Samples

Forty-five scrape samples were taken along 5 radial lines. Scrape samples covered approximately 10 cm^2 (1.5 in^2) of the surface and 1 cm (0.4 in) of depth (Figure 14). Three scrape samples were

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taken from the bottom set (bottom of the pool downstream of delta) of the delta deposit for grain size analysis.

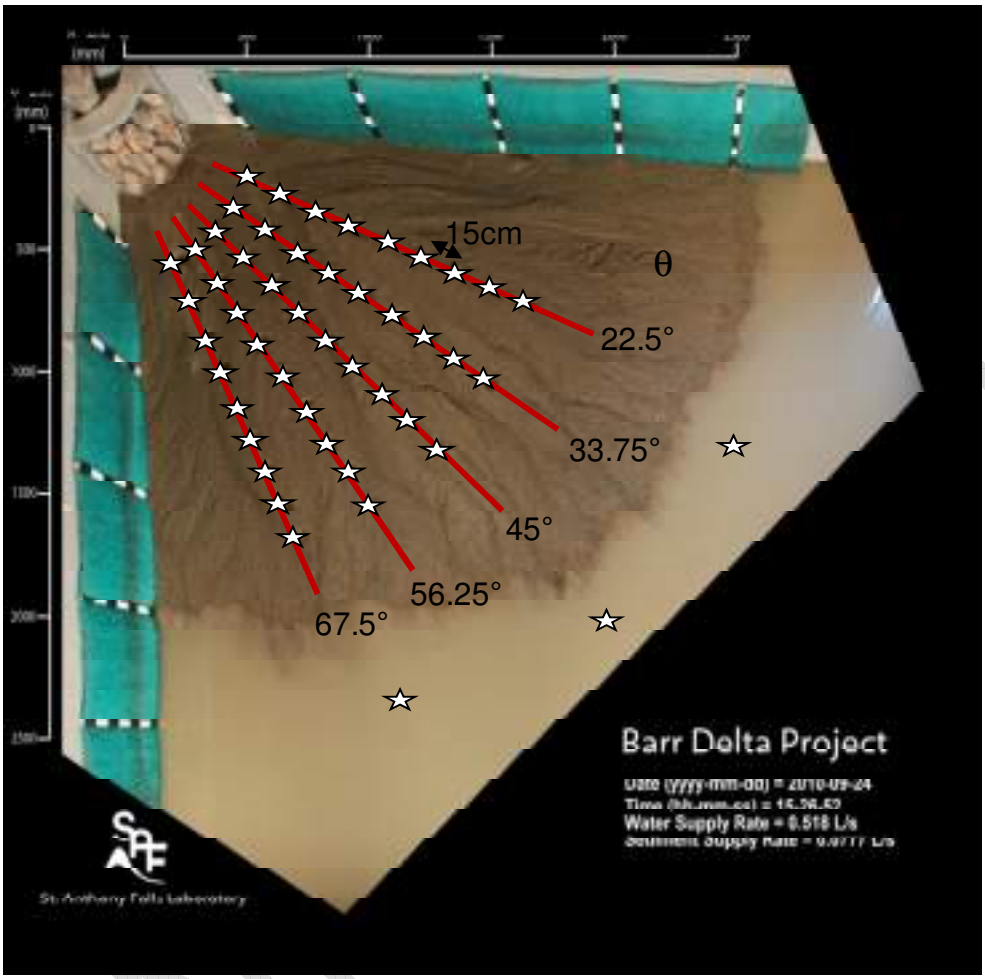

Figure 14 – Scrape sample locations

Digital Photographs

A digital SLR camera mounted above the basin and connected to a computer allowed continuous time-lapse documentation of the delta surface. Images were collected at a rate of 3 per minute. Images were post processed to correct for distortion.

Porosity Mini-Cores

At the end of each run, 6 mini-cores (2 cm, 0.75in deep) were taken to determine the porosity of the surface layer. A thin-walled, sharp-edged mini-core was first inserted 2 cm (0.75in) into the deposit. The deposit was then excavated away from around the mini-core and a blade was slid under the base of the mini-core to remove a known volume of saturated deposit. The samples were then weighed saturated and dry to determine water weight. The water weight was then used to determine void volume and porosity. Of these 6 mini-cores, three were taken at the upstream end of the delta – one each in the main channel, an old channel, and out of the channel. The other three were taken in the same locations at the downstream end of the deposit.

Freeze Slices

To record deposit structure, a 20-inch wide steel wedge with a vertical face on the upstream edge was inserted into the saturated deposit (Figure 15). The wedge was filled with dry ice and methanol. The two substances react to rapidly freeze the surrounding one-half inch of sediment to the wedge. When the wedge is removed, the frozen sediment is removed with it. The wedge is then filled with room temperature water. This delaminates the freeze slice from the working face of the wedge. At this stage the working face of the frozen sediment slice has residual ice buildup due to contact with the freeze core. To remove imperfections due to ice buildup, the working face is heated with a heat gun. This leaves one-eighth inch of slurry on top of three-eighths inch frozen sediment slice. The slurry is removed with a blade and the surface is brushed with a feather duster to remove any displaced particles from the blade. The remaining frozen slice of sediment preserves the delta stratigraphy and is ready to be photographed. For Run 2 two freeze slices were taken transverse to flow at 55 cm (1.8ft) and 150 cm (5ft) downstream of the feed point. These slices were photographed and subsampled for grain size analysis.

Figure 15a – Freeze slice apparatus.

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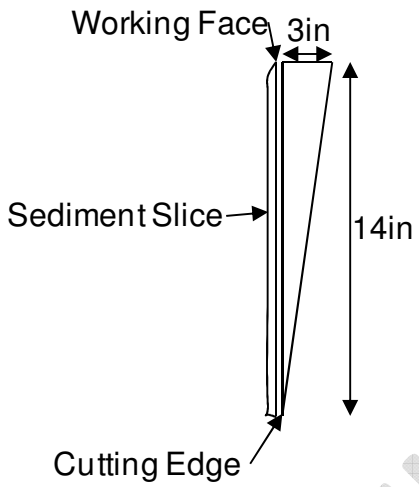

Figure 15b – Freeze Slice apparatus

Hydraulic conductivity samples

For Run 1 twenty full-depth piston cores were also taken along 3 radial lines within the beach. The hydraulic conductivity was tested on the cores using a rigid-walled, falling-head analysis. The benefit of this method is that the core cylinders can be inserted directly into the test apparatus without transferring the samples. Additionally, the percentage of fines in these samples exceeded the limits for the traditional constant head tests, and the hydraulic conductivities also exceed the limits for the traditional flexible-walled permeameter tests.

The test equations (Eq. 9 & 10) for the rigid-walled, falling head analysis are the same as for the flexible-walled permeameter.

$$k = \frac{aL}{At}\ln\left(\frac{h_1}{h_2}\right) \qquad (9)$$

$$\kappa = k\frac{\mu}{\rho g} \qquad (10)$$

Where:
- k = Hydraulic conductivity
- κ = Permeability
- a = Standpipe cross-sectional area
- A = Soil sample cross-sectional area
- L = Soil sample length
- h_1 = Water head at start of the test
- h_2 = Water head at end of the test
- t = Total test time
- μ = Water viscosity
- ρ = Water density
- g = Acceleration of gravity

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Other sediment sampling

During operation, samples of the pool water were taken for grain size analysis. The grain size analysis was performed using Horiba® laser diffraction. Laser diffraction was chosen in cases where sample volumes were too small for sieving or hydrometer tests.

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5.0 Phase II – Results

The Run 1 delta had a 2 m (6.5ft) radius and was approximately 20 cm (8in) thick. The radius of the Run 2 delta was 2.5 m (8.2ft) and approximately 15 cm (6in) thick. 20-second timelapse videos of the evolution of each of the runs are provided in Appendix D. The data acquired from Runs 1 and 2 are described below.

5.1 Pool and Scrape Sample Particle Size Analysis

5.1.1 Sieve Analysis of Scrape Samples

5.1.1.1 Run 1 Scrape Samples

The bulk material from Run 1 (preliminary grind) had a coarse fraction of about 41%. The coarse fraction is defined as the fraction of material by weight retained on a 74 micron sieve. 82% of the scrape samples taken from the beach had a coarse fraction of greater than 41%. The higher coarse fraction indicates that a portion of the fines bypassed the beach deposit and were flushed into the pool.

Figure 16a is an aerial photo of the Run 1 delta at the end of the growth phase. Figure 16b is the coarse fraction results from the scrape samples taken at that time across each radial sampling line and at various radii across the basin. The key indicates the location of the radial line relative to the left wall of the basin (looking downstream) and listed in radians and equivalent to 22.5, 33.75, 45, 56.25, and 67.5 degrees, respectively (Figure 14). Appendix B contains tabular results of the grain size analyses from all the scrape samples. Note in Figure 16b that the coarse fraction peaks at a radius of 55 cm. Figure 16a shows that the coarse peak at 55 cm is the same location as the flow transition from sheet flow to channelized flow. The general downward trend after the 55 cm radius confirms that the relative quantity of fines in the deposit increases with distance from the source. The variability in coarse fraction observed for the five radial transects are likely due to the chaotic nature of the braided system which means the delta surface includes a range of geomorphic features such as in-channel, channel bank, and floodplain. Similar evidence for local grain segregation was observed in other data: light and dark lenses observed in the suction cores and freeze slices taken from the final deposit(s) (See §5.4).

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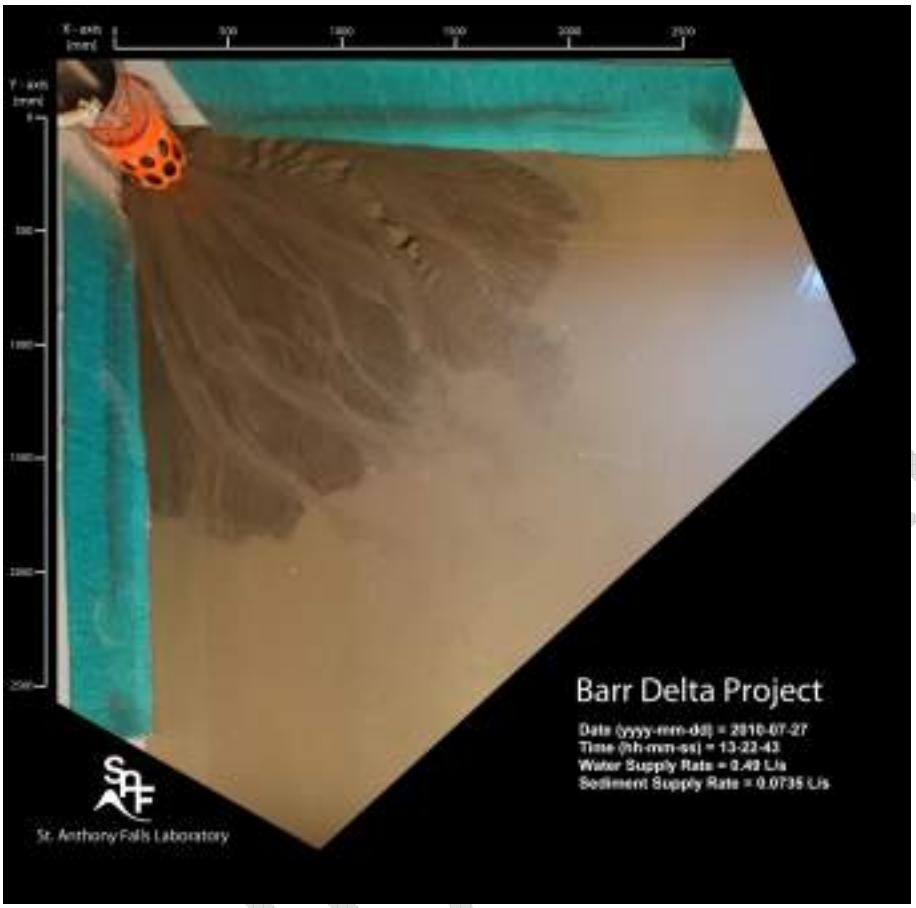

Figure 16a – Run 1 aerial photo of at the end of the growth phase.

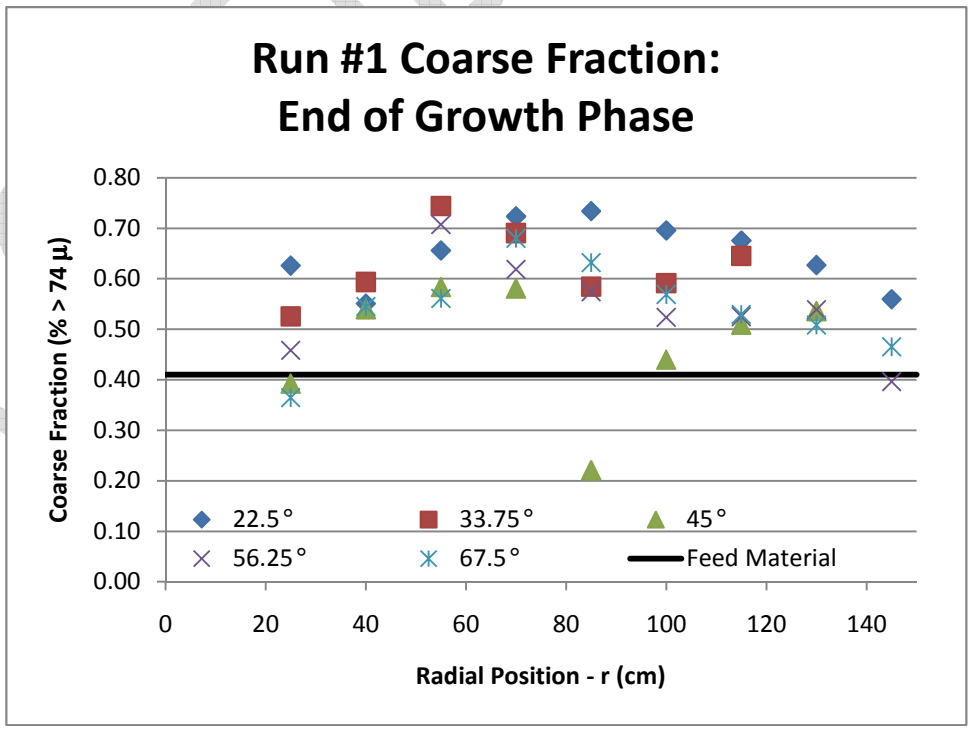

Figure 16b – Run 1 coarse fraction plot at the end of the growth phase.

Figure 16c is an aerial photo taken during Run 1 at the end of the falling phase (i.e. the final deposit), and Figure 16d is the results of the scrape samples taken at that time. Recall that during this falling phase the pool level was slowly lowered with the intent of promoting bypass of fine and preferential deposition of coarse material. In Run 1, a distinct single channel formed in the deposit at approximately the position of radial line at $\theta = 67.5°$ and this channel remained for most of the phase. All other radial lines for scrape samples were located in overbank settings. The data in Figure 16 may help to distinguish grain size distribution typical of in-channel versus overbank settings. The upper limit of the coarse fraction is reflected in the 5 scrape samples along the radial line at $\theta = 67.5°$that were taken from the main channel and range from 70-75%. The other samples taken from the floodplain range from 40-60% coarse material. These samples also show a slight downward trend indicative of downstream fining. Figure 16e compares the scrape results from each phase of Run 1. In general all samples were coarser than the input mixture indicating some loss of fine material. An upper limit is observed however as no deposit was coarser than 75% coarse sediments- even within an active channel.

After Run 1 was complete, the basin was slowly drained over a 25-day period of time. Three scrape samples were then taken from the bottomset, the subaqueous deposit on the sea floor less than 10cm (4in) beyond the delta toe. When sieved, 100% of each of these samples was less than 74 microns in diameter. These samples were saved for laser diffraction grain size analysis (§ 5.1.2). These results are expected since the bottomset is submerged by the pool, and the bottomset material is deposited via settling.

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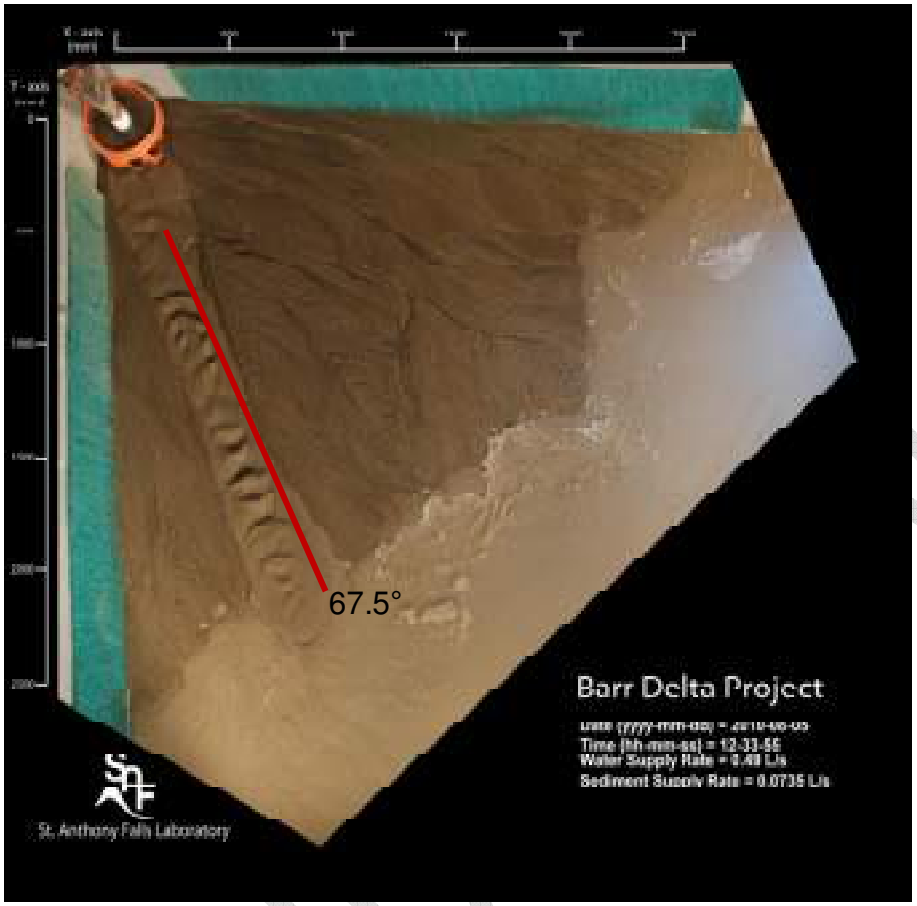

Figure 16c – Run 1 aerial photo of at the end of the falling phase.

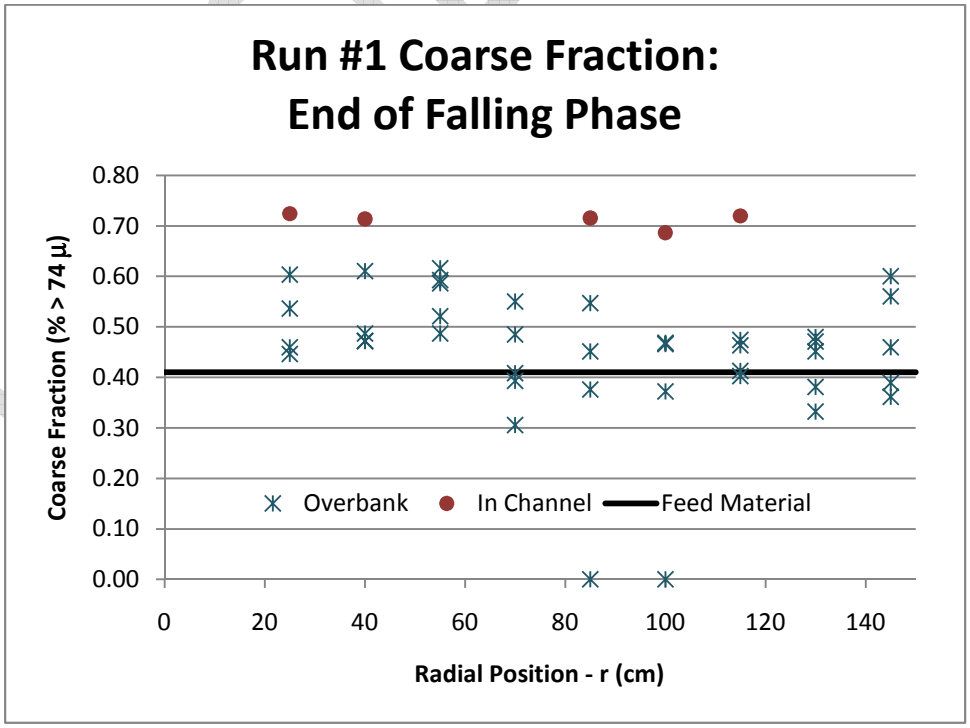

Figure 16d – Run 1 coarse fraction plot at the end of the falling phase.

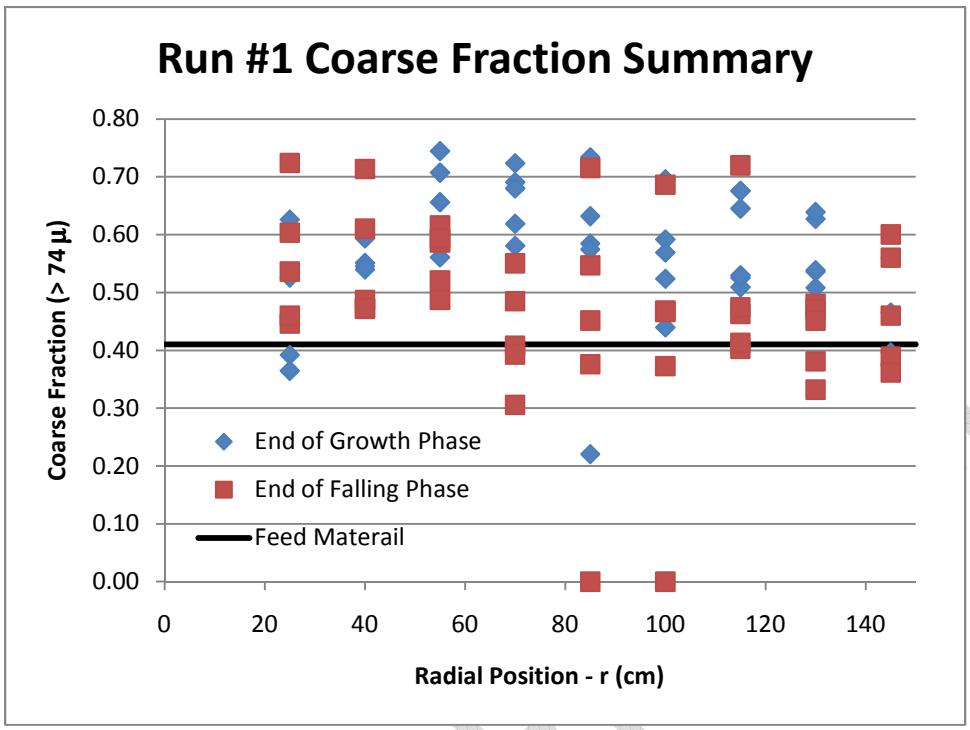

Figure 16e – Run 1 coarse fraction plot for all phases.

5.1.1.2 Run 2 Scrape Samples

The bulk material from Run 2 (final grind) also has a coarse fraction of about 41%. The higher coarse fractions found in 68% of the scrape samples indicate that a portion of the fines bypassed the beach deposit and were flushed into the pool. Figures 17a and 17b are the overhead photo and scrape sample results from the end of the growth phase. Like Run 1, there is a general downward trend in the coarse fraction indicative of downstream fining. The $\theta = 33.75°$ radial line sample does jump at the downstream end of the beach. This sample location was partially submerged by the pool which causes settling of coarse material.

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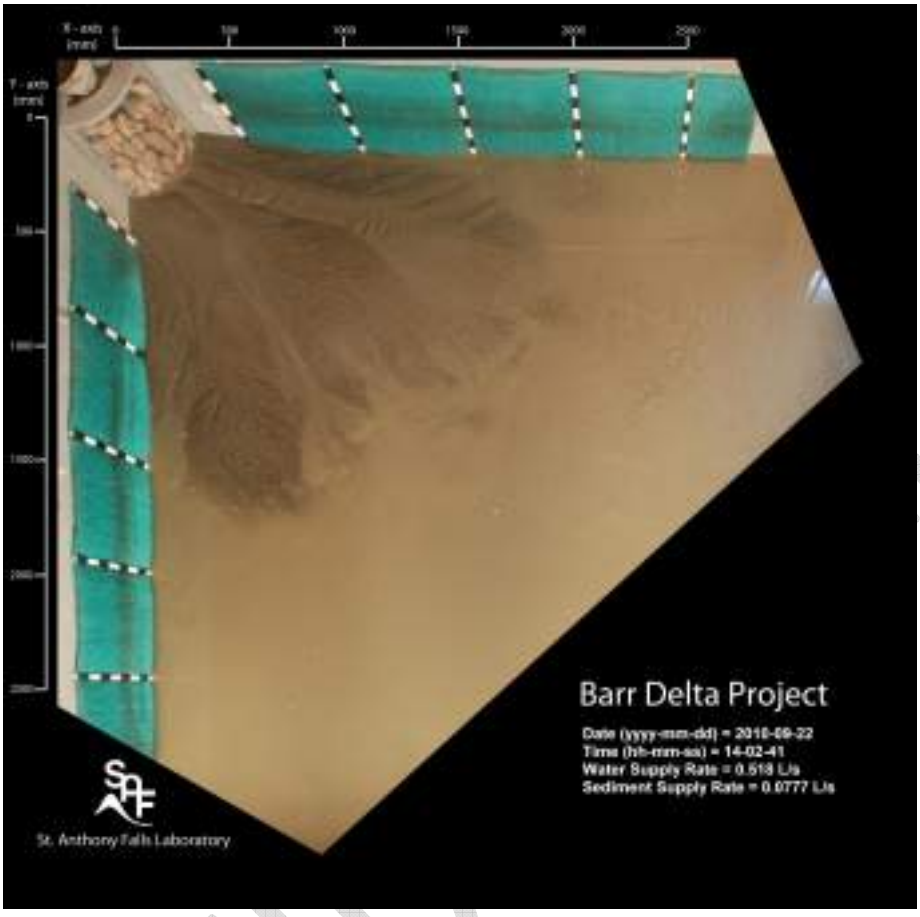

Figure 17a – Run 2 aerial photo of at the end of the growth phase.

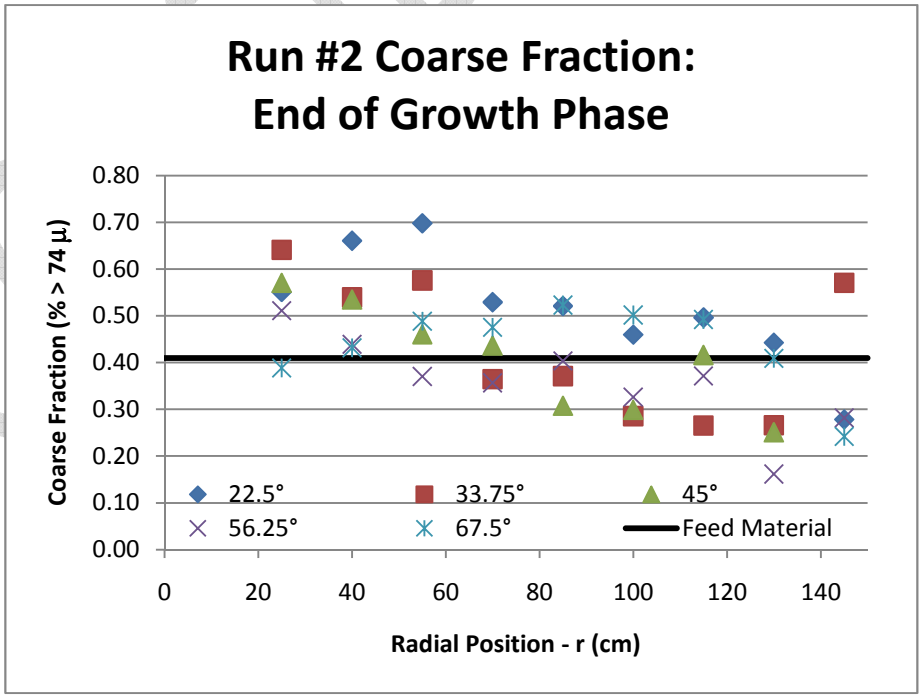

Figure 17b – Run 2 coarse fraction plot at the end of the growth phase.

Figure 17c is the overhead photo taken at the end of the building phase. The coarse fraction plot for the building phase is shown in Figure 17c. The results are similar to previously shown data where there is general fining of the deposit down slope and a variability in coarse fraction that ranges from 40% to 65%. The variability in the coarse fraction is largely due to the influence of bedforms and the braided channel morphology on the deposit. The image in Figure 17c highlights the chaotic nature of the fan surface with multiple braided channels and anti-dune bedforms (highlighted).

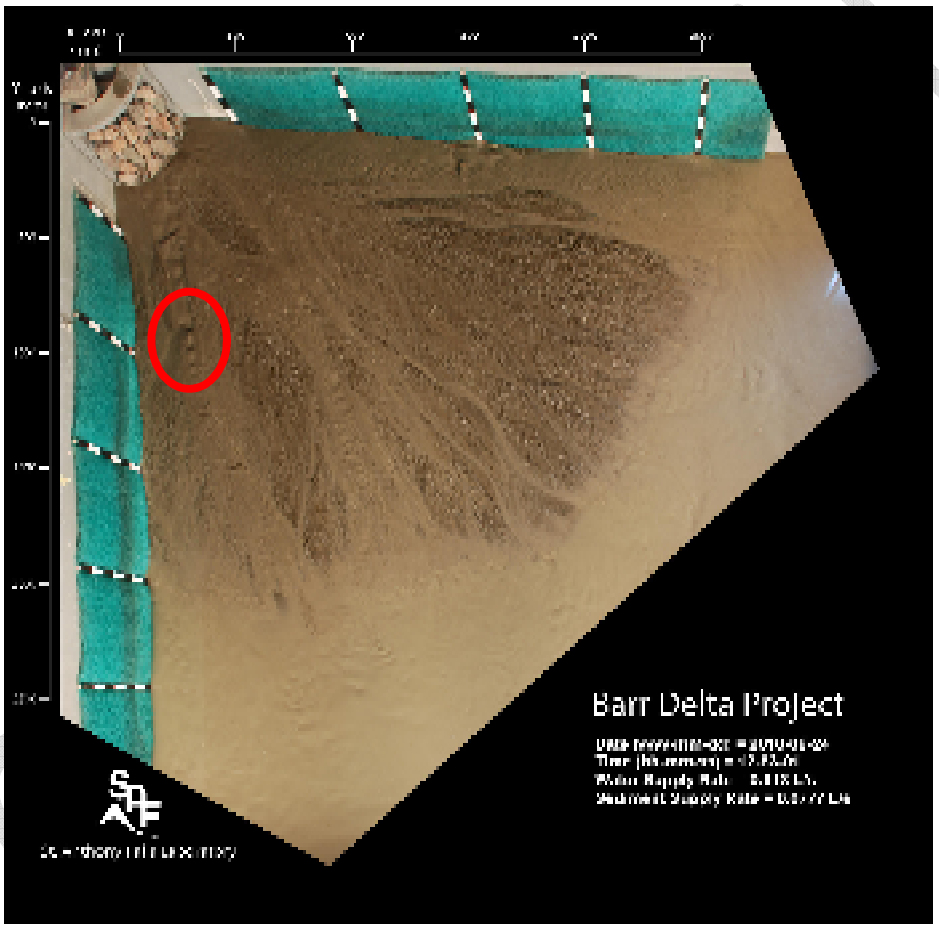

Figure 17c – Run 2 aerial photo of at the end of the building phase.

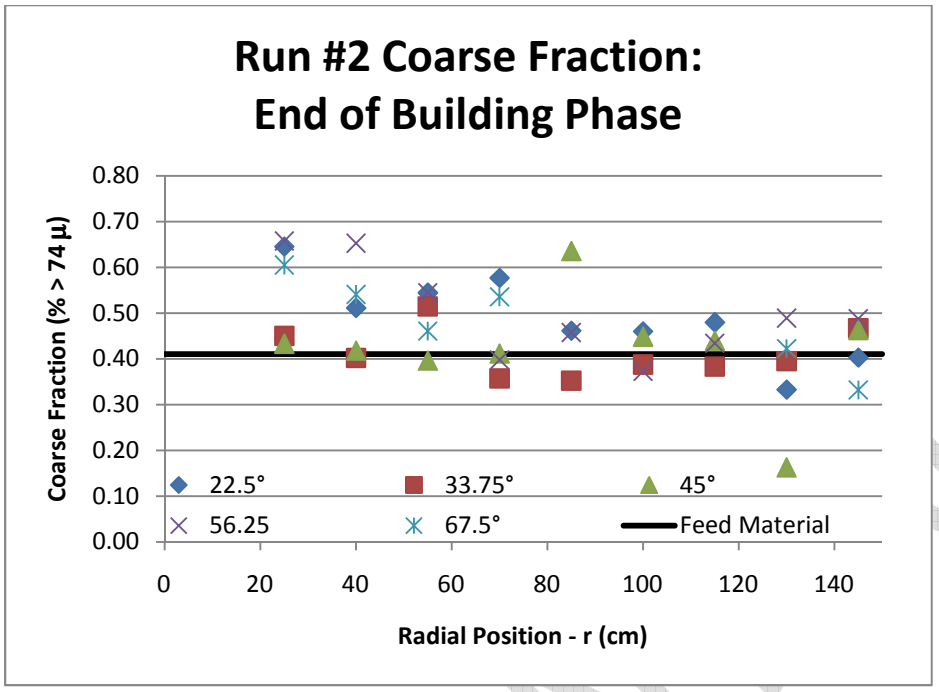

Figure 17d – Run 2 coarse fraction plot at the end of the building phase.

Figures 17e and 17f are the aerial photo and coarse fraction plot for the end of the falling phase. The photo highlights the anti-dune formations prevalent in the main channel. Figure 17g shows that the falling phase is generally the coarsest phase of Run 2, and the results from Figure 17f should be considered the worst case scenario for fines retention.

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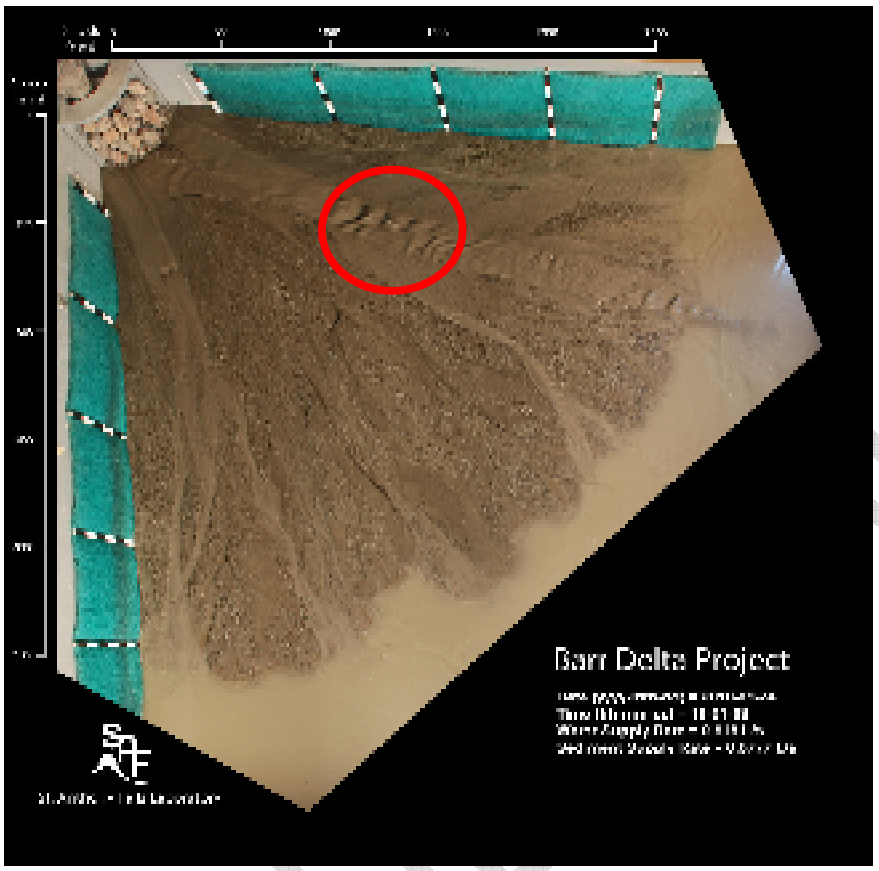

Figure 17e – Run 2 aerial photo of at the end of the falling phase.

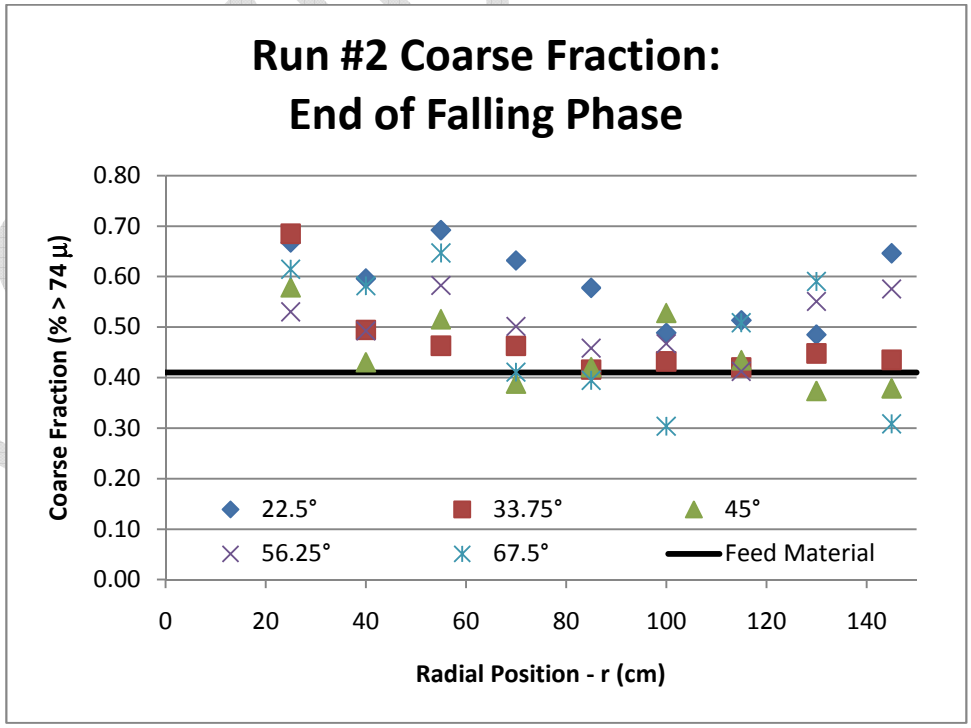

Figure 17f – Run 2 coarse fraction plot at the end of the falling phase.

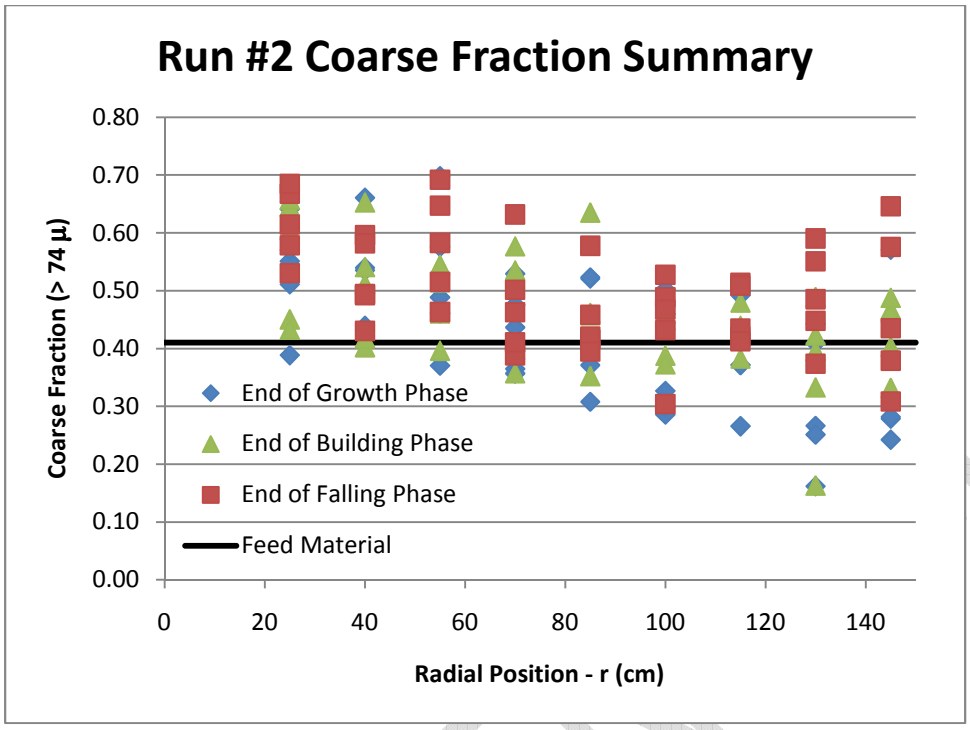

Figure 17g – Run 2 coarse fraction plot for all phases.

5.1.2 Horiba® Laser Diffraction Analysis of Pool Samples and Bottomset Scrape Samples

Water samples were taken from the pool during the run from two locations within the basin. The percent solids and sediment concentration of each of these samples is provided in Table 6. Figure 18 contains the grain size distribution for the pool samples plotted with the fine fraction of the bulk feed material. Grain size of these fine samples was performed using laser diffraction. All pool samples contain more fine material than the fine fraction of the bulk feed material. The pool sample that was taken near the shoreline where overland flow was entering the pool has the distribution that is most similar to the fine fraction of the bulk feed material. Samples collected away from the shoreline have finer distributions. The results suggest that the tailings can settle out of the water column. The estimated hydraulic residence time for water leaving the shoreline position to the basin outlet is about 85 minutes for the experiments reported here. The hydraulic residence time is the average amount of time needed to replace all of the water in the basin. It is equal to the volume of water in the basin divided by the flow rate of water entering the basin. This means that the sediment in the pool samples near the outlet have been in suspension for about 85 minutes. In other words, 85 minutes after entering the basin over 99.8% of delivered sediment has been deposited. Hydraulic residence time is actually an upper estimate of the transit time of sediment from the shoreline to basin outlet due to the presence of "short circuiting", the occurrence of preferential, faster flow paths in the water body.

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Table 6 – Solids fractions measured in pool samples

Run	Sample	Sample Location	% Solids	Concentration (kg/m^3)
1	1	At basin outlet	0.12%	1.2
1	2	At basin outlet	0.17%	1.7
2	2	Away from outlet	0.06%	0.6
2	4	Near shoreline	1.62%	15.9
2	6	At basin outlet	0.16%	1.6

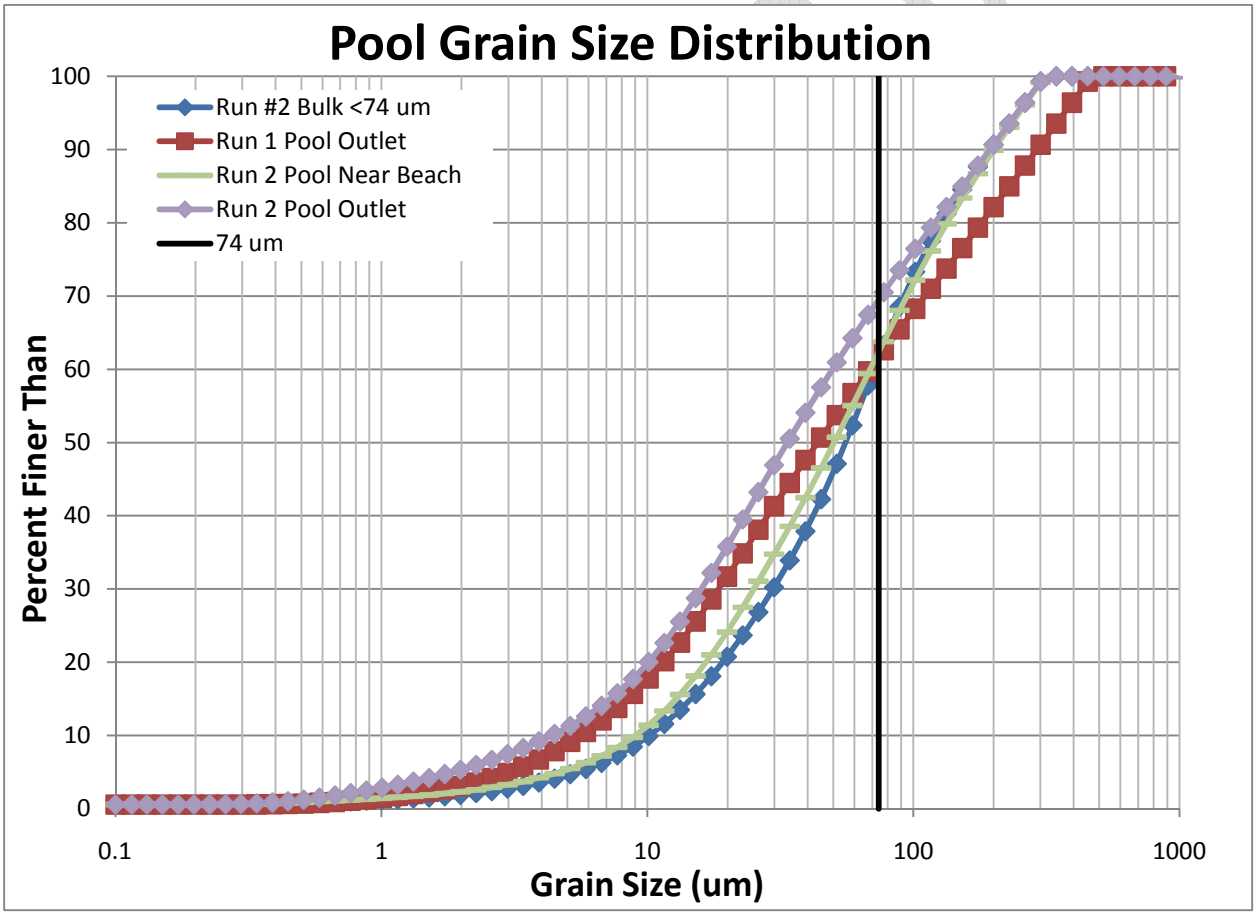

Figure 18 – Grain size distribution from pool sample.

Three bottomset scrape samples were taken from each final deposit. The grain size distributions provided in Figure 19 show that the bottomset particles fall within the fine fraction. The Run 1 bottomset scrape samples were taken within 10 cm (4in) of the toe of the deposit. The Run 2 bottomset scrape samples were taken about 30 cm (1ft) from the deposit toe. The coarser range of bottomset particles for Run 1 may be attributed to more settling near the toe. Figure 19 shows

that the Run 2 bottomset has fewer 1-74 micron size particles than the fine fraction of the bulk material. These missing particles were most likely trapped within the delta deposit.

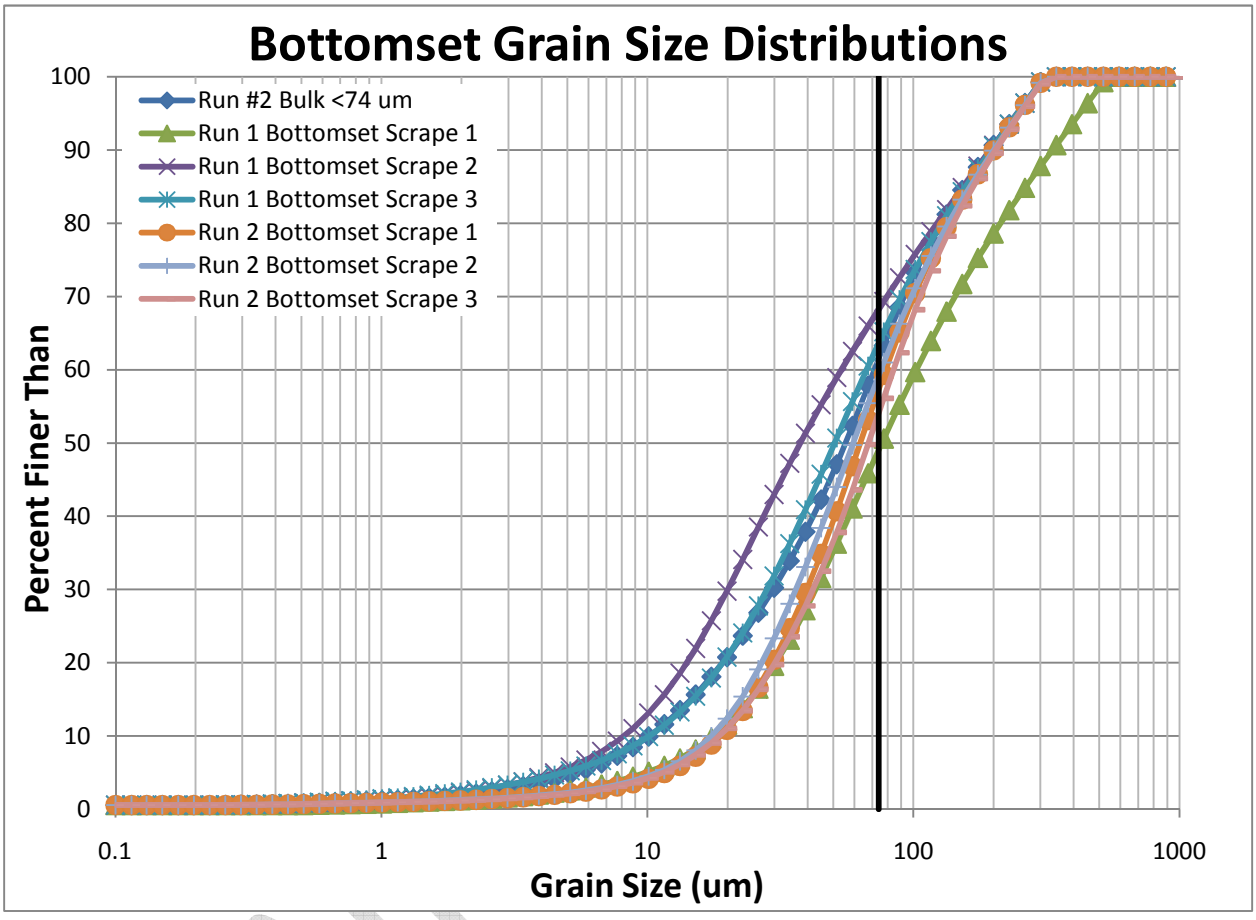

Figure 19 – Bottomset Scrape Sample Grain Size Distributions Measured Using Horiba® Laser Diffraction

5.2 Hydraulic Conductivity

Table 7 lists the results from the hydraulic conductivity tests. The data show that the hydraulic conductivity throughout the Run 1 deposit was quite low, although the values from various cores range over nearly one order of magnitude from 1.7×10^{-6} to 1.3×10^{-5} m/s. It is important to note that light and dark lenses were observed within the cores (§ 5.4.1) and that these lenses represent bodies of distinct grain size or fines content. The hydraulic conductivity test performed on the cores forces the vertically flowing water to pass through all elements of the deposit stratigraphy. Fine lenses are limiting layers which reduce the measured hydraulic conductivity. In the unconfined delta it is possible that the groundwater will simply flow around the fine lenses, depending on how the lenses are connected spatially. This would result in higher groundwater transport rates than are reflected by the hydraulic conductivity test. Section 5.4.2 has discussion of the connectivity of the fine and coarse lenses. Subject to this caveat about the possibility of bypassing the fine lenses, the trends of the measurements are descriptive of the sorting properties within the delta. The results, plotted in Figure 20, clearly show that vertical hydraulic conductivity changes with position in the delta. The primary factors influencing permeability are the D_{50} grain size and the degree of sorting (Beard and Weyl 1973). The decreasing trend shown

in Figure 20 is likely a result of the increasing fraction of fines with radial position (i.e. downstream fining). This is the expected trend in depositional fans and thus would be predicted for the field case as well.

Table 7 – Run 1 results of falling head hydraulic conductivity tests

Sample #	r (cm)	L (cm)	h_1 (cm)	h_2 (cm)	t (sec)	k (m/sec)	k (cm/sec)	k (ft/min)	κ^* (Darcy)
1	30	27	230	159.5	6723	1.0E-05	1.0E-03	2.1E-03	1.08
2	60	23.4	225.6	167.9	5269	9.3E-06	9.3E-04	1.8E-03	0.96
3	90	21.8	227.6	138	8093	9.6E-06	9.6E-04	1.9E-03	0.49
4	120	21.3	228.2	142.3	12360	5.8E-06	5.8E-04	1.1E-03	0.99
5	150	19.7	210.1	161.9	9850	3.7E-06	3.7E-04	7.3E-04	1.35
6	180	19.3	226.8	191.8	8393	2.7E-06	2.7E-04	5.4E-04	0.48
7	210	17.5	229.4	169.7	7956	4.7E-06	4.7E-04	9.3E-04	1.23
8	30	24.5	228.7	138.6	6695	1.3E-05	1.3E-03	2.6E-03	0.36
9	60	23.5	226.3	146.5	8400	8.6E-06	8.6E-04	1.7E-03	0.28
10	90	23	227	170.2	9047	5.2E-06	5.2E-04	1.0E-03	0.89
11	120	21	222.9	158.7	8892	5.7E-06	5.7E-04	1.1E-03	0.18
12	150	19.8	227.1	166.5	8887	4.9E-06	4.9E-04	9.7E-04	0.54
13	180	17.8	229.9	202.8	9110	1.7E-06	1.7E-04	3.4E-04	0.60
14	210	19	221.6	175.7	6817	4.6E-06	4.6E-04	9.0E-04	0.46
15	30	24.6	226.2	149.6	6071	1.2E-05	1.2E-03	2.3E-03	0.51
16	60	23.3	225.5	145.9	11128	6.5E-06	6.5E-04	1.3E-03	0.59
17	90	22.2	226.7	147.3	9779	6.9E-06	6.9E-04	1.4E-03	0.28
18	120	20.2	227.3	154.4	12428	4.5E-06	4.5E-04	8.8E-04	0.67
19	150	19.6	229.2	184.7	11241	2.7E-06	2.7E-04	5.3E-04	0.72
20	180	18	227.1	193.2	5958	3.5E-06	3.5E-04	6.8E-04	0.38

* Note: κ is permeability, 1 Darcy = 9.869×10^{-13} m² = 1.062×10^{-11} ft²

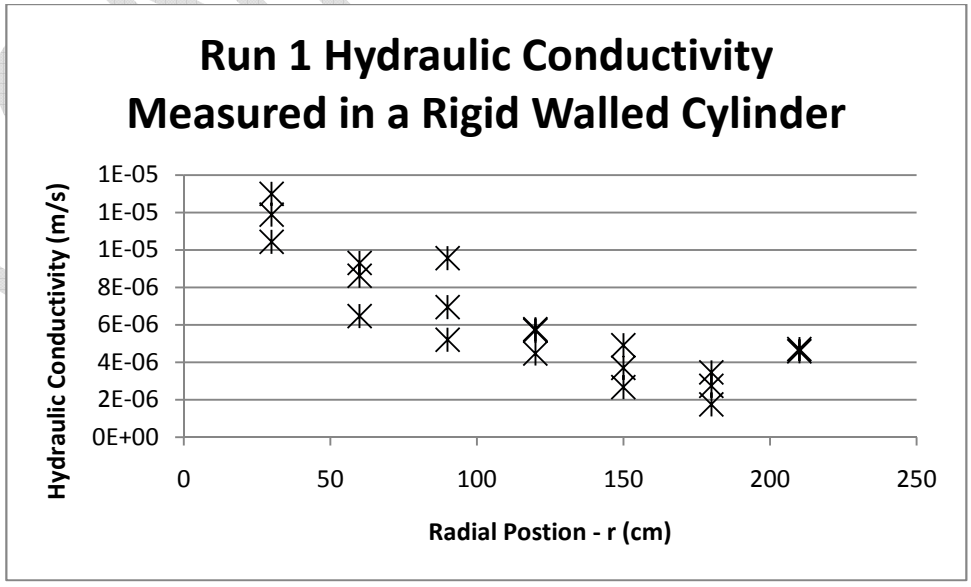

Figure 20 – Run 1 hydraulic conductivity variation with radial position along delta.

5.3 Porosity

The porosity of the Run 1 delta was measured using two methods. The first was mini-cores that measure porosity in the top 2 cm (0.75in) of the final delta surface. The second method is the bulk porosity measured from the full depth piston cores.

Table 8 lists the results from the surface porosity measurements. There is some variability at the different locations; however, there is no indication of spatial trends. Table 9 lists the bulk porosity from the Run 1 core samples.

Table 8 – Run 1 surface porosity taken via mini-cores

Sample Location	Porosity
Upstream in Main Channel	0.42
Upstream in Secondary Channel	0.37
Upstream out of Channel	0.37
Downstream in Main Channel	0.35
Downstream in Secondary Channel	0.42
Downstream out of Channel	0.42

Table 9 – Run 1 bulk porosity taken from suction cores.

Core #	r (cm)	Porosity
3	90	0.43
4	120	0.43
12	150	0.43

5.4 Stratigraphy

5.4.1 Suction Cores

After the Run 1 piston cores were tested for hydraulic conductivity, the four samples were extruded from the cylinder and split open for imaging. Figures 21a – 21d show images of the split cores. These images show light and dark bands within the deposit. Different color bands are associated with different grain sizes or sorting properties.

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Figure 21a – Grain image from piston core 01 (r = 30cm)

Figure 21b – Grain image from piston core 03 (r = 90cm)

Figure 21c – Grain image from piston core 04 (r = 120cm)

Figure 21d – Grain image from piston core 12 (r = 150cm)

5.4.2 Freeze Slices

As noted in Section 5.4.1 light and dark lenses were observed within the suction cores. The freeze slices, taken from the final deposit of Run 2, were intended to determine the lateral extent of the lenses and how interconnected they are. Figure 22a shows the freeze slice locations, and Figures 22b and c are images of frozen sediment slices from Run 2 taken at r = 55cm (1.8ft) and r = 150cm (5ft), respectively. Figure 22b shows that at the upstream end of the delta the dark lenses are large and often connected over the length of the deposit. The crack in the Figure 22b slice is due to the thawing process. It was not present within the deposit. In Figure 22c, at the downstream end of the delta, the dark lenses are more likely to be completely encased in fines.

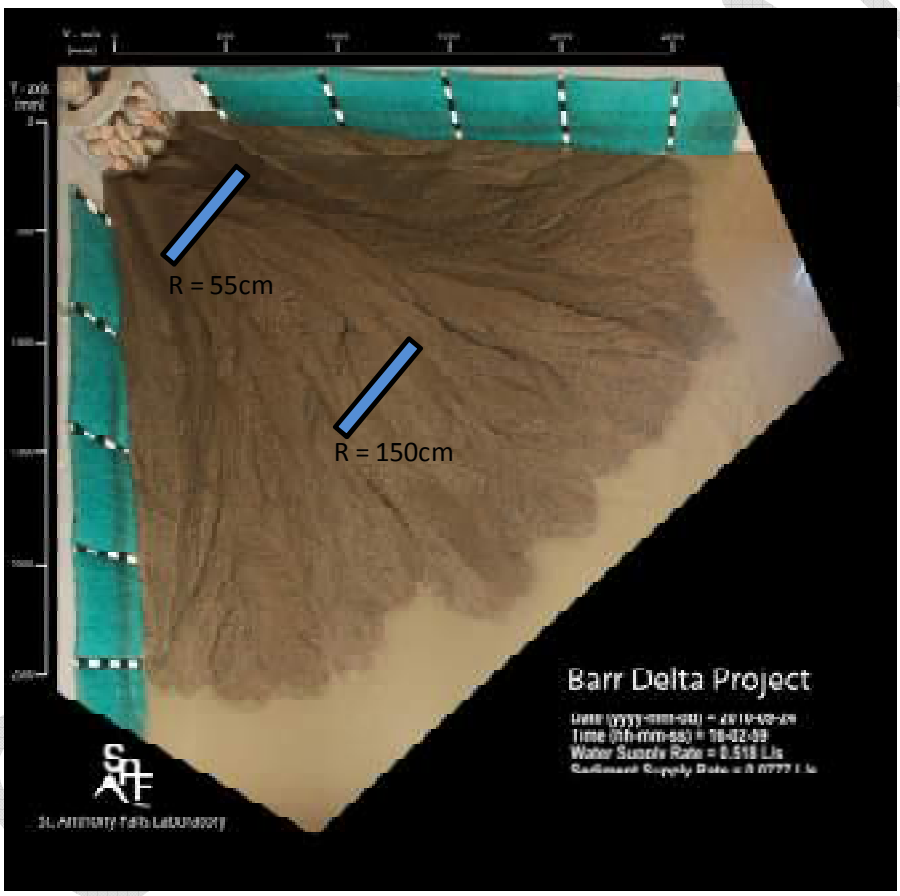

Figure 22a – Freeze slice locations.

Figure 22b – Run 2 freeze slice taken at r = 55cm. Looking upstream.

Figure 22c – Run 2 freeze slice taken at r = 150cm. Looking upstream.

5.4.2.1 Freeze Slice Grain Size Distributions

The freeze slices were subsampled to determine the grain size distributions from the light and dark lenses. The sampling locations are given in Figures 23a and 23b. The samples were analyzed for grain size distribution using Horiba® laser diffraction. The results provided in Figure 24 show that there is quite a bit of variability in the 10 to 74 um grain sizes. Figures 25a and 25b have the coarse fractions (above 74 microns) labeled directly on the freeze slice images. Note the darker lenses tend to have higher coarse fractions than the lighter lenses, though the trend seem to be less consistent at 150cm The coarse fraction observed in the freeze sliced are within the range of coarse fractions observed in the scrape samples (Fig. 17g).

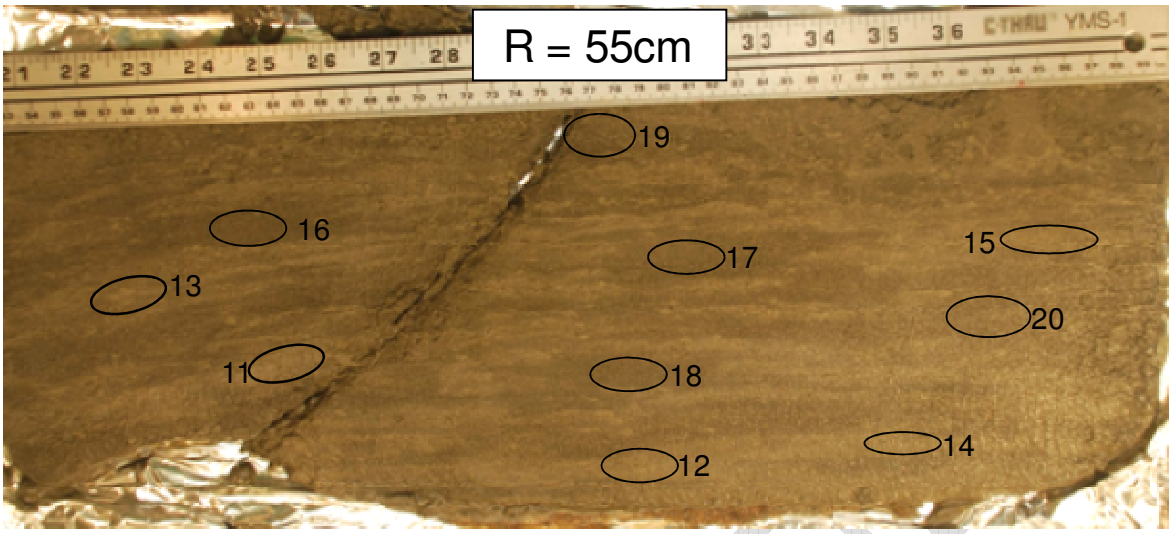

Figure 23a – Subsample Locations for the Freeze Slice at r = 55cm (The sample number are indicated)

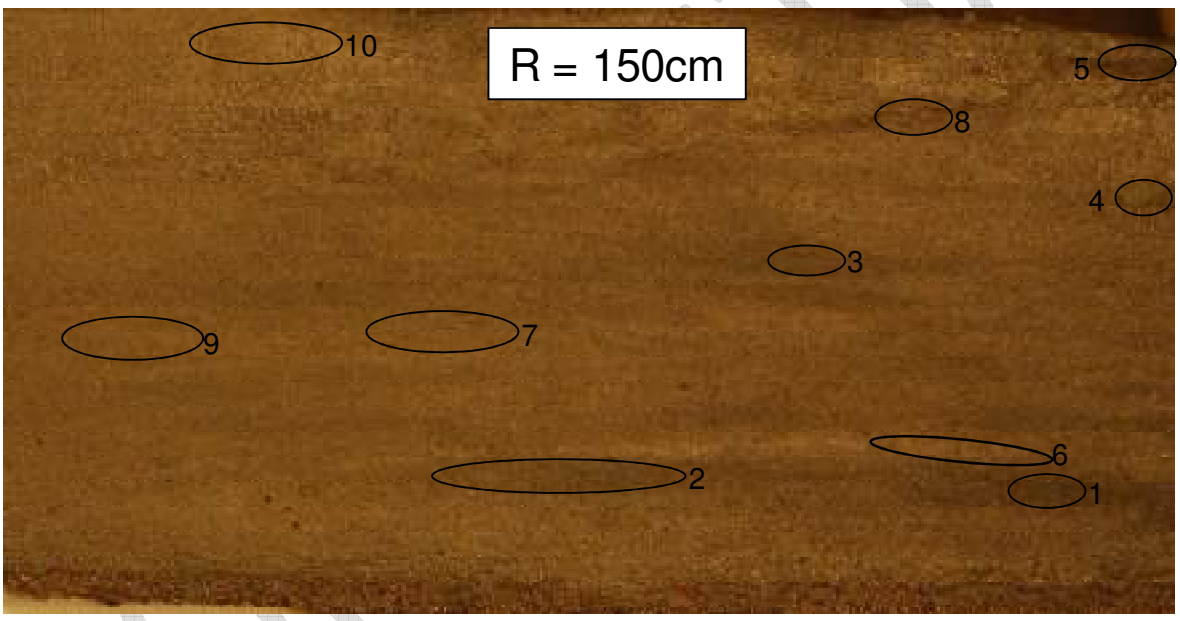

Figure 23b – Subsample Locations for the Freeze Slice at r = 150cm (The sample number are indicated)

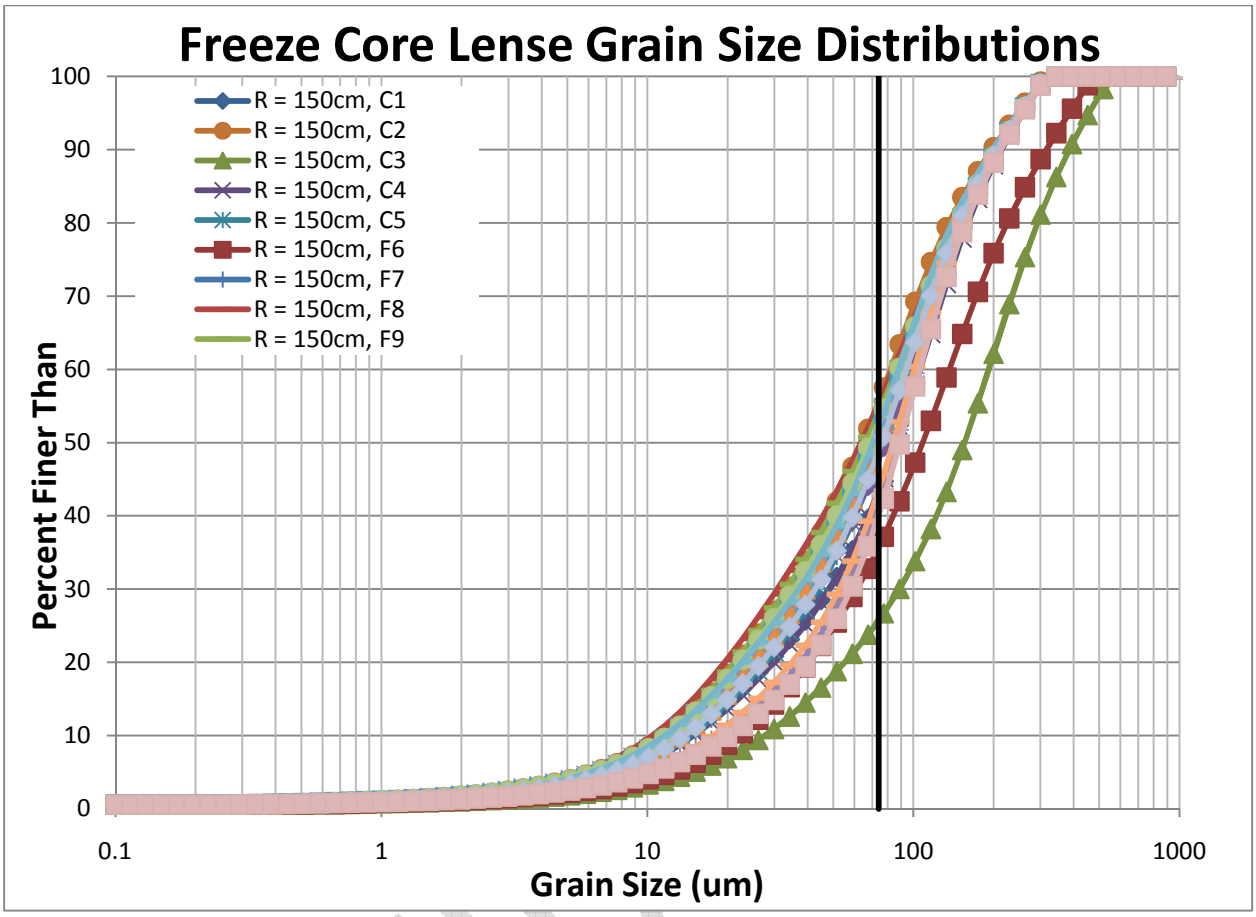

Figure 24 – Grain Size Distributions from Freeze Slice Subsamples

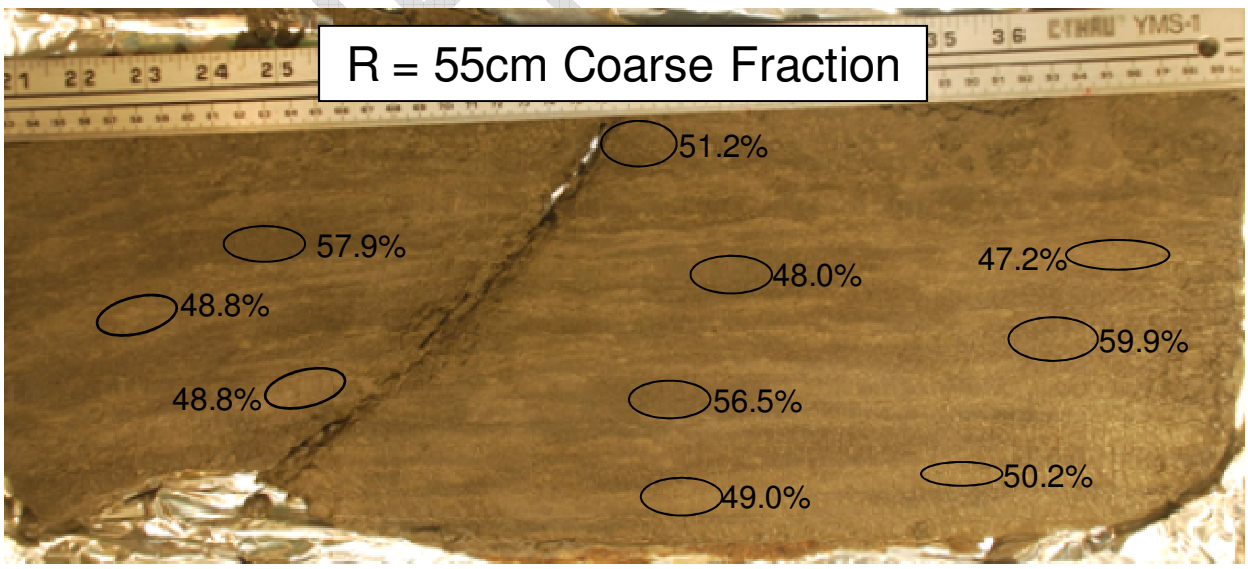

Figure 25a – Coarse Fraction (above 74um) from the Freeze Slice at r = 55cm

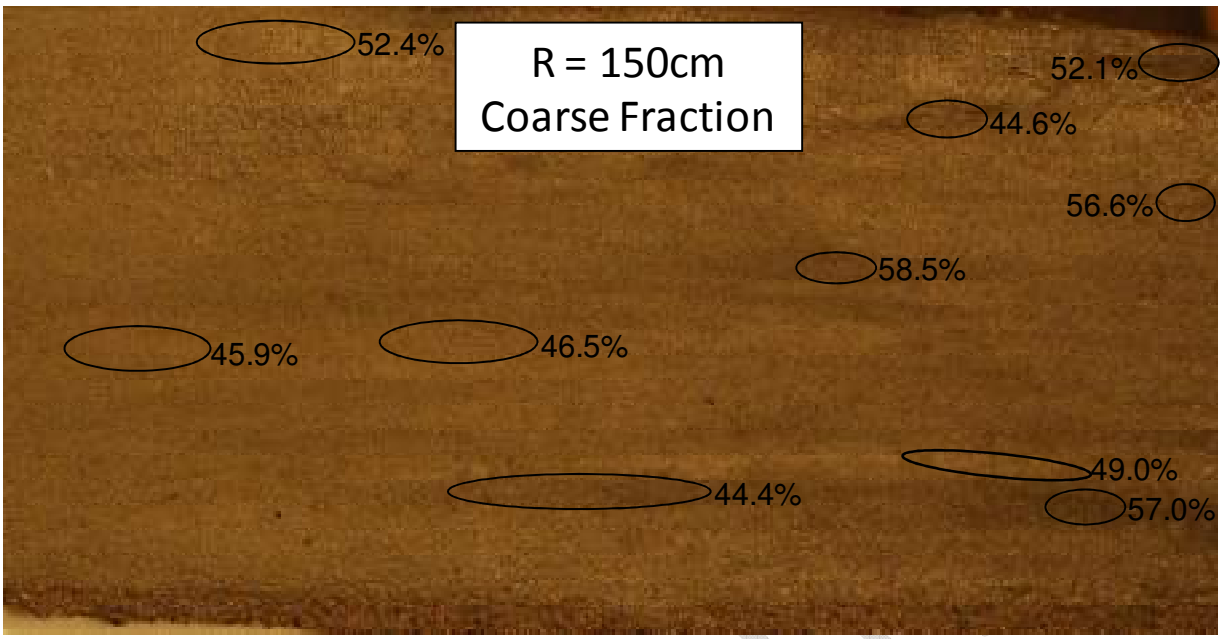

Figure 25b – Coarse Fraction (above 74um) from the Freeze Slice at r = 150cm

5.4.2.2 Freeze Slice Permeability

Although it is not possible to directly measure the permeability at each of the sampling points within the freeze slices, the permeability can be estimated from the grain size distribution. Beard and Weyl (1973) studied how the grain size distribution of artificially-mixed, wet-packed sand influences permeability. They found that permeability increases with increasing D_{50} (mean grain size) and decreases with an increasing sorting coefficient (D_{75}/D_{25}). D_{25}, D_{50}, and D_{75} are the grain diameters corresponding to 25, 50, and 75 percent finer than in Figure 24. The relationship between the sorting coefficient and permeability is stronger than the relationship between grain size and permeability. In other words, a coarser material is more conductive, but adding a small amount of fines fouls the coarse matrix and reduces flow. The matrix of grain size, sorting coefficients, and permeability developed by Beard and Weyl can be approximated using a power law equation. Because different materials are used, the permeability from Beard and Weyl's matrix may not predict the actual permeability of the tailings deposit; however, the range and spatial distribution of the subsample permeabilities should give a good relative indication about which zones of the freeze slice are more likely to allow or impede water flow.

Figures 26a and 26b provide the Beard and Weyl estimate of permeability (in mDarcy) for the different lenses of the freeze slices. The average Beard and Weyl permeability in the upstream freeze slice is higher than in the downstream freeze slice. This result is consistent with the rigid wall hydraulic conductivity measurements from Run 1 (Figure 20). At the downstream end the Beard and Weyl permeability is similar throughout the slice; however, at the upstream end the Beard and Weyl permeability is typically greater for the darker lenses. This result is consistent with the findings from Figure 25. We also note that the highest estimated permeabilities would control the bulk permeability for cases where the coarse (dark) depositional units are connected so as to provide an unbroken flow path through the deposit.

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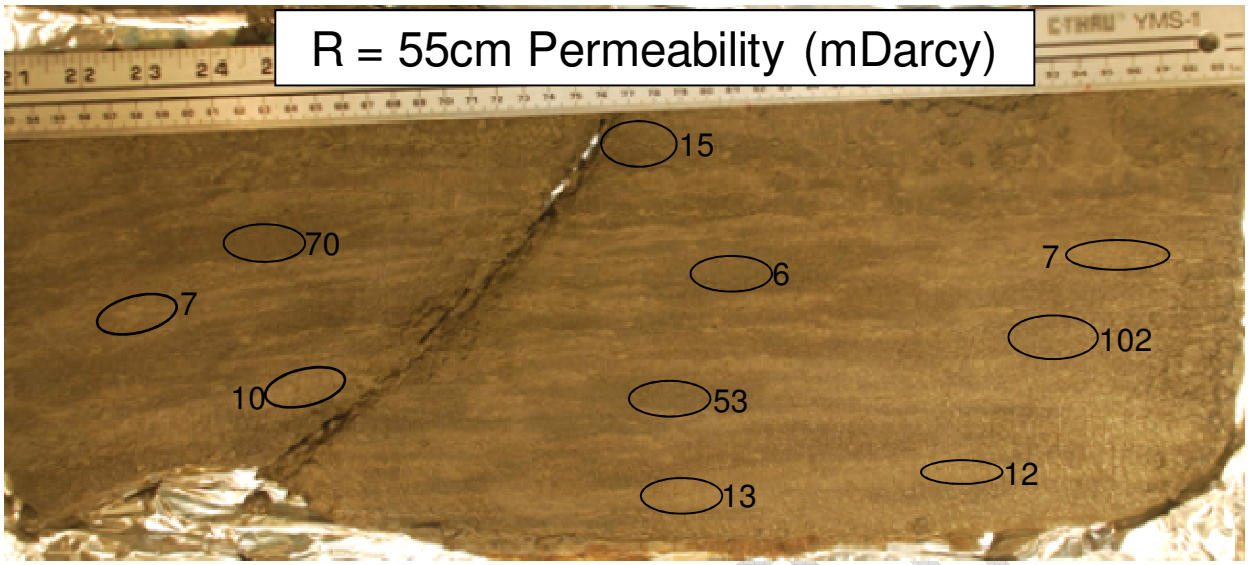

Figure 26a – Beard and Weyl permeability (in mDarcy) for the Freeze Slice at r = 55cm

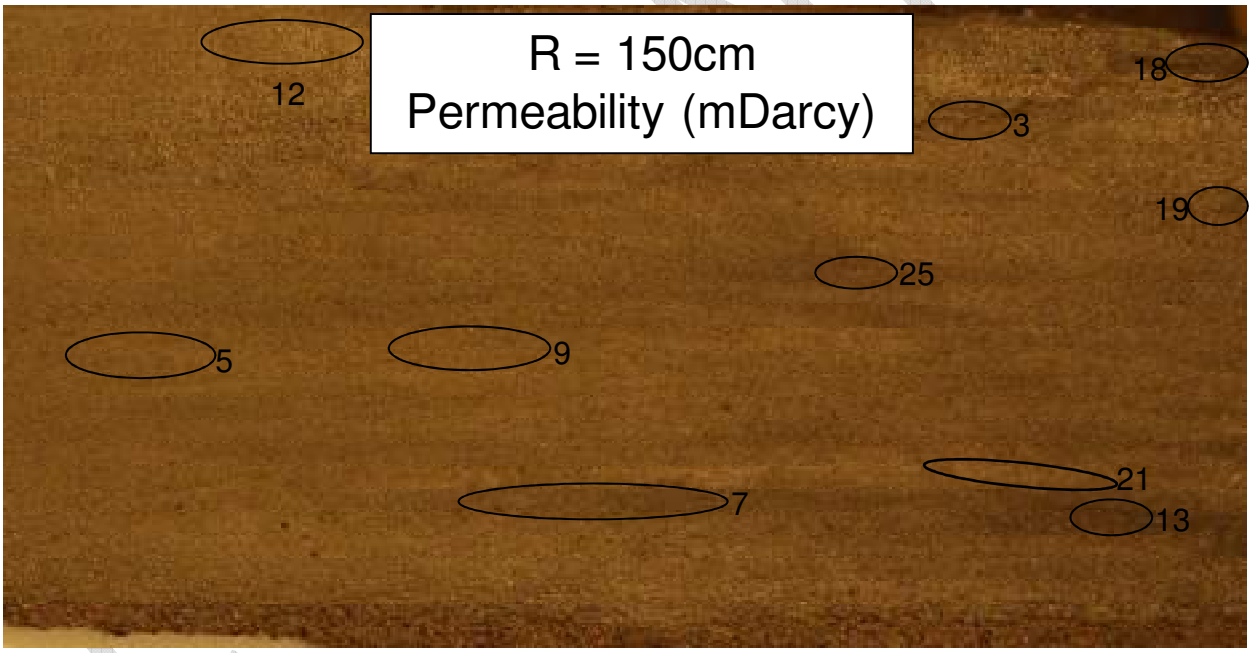

Figure 26b – Beard and Weyl permeability (in mDarcy) for the Freeze Slice at r = 150cm

5.5 Topographic Scans

The field delta geometry was evaluated as part of the one-dimensional experiments. The reduced slurry discharge required to preserve the aspect ratio resulted in a model delta geometry that does not directly translate to the field delta. For Run 1, the beach slope was on the order of 4%. Figure 27(a-g) shows topographic maps of the Run 1 and 2 deltas at the end of each phase. The figures provide further documentation of the variability of the surface topography and chaotic nature of the deposition process. It is interesting to note the shape of the shoreline and how it deviates

from an ideal cone-shape. Also, studying the distal portion of the deltas in all images shows distinct depositional lobes that appear to weave together as the distal fan is constructed. These are the processes that create variability in the grain size, coarse fraction, and visual variability that was observed in these tests.

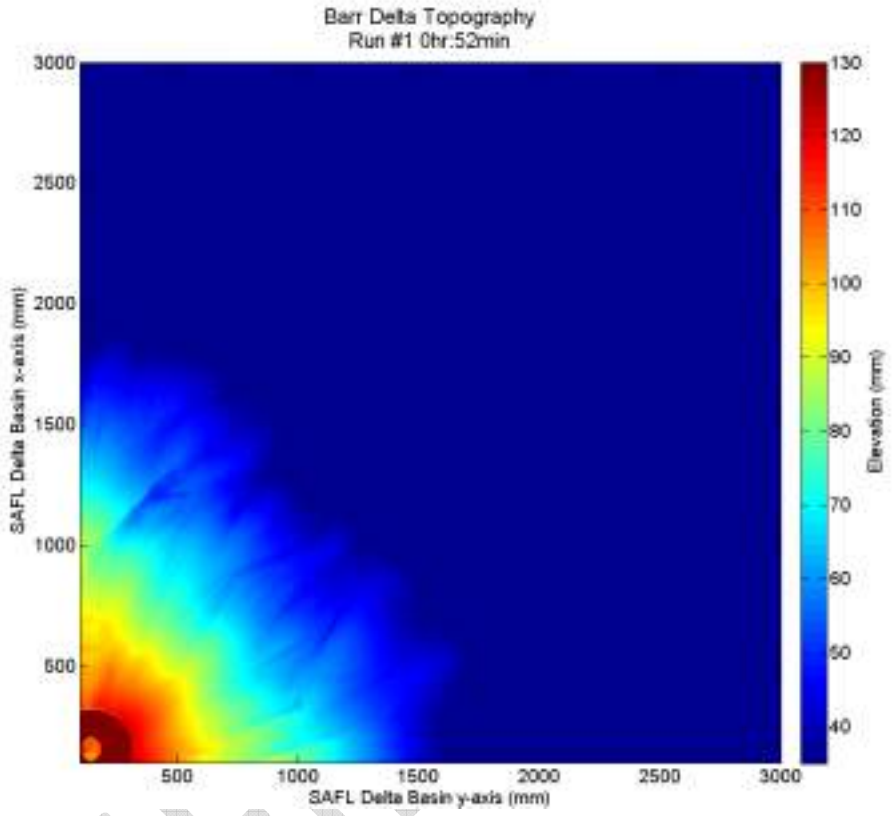

Figure 27a – Run 1 topography at the end of the growth phase

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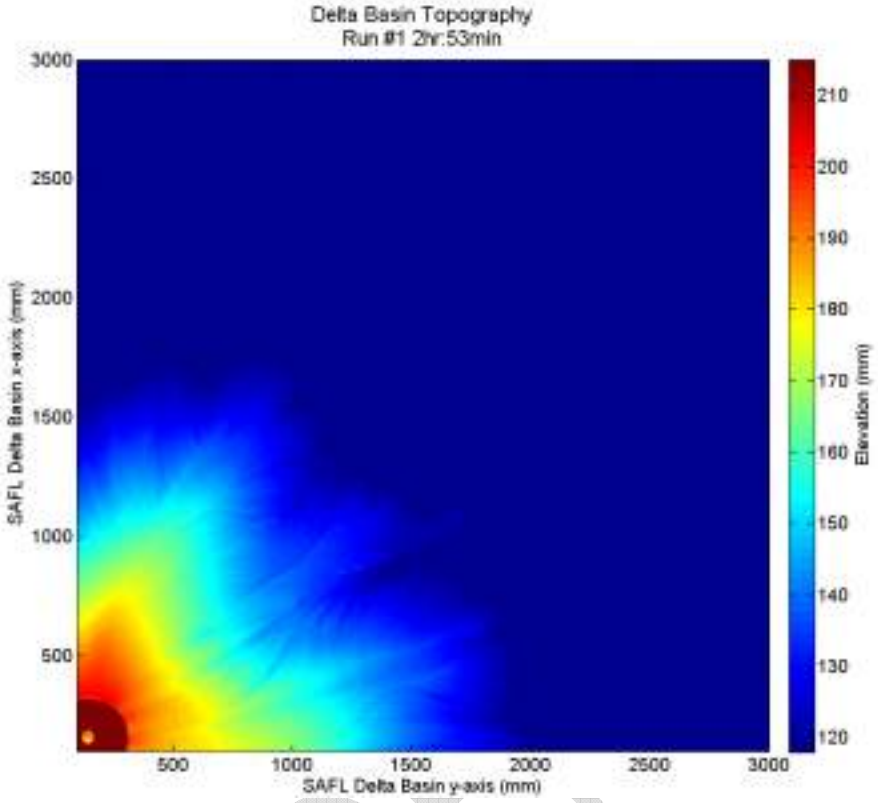

Figure 27b – Run 1 topography during the building phase

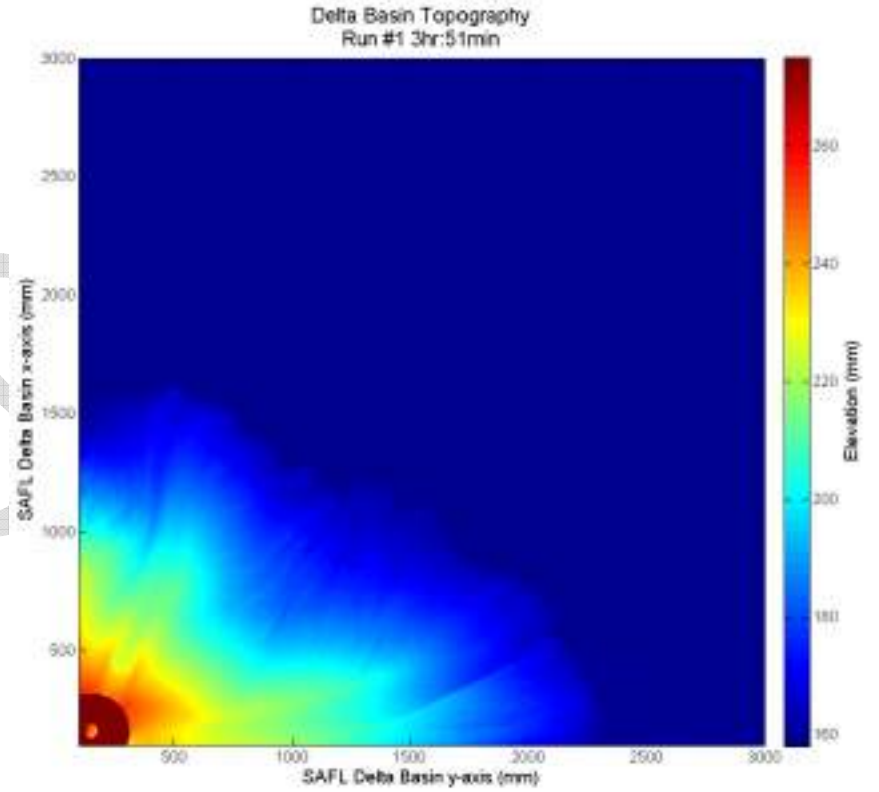

Figure 27c – Run 1 topography at the end of the building phase

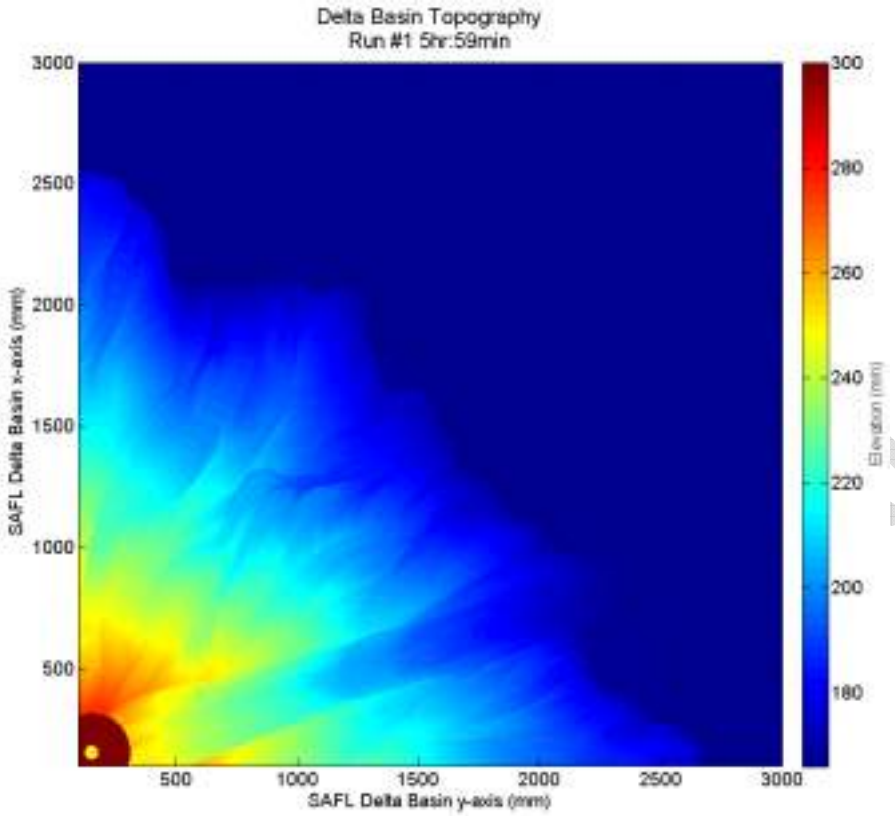

Figure 27d – Run 1 topography at the end of the falling phase

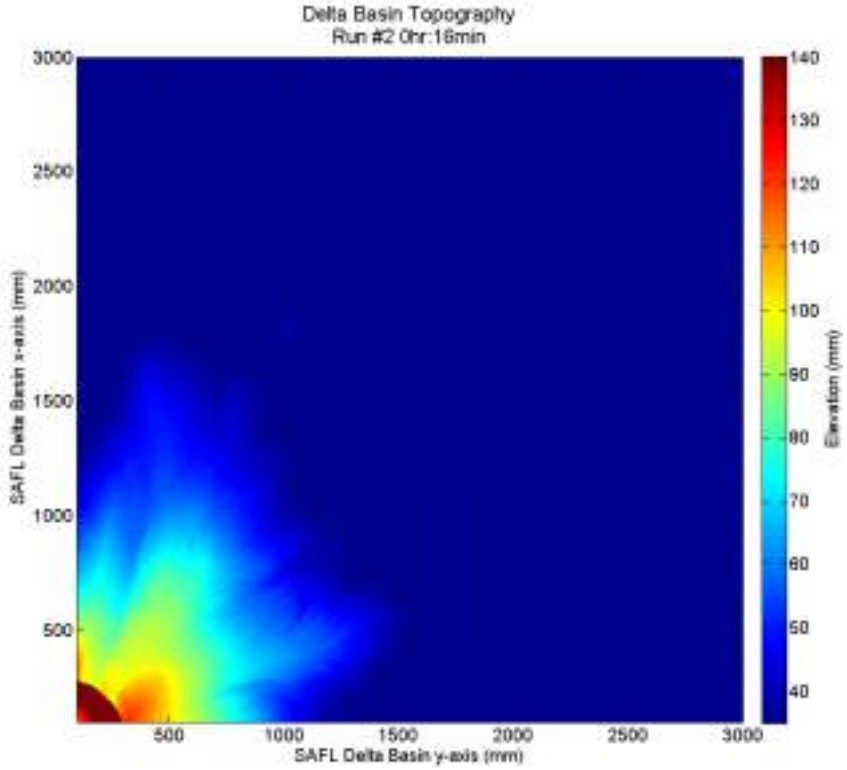

Figure 27e – Run 2 topography at the end of the growth phase

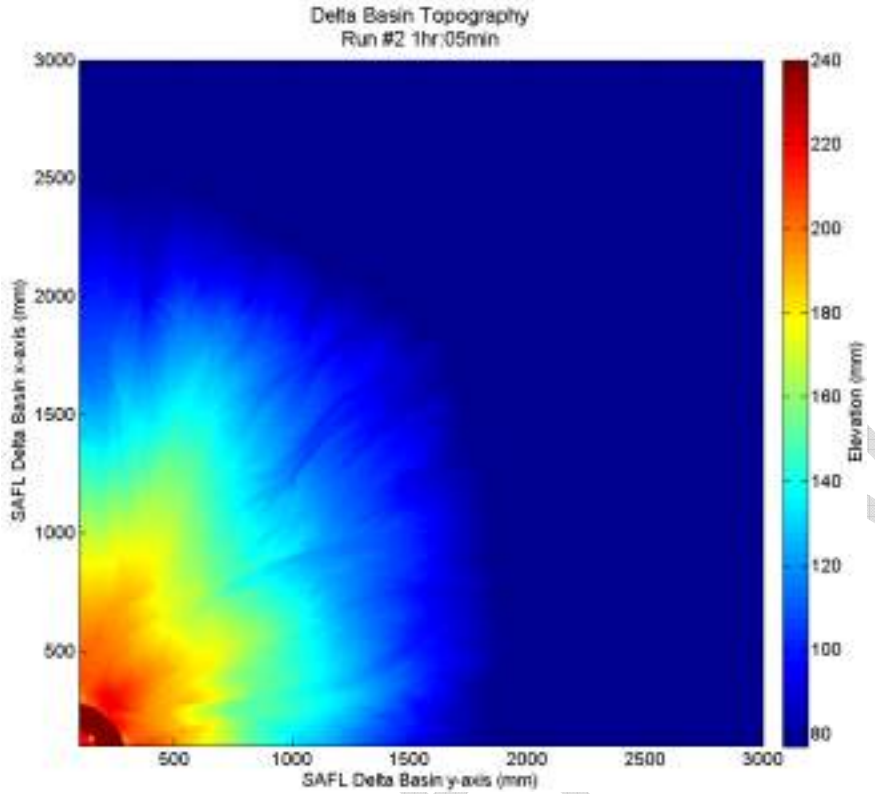

Figure 27f – Run 2 topography at the end of the building phase

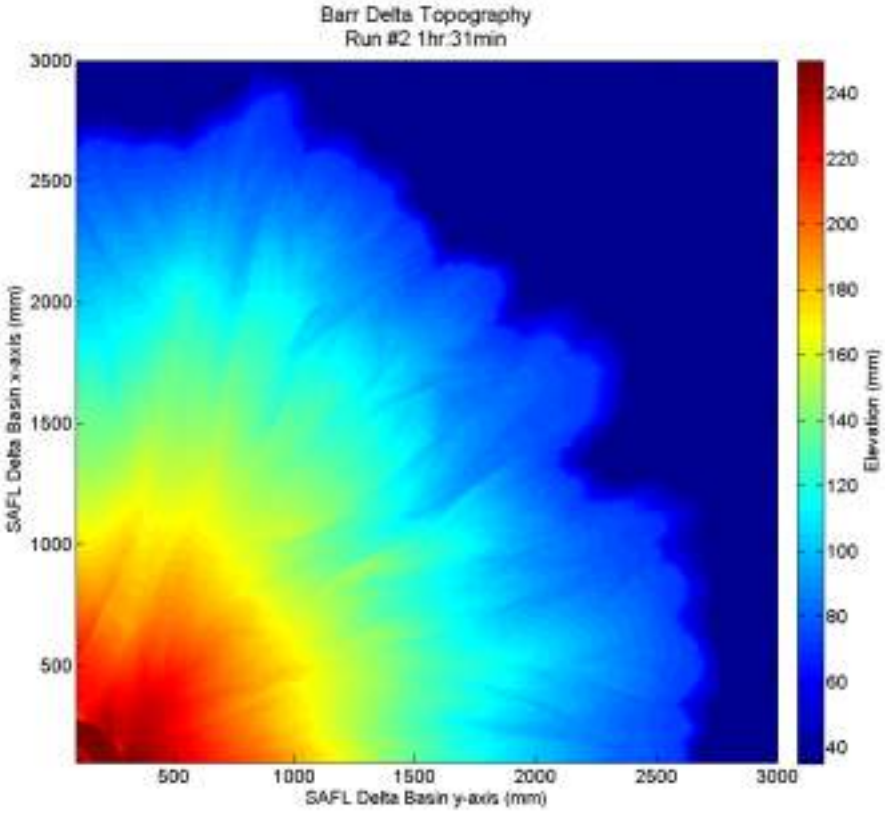

Figure 27g – Run 2 topography at the end of the falling phase

6.0 Phase II – Discussion

6.1 Phase II Results

The goals of the 2D experiments were to 1) determine the expected lower limit of fines concentration in the deposit, 2) determine the degree of grain size segregation both vertically and horizontally within the deposit and 3) evaluate the potential range of hydraulic conductivity throughout the model delta.

The lower limit of fines concentration and the degree of grain size segregation are both functions of fines retention within the deposit. There are several factors that likely influenced fine particle retention within the deposit. One such factor is that the grain size distribution and concentration of the tailings was conducive to fines retention. The range of particle sizes was sufficient and the concentration was high enough that in a shallow flow, such as the flow seen on a delta fan, the larger particles interact with the smaller particles strongly, and both are deposited together. This was observed in the generally high fines contents measured in the scrape samples and in the freeze cores.

The data consistently showed a downstream (nearer the shoreline) trend of increasing fines fraction within the deposit. This observation is likely representative of what would be observed in the field. The coarsest material will deposit in the proximal region of the delta and distally, where the energy of the flow is distributed across a larger fan surface and slopes are milder, fine material will be preferentially deposited. This process is depicted in the mild downward trend seen in Figures 16b, 16b, and 16d. Variability of downstream fining is likely due to the chaotic nature of deltaic transport and deposition of sediment. Braiding, avulsion (see step three of delta evolution below), and bar and bedform movement act to mix surface layers both vertically and horizontally, even as local channel processes create spatially distinct grain size zones. The effect of these processes can be seen in all of the aerial photos and their influence on fines retention is evident in the spikes seen in Figure 16d. Throughout the evolution of the delta, channels with flowing water and sediment sweep across the surface. This process likely accounts for the variability of the coarse fraction data seen in Figures 16e and 16g. This process also accounts for the light and dark lenses seen in Figures 26a and b. The larger, more pronounced lenses in the r = 55 cm freeze slice (Figure 26a) are due to narrower, deeper, and more active channels at the upstream end of the delta. At the downstream end much more of the delta is covered with unchannelized sheet flow. This results in the thinner, wider, and less distinct lenses seen in the r = 150 cm freeze slice (Figure 26b). Figures 28 show these channel characteristics and a sequence of channel evolution.

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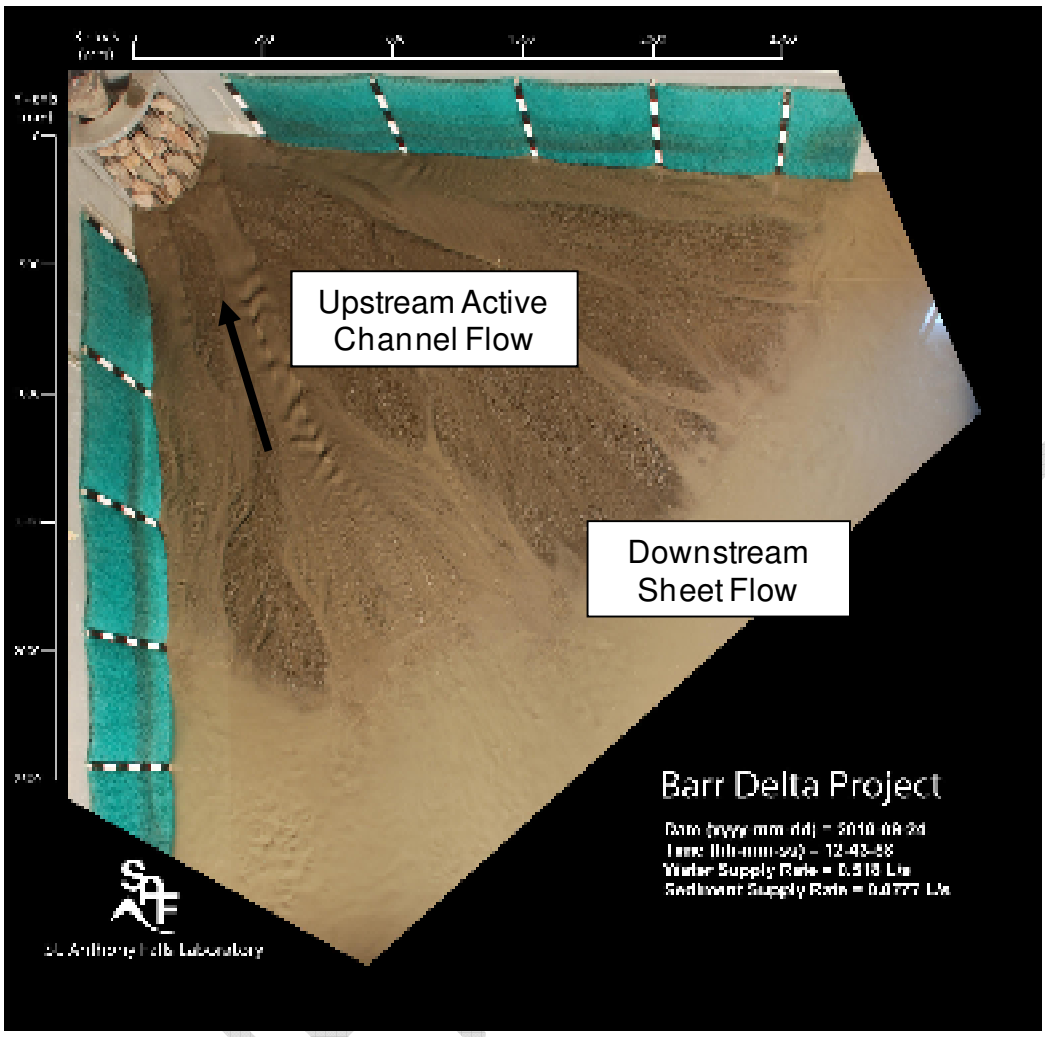

Figure 28a – Channel Evolution Step 1

As more sediment is delivered downstream, the downstream sheet flow zone grows a lobe and the channel cut begins to fill in.

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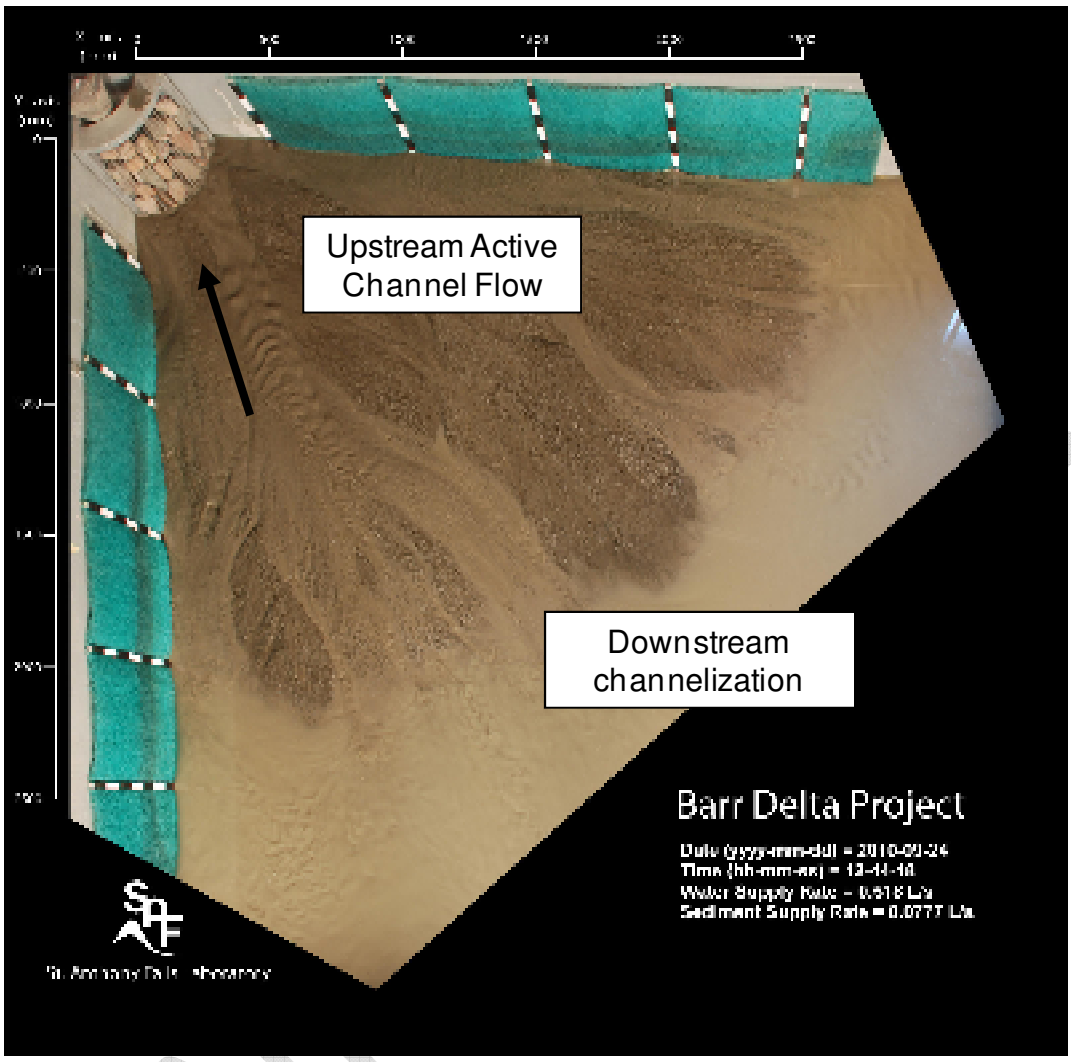

Figure 28b – Channel Evolution Step 2

Once the downstream deposit below the downstream lobe becomes too thick the downstream sheet flow splits into smaller channels, delivering sediment to the lower downstream areas. The upstream channel continues to fill in.

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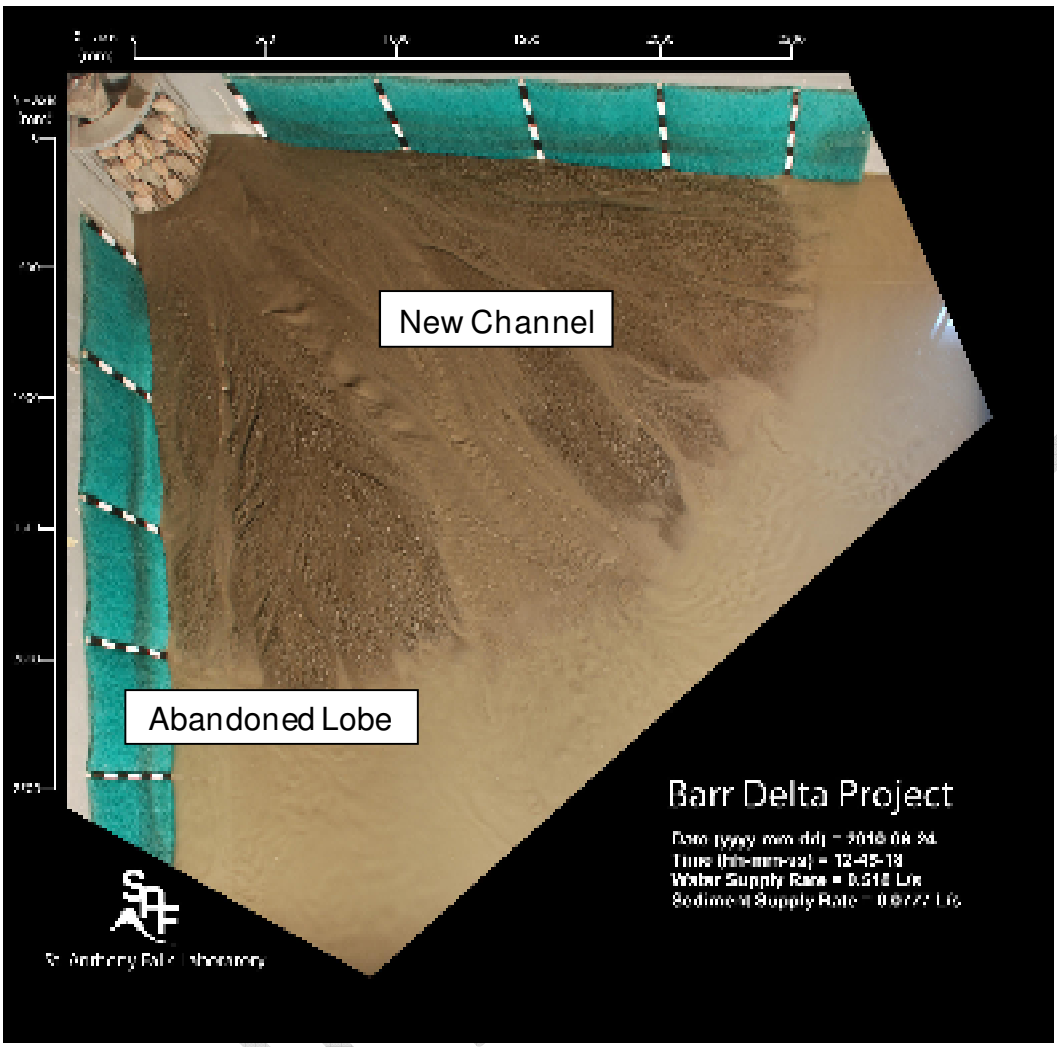

Figure 28c – Channel Evolution Step 3

Once the upstream channel fills in with sediment it avulses to a new location, cutting a new channel and abandoning the previous downstream lobe.

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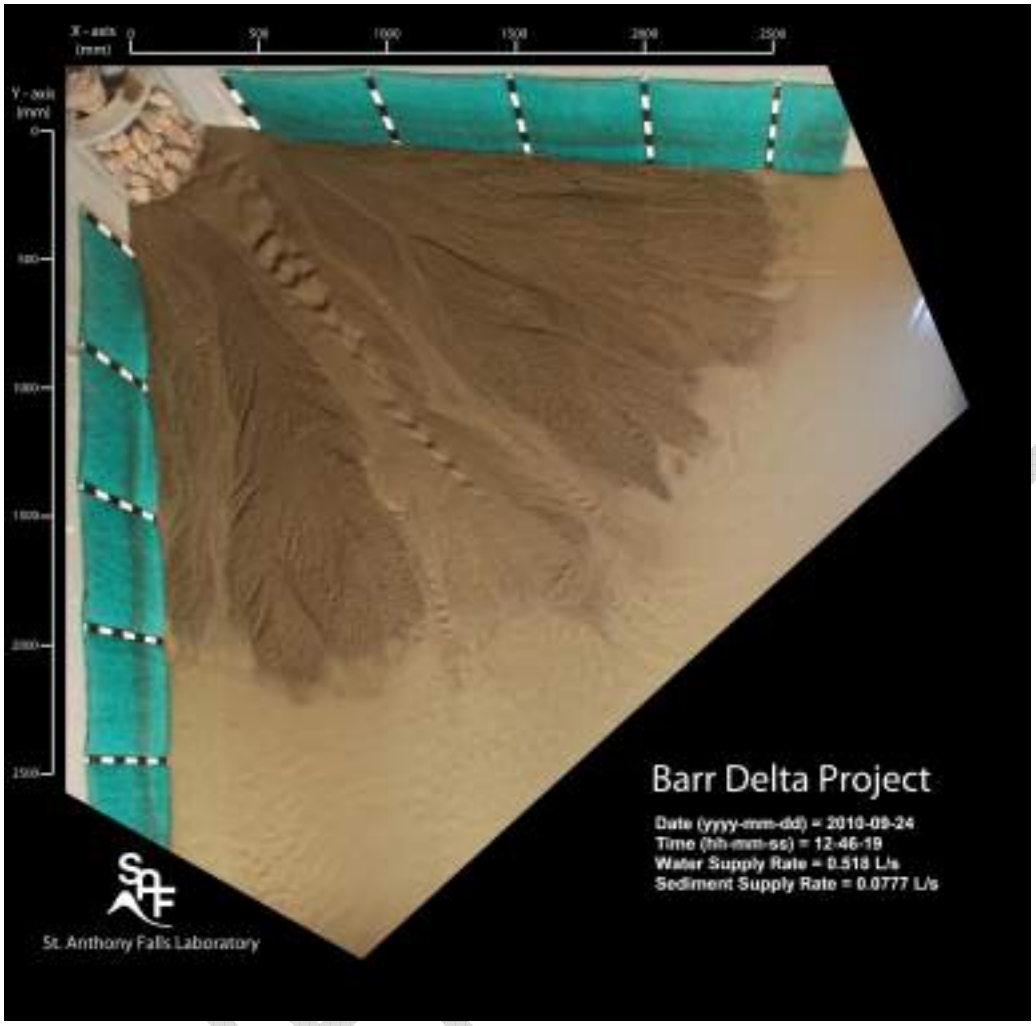

Figure 28d – Channel Evolution Step 4

Finally a new lobe forms at the downstream end of the new channel and the avulsion process start over again. This process has been observed over a range of scales and settings (Sheets and Hoyal 2009).

With these fines retention processes in mind, remember that the falling phase of the runs were designed to maximize channelization on the surface of the deposit. Maximum channelization results in minimum fines retention. This leads to the conclusion that the delta surface at the end of the falling phase should contain the absolute minimum fines that can be expected anywhere within the delta. This conclusion is supported by Figure 16g. From this conclusion one can expect a minimum of 30% fines retention everywhere within the delta.

The final deposit does show some grain size sorting characteristics. The upstream end of the delta spends more time under deep, narrow channels that generate thicker and more pronounced coarse lenses while the downstream end of the delta spends more time under shallower wider channels that generate thinner, more uniform lenses. In general, the results suggest that the field scale delta will have more pronounced coarse lenses at the upstream end of the fan with an

overall trend of downstream fining. Even these coarse lenses, however, should retain a fines fraction of ~30% or more.

At the upstream end the thickness and interconnected nature of the dark/coarse lenses indicate that infiltrating groundwater will likely bypass the light/fine lenses; consequently, groundwater transport will likely be greater at the upstream end of the delta than at the downstream end. This means that the hydraulic conductivity measurements taken in Section 5.2 can only be used as a lower limit of the effective hydraulic conductivity.

The laboratory experiment was designed to capture the main processes and channelization expected in the tailings pond delta. In other words, the laboratory delta is expected to grow and maintain its surface by the same mechanisms as the field scale delta. As such, the minimum fines retention and degree of grain size sorting seen in the experiment should be similar to that in the field scale delta. The field scale delta should also exhibit spatially variable hydraulic conductivity comparable to that seen in the laboratory scale delta.

6.2 Delta Drainage

Many experimental deltas have been built in the SAFL delta basins. In most cases the delta basin is drained at the end of the experiment and the delta is dry enough to be sliced in about a week. Six weeks after the completion of Run 1 the delta deposit was still near saturation. The following questions need to be addressed: 1) why did the laboratory scale delta retain water, and 2) will the field scale delta also retain water at near saturation? There are several possible explanations for this behavior which will be discussed in the following sections.

6.2.1 Low Permeability

Firstly, considering the primary difference between these laboratory scale deltas and previous delta experiments, the tailings used in this study had much higher fines content than previous experiments. In previous experiments the sediment was hand mixed. For both physical and safety reasons in previous experiments coal was used in place of the finest fraction of the sediment. The lower density of coal allows it to transport at a rate similar to the prototype (actual) fine material; however, the larger particle sizes of coal increase the permeability of the deposit. The inclusion of the “real” fine particles in the deposit greatly reduce the permeability of the deposit and thereby increase the drainage time; however, the decreased permeability alone is not sufficient to describe the prolonged water retention within the delta deposit.

6.2.2 Suspended Capillary Water

In order for water to be trapped in the delta deposit via capillary suction, the material must be unsaturated. In addition, soil suction and percent saturation are inversely related (see the SWCC provided by Daniel B. Stephens & Associates (DBS&A) Laboratory). The inclusion of fine material instead of coarser coal increases the soil suction pressures over previous experiments. The deposit was fully saturated at the end of the experimental run. The moment the delta began to drain the deposit became unsaturated. Figure 29 is a diagram of the system being investigated and the corresponding pressures.

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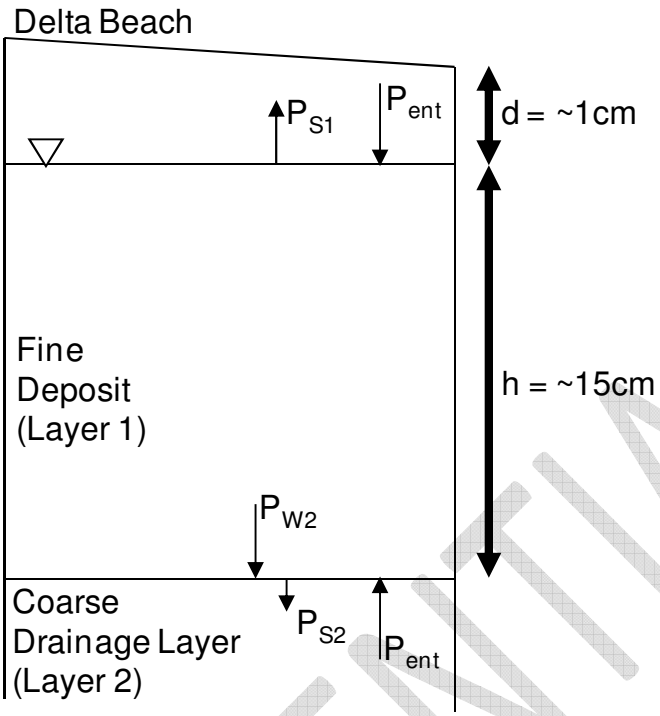

Figure 29 – Diagram of water column and associated pressures.

Where: P_{S1} = suction pressure of the deposit layer

P_{S2} = suction pressure of drainage layer

P_W = water pressure at bottom of the deposit

P_{ent} = atmospheric entrance pressure

The fine deposit layer has an upward suction pressure inversely related to the percent saturation. The coarse drainage layer has a much lower downward suction pressure. The water pressure balance for trapping water in the fine layer above a coarse layer is given by equation 11.

$$P_{S2} + P_W > P_{S1} \qquad (11)$$

The drainage layer suction pressure and the weight of the water act to drain water from the delta, while the deposit suction pressure acts to retain water within the delta. The suction pressure of the drainage layer is much lower than the suction pressure of the deposit and can be neglected. As such, if the water pressure due to the height of the deposit is equal to the suction pressure of the deposit, the water in the deposit will be retained. The water pressured generated by 15 cm (6in) of water is 1.47 kPa or 0.0174 bar. Based on the SWCC provided by DBS&A for the tailings material, the suction pressure of the deposit is equal to 0.0174 bar when the percent

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saturation is equal to 85-98%. In conclusion, suspended capillary water itself is sufficient to explain the water retention within the laboratory delta deposit.

6.2.3 Influence of Fluvial Deposition

The SWCC generated by DBS&A was created by testing artificially sorted and mixed tailings samples. It is entirely possible that the SWCC will be different for a fluvial deposit created from the same material. A flowing system may orient the particles differently or change the packing characteristics. This is a common problem in the construction industry. When mechanically placing construction fill it can be challenging to achieve compactions as high as field conditions and impossible to match them. The opposite is also true in laboratory settings; it can difficult to achieve loose compaction conditions or match fluvial deposition conditions.

Under fluvial deposition, angular fine particles may orient themselves and interlock in such a way that nearly all of the void space is composed of "micropores." Micropores are pore spaces so small that water contained within them will not drain by gravity and is only removed by suction and evaporation. The influence of fluvial deposition in itself is sufficient to explain the water retention found in the laboratory delta deposit.

6.2.4 Field Delta Drainage

The lack of drainage in the laboratory scale can be satisfactorily explained by suspended capillary water and/or fluvial deposition processes. Unfortunately the qualitative observations made of the laboratory scale delta drainage are insufficient to definitively say which process is responsible the water retention. It is most likely a combination of the processes.

In the field scale delta the thickness of the deposit means that suspended capillary water will not be able to retain water at near the saturation point as in the experiments, because the column pressure will be higher. On the other hand, if the fluvial deposition mechanism does actually generate the majority of void space as micropores, then the field scale delta could also retain a high percentage of water.

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7.0 Conclusions

Physical experiments were conducted to explore the transport and depositional characteristics of prototype tailings material. The experiments were conducted in two phases; the first phase used 1D flume experiments under field-scale conditions and Phase II involved 2D scaled experiments to investigate channelization and its consequences. The experimental findings are summarized here:

- Field scale slopes are anticipated to range from 0.5 to 2%. The 1% slope used for field scale design is reasonable.

- The delta will operate in a fluvial braided channelized regime characterized by multiple channels, rapid channel migration, and dynamic bar and bedform processes.

- The deposit will likely experience both vertical and horizontal sorting of grains. The coarsest portion of the deposit will be the proximal region of the delta and the distal region will have higher fines content.

- The multi-channel fluvial processes will create a deposit that has buried channels in the subsurface. The hydraulic interconnectedness of these channels has not been explored quantitatively in this study but visual observation suggests that channels will be connected vertically and laterally, especially in the upstream part of the deposit.

- Even under the most extreme plausible transport conditions, it was difficult to generate a deposit with less than 30% fines content in the deposit. This suggests that, even though channelization generally moves fine material offshore, bypassing the subaerial delta, a significant fraction of the fines in the tailings material is co-deposited with coarser material. These levels of trapped fines will significantly reduce conductivity relative to a well sorted deposit of the same median size.

- Hydraulic conductivities measured on piston cores taken from the laboratory deposits are relatively low due to conductivity in cores being limited by the presence of fine layers in the cores. Because 3D groundwater flow in the field may exploit connected high-conductivity pathways through the deposit, the values measured with cores represent the low end of conductivities expected for field conditions.

- The degree of water retention in the field scale deposit was inconclusive. The estimated deposit thickness and the SWCC for the tailings suggests that suction will not be great enough to keep the deposit saturated; however, internal structures such as lenses, discontinuities, or micro-pores associated with the natural deposition of the deposit may increase the suction pressure of the material.

There are several design alternatives that could potentially increase fines retention. Reducing the water content of the slurry such that the delta/beach behaves as a debris (“mud”) flow would be expected to enhance fines retention although no tests were done on this transport mode in this

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study. The delta could be operated such that there are periods of receding shoreline produced by raising the pool level. Switching between multiple spigot locations could also be timed to achieve a similar effect. These alternatives focus only on increasing fines retention. There may be other physical or operational considerations that make them unfeasible.

8.0 References

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Appendix A – Feed Material Grain Size Distribution

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Appendix B – Scrape Sample Grain Size Analysis

Run #	Phase	Sample #	θ	r (cm)	% > 74μ	% < 74μ	Coarse/Fine Fraction
1	End of Growth	1	π/8	25	0.63	0.37	1.67
1	End of Growth	2	π/8	40	0.55	0.45	1.23
1	End of Growth	3	π/8	55	0.66	0.34	1.91
1	End of Growth	4	π/8	70	0.72	0.28	2.62
1	End of Growth	5	π/8	85	0.73	0.27	2.76
1	End of Growth	6	π/8	100	0.70	0.30	2.29
1	End of Growth	7	π/8	115	0.68	0.32	2.08
1	End of Growth	8	π/8	130	0.63	0.37	1.68
1	End of Growth	9	π/8	145	0.56	0.44	1.27
1	End of Growth	10	3π/16	25	0.53	0.47	1.11
1	End of Growth	11	3π/16	40	0.59	0.41	1.46
1	End of Growth	12	3π/16	55	0.74	0.26	2.91
1	End of Growth	13	3π/16	70	0.69	0.31	2.23
1	End of Growth	14	3π/16	85	0.58	0.42	1.41
1	End of Growth	15	3π/16	100	0.59	0.41	1.45
1	End of Growth	16	3π/16	115	0.65	0.35	1.82
1	End of Growth	17	3π/16	130	0.64	0.36	1.77
1	End of Growth	19	π/4	25	0.39	0.61	0.64
1	End of Growth	20	π/4	40	0.54	0.46	1.17
1	End of Growth	21	π/4	55	0.58	0.42	1.40
1	End of Growth	22	π/4	70	0.58	0.42	1.38
1	End of Growth	23	π/4	85	0.22	0.78	0.28
1	End of Growth	24	π/4	100	0.44	0.56	0.78
1	End of Growth	25	π/4	115	0.51	0.49	1.04
1	End of Growth	26	π/4	130	0.54	0.46	1.15
1	End of Growth	28	5π/16	25	0.46	0.54	0.85
1	End of Growth	30	5π/16	55	0.71	0.29	2.41
1	End of Growth	31	5π/16	70	0.62	0.38	1.62
1	End of Growth	32	5π/16	85	0.57	0.43	1.35
1	End of Growth	33	5π/16	100	0.52	0.48	1.10
1	End of Growth	34	5π/16	115	0.53	0.47	1.11

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Run #	Phase	Sample #	θ	r (cm)	% > 74μ	% < 74μ	Coarse/Fine Fraction
1	End of Growth	35	5π/16	130	0.54	0.46	1.17
1	End of Growth	36	5π/16	145	0.40	0.60	0.66
1	End of Growth	37	3π/8	25	0.36	0.64	0.57
1	End of Growth	38	3π/8	40	0.55	0.45	1.20
1	End of Growth	39	3π/8	55	0.56	0.44	1.28
1	End of Growth	40	3π/8	70	0.68	0.32	2.12
1	End of Growth	41	3π/8	85	0.63	0.37	1.72
1	End of Growth	42	3π/8	100	0.57	0.43	1.32
1	End of Growth	43	3π/8	115	0.53	0.47	1.13
1	End of Growth	44	3π/8	130	0.51	0.49	1.03
1	End of Growth	45	3π/8	145	0.47	0.53	0.87
1	End of Falling	1	π/8	25	0.45	0.55	0.81
1	End of Falling	2	π/8	40	0.47	0.53	0.89
1	End of Falling	3	π/8	55	0.62	0.38	1.60
1	End of Falling	4	π/8	70	0.41	0.59	0.69
1	End of Falling	5	π/8	85	0.38	0.62	0.60
1	End of Falling	6	π/8	100	0.37	0.63	0.59
1	End of Falling	7	π/8	115	0.40	0.60	0.67
1	End of Falling	8	π/8	130	0.33	0.67	0.50
1	End of Falling	9	π/8	145	0.60	0.40	1.50
1	End of Falling	10	3π/16	25	0.54	0.46	1.16
1	End of Falling	11	3π/16	40	0.49	0.51	0.95
1	End of Falling	12	3π/16	55	0.59	0.41	1.42
1	End of Falling	13	3π/16	70	0.31	0.69	0.44
1	End of Falling	14	3π/16	85	0.55	0.45	1.21
1	End of Falling	15	3π/16	100	0.47	0.53	0.88
1	End of Falling	16	3π/16	115	0.41	0.59	0.70
1	End of Falling	17	3π/16	130	0.38	0.62	0.62
1	End of Falling	18	3π/16	145	0.46	0.54	0.85
1	End of Falling	19	π/4	25	0.46	0.54	0.85
1	End of Falling	20	π/4	40	0.47	0.53	0.90
1	End of Falling	21	π/4	55	0.49	0.51	0.95
1	End of Falling	22	π/4	70	0.48	0.52	0.94

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Run #	Phase	Sample #	θ	r (cm)	% > 74μ	% < 74μ	Coarse/Fine Fraction
1	End of Falling	23	π/4	85	0.45	0.55	0.82
1	End of Falling	24	π/4	100	0.47	0.53	0.87
1	End of Falling	25	π/4	115	0.46	0.54	0.86
1	End of Falling	26	π/4	130	0.48	0.52	0.92
1	End of Falling	27	π/4	145	0.39	0.61	0.64
1	End of Falling	28	5π/16	25	0.60	0.40	1.52
1	End of Falling	29	5π/16	40	0.61	0.39	1.57
1	End of Falling	30	5π/16	55	0.52	0.48	1.09
1	End of Falling	31	5π/16	70	0.39	0.61	0.65
1	End of Falling	32	5π/16	85	0.00	1.00	0.00
1	End of Falling	33	5π/16	100	0.00	1.00	0.00
1	End of Falling	34	5π/16	115	0.47	0.53	0.90
1	End of Falling	35	5π/16	130	0.47	0.53	0.89
1	End of Falling	36	5π/16	145	0.36	0.64	0.57
1	End of Falling	37	3π/8	25	0.72	0.28	2.63
1	End of Falling	38	3π/8	40	0.71	0.29	2.49
1	End of Falling	39	3π/8	55	0.59	0.41	1.46
1	End of Falling	40	3π/8	70	0.55	0.45	1.22
1	End of Falling	41	3π/8	85	0.72	0.28	2.51
1	End of Falling	42	3π/8	100	0.69	0.31	2.19
1	End of Falling	43	3π/8	115	0.72	0.28	2.57
1	End of Falling	44	3π/8	130	0.45	0.55	0.82
1	End of Falling	45	3π/8	145	0.56	0.44	1.27
2	End of Growth	1	π/8	25	0.55	0.45	1.23
2	End of Growth	2	π/8	40	0.66	0.34	1.95
2	End of Growth	3	π/8	55	0.70	0.30	2.31
2	End of Growth	4	π/8	70	0.53	0.47	1.12
2	End of Growth	5	π/8	85	0.52	0.48	1.09
2	End of Growth	6	π/8	100	0.46	0.54	0.85
2	End of Growth	7	π/8	115	0.50	0.50	0.99
2	End of Growth	8	π/8	130	0.44	0.56	0.79
2	End of Growth	9	π/8	145	0.28	0.72	0.39
2	End of Growth	10	3π/16	25	0.64	0.36	1.78

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Run #	Phase	Sample #	θ	r (cm)	% > 74μ	% < 74μ	Coarse/Fine Fraction
2	End of Growth	11	3π/16	40	0.54	0.46	1.17
2	End of Growth	12	3π/16	55	0.58	0.42	1.36
2	End of Growth	13	3π/16	70	0.36	0.64	0.57
2	End of Growth	14	3π/16	85	0.37	0.63	0.59
2	End of Growth	15	3π/16	100	0.29	0.71	0.40
2	End of Growth	16	3π/16	115	0.27	0.73	0.36
2	End of Growth	17	3π/16	130	0.27	0.73	0.36
2	End of Growth	18	3π/16	145	0.57	0.43	1.33
2	End of Growth	19	π/4	25	0.57	0.43	1.33
2	End of Growth	20	π/4	40	0.53	0.47	1.15
2	End of Growth	21	π/4	55	0.46	0.54	0.85
2	End of Growth	22	π/4	70	0.44	0.56	0.77
2	End of Growth	23	π/4	85	0.31	0.69	0.44
2	End of Growth	24	π/4	100	0.30	0.70	0.43
2	End of Growth	25	π/4	115	0.42	0.58	0.71
2	End of Growth	26	π/4	130	0.25	0.75	0.34
2	End of Growth	28	5π/16	25	0.51	0.49	1.04
2	End of Growth	29	5π/16	40	0.44	0.56	0.78
2	End of Growth	30	5π/16	55	0.37	0.63	0.59
2	End of Growth	31	5π/16	70	0.36	0.64	0.55
2	End of Growth	32	5π/16	85	0.40	0.60	0.68
2	End of Growth	33	5π/16	100	0.33	0.67	0.48
2	End of Growth	34	5π/16	115	0.37	0.63	0.59
2	End of Growth	35	5π/16	130	0.16	0.84	0.19
2	End of Growth	36	5π/16	145	0.28	0.72	0.39
2	End of Growth	37	3π/8	25	0.39	0.61	0.63
2	End of Growth	38	3π/8	40	0.43	0.57	0.76
2	End of Growth	39	3π/8	55	0.49	0.51	0.95
2	End of Growth	40	3π/8	70	0.48	0.52	0.91
2	End of Growth	41	3π/8	85	0.52	0.48	1.10
2	End of Growth	42	3π/8	100	0.50	0.50	1.01
2	End of Growth	43	3π/8	115	0.49	0.51	0.97
2	End of Growth	44	3π/8	130	0.41	0.59	0.69

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Run #	Phase	Sample #	θ	r (cm)	% > 74μ	% < 74μ	Coarse/Fine Fraction
2	End of Growth	45	3π/8	145	0.24	0.76	0.32
2	End of Building	1	π/8	25	0.65	0.35	1.82
2	End of Building	2	π/8	40	0.51	0.49	1.05
2	End of Building	3	π/8	55	0.54	0.46	1.19
2	End of Building	4	π/8	70	0.58	0.42	1.36
2	End of Building	5	π/8	85	0.46	0.54	0.86
2	End of Building	6	π/8	100	0.46	0.54	0.85
2	End of Building	7	π/8	115	0.48	0.52	0.92
2	End of Building	8	π/8	130	0.33	0.67	0.50
2	End of Building	9	π/8	145	0.40	0.60	0.67
2	End of Building	10	3π/16	25	0.45	0.55	0.82
2	End of Building	11	3π/16	40	0.40	0.60	0.67
2	End of Building	12	3π/16	55	0.51	0.49	1.06
2	End of Building	13	3π/16	70	0.36	0.64	0.56
2	End of Building	14	3π/16	85	0.35	0.65	0.54
2	End of Building	15	3π/16	100	0.39	0.61	0.63
2	End of Building	16	3π/16	115	0.38	0.62	0.62
2	End of Building	17	3π/16	130	0.40	0.60	0.65
2	End of Building	18	3π/16	145	0.47	0.53	0.88
2	End of Building	19	π/4	25	0.43	0.57	0.76
2	End of Building	20	π/4	40	0.42	0.58	0.72
2	End of Building	21	π/4	55	0.40	0.60	0.66
2	End of Building	22	π/4	70	0.41	0.59	0.70
2	End of Building	23	π/4	85	0.64	0.36	1.74
2	End of Building	24	π/4	100	0.45	0.55	0.81
2	End of Building	25	π/4	115	0.44	0.56	0.78
2	End of Building	26	π/4	130	0.16	0.84	0.19
2	End of Building	27	π/4	145	0.46	0.54	0.86
2	End of Building	28	5π/16	25	0.66	0.34	1.92
2	End of Building	29	5π/16	40	0.65	0.35	1.88
2	End of Building	30	5π/16	55	0.54	0.46	1.20
2	End of Building	31	5π/16	70	0.40	0.60	0.66
2	End of Building	32	5π/16	85	0.46	0.54	0.84

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Run #	Phase	Sample #	θ	r (cm)	% > 74μ	% < 74μ	Coarse/Fine Fraction
2	End of Building	33	$5\pi/16$	100	0.37	0.63	0.59
2	End of Building	34	$5\pi/16$	115	0.43	0.57	0.77
2	End of Building	35	$5\pi/16$	130	0.49	0.51	0.96
2	End of Building	36	$5\pi/16$	145	0.49	0.51	0.95
2	End of Building	37	$3\pi/8$	25	0.61	0.39	1.53
2	End of Building	38	$3\pi/8$	40	0.54	0.46	1.18
2	End of Building	39	$3\pi/8$	55	0.46	0.54	0.85
2	End of Building	40	$3\pi/8$	70	0.54	0.46	1.15
2	End of Building	44	$3\pi/8$	130	0.42	0.58	0.73
2	End of Building	45	$3\pi/8$	145	0.33	0.67	0.50
2	End of Falling	1	$\pi/8$	25	0.67	0.33	2.01
2	End of Falling	2	$\pi/8$	40	0.60	0.40	1.48
2	End of Falling	3	$\pi/8$	55	0.69	0.31	2.25
2	End of Falling	4	$\pi/8$	70	0.63	0.37	1.72
2	End of Falling	5	$\pi/8$	85	0.58	0.42	1.37
2	End of Falling	6	$\pi/8$	100	0.49	0.51	0.96
2	End of Falling	7	$\pi/8$	115	0.51	0.49	1.06
2	End of Falling	8	$\pi/8$	130	0.49	0.51	0.94
2	End of Falling	9	$\pi/8$	145	0.65	0.35	1.83
2	End of Falling	10	$3\pi/16$	25	0.68	0.32	2.17
2	End of Falling	11	$3\pi/16$	40	0.49	0.51	0.98
2	End of Falling	12	$3\pi/16$	55	0.46	0.54	0.86
2	End of Falling	13	$3\pi/16$	70	0.46	0.54	0.86
2	End of Falling	14	$3\pi/16$	85	0.42	0.58	0.71
2	End of Falling	15	$3\pi/16$	100	0.43	0.57	0.76
2	End of Falling	16	$3\pi/16$	115	0.42	0.58	0.72
2	End of Falling	17	$3\pi/16$	130	0.45	0.55	0.81
2	End of Falling	18	$3\pi/16$	145	0.44	0.56	0.77
2	End of Falling	19	$\pi/4$	25	0.58	0.42	1.37
2	End of Falling	20	$\pi/4$	40	0.43	0.57	0.76
2	End of Falling	21	$\pi/4$	55	0.52	0.48	1.06
2	End of Falling	22	$\pi/4$	70	0.39	0.61	0.63
2	End of Falling	23	$\pi/4$	85	0.42	0.58	0.73

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Run #	Phase	Sample #	θ	r (cm)	% > 74μ	% < 74μ	Coarse/Fine Fraction
2	End of Falling	24	π/4	100	0.53	0.47	1.12
2	End of Falling	25	π/4	115	0.43	0.57	0.77
2	End of Falling	26	π/4	130	0.37	0.63	0.60
2	End of Falling	27	π/4	145	0.38	0.62	0.61
2	End of Falling	28	5π/16	25	0.53	0.47	1.13
2	End of Falling	29	5π/16	40	0.49	0.51	0.97
2	End of Falling	30	5π/16	55	0.58	0.42	1.40
2	End of Falling	31	5π/16	70	0.50	0.50	1.01
2	End of Falling	32	5π/16	85	0.46	0.54	0.85
2	End of Falling	33	5π/16	100	0.47	0.53	0.88
2	End of Falling	34	5π/16	115	0.41	0.59	0.70
2	End of Falling	35	5π/16	130	0.55	0.45	1.23
2	End of Falling	36	5π/16	145	0.58	0.42	1.36
2	End of Falling	37	3π/8	25	0.61	0.39	1.59
2	End of Falling	38	3π/8	40	0.58	0.42	1.39
2	End of Falling	39	3π/8	55	0.65	0.35	1.83
2	End of Falling	40	3π/8	70	0.41	0.59	0.70
2	End of Falling	41	3π/8	85	0.39	0.61	0.65
2	End of Falling	42	3π/8	100	0.30	0.70	0.44
2	End of Falling	43	3π/8	115	0.51	0.49	1.04
2	End of Falling	44	3π/8	130	0.59	0.41	1.44
2	End of Falling	45	3π/8	145	0.31	0.69	0.45

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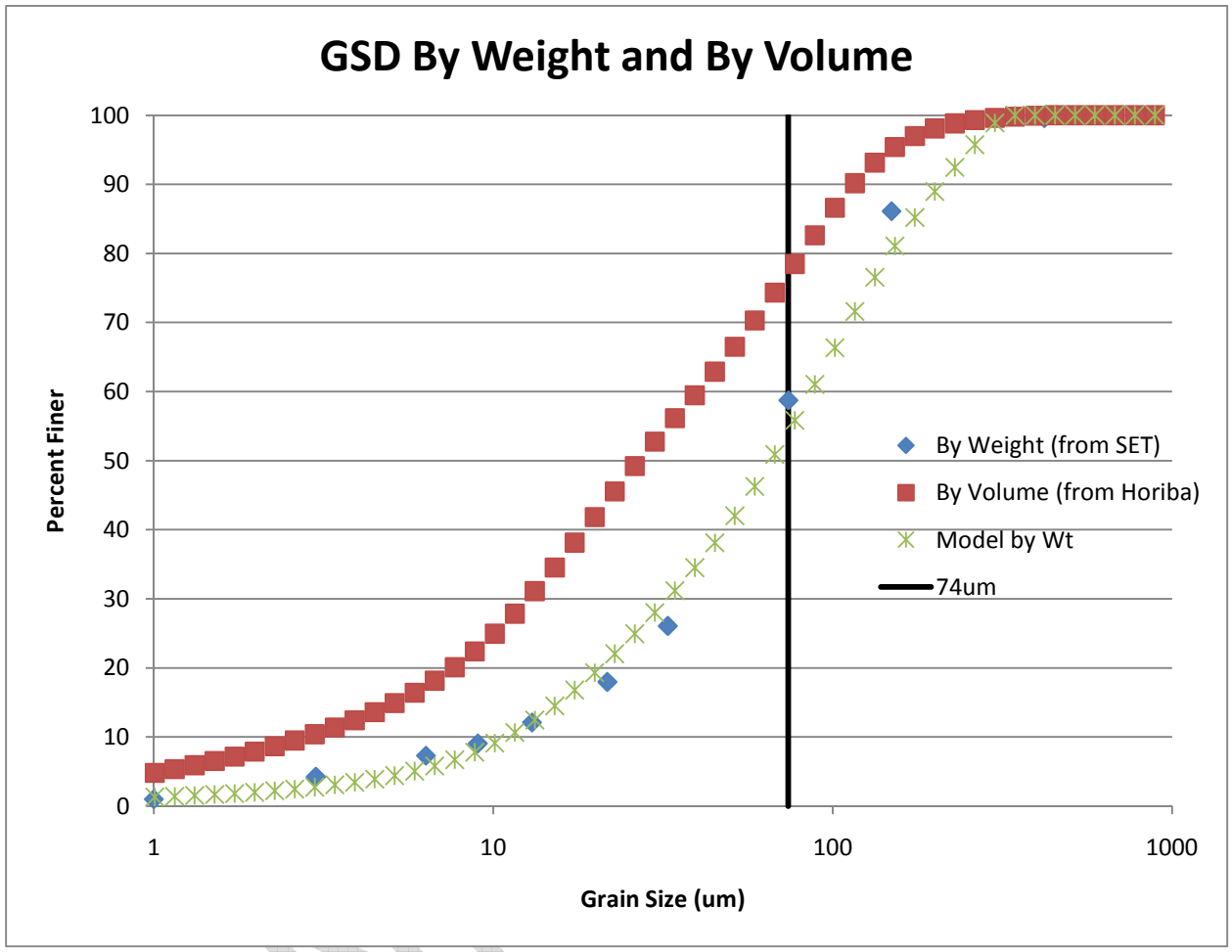
Appendix C – Comparison of Grain Size Distribution by Volume (Horiba® Laser Diffraction) and by Weight (Sieve and Hydrometer Tests from SET Labs)



The percent finer by weight can be estimated using the following equation:

$$\%Finer_W = \%Finer_V \cdot F \cdot [A \cdot ln(C \cdot Dia + E) + B] + G$$

Where: $\%Finer_W$ = Percent finer by weight

$\%Finer_V$ = Percnet Finer by volume

Dia = Particle diameter (um)

A = 0.214826

B = -0.21006

C = 0.83509

E = 4.470231

F = 1.006547

G = 0.574739

Appendix D – Videos

See supplemental CD or attached video files for Phase I video and Phase II 20-second time laps aerial videos of each run.

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Attachment F

Update on Tailings Humidity Cell Test Data

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Memo

To:	Peter Hinck	**Date:**	December 23, 2011
Company:	Barr	**From:**	Laura Donkervoort Stephen Day
Copy to:	Jim Scott, PolyMet	**Project #:**	1UP005.001
Subject:	Update on Tailings Humidity Cell Test Data, NorthMet Project – DRAFT		

1 Introduction

Kinetic geochemical testing of samples of tailings produced by pilot plant processing of NorthMet Project ores began in 2005 when the first process testing occurred. Subsequently, PolyMet has repeated similar pilot plant testing three times to optimize the flotation process and evaluate recovery of concentrate for different ore composites. Each time a pilot plant was operated, the resulting tailings samples have been submitted for the same geochemical test procedures including solids analysis for sulfur and trace element content and kinetic testing on whole samples and size fractions.

Fundamentally, the flowsheet of the flotation process has not changed except for copper sulfate not being used in the very first runs of Pilot Plant 1. Due to the improvement in recovery of sulfide minerals (and consequent reduction in sulfide content of tailings), copper sulfate has been used in all subsequent pilot plants. As a result, it is appropriate to consider all tailings results as a single dataset rather than individual datasets for each pilot plant run.

Separate tailings testwork has been initiated on four samples of LTVSMC tailings which will be used for construction of tailings embankments. As these tests are being performed on completely different materials, they are described separately at the end of this memorandum.

The purpose of the memorandum is to provide an update on tailings humidity cell data and their influence on inputs to water quality prediction models. A previous update (RS82 - Update on Use of Kinetic Test Data for Water Quality Predictions, February 2, 2009) was prepared with approximately 3 years data from NorthMet Project pilot plant samples generated in 2005 and 2006. Since that time, an additional three years of data have been collected for these tests and two additional data sets have been generated from tailings produced by the two additional pilot plants run in 2008 and 2009.

Due to the change in conceptualization of the overall modeling approach from the use of metal to sulfate ratios of release rates to the use mainly of sulfate release rates and metal to sulfur ratios in solids to predict pore water chemistry, this update report focuses mainly on a comparison of trends in pH and sulfate between the tests.

2 NorthMet Project Tailings Kinetic Tests

2.1 Status of Program

Table 1 lists all tailings humidity cells. The test program consists of conventional humidity cells with parallel tests using the MDNR reactor configuration on bulk tailings and tailings size fractions. All cells are currently running smoothly however, some repairs have occurred in addition to those stated in the RS82 report.

The filter fabrics in T55 and T62 humidity cells were found to be torn and allowing the breakthrough of solids into the leachate collection chamber. The cells were dismantled and repaired in September 2009. All solids were recovered and returned to the respective cells. Following these repairs, leachate chemistry re-stabilized within 2 weeks, returning to pre-repair levels. Arsenic levels were slower to decline, taking 7 weeks to return to pre-repair levels. This type of occurrence has been previously observed in the RS82 report.

All tests originally started are continuing and none of the tests have been terminated. Laboratory and analytical methods are unchanged from that developed in consultation with the MDNR and are consistent between all tests included in this update report.

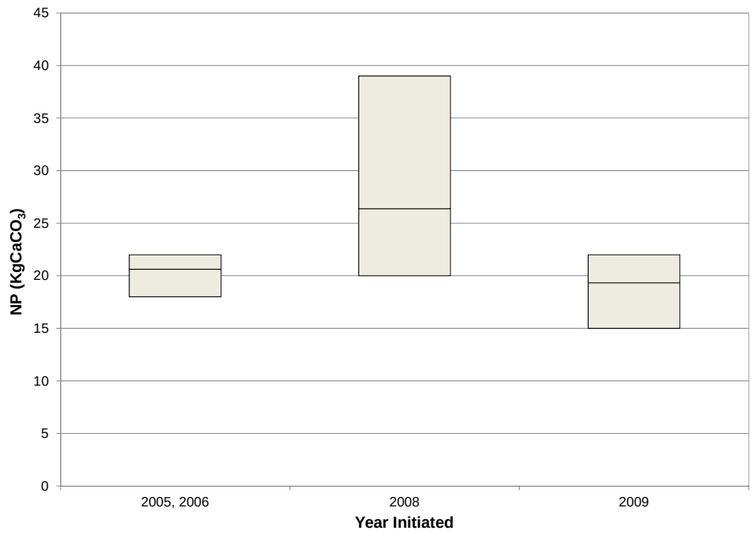
Table 1: Tailings Humidity Cells Used as Basis for Update Report

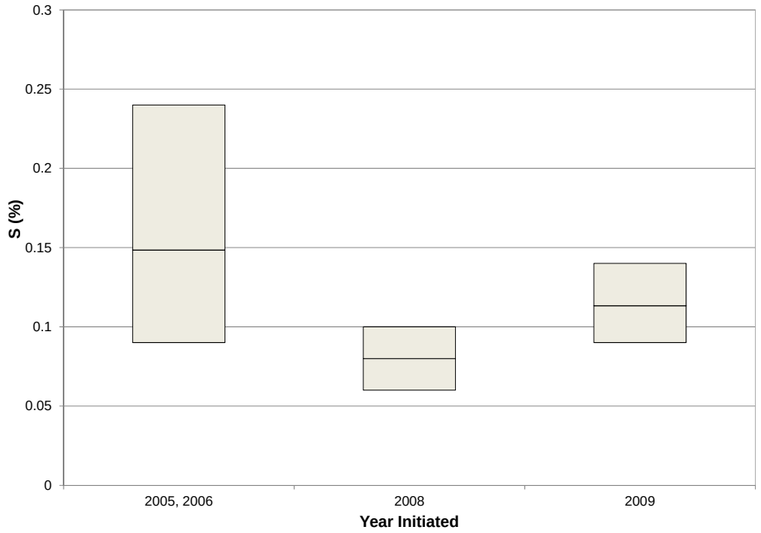
HCT ID	Fraction	HCT Full ID	Total Sulfur %	Initial Start Date	Total Weeks
T1	Whole	P1 (CuSO4)	0.1	9/8/2005	304
T2	Whole	P1 (no CuSO4)	0.23	9/8/2005	304
T3	Whole	P2 (no CuSO4)	0.2	9/8/2005	304
T4	Whole	P3 (CuSO4)	0.15	9/8/2005	304
T5	+100	Parcel 2 P2S +100 mesh	0.15	2/10/2006	282
T6	-100+200	Parcel 2 P2S -100 +200 mesh	0.17	2/10/2006	282
T7	-200	Parcel 2 P2S -200 mesh	0.24	2/10/2006	282
T8	+100	Parcel 1-2 PISCS +100 mesh	0.11	2/10/2006	282
T9	-100+200	Parcel 1-2 PISCS -100 +200 mesh	0.1	2/10/2006	282
T10	-200	Parcel 1-2 PISCS -200 mesh	0.09	2/10/2006	282
T11	+100	Parcel 3 P3S +100 mesh	0.11	2/10/2006	282
T12	-100+200	Parcel 3 P3S -100 +200 mesh	0.14	2/10/2006	282
T13	-200	Parcel 3 P3S -200 mesh	0.14	2/10/2006	282
T52	Whole	Pilot Plant 2 (0.30 Cu feed) Composite Bulk Tailings (as received)	0.07	7/8/2008	156
T53	+100	Pilot Plant 2 (0.30 Cu feed) Composite (+100 mesh)	0.08	7/8/2008	156
T54	-100+200	Pilot Plant 2 (0.30 Cu feed) Composite (-100 + 200 mesh)	0.06	7/8/2008	156
T55	-200	Pilot Plant 2 (0.30 Cu feed) Composite (-200 mesh)	0.09	7/8/2008	156
T56	Whole	Pilot Plant 3 (0.25 Cu feed) Composite Bulk Tailings (as received)	0.08	7/8/2008	156
T57	+100	Pilot Plant 3 (0.25 Cu feed) Composite (+100 mesh)	0.1	7/8/2008	156
T58	-100+200	Pilot Plant 3 (0.25 Cu feed) Composite (-100 + 200 mesh)	0.08	7/8/2008	156
T59	-200	Pilot Plant 3 (0.25 Cu feed) Composite (-200 mesh)	0.08	7/8/2008	156
T60	Whole	SCAV TAILS 9/30/09 1600	0.09	11/24/2009	84
T61	+100	SCAV TAILS 9/30/09 1600 +100	0.1	11/24/2009	84
T62	-100+200	SCAV TAILS 9/30/09 1600 -100+200	0.09	11/24/2009	84
T63	-200	SCAV TAILS 9/30/09 1600 -200	0.11	11/24/2009	84
T64	Whole	SCAV TAILS 10/1/09 09.00	0.13	11/24/2009	84
T65	+100	SCAV TAILS 10/1/09 09.00 +100	0.11	11/24/2009	84
T66	-100+200	SCAV TAILS 10/1/09 09.00 -100+200	0.14	11/24/2009	84
T67	-200	SCAV TAILS 10/1/09 09.00 -200	0.14	11/24/2009	84
T68	Whole	SCAV TAILS 10/1/09 17.00	0.12	11/24/2009	84
T69	+100	SCAV TAILS 10/1/09 17.00 +100	0.1	11/24/2009	84
T70	-100+200	SCAV TAILS 10/1/09 17.00 -100+200	0.1	11/24/2009	84
T71	-200	SCAV TAILS 10/1/09 17.00 -200	0.13	11/24/2009	84

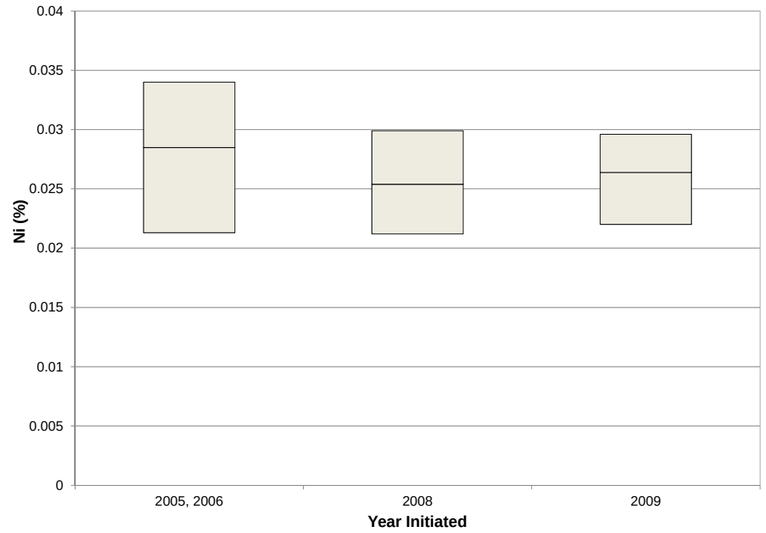
2.2 Comparison of Solids Data

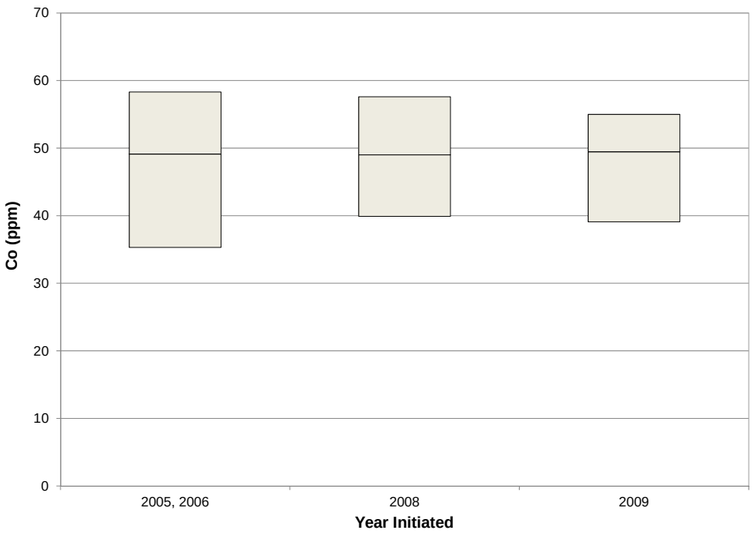
Solids data was compared between the three sets of humidity cells, (those initiated in 2005 & 2006, 2008 and 2009). Figure 1 shows the minimum, average and maximum values for key parameters.

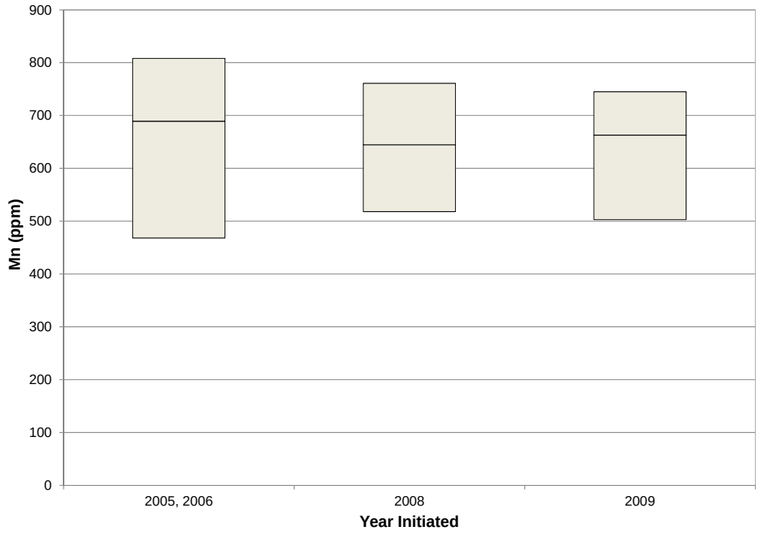
Sulfur concentrations showed the greatest range in the early dataset due to evaluation of the use of copper sulfate to optimize recovery of sulfide minerals by flotation. Subsequent pilot plants have used copper sulfate. Metal concentrations were very similar between the three data sets with the exception of copper, which showed slightly higher average and maximum values in the cells initiated 2005 and 2006.











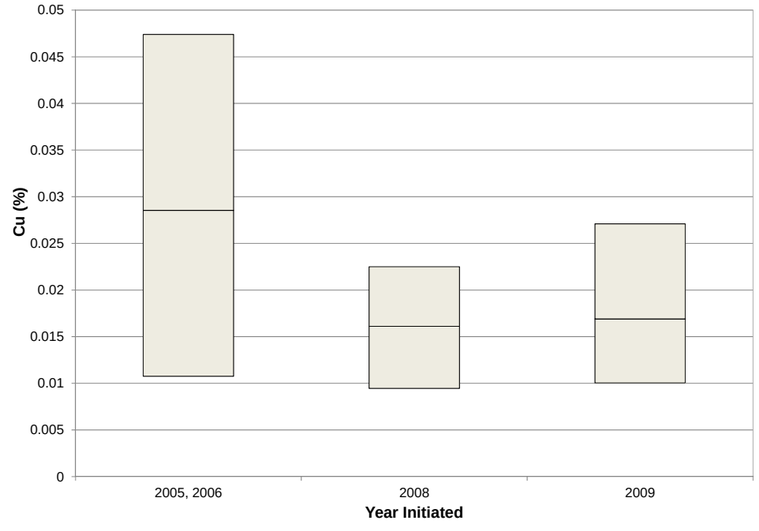


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Figure 1: Comparison of Solids data using Minimum, Average and Maximum Values – Key Results

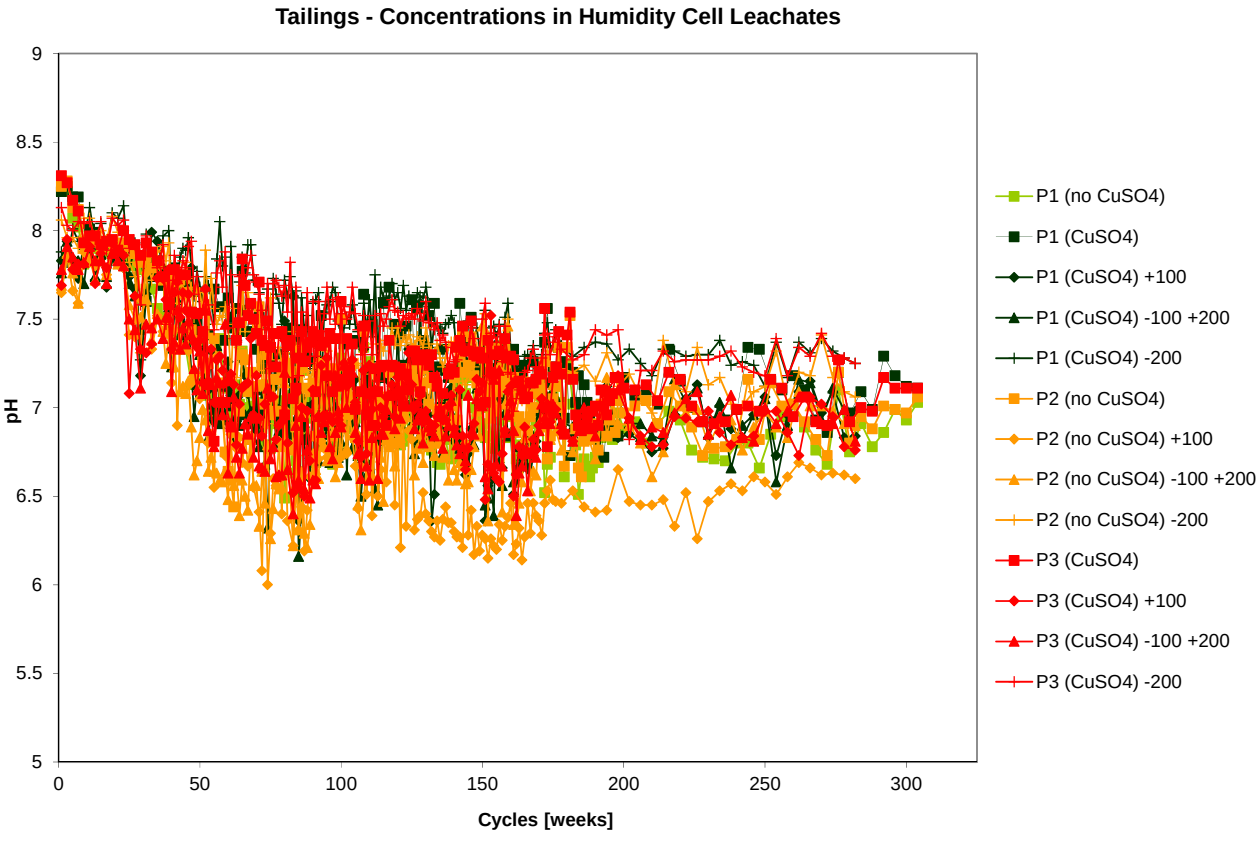
2.3 Description of Humidity Cell Results

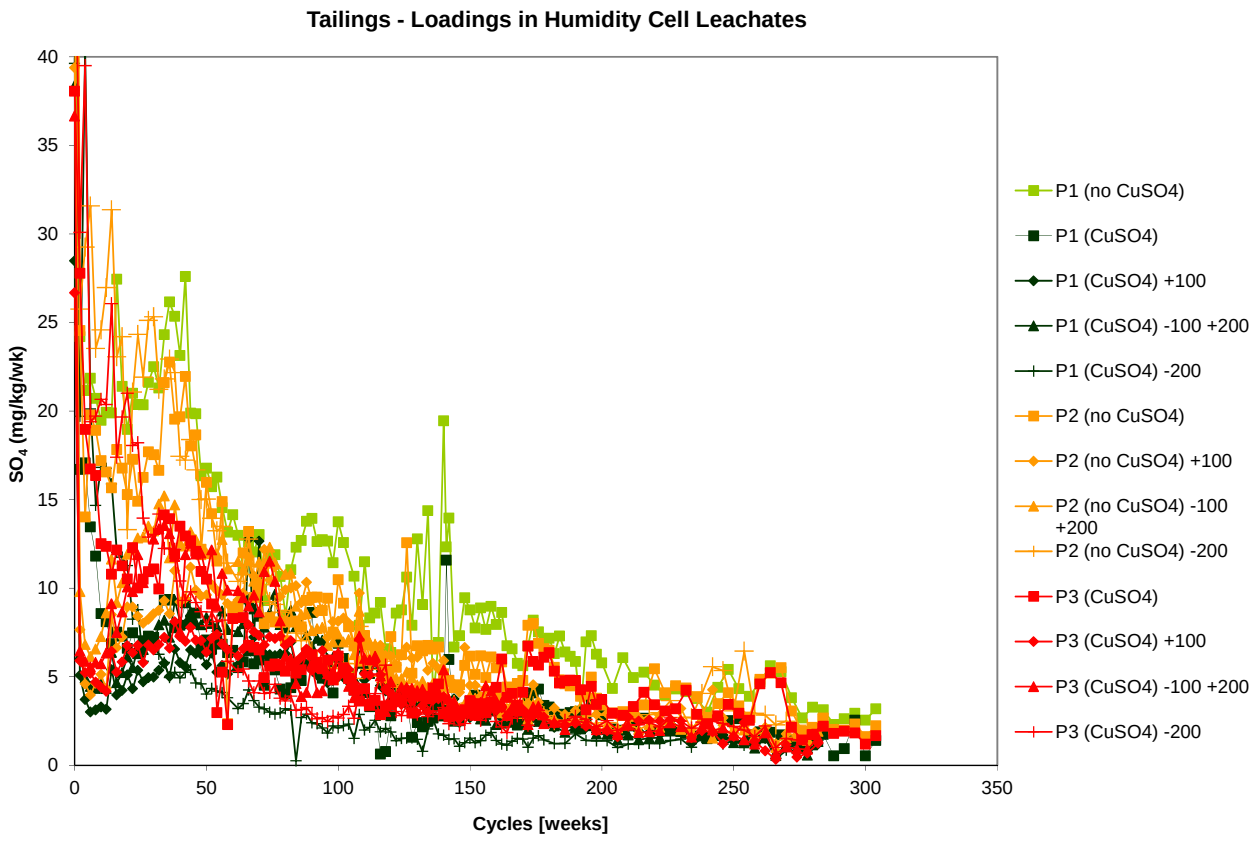
2.3.1 Tests Started in 2005 & 2006

All humidity cell tests showed an initial decline in pH (Figure 2). The decline was greatest for the two coarser particles size fractions and least for the -200 mesh fraction samples. The trend for bulk tailings was between the coarse and fine fractions. The range in leachate pHs following the main initial decline was 6.0 to 7.8. Leachate pHs remain somewhat erratic but no decline to lower pHs has been apparent and pH appears to be stabilizing with time. Variation reflects whether copper sulfate was used in the pilot mineral processing circuit to improve recovery of sulfide minerals during flotation. Higher pH leachates are generally associated with samples with lower sulfur contents produced by the use of copper sulfate.

Sulfate release has shown a decline from week zero to 200. After week 200, sulfate release has generally stabilized (Figure 2). Major ions have followed sulfate and were generally decreasing followed by stabilization after week 200. Alkalinity leaching from the two coarser particle sizes declined sharply and then stabilized, with the stable period starting at approximately week 75. In contrast, alkalinity from the -200 mesh samples continued to decline until stabilization at approximately week 125. Bulk tailings with higher sulfur contents have shown the similar trends to the two coarser particle sizes whereas bulk tailings with lower sulfur contents were closer to the -200 mesh samples with all bulk tailings samples having values between the coarse and fine fractions.

The main distinctive feature of leachates has been the increase in leaching of nickel (Figure 2) cobalt and manganese from the two coarse tailings fractions. This positive trend was apparent in the RS46 Draft 01 reporting period and was confirmed by the RS82 report. Since the RS82 report leachates have shown a decrease in these metals following their peak leaching rates at approximately week 100. However, values remain elevated relative to the fine size fraction and bulk samples. In addition, an increase in leaching of these metals in bulk samples was observed at approximately week 140 followed by a general decline but at an order of magnitude below the comparable coarse fraction samples. Copper leaching rates are stable at low values (<0.002 mg/kg/week).





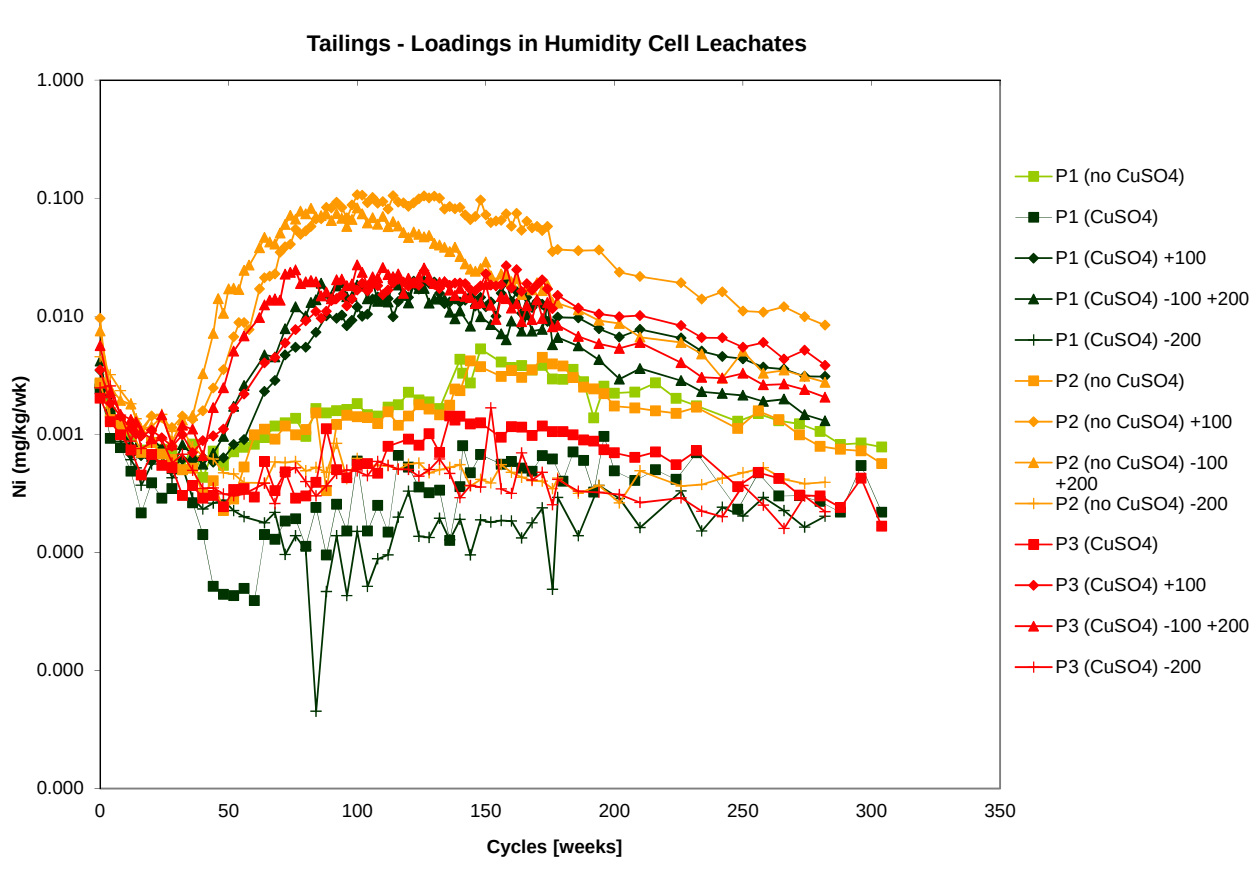


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Figure 2: Leachate pH, Sulfate and Nickel Loadings for 2005 & 2006 NorthMet Project Tailings

2.3.2 Tests Started in 2008

All humidity cell tests showed a slight initial decline in pH (Figure 3). The typical range in leachate pHs following the main initial decline was 7.25 to 7.78. pH does not appear to be related to fraction size as all pH measurements fluctuate in range of values. Leachate pHs remain somewhat erratic but no decline to lower pHs has been apparent, pH appears to be stabilizing with time.

Sulfate release initially declined sharply followed by a gradual decline (Figure 3). Major ions have followed sulfate but with calcium having a larger declining trend. Alkalinity has shown a consistent decreasing trend and is highest in the coarse particle size (+100) and lowest in the medium (+100-200) particle size.

For all fraction sizes, nickel showed an initial decline until week 80 at which time nickel release increased slightly and stabilized at levels slightly above the lowest release rates (Figure 3). The exception to this is the +100 mesh samples which showed an increase in nickel release at weeks 80 to 140 followed by a decrease to the lowest values of the test period. Copper release rates were more erratic but remained at low levels throughout the testing period in the range of 0.0002 mg/kg/week to 0.002 mg/kg/week.

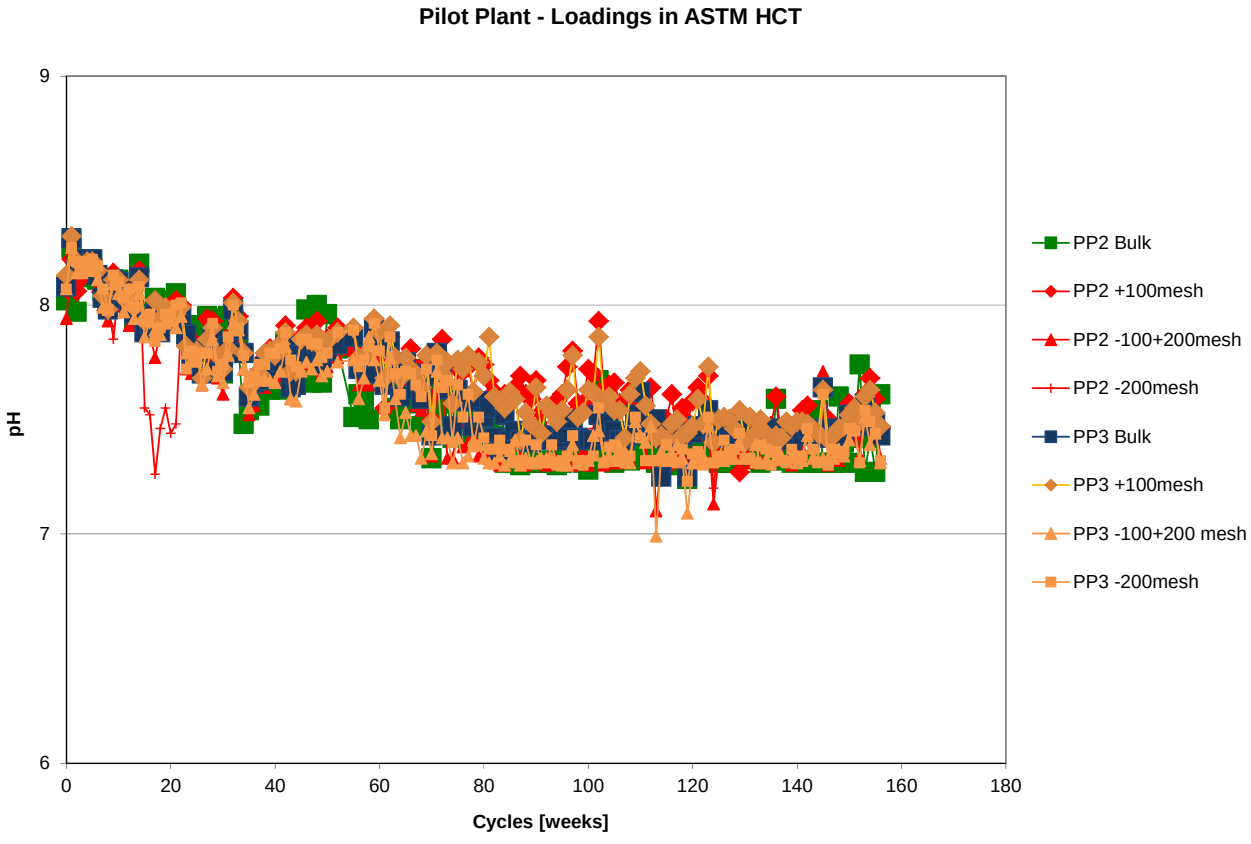
Results from these tests are distinctively different from the similar tests started in 2005 and 2006. Leachate pHs have not decreased to the same degree when comparing the same test period (first three years). As a result, accompanying order-of-magnitude increases in nickel concentrations have not been observed. Sulfate release for the 2008 tests also stabilized earlier.

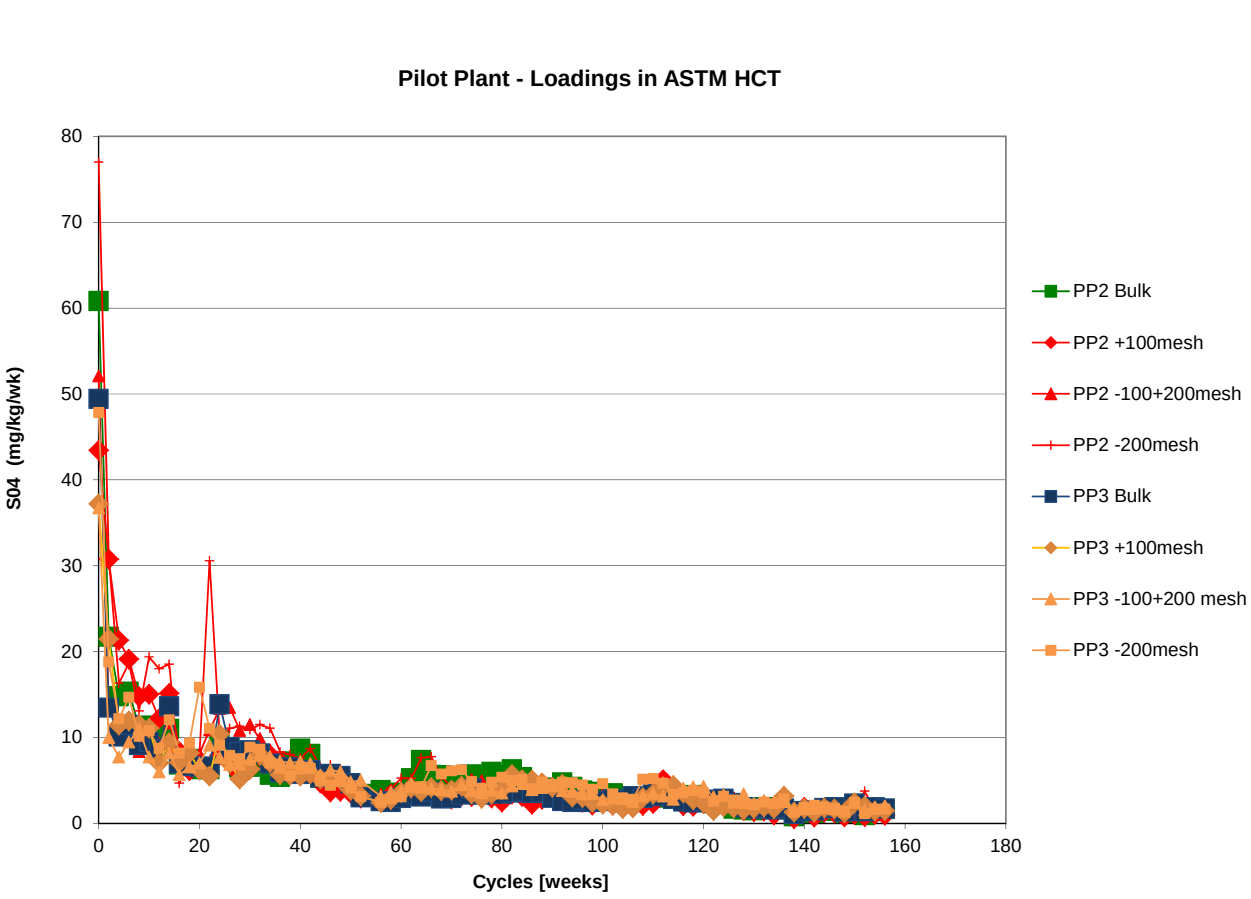
2.3.3 Tests Started in 2009

All humidity cell tests showed generally stable pH (Figure 4). However, a slight decline has occurred after 50 weeks of measurement in the -100 +200 mesh samples. The general trend is highest pHs are reported for the fine fraction and lowest for the -100 +200 mesh fraction with bulk tailings measurements between the coarse and fine fractions. The range in leachate pHs was 7.1 to 8.0.

Sulfate release initially declined sharply followed by a gradual decline (Figure 4). Major ions have followed sulfate but show a larger declining trend. Alkalinity showed an initial stable period from weeks 4 to 36 followed by a declining trend. This trend is most apparent in the coarse fraction.

The slight pH decrease at week 50 in the -200 +100 mesh tailings fractions was also accompanied by a slight rise in nickel (Figure4), cobalt and manganese release rates. All other size fractions have shown stable or decreasing release rates. Copper release rates have remained low for all cells throughout the test period in the range of 0.0001 mg/kg/week to 0.001 mg/kg/week.





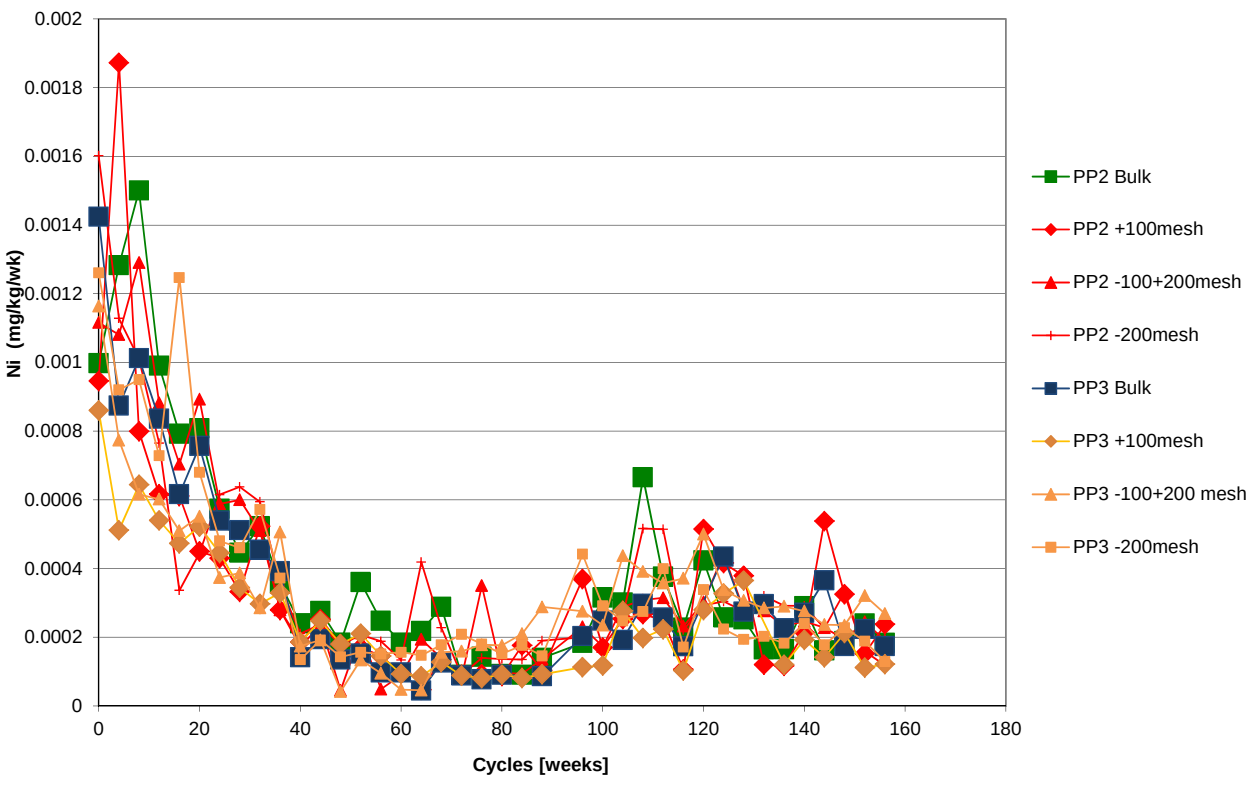


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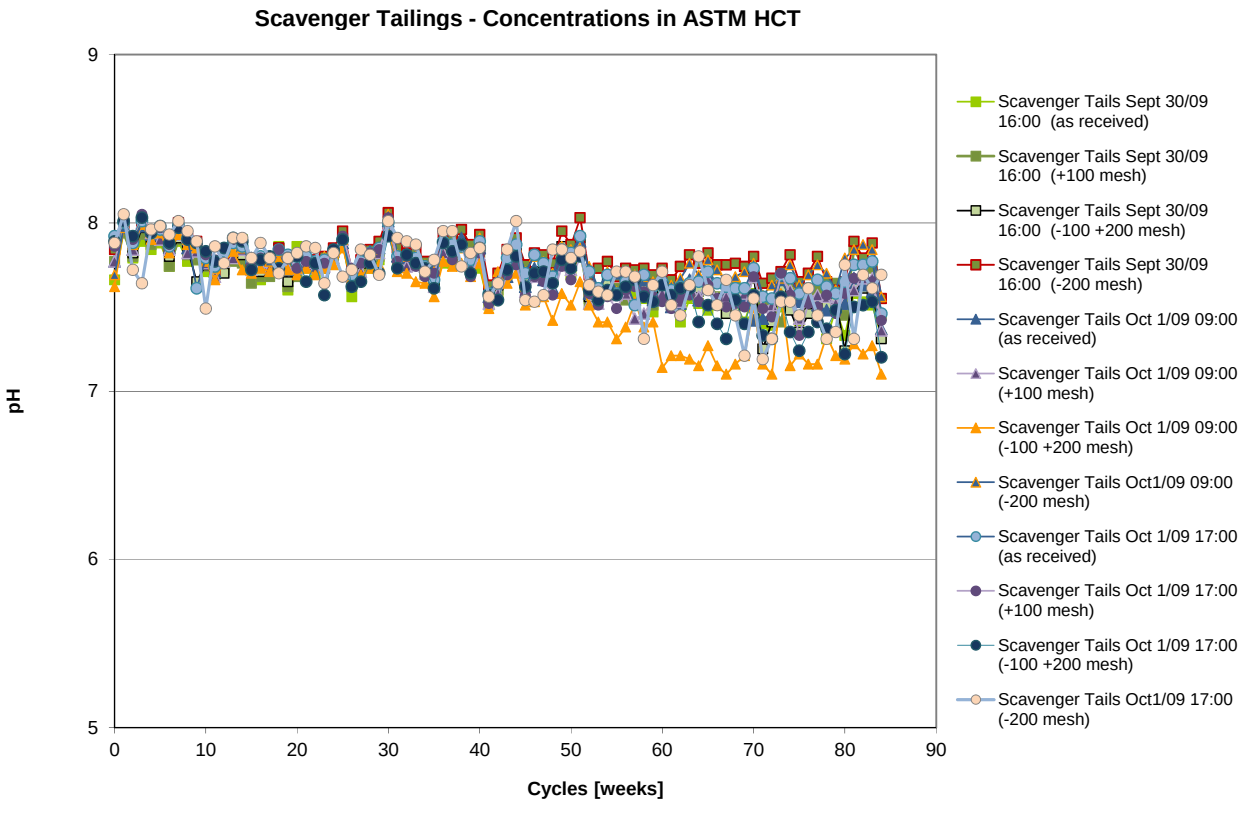
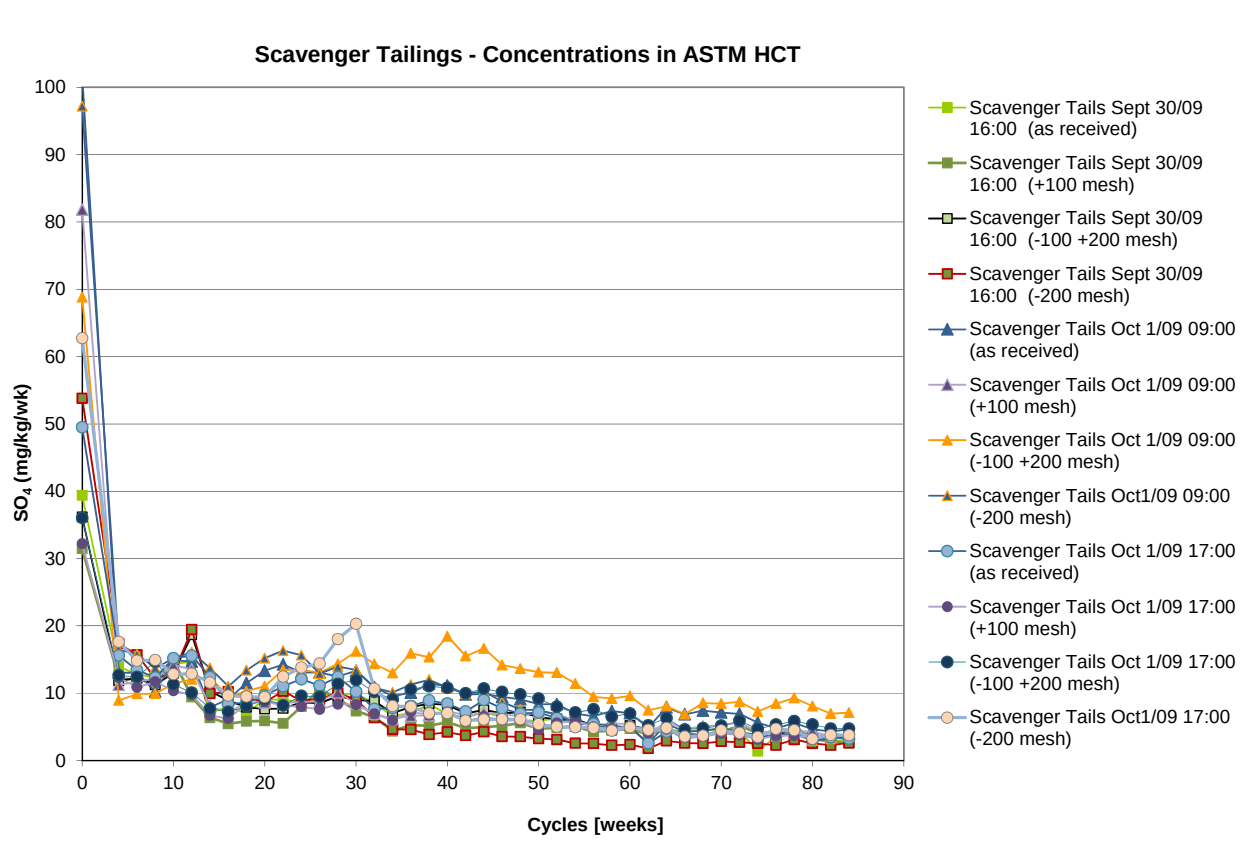
Figure 3: Leachate pH, Sulfate and Nickel Loadings for NorthMet 2008 Pilot Plant





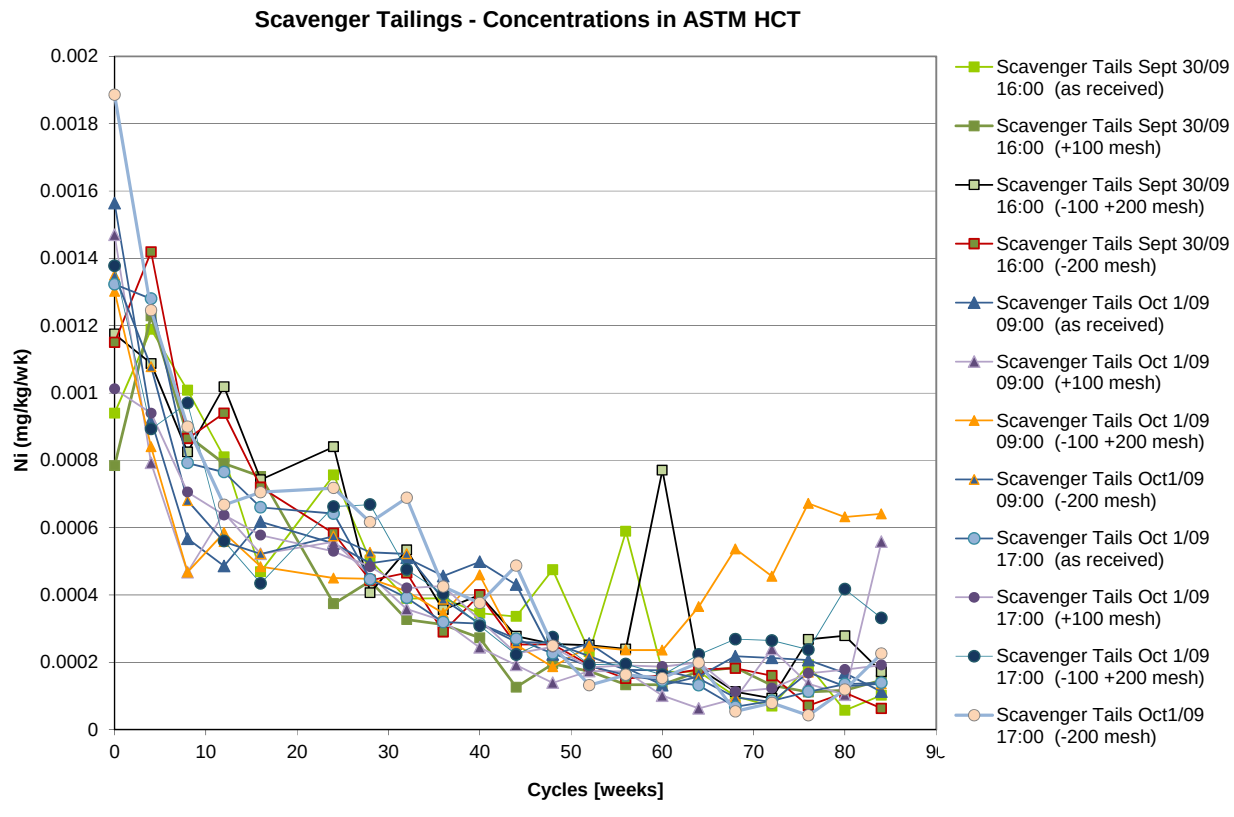


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Figure 4: Leachate pH, Sulfate and Nickel Loadings for NorthMet 2009 Scavenger Tailings

2.4 Discussion

The primary inputs to water quality modeling for the tailings are sulfide oxidation rates indicated by sulfate release from humidity cells. For the purpose of modeling, the sulfide oxidation reaction is assumed to be zero order, that is, the oxidation rate is independent of the amount of reactant (sulfide) remaining. This assumption was adopted to simplify the implementation of sulfide oxidation in the tailings water quality model and is conservative from the standpoint of estimating maximum expected concentrations of sulfate and other parameters. The current extended dataset provides an opportunity to evaluate the assumption and compare tailings produce by several metallurgical test programs.

Figure 5 shows sulfate release for all whole tailings samples being tested. Lines on the graph show the three different programs as different symbols and color-coding illustrates different initial sulfur concentrations. Sulfur concentrations exceeding 0.2% are shown as red; concentrations between 0.1 and 0.2% are shown as amber; and concentrations less than 0.1% are shown in green.

The overall distribution of rates shows that oxidation rates are correlated with sulfur content. Highest rates are associated with higher initial sulfur concentrations, and lowest rates are shown by samples with lowest sulfur concentrations.

Samples with sulfur concentrations exceeding 0.2% were only generated in the first pilot plant run when copper sulfate was not used at first to improve sulfide mineral flotation. Comparison between programs for the middle and lower sulfur ranges shows that there is little difference between results from the programs. In the middle sulfur range, oxidation rates for the initial pilot plant run samples were slightly higher than those in the same range produced in 2009, but the 2009 samples also had slightly lower initial sulfur concentrations. The same effect is seen in the lower sulfur range. The first pilot plant sample had the highest concentration of the four samples in this range and it appears yielded the highest sulfate release rates in that sulfur range. As a group, the testwork performed at various times shows similar relationships between sulfur content and sulfate release.

All tests showed that sulfate release declined over time. Mass balance indicates that the first and second sample set have been depleted of between 30% and 40% of the initial sulfur content indicating that the decline in sulfate release may be due to reduction in available oxidation rates. This implies the sulfide oxidation reaction is non-zero order and that the use of initial oxidation rates in modeling remains conservative.

Differences in pH trends and consequent release of nickel have been observed in the testwork. The initial dataset showed that pH decreased to a level at which nickel release could increase due to increased solubility of secondary nickel. This trend has not been observed to the same degree in subsequent testwork. The strong sensitivity of the test procedures to pH buffering by silicate mineral weathering has been observed previously and may be the reason for the observed differences. The 2008 tests suggest the tailings have greater buffering capacity than the 2005/2006 tests. Use of the first dataset to define sulfur concentrations which could result in significant metal leaching due to pH depression that could occur appears to be conservative.

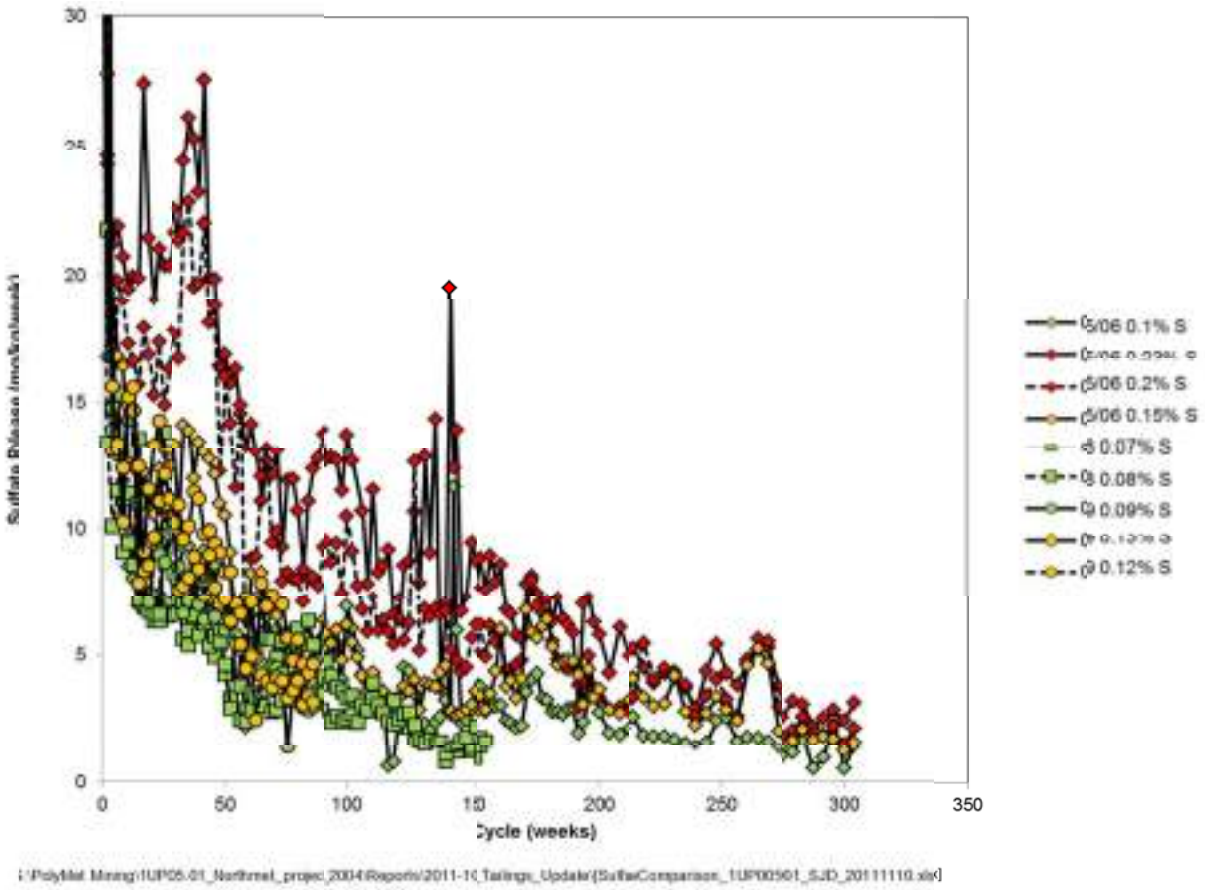

Figure 5: Comparison of Sulfate Leaching Rates as a Function of Sulfur Content

2.5 Conclusions

Review of ongoing and new humidity cell data shows that the assumptions used for oxidation rates in the water quality modelling are conservative when assuming the reaction order for sulfide oxidation, and that oxidation rates shown by tailings produced at different times are comparable.

Differences in pH trends are apparent between tailings produced by the different rounds of metallurgical testwork. However, the differences do not indicate a need to re-visit assumptions about sulfur management criteria (tailings to contain less than 0.2% sulfur). Ongoing testwork on tailings containing more than 0.2% sulfur for six years has shown stable pH as sulfur content has depleted to 40% of the starting composition.

3 LTVSMC Tailings Samples

3.1 Source of Samples

Twenty-nine samples of coarse LTVSMC tailings were obtained as part of a drilling program supervised by Barr Engineering to evaluate the geotechnical characteristics of the tailings. Ten drill holes were sampled in five different potential borrow areas (Figure 6).

Figure 6: LTVSMC Drilling Locations

3.2 Characteristics of Samples

Selected characteristics of the samples are shown in Table 2. Overall characteristics are similar to those for samples collected in an earlier drilling program and previously reported in RS46. Sulfur and trace element contents are very low. LTVSMC tailings samples are classified as non-acid generating due to the presence of carbonate minerals.

Results for the 29 samples were used to generate a sub-set of samples for humidity cell testing. In order to obtain sufficient sample for humidity cell testing (1 kg), composites were produced to represent coarse tailings in the four main borrow areas:

- 2E North Embankment
- 1E and 2E Separator
- 1E South Beach
- 2W North Embankment

Characteristics of the resulting composites are shown in Table 3. The sample from 2E North Embankment was tested in duplicate.

Table 2: Characteristics of LTVSMC Tailings

Statistic	As mg/kg	Co mg/kg	Cu mg/kg	Ni mg/kg	Carbonate %, C	Total S %, S
P5	14.88	5.94	5.28	2.78	0.638	<0.01
P50	25.5	7.9	7.3	4.1	1.45	0.03
P95	55.96	10.62	21.44	7.14	2.66	0.08

Source: \\Van-svr0.van.na.srk.ad\ge_projects\PolyMet Mining\1UP005.01_Northmet_project_2004\Testwork\2011-04_LTV_Static_Compilation\[LTV_Static_Compilation_1UP005001_SJD_20110428.xlsx]

Table 3: Characteristics of LTVSMC Composite Tailings Samples Tested in Humidity Cells

HCT ID	As mg/kg	Co mg/kg	Cu mg/kg	Ni mg/kg	Total S %
2E North Embankment Composite	28.7	8	7.8	4.1	0.03
2E North Embankment Composite (Duplicate)	22.7	6.9	7.7	3.8	0.03
1E and 2E Separator Composite	31.9	9.8	16.1	5.1	0.04
1E South Beach Composite	21.1	8.4	20.1	7	0.01
2W North Embankment Composite	46.9	8.1	8.5	4.6	0.06

Source: \\Van-svr0.van.na.srk.ad\ge_projects\PolyMet Mining\1UP005.01_Northmet_project_2004\Reports\2011-10_Tailings_Update\Re-Issue\[LTVSMC_Tailings_Data_1UP005.01_ld_rev00.xlsx

3.3 Humidity Cell Procedure

Shake flask testing of the samples prior to humidity cell testing indicated that the tailings were slightly oxidized due to weathering under field conditions. Since the purpose of humidity cells testing is to understand sulfide mineral oxidation rates, the samples were initially flushed with de-ionized water to remove soluble oxidation products. Following this procedure, the flushed samples were tested using the same procedures as used for NorthMet Project tailings.

3.4 Results

Selected humidity cell results from about 65 weeks of testing are shown in Figures 7 and 8.

Leachate pH values from all tests have been consistently between 7.3 and 8.1 and result mainly from leaching carbonate minerals. One of the 2E North Embankment samples showed consistently lower pH than the other tests. Likewise alkalinity, Ca and Mg were not comparable for the duplicate pair. All other parameters showed good reproducibility for the pair implying that the differences are due to heterogeneity in the carbonate content which did not affect elements associated with sulfide minerals.

Initial differences in sulfate release were observed. Higher release rates were observed for the two samples containing higher sulfur concentrations (2W North Embankment, and 1E and 2E Separator) but after about 25 weeks, differences disappeared and all samples yielded comparable low sulfate release. Leaching of trace elements occurred at very low rates. Arsenic, cobalt, manganese, molybdenum, selenium showed an initial decline like sulfate but most elements, including copper, nickel and zinc indicated no trend.

3.5 Conclusions

LTVSMC tailings leachate chemistry is dominated mainly by the buffering effects from dissolution of carbonate minerals. Initial differences in sulfate and some trace element release may be linked to differences in sulfide content but after about 25 weeks similar weathering behavior was shown regardless of sulfur content. Due to low sulfur content and expected long term effects of carbonate buffering, the tests have proceeded sufficiently to provide rates for input into water quality modeling. Release rates derived from these tests will be the average observed release over the 65-week testing period, which is conservative with respect to sulfate release.

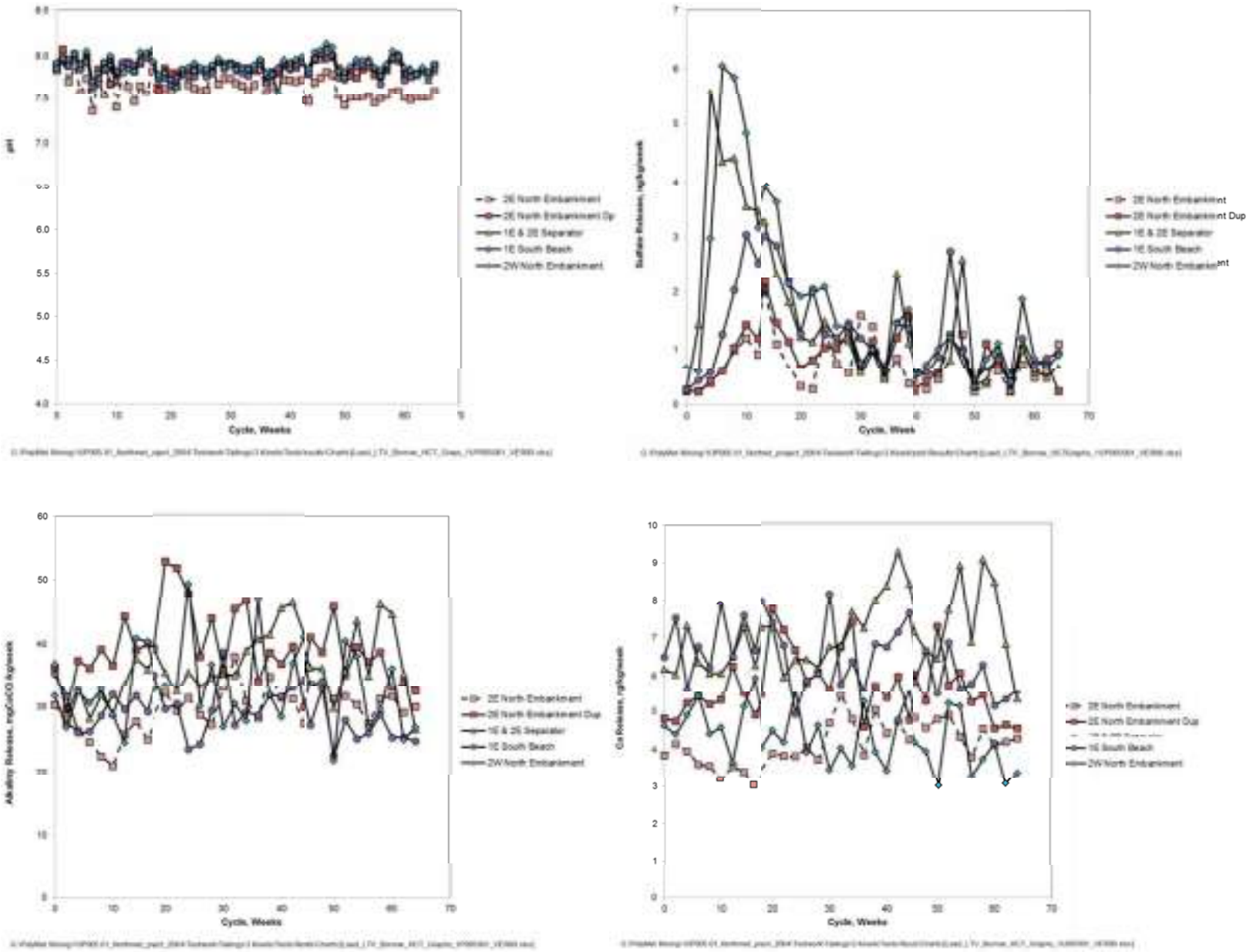

Figure 7: Major Ion Results for LTVSMC Humidity Cells

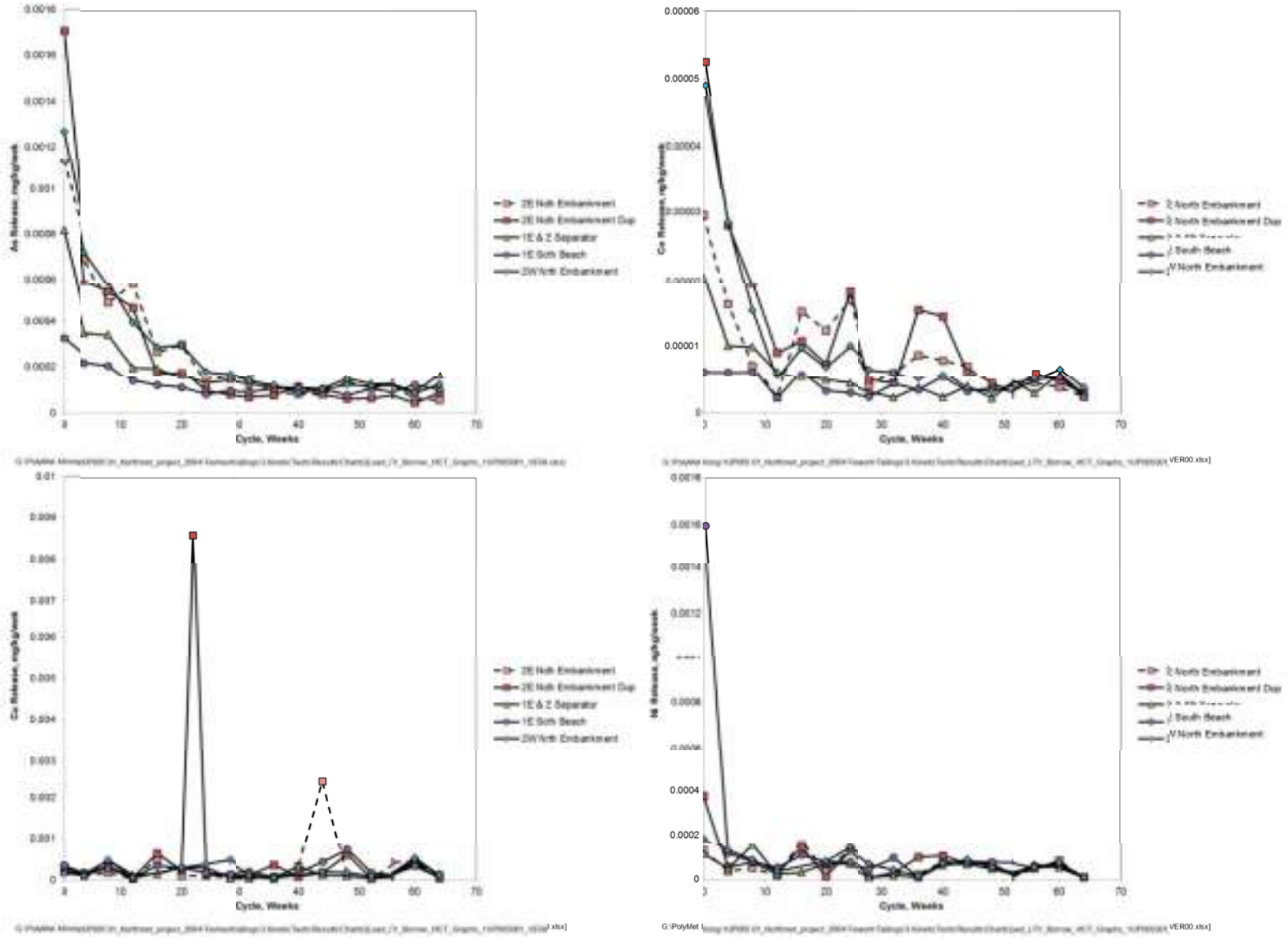

Figure 8: Trace Element Results for LTVSMC Humidity Cells

SRK Consulting (Canada) Inc.

Laura Donkervoort, M.Sc.
Staff Consultant (Geochemistry)

Stephen Day, M.Sc., P.Geo.
Corporate Consultant (Geochemistry)

Attachment G

Laboratory Report for Barr Engineering Company (Tailings Samples)

Laboratory Report for

Barr Engineering Company

Tailings Samples

October 22, 2010

Daniel B. Stephens & Associates, Inc.

5840 Osuna Road NE • Albuquerque, New Mexico 87109

October 22, 2010

Bethany S. Erfourth, PE
Barr Engineering Company
4700 W. 77th St.
Edina, MN 55435
(952) 832-2730

Re: DBS&A Laboratory Report for Barr Engineering Company Tailings Samples

Dear Mrs. Erfourth:

Enclosed is the final report for the Barr Engineering Company Tailings samples. Please review this report and provide any comments as samples will be held for a maximum of 30 days. After 30 days samples will be returned or disposed of in an appropriate manner.

All testing results were evaluated subjectively for consistency and reasonableness, and the results appear to be reasonably representative of the material tested. However, DBS&A does not assume any responsibility for interpretations or analyses based on the data enclosed, nor can we guarantee that these data are fully representative of the undisturbed materials at the field site. We recommend that careful evaluation of these laboratory results be made for your particular application.

The testing utilized to generate the enclosed final report employs methods that are standard for the industry. The results do not constitute a professional opinion by DBS&A, nor can the results affect any professional or expert opinions rendered with respect thereto by DBS&A. You have acknowledged that all the testing undertaken by us, and the final report provided, constitutes mere test results using standardized methods, and cannot be used to disqualify DBS&A from rendering any professional or expert opinion, having waived any claim of conflict of interest by DBS&A.

We are pleased to provide this service to Barr Engineering Company and look forward to future laboratory testing on other projects. If you have any questions about the enclosed data, please do not hesitate to call.

Sincerely,

DANIEL B. STEPHENS & ASSOCIATES, INC.
SOIL TESTING & RESEARCH LABORATORY

Joleen Hines
Laboratory Supervising Manager
Enclosure

Daniel B. Stephens & Associates, Inc.
Soil Testing & Research Laboratory
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Summaries

Summary of Tests Performed

Laboratory Sample Number	Initial Soil Properties[1]			Saturated Hydraulic Conductivity[2]			Moisture Characteristics[3]								Particle Size[4]			Specific Gravity[5]		Air Perm-eability	Atterberg Limits	Proctor Compaction
	G	VM	VD	CH	FH	FW	HC	PP	FP	DPP	RH	EP	WHC	K_{unsat}	DS	WS	H	F	C			
-200 fraction #1																		X				
-200 fraction #2																		X				
+200 fraction #1																		X				
+200 fraction #2																		X				
As Received																X	X	X				
Blend 1																X	X					
Blend 1 (38%)	X	X		X			X	X		X	X			X								
Blend 1 (45%)	X	X		X			X	X		X	X			X								
Blend 1 (55%)	X	X		X			X	X		X	X			X								
Blend 2																X	X					
Blend 2 (39%)	X	X		X			X	X		X	X			X								
Blend 2 (45%)	X	X		X			X	X		X	X			X								
Blend 2 (55%)	X	X		X			X	X		X	X			X								

[1] G = Gravimetric Moisture Content, VM = Volume Measurement Method, VD = Volume Displacement Method

[2] CH = Constant Head Rigid Wall, FH = Falling Head Rigid Wall, FW = Falling Head Rising Tail Flexible Wall

[3] HC = Hanging Column, PP = Pressure Plate, FP = Filter Paper, DPP = Dew Point Potentiometer, RH = Relative Humidity Box, EP = Effective Porosity, WHC = Water Holding Capacity, Kunsat = Calculated Unsaturated Hydraulic Conductivity

[4] DS = Dry Sieve, WS = Wet Sieve, H = Hydrometer

[5] F = Fine (<4.75mm), C = Coarse (>4.75mm)

Summary of Tests Performed (Continued)

Laboratory Sample Number	Initial Soil Properties[1]			Saturated Hydraulic Conductivity[2]			Moisture Characteristics[3]								Particle Size[4]			Specific Gravity[5]		Air Perm-eability	Atterberg Limits	Proctor Compaction
	G	VM	VD	CH	FH	FW	HC	PP	FP	DPP	RH	EP	WHC	K_{unsat}	DS	WS	H	F	C			
Blend 3																X	X					
Blend 3 (40%)	X	X		X			X	X		X	X			X								
Blend 3 (45%)	X	X		X			X	X		X	X			X								
Blend 3 (55%)	X	X		X			X	X		X	X			X								
Blend 4																X	X					
Blend 4 (44%)	X	X		X			X	X		X	X			X								
Blend 4 (50%)	X	X		X			X	X		X	X			X								
Blend 4 (55%)	X	X		X			X	X		X	X			X								

[1] G = Gravimetric Moisture Content, VM = Volume Measurement Method, VD = Volume Displacement Method

[2] CH = Constant Head Rigid Wall, FH = Falling Head Rigid Wall, FW = Falling Head Rising Tail Flexible Wall

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[4] DS = Dry Sieve, WS = Wet Sieve, H = Hydrometer

[5] F = Fine (<4.75mm), C = Coarse (>4.75mm)

Notes

One sample was received in four buckets. The four 1/2-3/4 full buckets were composited into one sample. The composited material was split on a #200 (0.075 mm) sieve by soaking the material in hot water overnight, followed by washing the material over the #200 sieve, again using hot water. All +200 (>0.075 mm) material and -200 (<0.075 mm) material was retained and dried.

Four separate blend samples were created using the dry +200 and -200 fractions by targeting the client provided percentages provided in the table below. All percentages are mass based.

Particle size analyses were performed on each blend to verify the actual percent passing the #200 sieve used to create each blend. The actual percent passing the #200 sieve used to create each blend was higher than the target value. This was due to material passing the #200 sieve remaining in the material retained on the #200 sieve; i.e. the initial washing and splitting process did not completely remove all material passing the #200 sieve from the material retained on the #200 sieve.

	% Passing #200	
	Target	Actual
Blend 1	75%	77%
Blend 2	50%	56%
Blend 3	30%	37%
Blend 4	10%	21%

The specific gravity (particle density) of the +200 and -200 fractions was measured in duplicate. The average of each the duplicate measurements was applied to each of the fractions. The particle density of each blend was then calculated based on the weighted average of the two fractions used for each blend.

The total porosity target remold values for each blend were initially 30%, 40%, and 50%. After determining that 30% total porosity was not achievable for Blend 4, the target total porosity values were revised to ‘as tight as possible’, 45% porosity, and 55% porosity for samples Blend 1, Blend 2, and Blend 3; and target remold values for Blend 4 were revised to ‘as tight as possible’, 50% porosity, and 55% porosity. The remolded samples were subjected to saturated hydraulic conductivity and moisture retention testing.

Summary of Sample Preparation/Volume Changes-Density Based

Sample Number	Target Remold Parameters[1] Moisture Content (%, g/g)	Target Remold Parameters[1] Dry Bulk Density (g/cm^3)	Actual Remold Data Moisture Content (%, g/g)	Actual Remold Data Dry Bulk Density (g/cm^3)	Actual Remold Data % of Target Density (%)	Volume Change Post Saturation[2] Dry Bulk Density (g/cm^3)	Volume Change Post Saturation[2] % Volume Change (%)	Volume Change Post Saturation[2] % of Target Density (%)	Volume Change Post Drying Curve[3] Dry Bulk Density (g/cm^3)	Volume Change Post Drying Curve[3] % Volume Change (%)	Volume Change Post Drying Curve[3] % of Target Density (%)
Blend 1 (38%)	8	1.85	8.3	1.85	100.0%	1.85	---	100%	1.85	---	100%
Blend 1 (45%)	8	1.64	8.3	1.64	99.6%	1.64	---	100%	1.70	(-) 3.6%	103%
Blend 1 (55%)	8	1.35	9.8	1.32	98.4%	1.60	(-) 17.0%	119%	1.86	(-) 29.0%	139%
Blend 2 (39%)	8	1.80	8.9	1.80	100.0%	1.80	---	100%	1.80	---	100%
Blend 2 (45%)	8	1.64	8.2	1.63	99.6%	1.69	(-) 3.4%	103%	1.79	(-) 8.9%	109%
Blend 2 (55%)	8	1.34	9.9	1.32	98.4%	1.67	(-) 20.8%	124%	1.92	(-) 31.1%	143%
Blend 3 (40%)	8	1.80	8.4	1.80	100.0%	1.80	---	100%	1.80	---	100%
Blend 3 (45%)	8	1.64	8.3	1.63	99.7%	1.69	(-) 3.1%	103%	1.77	(-) 7.8%	108%
Blend 3 (55%)	8	1.34	10.5	1.31	97.9%	1.73	(-) 24.3%	129%	1.89	(-) 30.5%	141%
Blend 4 (44%)	8	1.64	8.8	1.64	100.4%	1.64	---	100%	1.64	---	100%
Blend 4 (50%)	8	1.49	8.4	1.48	99.6%	1.55	(-) 4.4%	104%	1.58	(-) 6.1%	106%
Blend 4 (55%)	8	1.34	8.1	1.34	99.9%	1.54	(-) 13.0%	115%	1.60	(-) 16.2%	119%

[1]Target Remold Parameters: Provided by the client. See Notes page.

[2]Volume Change Post Saturation: Volume change measurements were obtained after saturated hydraulic conductivity testing.

[3]Volume Change Post Drying Curve: Volume change measurements were obtained throughout hanging column and pressure plate testing. The 'Volume Change Post Drying Curve' values represent the final sample dimensions after the last pressure plate point.

Notes:

(+) indicates sample swelling, (-) indicates sample settling, and "---" indicates no volume change occurred.

Summary of Sample Preparation/Volume Changes-Porosity Based

Sample Number	Target Remold Parameters[1]		Actual Remold Data			Volume Change Post Saturation[2]			Volume Change Post Drying Curve[3]		
	Moisture Content (%, g/g)	Calc. Porosity (%)	Moisture Content (%, g/g)	Calc. Porosity (%)	% of Target Porosity (%)	Calc. Porosity (%)	% Volume Change (%)	% of Target Porosity (%)	Calc. Porosity (%)	% Volume Change (%)	% of Target Porosity (%)
Blend 1 (38%)	8	38.1	8.3	38.1	100.0%	38.1	---	100.0%	38.1	---	100.0%
Blend 1 (45%)	8	45.0	8.3	45.3	100.6%	45.3	---	100.6%	43.2	(-) 3.6%	96.0%
Blend 1 (55%)	8	55.0	9.8	55.7	101.3%	46.6	(-) 17.0%	84.7%	37.6	(-) 29.0%	68.4%
Blend 2 (39%)	8	39.6	8.9	39.6	100.0%	39.6	---	100.0%	39.6	---	100.0%
Blend 2 (45%)	8	45.0	8.2	45.2	100.5%	43.3	(-) 3.4%	96.3%	39.9	(-) 8.9%	88.6%
Blend 2 (55%)	8	55.0	9.9	55.8	101.4%	44.2	(-) 20.8%	80.3%	35.8	(-) 31.1%	65.1%
Blend 3 (40%)	8	39.7	8.4	39.7	100.0%	39.7	---	100.0%	39.7	---	100.0%
Blend 3 (45%)	8	45.0	8.3	45.2	100.4%	43.4	(-) 3.1%	96.5%	40.6	(-) 7.8%	90.2%
Blend 3 (55%)	8	55.0	10.5	56.0	101.8%	41.9	(-) 24.3%	76.1%	36.7	(-) 30.5%	66.7%
Blend 4 (44%)	8	44.8	8.8	44.8	100.0%	44.8	---	100.0%	44.8	---	100.0%
Blend 4 (50%)	8	50.0	8.4	50.3	100.5%	47.9	(-) 4.4%	95.9%	47.0	(-) 6.1%	94.0%
Blend 4 (55%)	8	55.0	8.1	55.1	100.1%	48.4	(-) 13.0%	88.0%	46.4	(-) 16.2%	84.3%

[1]Target Remold Parameters: Provided by the client. See Notes page.

[2]Volume Change Post Saturation: Volume change measurements were obtained after saturated hydraulic conductivity testing.

[3]Volume Change Post Drying Curve: Volume change measurements were obtained throughout hanging column and pressure plate testing. The 'Volume Change Post Drying Curve' values represent the final sample dimensions after the last pressure plate point.

Notes:

(+) indicates sample swelling, (-) indicates sample settling, and "---" indicates no volume change occurred.

Summary of Initial Moisture Content, Dry Bulk Density Wet Bulk Density and Calculated Porosity

Sample Number	Moisture Content				Dry Bulk Density (g/cm^3)	Wet Bulk Density (g/cm^3)	Calculated Porosity (%)
	As Received		Remolded				
	Gravimetric (%, g/g)	Volumetric (%, cm^3/cm^3)	Gravimetric (%, g/g)	Volumetric (%, cm^3/cm^3)			
Blend 1 (38%)	NA	NA	8.3	15.3	1.85	2.00	38.1
Blend 1 (45%)	NA	NA	8.3	13.6	1.64	1.77	45.3
Blend 1 (55%)	NA	NA	9.8	13.0	1.32	1.45	55.7
Blend 2 (39%)	NA	NA	8.9	16.1	1.80	1.96	39.6
Blend 2 (45%)	NA	NA	8.2	13.4	1.63	1.77	45.2
Blend 2 (55%)	NA	NA	9.9	13.0	1.32	1.45	55.8
Blend 3 (40%)	NA	NA	8.4	15.1	1.80	1.95	39.7
Blend 3 (45%)	NA	NA	8.3	13.5	1.63	1.77	45.2
Blend 3 (55%)	NA	NA	10.5	13.8	1.31	1.45	56.0
Blend 4 (44%)	NA	NA	8.8	14.4	1.64	1.79	44.8
Blend 4 (50%)	NA	NA	8.4	12.5	1.48	1.60	50.3
Blend 4 (55%)	NA	NA	8.1	10.8	1.34	1.45	55.1

NA = Not analyzed

--- = This sample was not remolded

Summary of Saturated Hydraulic Conductivity Tests

Sample Number	K_{sat} (cm/sec)	Oversize Corrected K_{sat} (cm/sec)	Method of Analysis	
			Constant Head	Falling Head
Blend 1 (38%)	7.4E-05	NA	X	
Blend 1 (45%)	1.4E-04	NA	X	
Blend 1 (55%)	4.9E-04	NA	X	
Blend 2 (39%)	2.4E-04	NA	X	
Blend 2 (45%)	5.2E-04	NA	X	
Blend 2 (55%)	1.4E-03	NA	X	
Blend 3 (40%)	6.9E-04	NA	X	
Blend 3 (45%)	2.0E-03	NA	X	
Blend 3 (55%)	1.6E-03	NA	X	
Blend 4 (44%)	3.7E-03	NA	X	
Blend 4 (50%)	3.9E-03	NA	X	
Blend 4 (55%)	9.8E-03	NA	X	

--- = Oversize correction is unnecessary since coarse fraction < 5% of composite mass
NR = Not requested
NA = Not applicable

Summary of Moisture Characteristics of the Initial Drainage Curve

Sample Number	Pressure Head (-cm water)	Moisture Content (%, cm^3/cm^3)
Blend 1 (38%)	0	40.3
	17	39.9
	49	39.7
	123	39.7
	337	25.9
	14685	3.8
	38752	2.2
	124416	1.7
	861325	0.7
Blend 1 (45%)	0	45.0
	12	43.8
	31	42.8
	101	42.4 ‡‡
	337	25.9 ‡‡
	17439	3.9 ‡‡
	41200	2.5 ‡‡
	132268	1.3 ‡‡
	861325	0.7 ‡‡
Blend 1 (55%)	0	48.0 ‡‡
	12	40.8 ‡‡
	31	38.6 ‡‡
	101	37.4 ‡‡
	337	33.7 ‡‡
	17745	4.2 ‡‡
	44769	2.6 ‡‡
	109323	1.9 ‡‡
	861325	0.7 ‡‡

‡‡ Volume adjustments are applicable at this matric potential (see data sheet for this sample).

Summary of Moisture Characteristics of the Initial Drainage Curve (Continued)

Sample Number	Pressure Head (-cm water)	Moisture Content (%, cm^3/cm^3)
Blend 2 (39%)	0	39.9
	12	38.6
	31	38.2
	101	37.9
	337	16.9
	16419	2.5
	35489	1.8
	142772	1.3
	861325	0.6
Blend 2 (45%)	0	45.8 ‡‡
	12	43.5 ‡‡
	31	41.7 ‡‡
	101	40.2 ‡‡
	337	19.4 ‡‡
	14583	2.6 ‡‡
	35489	1.5 ‡‡
	126455	1.0 ‡‡
	861325	0.6 ‡‡
Blend 2 (55%)	0	46.6 ‡‡
	5	41.1 ‡‡
	21	39.4 ‡‡
	76	37.3 ‡‡
	337	21.6 ‡‡
	17541	2.9 ‡‡
	44565	1.7 ‡‡
	106875	1.4 ‡‡
	861325	0.7 ‡‡

‡‡ Volume adjustments are applicable at this matric potential (see data sheet for this sample).

Summary of Moisture Characteristics of the Initial Drainage Curve (Continued)

Sample Number	Pressure Head (-cm water)	Moisture Content (%, cm^3/cm^3)
Blend 3 (40%)	0	39.7
	12	39.1
	31	38.8
	101	27.0
	337	12.3
	16215	3.2
	36611	2.9
	138387	2.0
	861325	0.6
Blend 3 (45%)	0	44.9 ‡‡
	5	42.9 ‡‡
	21	41.5 ‡‡
	76	39.7 ‡‡
	337	12.1 ‡‡
	14481	2.0 ‡‡
	41098	1.3 ‡‡
	141752	0.9 ‡‡
	861325	0.6 ‡‡
Blend 3 (55%)	0	44.0 ‡‡
	5	38.7 ‡‡
	21	37.1 ‡‡
	76	36.7 ‡‡
	337	12.1 ‡‡
	12951	1.9 ‡‡
	34673	1.3 ‡‡
	123396	1.0 ‡‡
	861325	0.6 ‡‡

‡‡ Volume adjustments are applicable at this matric potential (see data sheet for this sample).

Summary of Moisture Characteristics of the Initial Drainage Curve (Continued)

Sample Number	Pressure Head (-cm water)	Moisture Content (%, cm^3/cm^3)
Blend 4 (44%)	0	43.1
	5	42.8
	21	42.5
	76	32.7
	337	5.3
	19274	1.2
	57619	0.9
	151440	0.7
	861325	0.5
Blend 4 (50%)	0	42.2 ‡‡
	6	42.1 ‡‡
	30	41.4 ‡‡
	71	26.3 ‡‡
	337	4.5 ‡‡
	16929	1.4 ‡‡
	46197	1.0 ‡‡
	265658	0.8 ‡‡
	861325	0.4 ‡‡
Blend 4 (55%)	0	41.6 ‡‡
	6	40.7 ‡‡
	30	39.8 ‡‡
	71	29.3 ‡‡
	337	4.7 ‡‡
	6323	2.0 ‡‡
	66695	0.6 ‡‡
	105345	0.7 ‡‡
	861325	0.5 ‡‡

‡‡ Volume adjustments are applicable at this matric potential (see data sheet for this sample).

Summary of Calculated Unsaturated Hydraulic Properties

Sample Number	α (cm^{-1})	N (dimensionless)	θ_r (% vol)	θ_s (% vol)	Oversize Corrected θ_r (% vol)	Oversize Corrected θ_s (% vol)
Blend 1 (38%)	0.0028	4.4060	2.10	39.97	NA	NA
Blend 1 (45%)	0.0037	2.3302	2.04	44.09	NA	NA
Blend 1 (55%)	0.0032	1.5352	0.00	42.21	NA	NA
Blend 2 (39%)	0.0039	3.4263	1.55	38.91	NA	NA
Blend 2 (45%)	0.0049	2.3810	1.36	43.86	NA	NA
Blend 2 (55%)	0.0097	1.5315	0.20	43.07	NA	NA
Blend 3 (40%)	0.0112	1.9686	2.08	39.94	NA	NA
Blend 3 (45%)	0.0065	2.6280	1.17	43.17	NA	NA
Blend 3 (55%)	0.0066	2.4829	1.18	40.08	NA	NA
Blend 4 (44%)	0.0106	2.7451	0.81	42.95	NA	NA
Blend 4 (50%)	0.0148	2.7336	1.07	42.67	NA	NA
Blend 4 (55%)	0.0127	2.6317	0.95	41.31	NA	NA

--- = Oversize correction is unnecessary since coarse fraction < 5% of composite mass

NR = Not requested

NA = Not applicable

Summary of Particle Size Characteristics

Sample Number	d_{10} (mm)	d_{50} (mm)	d_{60} (mm)	C_u	C_c	Method	ASTM Classification	USDA Classification
As Received	0.010	0.076	0.096	9.6	2.1	WS/H	Silty sand (SM)	Sandy Loam
Blend 1	0.0037	0.037	0.048	13	1.6	WS/H	Silt with sand (ML)s	Silt Loam
Blend 2	0.0056	0.062	0.082	15	2.2	WS/H	Sandy silt s(ML)	Sandy Loam
Blend 3	0.013	0.092	0.11	8.5	2.4	WS/H	Silty sand (SM)	Loamy Sand
Blend 4	0.051	0.11	0.14	2.7	1.0	WS/H	Silty sand (SM)	Sand

d_{50} = Median particle diameter

Est = Reported values for d_{10}, C_u, C_c, and soil classification are estimates, since extrapolation was required to obtain the d_{10} diameter

$$C_u = \frac{d_{60}}{d_{10}}$$

$$C_c = \frac{(d_{30})^2}{(d_{10})(d_{60})}$$

DS = Dry sieve

H = Hydrometer

WS = Wet sieve

† Greater than 10% of sample is coarse material

Percent Gravel, Sand, Silt and Clay*

Sample Number	% Gravel (>4.75mm)	% Sand (<4.75mm, >0.075mm)	% Silt (<0.075mm, >0.002mm)	% Clay (<0.002mm)
As Received	0.0	50.3	43.8	5.9
Blend 1	0.0	22.9	70.0	7.1
Blend 2	0.0	43.5	51.1	5.4
Blend 3	0.0	62.6	34.0	3.4
Blend 4	0.0	78.6	19.3	2.1

*USCS classification does not classify clay fraction based on particle size. USDA definition of clay (<0.002mm) used in this table.

Summary of Specific Gravity Tests

	<4.75mm Material		>4.75mm Material		Bulk Sample
Sample Number	Specific Gravity	Percent of Bulk Sample	Specific Gravity	Percent of Bulk Sample	Specific Gravity
-200 fraction #1	3.01	100.0	---	0.0	3.01
-200 fraction #2	2.99	100.0	---	0.0	2.99
-200 Average	3.00	100.0	---	0.0	3.00
+200 fraction #1	2.97	100.0	---	0.0	2.97
+200 fraction #2	2.98	100.0	---	0.0	2.98
+200 Average	2.98	100.0	---	0.0	2.98
As Received	2.95	100.0	---	0.0	2.95

--- = Unnecessary since specified fraction <5% of composite mass

* = Based on specific gravity of material < 4.75 mm

Laboratory Data and Graphical Plots

Initial Properties

Summary of Initial Moisture Content, Dry Bulk Density Wet Bulk Density and Calculated Porosity

Sample Number	Moisture Content				Dry Bulk Density (g/cm^3)	Wet Bulk Density (g/cm^3)	Calculated Porosity (%)
	As Received		Remolded				
	Gravimetric (%, g/g)	Volumetric (%, cm^3/cm^3)	Gravimetric (%, g/g)	Volumetric (%, cm^3/cm^3)			
Blend 1 (38%)	NA	NA	8.3	15.3	1.85	2.00	38.1
Blend 1 (45%)	NA	NA	8.3	13.6	1.64	1.77	45.3
Blend 1 (55%)	NA	NA	9.8	13.0	1.32	1.45	55.7
Blend 2 (39%)	NA	NA	8.9	16.1	1.80	1.96	39.6
Blend 2 (45%)	NA	NA	8.2	13.4	1.63	1.77	45.2
Blend 2 (55%)	NA	NA	9.9	13.0	1.32	1.45	55.8
Blend 3 (40%)	NA	NA	8.4	15.1	1.80	1.95	39.7
Blend 3 (45%)	NA	NA	8.3	13.5	1.63	1.77	45.2
Blend 3 (55%)	NA	NA	10.5	13.8	1.31	1.45	56.0
Blend 4 (44%)	NA	NA	8.8	14.4	1.64	1.79	44.8
Blend 4 (50%)	NA	NA	8.4	12.5	1.48	1.60	50.3
Blend 4 (55%)	NA	NA	8.1	10.8	1.34	1.45	55.1

NA = Not analyzed

--- = This sample was not remolded

Data for Initial Moisture Content, Bulk Density, Porosity, and Percent Saturation

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 1 (38%)
Ring Number: NA
Depth: NA

	As Received	Remolded
Test Date:	NA	24-Aug-10
Field weight of sample* (g)*:*		375.07
Tare weight, ring (g)*:*		88.69
Tare weight, pan/plate (g)*:*		0.00
Tare weight, other (g)*:*		0.00
Dry weight of sample (g)*:*		264.49
Sample volume (cm^3)*:*		142.91
Measured particle density (g/cm^3)*:*		2.99
Gravimetric Moisture Content (% g/g)*:*		8.3
Volumetric Moisture Content (% vol)*:*		15.3
Dry bulk density (g/cm^3)*:*		1.85
Wet bulk density (g/cm^3)*:*		2.00
Calculated Porosity (% vol)*:*		38.1
Percent Saturation:		40.2

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Comments:

* Weight including tares
NA = Not analyzed
--- = This sample was not remolded

Data for Initial Moisture Content, Bulk Density, Porosity, and Percent Saturation

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 1 (45%)
Ring Number: NA
Depth: NA

	As Received	Remolded
Test Date:	NA	25-Aug-10
Field weight of sample* (g)*:*		341.79
Tare weight, ring (g)*:*		87.09
Tare weight, pan/plate (g)*:*		0.00
Tare weight, other (g)*:*		0.00
Dry weight of sample (g)*:*		235.17
Sample volume (cm^3)*:*		143.74
Measured particle density (g/cm^3)*:*		2.99
Gravimetric Moisture Content (% g/g)*:*		8.3
Volumetric Moisture Content (% vol)*:*		13.6
Dry bulk density (g/cm^3)*:*		1.64
Wet bulk density (g/cm^3)*:*		1.77
Calculated Porosity (% vol)*:*		45.3
Percent Saturation:		30.0

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Comments:

* Weight including tares
NA = Not analyzed
--- = This sample was not remolded

Data for Initial Moisture Content, Bulk Density, Porosity, and Percent Saturation

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 1 (55%)
Ring Number: NA
Depth: NA

	As Received	Remolded
Test Date:	NA	25-Aug-10
Field weight of sample* (g)*:*		289.35
Tare weight, ring (g)*:*		85.21
Tare weight, pan/plate (g)*:*		0.00
Tare weight, other (g)*:*		0.00
Dry weight of sample (g)*:*		185.91
Sample volume (cm^3)*:*		140.40
Measured particle density (g/cm^3)*:*		2.99
Gravimetric Moisture Content (% g/g)*:*		9.8
Volumetric Moisture Content (% vol)*:*		13.0
Dry bulk density (g/cm^3)*:*		1.32
Wet bulk density (g/cm^3)*:*		1.45
Calculated Porosity (% vol)*:*		55.7
Percent Saturation:		23.3

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Comments:

* Weight including tares
NA = Not analyzed
--- = This sample was not remolded

Data for Initial Moisture Content, Bulk Density, Porosity, and Percent Saturation

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 2 (39%)
Ring Number: NA
Depth: NA

	As Received	Remolded
Test Date:	NA	25-Aug-10
Field weight of sample* (g)*:*		377.41
Tare weight, ring (g)*:*		89.02
Tare weight, pan/plate (g)*:*		0.00
Tare weight, other (g)*:*		0.00
Dry weight of sample (g)*:*		264.75
Sample volume (cm^3)*:*		147.03
Measured particle density (g/cm^3)*:*		2.98
Gravimetric Moisture Content (% g/g)*:*		8.9
Volumetric Moisture Content (% vol)*:*		16.1
Dry bulk density (g/cm^3)*:*		1.80
Wet bulk density (g/cm^3)*:*		1.96
Calculated Porosity (% vol)*:*		39.6
Percent Saturation:		40.6

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Comments:

* Weight including tares
NA = Not analyzed
--- = This sample was not remolded

Data for Initial Moisture Content, Bulk Density, Porosity, and Percent Saturation

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 2 (45%)
Ring Number: NA
Depth: NA

	As Received	Remolded
Test Date:	NA	25-Aug-10
Field weight of sample* (g)*:*		357.60
Tare weight, ring (g)*:*		110.15
Tare weight, pan/plate (g)*:*		0.00
Tare weight, other (g)*:*		0.00
Dry weight of sample (g)*:*		228.69
Sample volume (cm^3)*:*		139.97
Measured particle density (g/cm^3)*:*		2.98
Gravimetric Moisture Content (% g/g)*:*		8.2
Volumetric Moisture Content (% vol)*:*		13.4
Dry bulk density (g/cm^3)*:*		1.63
Wet bulk density (g/cm^3)*:*		1.77
Calculated Porosity (% vol)*:*		45.2
Percent Saturation:		29.6

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Comments:

* Weight including tares
NA = Not analyzed
--- = This sample was not remolded

Data for Initial Moisture Content, Bulk Density, Porosity, and Percent Saturation

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 2 (55%)
Ring Number: NA
Depth: NA

	As Received	Remolded
Test Date:	NA	25-Aug-10
Field weight of sample* (g)*:*		320.64
Tare weight, ring (g)*:*		116.30
Tare weight, pan/plate (g)*:*		0.00
Tare weight, other (g)*:*		0.00
Dry weight of sample (g)*:*		185.98
Sample volume (cm^3)*:*		140.88
Measured particle density (g/cm^3)*:*		2.98
Gravimetric Moisture Content (% g/g)*:*		9.9
Volumetric Moisture Content (% vol)*:*		13.0
Dry bulk density (g/cm^3)*:*		1.32
Wet bulk density (g/cm^3)*:*		1.45
Calculated Porosity (% vol)*:*		55.8
Percent Saturation:		23.4

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Comments:

* Weight including tares
NA = Not analyzed
--- = This sample was not remolded

Data for Initial Moisture Content, Bulk Density, Porosity, and Percent Saturation

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 3 (40%)
Ring Number: NA
Depth: NA

	As Received	Remolded
Test Date:	NA	25-Aug-10
Field weight of sample* (g)*:*		366.15
Tare weight, ring (g)*:*		87.14
Tare weight, pan/plate (g)*:*		0.00
Tare weight, other (g)*:*		0.00
Dry weight of sample (g)*:*		257.44
Sample volume (cm^3)*:*		143.29
Measured particle density (g/cm^3)*:*		2.98
Gravimetric Moisture Content (% g/g)*:*		8.4
Volumetric Moisture Content (% vol)*:*		15.1
Dry bulk density (g/cm^3)*:*		1.80
Wet bulk density (g/cm^3)*:*		1.95
Calculated Porosity (% vol)*:*		39.7
Percent Saturation:		37.9

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Comments:

* Weight including tares
NA = Not analyzed
--- = This sample was not remolded

Data for Initial Moisture Content, Bulk Density, Porosity, and Percent Saturation

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 3 (45%)
Ring Number: NA
Depth: NA

	As Received	Remolded
Test Date:	NA	25-Aug-10
Field weight of sample* (g)*:*		339.40
Tare weight, ring (g)*:*		87.13
Tare weight, pan/plate (g)*:*		0.00
Tare weight, other (g)*:*		0.00
Dry weight of sample (g)*:*		232.98
Sample volume (cm^3)*:*		142.72
Measured particle density (g/cm^3)*:*		2.98
Gravimetric Moisture Content (% g/g)*:*		8.3
Volumetric Moisture Content (% vol)*:*		13.5
Dry bulk density (g/cm^3)*:*		1.63
Wet bulk density (g/cm^3)*:*		1.77
Calculated Porosity (% vol)*:*		45.2
Percent Saturation:		29.9

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Comments:

* Weight including tares
NA = Not analyzed
--- = This sample was not remolded

Data for Initial Moisture Content, Bulk Density, Porosity, and Percent Saturation

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 3 (55%)
Ring Number: NA
Depth: NA

	As Received	Remolded
Test Date:	NA	25-Aug-10
Field weight of sample* (g)*:*		300.46
Tare weight, ring (g)*:*		88.41
Tare weight, pan/plate (g)*:*		0.00
Tare weight, other (g)*:*		0.00
Dry weight of sample (g)*:*		191.84
Sample volume (cm^3)*:*		146.31
Measured particle density (g/cm^3)*:*		2.98
Gravimetric Moisture Content (% g/g)*:*		10.5
Volumetric Moisture Content (% vol)*:*		13.8
Dry bulk density (g/cm^3)*:*		1.31
Wet bulk density (g/cm^3)*:*		1.45
Calculated Porosity (% vol)*:*		56.0
Percent Saturation:		24.7

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Comments:

* Weight including tares
NA = Not analyzed
--- = This sample was not remolded

Data for Initial Moisture Content, Bulk Density, Porosity, and Percent Saturation

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 4 (44%)
Ring Number: NA
Depth: NA

	As Received	Remolded
Test Date:	NA	24-Aug-10
Field weight of sample* (g)*:*		345.02
Tare weight, ring (g)*:*		87.15
Tare weight, pan/plate (g)*:*		0.00
Tare weight, other (g)*:*		0.00
Dry weight of sample (g)*:*		237.11
Sample volume (cm^3)*:*		144.43
Measured particle density (g/cm^3)*:*		2.98
Gravimetric Moisture Content (% g/g)*:*		8.8
Volumetric Moisture Content (% vol)*:*		14.4
Dry bulk density (g/cm^3)*:*		1.64
Wet bulk density (g/cm^3)*:*		1.79
Calculated Porosity (% vol)*:*		44.8
Percent Saturation:		32.1

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Comments:

* Weight including tares
NA = Not analyzed
--- = This sample was not remolded

Data for Initial Moisture Content, Bulk Density, Porosity, and Percent Saturation

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 4 (50%)
Ring Number: NA
Depth: NA

	As Received	Remolded
Test Date:	NA	10-Sep-10
Field weight of sample* (g)*:*		319.29
Tare weight, ring (g)*:*		88.63
Tare weight, pan/plate (g)*:*		0.00
Tare weight, other (g)*:*		0.00
Dry weight of sample (g)*:*		212.69
Sample volume (cm^3)*:*		143.72
Measured particle density (g/cm^3)*:*		2.98
Gravimetric Moisture Content (% g/g)*:*		8.4
Volumetric Moisture Content (% vol)*:*		12.5
Dry bulk density (g/cm^3)*:*		1.48
Wet bulk density (g/cm^3)*:*		1.60
Calculated Porosity (% vol)*:*		50.3
Percent Saturation:		24.9

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Comments:

* Weight including tares
NA = Not analyzed
--- = This sample was not remolded

Data for Initial Moisture Content, Bulk Density, Porosity, and Percent Saturation

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 4 (55%)
Ring Number: NA
Depth: NA

	As Received	Remolded
Test Date:	NA	10-Sep-10
Field weight of sample* (g)*:*		297.39
Tare weight, ring (g)*:*		88.93
Tare weight, pan/plate (g)*:*		0.00
Tare weight, other (g)*:*		0.00
Dry weight of sample (g)*:*		192.87
Sample volume (cm^3)*:*		144.26
Measured particle density (g/cm^3)*:*		2.98
Gravimetric Moisture Content (% g/g)*:*		8.1
Volumetric Moisture Content (% vol)*:*		10.8
Dry bulk density (g/cm^3)*:*		1.34
Wet bulk density (g/cm^3)*:*		1.45
Calculated Porosity (% vol)*:*		55.1
Percent Saturation:		19.6

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Comments:

* Weight including tares
NA = Not analyzed
--- = This sample was not remolded

Saturated Hydraulic Conductivity

Summary of Saturated Hydraulic Conductivity Tests

Sample Number	K_{sat} (cm/sec)	Oversize Corrected K_{sat} (cm/sec)	Method of Analysis	
			Constant Head	Falling Head
Blend 1 (38%)	7.4E-05	NA	X	
Blend 1 (45%)	1.4E-04	NA	X	
Blend 1 (55%)	4.9E-04	NA	X	
Blend 2 (39%)	2.4E-04	NA	X	
Blend 2 (45%)	5.2E-04	NA	X	
Blend 2 (55%)	1.4E-03	NA	X	
Blend 3 (40%)	6.9E-04	NA	X	
Blend 3 (45%)	2.0E-03	NA	X	
Blend 3 (55%)	1.6E-03	NA	X	
Blend 4 (44%)	3.7E-03	NA	X	
Blend 4 (50%)	3.9E-03	NA	X	
Blend 4 (55%)	9.8E-03	NA	X	

--- = Oversize correction is unnecessary since coarse fraction < 5% of composite mass
NR = Not requested
NA = Not applicable

Saturated Hydraulic Conductivity
Constant Head Method

Job name: Barr Engineering Company
Job number: LB10.0170.00
Sample number: Blend 1 (38%)
Ring Number: NA
Depth: NA

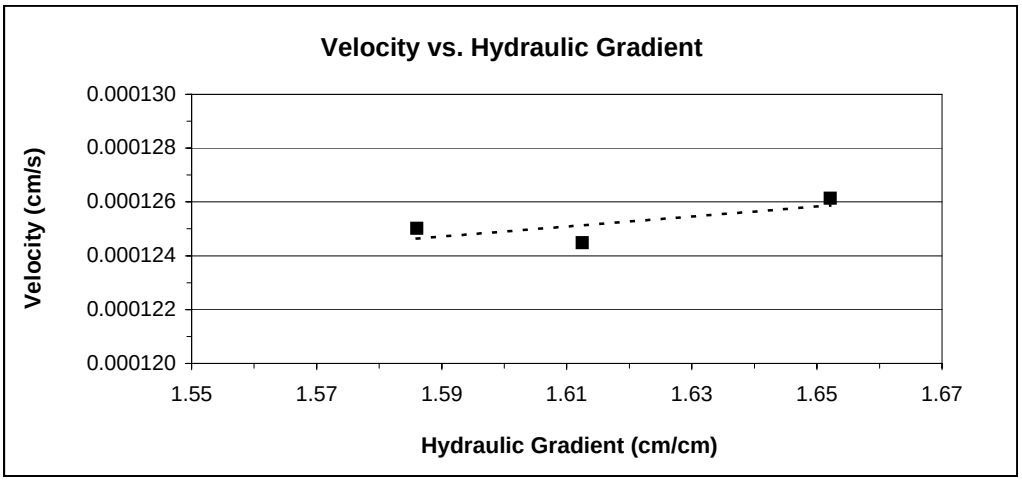
Type of water used: TAP
Collection vessel tare (g)*:* 10.95
Sample length (cm)*:* 7.57
Sample diameter (cm)*:* 4.90
Sample x-sectional area (cm^2)*:* 18.89

Date	Time	Temp (°C)	Head (cm)	Q + Tare (g)	Q (cm^3)	Elapsed time (sec)	Ksat (cm/sec)	Ksat @ 20°C (cm/sec)
Test # 1:								
27-Aug-10	9:09:40	22.0	12.5	12.42	1.5	617	7.6E-05	7.3E-05
27-Aug-10	9:19:57							
Test # 2:								
27-Aug-10	11:02:28	22.0	12.2	12.57	1.6	689	7.7E-05	7.4E-05
27-Aug-10	11:13:57							
Test # 3:								
27-Aug-10	11:44:57	22.0	12	11.93	1.0	415	7.9E-05	7.5E-05
27-Aug-10	11:51:52							

Average Ksat* (cm/sec)*: **7.4E-05**
Oversize Corrected Ksat* (cm/sec)*: **NA**

Comments:

--- = Oversize correction is unnecessary since coarse fraction < 5% of composite mass
NA = Not applicable



Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Saturated Hydraulic Conductivity
Constant Head Method

Job name: Barr Engineering Company
Job number: LB10.0170.00
Sample number: Blend 1 (45%)
Ring Number: NA
Depth: NA

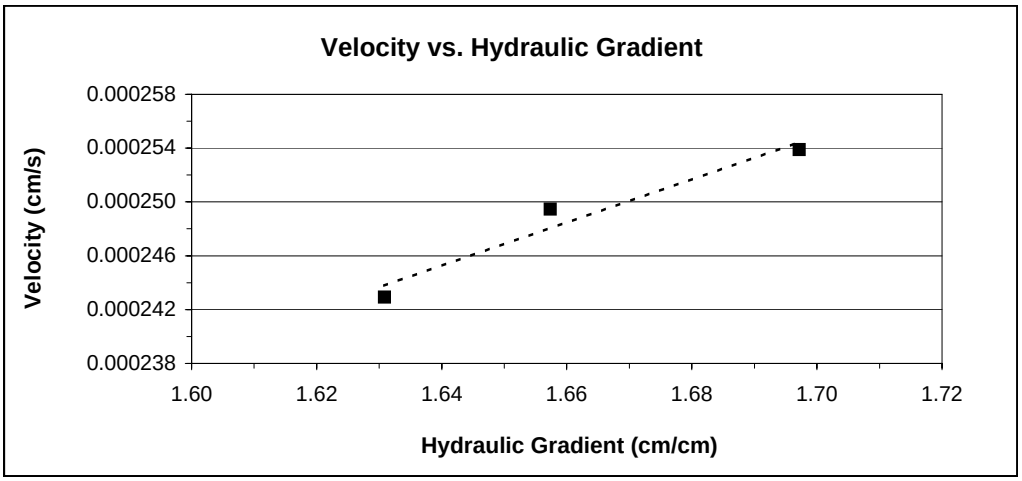
Type of water used: TAP
Collection vessel tare (g)*:* 11.00
Sample length (cm)*:* 7.54
Sample diameter (cm)*:* 4.93
Sample x-sectional area (cm^2)*:* 19.06

Date	Time	Temp (°C)	Head (cm)	Q + Tare (g)	Q (cm^3)	Elapsed time (sec)	Ksat (cm/sec)	Ksat @ 20°C (cm/sec)
Test # 1:								
27-Aug-10	10:12:58	22.0	12.8	14.15	3.2	651	1.5E-04	1.4E-04
27-Aug-10	10:23:49							
Test # 2:								
27-Aug-10	11:02:49	22.0	12.5	13.13	2.1	448	1.5E-04	1.4E-04
27-Aug-10	11:10:17							
Test # 3:								
27-Aug-10	11:45:18	22.0	12.3	13.25	2.3	486	1.5E-04	1.4E-04
27-Aug-10	11:53:24							

Average Ksat* (cm/sec)*: **1.4E-04**
Oversize Corrected Ksat* (cm/sec)*: **NA**

Comments:

--- = Oversize correction is unnecessary since coarse fraction < 5% of composite mass
NA = Not applicable



Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Saturated Hydraulic Conductivity
Constant Head Method

Job name: Barr Engineering Company
Job number: LB10.0170.00
Sample number: Blend 1 (55%)
Ring Number: NA
Depth: NA

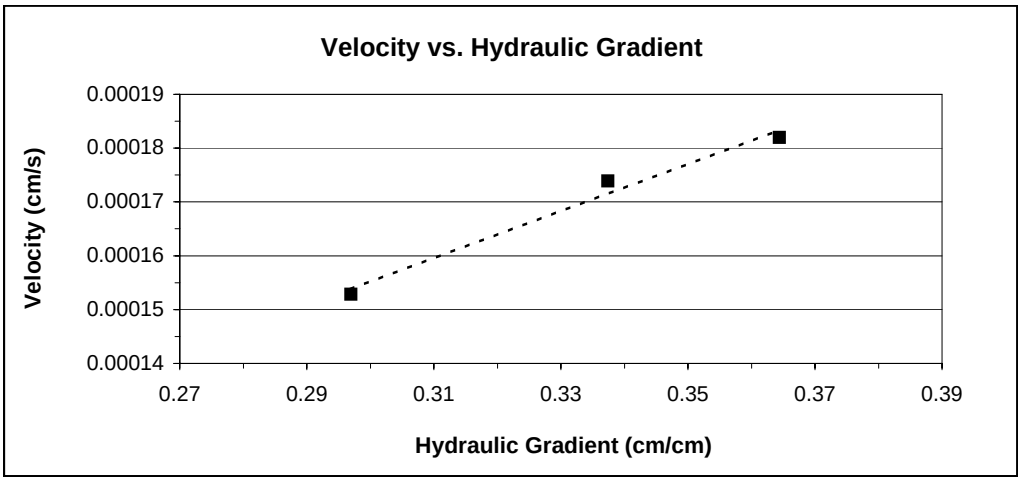
Type of water used: TAP
Collection vessel tare (g)*:* 10.96
Sample length (cm)*:* 7.41
Sample diameter (cm)*:* 4.91
Sample x-sectional area (cm^2)*:* 18.95

Date	Time	Temp (°C)	Head (cm)	Q + Tare (g)	Q (cm^3)	Elapsed time (sec)	Ksat (cm/sec)	Ksat @ 20°C (cm/sec)
Test # 1:								
31-Aug-10	11:17:19	22.0	2.2	12.22	1.3	435	5.1E-04	4.9E-04
31-Aug-10	11:24:34							
Test # 2:								
31-Aug-10	11:43:49	22.0	2.5	13.25	2.3	695	5.2E-04	4.9E-04
31-Aug-10	11:55:24							
Test # 3:								
31-Aug-10	13:05:26	22.5	2.7	13.16	2.2	638	5.0E-04	4.7E-04
31-Aug-10	13:16:04							

Average Ksat* (cm/sec)*: **4.9E-04**
Oversize Corrected Ksat* (cm/sec)*: **NA**

Comments:

--- = Oversize correction is unnecessary since coarse fraction < 5% of composite mass
NA = Not applicable



Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Saturated Hydraulic Conductivity
Constant Head Method

Job name: Barr Engineering Company
Job number: LB10.0170.00
Sample number: Blend 2 (39%)
Ring Number: NA
Depth: NA

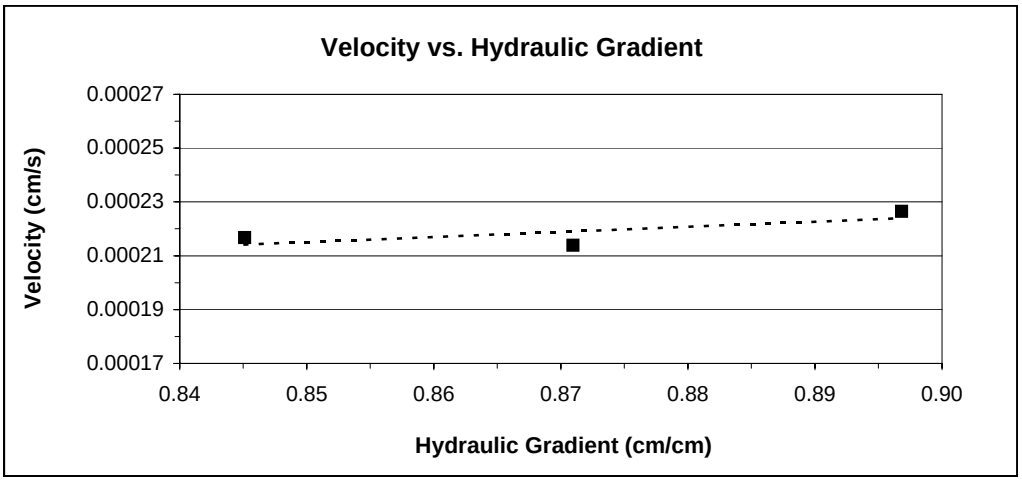
Type of water used: TAP
Collection vessel tare (g)*:* 11.01
Sample length (cm)*:* 7.74
Sample diameter (cm)*:* 4.92
Sample x-sectional area (cm^2)*:* 19.00

Date	Time	Temp (°C)	Head (cm)	Q + Tare (g)	Q (cm^3)	Elapsed time (sec)	Ksat (cm/sec)	Ksat @ 20°C (cm/sec)
Test # 1:								
31-Aug-10	11:01:35	22.0	6.9	13.55	2.5	590	2.5E-04	2.4E-04
31-Aug-10	11:11:25							
Test # 2:								
31-Aug-10	11:43:55	22.0	6.7	13.14	2.1	524	2.5E-04	2.4E-04
31-Aug-10	11:52:39							
Test # 3:								
31-Aug-10	12:22:20	22.0	6.5	12.60	1.6	386	2.6E-04	2.5E-04
31-Aug-10	12:28:46							

Average Ksat* (cm/sec)*: **2.4E-04**
Oversize Corrected Ksat* (cm/sec)*: **NA**

Comments:

--- = Oversize correction is unnecessary since coarse fraction < 5% of composite mass
NA = Not applicable



Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Saturated Hydraulic Conductivity
Constant Head Method

Job name: Barr Engineering Company
Job number: LB10.0170.00
Sample number: Blend 2 (45%)
Ring Number: NA
Depth: NA

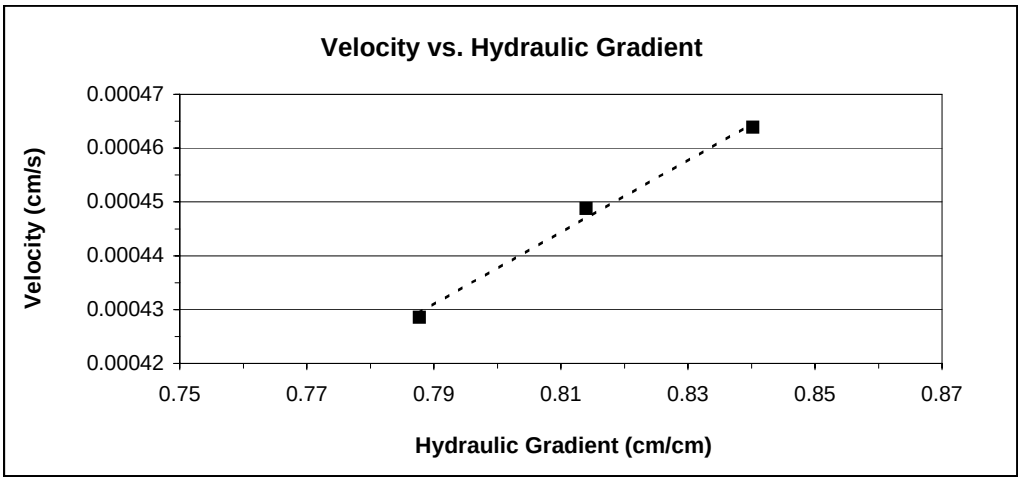
Type of water used: TAP
Collection vessel tare (g)*:* 10.95
Sample length (cm)*:* 7.62
Sample diameter (cm)*:* 4.84
Sample x-sectional area (cm^2)*:* 18.38

Date	Time	Temp (°C)	Head (cm)	Q + Tare (g)	Q (cm^3)	Elapsed time (sec)	Ksat (cm/sec)	Ksat @ 20°C (cm/sec)
Test # 1:								
27-Aug-10	11:01:12	22.0	6.4	15.28	4.3	508	5.5E-04	5.3E-04
27-Aug-10	11:09:40							
Test # 2:								
27-Aug-10	11:43:20	22.0	6.2	13.49	2.5	308	5.5E-04	5.3E-04
27-Aug-10	11:48:28							
Test # 3:								
27-Aug-10	15:25:56	22.0	6	14.10	3.2	400	5.4E-04	5.2E-04
27-Aug-10	15:32:36							

Average Ksat* (cm/sec)*: **5.2E-04**
Oversize Corrected Ksat* (cm/sec)*: **NA**

Comments:

--- = Oversize correction is unnecessary since coarse fraction < 5% of composite mass
NA = Not applicable



Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Saturated Hydraulic Conductivity
Constant Head Method

Job name: Barr Engineering Company
Job number: LB10.0170.00
Sample number: Blend 2 (55%)
Ring Number: NA
Depth: NA

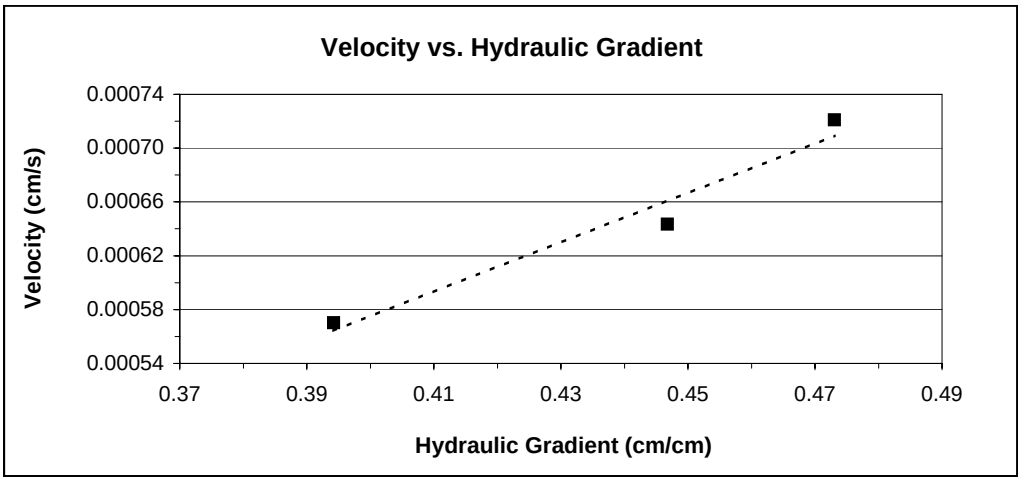
Type of water used: TAP
Collection vessel tare (g)*:* 10.93
Sample length (cm)*:* 7.61
Sample diameter (cm)*:* 4.86
Sample x-sectional area (cm^2)*:* 18.51

Date	Time	Temp (°C)	Head (cm)	Q + Tare (g)	Q (cm^3)	Elapsed time (sec)	Ksat (cm/sec)	Ksat @ 20°C (cm/sec)
Test # 1:								
31-Aug-10	10:52:46	22.0	3.6	14.00	3.1	230	1.5E-03	1.5E-03
31-Aug-10	10:56:36							
Test # 2:								
31-Aug-10	11:18:00	22.0	3.4	13.86	2.9	246	1.4E-03	1.4E-03
31-Aug-10	11:22:06							
Test # 3:								
31-Aug-10	11:43:30	22.0	3	17.38	6.5	611	1.4E-03	1.4E-03
31-Aug-10	11:53:41							

Average Ksat* (cm/sec)*: **1.4E-03**
Oversize Corrected Ksat* (cm/sec)*: **NA**

Comments:

--- = Oversize correction is unnecessary since coarse fraction < 5% of composite mass
NA = Not applicable



Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Saturated Hydraulic Conductivity
Constant Head Method

Job name:	Barr Engineering Company	*Type of water used:*	TAP
Job number:	LB10.0170.00	*Collection vessel tare* (g)*:*	10.92
Sample number:	Blend 3 (40%)	*Sample length* (cm)*:*	7.58
Ring Number:	NA	*Sample diameter* (cm)*:*	4.91
Depth:	NA	*Sample x-sectional area* (cm^2)*:*	18.90

Date	Time	Temp (°C)	Head (cm)	Q + Tare (g)	Q (cm^3)	Elapsed time (sec)	Ksat (cm/sec)	Ksat @ 20°C (cm/sec)
Test # 1:								
27-Aug-10	10:10:51	22.0	4	14.25	3.3	462	7.2E-04	6.9E-04
27-Aug-10	10:18:33							
Test # 2:								
27-Aug-10	11:01:00	22.0	3.8	14.13	3.2	470	7.2E-04	6.9E-04
27-Aug-10	11:08:50							
Test # 3:								
27-Aug-10	11:42:58	22.0	3.6	13.30	2.4	369	7.2E-04	6.9E-04
27-Aug-10	11:49:07							

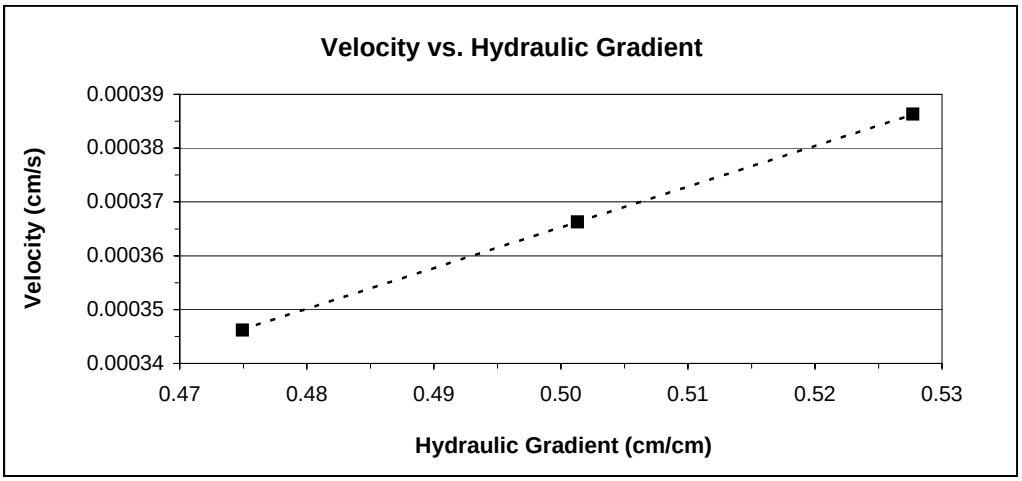
Average Ksat* (cm/sec)*: **6.9E-04**

Oversize Corrected Ksat* (cm/sec)*: **NA**

Comments:

--- = Oversize correction is unnecessary since coarse fraction < 5% of composite mass

NA = Not applicable



Laboratory analysis by: K. Wright

Data entered by: K. Wright

Checked by: J. Hines

Saturated Hydraulic Conductivity
Constant Head Method

Job name:	Barr Engineering Company	*Type of water used:*	TAP
Job number:	LB10.0170.00	*Collection vessel tare* (g)*:*	10.98
Sample number:	Blend 3 (45%)	*Sample length* (cm)*:*	7.56
Ring Number:	NA	*Sample diameter* (cm)*:*	4.90
Depth:	NA	*Sample x-sectional area* (cm^2)*:*	18.89

Date	Time	Temp (°C)	Head (cm)	Q + Tare (g)	Q (cm^3)	Elapsed time (sec)	Ksat (cm/sec)	Ksat @ 20°C (cm/sec)
Test # 1:								
27-Aug-10	10:10:37	22.0	3.5	20.69	9.7	527	2.1E-03	2.0E-03
27-Aug-10	10:19:24							
Test # 2:								
27-Aug-10	11:00:49	22.0	3.3	18.71	7.7	449	2.1E-03	2.0E-03
27-Aug-10	11:08:18							
Test # 3:								
27-Aug-10	11:42:34	22.0	3.1	15.43	4.5	278	2.1E-03	2.0E-03
27-Aug-10	11:47:12							

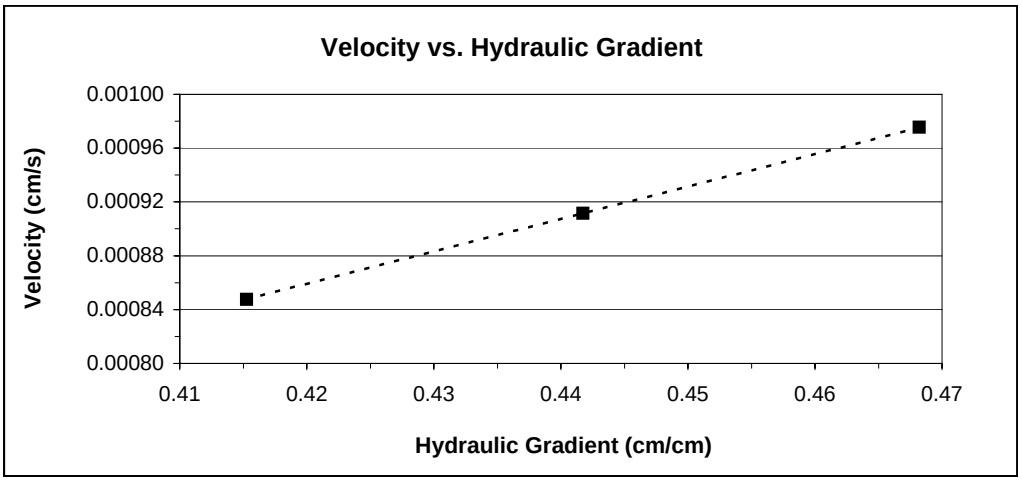
Average Ksat* (cm/sec)*: **2.0E-03**

Oversize Corrected Ksat* (cm/sec)*: **NA**

Comments:

--- = Oversize correction is unnecessary since coarse fraction < 5% of composite mass

NA = Not applicable



Laboratory analysis by: K. Wright

Data entered by: K. Wright

Checked by: J. Hines

Saturated Hydraulic Conductivity
Constant Head Method

Job name:	Barr Engineering Company	*Type of water used:*	TAP
Job number:	LB10.0170.00	*Collection vessel tare* (g)*:*	10.96
Sample number:	Blend 3 (55%)	*Sample length* (cm)*:*	7.71
Ring Number:	NA	*Sample diameter* (cm)*:*	4.92
Depth:	NA	*Sample x-sectional area* (cm^2)*:*	18.99

Date	Time	Temp (°C)	Head (cm)	Q + Tare (g)	Q (cm^3)	Elapsed time (sec)	Ksat (cm/sec)	Ksat @ 20°C (cm/sec)
Test # 1:								
27-Aug-10	10:11:45	22.0	3.6	15.80	4.8	327	1.7E-03	1.6E-03
27-Aug-10	10:17:12							
Test # 2:								
27-Aug-10	11:01:59	22.0	3.1	14.96	4.0	301	1.7E-03	1.7E-03
27-Aug-10	11:07:00							
Test # 3:								
27-Aug-10	11:44:18	22.0	3.2	15.18	4.2	359	1.5E-03	1.4E-03
27-Aug-10	11:50:17							

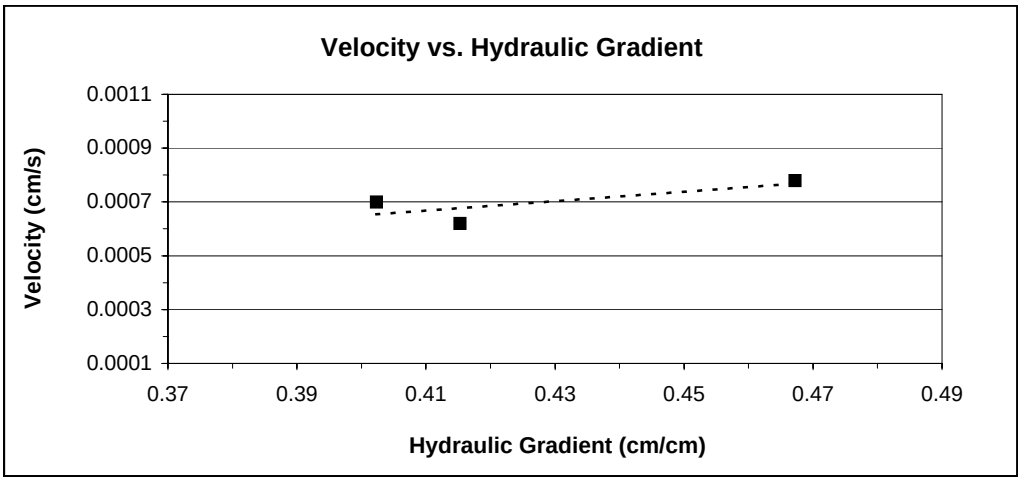
Average Ksat* (cm/sec)*: **1.6E-03**

Oversize Corrected Ksat* (cm/sec)*: **NA**

Comments:

--- = Oversize correction is unnecessary since coarse fraction < 5% of composite mass

NA = Not applicable



Laboratory analysis by: K. Wright

Data entered by: K. Wright

Checked by: J. Hines

Saturated Hydraulic Conductivity
Constant Head Method

Job name: Barr Engineering Company
Job number: LB10.0170.00
Sample number: Blend 4 (44%)
Ring Number: NA
Depth: NA

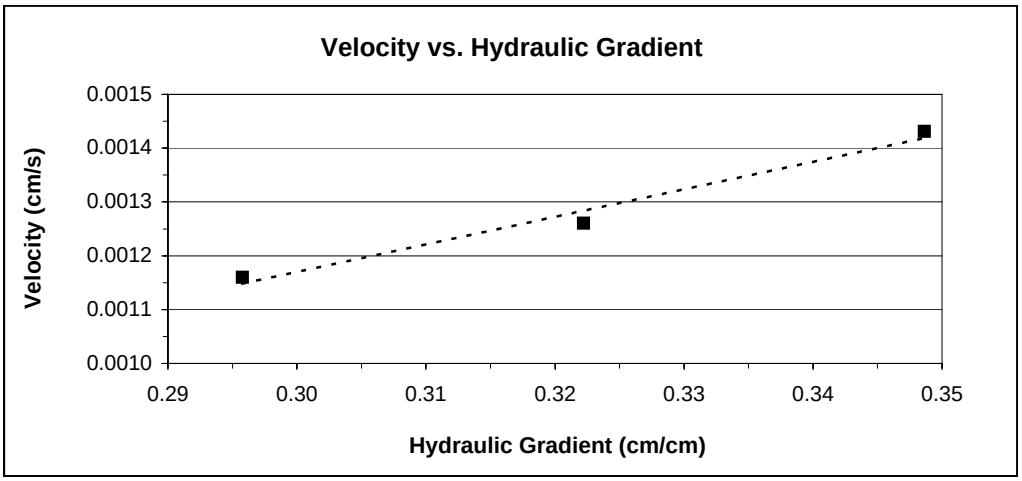
Type of water used: TAP
Collection vessel tare (g)*:* 10.92
Sample length (cm)*:* 7.57
Sample diameter (cm)*:* 4.93
Sample x-sectional area (cm^2)*:* 19.09

Date	Time	Temp (°C)	Head (cm)	Q + Tare (g)	Q (cm^3)	Elapsed time (sec)	Ksat (cm/sec)	Ksat @ 20°C (cm/sec)
Test # 1:								
27-Aug-10	10:12:47	22.0	2.6	17.01	6.1	231	4.0E-03	3.8E-03
27-Aug-10	10:16:38							
Test # 2:								
27-Aug-10	11:02:38	22.0	2.4	15.91	5.0	216	3.8E-03	3.6E-03
27-Aug-10	11:06:14							
Test # 3:								
27-Aug-10	11:45:06	22.0	2.2	16.81	5.9	278	3.8E-03	3.6E-03
27-Aug-10	11:49:44							

Average Ksat* (cm/sec)*: **3.7E-03**
Oversize Corrected Ksat* (cm/sec)*: **NA**

Comments:

--- = Oversize correction is unnecessary since coarse fraction < 5% of composite mass
NA = Not applicable



Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Saturated Hydraulic Conductivity
Constant Head Method

Job name:	Barr Engineering Company	*Type of water used:*	TAP
Job number:	LB10.0170.00	*Collection vessel tare* (g)*:*	10.94
Sample number:	Blend 4 (50%)	*Sample length* (cm)*:*	7.61
Ring Number:	NA	*Sample diameter* (cm)*:*	4.91
Depth:	NA	*Sample x-sectional area* (cm^2)*:*	18.90

Date	Time	Temp (°C)	Head (cm)	Q + Tare (g)	Q (cm^3)	Elapsed time (sec)	Ksat (cm/sec)	Ksat @ 20°C (cm/sec)
Test # 1:								
13-Sep-10	10:01:53	22.0	3.5	20.27	9.3	268	4.0E-03	3.8E-03
13-Sep-10	10:06:21							
Test # 2:								
13-Sep-10	10:29:52	22.0	3.3	18.92	8.0	244	4.0E-03	3.8E-03
13-Sep-10	10:33:56							
Test # 3:								
13-Sep-10	10:54:06	22.0	3	17.64	6.7	216	4.2E-03	4.0E-03
13-Sep-10	10:57:42							

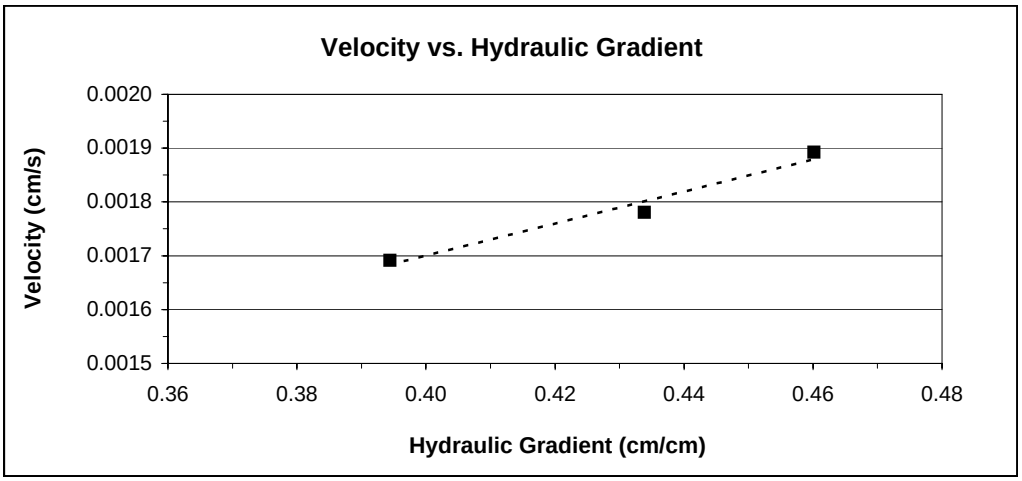
Average Ksat* (cm/sec)*: **3.9E-03**

Oversize Corrected Ksat* (cm/sec)*: **NA**

Comments:

--- = Oversize correction is unnecessary since coarse fraction < 5% of composite mass

NA = Not applicable



Laboratory analysis by: K. Wright

Data entered by: K. Wright

Checked by: J. Hines

Saturated Hydraulic Conductivity
Constant Head Method

Job name:	Barr Engineering Company	*Type of water used:*	TAP
Job number:	LB10.0170.00	*Collection vessel tare* (g)*:*	10.93
Sample number:	Blend 4 (55%)	*Sample length* (cm)*:*	7.63
Ring Number:	NA	*Sample diameter* (cm)*:*	4.91
Depth:	NA	*Sample x-sectional area* (cm^2)*:*	18.91

Date	Time	Temp (°C)	Head (cm)	Q + Tare (g)	Q (cm^3)	Elapsed time (sec)	Ksat (cm/sec)	Ksat @ 20°C (cm/sec)
Test # 1:								
13-Sep-10	10:02:00	22.0	3.2	26.89	16.0	208	9.7E-03	9.2E-03
13-Sep-10	10:05:28							
Test # 2:								
13-Sep-10	10:30:02	22.0	2.8	23.15	12.2	170	1.0E-02	9.9E-03
13-Sep-10	10:32:52							
Test # 3:								
13-Sep-10	10:54:14	22.0	2.5	21.68	10.8	162	1.1E-02	1.0E-02
13-Sep-10	10:56:56							

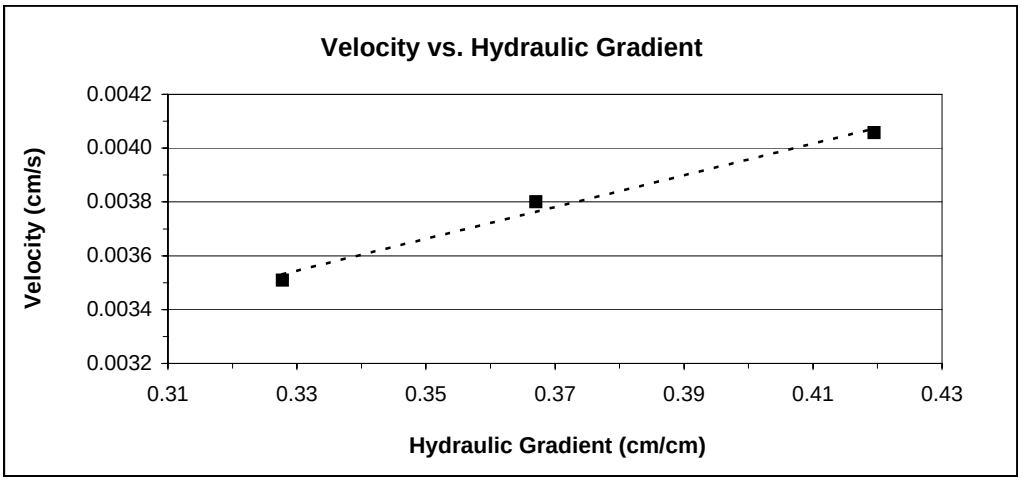
Average Ksat* (cm/sec)*: **9.8E-03**

Oversize Corrected Ksat* (cm/sec)*: **NA**

Comments:

--- = Oversize correction is unnecessary since coarse fraction < 5% of composite mass

NA = Not applicable



Laboratory analysis by: K. Wright

Data entered by: K. Wright

Checked by: J. Hines

Moisture Retention Characteristics

Summary of Moisture Characteristics of the Initial Drainage Curve

Sample Number	Pressure Head (-cm water)	Moisture Content (%, cm^3/cm^3)
Blend 1 (38%)	0	40.3
	17	39.9
	49	39.7
	123	39.7
	337	25.9
	14685	3.8
	38752	2.2
	124416	1.7
	861325	0.7
Blend 1 (45%)	0	45.0
	12	43.8
	31	42.8
	101	42.4 ‡‡
	337	25.9 ‡‡
	17439	3.9 ‡‡
	41200	2.5 ‡‡
	132268	1.3 ‡‡
	861325	0.7 ‡‡
Blend 1 (55%)	0	48.0 ‡‡
	12	40.8 ‡‡
	31	38.6 ‡‡
	101	37.4 ‡‡
	337	33.7 ‡‡
	17745	4.2 ‡‡
	44769	2.6 ‡‡
	109323	1.9 ‡‡
	861325	0.7 ‡‡

‡‡ Volume adjustments are applicable at this matric potential (see data sheet for this sample).

Summary of Moisture Characteristics of the Initial Drainage Curve (Continued)

Sample Number	Pressure Head (-cm water)	Moisture Content (%, cm^3/cm^3)
Blend 2 (39%)	0	39.9
	12	38.6
	31	38.2
	101	37.9
	337	16.9
	16419	2.5
	35489	1.8
	142772	1.3
	861325	0.6
Blend 2 (45%)	0	45.8 ‡‡
	12	43.5 ‡‡
	31	41.7 ‡‡
	101	40.2 ‡‡
	337	19.4 ‡‡
	14583	2.6 ‡‡
	35489	1.5 ‡‡
	126455	1.0 ‡‡
	861325	0.6 ‡‡
Blend 2 (55%)	0	46.6 ‡‡
	5	41.1 ‡‡
	21	39.4 ‡‡
	76	37.3 ‡‡
	337	21.6 ‡‡
	17541	2.9 ‡‡
	44565	1.7 ‡‡
	106875	1.4 ‡‡
	861325	0.7 ‡‡

‡‡ Volume adjustments are applicable at this matric potential (see data sheet for this sample).

Summary of Moisture Characteristics of the Initial Drainage Curve (Continued)

Sample Number	Pressure Head (-cm water)	Moisture Content (%, cm^3/cm^3)
Blend 3 (40%)	0	39.7
	12	39.1
	31	38.8
	101	27.0
	337	12.3
	16215	3.2
	36611	2.9
	138387	2.0
	861325	0.6
Blend 3 (45%)	0	44.9 ‡‡
	5	42.9 ‡‡
	21	41.5 ‡‡
	76	39.7 ‡‡
	337	12.1 ‡‡
	14481	2.0 ‡‡
	41098	1.3 ‡‡
	141752	0.9 ‡‡
	861325	0.6 ‡‡
Blend 3 (55%)	0	44.0 ‡‡
	5	38.7 ‡‡
	21	37.1 ‡‡
	76	36.7 ‡‡
	337	12.1 ‡‡
	12951	1.9 ‡‡
	34673	1.3 ‡‡
	123396	1.0 ‡‡
	861325	0.6 ‡‡

‡‡ Volume adjustments are applicable at this matric potential (see data sheet for this sample).

Summary of Moisture Characteristics of the Initial Drainage Curve (Continued)

Sample Number	Pressure Head (-cm water)	Moisture Content (%, cm^3/cm^3)
Blend 4 (44%)	0	43.1
	5	42.8
	21	42.5
	76	32.7
	337	5.3
	19274	1.2
	57619	0.9
	151440	0.7
	861325	0.5
Blend 4 (50%)	0	42.2 ‡‡
	6	42.1 ‡‡
	30	41.4 ‡‡
	71	26.3 ‡‡
	337	4.5 ‡‡
	16929	1.4 ‡‡
	46197	1.0 ‡‡
	265658	0.8 ‡‡
	861325	0.4 ‡‡
Blend 4 (55%)	0	41.6 ‡‡
	6	40.7 ‡‡
	30	39.8 ‡‡
	71	29.3 ‡‡
	337	4.7 ‡‡
	6323	2.0 ‡‡
	66695	0.6 ‡‡
	105345	0.7 ‡‡
	861325	0.5 ‡‡

‡‡ Volume adjustments are applicable at this matric potential (see data sheet for this sample).

Summary of Calculated Unsaturated Hydraulic Properties

Sample Number	α (cm^{-1})	N (dimensionless)	θ_r (% vol)	θ_s (% vol)	Oversize Corrected θ_r (% vol)	Oversize Corrected θ_s (% vol)
Blend 1 (38%)	0.0028	4.4060	2.10	39.97	NA	NA
Blend 1 (45%)	0.0037	2.3302	2.04	44.09	NA	NA
Blend 1 (55%)	0.0032	1.5352	0.00	42.21	NA	NA
Blend 2 (39%)	0.0039	3.4263	1.55	38.91	NA	NA
Blend 2 (45%)	0.0049	2.3810	1.36	43.86	NA	NA
Blend 2 (55%)	0.0097	1.5315	0.20	43.07	NA	NA
Blend 3 (40%)	0.0112	1.9686	2.08	39.94	NA	NA
Blend 3 (45%)	0.0065	2.6280	1.17	43.17	NA	NA
Blend 3 (55%)	0.0066	2.4829	1.18	40.08	NA	NA
Blend 4 (44%)	0.0106	2.7451	0.81	42.95	NA	NA
Blend 4 (50%)	0.0148	2.7336	1.07	42.67	NA	NA
Blend 4 (55%)	0.0127	2.6317	0.95	41.31	NA	NA

--- = Oversize correction is unnecessary since coarse fraction < 5% of composite mass

NR = Not requested

NA = Not applicable

Moisture Retention Data

Hanging Column / Pressure Plate

(Soil-Water Characteristic Curve)

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 1 (38%)
Ring Number: NA
Depth: NA

Dry wt. of sample (g)*:* 264.49
Tare wt., ring (g)*:* 88.69
Tare wt., screen & clamp (g)*:* 26.14
Initial sample volume (cm^3)*:* 142.91
Initial dry bulk density (g/cm^3)*:* 1.85
Measured particle density (g/cm^3)*:* 2.99
Initial calculated total porosity (%)*:* 38.07

	Date	Time	Weight* (g)	Matric Potential (-cm water)	Moisture Content† (% vol)
Hanging column:	2-Sep-10	16:20	436.88	0	40.28
	13-Sep-10	9:35	436.34	17.0	39.90
	19-Sep-10	9:05	436.09	49.0	39.72
	26-Sep-10	8:25	436.05	123.0	39.70
Pressure plate:	4-Oct-10	10:55	416.31	337	25.88

Volume Adjusted Data [1]

	Matric Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calculated Porosity (%)
Hanging column:	0.0	---	---	---	---
	17.0	---	---	---	---
	49.0	---	---	---	---
	123.0	---	---	---	---
Pressure plate:	337	---	---	---	---

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent each of the volume change measurements obtained after saturated hydraulic conductivity testing and throughout hanging column/pressure plate testing. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

† Assumed density of water is 1.0 g/cm^3

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Technician Notes:

Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Moisture Retention Data

Dew Point Potentiometer / Relative Humidity Box

(Soil-Water Characteristic Curve)

Sample Number: Blend 1 (38%)

Initial sample bulk density (g/cm^3)*:* 1.85
Fraction of bulk sample used (<2.00mm fraction) (%): 100.00

Dry weight of dew point potentiometer sample* (g)*:* 152.68
Tare weight, jar (g)*:* 119.36

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)
Dew point potentiometer:	23-Sep-10	11:15	153.36	14685	3.78
	23-Sep-10	10:56	153.08	38752	2.22
	23-Sep-10	15:23	152.98	124416	1.67

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Dew point potentiometer:	14685	---	---	---	---
	38752	---	---	---	---
	124416	---	---	---	---

Dry weight of relative humidity box sample* (g)*:* 66.13
Tare weight (g)*:* 31.76

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)
Relative humidity box:	16-Sep-10	10:45	66.27	861325	0.74

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Relative humidity box:	861325	---	---	---	---

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent the volume change measurements obtained after the last hanging column or pressure plate point. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

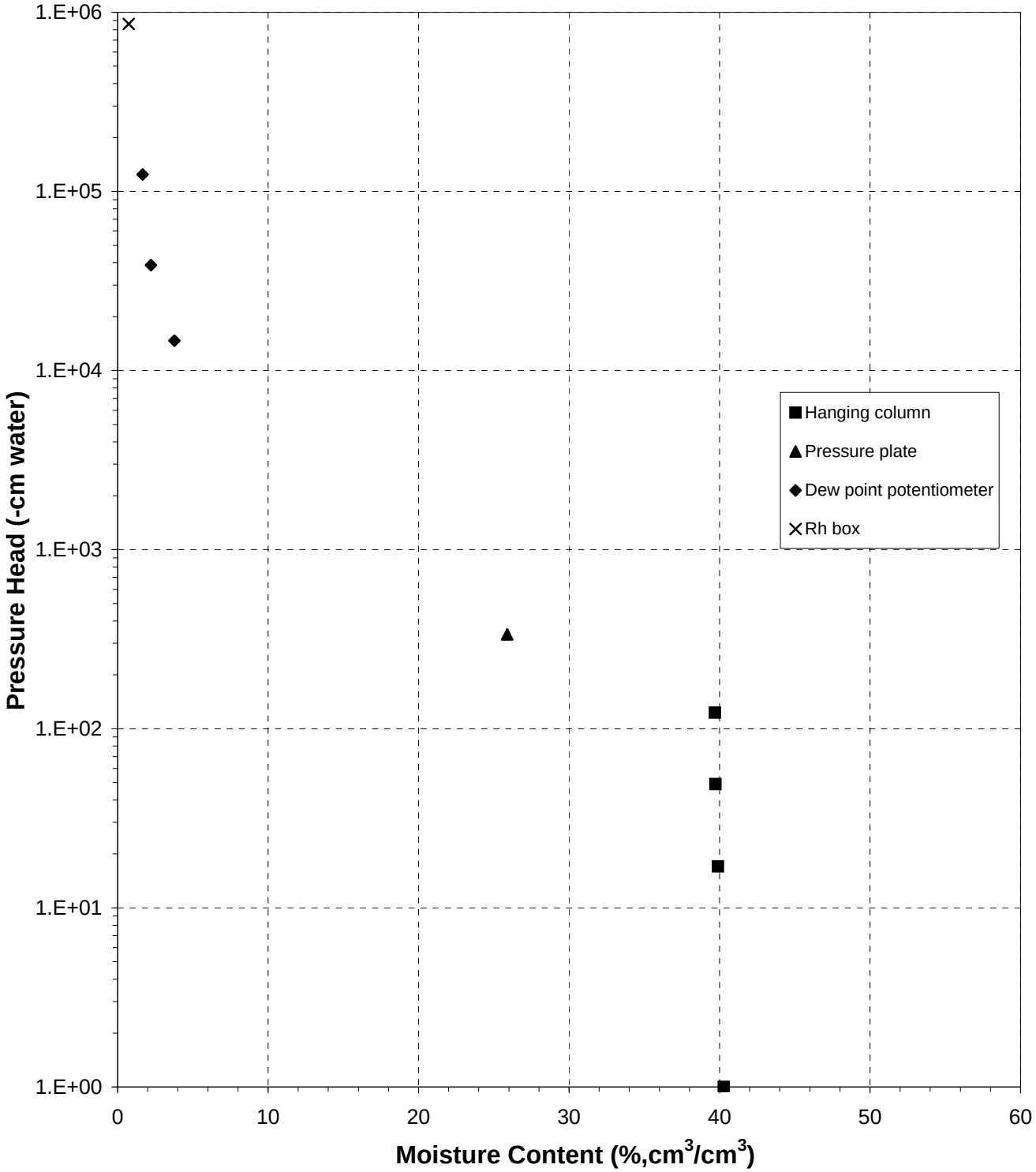
† Adjusted for >2.00mm (#10 sieve) material not used in DPP/RH testing. Assumed moisture content of material >2.00mm is zero, and assumed density of water is 1.0 g/cm^3.

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Laboratory analysis by: M. Vigil
Data entered by: K. Wright
Checked by: J. Hines

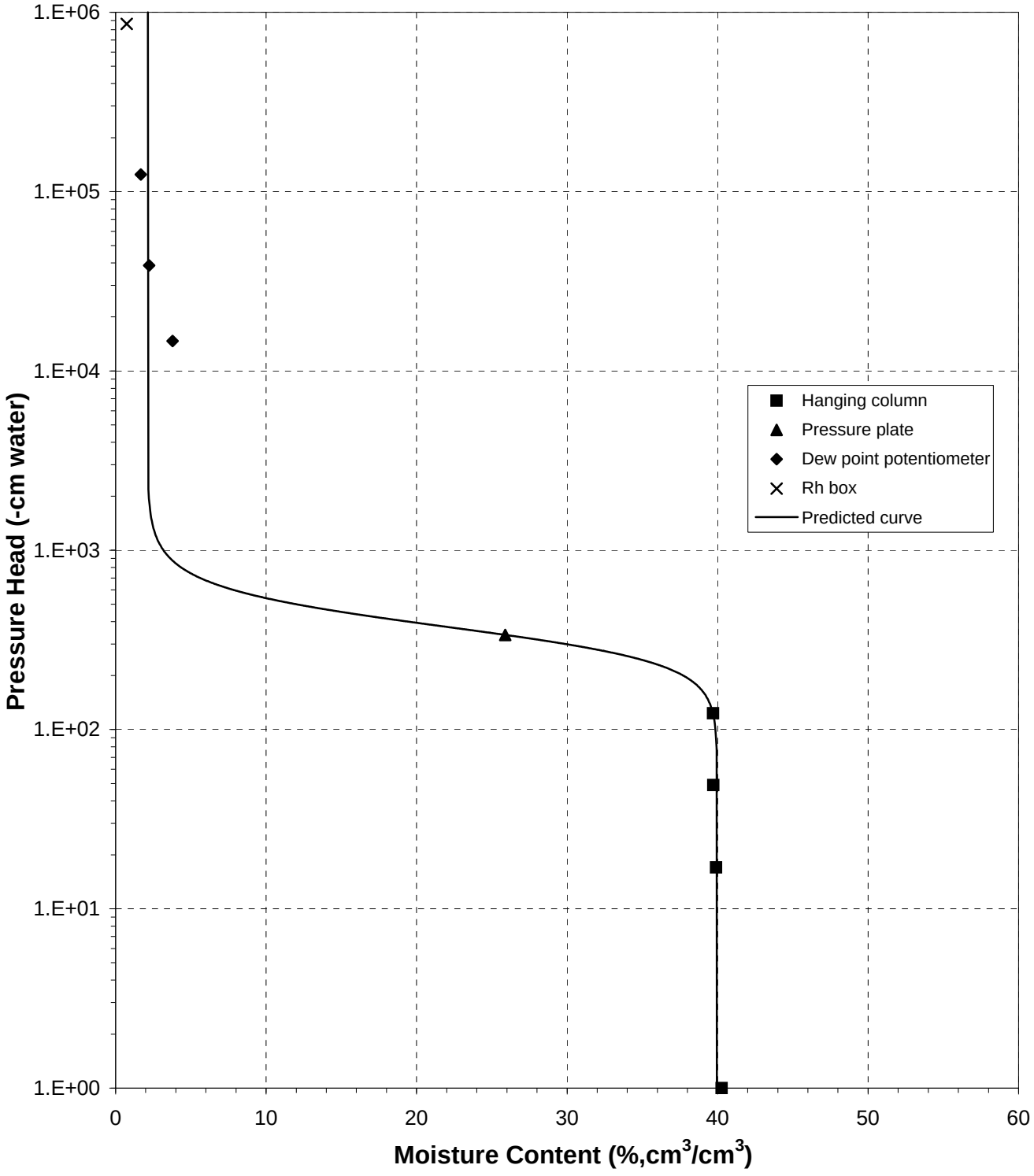
Water Retention Data Points

Sample Number: Blend 1 (38%)



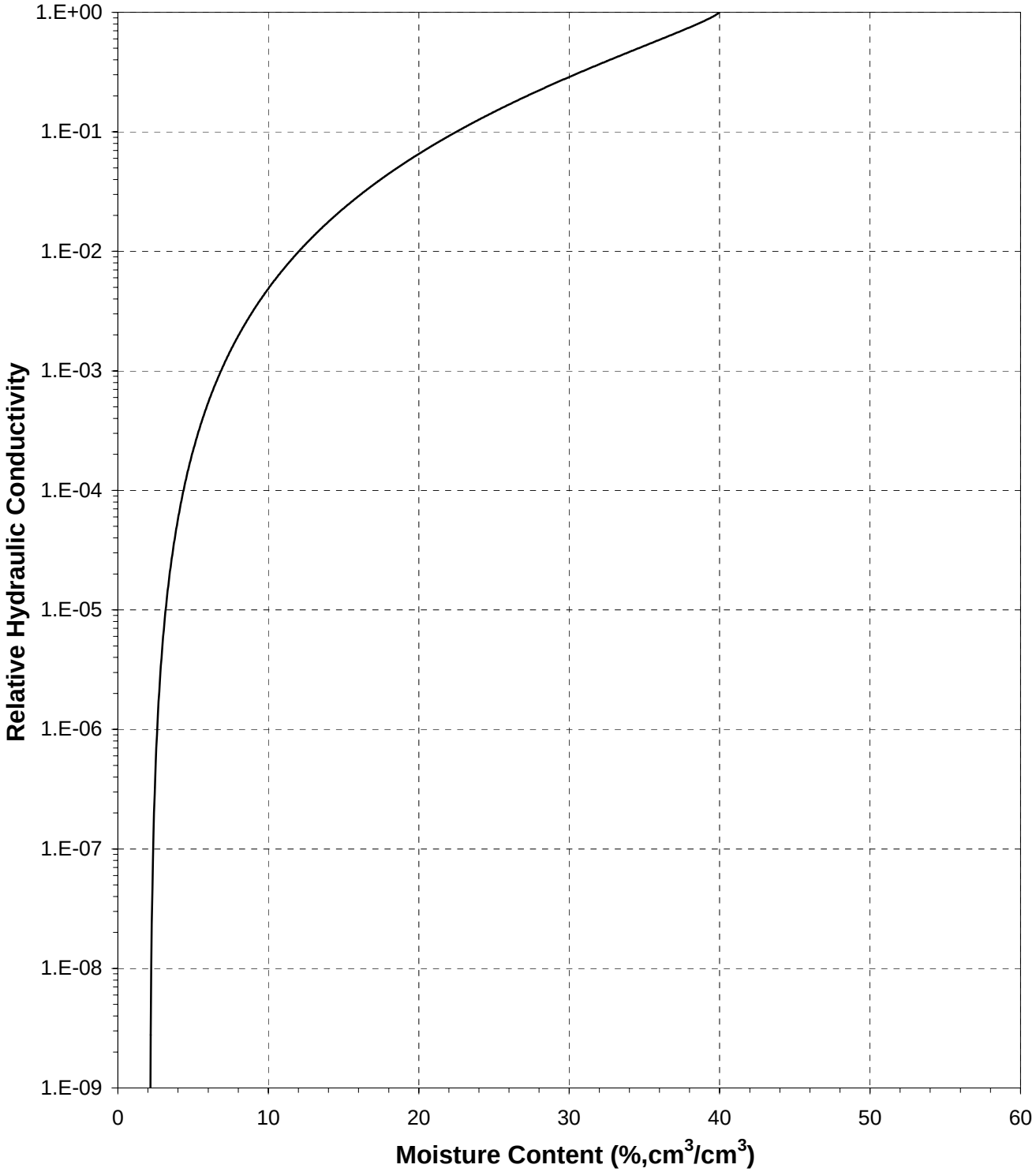
Predicted Water Retention Curve and Data Points

Sample Number: Blend 1 (38%)



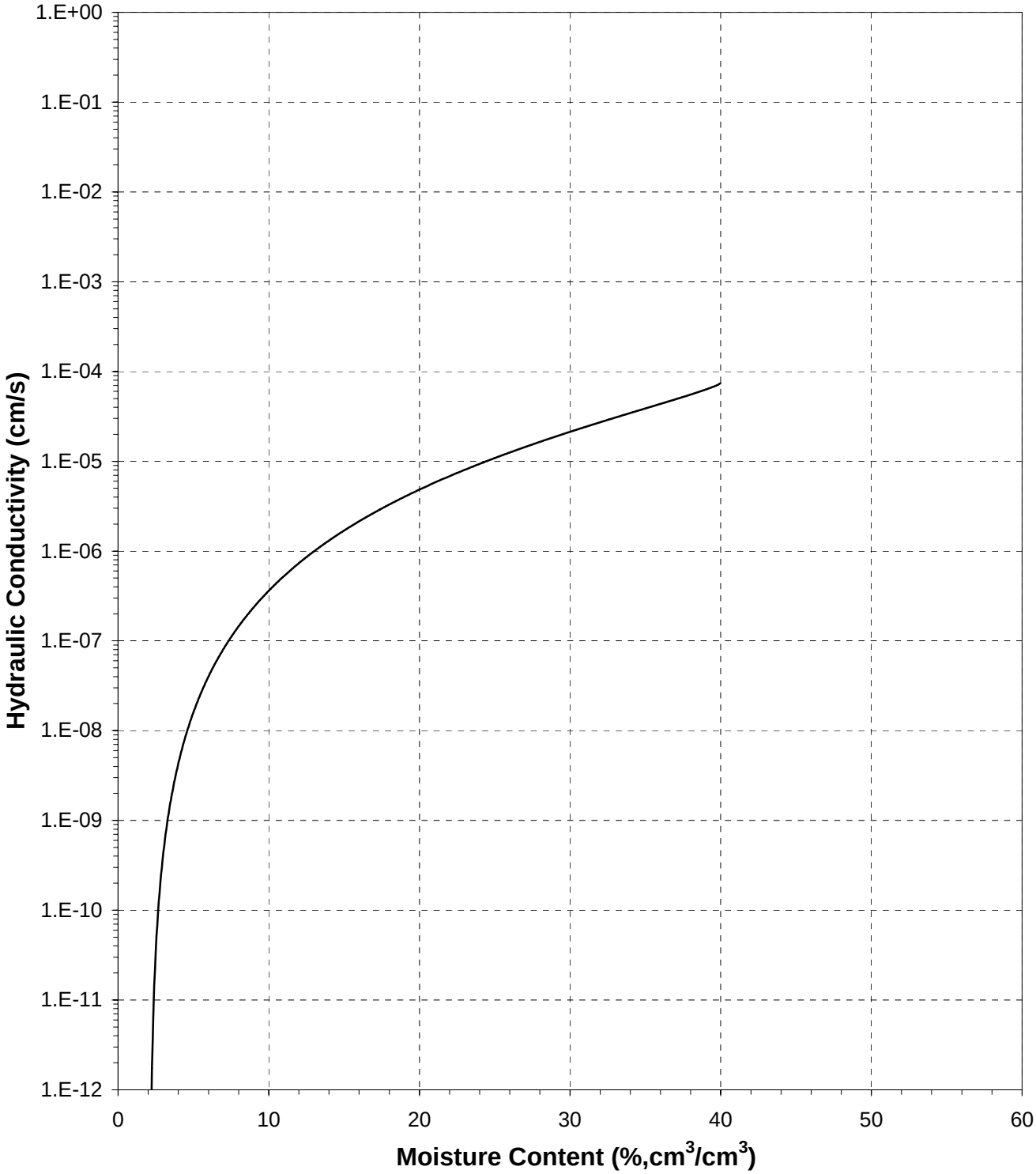
Plot of Relative Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 1 (38%)



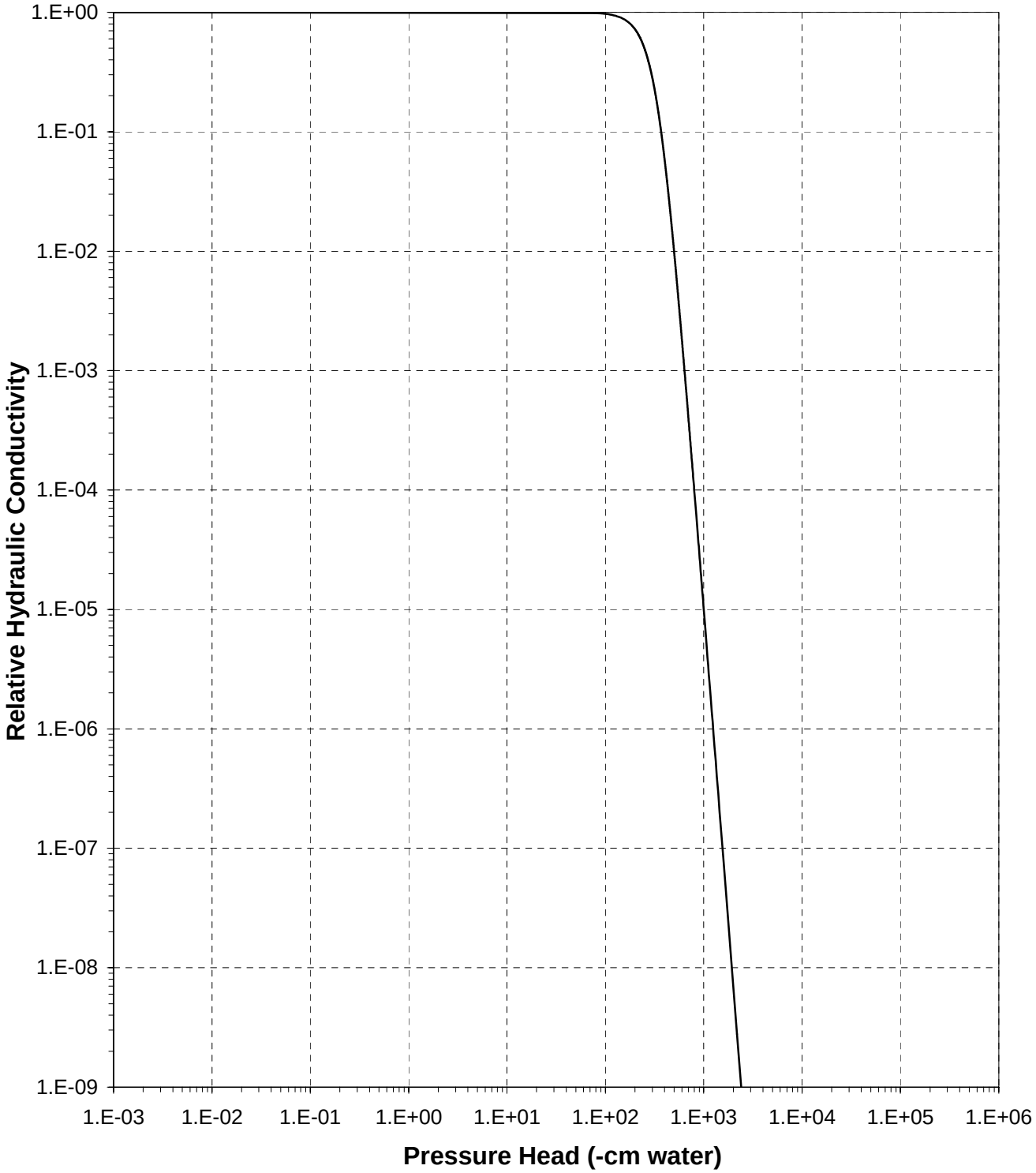
Plot of Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 1 (38%)



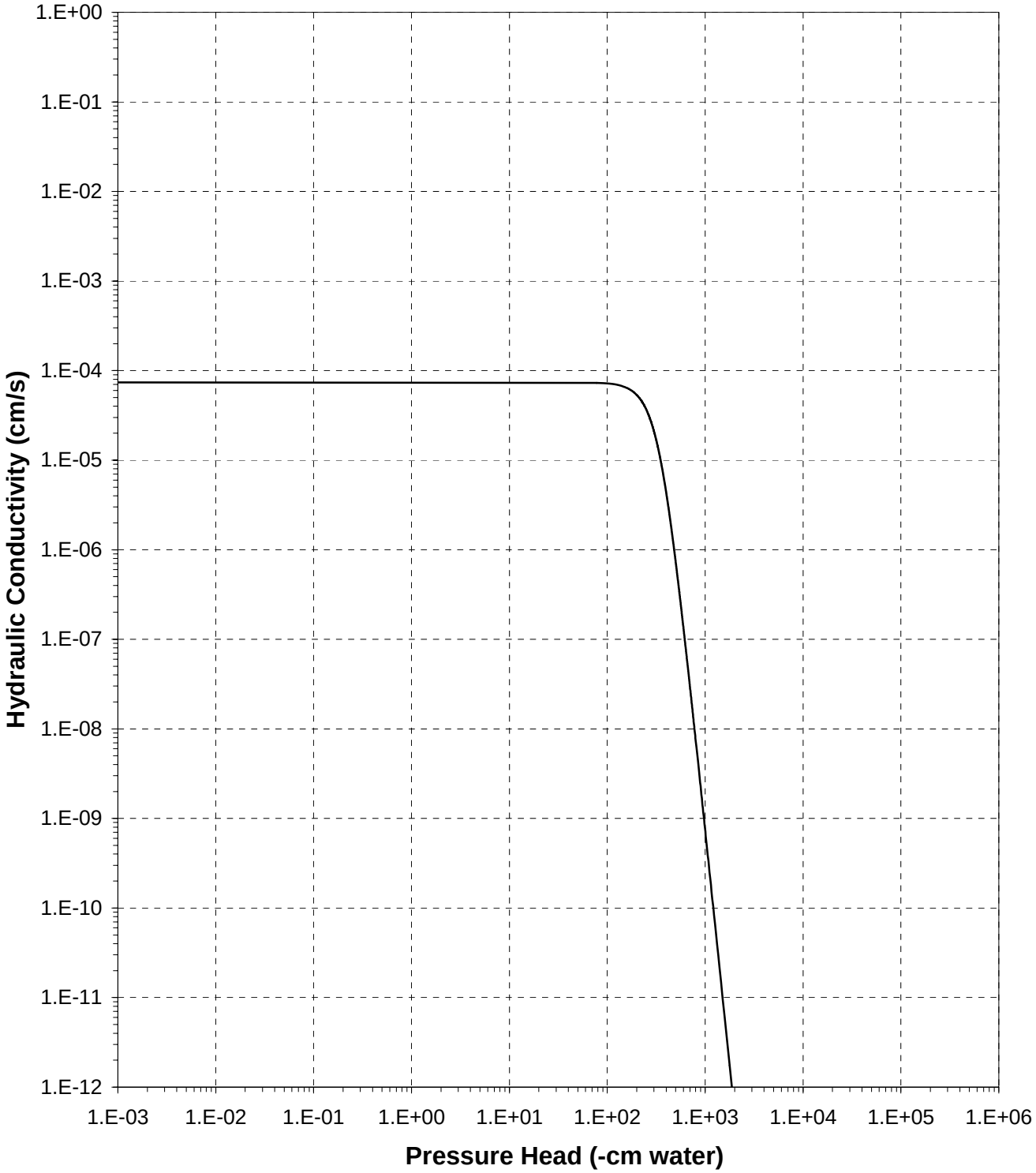
Plot of Relative Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 1 (38%)



Plot of Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 1 (38%)



Moisture Retention Data

Hanging Column / Pressure Plate

(Soil-Water Characteristic Curve)

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 1 (45%)
Ring Number: NA
Depth: NA

Dry wt. of sample (g)*:* 235.17
Tare wt., ring (g)*:* 87.09
Tare wt., screen & clamp (g)*:* 27.68
Initial sample volume (cm^3)*:* 143.74
Initial dry bulk density (g/cm^3)*:* 1.64
Measured particle density (g/cm^3)*:* 2.99
Initial calculated total porosity (*%*)*:* 45.25

	Date	Time	Weight* (g)	Matric Potential (-cm water)	Moisture Content † (% vol)	
Hanging column:	7-Sep-10	10:40	414.56	0	44.96	
	13-Sep-10	10:25	412.95	12.0	43.84	
	19-Sep-10	10:25	411.51	31.0	42.84	
	26-Sep-10	8:50	408.89	101.0	42.37	‡‡
Pressure plate:	4-Oct-10	12:45	385.77	337	25.87	‡‡

Volume Adjusted Data [1]

	Matric Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calculated Porosity (%)
Hanging column:	0.0	---	---	---	---
	12.0	---	---	---	---
	31.0	---	---	---	---
	101.0	139.14	-3.20%	1.69	43.45
Pressure plate:	337	138.51	-3.63%	1.70	43.19

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent each of the volume change measurements obtained after saturated hydraulic conductivity testing and throughout hanging column/pressure plate testing. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

† Assumed density of water is 1.0 g/cm^3

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Technician Notes:

Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Moisture Retention Data

Dew Point Potentiometer / Relative Humidity Box

(Soil-Water Characteristic Curve)

Sample Number: Blend 1 (45%)

Initial sample bulk density (g/cm^3)*:* 1.64

Fraction of bulk sample used (<2.00mm fraction) (%): 100.00

Dry weight of dew point potentiometer sample* (g)*:* 152.33

Tare weight, jar (g)*:* 112.72

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)	
Dew point potentiometer:	23-Sep-10	11:34	153.25	17439	3.94	‡‡
	23-Sep-10	11:10	152.91	41200	2.49	‡‡
	24-Sep-10	8:48	152.64	132268	1.33	‡‡

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Dew point potentiometer:	17439	138.51	-3.63%	1.70	43.19
	41200	138.51	-3.63%	1.70	43.19
	132268	138.51	-3.63%	1.70	43.19

Dry weight of relative humidity box sample* (g)*:* 71.84

Tare weight (g)*:* 39.94

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)	
Relative humidity box:	16-Sep-10	10:45	71.97	861325	0.67	‡‡

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Relative humidity box:	861325	138.51	-3.63%	1.70	43.19

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent the volume change measurements obtained after the last hanging column or pressure plate point. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

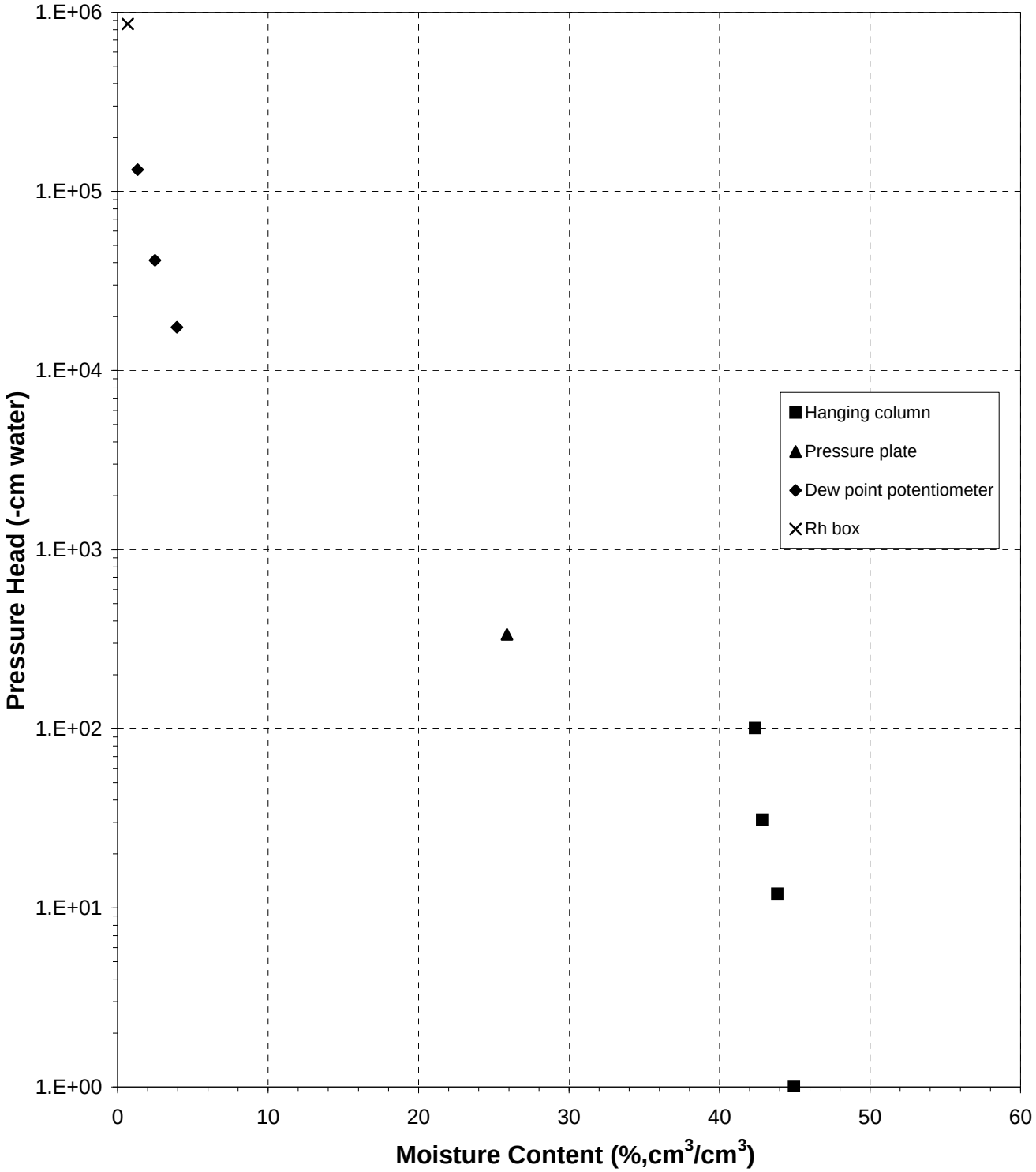
† Adjusted for >2.00mm (#10 sieve) material not used in DPP/RH testing. Assumed moisture content of material >2.00mm is zero, and assumed density of water is 1.0 g/cm^3.

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Laboratory analysis by: M. Vigil
Data entered by: K. Wright
Checked by: J. Hines

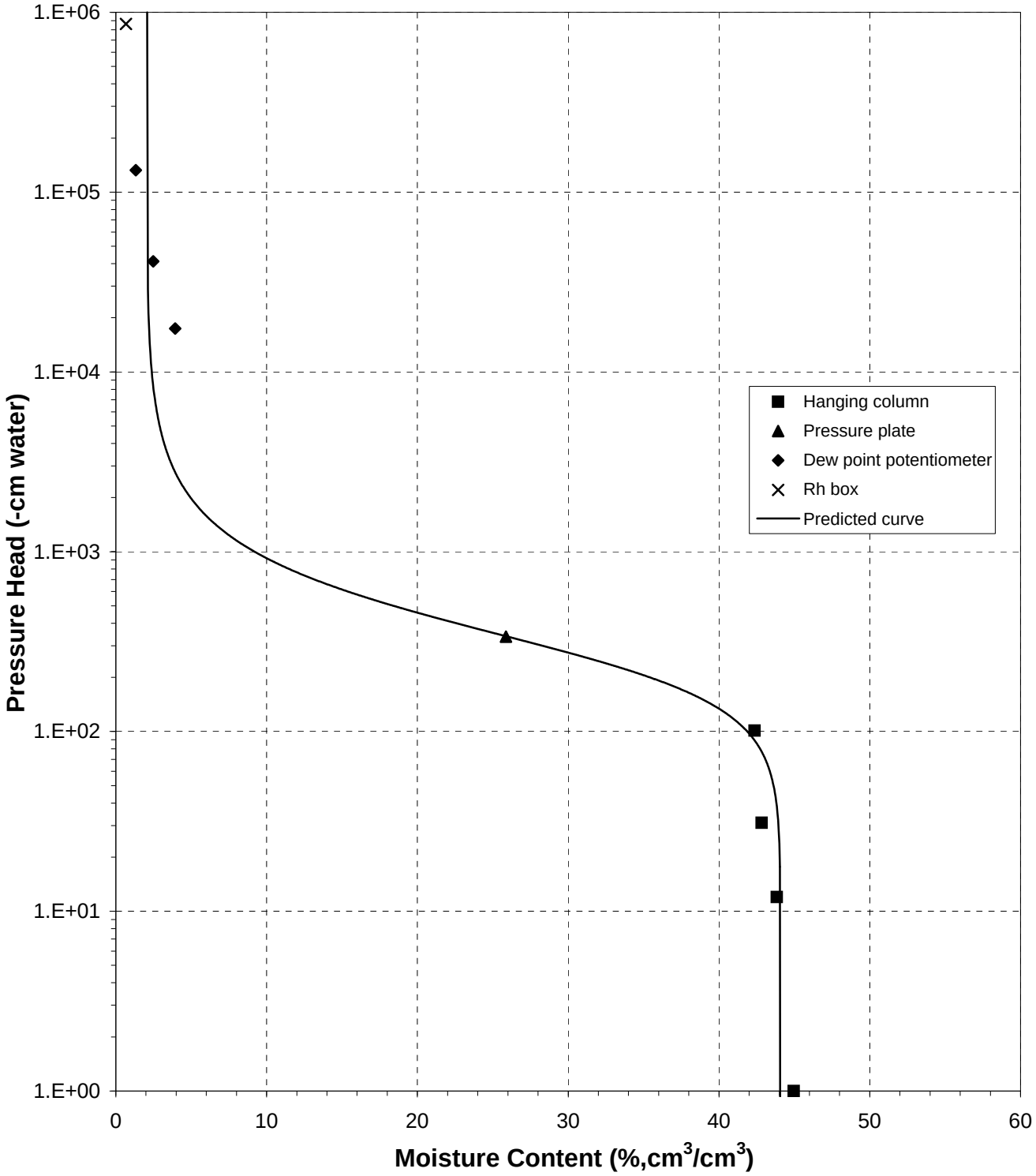
Water Retention Data Points

Sample Number: Blend 1 (45%)



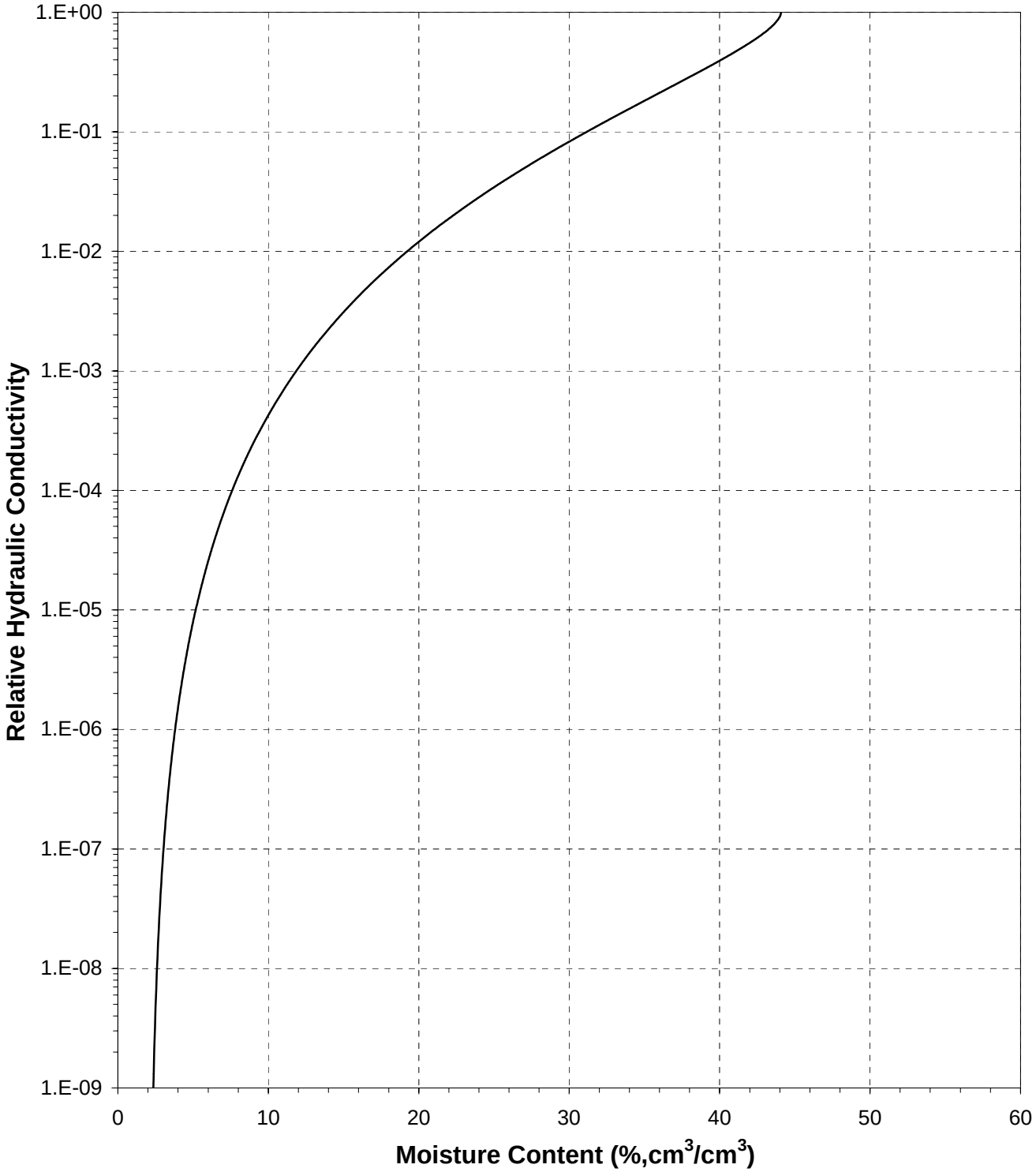
Predicted Water Retention Curve and Data Points

Sample Number: Blend 1 (45%)



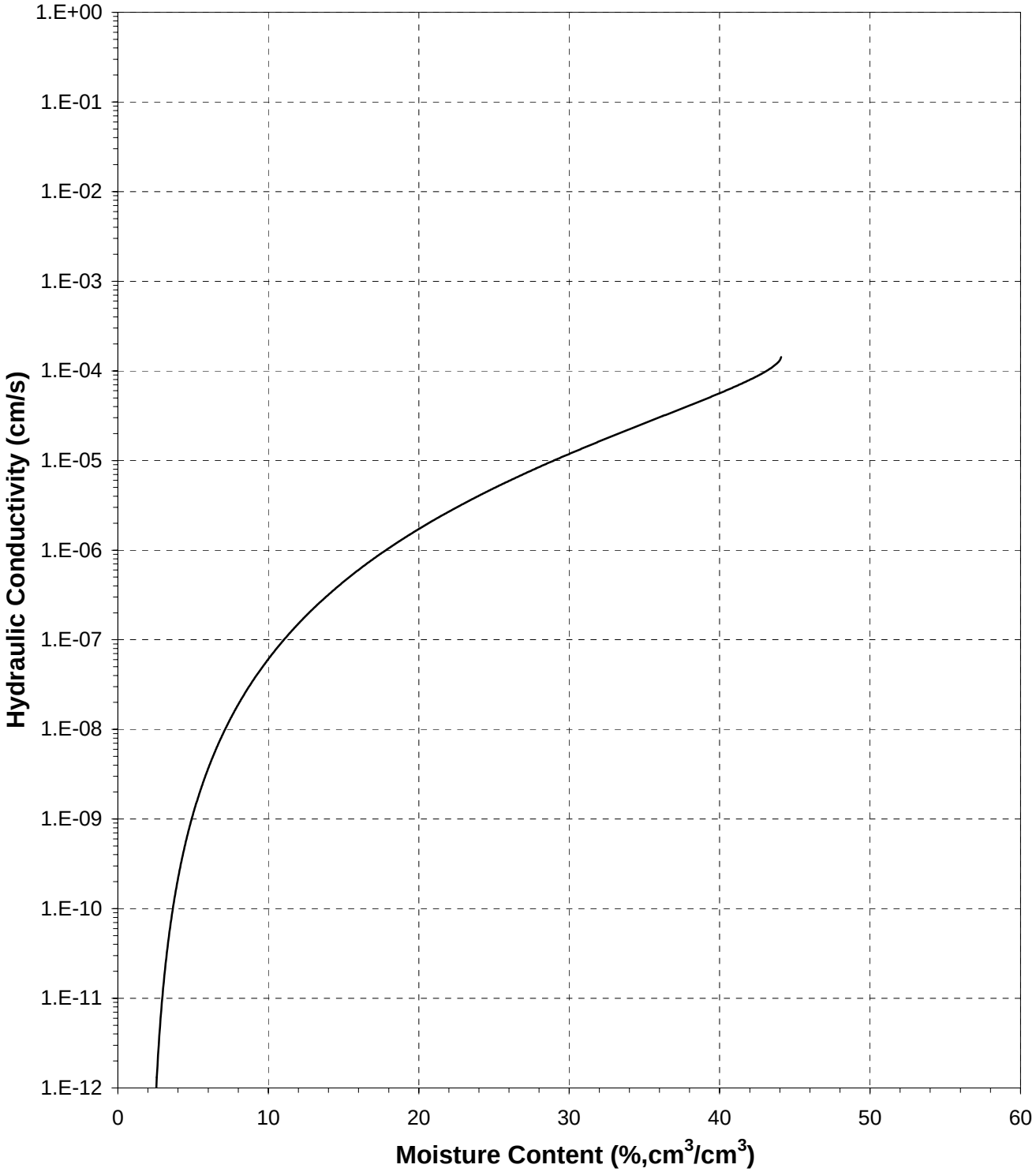
Plot of Relative Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 1 (45%)



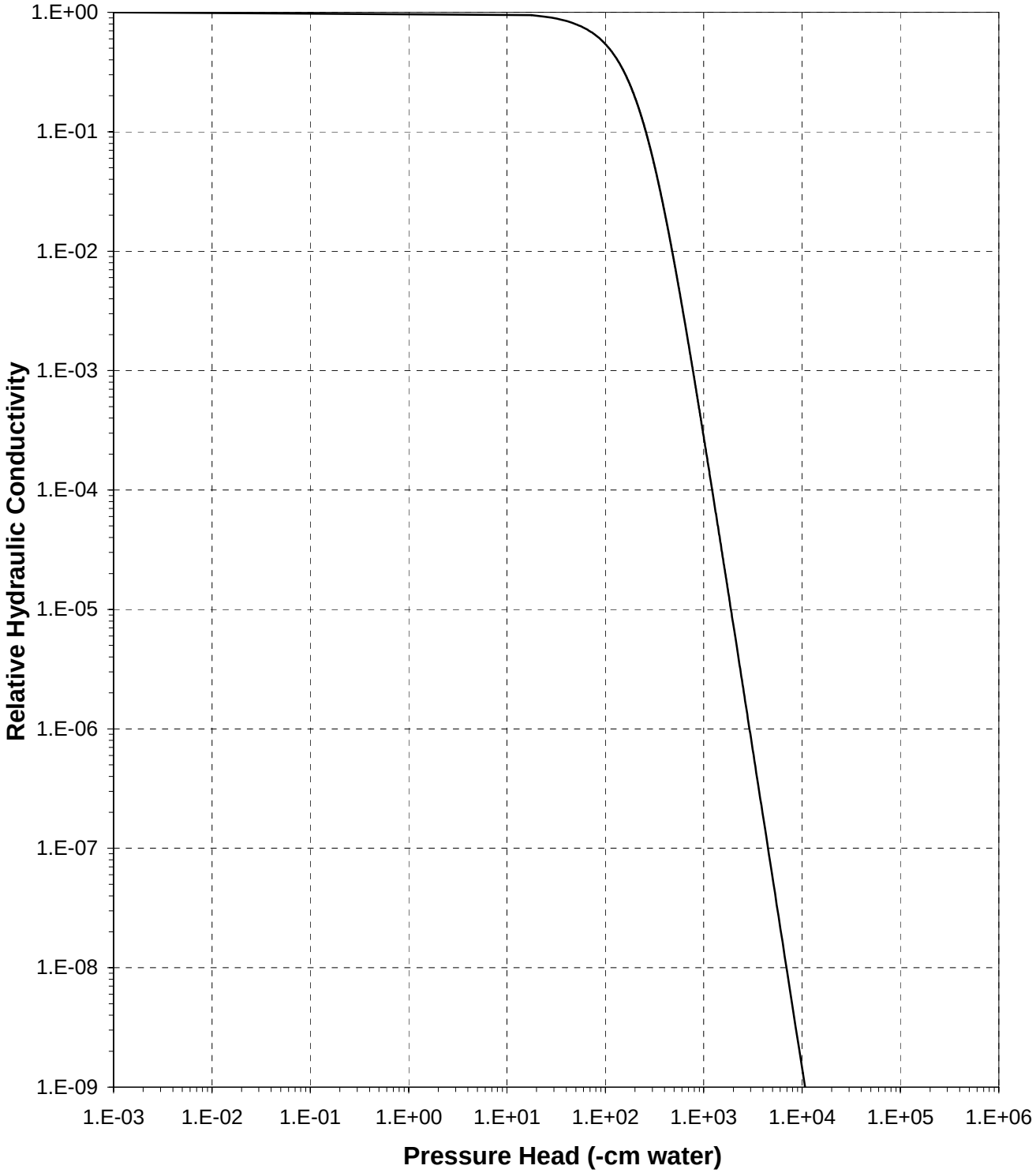
Plot of Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 1 (45%)



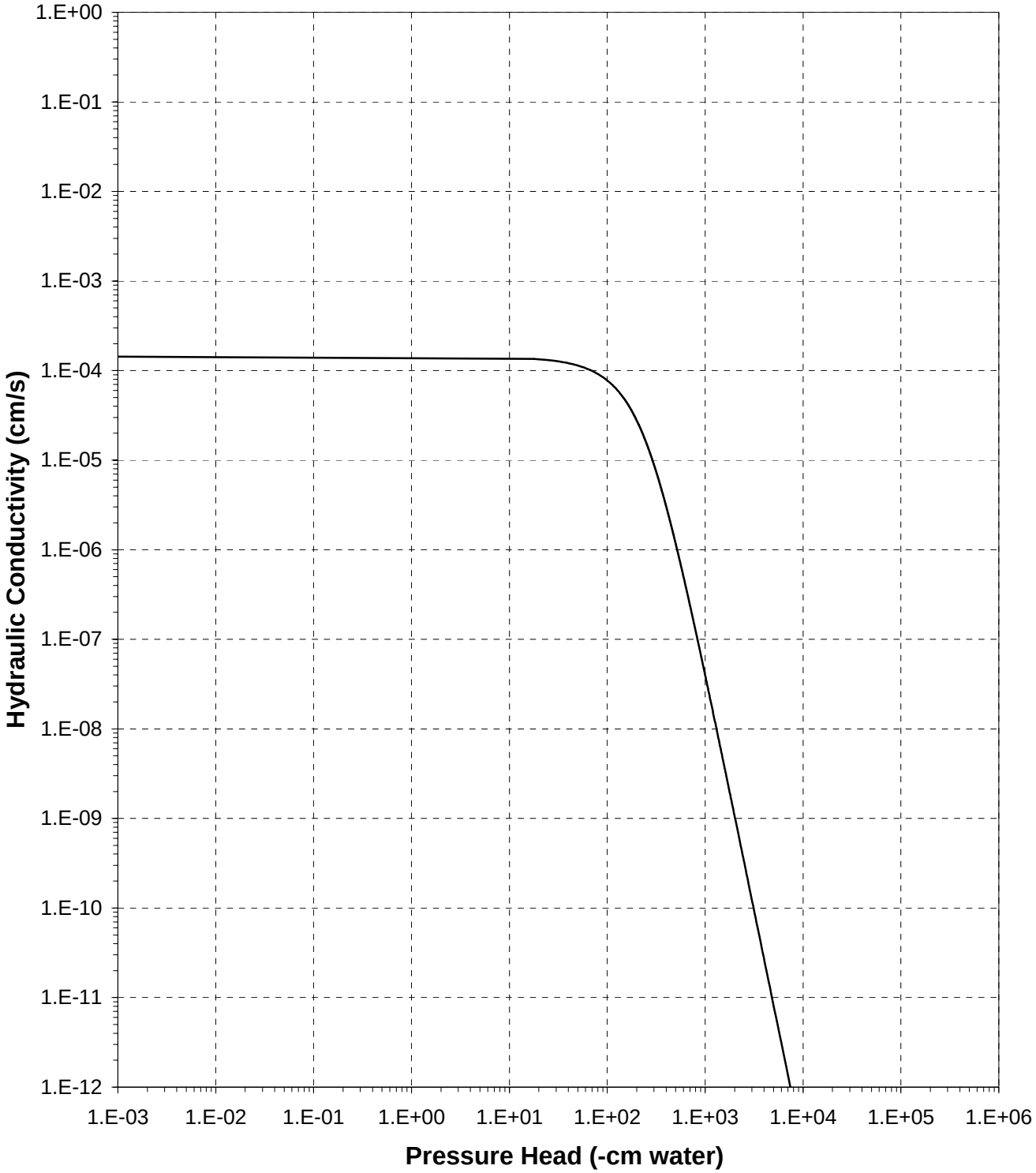
Plot of Relative Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 1 (45%)



Plot of Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 1 (45%)



Moisture Retention Data

Hanging Column / Pressure Plate

(Soil-Water Characteristic Curve)

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 1 (55%)
Ring Number: NA
Depth: NA

Dry wt. of sample (g)*:* 185.91
Tare wt., ring (g)*:* 85.21
Tare wt., screen & clamp (g)*:* 27.76
Initial sample volume (cm^3)*:* 140.40
Initial dry bulk density (g/cm^3)*:* 1.32
Measured particle density (g/cm^3)*:* 2.99
Initial calculated total porosity (%)*:* 55.69

	Date	Time	Weight* (g)	Matric Potential (-cm water)	Moisture Content† (% vol)	
Hanging column:	7-Sep-10	10:50	354.83	0	48.03	‡‡
	13-Sep-10	10:25	341.64	12.0	40.83	‡‡
	19-Sep-10	10:30	338.56	31.0	38.62	‡‡
	26-Sep-10	8:55	336.23	101.0	37.36	‡‡
Pressure plate:	4-Oct-10	12:30	332.45	337	33.67	‡‡

Volume Adjusted Data [1]

	Matric Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calculated Porosity (%)
Hanging column:	0.0	116.48	-17.03%	1.60	46.60
	12.0	104.74	-25.40%	1.78	40.61
	31.0	102.75	-26.82%	1.81	39.46
	101.0	99.98	-28.79%	1.86	37.78
Pressure plate:	337	99.70	-28.99%	1.86	37.60

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent each of the volume change measurements obtained after saturated hydraulic conductivity testing and throughout hanging column/pressure plate testing. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

† Assumed density of water is 1.0 g/cm^3

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Technician Notes:

Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Moisture Retention Data

Dew Point Potentiometer / Relative Humidity Box

(Soil-Water Characteristic Curve)

Sample Number: Blend 1 (55%)

Initial sample bulk density (g/cm^3)*:* 1.32
Fraction of bulk sample used (<2.00mm fraction) (%): 100.00

Dry weight of dew point potentiometer sample* (g)*:* 146.36
Tare weight, jar (g)*:* 114.27

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)	
Dew point potentiometer:	23-Sep-10	14:49	147.09	17745	4.24	‡‡
	23-Sep-10	14:21	146.80	44769	2.56	‡‡
	23-Sep-10	14:08	146.69	109323	1.92	‡‡

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Dew point potentiometer:	17745	99.70	-28.99%	1.86	37.60
	44769	99.70	-28.99%	1.86	37.60
	109323	99.70	-28.99%	1.86	37.60

Dry weight of relative humidity box sample* (g)*:* 71.35
Tare weight (g)*:* 40.00

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)	
Relative humidity box:	16-Sep-10	10:45	71.47	861325	0.73	‡‡

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Relative humidity box:	861325	99.70	-28.99%	1.86	37.60

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent the volume change measurements obtained after the last hanging column or pressure plate point. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

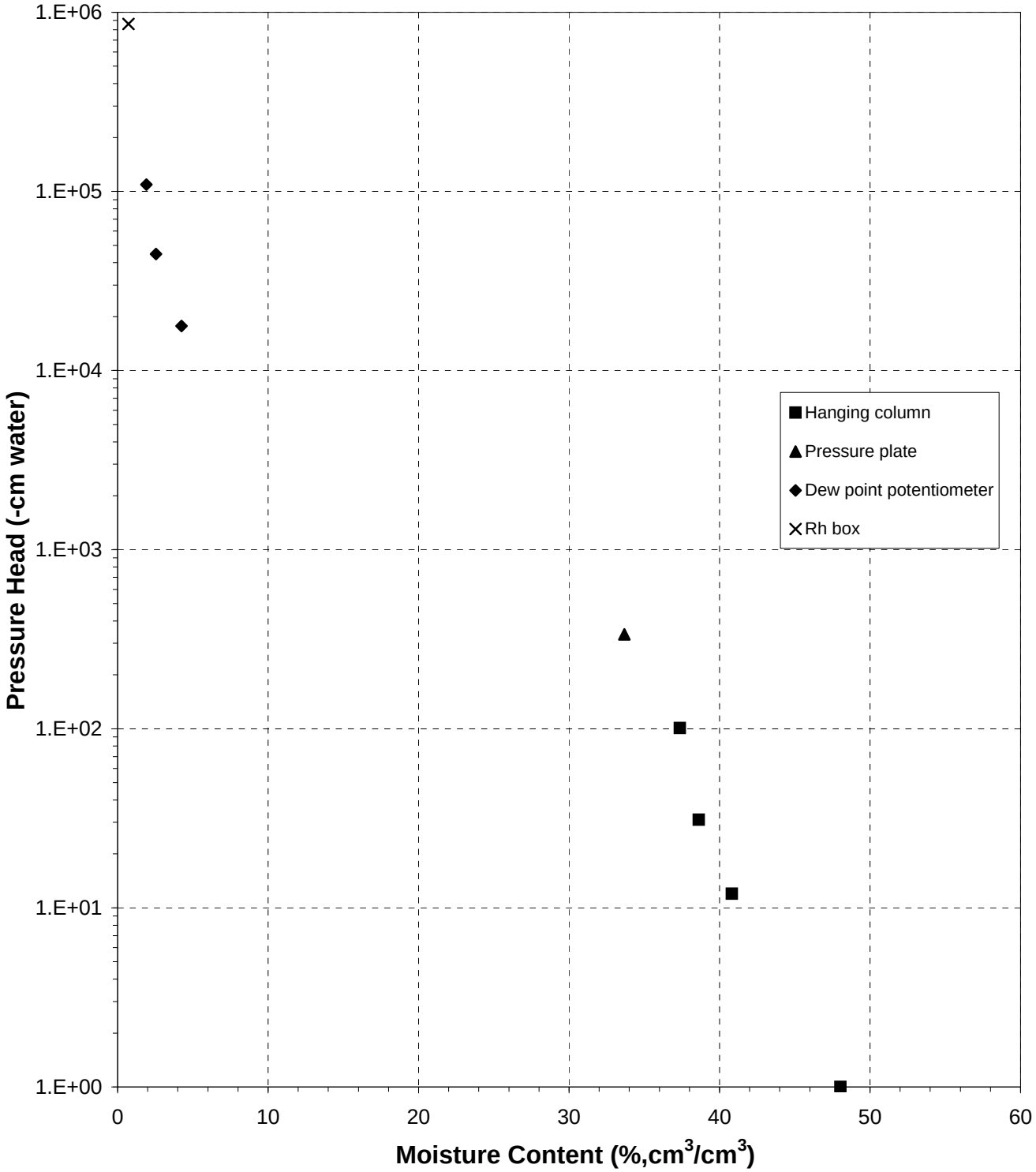
† Adjusted for >2.00mm (#10 sieve) material not used in DPP/RH testing. Assumed moisture content of material >2.00mm is zero, and assumed density of water is 1.0 g/cm^3.

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Laboratory analysis by: M. Vigil
Data entered by: K. Wright
Checked by: J. Hines

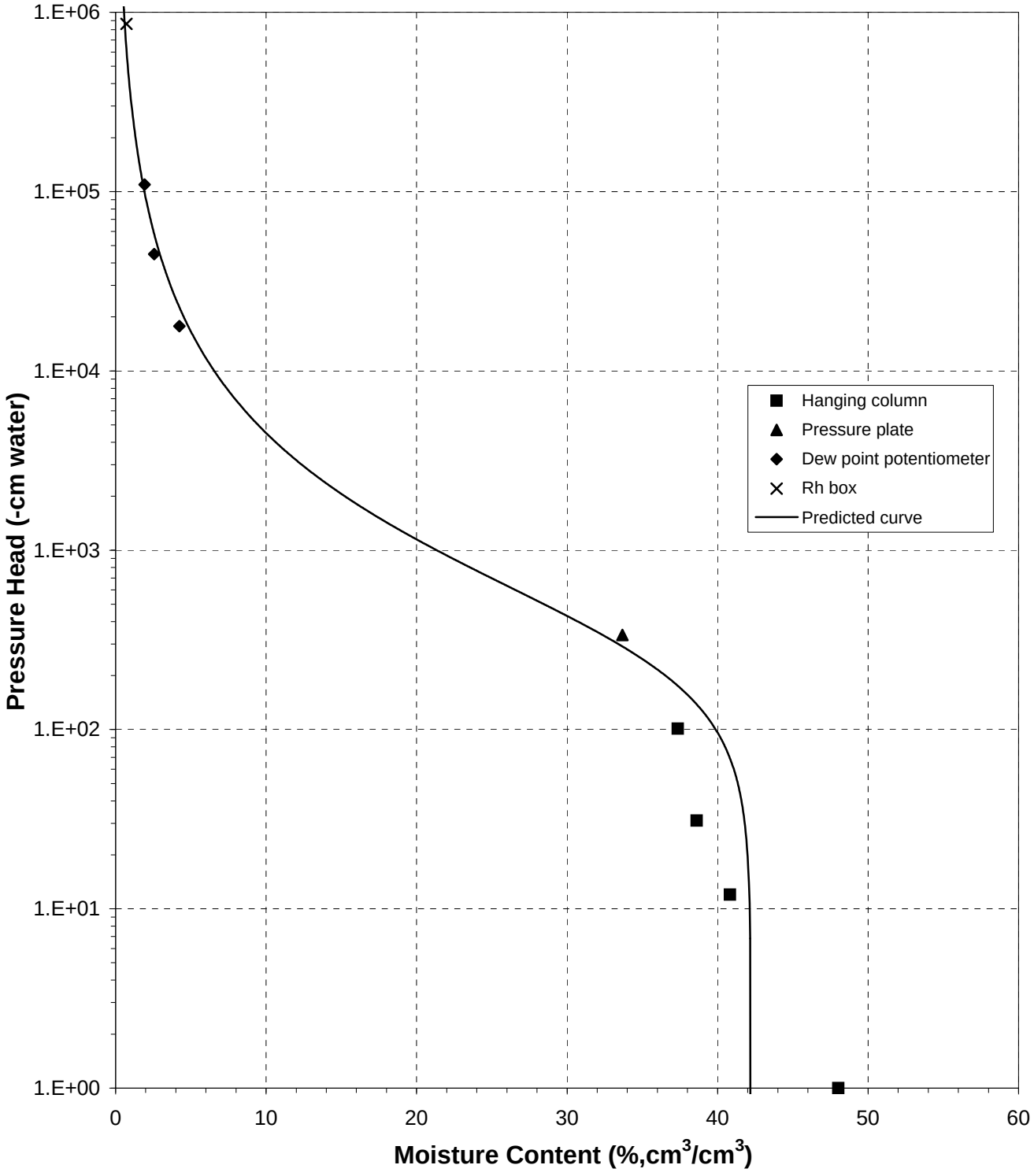
Water Retention Data Points

Sample Number: Blend 1 (55%)



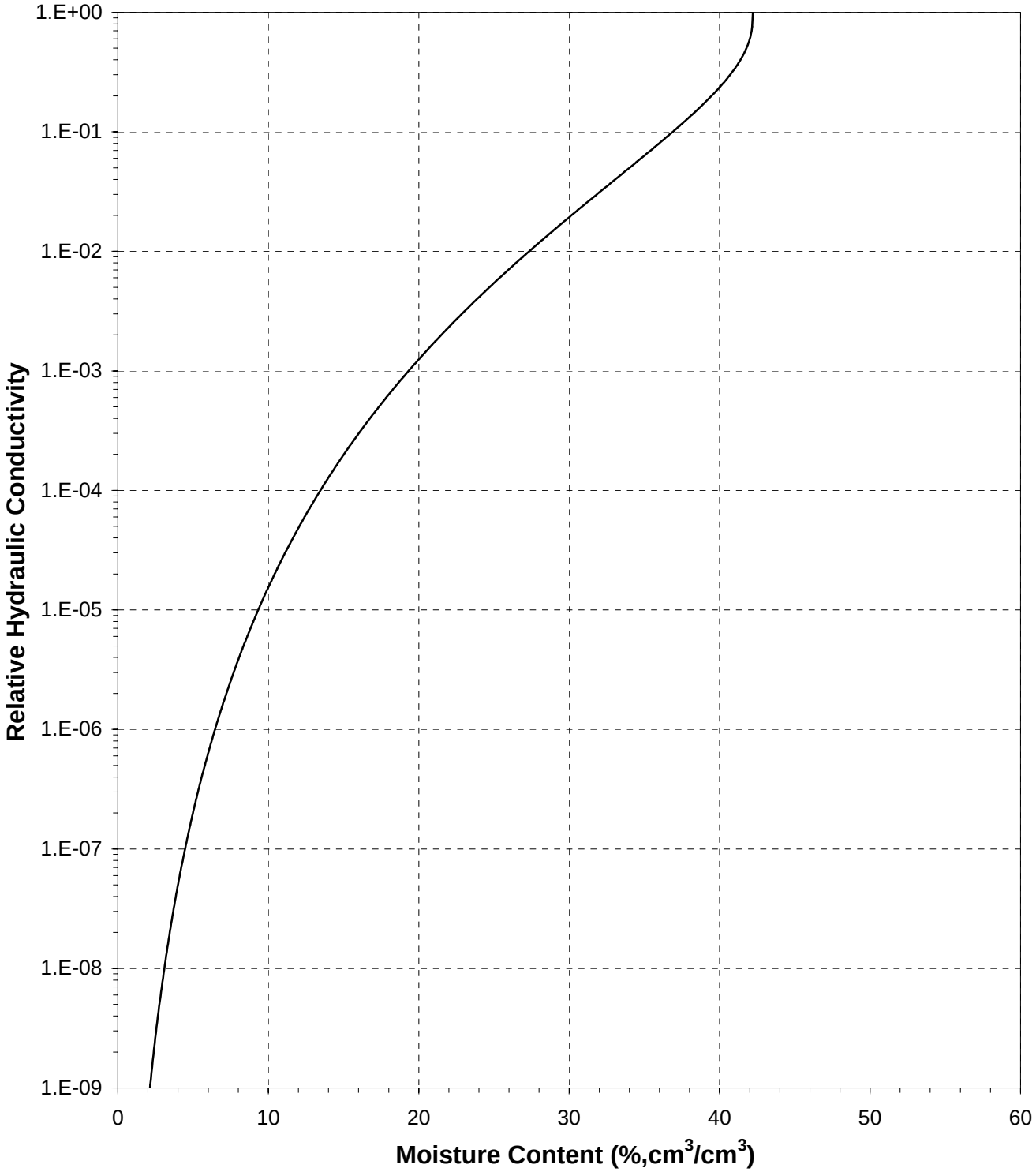
Predicted Water Retention Curve and Data Points

Sample Number: Blend 1 (55%)



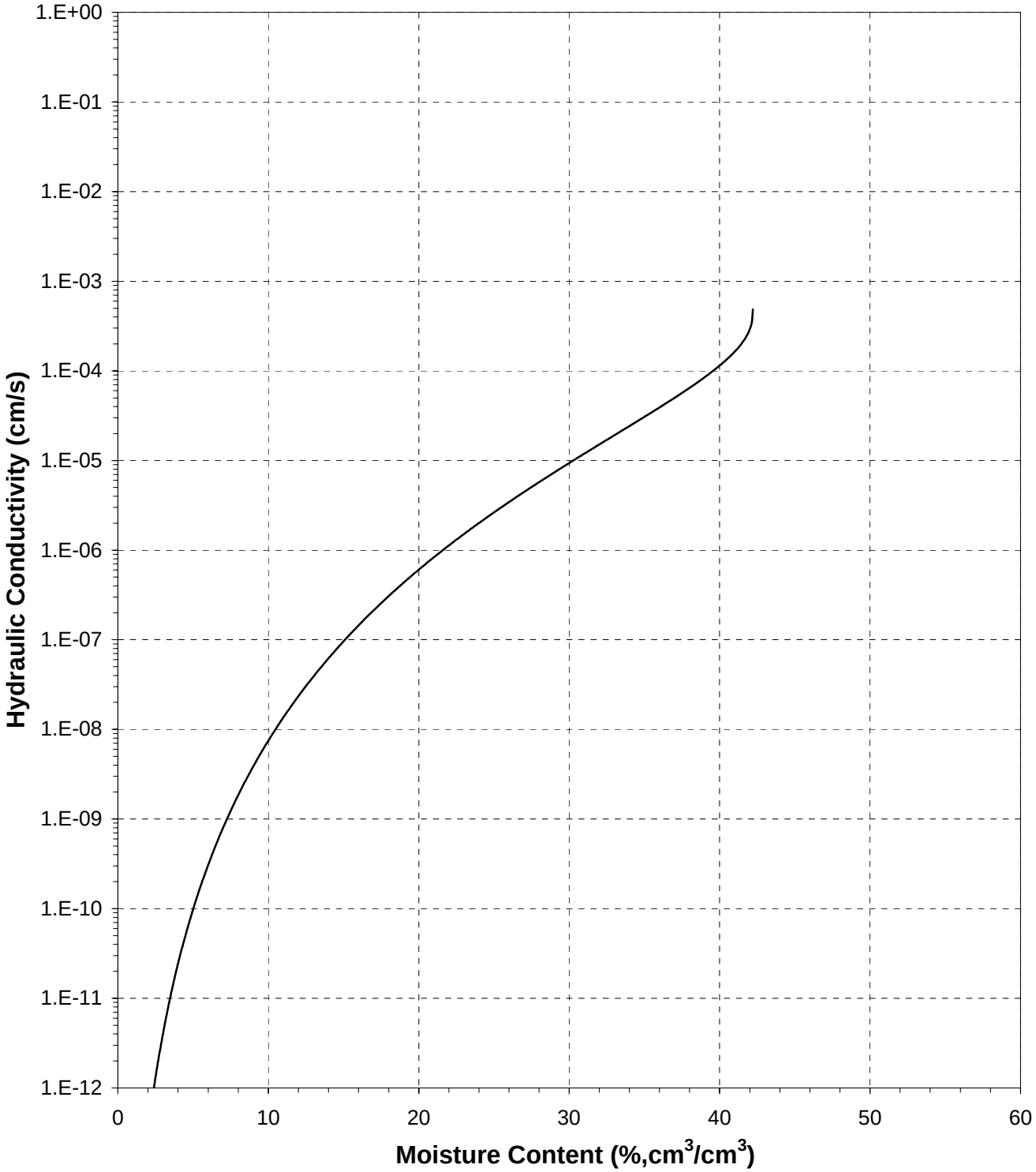
Plot of Relative Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 1 (55%)



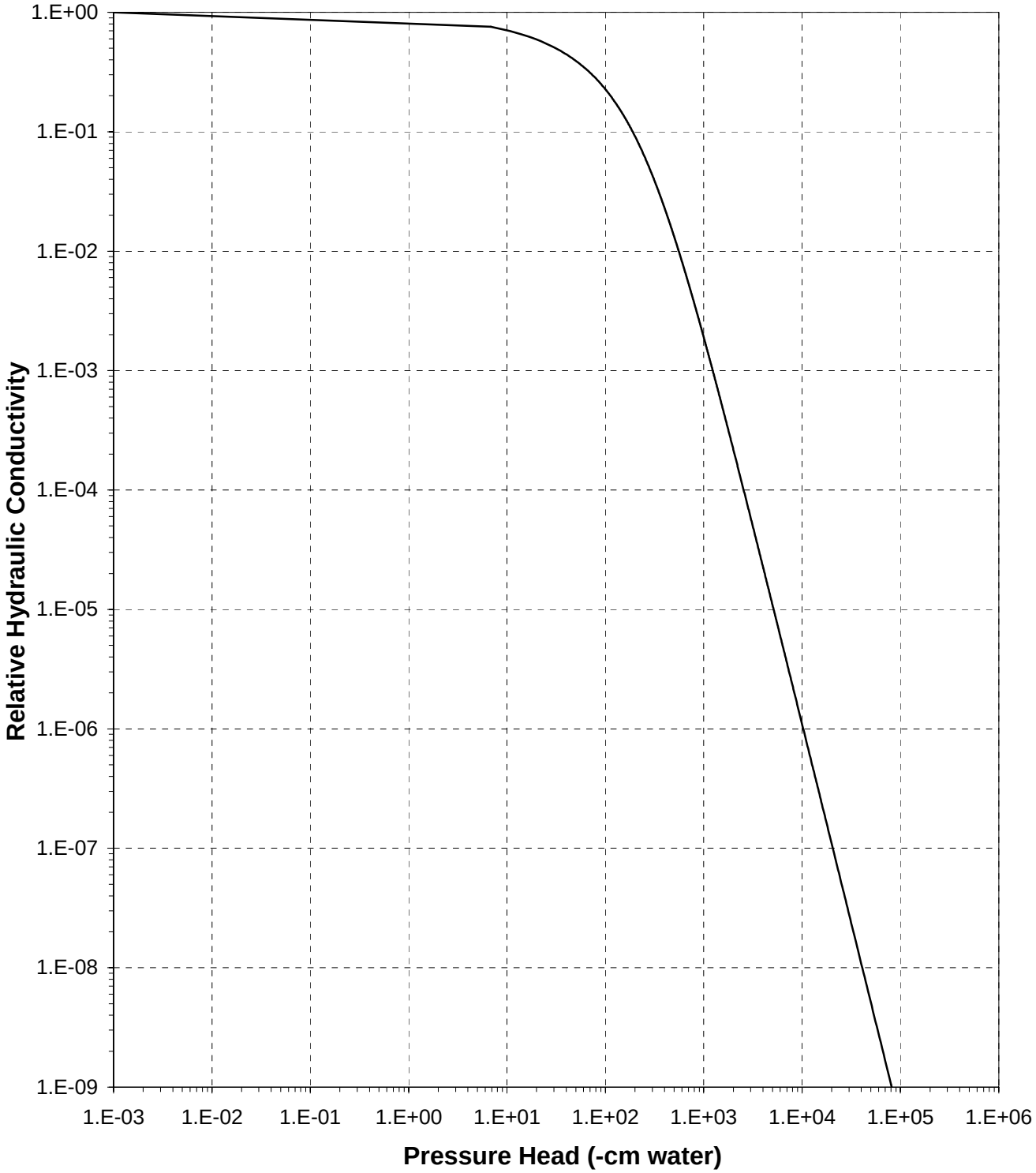
Plot of Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 1 (55%)



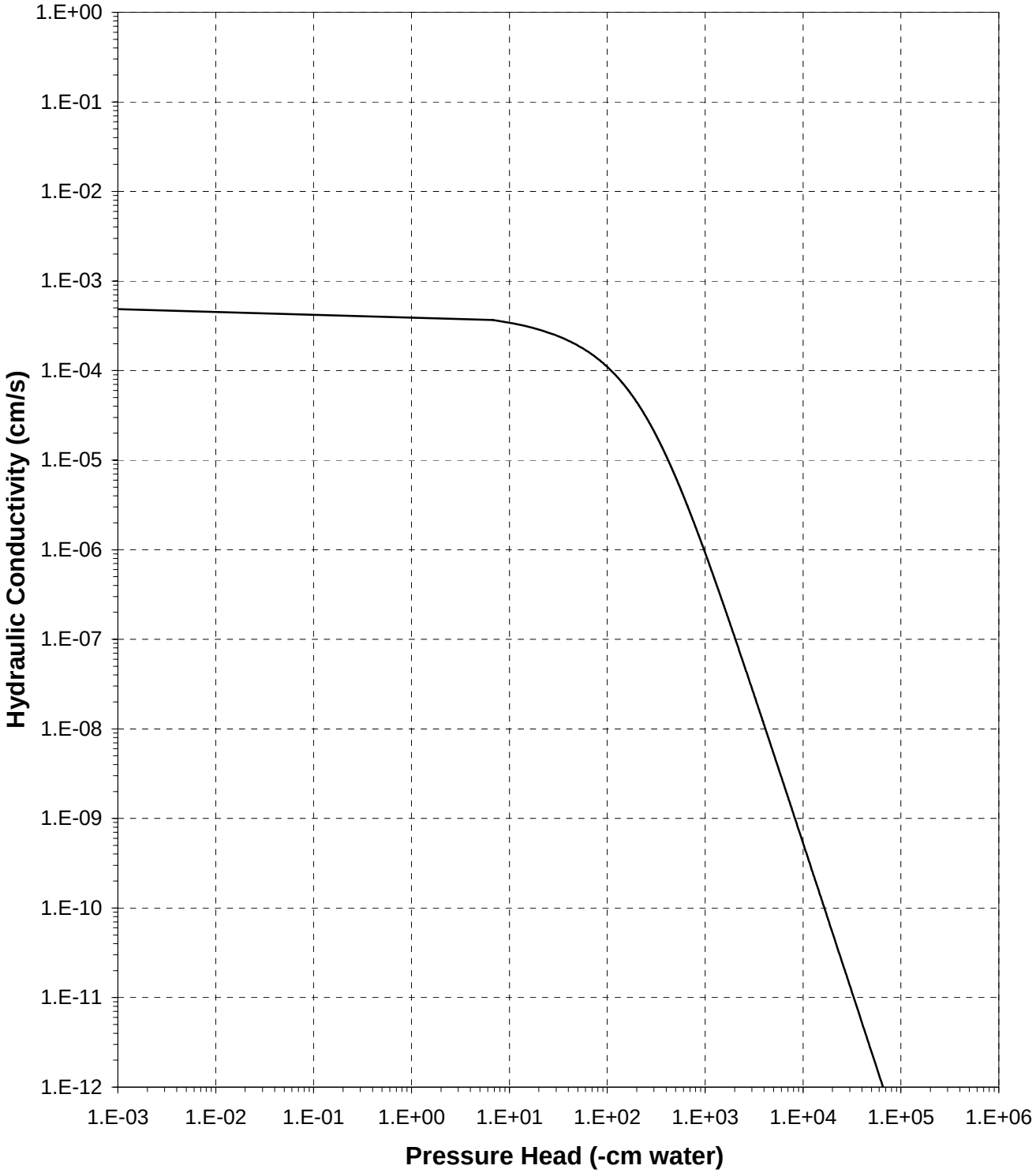
Plot of Relative Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 1 (55%)



Plot of Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 1 (55%)



Moisture Retention Data

Hanging Column / Pressure Plate

(Soil-Water Characteristic Curve)

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 2 (39%)
Ring Number: NA
Depth: NA

Dry wt. of sample (g)*:* 264.75
Tare wt., ring (g)*:* 89.02
Tare wt., screen & clamp (g)*:* 25.01
Initial sample volume (cm^3)*:* 147.03
Initial dry bulk density (g/cm^3)*:* 1.80
Measured particle density (g/cm^3)*:* 2.98
Initial calculated total porosity (%)*:* 39.65

	Date	Time	Weight* (g)	Matric Potential (-cm water)	Moisture Content† (% vol)
Hanging column:	7-Sep-10	10:30	437.43	0	39.89
	13-Sep-10	10:20	435.52	12.0	38.59
	19-Sep-10	10:25	434.95	31.0	38.20
	26-Sep-10	8:50	434.51	101.0	37.90
Pressure plate:	4-Oct-10	12:40	403.57	337	16.86

Volume Adjusted Data [1]

	Matric Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calculated Porosity (%)
Hanging column:	0.0	---	---	---	---
	12.0	---	---	---	---
	31.0	---	---	---	---
	101.0	---	---	---	---
Pressure plate:	337	---	---	---	---

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent each of the volume change measurements obtained after saturated hydraulic conductivity testing and throughout hanging column/pressure plate testing. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

† Assumed density of water is 1.0 g/cm^3

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Technician Notes:

Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Moisture Retention Data

Dew Point Potentiometer / Relative Humidity Box

(Soil-Water Characteristic Curve)

Sample Number: Blend 2 (39%)

Initial sample bulk density (g/cm^3)*:* 1.80
Fraction of bulk sample used (<2.00mm fraction) (%): 100.00

Dry weight of dew point potentiometer sample* (g)*:* 151.18
Tare weight, jar (g)*:* 118.41

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content† (% vol)
Dew point potentiometer:	24-Sep-10	10:15	151.64	16419	2.53
	24-Sep-10	9:57	151.51	35489	1.81
	23-Sep-10	9:32	151.41	142772	1.26

Volume Adjusted Data[1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change[2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Dew point potentiometer:	16419	---	---	---	---
	35489	---	---	---	---
	142772	---	---	---	---

Dry weight of relative humidity box sample* (g)*:* 74.86
Tare weight (g)*:* 41.72

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content† (% vol)
Relative humidity box:	16-Sep-10	10:45	74.97	861325	0.59

Volume Adjusted Data[1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change[2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Relative humidity box:	861325	---	---	---	---

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent the volume change measurements obtained after the last hanging column or pressure plate point. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

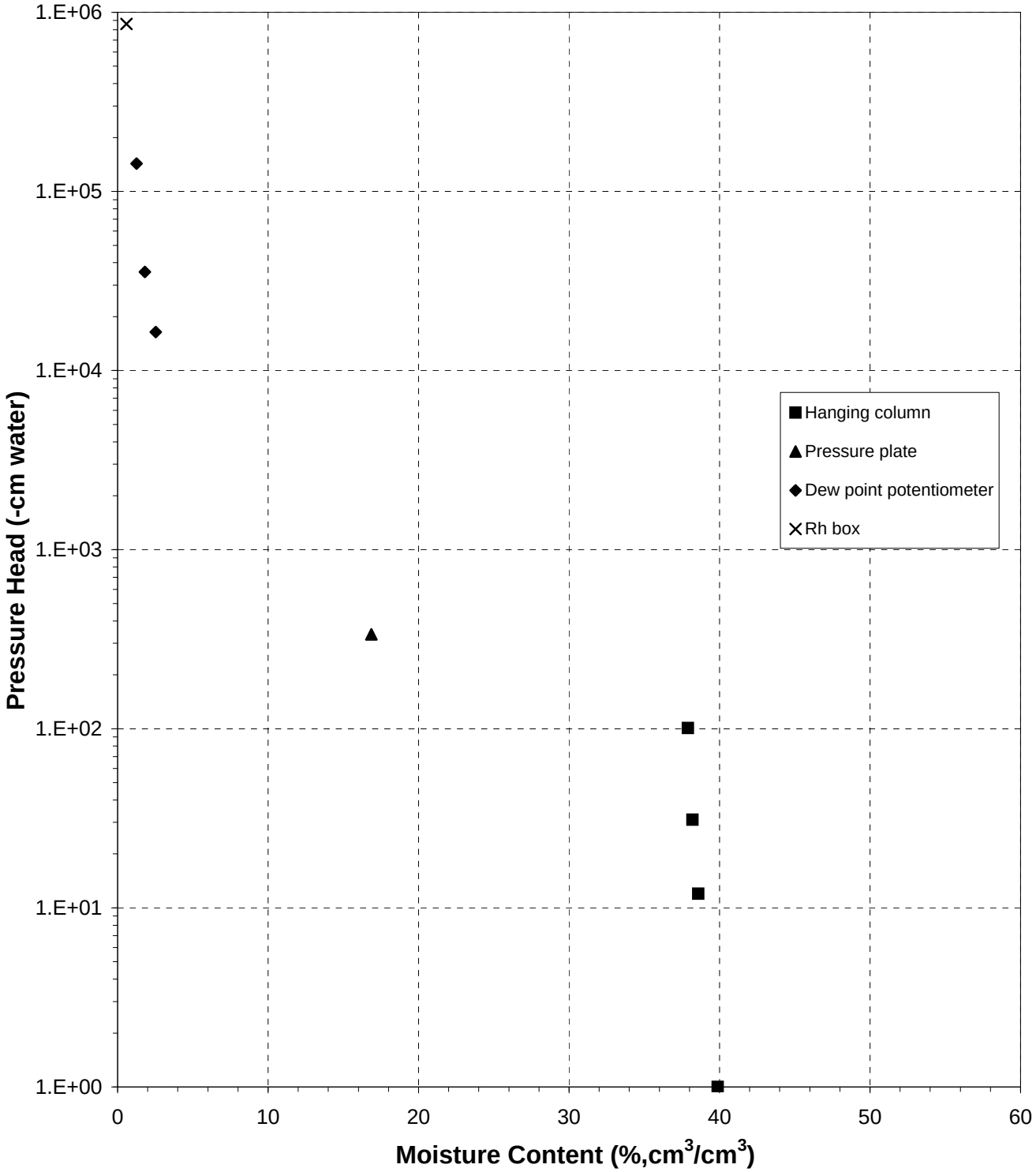
† Adjusted for >2.00mm (#10 sieve) material not used in DPP/RH testing. Assumed moisture content of material >2.00mm is zero, and assumed density of water is 1.0 g/cm^3.

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Laboratory analysis by: M. Vigil
Data entered by: K. Wright
Checked by: J. Hines

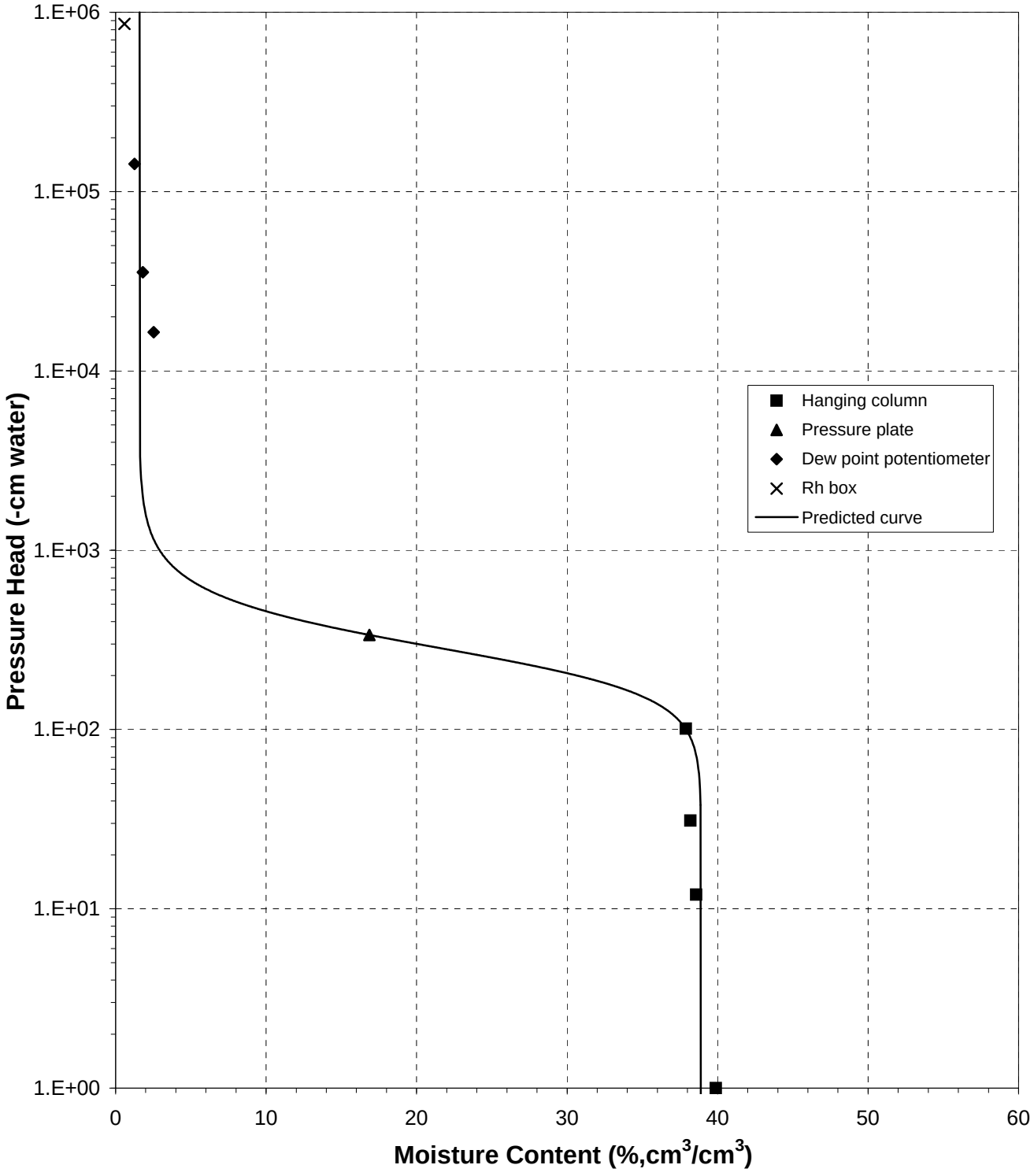
Water Retention Data Points

Sample Number: Blend 2 (39%)



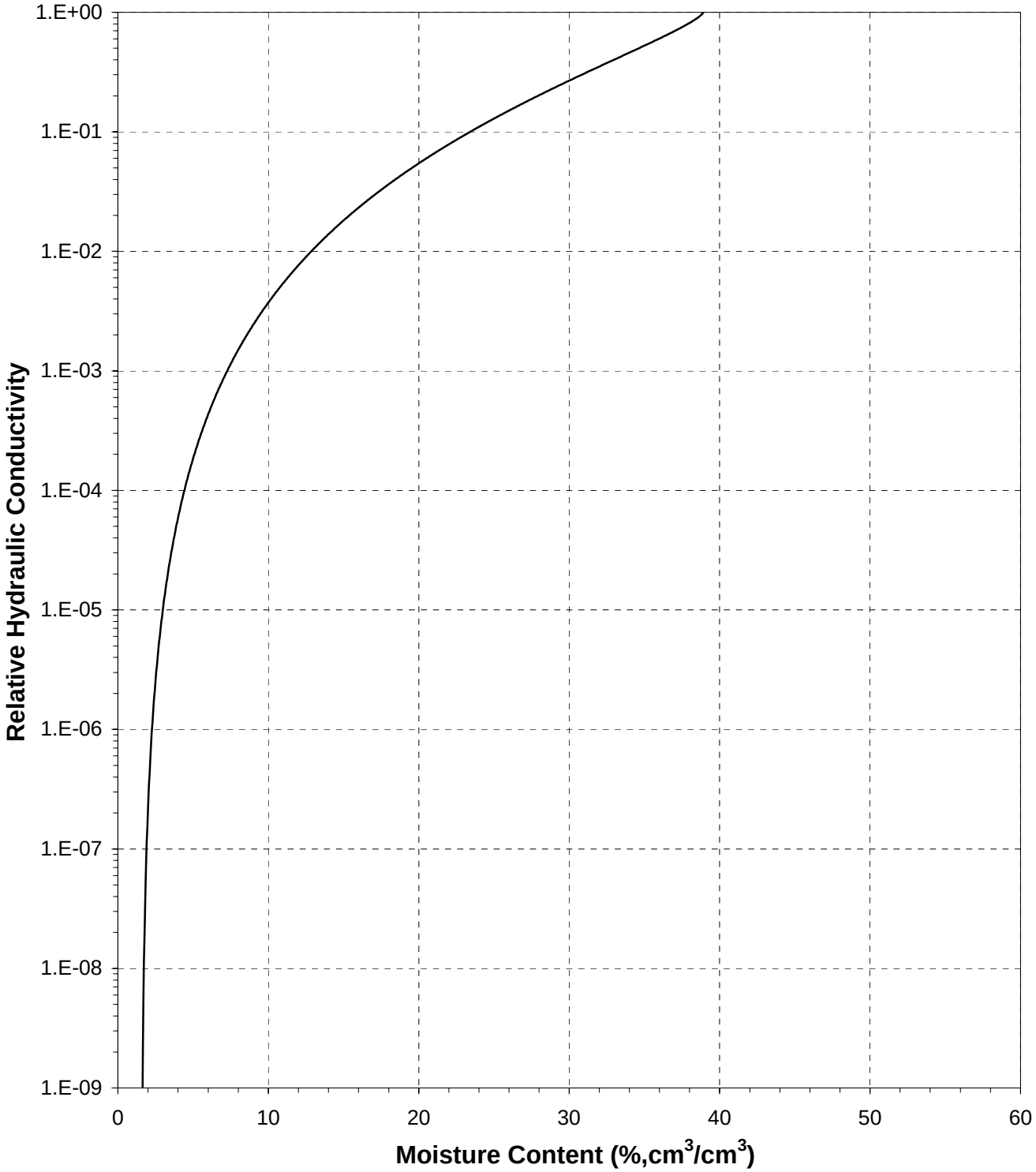
Predicted Water Retention Curve and Data Points

Sample Number: Blend 2 (39%)



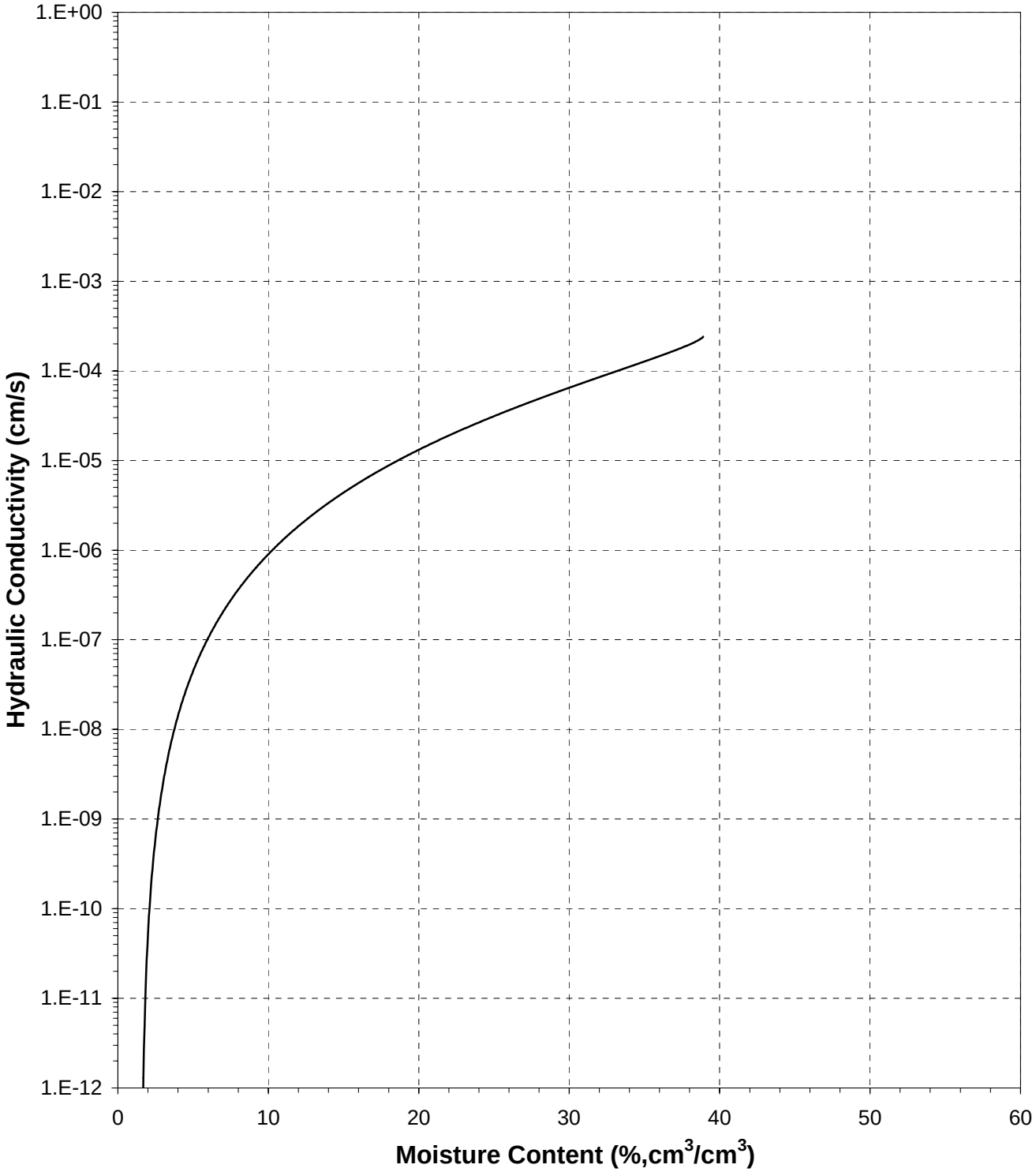
Plot of Relative Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 2 (39%)



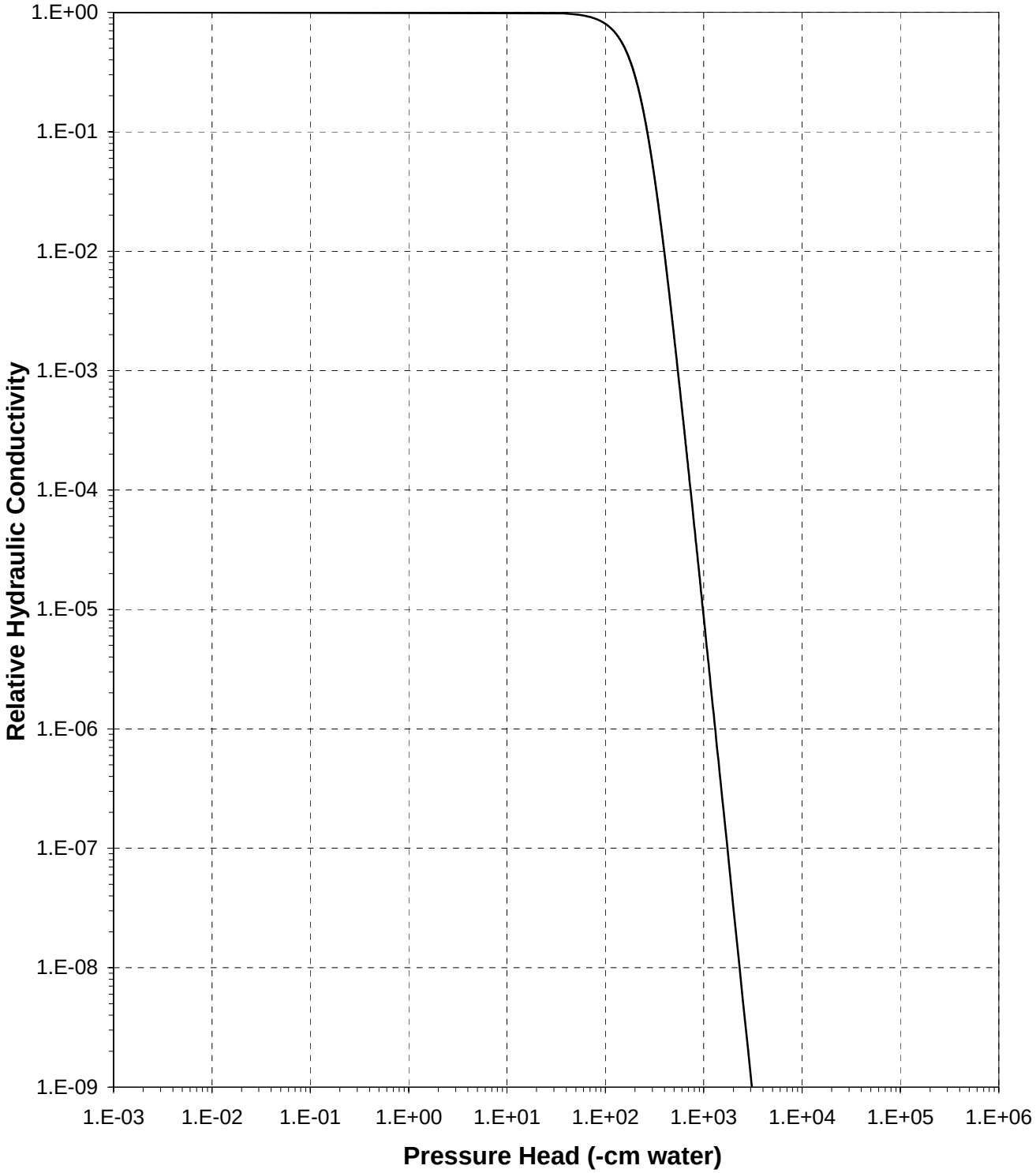
Plot of Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 2 (39%)



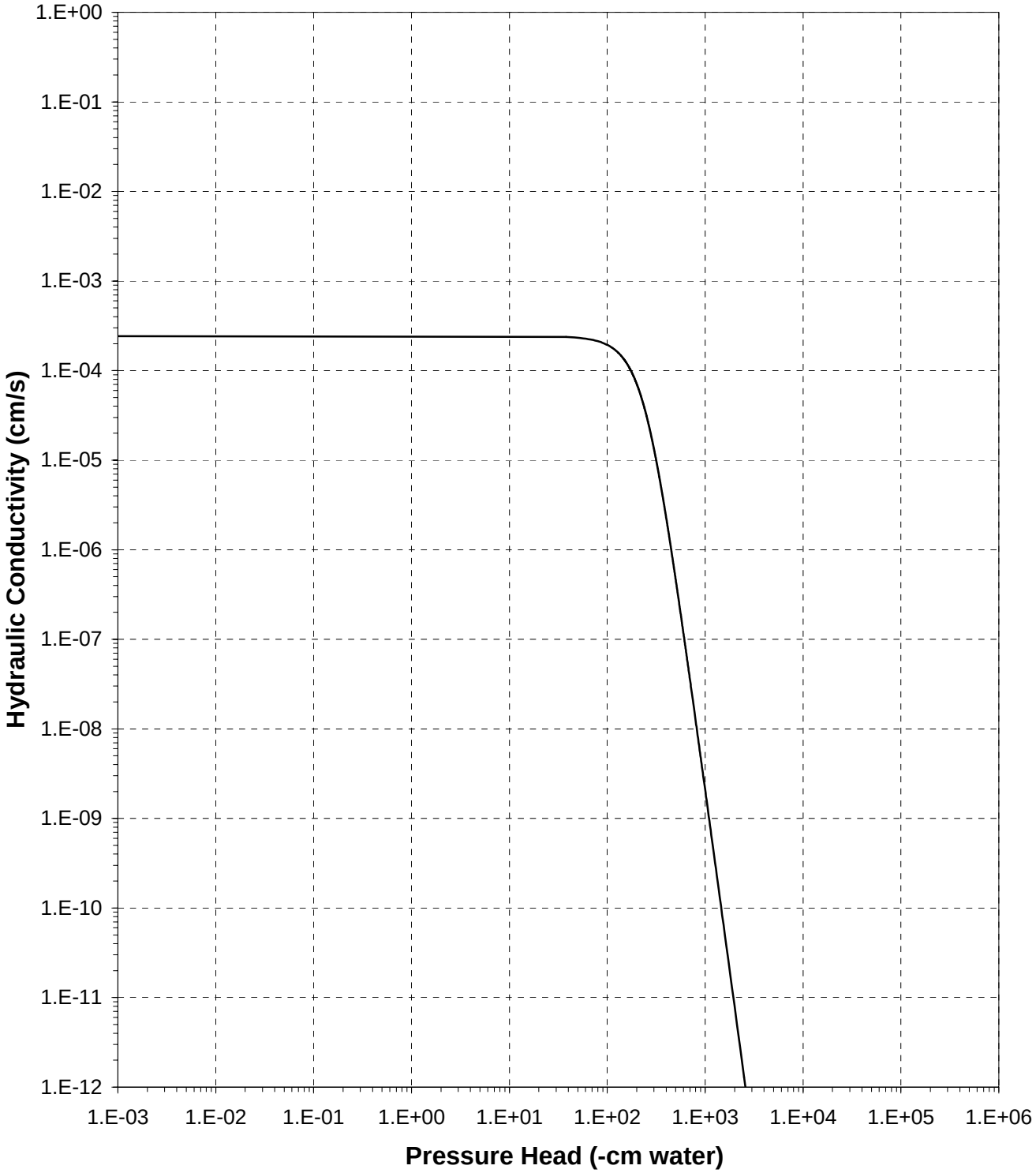
Plot of Relative Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 2 (39%)



Plot of Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 2 (39%)



Moisture Retention Data

Hanging Column / Pressure Plate

(Soil-Water Characteristic Curve)

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 2 (45%)
Ring Number: NA
Depth: NA

Dry wt. of sample (g)*:* 228.69
Tare wt., ring (g)*:* 110.15
Tare wt., screen & clamp (g)*:* 25.25
Initial sample volume (cm^3)*:* 139.97
Initial dry bulk density (g/cm^3)*:* 1.63
Measured particle density (g/cm^3)*:* 2.98
Initial calculated total porosity (%)*:* 45.24

	Date	Time	Weight* (g)	Matric Potential (-cm water)	Moisture Content† (% vol)	
Hanging column:	7-Sep-10	10:25	426.02	0	45.78	‡‡
	13-Sep-10	10:20	420.90	12.0	43.48	‡‡
	19-Sep-10	10:20	418.04	31.0	41.70	‡‡
	26-Sep-10	8:45	415.34	101.0	40.15	‡‡
Pressure plate:	4-Oct-10	12:35	388.83	337	19.41	‡‡

Volume Adjusted Data [1]

	Matric Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calculated Porosity (%)
Hanging column:	0.0	135.26	-3.36%	1.69	43.33
	12.0	130.65	-6.66%	1.75	41.33
	31.0	129.36	-7.58%	1.77	40.75
	101.0	127.64	-8.81%	1.79	39.95
Pressure plate:	337	127.49	-8.91%	1.79	39.88

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent each of the volume change measurements obtained after saturated hydraulic conductivity testing and throughout hanging column/pressure plate testing. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

† Assumed density of water is 1.0 g/cm^3

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Technician Notes:

Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Moisture Retention Data

Dew Point Potentiometer / Relative Humidity Box

(Soil-Water Characteristic Curve)

Sample Number: Blend 2 (45%)

Initial sample bulk density (g/cm^3)*:* 1.63
Fraction of bulk sample used (<2.00mm fraction) (%): 100.00

Dry weight of dew point potentiometer sample* (g)*:* 152.65
Tare weight, jar (g)*:* 114.03

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)	
Dew point potentiometer:	23-Sep-10	14:43	153.20	14583	2.55	‡‡
	23-Sep-10	14:27	152.97	35489	1.49	‡‡
	23-Sep-10	13:59	152.87	126455	1.02	‡‡

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Dew point potentiometer:	14583	127.49	-8.91%	1.79	39.88
	35489	127.49	-8.91%	1.79	39.88
	126455	127.49	-8.91%	1.79	39.88

Dry weight of relative humidity box sample* (g)*:* 76.30
Tare weight (g)*:* 40.79

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)	
Relative humidity box:	16-Sep-10	10:45	76.42	861325	0.59	‡‡

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Relative humidity box:	861325	127.49	-8.91%	1.79	39.88

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent the volume change measurements obtained after the last hanging column or pressure plate point. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

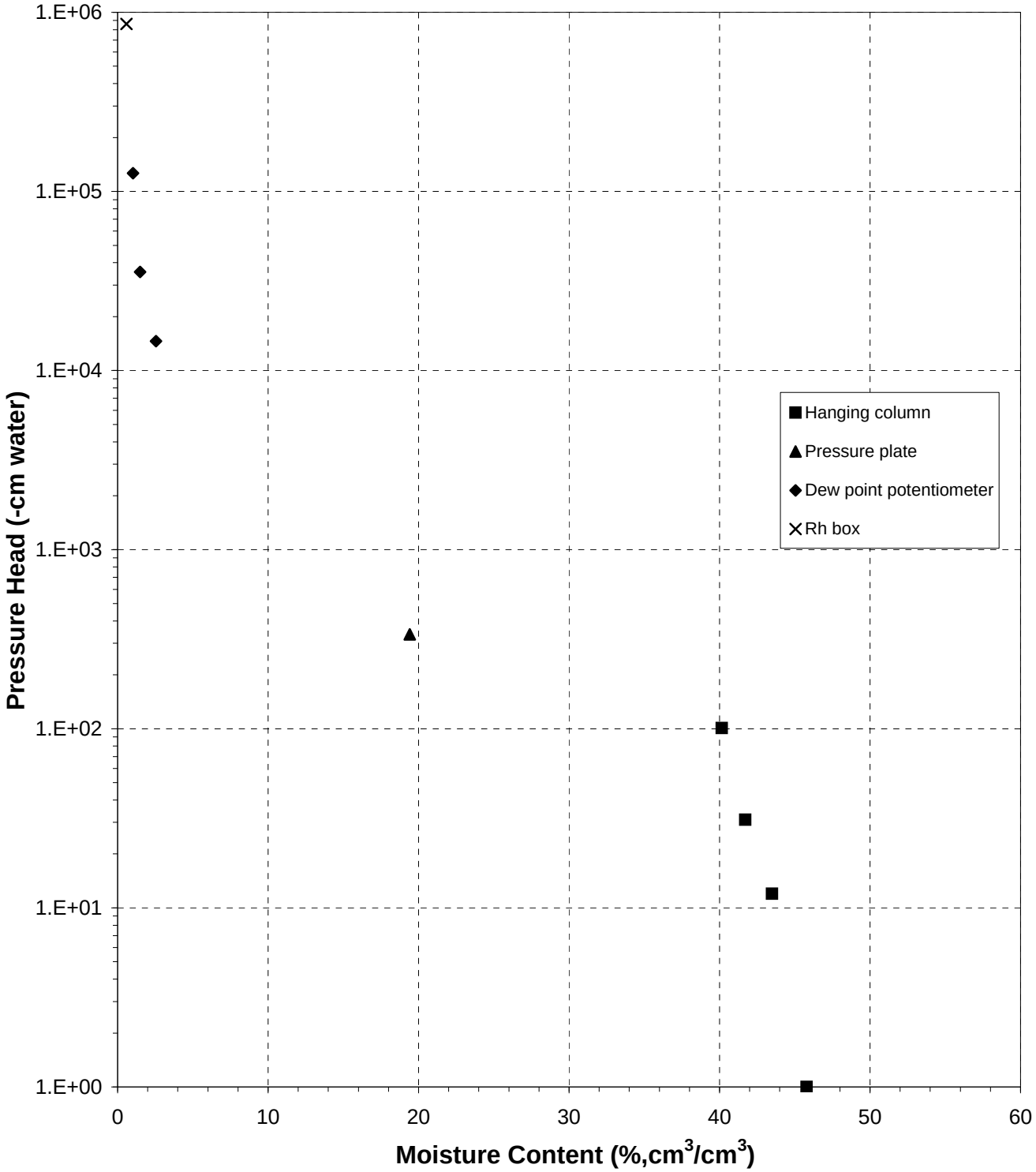
† Adjusted for >2.00mm (#10 sieve) material not used in DPP/RH testing. Assumed moisture content of material >2.00mm is zero, and assumed density of water is 1.0 g/cm^3.

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Laboratory analysis by: M. Vigil
Data entered by: K. Wright
Checked by: J. Hines

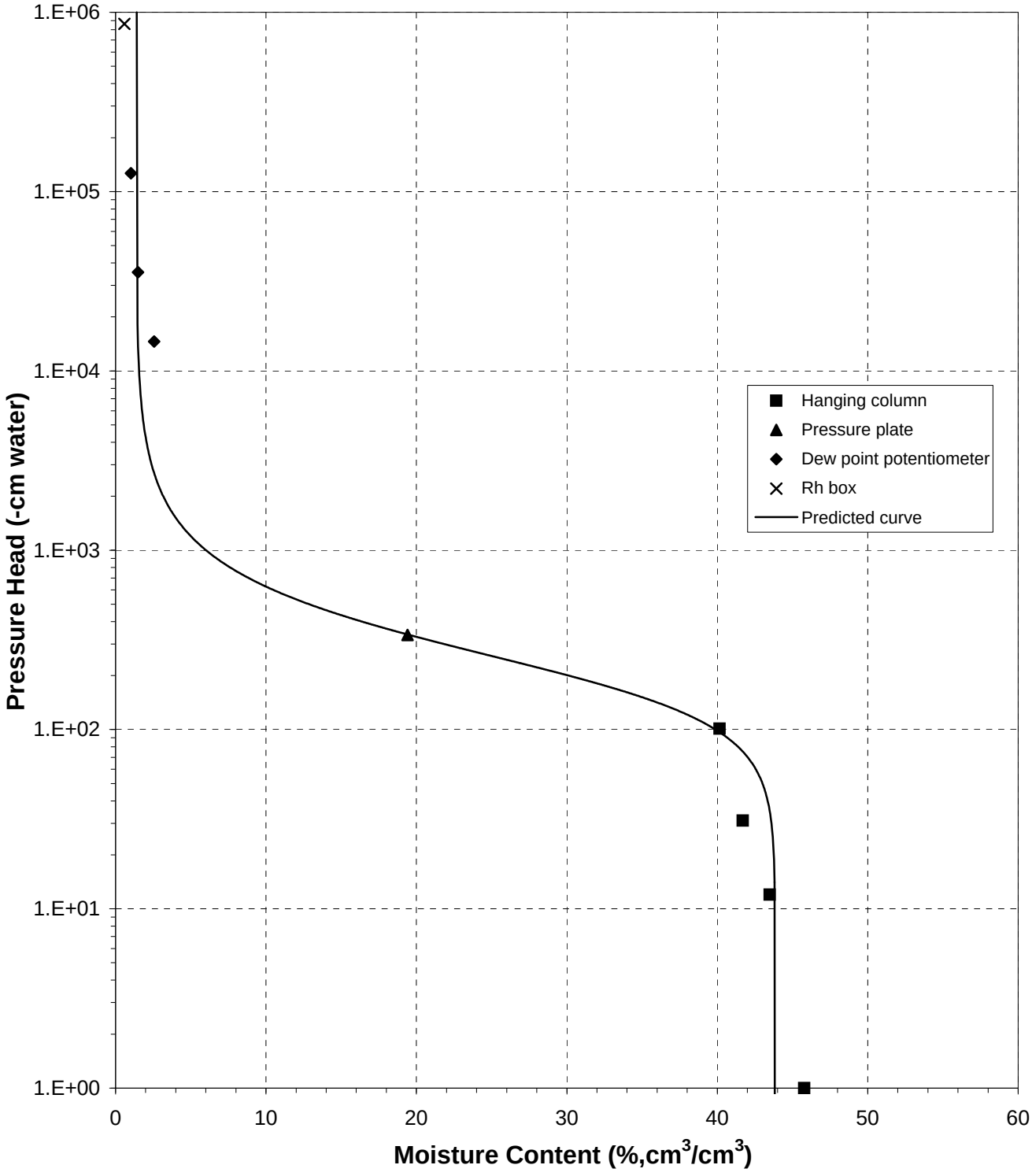
Water Retention Data Points

Sample Number: Blend 2 (45%)



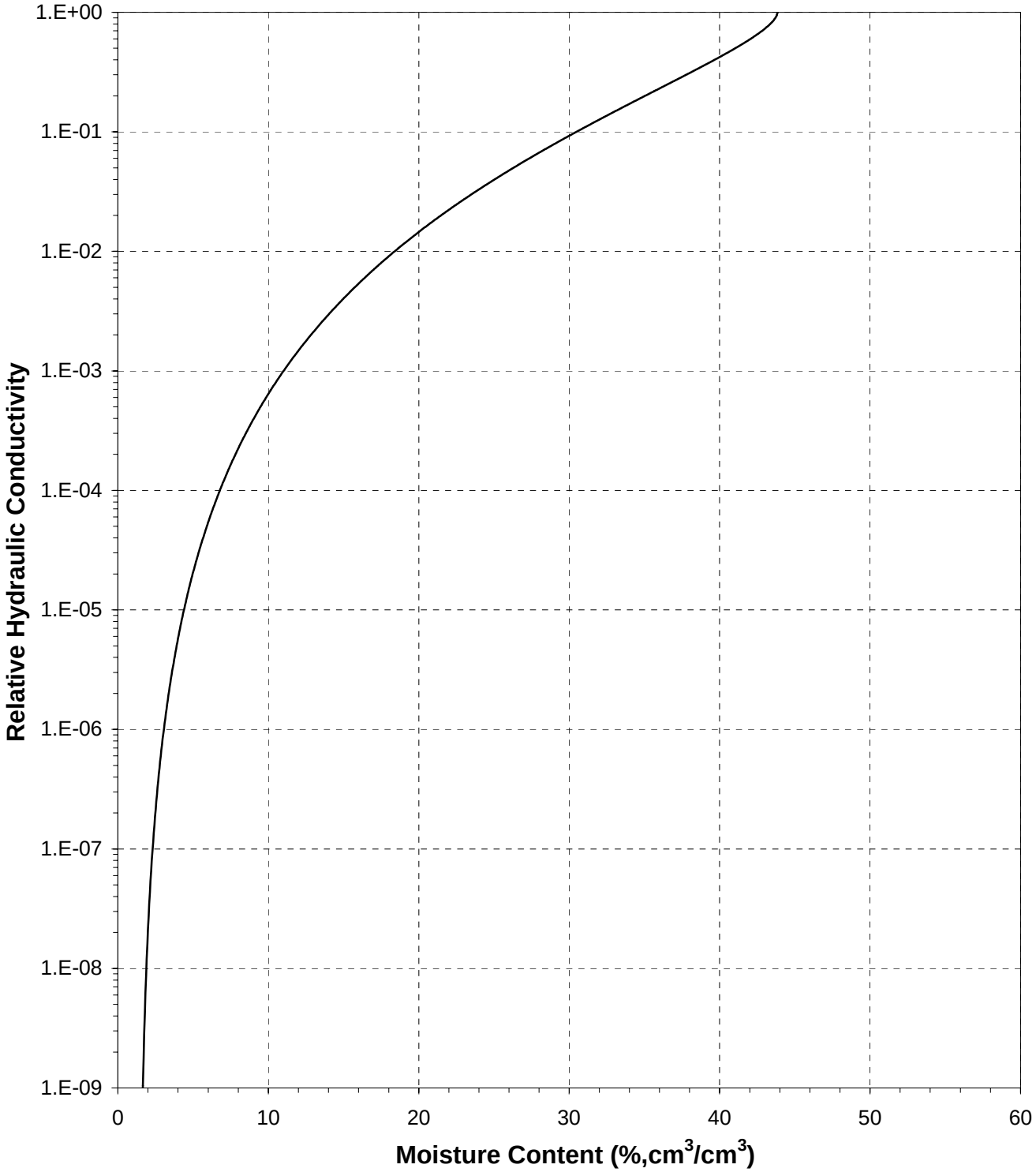
Predicted Water Retention Curve and Data Points

Sample Number: Blend 2 (45%)



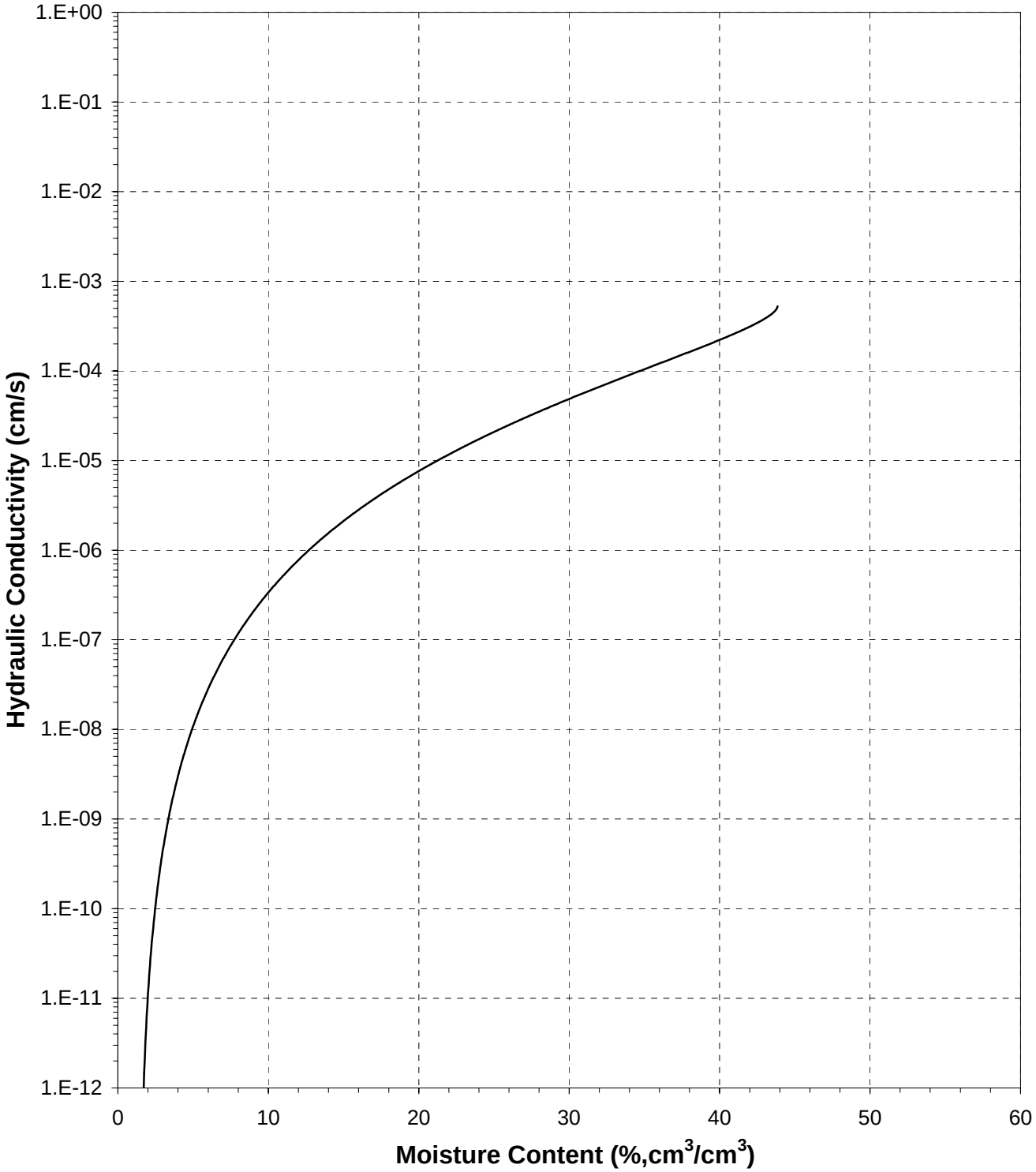
Plot of Relative Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 2 (45%)



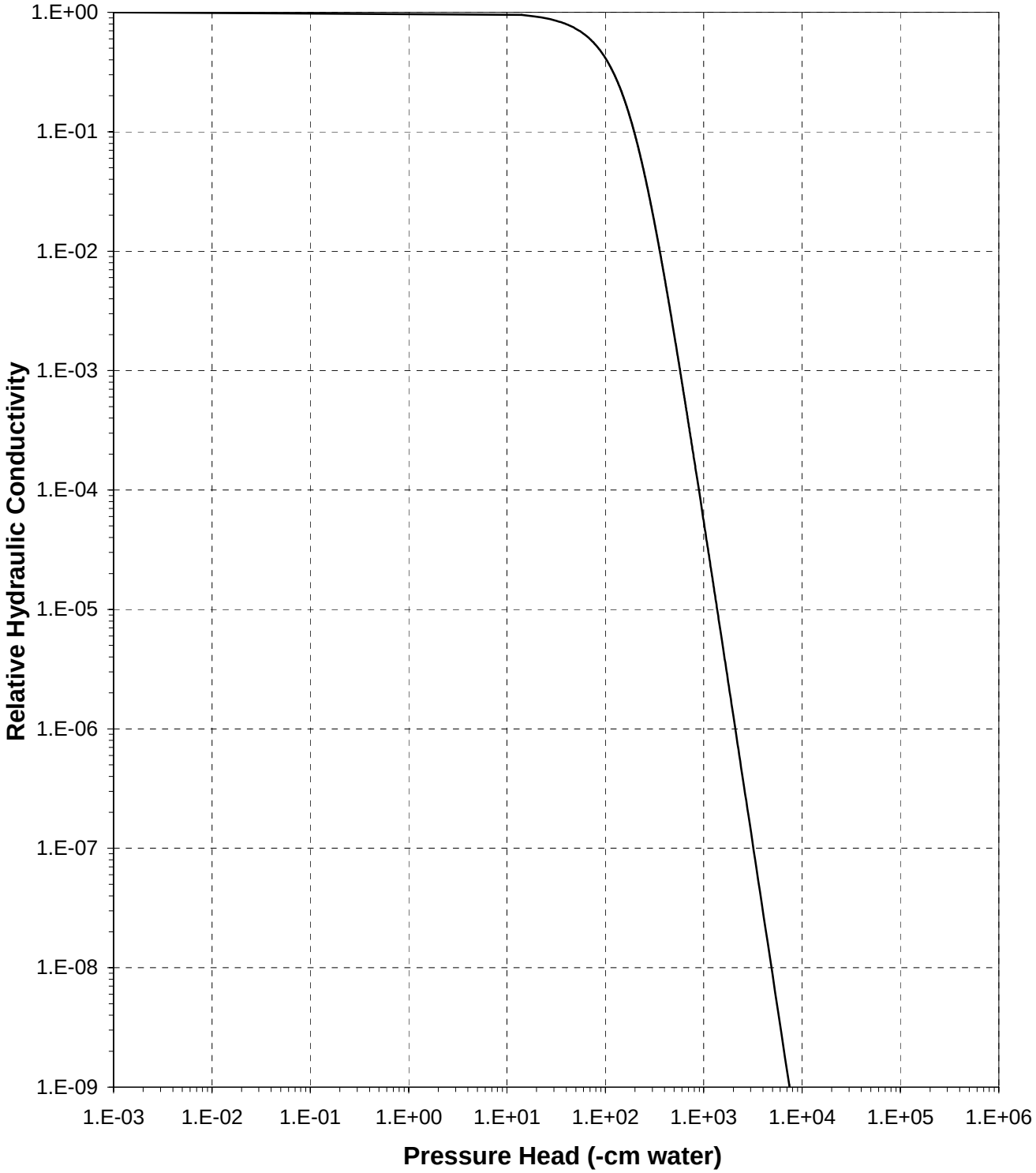
Plot of Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 2 (45%)



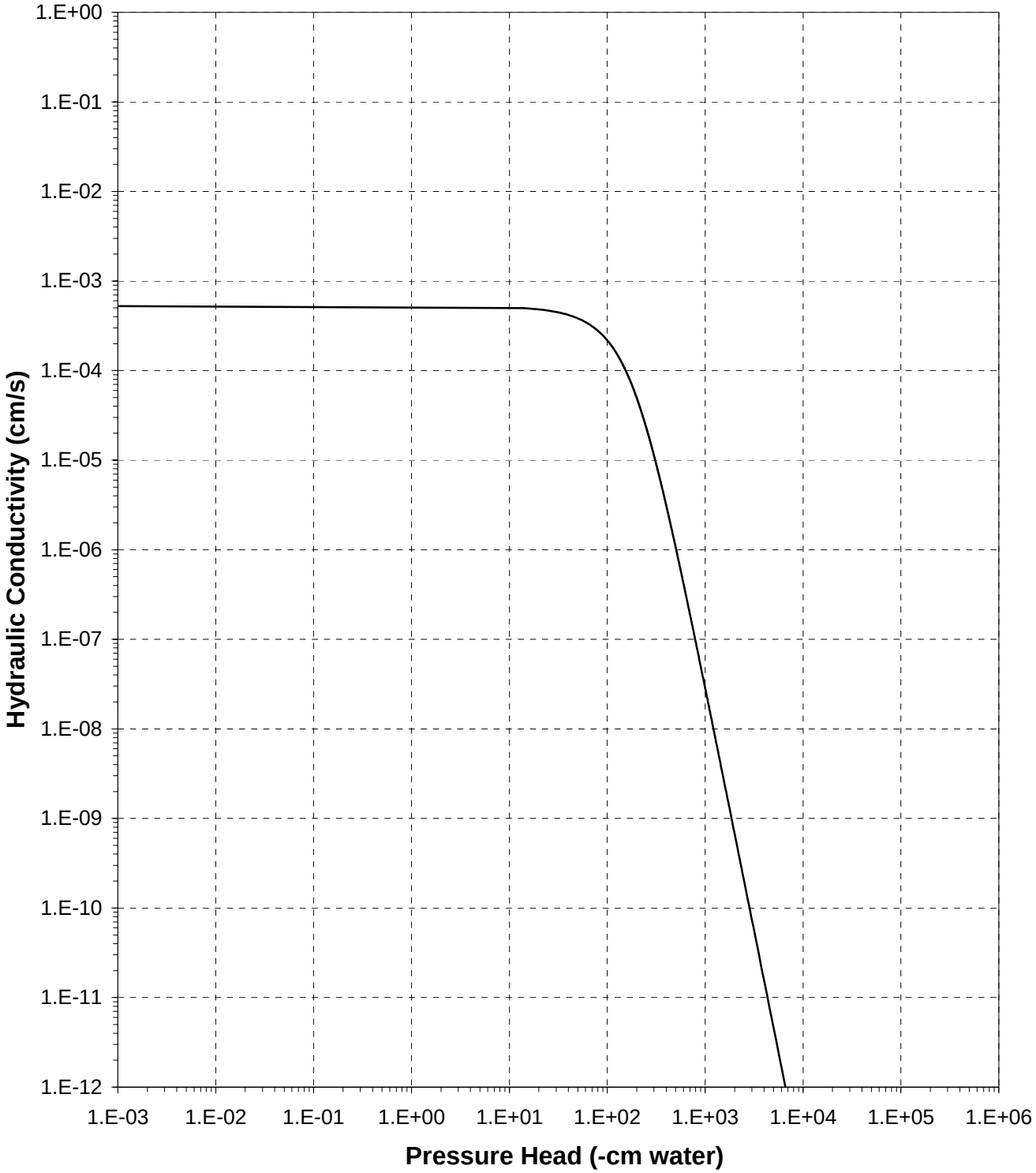
Plot of Relative Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 2 (45%)



Plot of Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 2 (45%)



Moisture Retention Data

Hanging Column / Pressure Plate

(Soil-Water Characteristic Curve)

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 2 (55%)
Ring Number: NA
Depth: NA

Dry wt. of sample (g)*:* 185.98
Tare wt., ring (g)*:* 116.30
Tare wt., screen & clamp (g)*:* 27.98
Initial sample volume (cm^3)*:* 140.88
Initial dry bulk density (g/cm^3)*:* 1.32
Measured particle density (g/cm^3)*:* 2.98
Initial calculated total porosity (%)*:* 55.75

	Date	Time	Weight* (g)	Matric Potential (-cm water)	Moisture Content† (% vol)	
Hanging column:	7-Sep-10	12:00	382.24	0	46.56	‡‡
	13-Sep-10	13:10	373.04	5.0	41.07	‡‡
	19-Sep-10	11:40	368.69	21.0	39.43	‡‡
	26-Sep-10	9:45	366.56	76.0	37.32	‡‡
Pressure plate:	4-Oct-10	13:20	351.25	337	21.61	‡‡

Volume Adjusted Data [1]

	Matric Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calculated Porosity (%)
Hanging column:	0.0	111.65	-20.75%	1.67	44.17
	5.0	104.15	-26.07%	1.79	40.15
	21.0	97.47	-30.81%	1.91	36.05
	76.0	97.27	-30.96%	1.91	35.91
Pressure plate:	337	97.12	-31.06%	1.92	35.82

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent each of the volume change measurements obtained after saturated hydraulic conductivity testing and throughout hanging column/pressure plate testing. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

† Assumed density of water is 1.0 g/cm^3

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Technician Notes:

Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Moisture Retention Data

Dew Point Potentiometer / Relative Humidity Box

(Soil-Water Characteristic Curve)

Sample Number: Blend 2 (55%)

Initial sample bulk density (g/cm^3)*:* 1.32
Fraction of bulk sample used (<2.00mm fraction) (%): 100.00

Dry weight of dew point potentiometer sample* (g)*:* 152.10
Tare weight, jar (g)*:* 117.44

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)	
Dew point potentiometer:	23-Sep-10	12:33	152.63	17541	2.93	‡‡
	23-Sep-10	12:26	152.41	44565	1.71	‡‡
	24-Sep-10	8:55	152.35	106875	1.38	‡‡

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Dew point potentiometer:	17541	97.12	-31.06%	1.92	35.82
	44565	97.12	-31.06%	1.92	35.82
	106875	97.12	-31.06%	1.92	35.82

Dry weight of relative humidity box sample* (g)*:* 69.20
Tare weight (g)*:* 36.88

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)	
Relative humidity box:	16-Sep-10	10:45	69.32	861325	0.71	‡‡

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Relative humidity box:	861325	97.12	-31.06%	1.92	35.82

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent the volume change measurements obtained after the last hanging column or pressure plate point. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

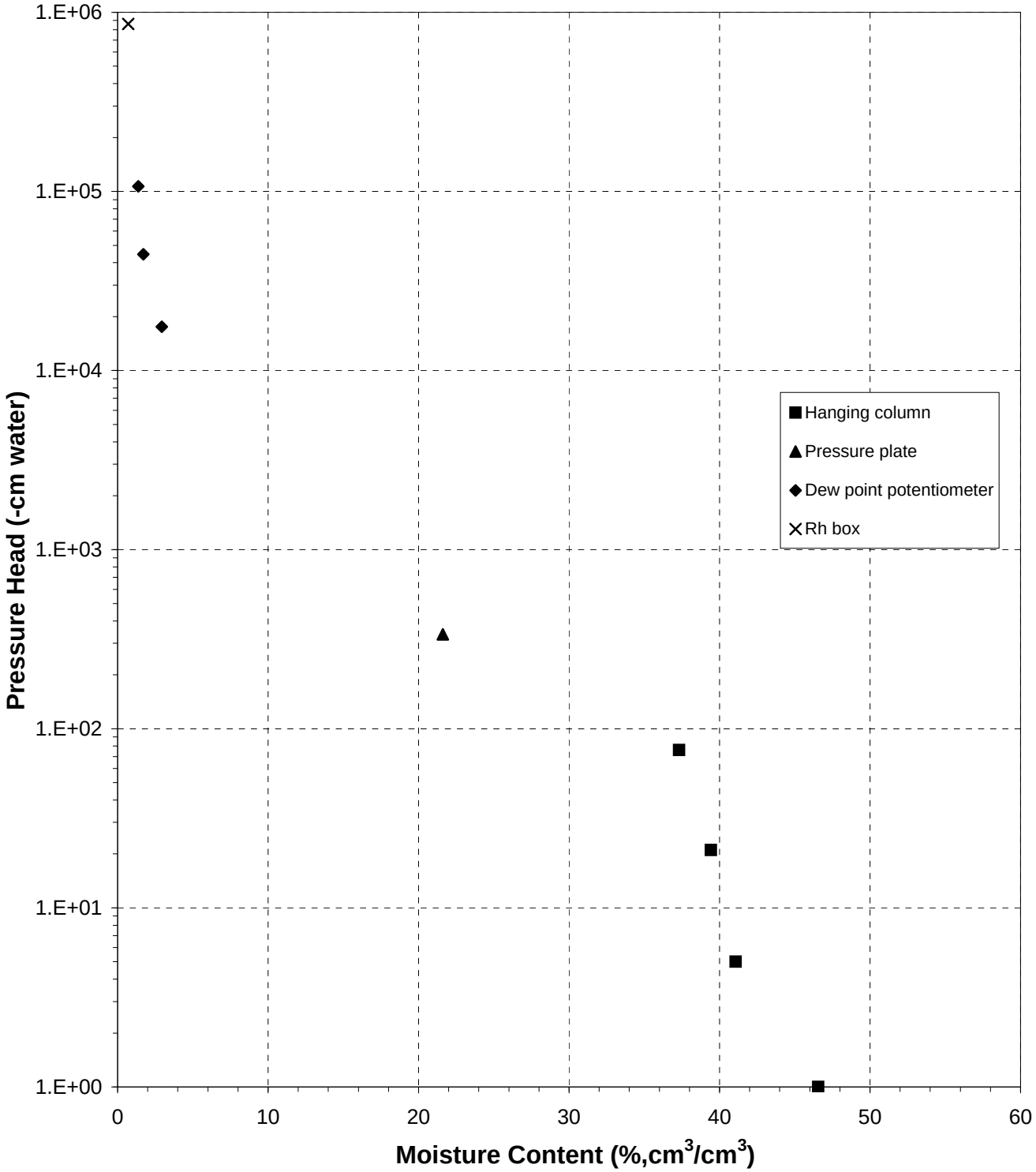
† Adjusted for >2.00mm (#10 sieve) material not used in DPP/RH testing. Assumed moisture content of material >2.00mm is zero, and assumed density of water is 1.0 g/cm^3.

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Laboratory analysis by: M. Vigil
Data entered by: K. Wright
Checked by: J. Hines

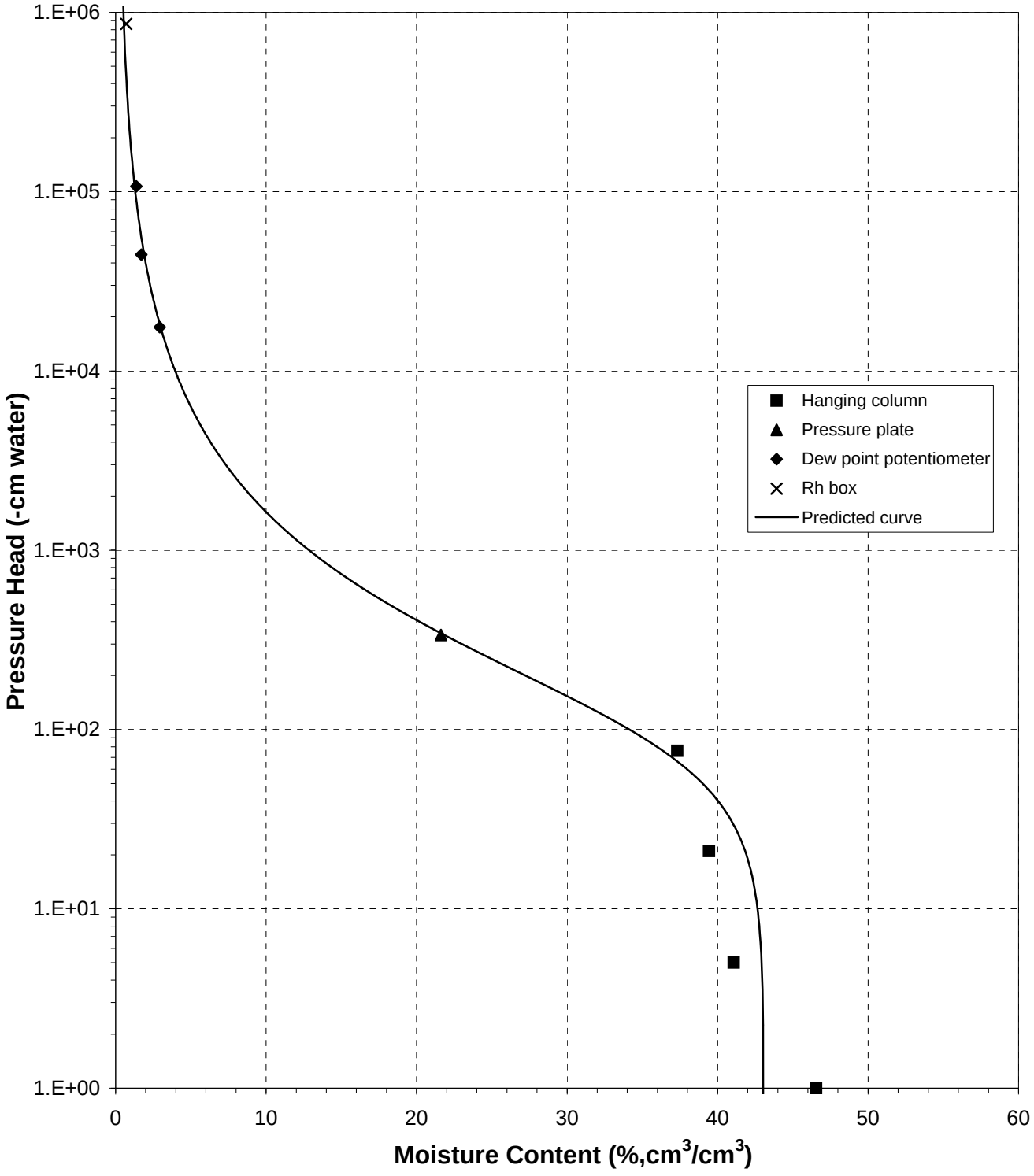
Water Retention Data Points

Sample Number: Blend 2 (55%)



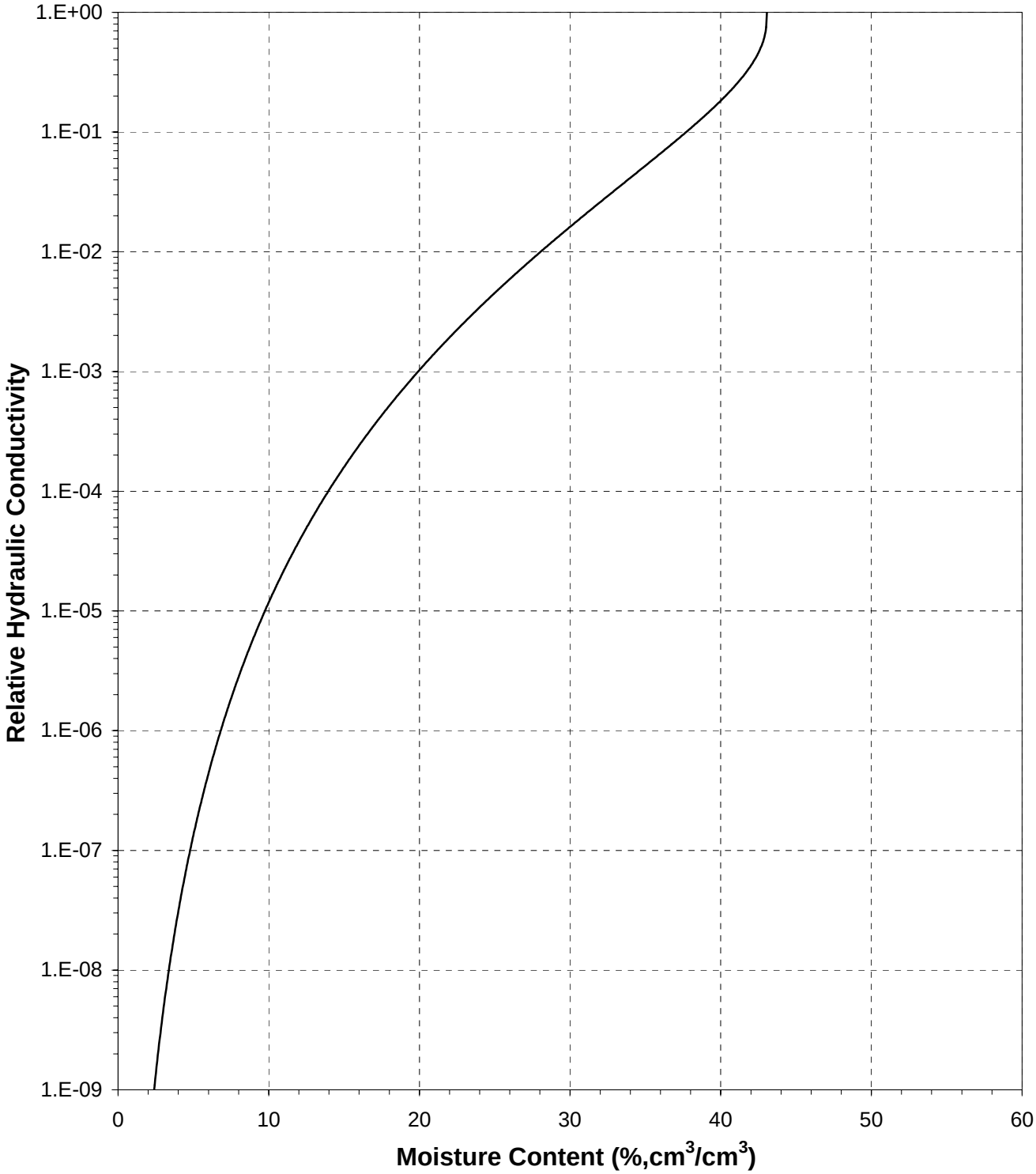
Predicted Water Retention Curve and Data Points

Sample Number: Blend 2 (55%)



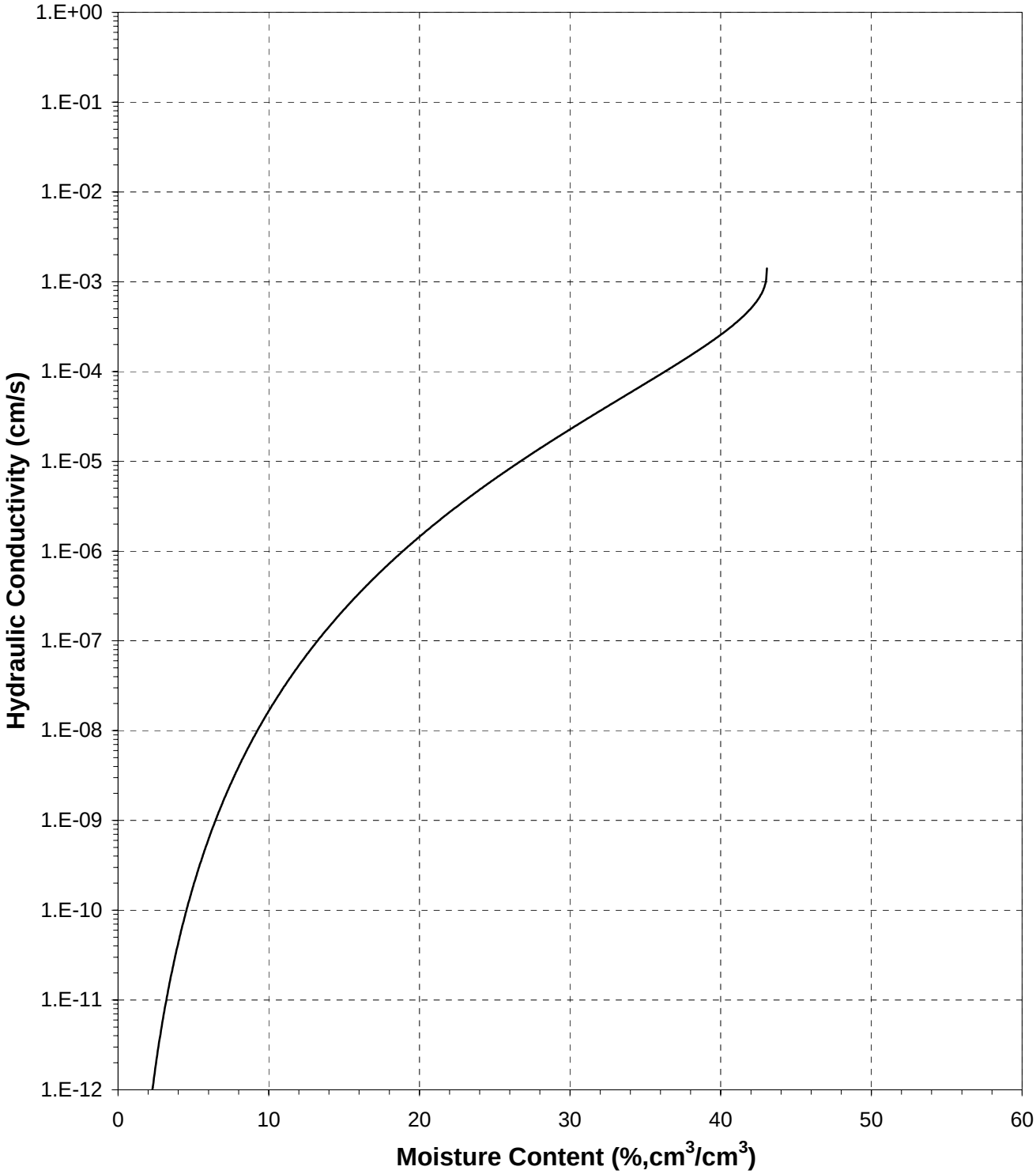
Plot of Relative Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 2 (55%)



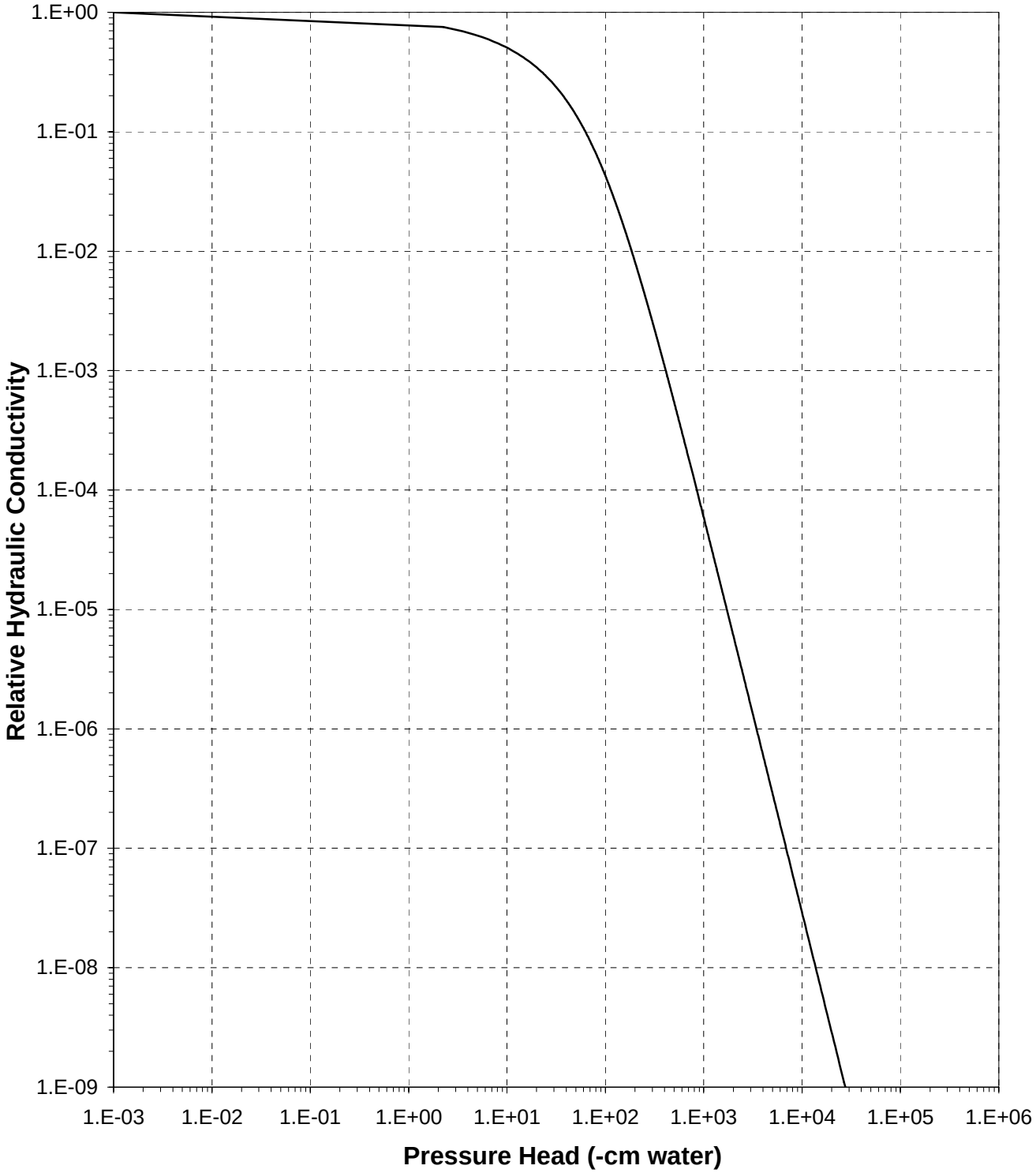
Plot of Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 2 (55%)



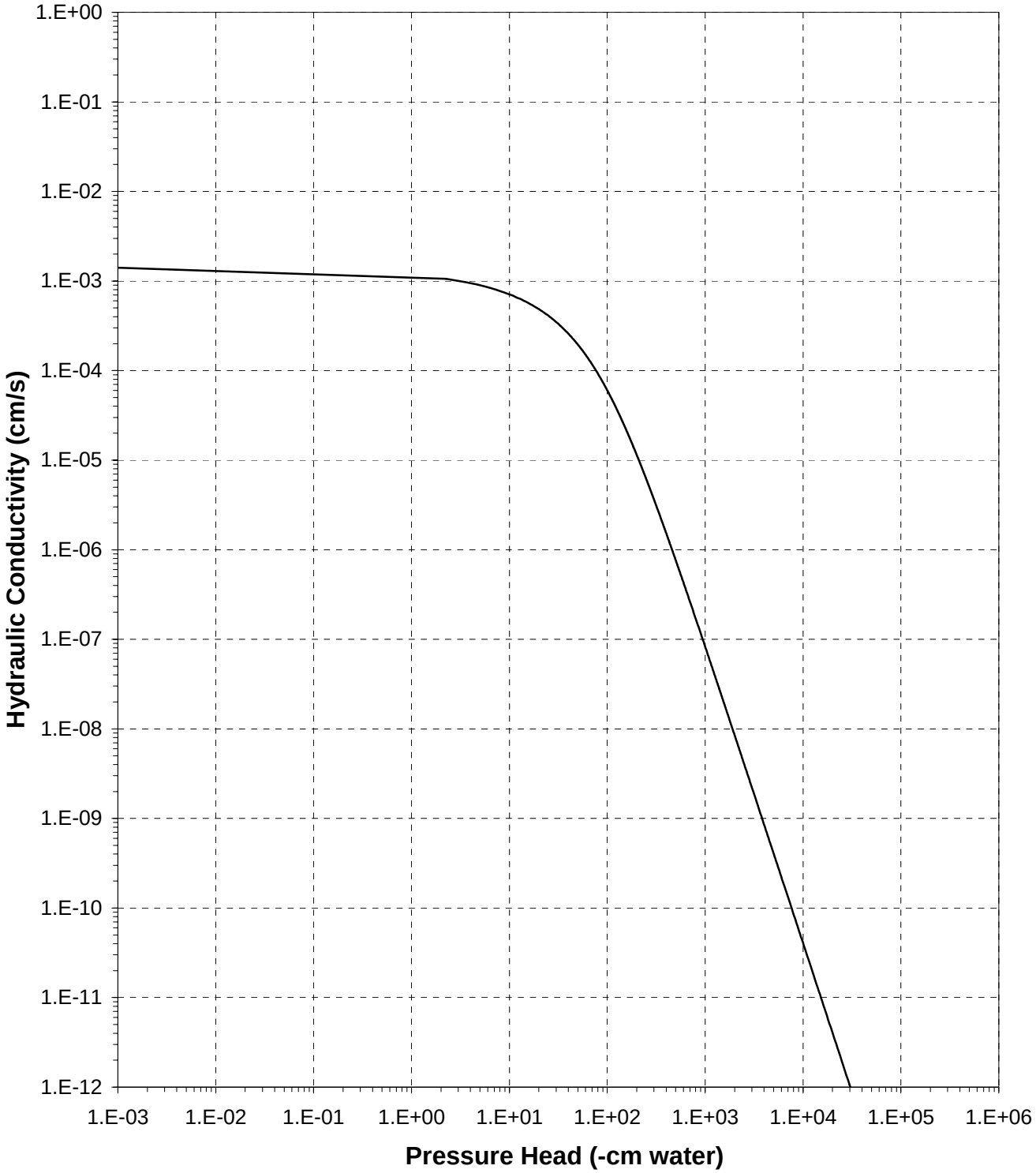
Plot of Relative Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 2 (55%)



Plot of Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 2 (55%)



Moisture Retention Data

Hanging Column / Pressure Plate

(Soil-Water Characteristic Curve)

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 3 (40%)
Ring Number: NA
Depth: NA

Dry wt. of sample (g)*:* 257.44
Tare wt., ring (g)*:* 87.14
Tare wt., screen & clamp (g)*:* 27.77
Initial sample volume (cm^3)*:* 143.29
Initial dry bulk density (g/cm^3)*:* 1.80
Measured particle density (g/cm^3)*:* 2.98
Initial calculated total porosity (%)*:* 39.69

	Date	Time	Weight* (g)	Matric Potential (-cm water)	Moisture Content† (% vol)
Hanging column:	7-Sep-10	11:00	429.21	0	39.68
	13-Sep-10	10:35	428.33	12.0	39.07
	19-Sep-10	10:30	428.01	31.0	38.84
	26-Sep-10	8:40	411.02	101.0	26.99
Pressure plate:	4-Oct-10	11:00	389.97	337	12.30

Volume Adjusted Data [1]

	Matric Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calculated Porosity (%)
Hanging column:	0.0	---	---	---	---
	12.0	---	---	---	---
	31.0	---	---	---	---
	101.0	---	---	---	---
Pressure plate:	337	---	---	---	---

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent each of the volume change measurements obtained after saturated hydraulic conductivity testing and throughout hanging column/pressure plate testing. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

† Assumed density of water is 1.0 g/cm^3

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Technician Notes:

Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Moisture Retention Data

Dew Point Potentiometer / Relative Humidity Box

(Soil-Water Characteristic Curve)

Sample Number: Blend 3 (40%)

Initial sample bulk density (g/cm^3)*:* 1.80

Fraction of bulk sample used (<2.00mm fraction) (%): 100.00

Dry weight of dew point potentiometer sample* (g)*:* 145.09

Tare weight, jar (g)*:* 113.91

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content† (% vol)
Dew point potentiometer:	21-Oct-10	10:03	145.64	16215	3.17
	19-Oct-10	13:30	145.59	36611	2.88
	21-Oct-10	8:33	145.44	138387	2.02

Volume Adjusted Data[1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change[2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Dew point potentiometer:	16215	---	---	---	---
	36611	---	---	---	---
	138387	---	---	---	---

Dry weight of relative humidity box sample* (g)*:* 80.32

Tare weight (g)*:* 47.61

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content† (% vol)
Relative humidity box:	16-Sep-10	11:05	80.44	861325	0.64

Volume Adjusted Data[1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change[2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Relative humidity box:	861325	---	---	---	---

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent the volume change measurements obtained after the last hanging column or pressure plate point. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

† Adjusted for >2.00mm (#10 sieve) material not used in DPP/RH testing. Assumed moisture content of material >2.00mm is zero, and assumed density of water is 1.0 g/cm^3.

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

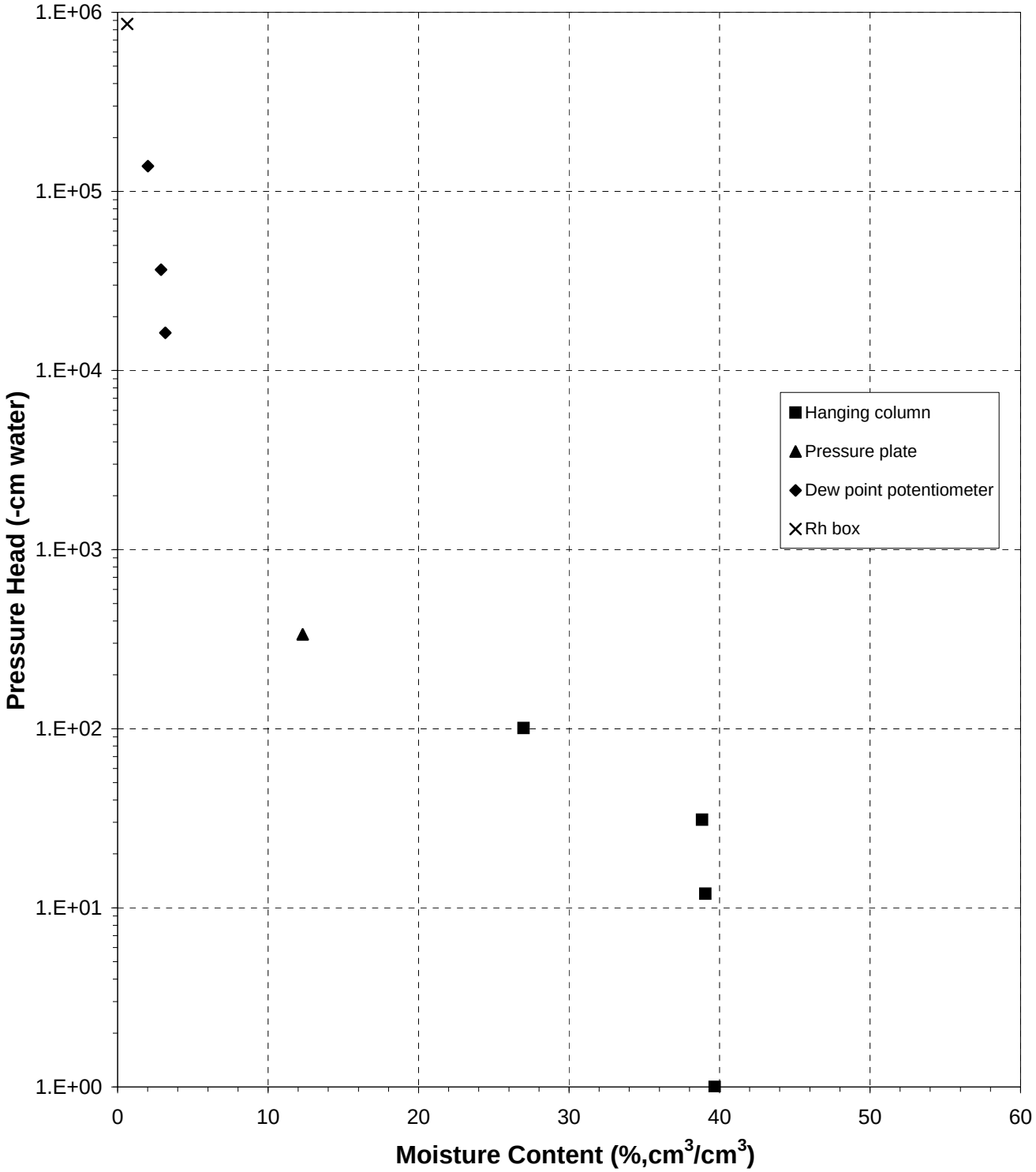
Laboratory analysis by: K. Wright/M. Vigil

Data entered by: K. Wright

Checked by: J. Hines

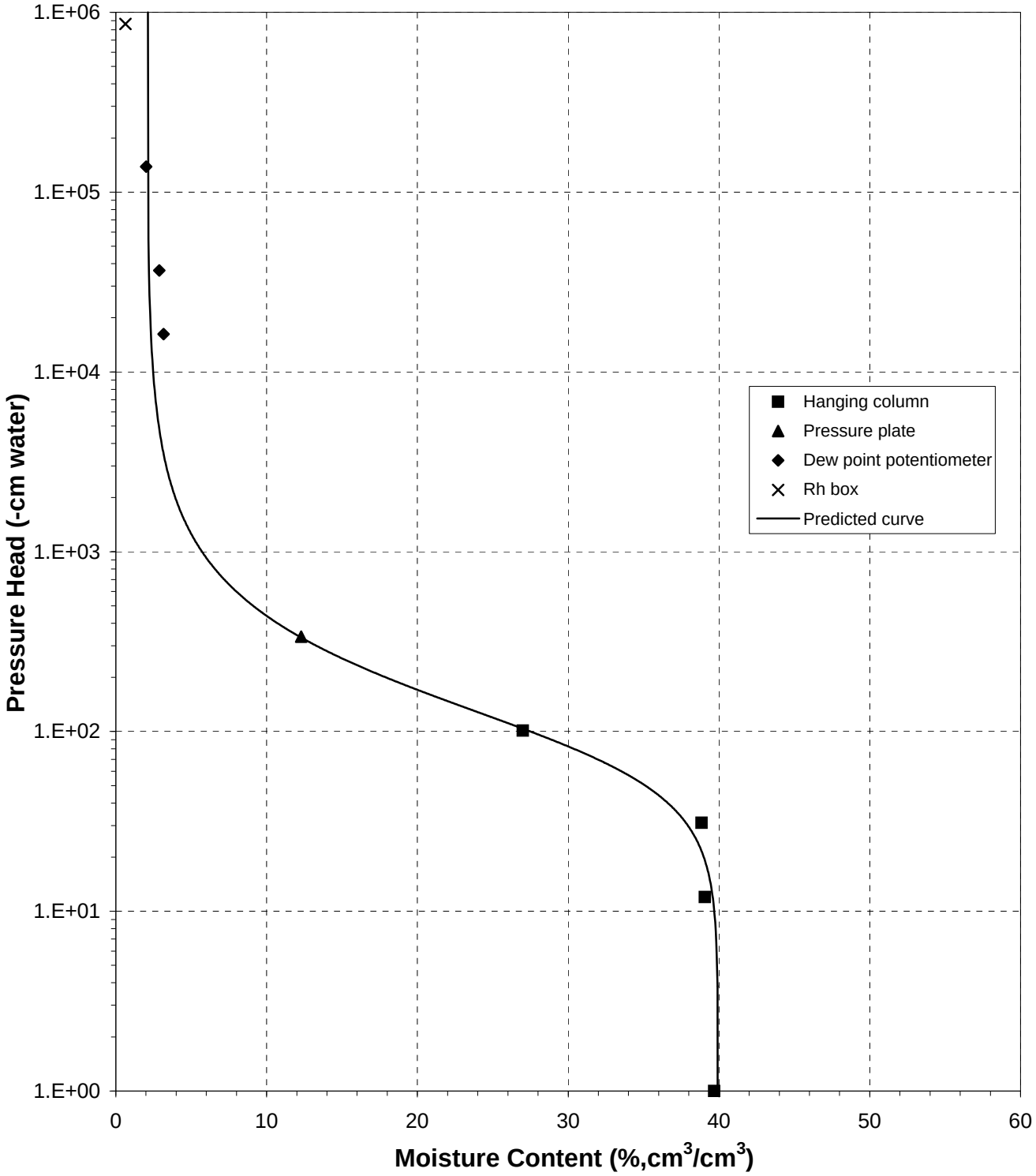
Water Retention Data Points

Sample Number: Blend 3 (40%)



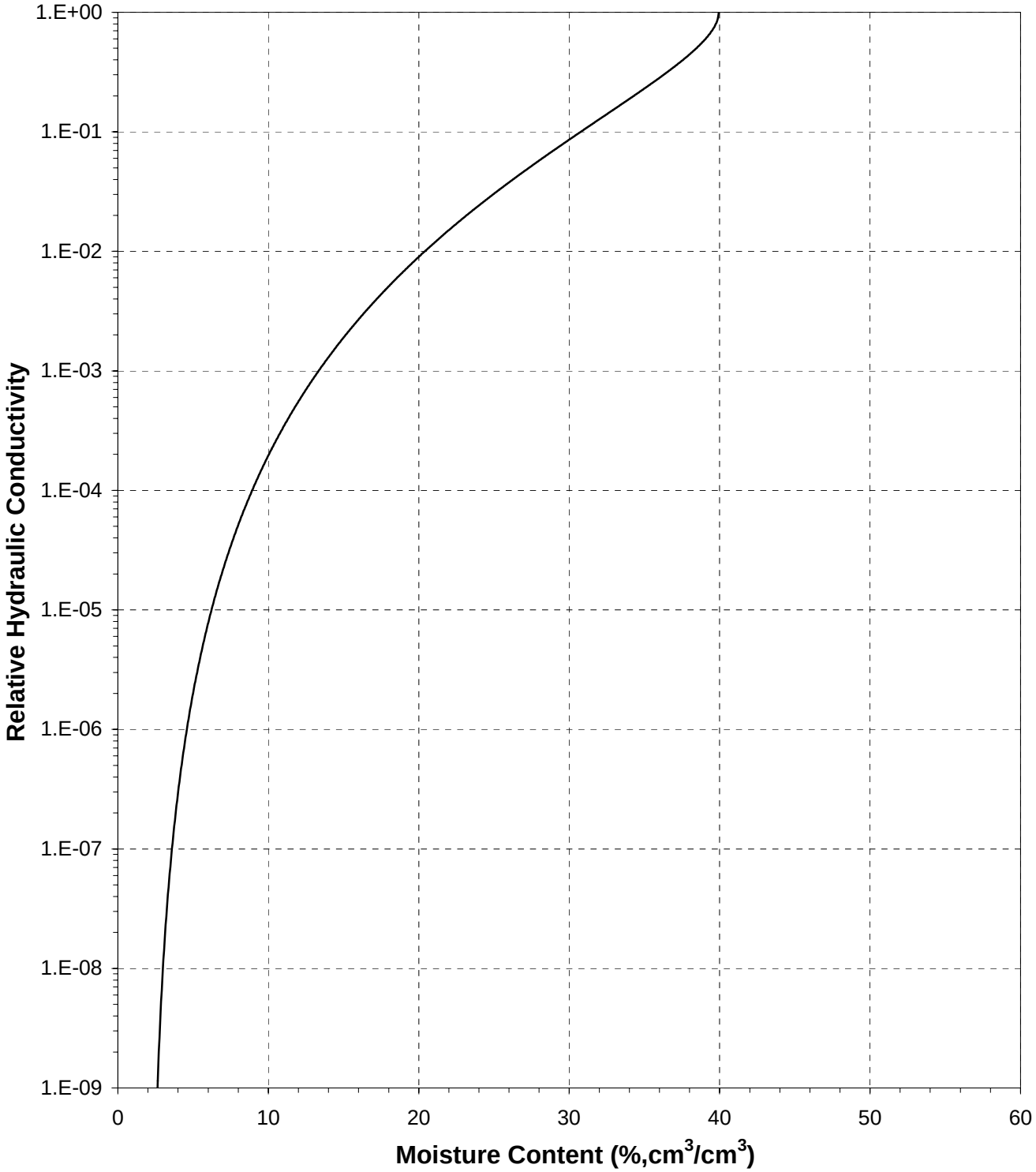
Predicted Water Retention Curve and Data Points

Sample Number: Blend 3 (40%)



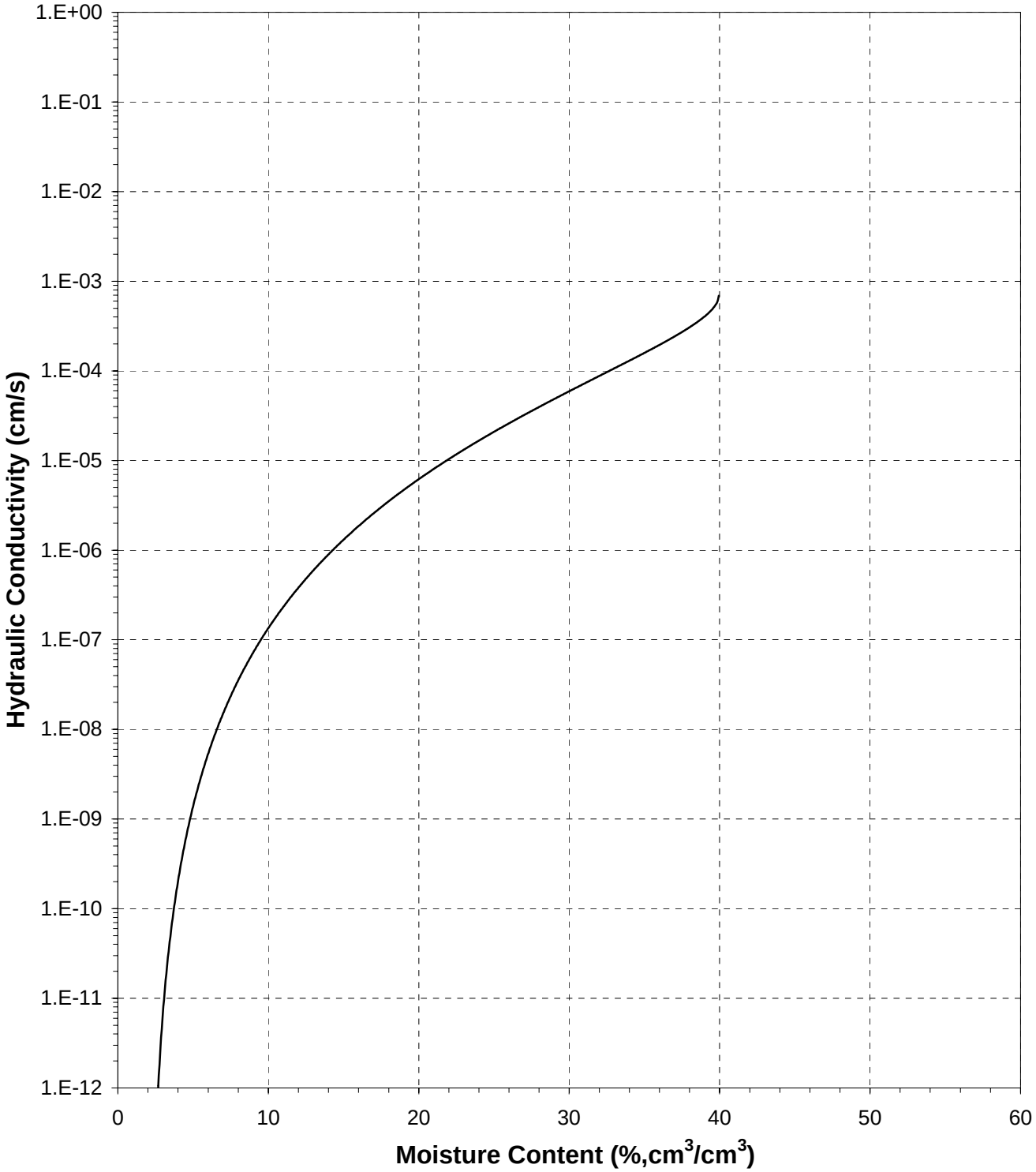
Plot of Relative Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 3 (40%)



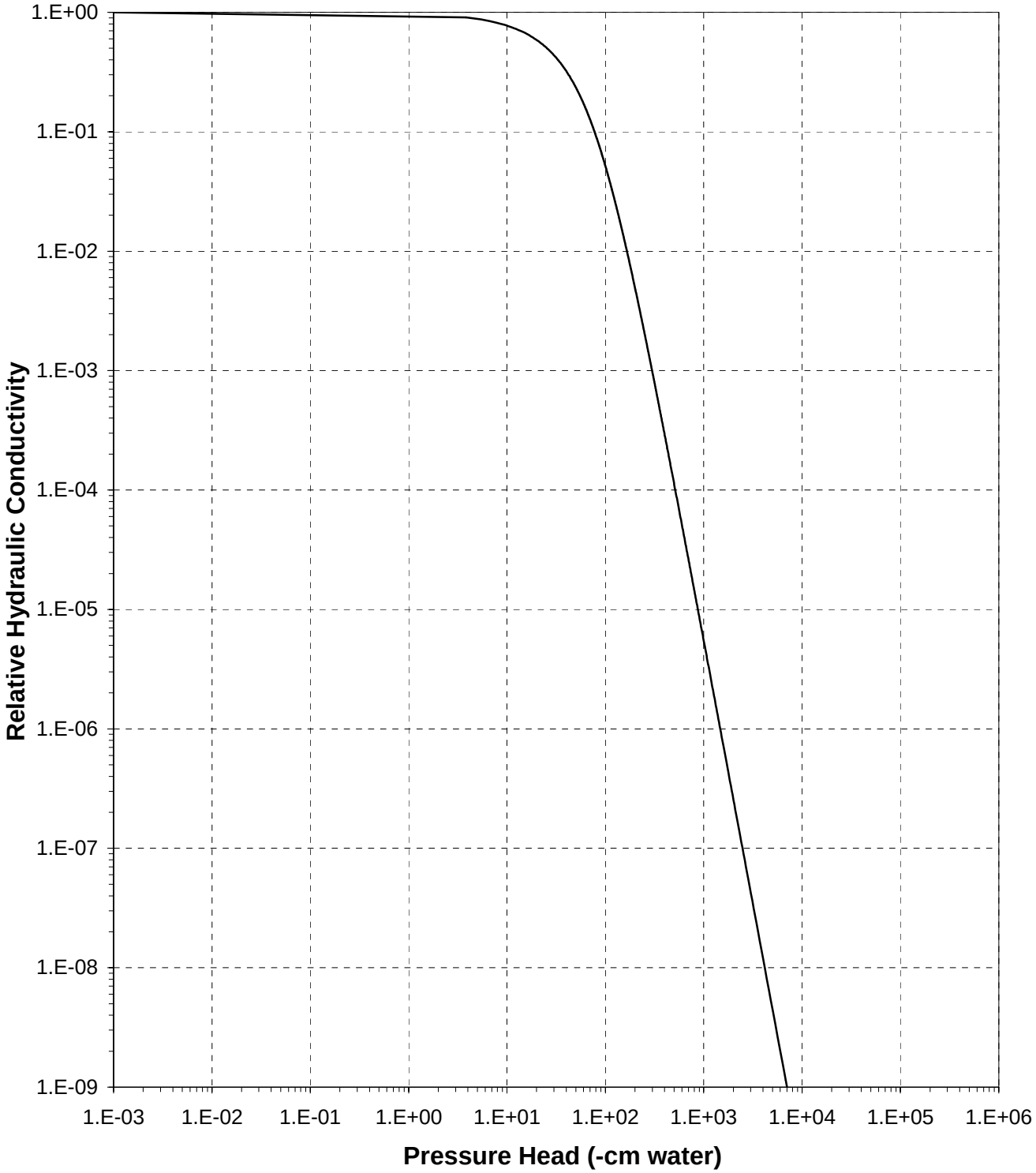
Plot of Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 3 (40%)



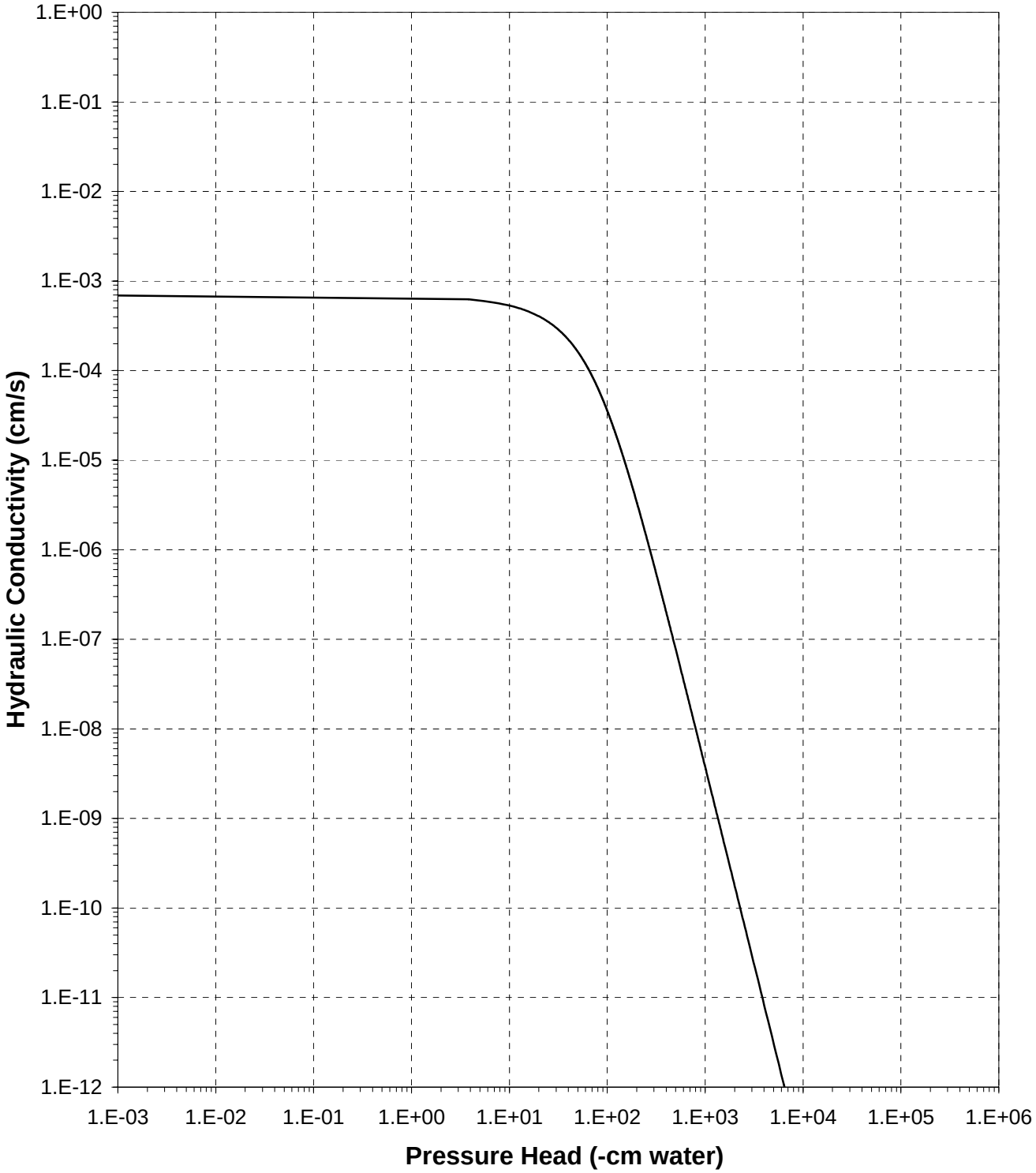
Plot of Relative Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 3 (40%)



Plot of Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 3 (40%)



Moisture Retention Data

Hanging Column / Pressure Plate

(Soil-Water Characteristic Curve)

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 3 (45%)
Ring Number: NA
Depth: NA

Dry wt. of sample (g)*:* 232.98
Tare wt., ring (g)*:* 87.13
Tare wt., screen & clamp (g)*:* 26.42
Initial sample volume (cm^3)*:* 142.72
Initial dry bulk density (g/cm^3)*:* 1.63
Measured particle density (g/cm^3)*:* 2.98
Initial calculated total porosity (%)*:* 45.20

	Date	Time	Weight* (g)	Matric Potential (-cm water)	Moisture Content† (% vol)	
Hanging column:	7-Sep-10	14:45	408.54	0	44.86	‡‡
	13-Sep-10	13:15	404.95	5.0	42.89	‡‡
	19-Sep-10	11:30	401.62	21.0	41.52	‡‡
	26-Sep-10	9:50	399.04	76.0	39.70	‡‡
Pressure plate:	4-Oct-10	13:15	362.41	337	12.06	‡‡

Volume Adjusted Data [1]

	Matric Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calculated Porosity (%)
Hanging column:	0.0	138.24	-3.14%	1.69	43.43
	5.0	136.22	-4.55%	1.71	42.59
	21.0	132.67	-7.04%	1.76	41.05
	76.0	132.27	-7.32%	1.76	40.87
Pressure plate:	337	131.63	-7.77%	1.77	40.59

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent each of the volume change measurements obtained after saturated hydraulic conductivity testing and throughout hanging column/pressure plate testing. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

† Assumed density of water is 1.0 g/cm^3

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Technician Notes:

Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Moisture Retention Data

Dew Point Potentiometer / Relative Humidity Box

(Soil-Water Characteristic Curve)

Sample Number: Blend 3 (45%)

Initial sample bulk density (g/cm^3)*:* 1.63
Fraction of bulk sample used (<2.00mm fraction) (%): 100.00

Dry weight of dew point potentiometer sample* (g)*:* 146.29
Tare weight, jar (g)*:* 115.51

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)	
Dew point potentiometer:	24-Sep-10	14:49	146.64	14481	1.99	‡‡
	24-Sep-10	14:37	146.51	41098	1.29	‡‡
	24-Sep-10	11:36	146.44	141752	0.86	‡‡

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Dew point potentiometer:	14481	131.63	-7.77%	1.77	40.59
	41098	131.63	-7.77%	1.77	40.59
	141752	131.63	-7.77%	1.77	40.59

Dry weight of relative humidity box sample* (g)*:* 75.89
Tare weight (g)*:* 38.33

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)	
Relative humidity box:	16-Sep-10	11:05	76.02	861325	0.59	‡‡

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Relative humidity box:	861325	131.63	-7.77%	1.77	40.59

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent the volume change measurements obtained after the last hanging column or pressure plate point. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

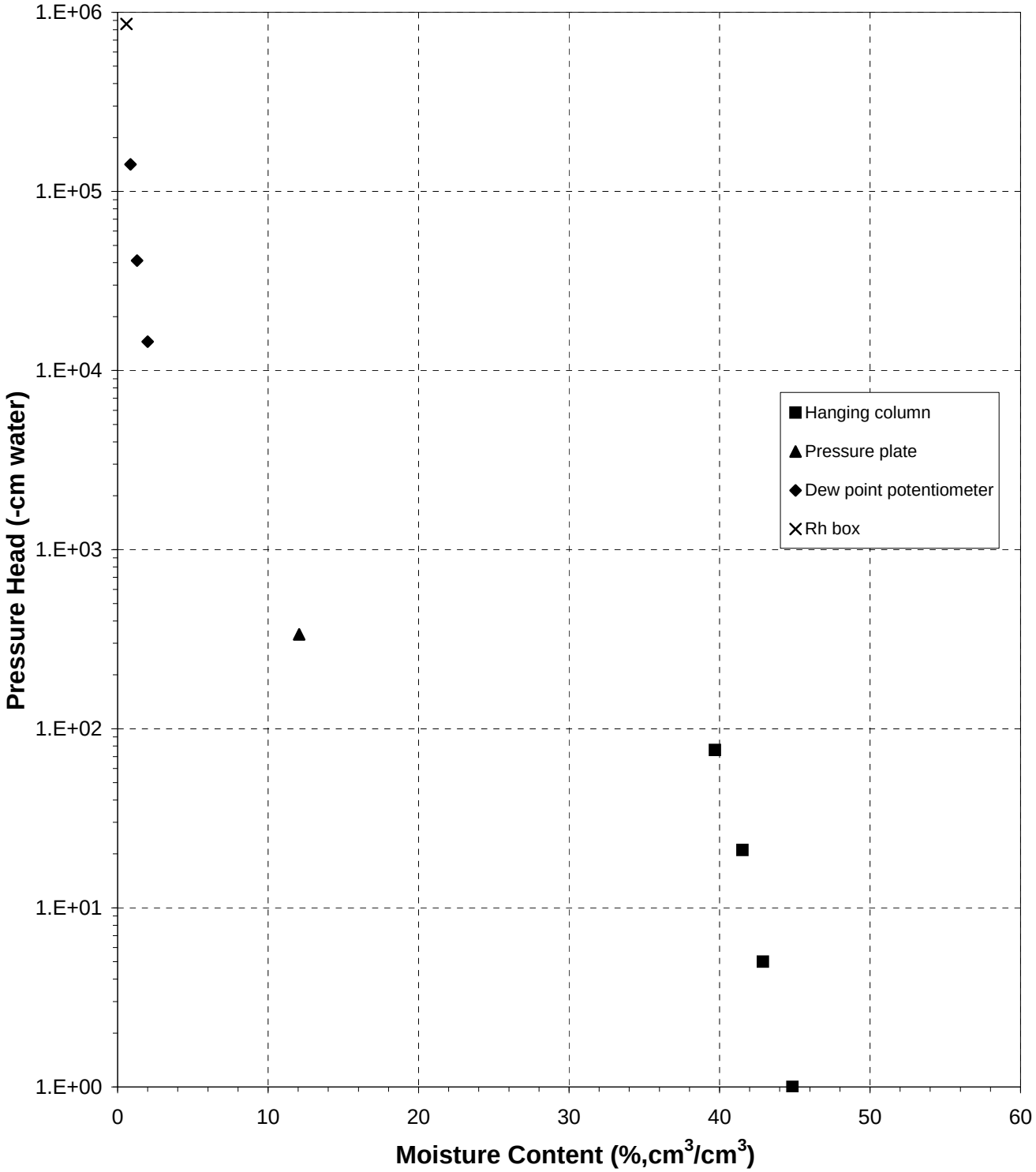
† Adjusted for >2.00mm (#10 sieve) material not used in DPP/RH testing. Assumed moisture content of material >2.00mm is zero, and assumed density of water is 1.0 g/cm^3.

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Laboratory analysis by: M. Vigil
Data entered by: K. Wright
Checked by: J. Hines

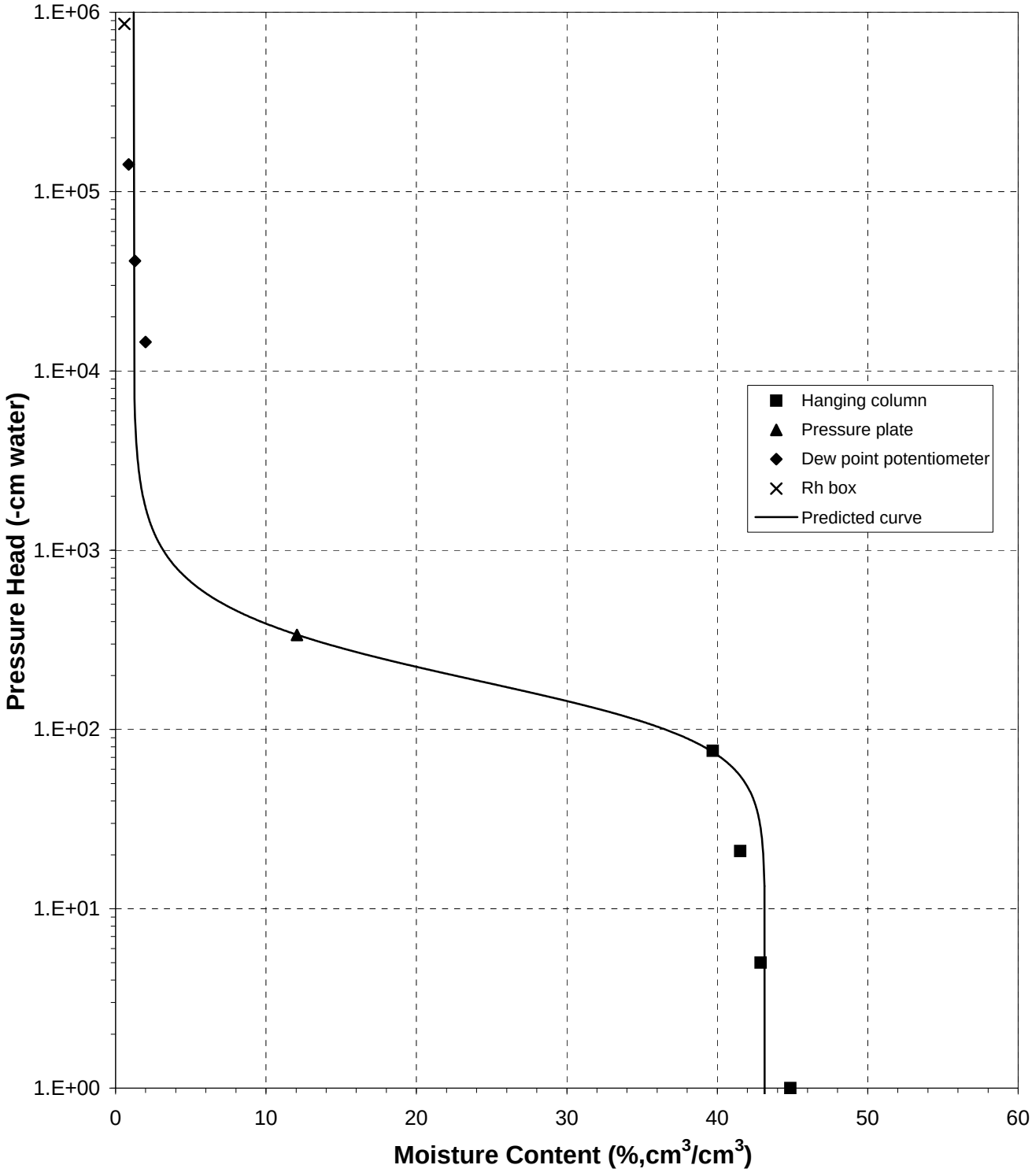
Water Retention Data Points

Sample Number: Blend 3 (45%)



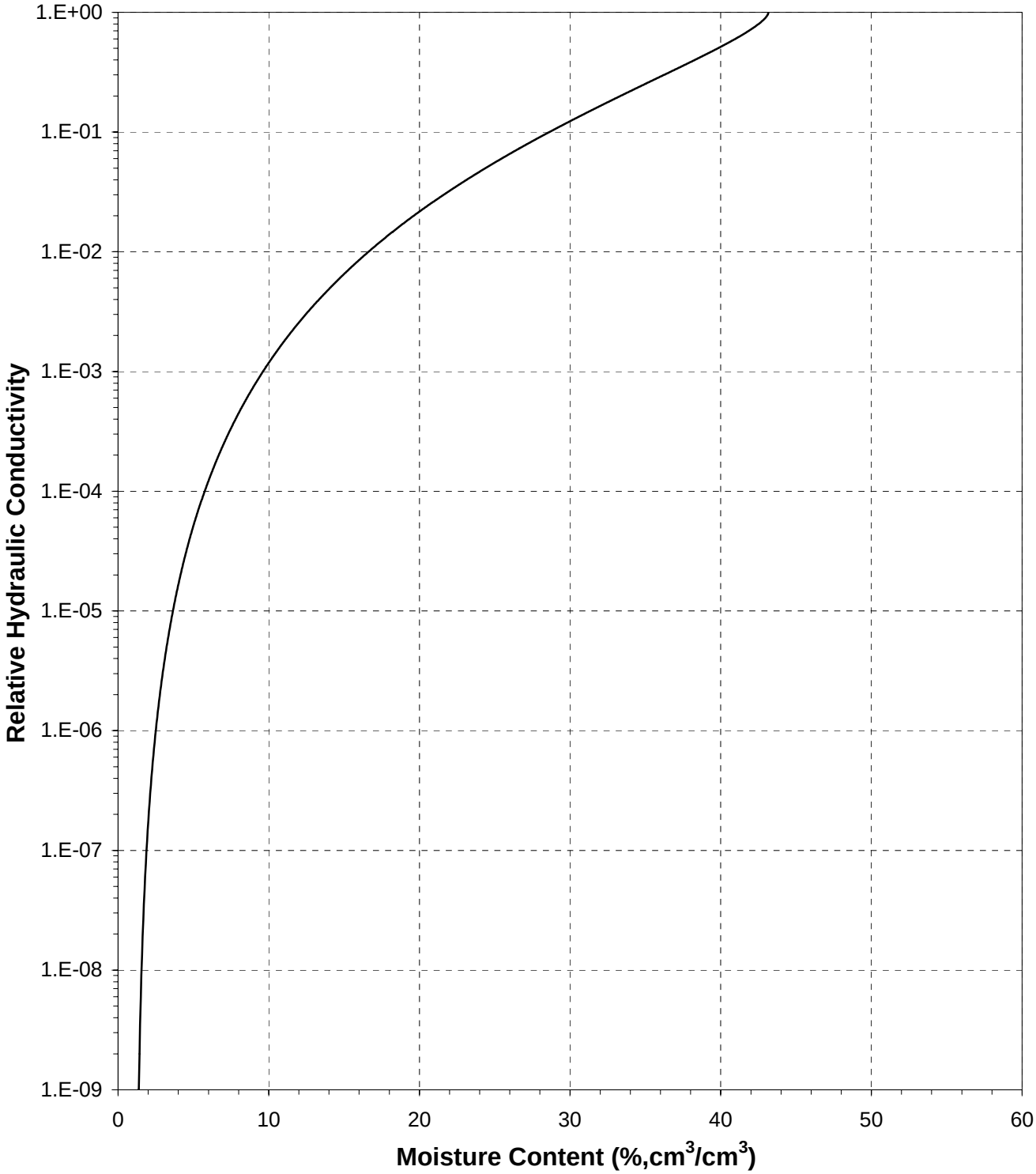
Predicted Water Retention Curve and Data Points

Sample Number: Blend 3 (45%)



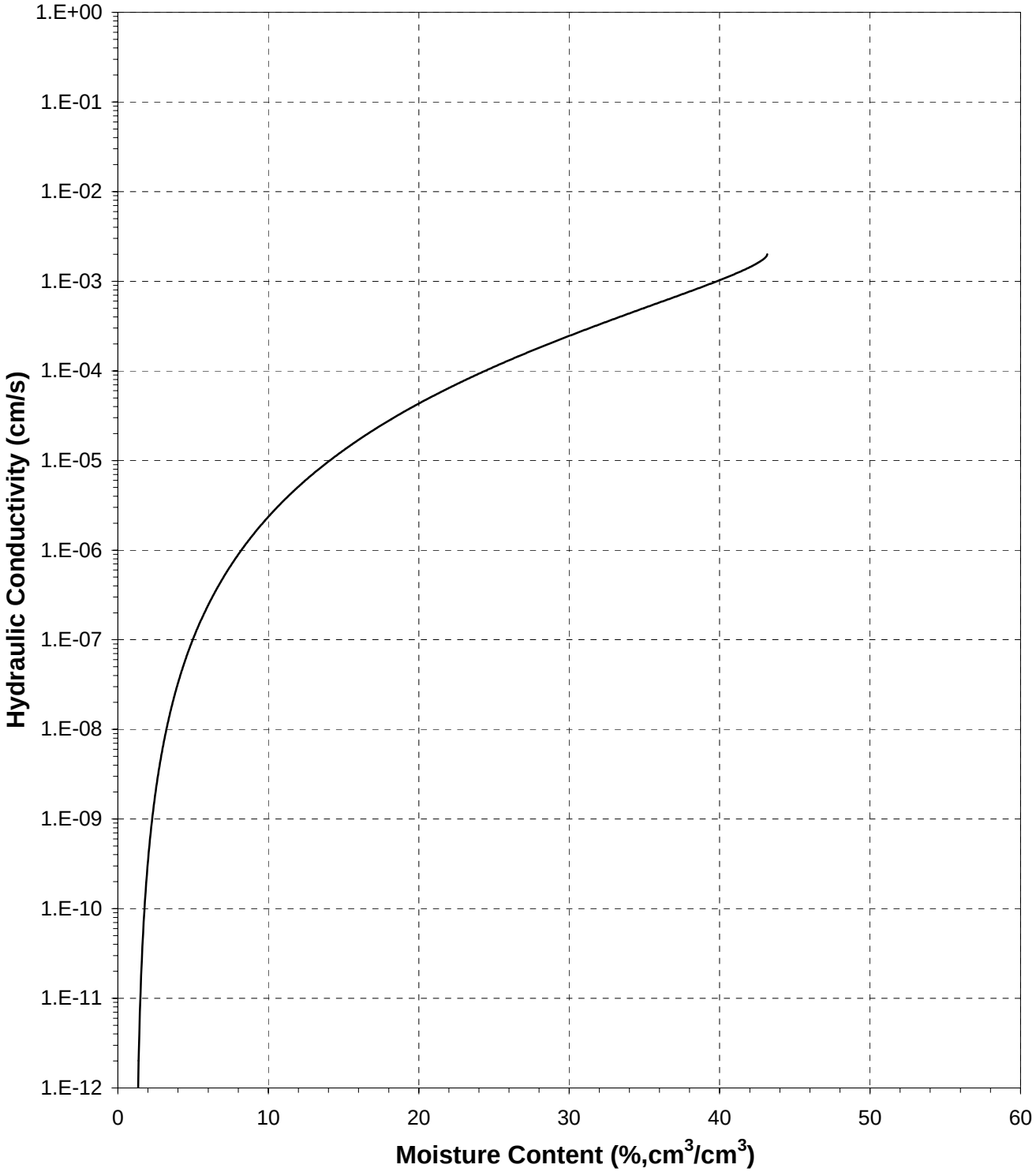
Plot of Relative Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 3 (45%)



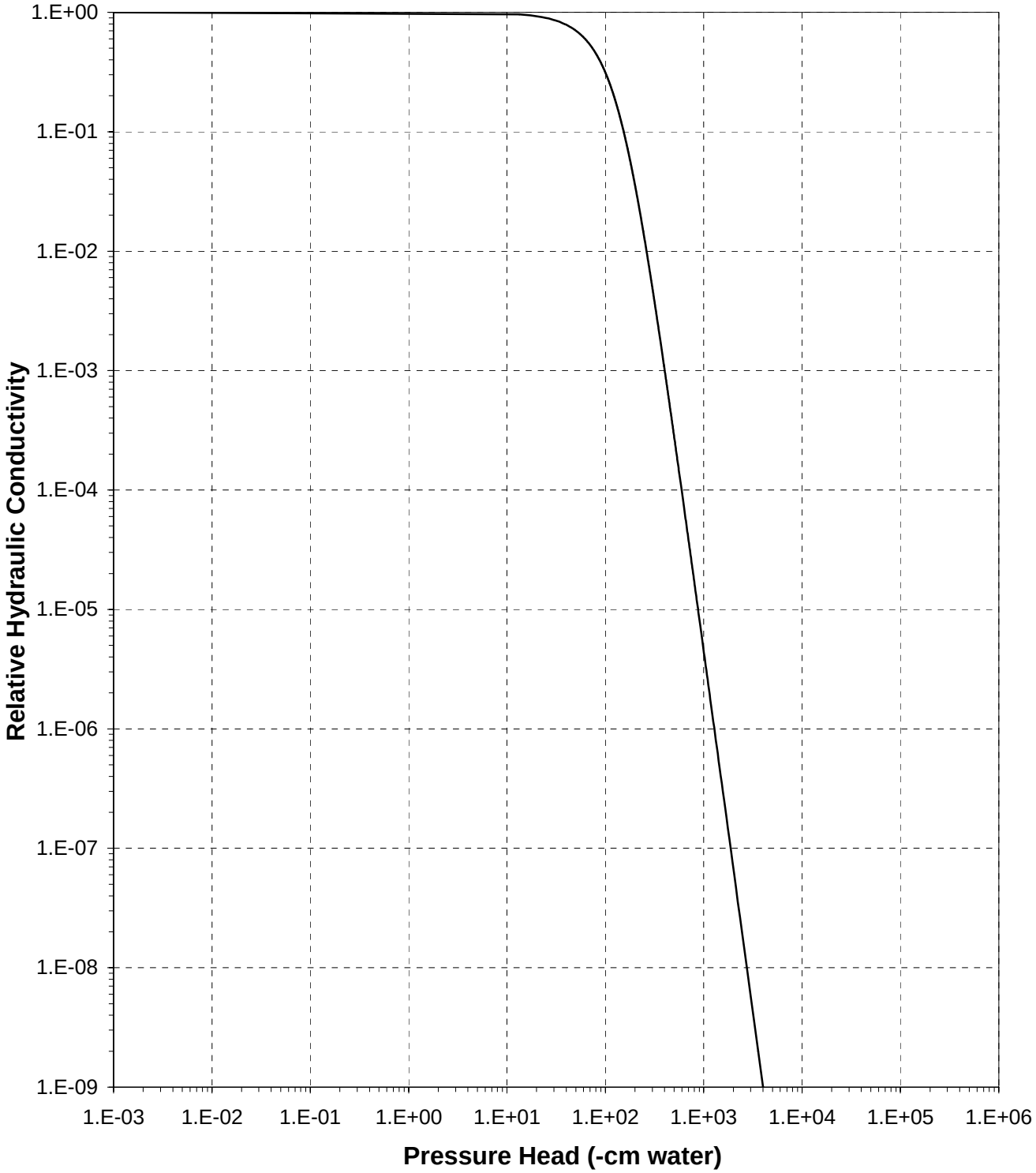
Plot of Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 3 (45%)



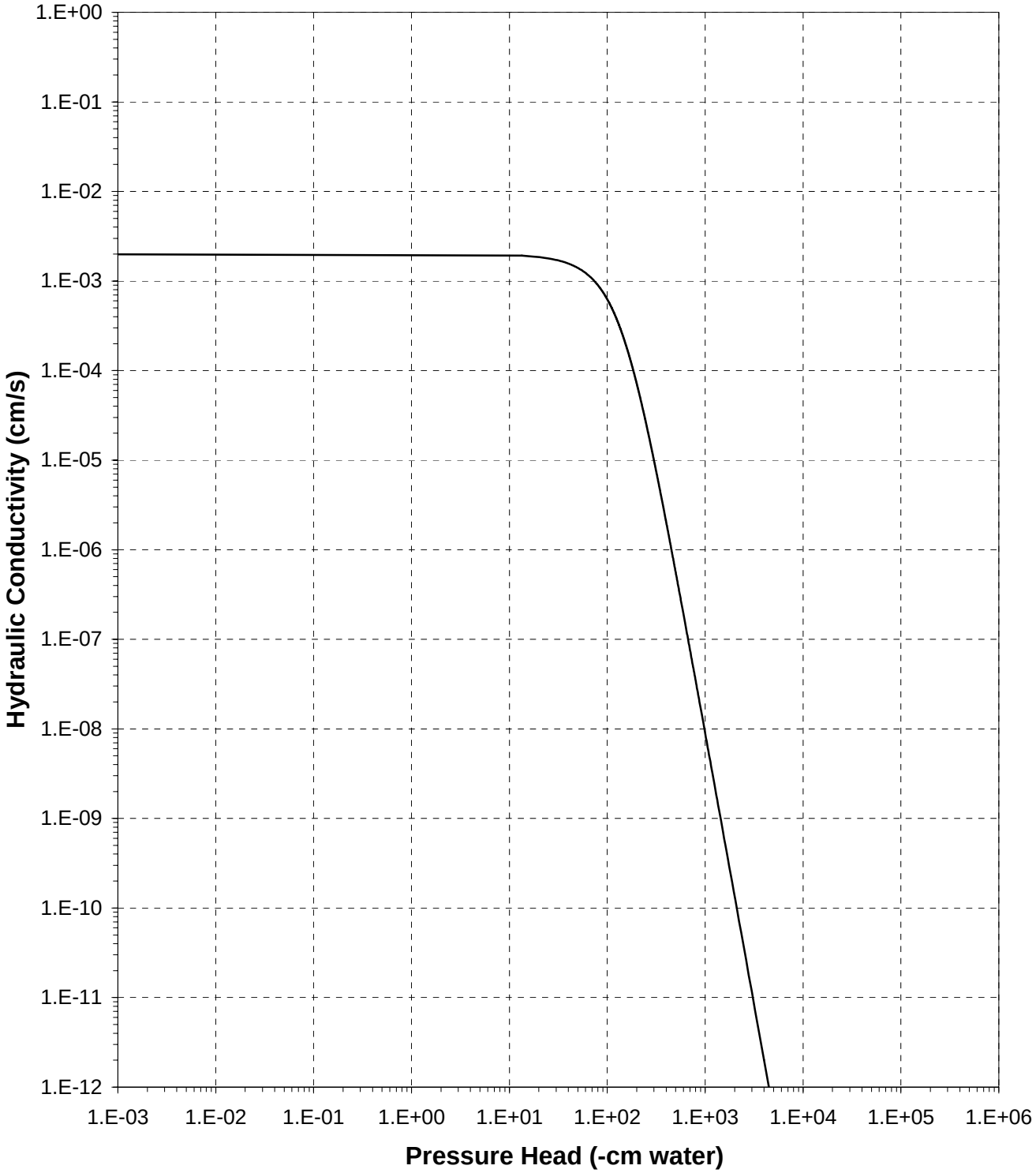
Plot of Relative Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 3 (45%)



Plot of Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 3 (45%)



Moisture Retention Data

Hanging Column / Pressure Plate

(Soil-Water Characteristic Curve)

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 3 (55%)
Ring Number: NA
Depth: NA

Dry wt. of sample (g)*:* 191.84
Tare wt., ring (g)*:* 88.41
Tare wt., screen & clamp (g)*:* 27.74
Initial sample volume (cm^3)*:* 146.31
Initial dry bulk density (g/cm^3)*:* 1.31
Measured particle density (g/cm^3)*:* 2.98
Initial calculated total porosity (%)*:* 55.98

	Date	Time	Weight* (g)	Matric Potential (-cm water)	Moisture Content† (% vol)	
Hanging column:	7-Sep-10	14:35	356.77	0	44.03	‡‡
	13-Sep-10	13:05	348.83	5.0	38.74	‡‡
	19-Sep-10	11:40	346.42	21.0	37.05	‡‡
	26-Sep-10	9:40	345.39	76.0	36.71	‡‡
Pressure plate:	4-Oct-10	13:25	320.33	337	12.13	‡‡

Volume Adjusted Data [1]

	Matric Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calculated Porosity (%)
Hanging column:	0.0	110.78	-24.28%	1.73	41.87
	5.0	105.42	-27.94%	1.82	38.91
	21.0	103.72	-29.11%	1.85	37.91
	76.0	101.87	-30.37%	1.88	36.79
Pressure plate:	337	101.74	-30.46%	1.89	36.70

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent each of the volume change measurements obtained after saturated hydraulic conductivity testing and throughout hanging column/pressure plate testing. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

† Assumed density of water is 1.0 g/cm^3

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Technician Notes:

Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Moisture Retention Data

Dew Point Potentiometer / Relative Humidity Box

(Soil-Water Characteristic Curve)

Sample Number: Blend 3 (55%)

Initial sample bulk density (g/cm^3)*:* 1.31
Fraction of bulk sample used (<2.00mm fraction) (%): 100.00

Dry weight of dew point potentiometer sample* (g)*:* 143.99
Tare weight, jar (g)*:* 112.60

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)	
Dew point potentiometer:	23-Sep-10	16:17	144.31	12951	1.92	‡‡
	23-Sep-10	16:02	144.21	34673	1.32	‡‡
	23-Sep-10	15:36	144.15	123396	0.96	‡‡

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Dew point potentiometer:	12951	101.74	-30.46%	1.89	36.70
	34673	101.74	-30.46%	1.89	36.70
	123396	101.74	-30.46%	1.89	36.70

Dry weight of relative humidity box sample* (g)*:* 74.59
Tare weight (g)*:* 37.81

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)	
Relative humidity box:	16-Sep-10	11:05	74.71	861325	0.63	‡‡

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Relative humidity box:	861325	101.74	-30.46%	1.89	36.70

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent the volume change measurements obtained after the last hanging column or pressure plate point. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

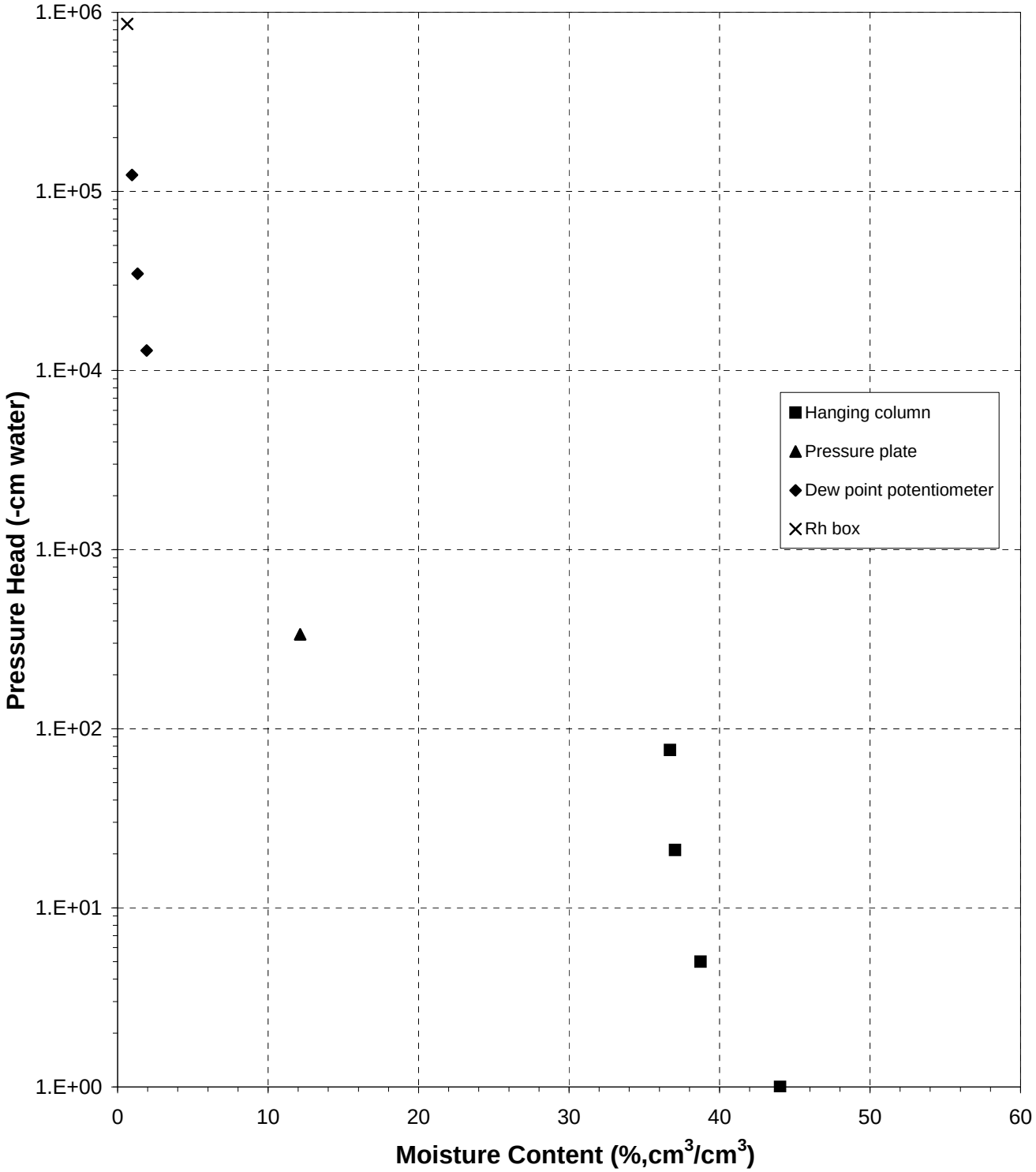
† Adjusted for >2.00mm (#10 sieve) material not used in DPP/RH testing. Assumed moisture content of material >2.00mm is zero, and assumed density of water is 1.0 g/cm^3.

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Laboratory analysis by: M. Vigil
Data entered by: K. Wright
Checked by: J. Hines

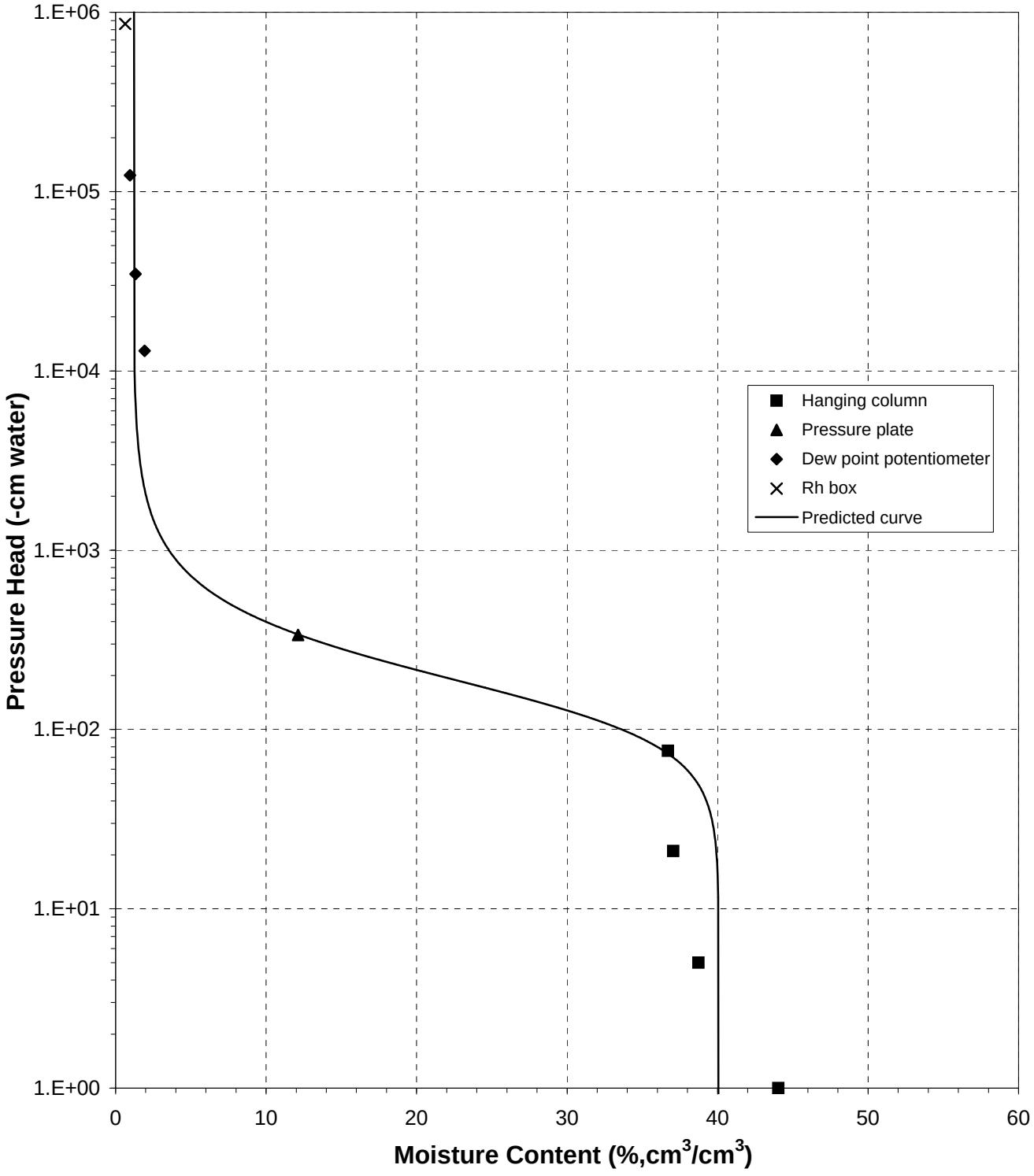
Water Retention Data Points

Sample Number: Blend 3 (55%)



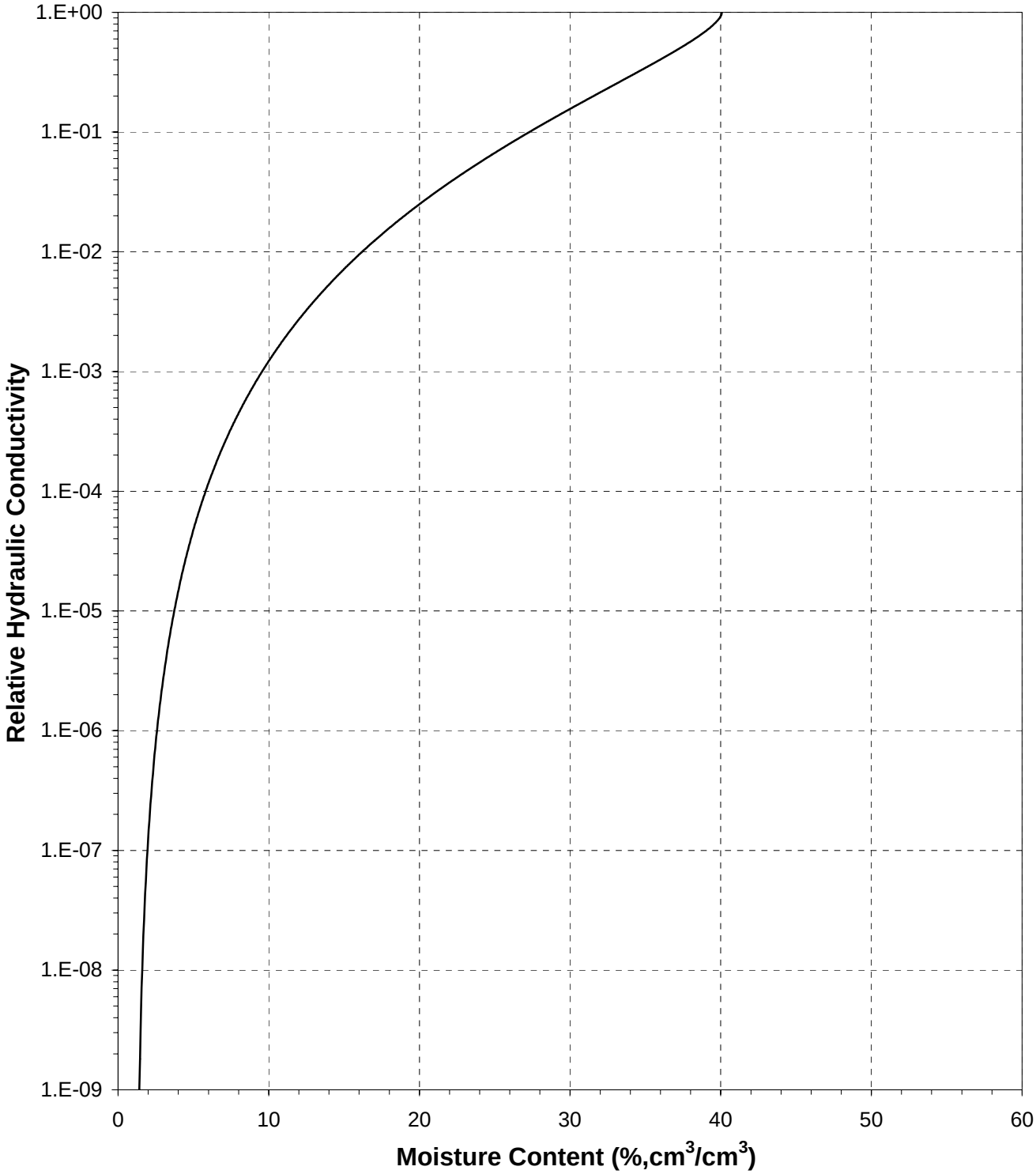
Predicted Water Retention Curve and Data Points

Sample Number: Blend 3 (55%)



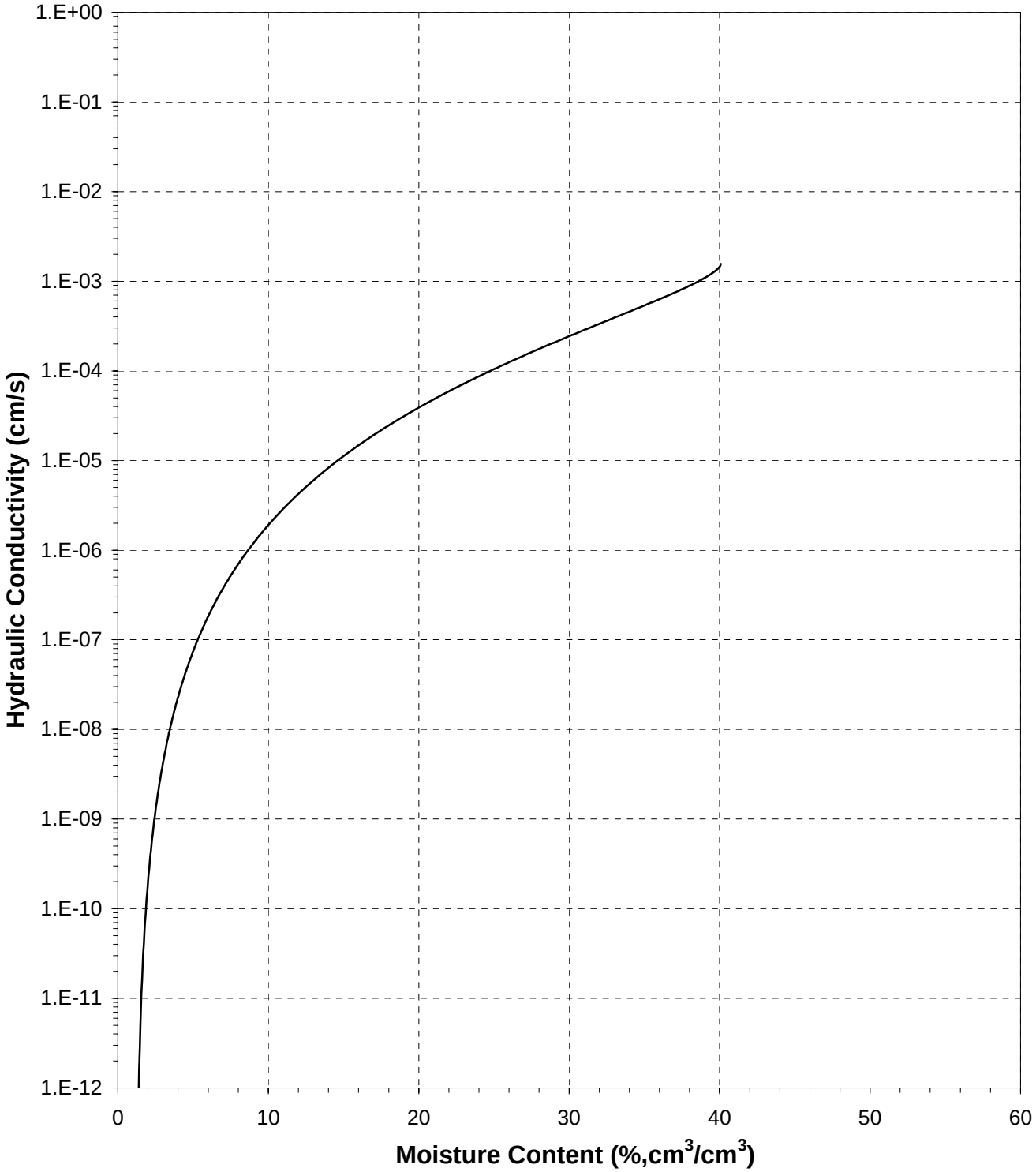
Plot of Relative Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 3 (55%)



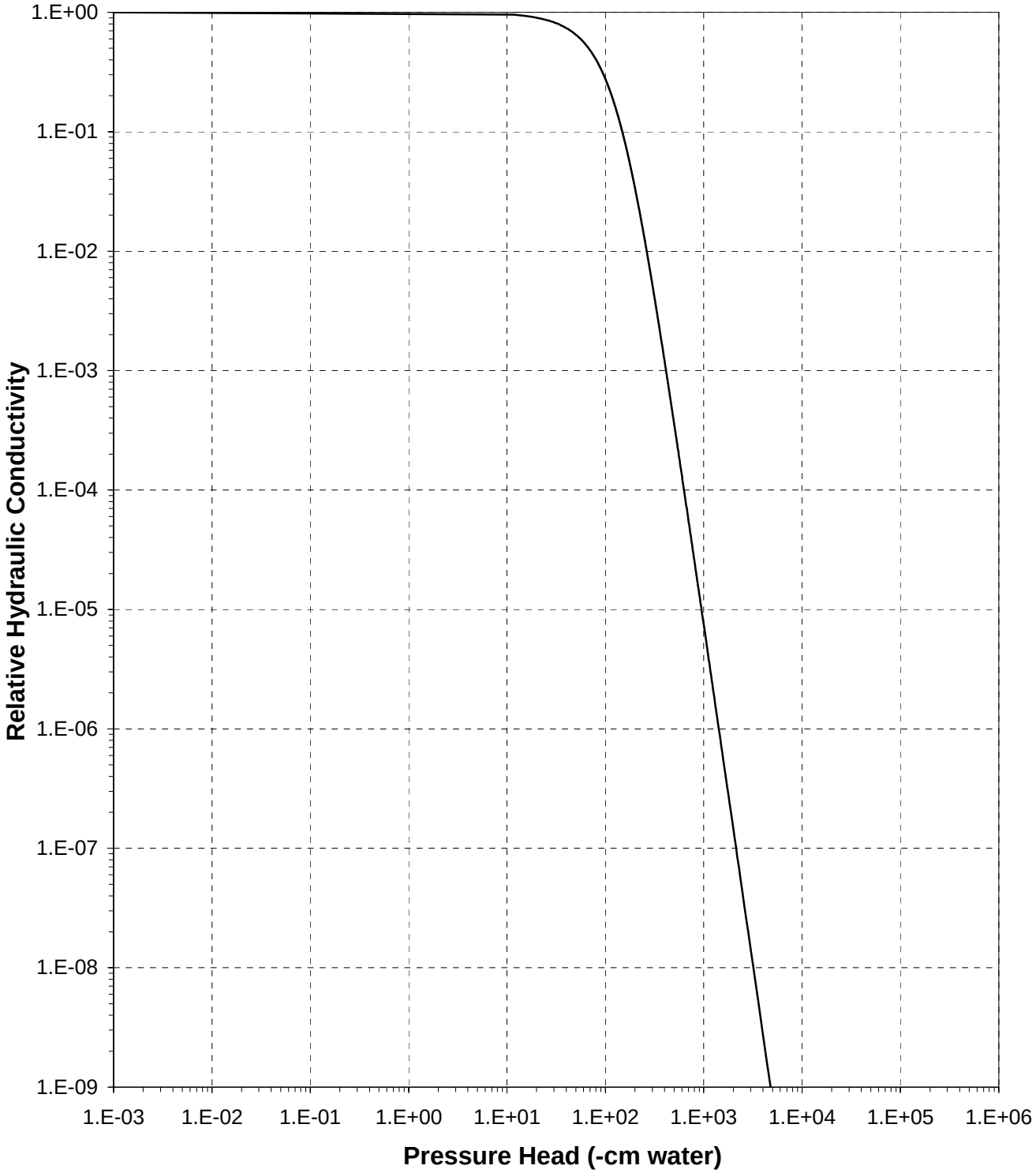
Plot of Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 3 (55%)



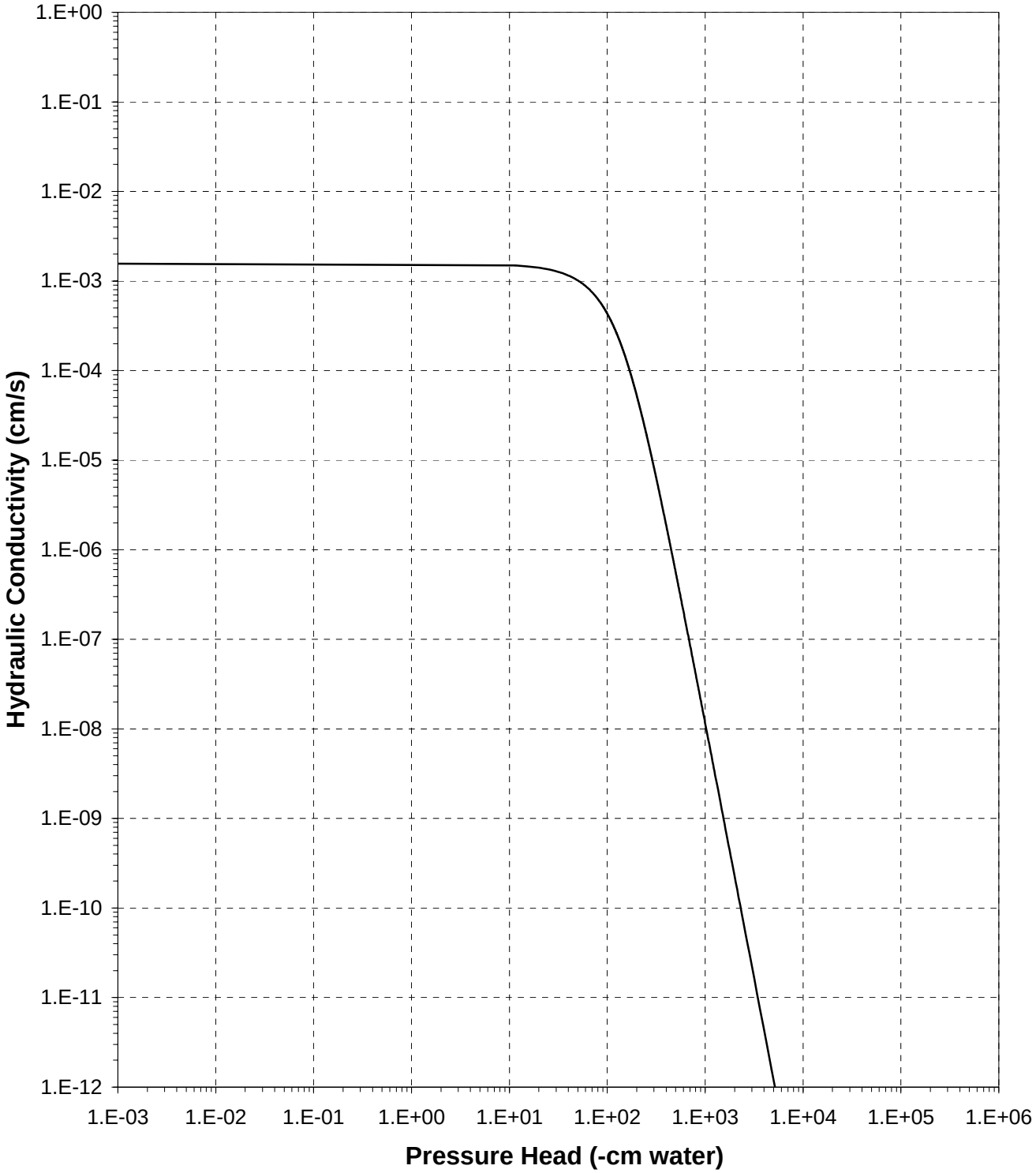
Plot of Relative Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 3 (55%)



Plot of Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 3 (55%)



Moisture Retention Data

Hanging Column / Pressure Plate

(Soil-Water Characteristic Curve)

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 4 (44%)
Ring Number: NA
Depth: NA

Dry wt. of sample (g)*:* 237.11
Tare wt., ring (g)*:* 87.15
Tare wt., screen & clamp (g)*:* 27.65
Initial sample volume (cm^3)*:* 144.43
Initial dry bulk density (g/cm^3)*:* 1.64
Measured particle density (g/cm^3)*:* 2.98
Initial calculated total porosity (%)*:* 44.82

	Date	Time	Weight* (g)	Matric Potential (-cm water)	Moisture Content† (% vol)
Hanging column:	7-Sep-10	14:25	414.20	0	43.13
	13-Sep-10	13:15	413.66	5.0	42.75
	19-Sep-10	11:30	413.34	21.0	42.53
	26-Sep-10	9:50	399.08	76.0	32.66
Pressure plate:	4-Oct-10	13:05	359.55	337	5.29

Volume Adjusted Data [1]

	Matric Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calculated Porosity (%)
Hanging column:	0.0	---	---	---	---
	5.0	---	---	---	---
	21.0	---	---	---	---
	76.0	---	---	---	---
Pressure plate:	337	---	---	---	---

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent each of the volume change measurements obtained after saturated hydraulic conductivity testing and throughout hanging column/pressure plate testing. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

† Assumed density of water is 1.0 g/cm^3

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Technician Notes:

Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Moisture Retention Data

Dew Point Potentiometer / Relative Humidity Box

(Soil-Water Characteristic Curve)

Sample Number: Blend 4 (44%)

Initial sample bulk density (g/cm^3)*:* 1.64

Fraction of bulk sample used (<2.00mm fraction) (%): 100.00

Dry weight of dew point potentiometer sample* (g)*:* 181.08

Tare weight, jar (g)*:* 113.12

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)
Dew point potentiometer:	21-Oct-10	9:48	181.58	19274	1.21
	21-Oct-10	8:51	181.46	57619	0.92
	21-Oct-10	14:04	181.35	151440	0.65

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Dew point potentiometer:	19274	---	---	---	---
	57619	---	---	---	---
	151440	---	---	---	---

Dry weight of relative humidity box sample* (g)*:* 79.49

Tare weight (g)*:* 44.10

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)
Relative humidity box:	16-Sep-10	11:05	79.59	861325	0.48

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Relative humidity box:	861325	---	---	---	---

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent the volume change measurements obtained after the last hanging column or pressure plate point. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

† Adjusted for >2.00mm (#10 sieve) material not used in DPP/RH testing. Assumed moisture content of material >2.00mm is zero, and assumed density of water is 1.0 g/cm^3.

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

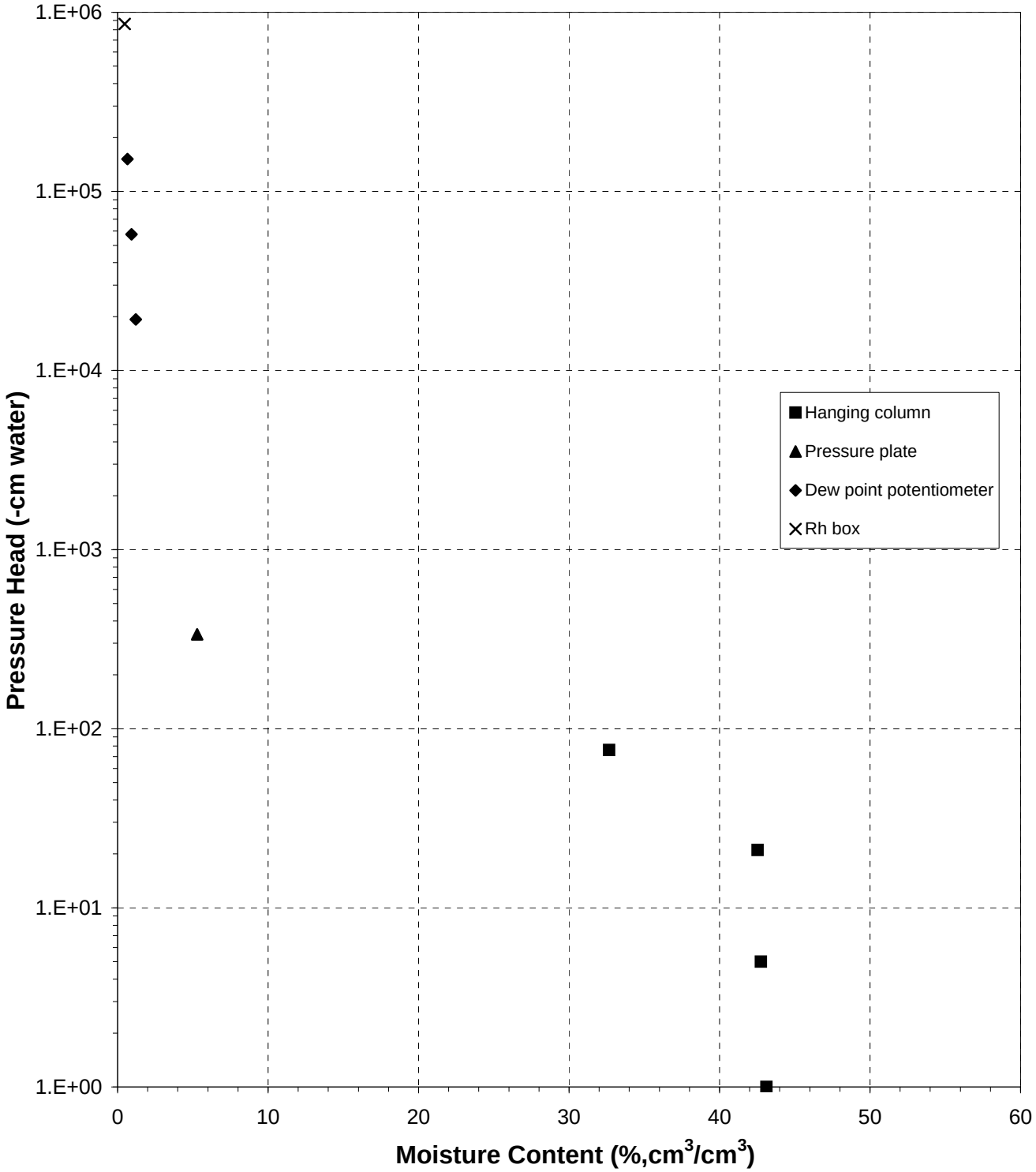
Laboratory analysis by: K. Wright/M. Vigil

Data entered by: K. Wright

Checked by: J. Hines

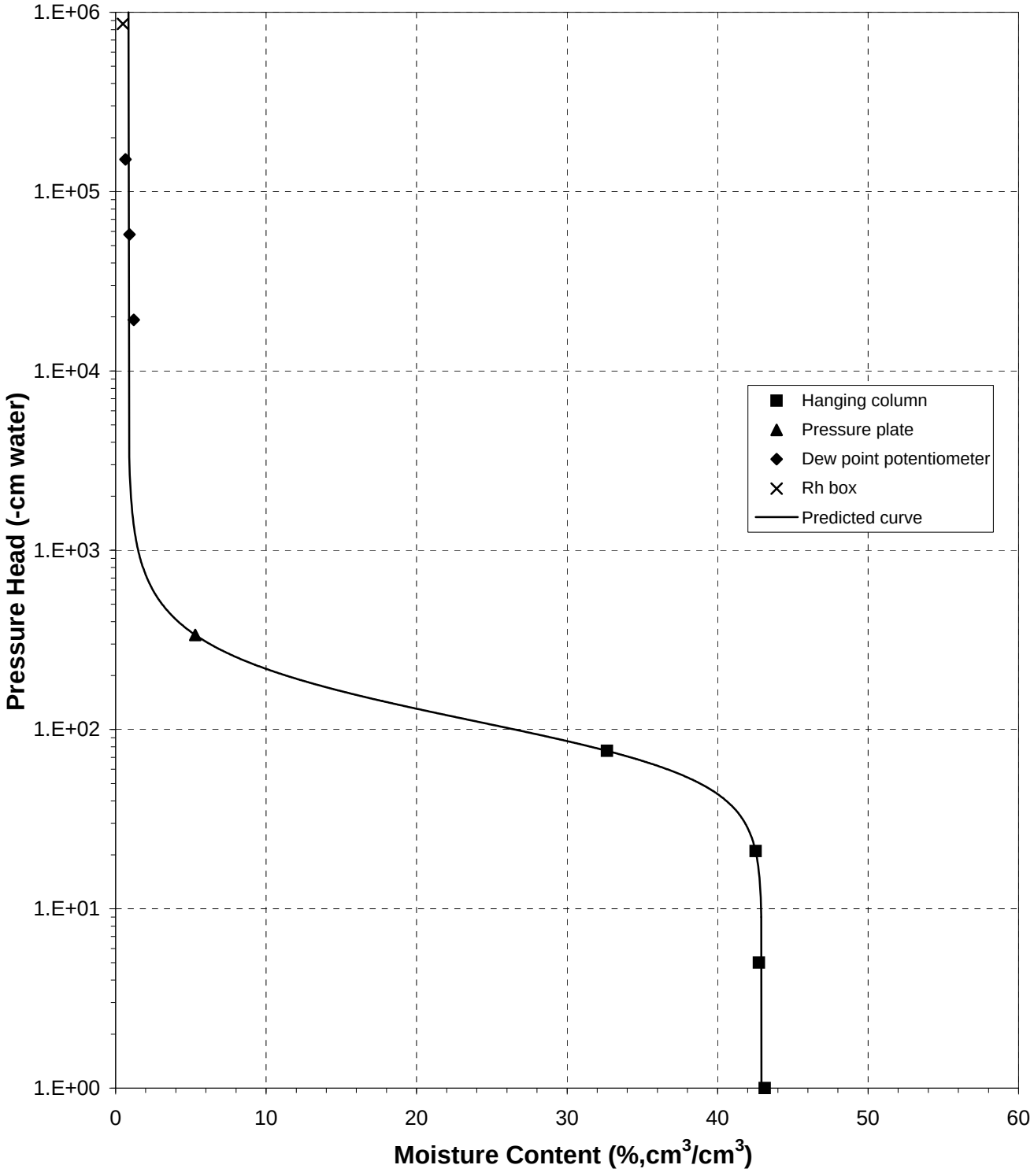
Water Retention Data Points

Sample Number: Blend 4 (44%)



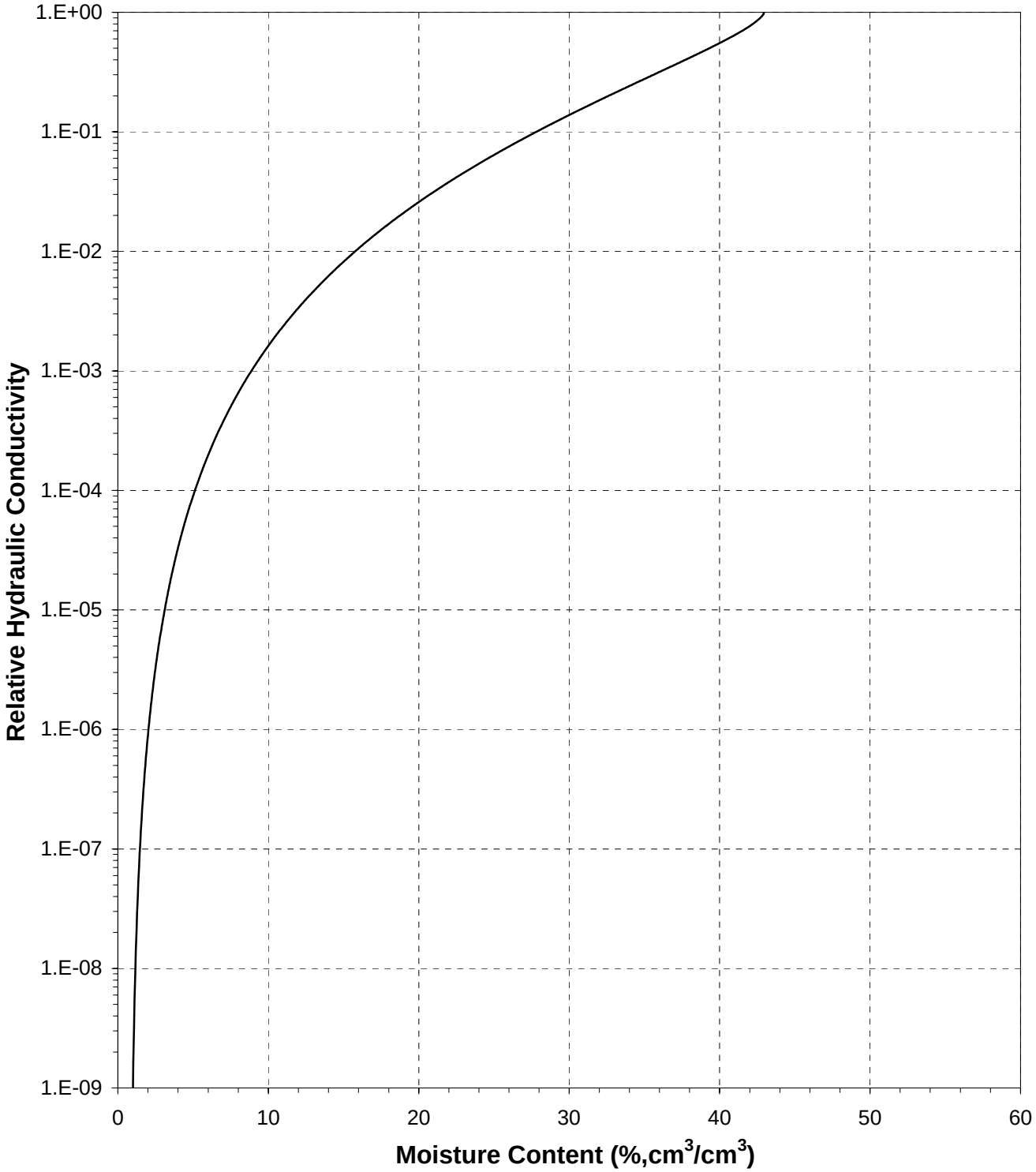
Predicted Water Retention Curve and Data Points

Sample Number: Blend 4 (44%)



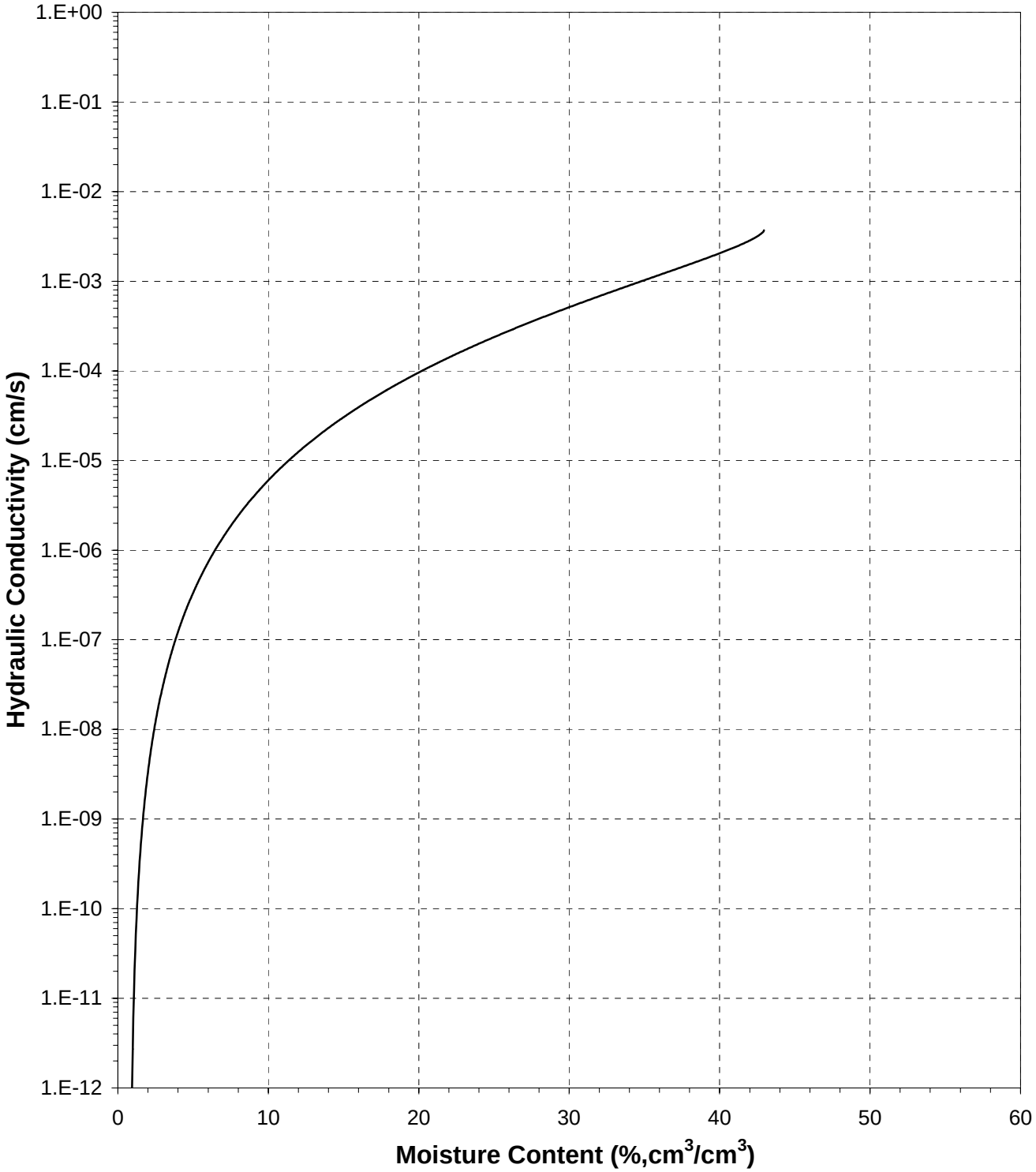
Plot of Relative Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 4 (44%)



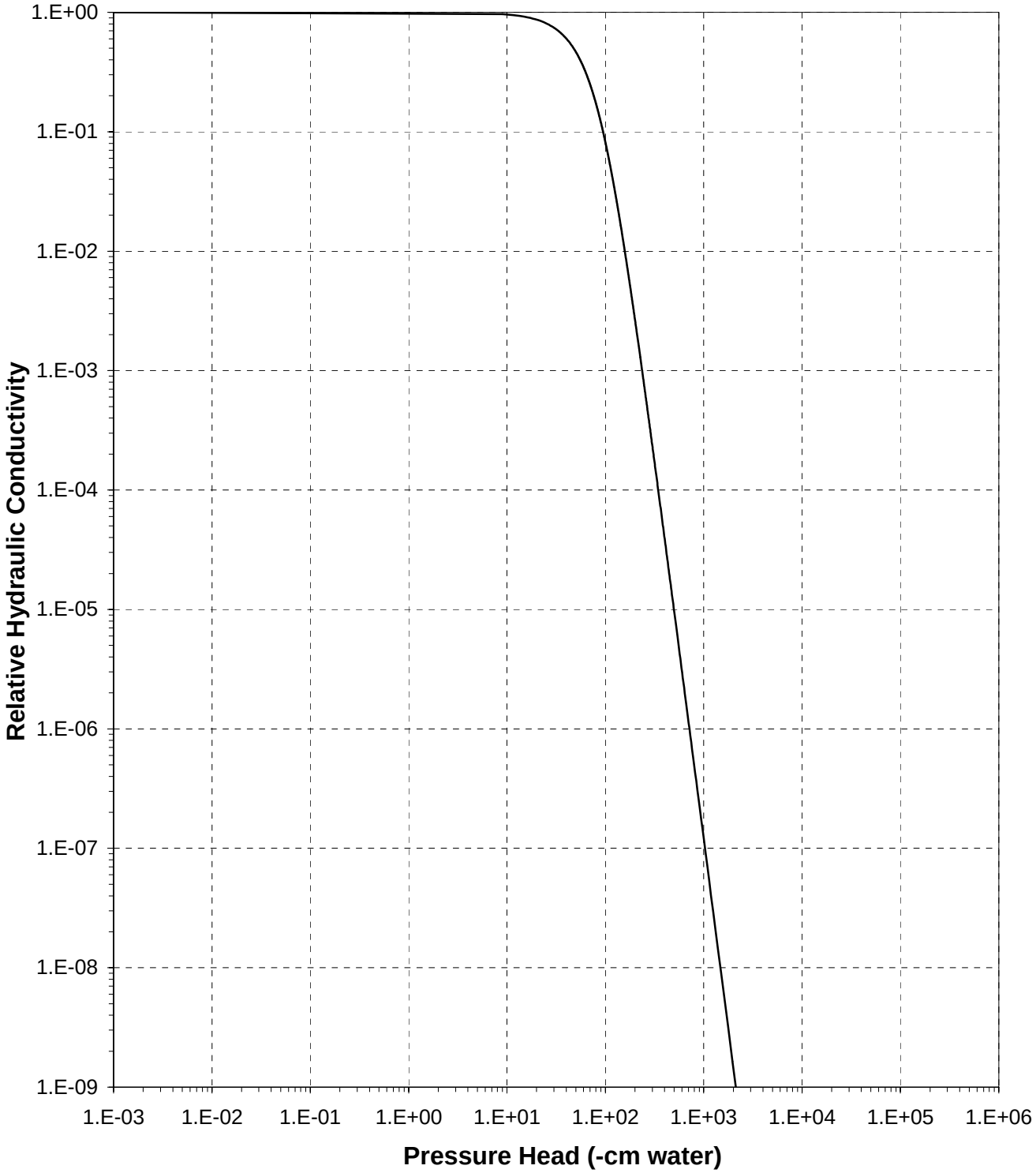
Plot of Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 4 (44%)



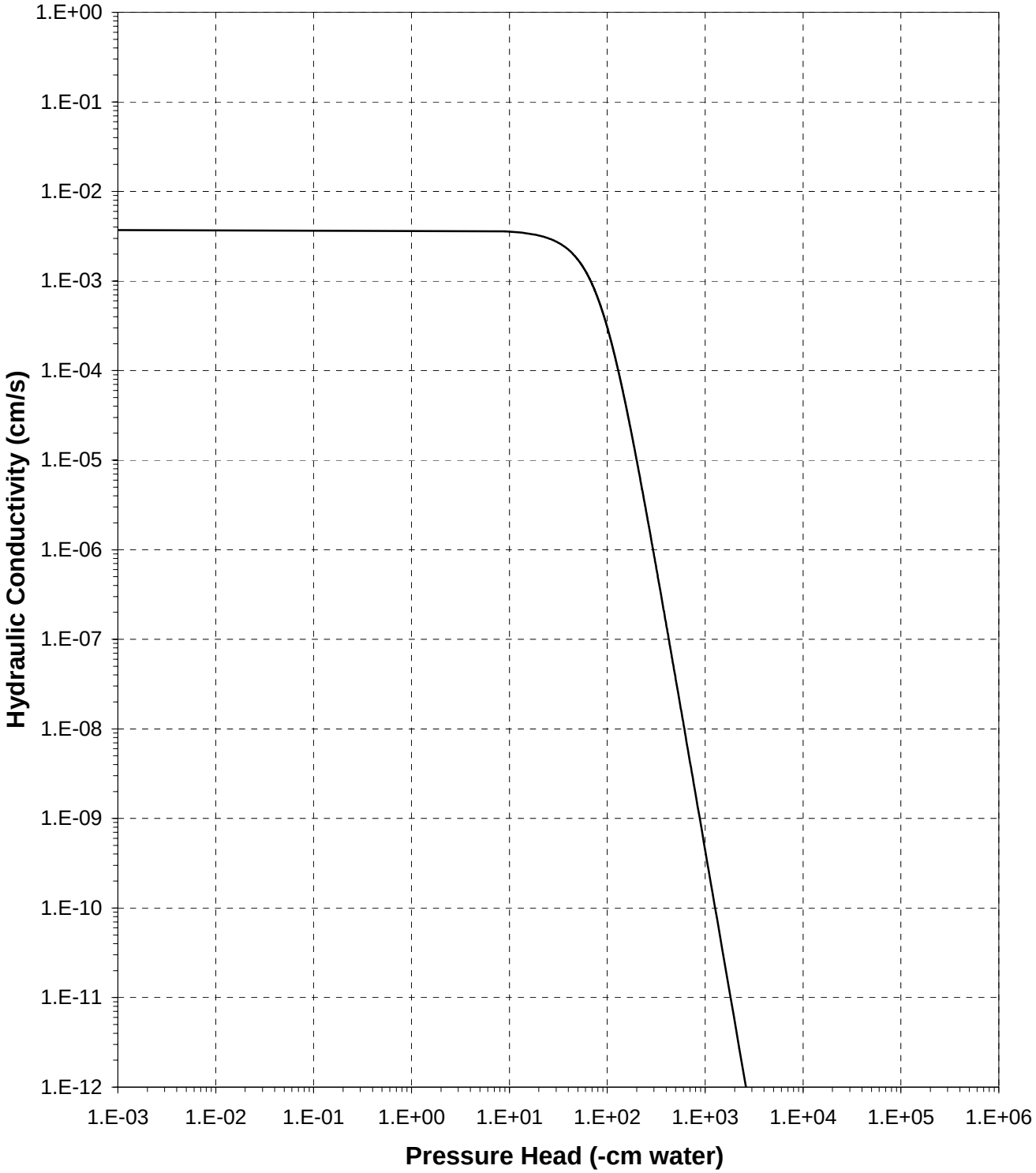
Plot of Relative Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 4 (44%)



Plot of Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 4 (44%)



Moisture Retention Data

Hanging Column / Pressure Plate

(Soil-Water Characteristic Curve)

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 4 (50%)
Ring Number: NA
Depth: NA

Dry wt. of sample (g)*:* 212.69
Tare wt., ring (g)*:* 88.63
Tare wt., screen & clamp (g)*:* 27.95
Initial sample volume (cm^3)*:* 143.72
Initial dry bulk density (g/cm^3)*:* 1.48
Measured particle density (g/cm^3)*:* 2.98
Initial calculated total porosity (%)*:* 50.26

	Date	Time	Weight* (g)	Matric Potential (-cm water)	Moisture Content† (% vol)	
Hanging column:	13-Sep-10	15:50	387.27	0	42.23	‡‡
	19-Sep-10	13:05	386.58	6.0	42.12	‡‡
	26-Sep-10	11:05	385.28	30.0	41.37	‡‡
	2-Oct-10	9:10	364.80	71.0	26.26	‡‡
Pressure plate:	10-Oct-10	9:15	335.29	337	4.46	‡‡

Volume Adjusted Data [1]

	Matric Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calculated Porosity (%)
Hanging column:	0.0	137.34	-4.44%	1.55	47.94
	6.0	136.07	-5.32%	1.56	47.46
	30.0	135.39	-5.80%	1.57	47.20
	71.0	135.29	-5.86%	1.57	47.16
Pressure plate:	337	134.92	-6.13%	1.58	47.01

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent each of the volume change measurements obtained after saturated hydraulic conductivity testing and throughout hanging column/pressure plate testing. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

† Assumed density of water is 1.0 g/cm^3

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Technician Notes:

Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Moisture Retention Data

Dew Point Potentiometer / Relative Humidity Box

(Soil-Water Characteristic Curve)

Sample Number: Blend 4 (50%)

Initial sample bulk density (g/cm^3)*:* 1.48
Fraction of bulk sample used (<2.00mm fraction) (%): 100.00

Dry weight of dew point potentiometer sample* (g)*:* 164.74
Tare weight, jar (g)*:* 113.21

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)	
Dew point potentiometer:	21-Oct-10	9:24	165.16	16929	1.44	‡‡
	21-Oct-10	8:42	165.04	46197	1.03	‡‡
	21-Oct-10	11:35	164.96	265658	0.76	‡‡

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Dew point potentiometer:	16929	120.23	-16.34%	1.77	40.54
	46197	120.23	-16.34%	1.77	40.54
	265658	120.23	-16.34%	1.77	40.54

Dry weight of relative humidity box sample* (g)*:* 72.51
Tare weight (g)*:* 41.63

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)	
Relative humidity box:	16-Sep-10	11:05	72.58	861325	0.38	‡‡

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Relative humidity box:	861325	134.92	-6.13%	1.58	47.01

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent the volume change measurements obtained after the last hanging column or pressure plate point. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

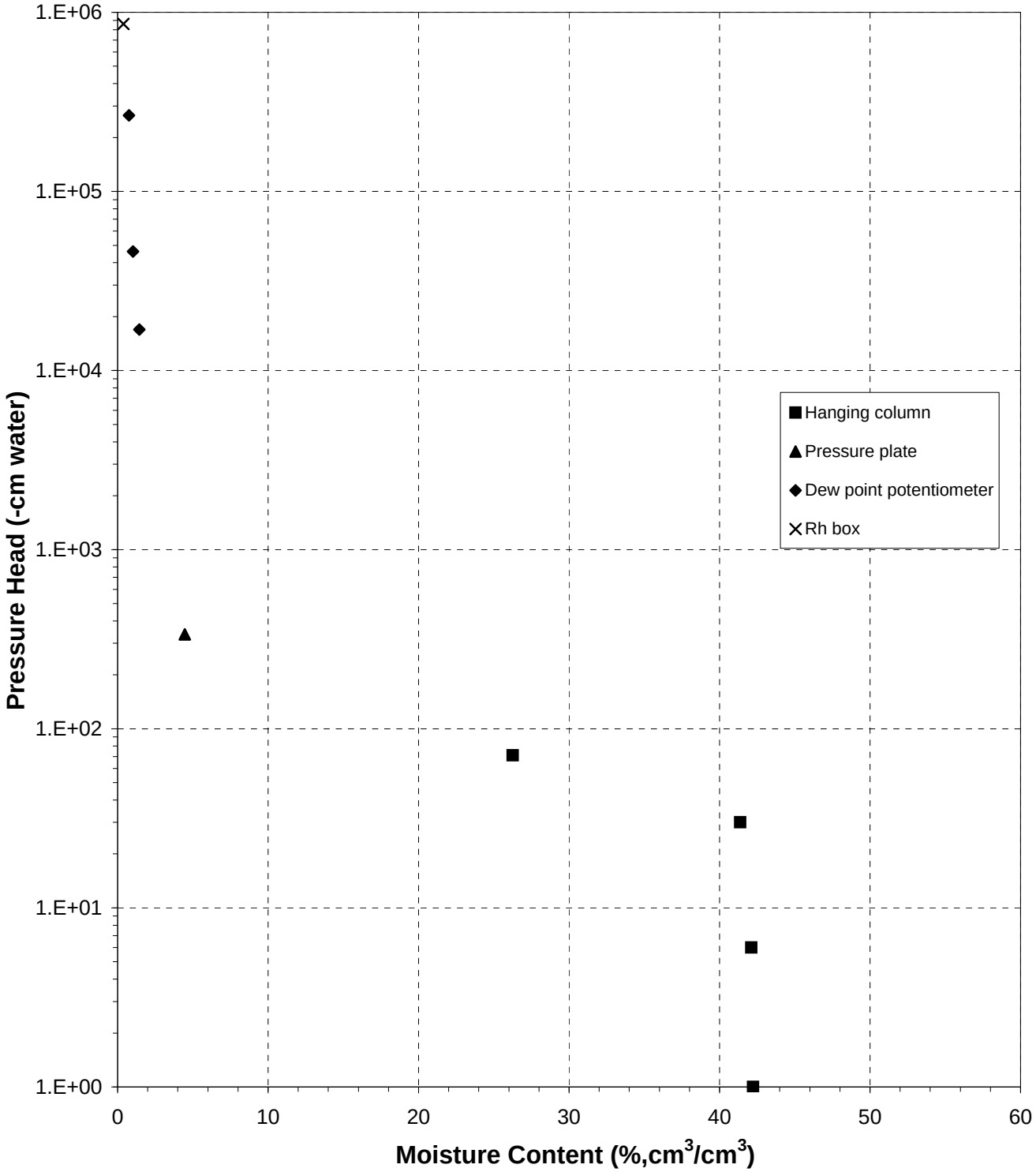
† Adjusted for >2.00mm (#10 sieve) material not used in DPP/RH testing. Assumed moisture content of material >2.00mm is zero, and assumed density of water is 1.0 g/cm^3.

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Laboratory analysis by: K. Wright/M. Vigil
Data entered by: K. Wright
Checked by: J. Hines

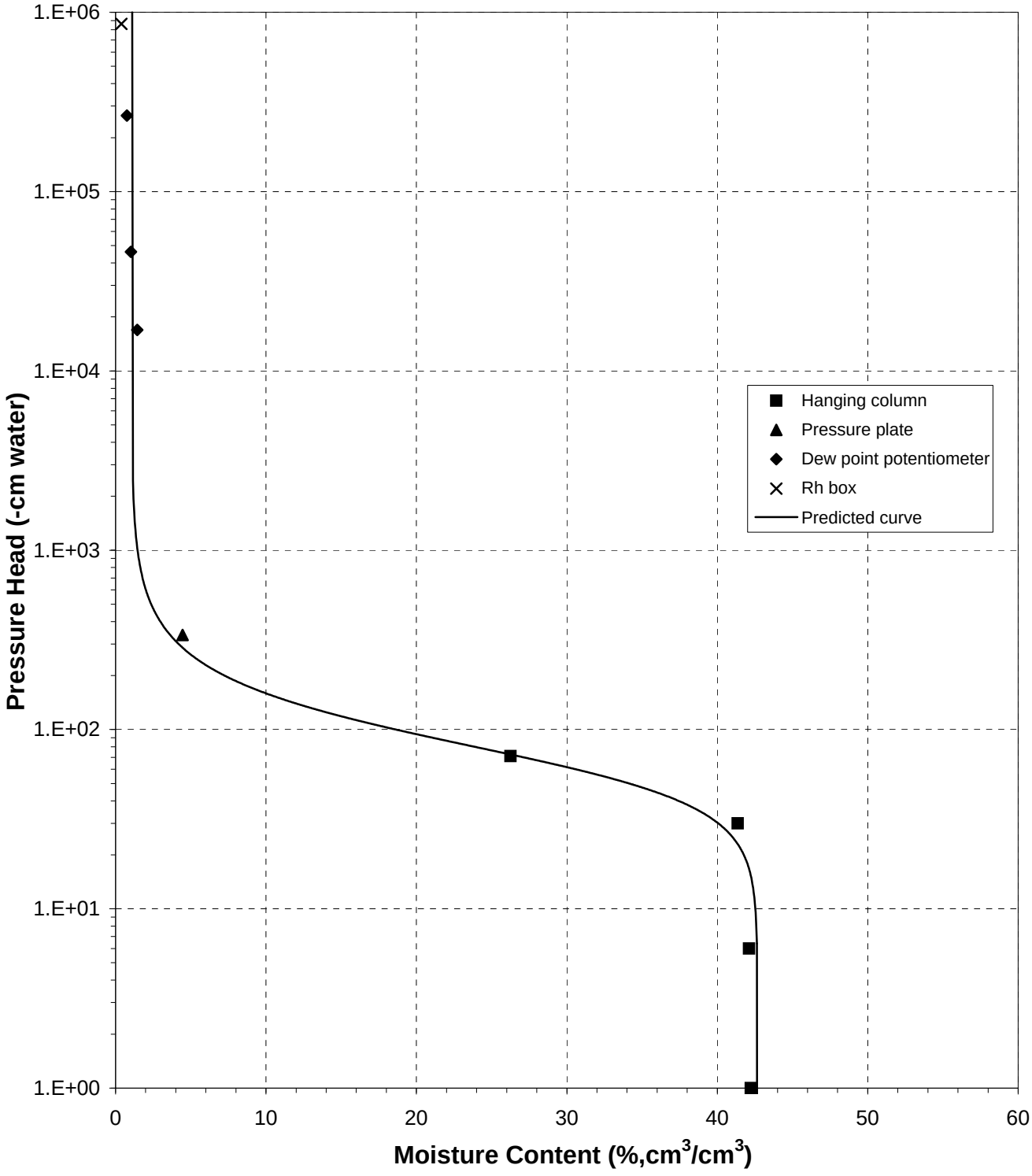
Water Retention Data Points

Sample Number: Blend 4 (50%)



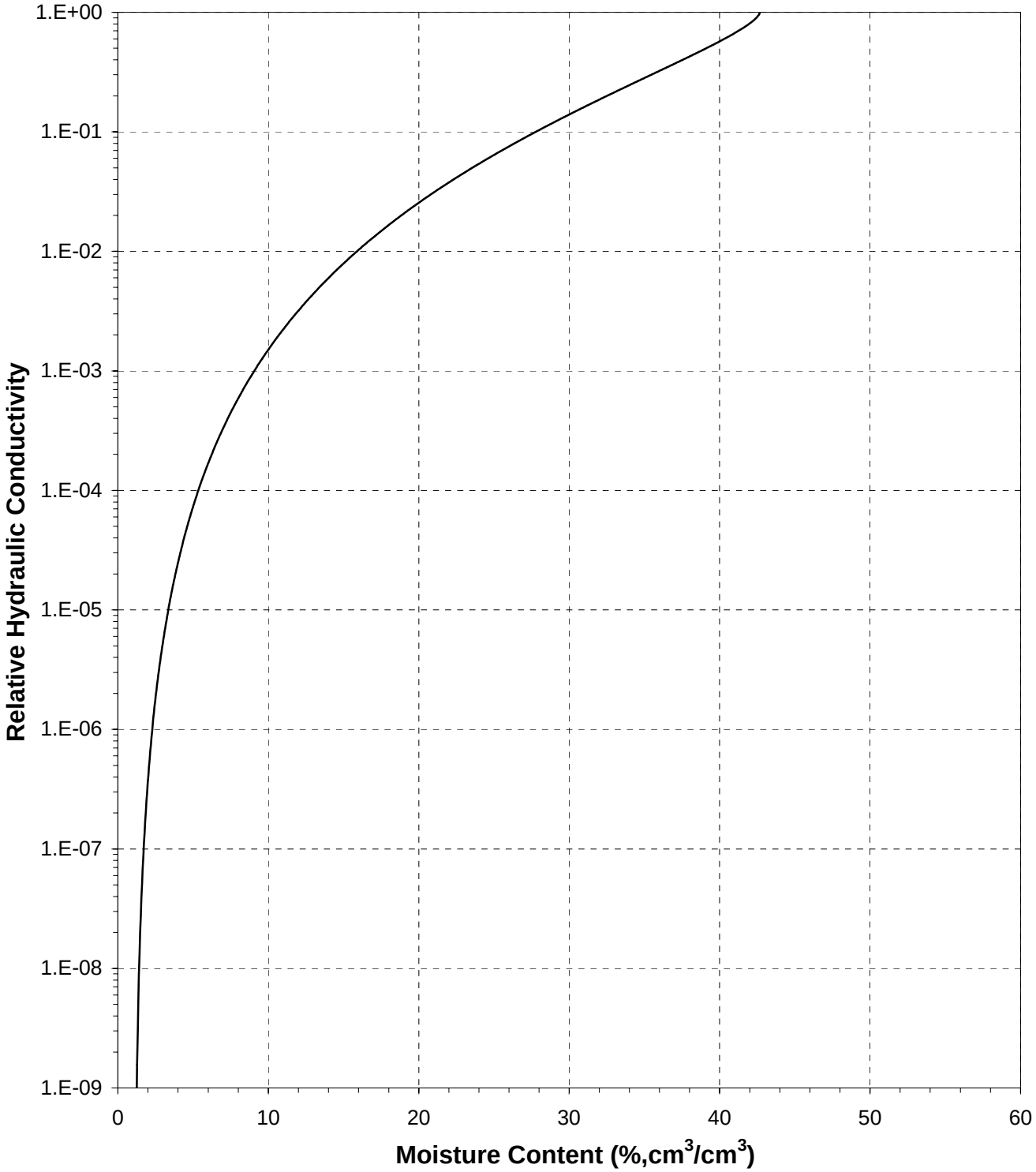
Predicted Water Retention Curve and Data Points

Sample Number: Blend 4 (50%)



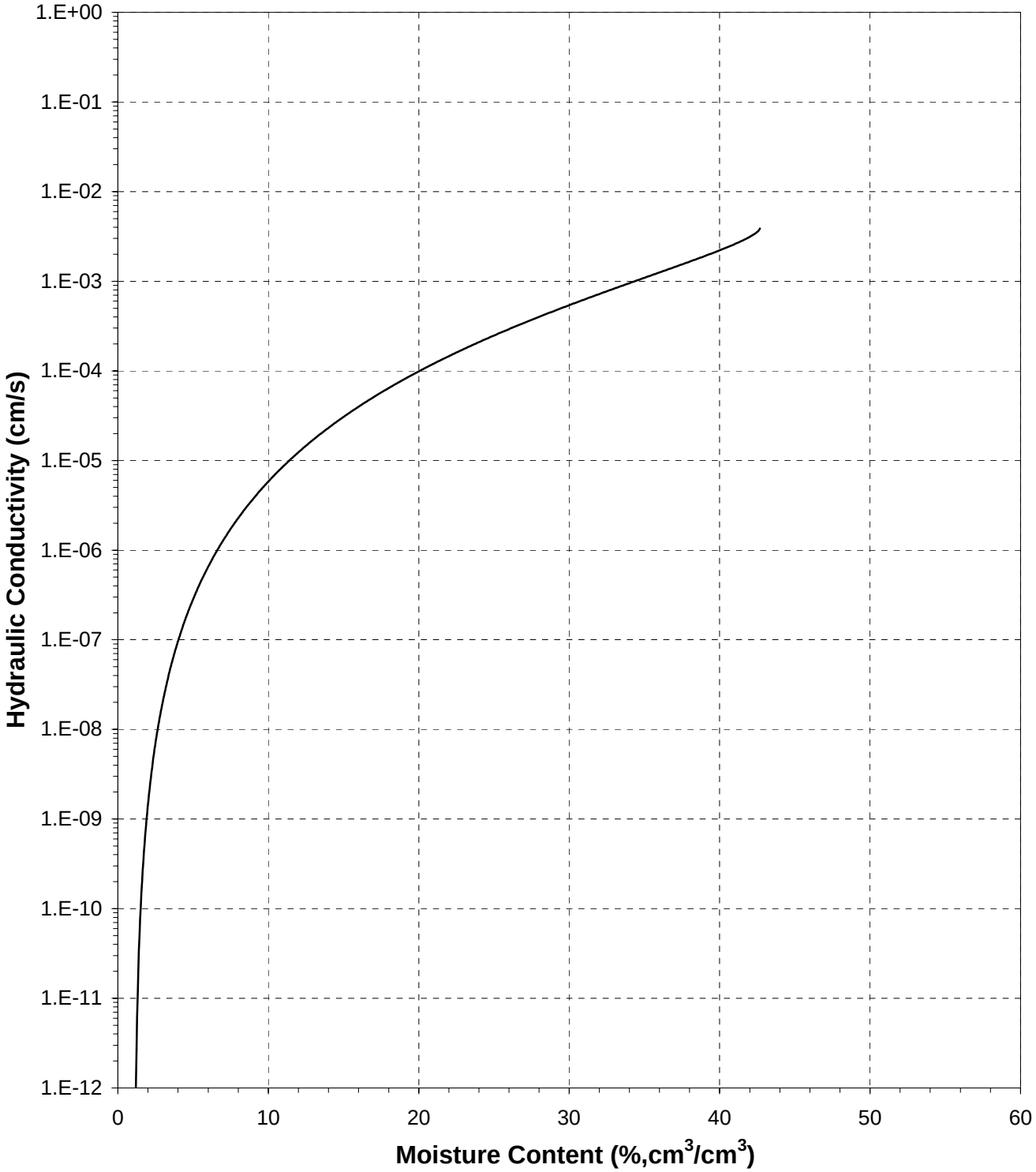
Plot of Relative Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 4 (50%)



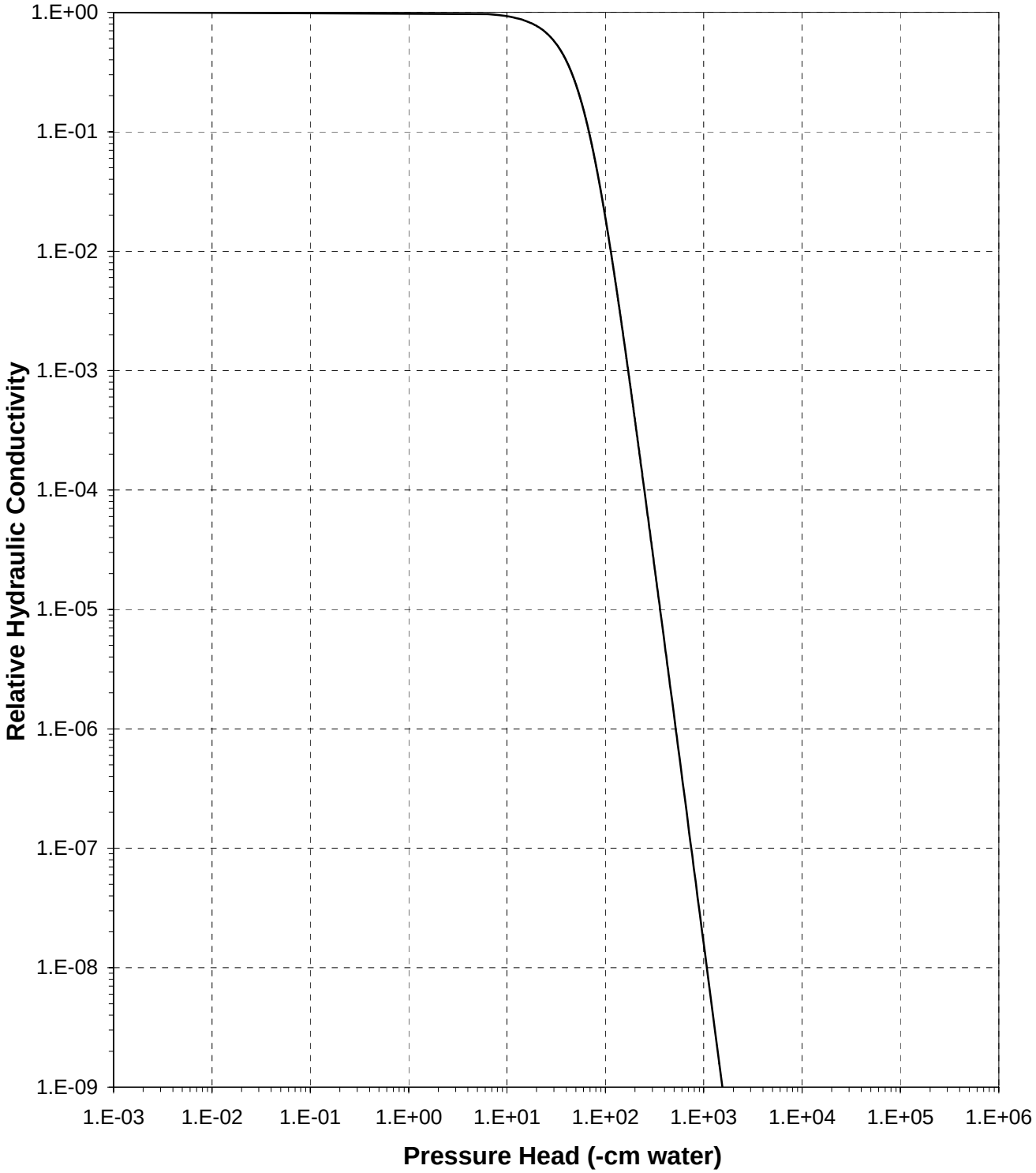
Plot of Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 4 (50%)



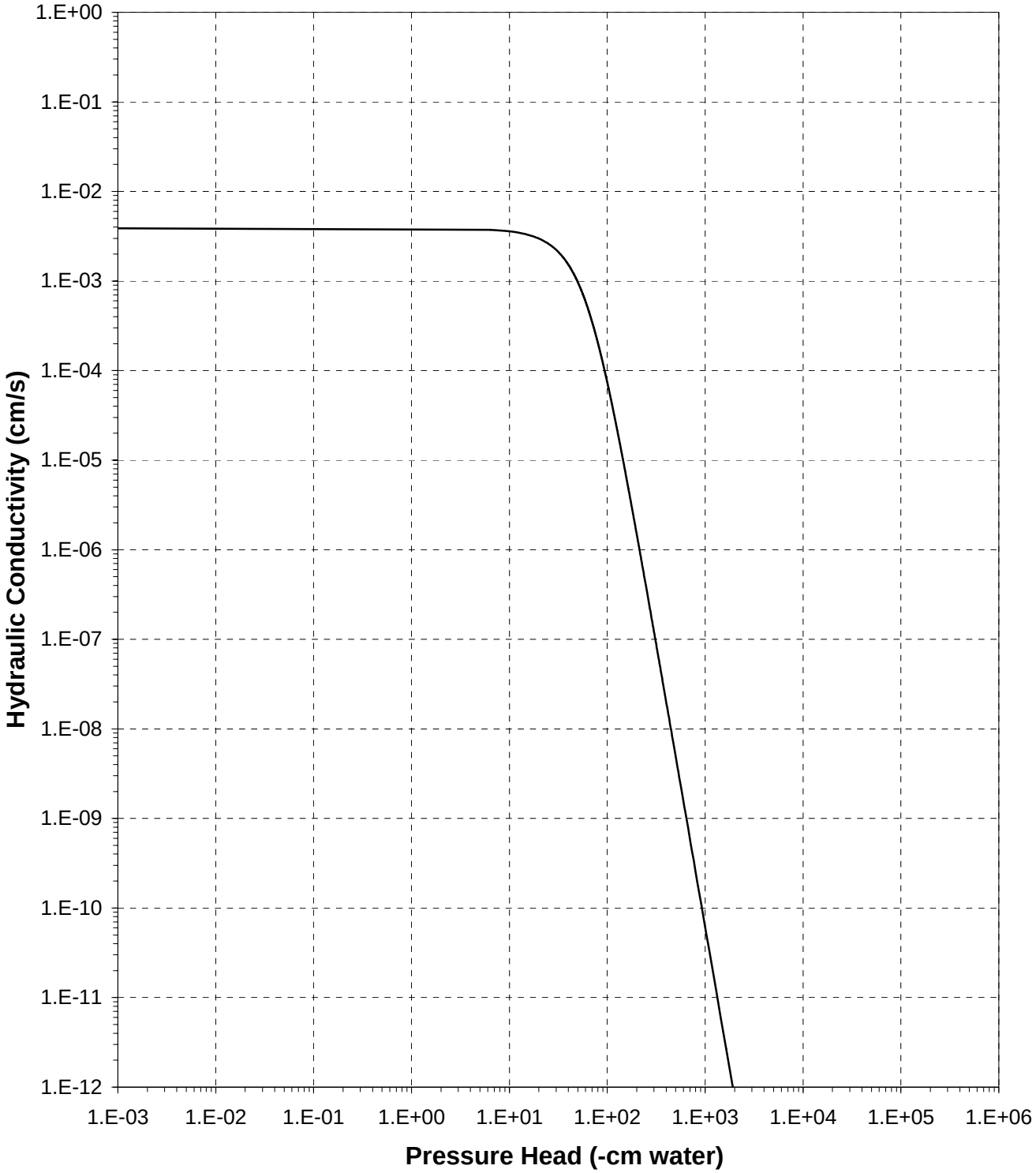
Plot of Relative Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 4 (50%)



Plot of Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 4 (50%)



Moisture Retention Data

Hanging Column / Pressure Plate

(Soil-Water Characteristic Curve)

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 4 (55%)
Ring Number: NA
Depth: NA

Dry wt. of sample (g)*:* 192.87
Tare wt., ring (g)*:* 88.93
Tare wt., screen & clamp (g)*:* 29.07
Initial sample volume (cm^3)*:* 144.26
Initial dry bulk density (g/cm^3)*:* 1.34
Measured particle density (g/cm^3)*:* 2.98
Initial calculated total porosity (%)*:* 55.06

	Date	Time	Weight* (g)	Matric Potential (-cm water)	Moisture Content† (% vol)	
Hanging column:	13-Sep-10	15:45	363.11	0	41.60	‡‡
	19-Sep-10	13:10	360.86	6.0	40.74	‡‡
	26-Sep-10	11:15	359.62	30.0	39.78	‡‡
	2-Oct-10	9:15	346.67	71.0	29.27	‡‡
Pressure plate:	10-Oct-10	9:15	316.60	337	4.74	‡‡

Volume Adjusted Data [1]

	Matric Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calculated Porosity (%)
Hanging column:	0.0	125.57	-12.95%	1.54	48.37
	6.0	122.72	-14.93%	1.57	47.17
	30.0	122.55	-15.05%	1.57	47.10
	71.0	122.32	-15.21%	1.58	47.00
Pressure plate:	337	120.92	-16.18%	1.60	46.39

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent each of the volume change measurements obtained after saturated hydraulic conductivity testing and throughout hanging column/pressure plate testing. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

† Assumed density of water is 1.0 g/cm^3

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Technician Notes:

Laboratory analysis by: K. Wright
Data entered by: K. Wright
Checked by: J. Hines

Moisture Retention Data

Dew Point Potentiometer / Relative Humidity Box

(Soil-Water Characteristic Curve)

Sample Number: Blend 4 (55%)

Initial sample bulk density (g/cm^3)*:* 1.34
Fraction of bulk sample used (<2.00mm fraction) (%): 100.00

Dry weight of dew point potentiometer sample* (g)*:* 152.16
Tare weight, jar (g)*:* 117.16

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)	
Dew point potentiometer:	19-Oct-10	13:04	152.58	6323	2.00	‡‡
	20-Oct-10	9:20	152.29	66695	0.62	‡‡
	20-Oct-10	13:05	152.31	105345	0.72	‡‡

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Dew point potentiometer:	6323	114.77	-20.44%	1.68	43.52
	66695	114.77	-20.44%	1.68	43.52
	105345	114.77	-20.44%	1.68	43.52

Dry weight of relative humidity box sample* (g)*:* 73.55
Tare weight (g)*:* 42.30

	Date	Time	Weight* (g)	Water Potential (-cm water)	Moisture Content † (% vol)	
Relative humidity box:	16-Sep-10	11:05	73.64	861325	0.45	‡‡

Volume Adjusted Data [1]

	Water Potential (-cm water)	Adjusted Volume (cm^3)	% Volume Change [2] (%)	Adjusted Density (g/cm^3)	Adjusted Calc. Porosity (%)
Relative humidity box:	861325	120.92	-16.18%	1.60	46.39

Comments:

[1] Applicable if the sample experienced volume changes during testing. 'Volume Adjusted' values represent the volume change measurements obtained after the last hanging column or pressure plate point. "---" indicates no volume changes occurred.

[2] Represents percent volume change from original sample volume. A '+' denotes measured sample swelling, a '-' denotes measured sample settling, and '---' denotes no volume change occurred.

* Weight including tares

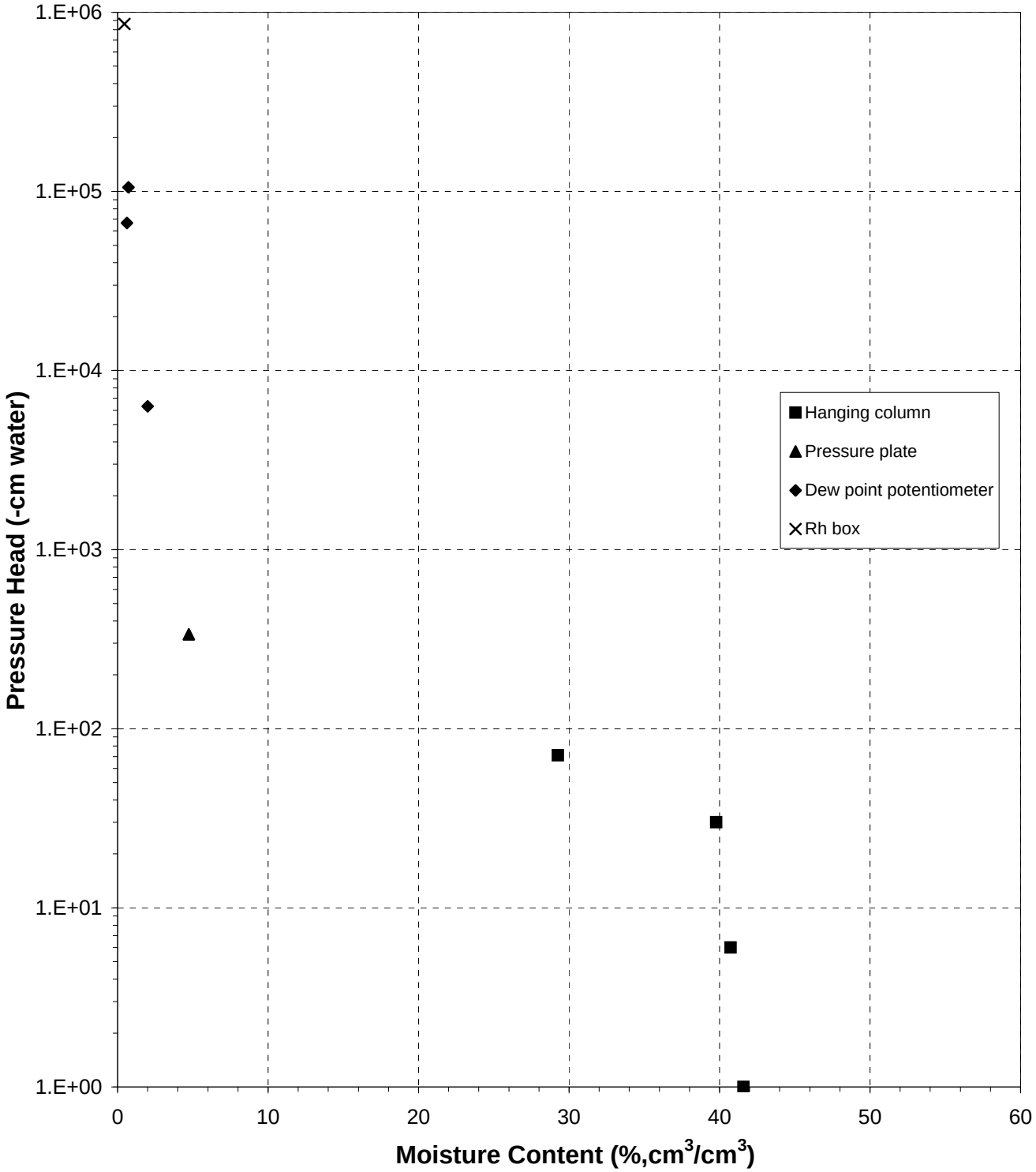
† Adjusted for >2.00mm (#10 sieve) material not used in DPP/RH testing. Assumed moisture content of material >2.00mm is zero, and assumed density of water is 1.0 g/cm^3.

‡‡ Volume adjustments are applicable at this matric potential (see comment #1).

Laboratory analysis by: K. Wright/M. Vigil
Data entered by: K. Wright
Checked by: J. Hines

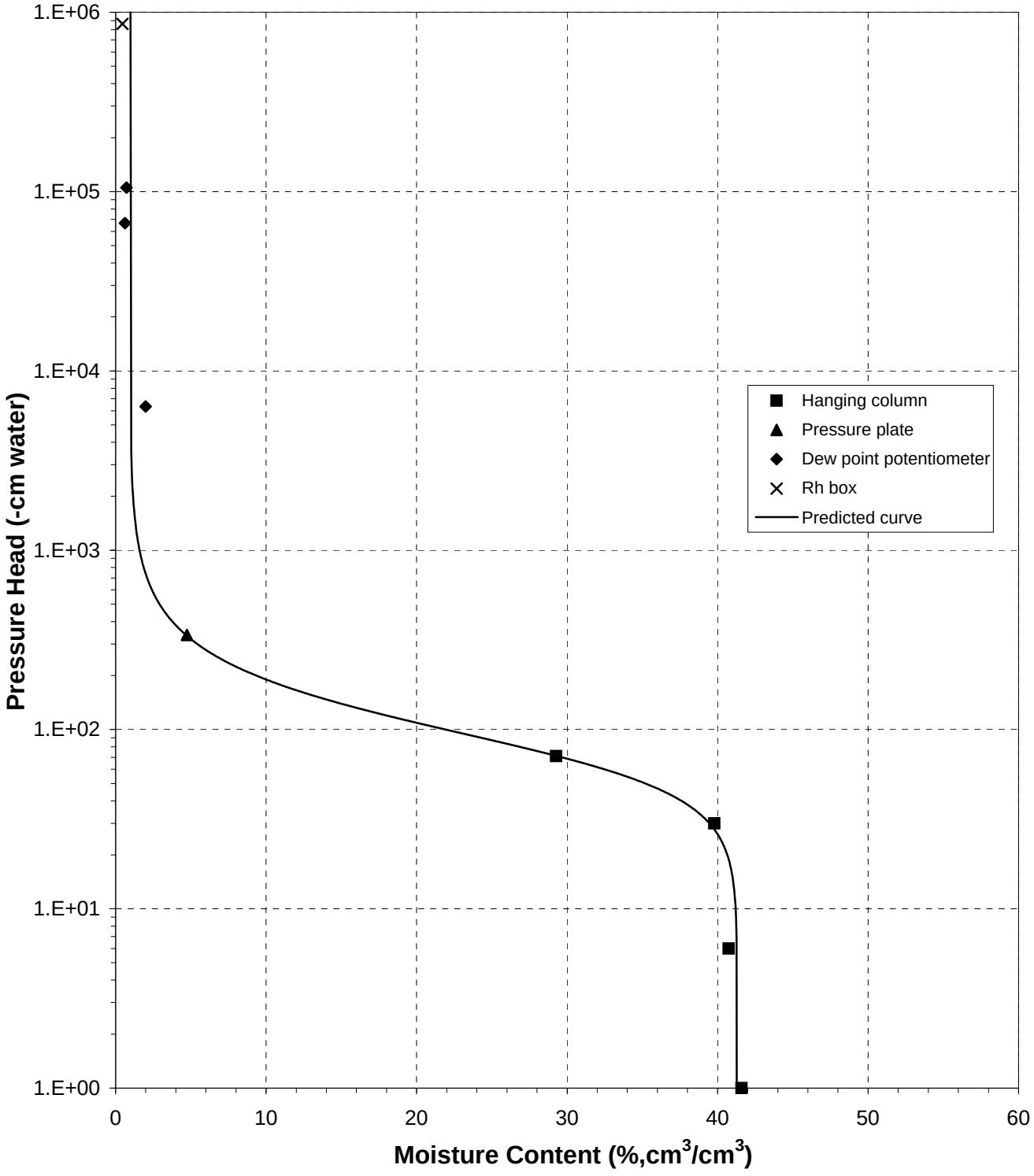
Water Retention Data Points

Sample Number: Blend 4 (55%)



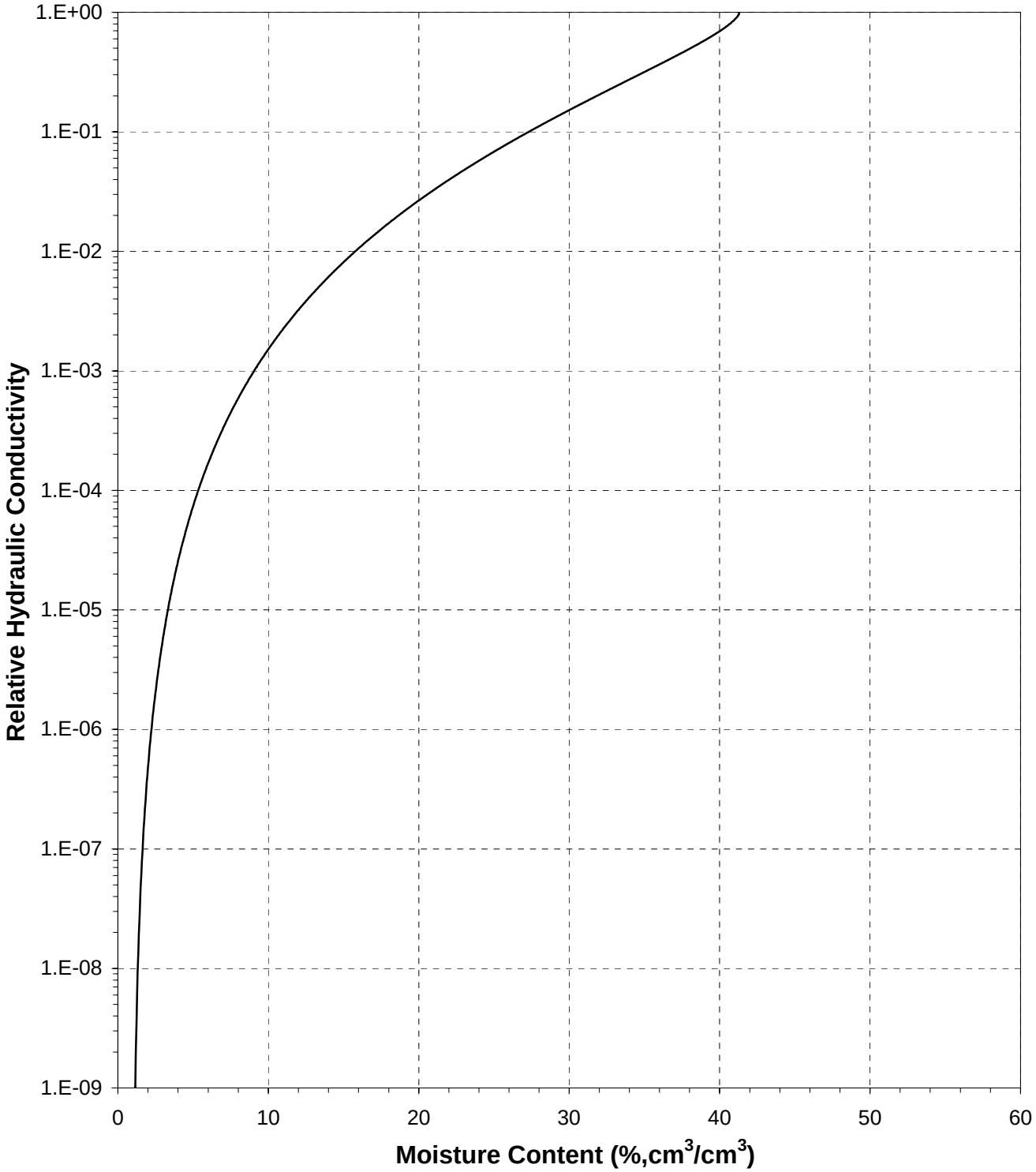
Predicted Water Retention Curve and Data Points

Sample Number: Blend 4 (55%)



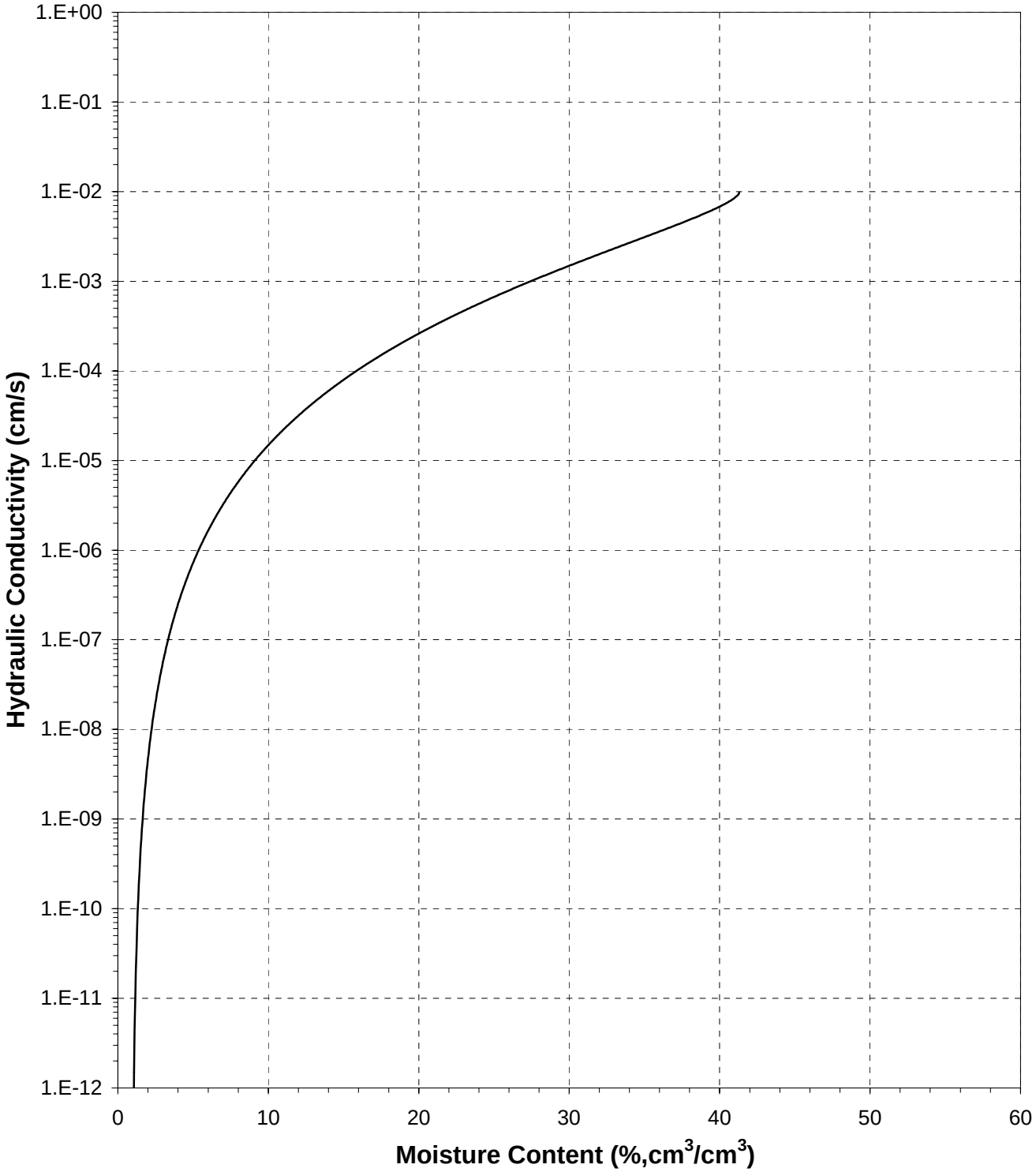
Plot of Relative Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 4 (55%)



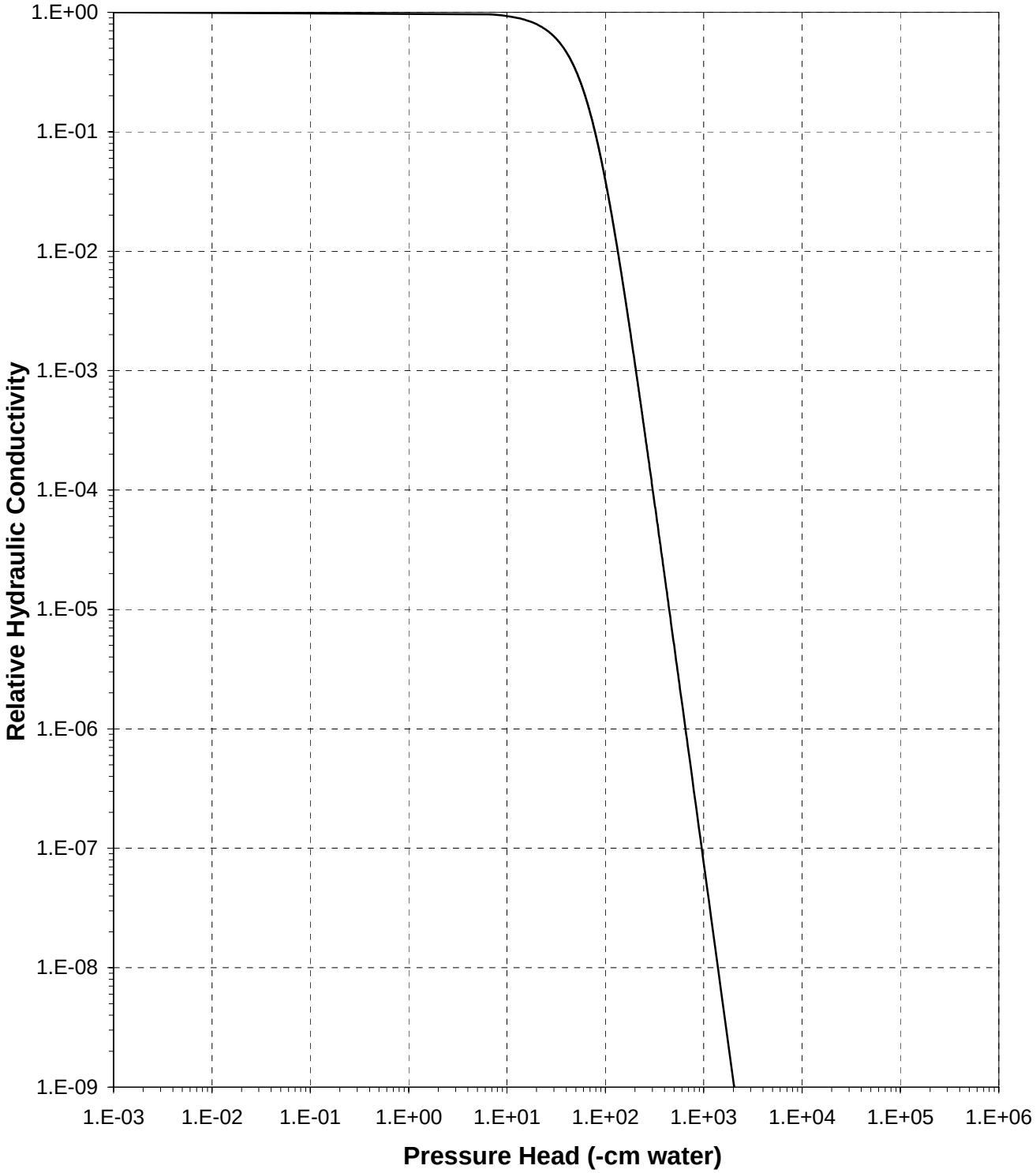
Plot of Hydraulic Conductivity vs Moisture Content

Sample Number: Blend 4 (55%)



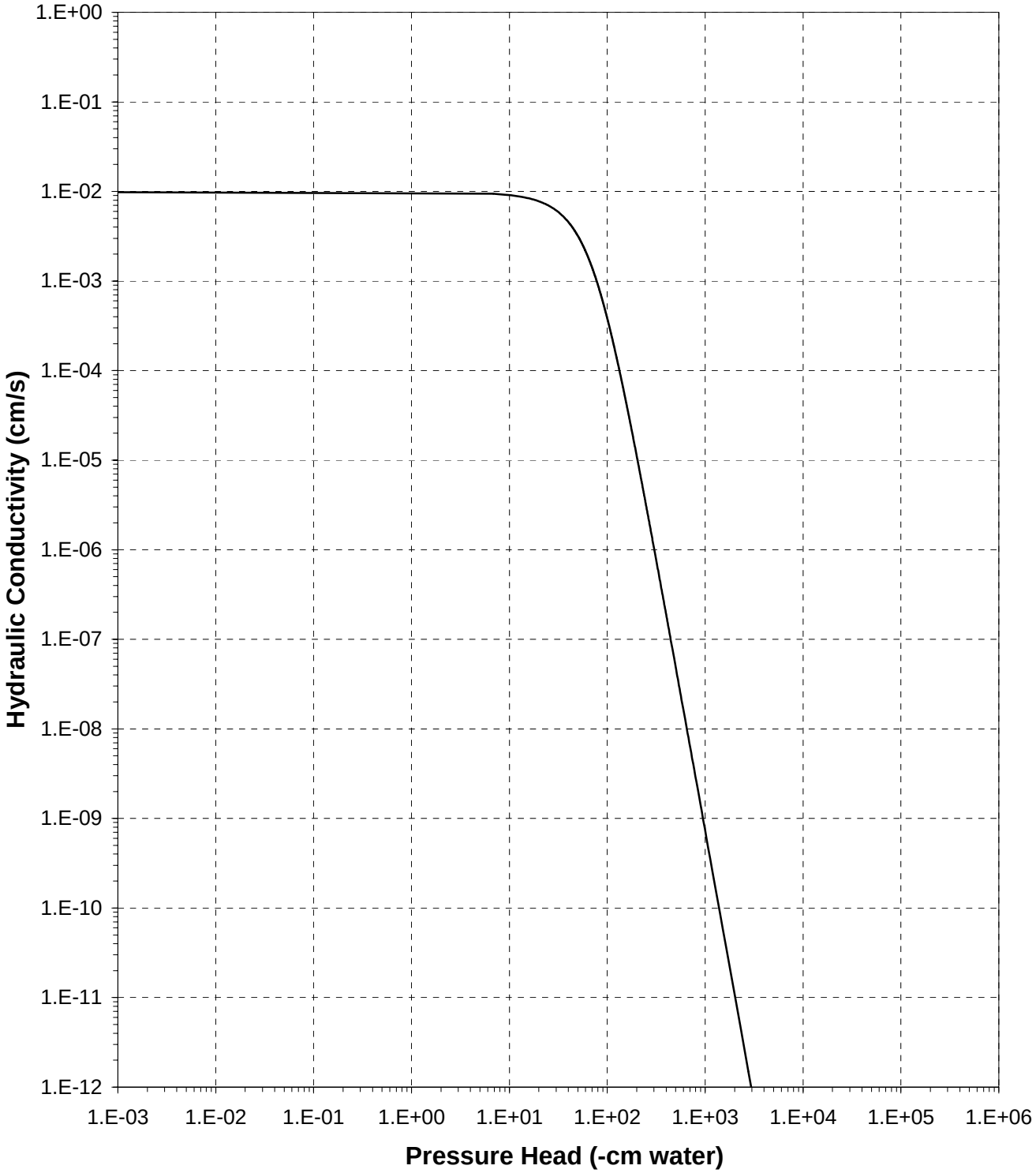
Plot of Relative Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 4 (55%)



Plot of Hydraulic Conductivity vs Pressure Head

Sample Number: Blend 4 (55%)



Particle Size Analysis

Summary of Particle Size Characteristics

Sample Number	d_{10} (mm)	d_{50} (mm)	d_{60} (mm)	C_u	C_c	Method	ASTM Classification	USDA Classification
As Received	0.010	0.076	0.096	9.6	2.1	WS/H	Silty sand (SM)	Sandy Loam
Blend 1	0.0037	0.037	0.048	13	1.6	WS/H	Silt with sand (ML)s	Silt Loam
Blend 2	0.0056	0.062	0.082	15	2.2	WS/H	Sandy silt s(ML)	Sandy Loam
Blend 3	0.013	0.092	0.11	8.5	2.4	WS/H	Silty sand (SM)	Loamy Sand
Blend 4	0.051	0.11	0.14	2.7	1.0	WS/H	Silty sand (SM)	Sand

d_{50} = Median particle diameter

Est = Reported values for d_{10}, C_u, C_c, and soil classification are estimates, since extrapolation was required to obtain the d_{10} diameter

$$C_u = \frac{d_{60}}{d_{10}}$$

$$C_c = \frac{(d_{30})^2}{(d_{10})(d_{60})}$$

DS = Dry sieve

H = Hydrometer

WS = Wet sieve

† Greater than 10% of sample is coarse material

Percent Gravel, Sand, Silt and Clay*

Sample Number	% Gravel (>4.75mm)	% Sand (<4.75mm, >0.075mm)	% Silt (<0.075mm, >0.002mm)	% Clay (<0.002mm)
As Received	0.0	50.3	43.8	5.9
Blend 1	0.0	22.9	70.0	7.1
Blend 2	0.0	43.5	51.1	5.4
Blend 3	0.0	62.6	34.0	3.4
Blend 4	0.0	78.6	19.3	2.1

*USCS classification does not classify clay fraction based on particle size. USDA definition of clay (<0.002mm) used in this table.

Particle Size Analysis
Wet Sieve Data (#10 Split)

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: As Received
Ring Number: NA
Depth: NA
Test Date: 22-Aug-10

Initial Dry Weight of Sample (g): 494.56
Weight Passing #10 (g): 494.56
Weight Retained #10 (g): 0.00
Weight of Hydrometer Sample (g): 46.39
Calculated Weight of Sieve Sample (g): 46.39

Shape: Angular
Hardness: Soft

Test Fraction	Sieve Number	Diameter (mm)	Wt. Retained	Cum Wt. Retained	Wt. Passing	% Passing
+10						
	3"	75	0.00	0.00	494.56	100.00
	2"	50	0.00	0.00	494.56	100.00
	1.5"	38.1	0.00	0.00	494.56	100.00
	1"	25	0.00	0.00	494.56	100.00
	3/4"	19.0	0.00	0.00	494.56	100.00
	3/8"	9.5	0.00	0.00	494.56	100.00
	4	4.75	0.00	0.00	494.56	100.00
	10	2.00	0.00	0.00	494.56	100.00
-10			(Based on calculated sieve wt.)			
	20	0.85	0.02	0.02	46.37	99.96
	40	0.425	0.08	0.10	46.29	99.78
	60	0.250	1.28	1.38	45.01	97.03
	140	0.106	15.37	16.75	29.64	63.89
	200	0.075	6.59	23.34	23.05	49.69
	dry pan		0.89	24.23	22.16	
	wet pan			22.16	0.00	

d_{10} (mm): 0.010
d_{16} (mm): 0.022
d_{30} (mm): 0.045
d_{50} (mm): 0.076
d_{60} (mm): 0.096
d_{84} (mm): 0.18

Median Particle Diameter --d_{50} (mm): 0.076
Uniformity Coefficient, Cu --[d_{60}/d_{10}] (mm): 9.6
Coefficient of Curvature, Cc --[$(d_{30})^2/(d_{10}*d_{60})$] (mm): 2.1
Mean Particle Diameter --[$(d_{16}+d_{50}+d_{84})/3$] (mm): 0.093

Classification of fines (visual method): ML

ASTM Soil Classification: Silty sand (SM)
USDA Soil Classification: Sandy Loam

Laboratory analysis by: K. Wright
Data entered by: M. Vigil
Checked by: J. Hines

Particle Size Analysis
Hydrometer Data

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: As Received
Ring Number: NA
Depth: NA

Test Date: 18-Aug-10
Start Time: 9:12

Type of Water Used: DISTILLED
Reaction with H_2O_2*:* NA
Dispersant:* $(NaPO_3)_6$
Measured particle density: 2.95

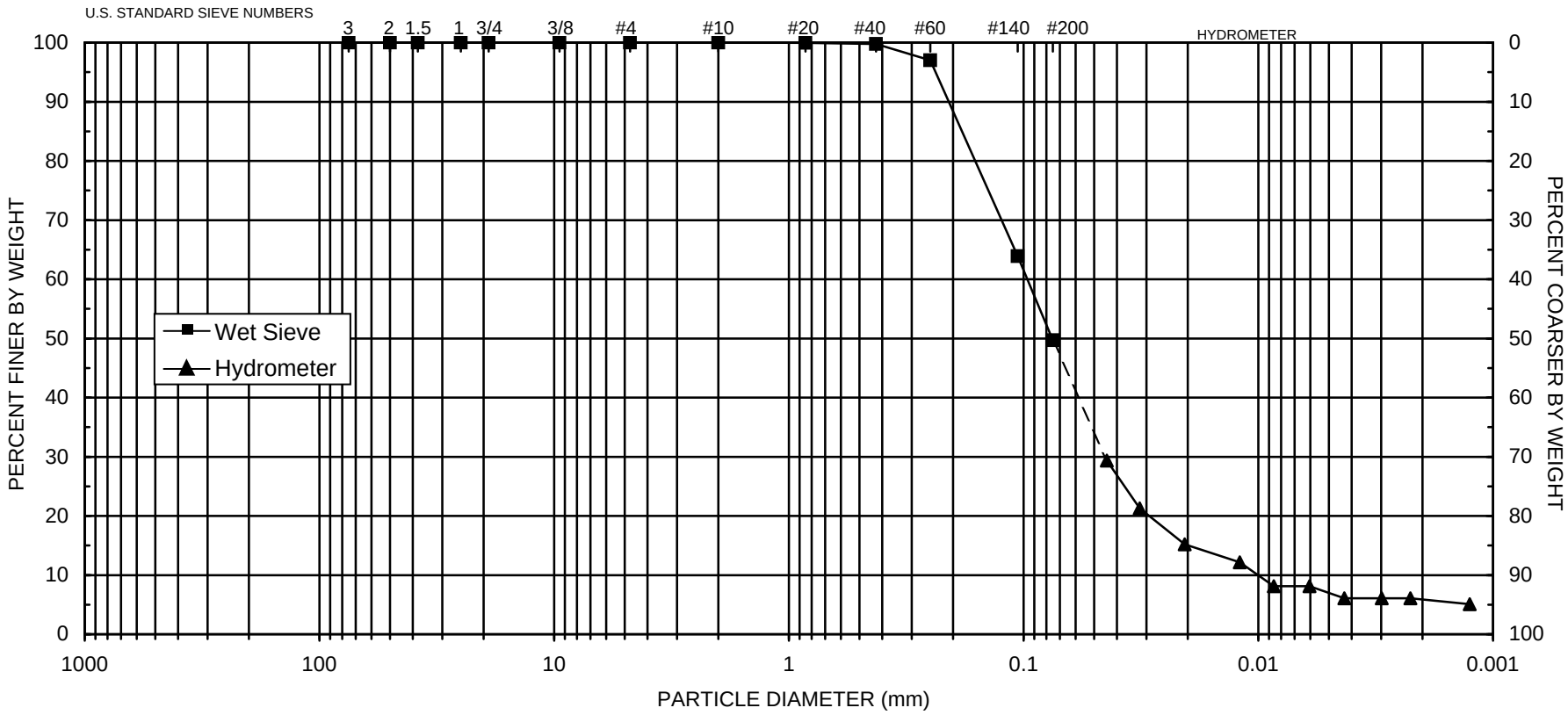
Initial Wt. (g)*:* 46.39
Total Sample Wt. (g)*:* 494.56
Wt. Passing #10 (g): 494.56

Date	Time (min)	Temp (°C)	R (g/L)	R_L (g/L)	R_{corr} (g/L)	L (cm)	D (mm)	P (%)	% Finer
18-Aug-10	1	21.8	20.5	6.0	14.5	12.9	0.04416	29.4	29.4
	2	21.8	16.5	6.0	10.5	13.6	0.03200	21.3	21.3
	5	21.8	13.5	6.0	7.5	14.1	0.02060	15.2	15.2
	15	21.8	12.0	6.0	6.0	14.3	0.01200	12.2	12.2
	30	21.8	10.0	6.0	4.0	14.7	0.00858	8.1	8.1
	60	21.9	10.0	6.0	4.0	14.7	0.00606	8.1	8.1
	120	22.0	9.0	6.0	3.0	14.8	0.00430	6.1	6.1
	250	22.0	9.0	6.0	3.0	14.8	0.00298	6.1	6.1
	440	22.0	9.0	6.0	3.0	14.8	0.00225	6.1	6.1
19-Aug-10	1425	21.8	8.5	6.0	2.5	14.9	0.00126	5.1	5.1

Comments:

* Dispersion device: mechanically operated stirring device

Laboratory analysis by: K. Wright
Data entered by: M. Vigil
Checked by: J. Hines



UNIFIED	COBBLES	GRAVEL		SAND			SILT OR CLAY				
		Coarse	Fine	Coarse	Medium	Fine					
USDA	COBBLES	GRAVEL		SAND				SILT	CLAY		
				Very coarse	Coarse	Medium	Fine	Very fine			

d_{10} = **0.010**	d_{30} = **0.045**	d_{50} = **0.076**	d_{60} = **0.096**	C_u = **9.6**	C_c = **2.1**

SAMPLE NUMBER	DEPTH	ASTM CLASSIFICATION	USDA CLASSIFICATION
As Received	**NA**	**Silty sand (SM)**	**Sandy Loam**

Particle Size Analysis
Wet Sieve Data (#10 Split)

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 1
Ring Number: NA
Depth: NA
Test Date: 7-Sep-10

Initial Dry Weight of Sample (g): 52.03
Weight Passing #10 (g): 52.03
Weight Retained #10 (g): 0.00
Weight of Hydrometer Sample (g): 52.03
Calculated Weight of Sieve Sample (g): 52.03

Shape: Angular
Hardness: Soft

Test Fraction	Sieve Number	Diameter (mm)	Wt. Retained	Cum Wt. Retained	Wt. Passing	% Passing
+10						
	3"	75	0.00	0.00	52.03	100.00
	2"	50	0.00	0.00	52.03	100.00
	1.5"	38.1	0.00	0.00	52.03	100.00
	1"	25	0.00	0.00	52.03	100.00
	3/4"	19.0	0.00	0.00	52.03	100.00
	3/8"	9.5	0.00	0.00	52.03	100.00
	4	4.75	0.00	0.00	52.03	100.00
	10	2.00	0.00	0.00	52.03	100.00
-10			(Based on calculated sieve wt.)			
	20	0.85	0.00	0.00	52.03	100.00
	40	0.425	0.03	0.03	52.00	99.94
	60	0.250	0.80	0.83	51.20	98.40
	140	0.106	7.87	8.70	43.33	83.28
	200	0.075	3.24	11.94	40.09	77.05
	dry pan		0.81	12.75	39.28	
	wet pan			39.28	0.00	

d_{10} (mm): 0.0037
d_{16} (mm): 0.0072
d_{30} (mm): 0.017
d_{50} (mm): 0.037
d_{60} (mm): 0.048
d_{84} (mm): 0.11

Median Particle Diameter --d_{50} (mm): 0.037
Uniformity Coefficient, Cu --$[d_{60}/d_{10}]$ (mm): 13
Coefficient of Curvature, Cc --$[(d_{30})^2/(d_{10}*d_{60})]$ (mm): 1.6
Mean Particle Diameter --$[(d_{16}+d_{50}+d_{84})/3]$ (mm): 0.051

Classification of fines (visual method): ML

ASTM Soil Classification: Silt with sand (ML)s
USDA Soil Classification: Silt Loam

Laboratory analysis by: J. Ayarbe/M. Vigil
Data entered by: C. Krous
Checked by: J. Hines

Particle Size Analysis
Hydrometer Data

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 1
Ring Number: NA
Depth: NA
Test Date: 2-Sep-10
Start Time: 9:00

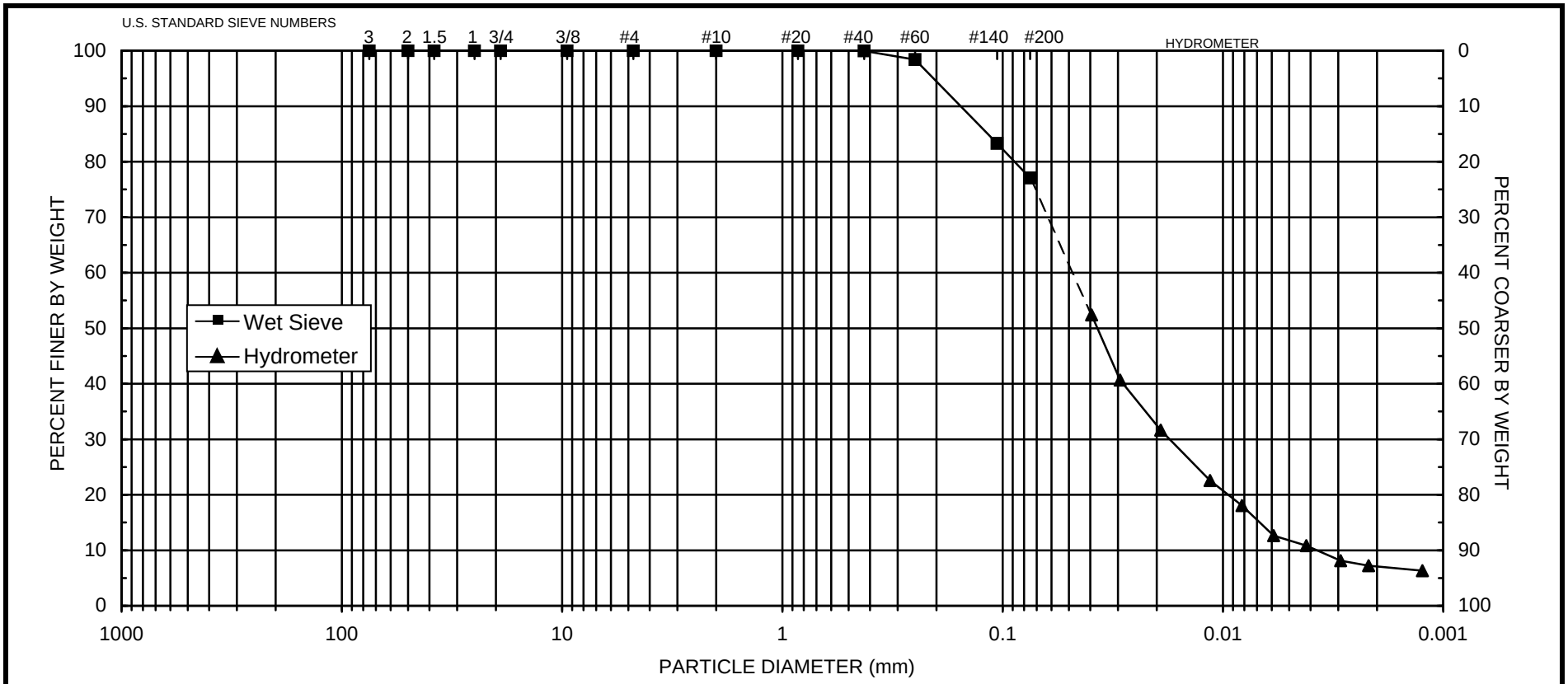
Type of Water Used: DISTILLED
Reaction with H_2O_2*:* NA
Dispersant:* $(NaPO_3)_6$
Measured particle density: 2.99
Initial Wt. (g)*:* 52.03
Total Sample Wt. (g)*:* 52.03
Wt. Passing #10 (g): 52.03

Date	Time (min)	Temp (°C)	R (g/L)	R_L (g/L)	R_{corr} (g/L)	L (cm)	D (mm)	P (%)	% Finer
2-Sep-10	1	21.7	35.0	6.0	29.0	10.6	0.03951	52.4	52.4
	2	21.7	28.5	6.0	22.5	11.6	0.02931	40.6	40.6
	5	21.7	23.5	6.0	17.5	12.4	0.01918	31.6	31.6
	15	21.7	18.5	6.0	12.5	13.3	0.01143	22.6	22.6
	30	21.7	16.0	6.0	10.0	13.7	0.00821	18.1	18.1
	60	21.9	13.0	6.0	7.0	14.2	0.00589	12.6	12.6
	120	22.0	12.0	6.0	6.0	14.3	0.00419	10.8	10.8
	250	22.0	10.5	6.0	4.5	14.6	0.00293	8.1	8.1
	450	22.1	10.0	6.0	4.0	14.7	0.00218	7.2	7.2
3-Sep-10	1410	21.7	9.5	6.0	3.5	14.7	0.00124	6.3	6.3

Comments:

* Dispersion device: mechanically operated stirring device

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines



UNIFIED	COBBLES	GRAVEL		SAND			SILT OR CLAY			
		Coarse	Fine	Coarse	Medium	Fine				
USDA	COBBLES	GRAVEL		SAND				SILT	CLAY	
				Very coarse	Coarse	Medium	Fine	Very fine		

d_{10} = **0.0037**	d_{30} = **0.017**	d_{50} = **0.037**	d_{60} = **0.048**	C_u = **13**	C_c = **1.6**

SAMPLE NUMBER	DEPTH	ASTM CLASSIFICATION	USDA CLASSIFICATION
Blend 1	**NA**	**Silt with sand (ML)s**	**Silt Loam**

Particle Size Analysis
Wet Sieve Data (#10 Split)

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 2
Ring Number: NA
Depth: NA
Test Date: 7-Sep-10

Initial Dry Weight of Sample (g): 49.18
Weight Passing #10 (g): 49.18
Weight Retained #10 (g): 0.00
Weight of Hydrometer Sample (g): 49.18
Calculated Weight of Sieve Sample (g): 49.18

Shape: Angular
Hardness: Soft

Test Fraction	Sieve Number	Diameter (mm)	Wt. Retained	Cum Wt. Retained	Wt. Passing	% Passing
+10						
	3"	75	0.00	0.00	49.18	100.00
	2"	50	0.00	0.00	49.18	100.00
	1.5"	38.1	0.00	0.00	49.18	100.00
	1"	25	0.00	0.00	49.18	100.00
	3/4"	19.0	0.00	0.00	49.18	100.00
	3/8"	9.5	0.00	0.00	49.18	100.00
	4	4.75	0.00	0.00	49.18	100.00
	10	2.00	0.00	0.00	49.18	100.00
-10			(Based on calculated sieve wt.)			
	20	0.85	0.00	0.00	49.18	100.00
	40	0.425	0.07	0.07	49.11	99.86
	60	0.250	1.14	1.21	47.97	97.54
	140	0.106	13.86	15.07	34.11	69.36
	200	0.075	6.34	21.41	27.77	56.47
	dry pan		2.08	23.49	25.69	
	wet pan			25.69	0.00	

d_{10} (mm): 0.0056
d_{16} (mm): 0.012
d_{30} (mm): 0.032
d_{50} (mm): 0.062
d_{60} (mm): 0.082
d_{84} (mm): 0.17

Median Particle Diameter --d_{50} (mm): 0.062
Uniformity Coefficient, Cu --[d_{60}/d_{10}] (mm): 15
Coefficient of Curvature, Cc --[$(d_{30})^2/(d_{10}*d_{60})$] (mm): 2.2
Mean Particle Diameter --[$(d_{16}+d_{50}+d_{84})/3$] (mm): 0.081

Classification of fines (visual method): ML

ASTM Soil Classification: Sandy silt s(ML)
USDA Soil Classification: Sandy Loam

Laboratory analysis by: D. O'Dowd/M. Vigil
Data entered by: C. Krous
Checked by: J. Hines

Particle Size Analysis
Hydrometer Data

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 2
Ring Number: NA
Depth: NA
Test Date: 2-Sep-10
Start Time: 9:06

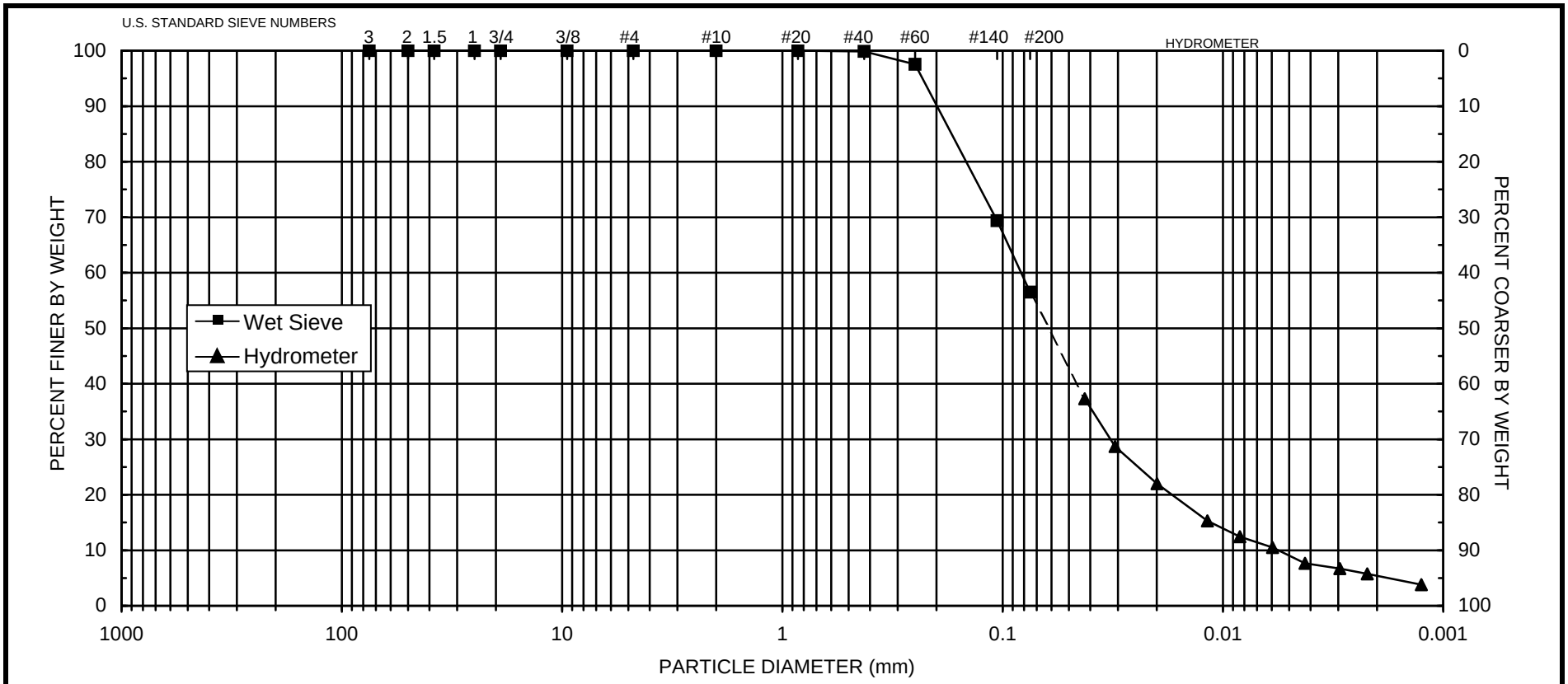
Type of Water Used: DISTILLED
Reaction with H_2O_2*:* NA
Dispersant:* $(NaPO_3)_6$
Measured particle density: 2.98
Initial Wt. (g)*:* 49.18
Total Sample Wt. (g)*:* 49.18
Wt. Passing #10 (g): 49.18

Date	Time (min)	Temp (°C)	R (g/L)	R_L (g/L)	R_{corr} (g/L)	L (cm)	D (mm)	P (%)	% Finer
2-Sep-10	1	21.7	25.5	6.0	19.5	12.1	0.04237	37.3	37.3
	2	21.7	21.0	6.0	15.0	12.9	0.03086	28.7	28.7
	5	21.7	17.5	6.0	11.5	13.4	0.01995	22.0	22.0
	15	21.7	14.0	6.0	8.0	14.0	0.01176	15.3	15.3
	30	21.8	12.5	6.0	6.5	14.3	0.00838	12.4	12.4
	60	21.9	11.5	6.0	5.5	14.4	0.00595	10.5	10.5
	120	22.0	10.0	6.0	4.0	14.7	0.00424	7.6	7.6
	250	22.0	9.5	6.0	3.5	14.7	0.00295	6.7	6.7
	445	22.1	9.0	6.0	3.0	14.8	0.00221	5.7	5.7
3-Sep-10	1405	21.7	8.0	6.0	2.0	15.0	0.00126	3.8	3.8

Comments:

* Dispersion device: mechanically operated stirring device

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines



UNIFIED	COBBLES	GRAVEL		SAND			SILT OR CLAY			
		Coarse	Fine	Coarse	Medium	Fine				
USDA	COBBLES	GRAVEL		SAND				SILT	CLAY	
				Very coarse	Coarse	Medium	Fine	Very fine		

d_{10} = **0.0056**	d_{30} = **0.032**	d_{50} = **0.062**	d_{60} = **0.082**	C_u = **15**	C_c = **2.2**

SAMPLE NUMBER	DEPTH	ASTM CLASSIFICATION	USDA CLASSIFICATION
Blend 2	**NA**	**Sandy silt s(ML)**	**Sandy Loam**

Particle Size Analysis
Wet Sieve Data (#10 Split)

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 3
Ring Number: NA
Depth: NA
Test Date: 7-Sep-10

Initial Dry Weight of Sample (g): 50.11
Weight Passing #10 (g): 50.11
Weight Retained #10 (g): 0.00
Weight of Hydrometer Sample (g): 50.11
Calculated Weight of Sieve Sample (g): 50.11

Shape: Angular
Hardness: Soft

Test Fraction	Sieve Number	Diameter (mm)	Wt. Retained	Cum Wt. Retained	Wt. Passing	% Passing
+10						
	3"	75	0.00	0.00	50.11	100.00
	2"	50	0.00	0.00	50.11	100.00
	1.5"	38.1	0.00	0.00	50.11	100.00
	1"	25	0.00	0.00	50.11	100.00
	3/4"	19.0	0.00	0.00	50.11	100.00
	3/8"	9.5	0.00	0.00	50.11	100.00
	4	4.75	0.00	0.00	50.11	100.00
	10	2.00	0.00	0.00	50.11	100.00
-10			(Based on calculated sieve wt.)			
	20	0.85	0.00	0.00	50.11	100.00
	40	0.425	0.07	0.07	50.04	99.86
	60	0.250	1.53	1.60	48.51	96.81
	140	0.106	19.33	20.93	29.18	58.23
	200	0.075	10.46	31.39	18.72	37.36
	dry pan		2.16	33.55	16.56	
	wet pan			16.56	0.00	

d_{10} (mm): 0.013
d_{16} (mm): 0.028
d_{30} (mm): 0.058
d_{50} (mm): 0.092
d_{60} (mm): 0.11
d_{84} (mm): 0.19

Median Particle Diameter --d_{50} (mm): 0.092
Uniformity Coefficient, Cu --[d_{60}/d_{10}] (mm): 8.5
Coefficient of Curvature, Cc --[$(d_{30})^2/(d_{10}*d_{60})$] (mm): 2.4
Mean Particle Diameter --[$(d_{16}+d_{50}+d_{84})/3$] (mm): 0.10

Classification of fines (visual method): ML

ASTM Soil Classification: Silty sand (SM)
USDA Soil Classification: Loamy Sand

Laboratory analysis by: D. O'Dowd/M. Vigil
Data entered by: C. Krous
Checked by: J. Hines

Particle Size Analysis
Hydrometer Data

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 3
Ring Number: NA
Depth: NA

Test Date: 2-Sep-10
Start Time: 9:12

Type of Water Used: DISTILLED
Reaction with H_2O_2*:* NA
Dispersant:* $(NaPO_3)_6$
Measured particle density: 2.98

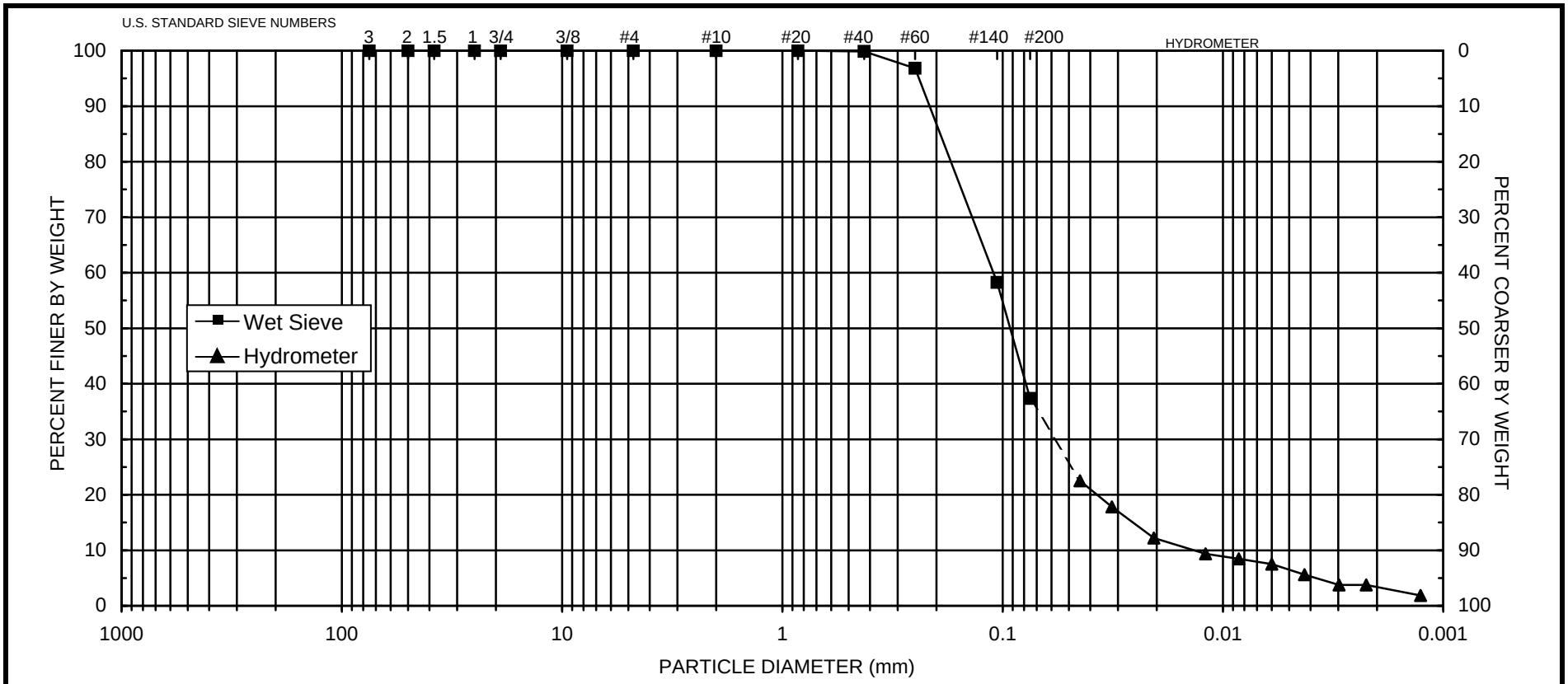
Initial Wt. (g)*:* 50.11
Total Sample Wt. (g)*:* 50.11
Wt. Passing #10 (g): 50.11

Date	Time (min)	Temp (°C)	R (g/L)	R_L (g/L)	R_{corr} (g/L)	L (cm)	D (mm)	P (%)	% Finer
2-Sep-10	1	21.7	18.0	6.0	12.0	13.3	0.04452	22.5	22.5
	2	21.7	15.5	6.0	9.5	13.8	0.03196	17.8	17.8
	5	21.7	12.5	6.0	6.5	14.3	0.02057	12.2	12.2
	15	21.7	11.0	6.0	5.0	14.5	0.01198	9.4	9.4
	30	21.8	10.5	6.0	4.5	14.6	0.00848	8.4	8.4
	60	21.9	10.0	6.0	4.0	14.7	0.00601	7.5	7.5
	120	22.0	9.0	6.0	3.0	14.8	0.00427	5.6	5.6
	250	22.0	8.0	6.0	2.0	15.0	0.00297	3.8	3.8
	440	22.1	8.0	6.0	2.0	15.0	0.00224	3.8	3.8
3-Sep-10	1400	21.7	7.0	6.0	1.0	15.2	0.00127	1.9	1.9

Comments:

* Dispersion device: mechanically operated stirring device

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines



UNIFIED	COBBLES	GRAVEL		SAND			SILT OR CLAY			
		Coarse	Fine	Coarse	Medium	Fine				
USDA	COBBLES	GRAVEL		SAND				SILT	CLAY	
				Very coarse	Coarse	Medium	Fine	Very fine		

d_{10} = **0.013**	d_{30} = **0.058**	d_{50} = **0.092**	d_{60} = **0.11**	C_u = **8.5**	C_c = **2.4**

SAMPLE NUMBER	DEPTH	ASTM CLASSIFICATION	USDA CLASSIFICATION
Blend 3	**NA**	**Silty sand (SM)**	**Loamy Sand**

Particle Size Analysis
Wet Sieve Data (#10 Split)

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 4
Ring Number: NA
Depth: NA
Test Date: 7-Sep-10

Initial Dry Weight of Sample (g): 56.56
Weight Passing #10 (g): 56.56
Weight Retained #10 (g): 0.00
Weight of Hydrometer Sample (g): 56.56
Calculated Weight of Sieve Sample (g): 56.56

Shape: Angular
Hardness: Soft

Test Fraction	Sieve Number	Diameter (mm)	Wt. Retained	Cum Wt. Retained	Wt. Passing	% Passing
+10						
	3"	75	0.00	0.00	56.56	100.00
	2"	50	0.00	0.00	56.56	100.00
	1.5"	38.1	0.00	0.00	56.56	100.00
	1"	25	0.00	0.00	56.56	100.00
	3/4"	19.0	0.00	0.00	56.56	100.00
	3/8"	9.5	0.00	0.00	56.56	100.00
	4	4.75	0.00	0.00	56.56	100.00
	10	2.00	0.00	0.00	56.56	100.00
-10			(Based on calculated sieve wt.)			
	20	0.85	0.00	0.00	56.56	100.00
	40	0.425	0.08	0.08	56.48	99.86
	60	0.250	2.43	2.51	54.05	95.56
	140	0.106	28.06	30.57	25.99	45.95
	200	0.075	13.88	44.45	12.11	21.41
	dry pan		4.05	48.50	8.06	
	wet pan			8.06	0.00	

d_{10} (mm): 0.051
d_{16} (mm): 0.062
d_{30} (mm): 0.085
d_{50} (mm): 0.11
d_{60} (mm): 0.14
d_{84} (mm): 0.20

Median Particle Diameter --d_{50} (mm): 0.11
Uniformity Coefficient, Cu --[d_{60}/d_{10}] (mm): 2.7
Coefficient of Curvature, Cc --[$(d_{30})^2/(d_{10}*d_{60})$] (mm): 1.0
Mean Particle Diameter --[$(d_{16}+d_{50}+d_{84})/3$] (mm): 0.12

Classification of fines (visual method): ML

ASTM Soil Classification: Silty sand (SM)
USDA Soil Classification: Sand

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Particle Size Analysis
Hydrometer Data

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: Blend 4
Ring Number: NA
Depth: NA

Test Date: 2-Sep-10
Start Time: 9:18

Type of Water Used: DISTILLED
Reaction with H_2O_2*:* NA
Dispersant:* $(NaPO_3)_6$
Measured particle density: 2.97

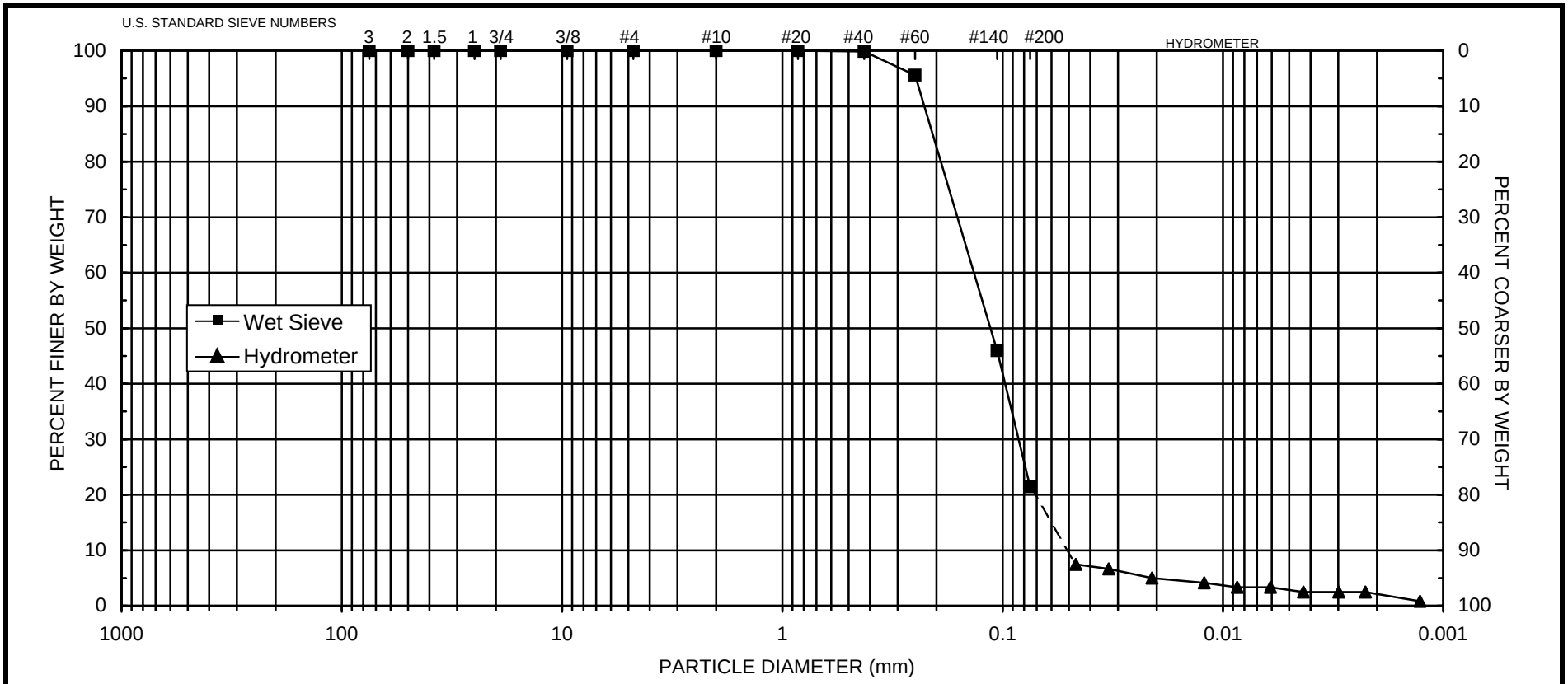
Initial Wt. (g)*:* 56.56
Total Sample Wt. (g)*:* 56.56
Wt. Passing #10 (g): 56.56

Date	Time (min)	Temp (°C)	R (g/L)	R_L (g/L)	R_{corr} (g/L)	L (cm)	D (mm)	P (%)	% Finer
2-Sep-10	1	21.7	10.5	6.0	4.5	14.6	0.04657	7.5	7.5
	2	21.7	10.0	6.0	4.0	14.7	0.03303	6.6	6.6
	5	21.7	9.0	6.0	3.0	14.8	0.02100	5.0	5.0
	15	21.7	8.5	6.0	2.5	14.9	0.01216	4.2	4.2
	30	21.8	8.0	6.0	2.0	15.0	0.00861	3.3	3.3
	60	21.9	8.0	6.0	2.0	15.0	0.00608	3.3	3.3
	120	22.0	7.5	6.0	1.5	15.1	0.00431	2.5	2.5
	250	22.0	7.5	6.0	1.5	15.1	0.00298	2.5	2.5
	436	22.1	7.5	6.0	1.5	15.1	0.00226	2.5	2.5
3-Sep-10	1395	21.7	6.5	6.0	0.5	15.2	0.00127	0.8	0.8

Comments:

* Dispersion device: mechanically operated stirring device

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines



UNIFIED	COBBLES	GRAVEL		SAND			SILT OR CLAY			
		Coarse	Fine	Coarse	Medium	Fine				
USDA	COBBLES	GRAVEL		SAND				SILT	CLAY	
				Very coarse	Coarse	Medium	Fine	Very fine		

d_{10} = **0.051**	d_{30} = **0.085**	d_{50} = **0.11**	d_{60} = **0.14**	C_u = **2.7**	C_c = **1.0**

SAMPLE NUMBER	DEPTH	ASTM CLASSIFICATION	USDA CLASSIFICATION
Blend 4	**NA**	**Silty sand (SM)**	**Sand**

Specific Gravity

Summary of Specific Gravity Tests

	<4.75mm Material		>4.75mm Material		Bulk Sample
Sample Number	Specific Gravity	Percent of Bulk Sample	Specific Gravity	Percent of Bulk Sample	Specific Gravity
-200 fraction #1	3.01	100.0	---	0.0	3.01
-200 fraction #2	2.99	100.0	---	0.0	2.99
-200 Average	3.00	100.0	---	0.0	3.00
+200 fraction #1	2.97	100.0	---	0.0	2.97
+200 fraction #2	2.98	100.0	---	0.0	2.98
+200 Average	2.98	100.0	---	0.0	2.98
As Received	2.95	100.0	---	0.0	2.95

--- = Unnecessary since specified fraction <5% of composite mass

* = Based on specific gravity of material < 4.75 mm

Data for Specific Gravity for Sample: -200 fraction #1

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: -200 fraction #1
Ring Number: NA
Depth: NA

ASTM D854 (<4.75mm Fraction)

Test Date:	22-Aug-10	
Percent of Test Sample (% g/g):	100.00	
Percent of Bulk Sample (% g/g):	100.00	
	Trial 1	*Trial 2*
Weight of pycnometer filled w/air (g)*:*	93.43	90.54
Weight of pycnometer filled w/soil (g)*:*	143.35	141.09
Weight of pycnometer filled w/soil & water (g)*:*	375.91	373.59
Weight of pycnometer filled w/water (g)*:*	342.57	339.86
Observed temperature (°C)*:*	22.30	22.30
Density of water at observed temperature (g/cm^3)*:*	0.9977	0.9977
Specific Gravity (g/g)*:*	3.01	3.01
Correction factor, K:	0.9995	0.9995
Specific Gravity at 20°C (g/g)*:*	3.01	3.00
Average Specific Gravity at 20°C (g/g):	3.01	
Average Particle Density at 20°C (g/cm^3):	3.00	

ASTM C127 (>4.75mm Fraction)

Test Date:	---	
Percent of Test Sample (% g/g):	0.00	
Percent of Bulk Sample (% g/g):	0.00	
Tare Weight (g):	---	--- = Test unnecessary since specified
Saturated Surface Dry (SSD) mass in Air & Tare (g):	---	fraction <5% of composite mass.
Saturated Apparent mass in Water & Tare (g):	---	
Oven Dry (OD) mass in Air & Tare (g):	---	
Observed Temperature (°C):	---	
Density of water at observed temperature (g/m^3):	---	
SSD Specific Gravity (g/g):	---	
Apparent Specific Gravity (g/g):	---	
OD Specific Gravity (g/g):	---	
Percent Absorption (%):	---	
Correction Factor, K:	---	
Average Specific Gravity (Apparent) at 20°C:*	---	
Average Particle Density (Apparent) at 20°C (g/cm^3)**:*	---	
Specific Gravity (Apparent) at 20°C*:	**3.01**	** Weighted harmonic average,*
Particle Density (Apparent) at 20°C (g/cm^3)* :	3.00	*if more than one fraction used.*

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Data for Specific Gravity for Sample: -200 fraction #2

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: -200 fraction #2
Ring Number: NA
Depth: NA

ASTM D854 (<4.75mm Fraction)

Test Date:	22-Aug-10	
Percent of Test Sample (% g/g):	100.00	
Percent of Bulk Sample (% g/g):	100.00	
	Trial 1	*Trial 2*
Weight of pycnometer filled w/air (g)*:*	90.62	90.89
Weight of pycnometer filled w/soil (g)*:*	140.55	140.93
Weight of pycnometer filled w/soil & water (g)*:*	373.07	373.53
Weight of pycnometer filled w/water (g)*:*	339.85	340.17
Observed temperature (°C)*:*	22.40	22.30
Density of water at observed temperature (g/cm^3)*:*	0.9977	0.9977
Specific Gravity (g/g)*:*	2.99	3.00
Correction factor, K:	0.9995	0.9995
Specific Gravity at 20°C (g/g)*:*	2.99	3.00
Average Specific Gravity at 20°C (g/g):	2.99	
Average Particle Density at 20°C (g/cm^3):	2.99	

ASTM C127 (>4.75mm Fraction)

Test Date:	---	
Percent of Test Sample (% g/g):	0.00	
Percent of Bulk Sample (% g/g):	0.00	
Tare Weight (g):	---	--- = Test unnecessary since specified fraction <5% of composite mass.
Saturated Surface Dry (SSD) mass in Air & Tare (g):	---	
Saturated Apparent mass in Water & Tare (g):	---	
Oven Dry (OD) mass in Air & Tare (g):	---	
Observed Temperature (°C):	---	
Density of water at observed temperature (g/m^3):	---	
SSD Specific Gravity (g/g):	---	
Apparent Specific Gravity (g/g):	---	
OD Specific Gravity (g/g):	---	
Percent Absorption (%):	---	
Correction Factor, K:	---	
Average Specific Gravity (Apparent) at 20°C:*	---	
Average Particle Density (Apparent) at 20°C (g/cm^3)**:*	---	
Specific Gravity (Apparent) at 20°C*:	**2.99**	** Weighted harmonic average, if more than one fraction used.*
Particle Density (Apparent) at 20°C (g/cm^3)* :	2.99	

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Data for Specific Gravity for Sample: +200 fraction #1

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: +200 fraction #1
Ring Number: NA
Depth: NA

ASTM D854 (<4.75mm Fraction)

Test Date:	22-Aug-10	
Percent of Test Sample (% g/g):	100.00	
Percent of Bulk Sample (% g/g):	100.00	
	Trial 1	*Trial 2*
Weight of pycnometer filled w/air (g)*:*	93.08	92.48
Weight of pycnometer filled w/soil (g)*:*	147.04	142.89
Weight of pycnometer filled w/soil & water (g)*:*	378.14	375.28
Weight of pycnometer filled w/water (g)*:*	342.32	341.82
Observed temperature (°C)*:*	22.20	22.20
Density of water at observed temperature (g/cm^3)*:*	0.9977	0.9977
Specific Gravity (g/g)*:*	2.98	2.97
Correction factor, K:	0.9995	0.9995
Specific Gravity at 20°C (g/g)*:*	2.97	2.97
Average Specific Gravity at 20°C (g/g):	2.97	
Average Particle Density at 20°C (g/cm^3):	2.97	

ASTM C127 (>4.75mm Fraction)

Test Date:	---	
Percent of Test Sample (% g/g):	0.00	
Percent of Bulk Sample (% g/g):	0.00	
Tare Weight (g):	---	--- = Test unnecessary since specified fraction <5% of composite mass.
Saturated Surface Dry (SSD) mass in Air & Tare (g):	---	
Saturated Apparent mass in Water & Tare (g):	---	
Oven Dry (OD) mass in Air & Tare (g):	---	
Observed Temperature (°C):	---	
Density of water at observed temperature (g/m^3):	---	
SSD Specific Gravity (g/g):	---	
Apparent Specific Gravity (g/g):	---	
OD Specific Gravity (g/g):	---	
Percent Absorption (%):	---	
Correction Factor, K:	---	
Average Specific Gravity (Apparent) at 20°C:*	---	
Average Particle Density (Apparent) at 20°C (g/cm^3)**:*	---	
Specific Gravity (Apparent) at 20°C*:	**2.97**	** Weighted harmonic average, if more than one fraction used.*
Particle Density (Apparent) at 20°C (g/cm^3)* :	2.97	

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Data for Specific Gravity for Sample: +200 fraction #2

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: +200 fraction #2
Ring Number: NA
Depth: NA

ASTM D854 (<4.75mm Fraction)

Test Date:	22-Aug-10	
Percent of Test Sample (% g/g):	100.00	
Percent of Bulk Sample (% g/g):	100.00	
	Trial 1	*Trial 2*
Weight of pycnometer filled w/air (g)*:*	93.36	90.53
Weight of pycnometer filled w/soil (g)*:*	145.05	141.06
Weight of pycnometer filled w/soil & water (g)*:*	376.92	373.33
Weight of pycnometer filled w/water (g)*:*	342.53	339.82
Observed temperature (°C)*:*	22.30	22.20
Density of water at observed temperature (g/cm^3)*:*	0.9977	0.9977
Specific Gravity (g/g)*:*	2.99	2.97
Correction factor, K:	0.9995	0.9995
Specific Gravity at 20°C (g/g)*:*	2.99	2.97
Average Specific Gravity at 20°C (g/g):	2.98	
Average Particle Density at 20°C (g/cm^3):	2.97	

ASTM C127 (>4.75mm Fraction)

Test Date:	---	
Percent of Test Sample (% g/g):	0.00	
Percent of Bulk Sample (% g/g):	0.00	
Tare Weight (g):	---	--- = Test unnecessary since specified fraction <5% of composite mass.
Saturated Surface Dry (SSD) mass in Air & Tare (g):	---	
Saturated Apparent mass in Water & Tare (g):	---	
Oven Dry (OD) mass in Air & Tare (g):	---	
Observed Temperature (°C):	---	
Density of water at observed temperature (g/m^3):	---	
SSD Specific Gravity (g/g):	---	
Apparent Specific Gravity (g/g):	---	
OD Specific Gravity (g/g):	---	
Percent Absorption (%):	---	
Correction Factor, K:	---	
Average Specific Gravity (Apparent) at 20°C:*	---	
Average Particle Density (Apparent) at 20°C (g/cm^3)**:*	---	
Specific Gravity (Apparent) at 20°C*:	**2.98**	** Weighted harmonic average, if more than one fraction used.*
Particle Density (Apparent) at 20°C (g/cm^3)* :	2.97	

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Data for Specific Gravity for Sample: As Received

Job Name: Barr Engineering Company
Job Number: LB10.0170.00
Sample Number: As Received
Ring Number: NA
Depth: NA

ASTM D854 (<4.75mm Fraction)

Test Date:	17-Aug-10	
Percent of Test Sample (% g/g):	100.00	
Percent of Bulk Sample (% g/g):	100.00	
	Trial 1	*Trial 2*
Weight of pycnometer filled w/air (g)*:*	92.14	92.00
Weight of pycnometer filled w/soil (g)*:*	144.17	145.97
Weight of pycnometer filled w/soil & water (g)*:*	375.81	376.91
Weight of pycnometer filled w/water (g)*:*	341.39	341.23
Observed temperature (°C)*:*	22.00	22.00
Density of water at observed temperature (g/cm^3)*:*	0.9978	0.9978
Specific Gravity (g/g)*:*	2.95	2.95
Correction factor, K:	0.9996	0.9996
Specific Gravity at 20°C (g/g)*:*	2.95	2.95
Average Specific Gravity at 20°C (g/g):	2.95	
Average Particle Density at 20°C (g/cm^3):	2.95	

ASTM C127 (>4.75mm Fraction)

Test Date:	---	
Percent of Test Sample (% g/g):	0.00	
Percent of Bulk Sample (% g/g):	0.00	
Tare Weight (g):	---	--- = Test unnecessary since specified
Saturated Surface Dry (SSD) mass in Air & Tare (g):	---	fraction <5% of composite mass.
Saturated Apparent mass in Water & Tare (g):	---	
Oven Dry (OD) mass in Air & Tare (g):	---	
Observed Temperature (°C):	---	
Density of water at observed temperature (g/m^3):	---	
SSD Specific Gravity (g/g):	---	
Apparent Specific Gravity (g/g):	---	
OD Specific Gravity (g/g):	---	
Percent Absorption (%):	---	
Correction Factor, K:	---	
Average Specific Gravity (Apparent) at 20°C:*	---	
Average Particle Density (Apparent) at 20°C (g/cm^3)**:*	---	
Specific Gravity (Apparent) at 20°C*:	**2.95**	** Weighted harmonic average,*
Particle Density (Apparent) at 20°C (g/cm^3)* :	2.95	*if more than one fraction used.*

Laboratory analysis by: K. Wright
Data entered by: C. Krous
Checked by: J. Hines

Laboratory Tests and Methods

Tests and Methods

Dry Bulk Density:	ASTM D7263
Moisture Content:	ASTM D7263
Calculated Porosity:	ASTM D7263
Saturated Hydraulic Conductivity:	
Constant Head: (Rigid Wall)	ASTM D 2434 (modified apparatus)
Hanging Column Method:	ASTM D6836 (modified apparatus)
Pressure Plate Method:	ASTM D6836 (modified apparatus)
Water Potential (Dewpoint Potentiometer) Method:	ASTM D6836
Relative Humidity (Box) Method:	Karathanasis & Hajek. 1982. Quantitative Evaluation of Water Adsorption on Soil Clays. SSA Journal 46:1321-1325; Campbell, G. and G. Gee. 1986. Water Potential: Miscellaneous Methods. Chp. 25, pp. 631-632, in A. Klute (ed.), Methods of Soil Analysis, American Society of Agronomy, Madison, WI
Moisture Retention Characteristics & Calculated Unsaturated Hydraulic Conductivity:	ASTM D6836; van Genuchten, M.T. 1980. A closed-form equation for predicting the hydraulic conductivity of unsaturated soils. SSSAJ 44:892-898; van Genuchten, M.T., F.J. Leij, and S.R. Yates. 1991. The RETC code for quantifying the hydraulic functions of unsaturated soils. Robert S. Kerr Environmental Research Laboratory, Office of Research and Development, U.S. Environmental Protection Agency, Ada, Oklahoma. EPA/600/2091/065. December 1991
Specific Gravity Fine	ASTM D854
Particle Size Analysis:	ASTM D422

Attachment 2-25

Industrial Chemical Additives (Beneficiation and Hydromet)
March 13, 2015.

Minnesota Pollution Control Agency
520 Lafayette Road North
St. Paul, MN 55155-4194

Industrial Chemical Additives Attachment

NPDES/SDS Permit Program

Doc Type: Permit Application

The National Pollutant Discharge Elimination System (NPDES)/State Disposal System (SDS) Permit Program regulates wastewater discharges to land and surface waters. This is an attachment to the Industrial Applications for facilities with multiple chemical additives.

Complete the attachment by typing or printing in black ink. Attach additional sheets as necessary. For more information, please contact the Minnesota Pollution Control Agency (MPCA) at: In Metro Area: 651-296-6300 or Outside Metro Area: 800-657-3864.

Permittee name: PolyMet Mining Inc. - THIS APPLICATION FORM IS PROVIDED TO FOTH AS DRAFT (3/13/2015) **Permit number:** MN TBD

Chemical	**Purpose**	**Location of chemical addition in process** (e.g., to raw water supply, at greensand filter, before RO unit #2, etc.)	**Amount/duration/ frequency of addition**	**Average rate of use** (weight or volume per day)	**Maximum rate of use** (weight or volume per day)
SIPX (sodium isopropyl xanthate)	Collector: Selectively adsorb minerals based on hydrophobicity of the collector and mineral	Flotation Circuit, specifically the Flotation Roughers, Scavengers, Cleaner Circuit	Continuous	2.74 t/day (1,000 t/year)	4.79 t/day (1,750 t/year)
PAX (potassium amyl xanthate)	Collector: Selectively adsorb minerals based on hydrophobicity of the collector and mineral	Flotation Circuit, specifically the Flotation Roughers, Scavengers, and Cleaner Flotation Cells	Continuous	2.74 t/day (1,000 t/year)	4.79 t/day (1,750 t/year)
MIBC (methyl isobutyl carbinol 100% solution)	Frother: Used to improve stability of froth bubbles as they rise through the flotation cells	Flotation Circuit, specifically the Flotation Roughers, Scavengers, and Cleaner Flotation Cells	Continuous	2.88 t/day (1,050 t/year)	4.11 t/day (1,500 t/year)
Frother (F-160-05)	Frother: Used to improve stability of froth bubbles as they ris through the flotation cells (Potential substitute for MIBC)	Flotation Circuit, specifically the Flotation Roughers, Scavengers, and Cleaner Flotation Cells	Continuous	2.88 t/day (1,050 t/year)	4.11 t/day (1,500 t/year)
Frother (F-160-13)	Frother: Used to improve stability of froth bubbles as they ris through the flotation cells (Potential substitute for MIBC)	Flotation Circuit, specifically the Flotation Roughers, Scavengers, and Cleaner Flotation Cells	Continuous	2.88 t/day (1,050 t/year)	4.11 t/day (1,500 t/year)
Frother (DVS4U038)	Frother: Used to improve stability of froth bubbles as they ris through the flotation cells (Potential substitute for MIBC)	Flotation Circuit, specifically the Flotation Roughers, Scavengers, and Cleaner Flotation Cells	Continuous	2.88 t/day (1,050 t/year)	4.11 t/day (1,500 t/year)
Copper Sulfate	Activator: Used as an activator to increse the available adsorption sites on the mineral to allow for adsorption by the Collector	Flotation Circuit, specifically Scavenger Cells	Continuous	1.71 t/day (625 t/year)	2.05 t/day (750 t/year)
Flocculant (MagnaFloc 10)	Flocculant: Promote flocculation of suspended particles in liquors	Flotation Circuit, specifically Concentrate Thickeners	Continuous	0.082 t/day	0.14 t/day

				(30 t/year)	(50 t/year)
Flocculant (MagnaFloc 455)	Flocculant: Promote flocculation of suspended particles in liquors (Potential substitute for Magnafloc 10)	Flotation Circuit, specifically Concentrate Thickeners	Continuous	0.07 t/day (25 t/year)	0.14 t/day (50 t/year)
Flocculant (Neo NS 6655)	Flocculant: Promote flocculation of suspended particles in liquors (Potential substitute for Magnafloc 10/455)	Flotation Circuit, specifically Concentrate Thickeners	Continuous	0.07 t/day (25 t/year)	0.14 t/day (50 t/year)
Flocculant (NALCO 83949)	Flocculant: Promote flocculation of suspended particles in liquors (Potential substitute for Magnafloc 10/455)	Flotation Circuit, specifically Concentrate Thickeners	Continuous	0.07 t/day (25 t/year)	0.14 t/day (50 t/year)
Flocculant (NALCO 9877 Pulv)	Flocculant: Promote flocculation of suspended particles in liquors (Potential substitute for Magnafloc 10/455)	Flotation Circuit, specifically Concentrate Thickeners	Continuous	0.07 t/day (25 t/year)	0.14 t/day (50 t/year)
CMC (Carboxyl methyl cellulose)	Flocculant: Used to depress gangue minerals in flotation cells to improve selectivity towards Cu Ni minerals	Flotation Circuit, specifically Rougher and Pyrhotite Cleaner Flotation Cells	Continuous	3.29 t/day (1,200 t/year)	4.79 t/day (1,750 t/year)
Lime slurry	pH Modifier: Used to regulate pH in the Flotation Circuit	Flotation Circuit, specifically the Separation Cleaner Flotation Cells	Continuous	28.15 t/day as hydrated lime (10,274 t/year as hydrated lime)	41.10 t/day as hydrated lime (15,000 t/year as hydrated lime)

***Remember to attach the *Material Safety Data Sheets*, complete product labels and any other information on chemical composition, aquatic toxicity, human health, and environmental fate for each chemical additive.**

Please make a copy for your records.

Refer to the *Transmittal Form* for mailing instructions.

Minnesota Pollution Control Agency
520 Lafayette Road North
St. Paul, MN 55155-4194

Industrial Chemical Additives Attachment

NPDES/SDS Permit Program

Doc Type: Permit Application

The National Pollutant Discharge Elimination System (NPDES)/State Disposal System (SDS) Permit Program regulates wastewater discharges to land and surface waters. This is an attachment to the Industrial Applications for facilities with multiple chemical additives.

Complete the attachment by typing or printing in black ink. Attach additional sheets as necessary. For more information, please contact the Minnesota Pollution Control Agency (MPCA) at: In Metro Area: 651-296-6300 or Outside Metro Area: 800-657-3864.

Permittee name: PolyMet Mining Inc. - THIS APPLICATION FORM IS PROVIDED TO FOTH AS DRAFT (3/13/2015) **Permit number:** MN TBD

Chemical	**Purpose**	**Location of chemical addition in process** (e.g., to raw water supply, at greensand filter, before RO unit #2, etc.)	**Amount/duration/ frequency of addition**	**Average rate of use** (weight or volume per day)	**Maximum rate of use** (weight or volume per day)
Sodium hydrosulfide (30% solution)	Cementation of copper from solution as CuS	Hydromet, specifically copper cementation	Continuous	3.17 t/day (1,160 t/year)	4.10 t/day (1,750 t/year)
Caustic soda (Sodium hydroxide, 50% solution)	Increase pH of off-gases by removing traces of H2S and SO2 in vent scrubbers	Hydromet, specifically the plant scrubber	Continuous	57.53 gal/day (21,000 gal/year)	82.19 gal/day (30,000 gal/year)
Sulfuric acid (93% solution)	Used as wash water for leach residue filter	Hydromet, specifically the residue filter wash water	Continuous	0.47 t/day (170 t/year)	0.68 t/day (250 t/year)
Hydrochloric acid (32% solution)	Addition of chloride used to promote mineral leaching	Hydromet, specifically the autoclave	Continuous	13.70 t/day (5,000 t/year)	20.55 t/day (7,500 t/year)
Flocculant (MagnaFloc 342)	Promote flocculation of suspended particles in liquors	Hydromet, specifically mixed hydroxide precipitation	Continuous	0.06 t/day (21 t/year)	0.11 t/day (40 t/year)
Flocculant (NALCO 9877 Pulv)	Promote flocculation of suspended particles in liquors (Potential substitute for MagnaFloc 342)	Hydromet, specifically mixed hydroxide precipitation	Continuous	0.11 t/day (40 t/year)	0.21 t/day (75 t/year)
Flocculant (MagnaFloc 155)	Promote flocculation of suspended particles in liquors	Hydromet, specifically mixed hydroxide precipitation	Continuous	0.11 t/day (40 t/year)	0.21 t/day (75 t/year)
Flocculant (Neo NS 6670)	Promote flocculation of suspended particles in liquors (Potential substitute for MagnaFloc 342 or 155)	Hydromet, specifically mixed hydroxide precipitation	Continuous	0.11 t/day (40 t/year)	0.21 t/day (75 t/year)
Flocculant (NALCO	Promotes flocculation of suspended	Hydromet, specifically mixed hydroxide	Continuous	0.11 t/day	0.21 t/day

8173 PULV)	particles in liquors (Potential substitute for MagaFloc 342 or 155)	precipitation		(40 t/year)	(75 t/year)
Flocculant (MagnaFloc 351)	Promote flocculation of suspended particles in liquors	Hydromet, specifically in the leach residue thickener, PGM thickener, and CuS cementation thickener	Continuous	0.27 t/day (100 t/year)	0.41 t/day (150 t/year)
Flocculant (Neo NS 6500)	Promote flocculation of suspended particles in liquors (Potential substitute for MagnaFloc 351)	Hydromet, specifically in the leach residue thickener, PGM thickener, and CuS cementation thickener	Continuous	0.41 t/day (150 t/year)	0.55 t/day (200 t/year)
Flocculant (NALCO 9876 PULV)	Promote flocculation of suspended particles in liquors (Potential substitute for MagnaFloc 351)	Hydromet, specifically in the leach residue thickener, PGM thickener, and CuS cementation thickener	Continuous	0.41 t/day (150 t/year)	0.68 t/day (250 t/year)
Sulfur dioxide (liquid)	Used to reduce ferric ions to ferrous ions	Hydromet, specifically iron reduction and PGM precipitation	Continuous	4.14 t/day (1,510 t/year)	6.16 t/day (2,250 t/year)
Limestone (lump)	Used to promote precipitation of Fe and Al	Hydromet, specifically in iron removal	Continuous	276.71 t/day (101,000 t/year)	410.96 t/day (150,000 t/year)
Limestone (ground)	Used to promote precipitation of Fe and Al (Potential substitute fo lump limestone)	Hydromet, specifically in iron removal	Continuous	276.71 t/day (101,000 t/year)	410.96 t/day (150,000 t/year)
Lime (dry)	Used to promote precipitation of Ni and Co sulfates as Ni and Co hydroxides (mixed hydroxide precipitates)	Hydromet, specifically mixed hydroxide precipitation	Continuous	10.55 t/day as CaO (3,850 t/year as CaO)	16.44 t/day as CaO (6,000 t/year as CaO)
Magnesium hydroxide (60% slurry)	Used to promote precipitation of Ni and Co sulfates as Ni and Co hydroxides (mixed hydroxide precipitate)	Hydromet, specifically mixed hydroxide precipitation	Continuous	16.44 t/day (6,000 t/year)	24.66 t/day (9,000 t/year)
Magnesium hydroxide (dry)	Used to promote precipitation of Ni and Co sulfates as Ni and Co hydroxides (mixed hydroxide precipitate) (Potential substitute for magnesium hydroxide - 60% slurry)	Hydromet, specifically mixed hydroxide precipitation	Continuous	16.44 t/day (6,000 t/year)	24.66 t/day (9,000 t/year)

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