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File Code: 2670
Date: DEC 14 2016

Neil Kornze
Director
Bureau of Land Management
1849 C. Street NW, Rm. 5665
Washington, DC 20240

Dear Director Kornze:

On June 3, 2016, the Bureau of Land Management (BLM) requested the Forest Service (FS) provide a decision on whether it consents to renewal of two leases currently held by Twin Metals Minnesota (TMM) for lands within the Superior National Forest (SNF) in northern Minnesota. These two Preference Right leases, MNES-01352 and MNES-01353, lie directly adjacent to and within three miles of the Boundary Waters Canoe Area Wilderness (BWCAW), respectively. The FS has considered the environmental conditions, nature and uses of the BWCAW by the public and tribes, economic benefits of mineral development and wilderness recreation, potential environmental consequences of mineral development on the leases, public opinion, rarity of copper-nickel sulfide ore mining in this region, and current laws and policy to inform the agency's decision.

Based on this analysis, I find unacceptable the inherent potential risk that development of a regionally-untested copper-nickel sulfide ore mine within the same watershed as the BWCAW might cause serious and irreplaceable harm to this unique, iconic, and irreplaceable wilderness area. Therefore, the FS does not consent to renewal of Preference Right leases MNES-01352 and MNES-01353. A summary of the basis for my decision follows.

The BWCAW Is an Irreplaceable Resource

The 1.1 million acre the BWCAW is located in the northern third of the SNF in Minnesota, extending nearly 200 miles along the international boundary with Canada. It is the only large-scale protected sub-boreal forest in the lower 48 United States. The SNF holds 20 percent of the National Forest System's fresh water supply. These healthy forests with extremely high water quality also provide a host of watershed benefits, such as purifying water, sustaining surface water and ground water flow, maintaining fish habitats, controlling erosion, and stabilizing streambanks.

In addition to the existing high quality of the waters, the dramatic hydrogeology and interconnectedness of BWCAW's forests, lakes, streams, and wetlands make the region unique and susceptible to degradation. The BWCAW includes nearly 2,000 pristine lakes ranging in size from 10 acres to 10,000 acres, and more than 1,200 miles of canoe routes.

With Voyageurs National Park and Quetico Provincial Park, BWCAW is part of an international network of conserved land and wilderness. Quetico Provincial Park, located in Ontario, Canada, lies within the same Rainy River watershed as the BWCAW. Quetico Provincial Park is an iconic wilderness class park, world renowned as a destination for backcountry canoeing with over 2,000 lakes and over one million acres of remote water-based wilderness. Together, Quetico and BWCAW form a core wilderness area of over two million acres.

Located northwest of the BWCAW, Voyageurs National Park was established by Congress in 1971 to preserve and interpret fur trade history and the importance of canoe travel routes in northern Minnesota. The park is at the southern edge of the boreal forest, and lies within the same Rainy River watershed as the BWCAW. It features spectacular canoeing and boating routes along with hiking trails exploring portage routes used by American Indians, early fur traders, and gold miners. Approximately 240,000 people visit Voyageurs National Park every year.

Just south of the BWCAW the Laurentian Divide separates three river systems: one flowing north to Hudson Bay; the Laurentian system flowing eastward towards the Atlantic through the Great Lakes, and the Mississippi system, flowing south to the Gulf of Mexico. TMM's two leases subject to FS decision are located in the Rainy River Watershed, which drains into the BWCAW, Quetico Provincial Park and Voyageurs National Park. There are four HUC (Hydrologic Unit Code) -10 sub-watersheds in the area of the leases and potential project site—Birch Lake, Stony River, Isabella River and Kawishiwi River. Surface water flows north and west from Birch Lake and the Kawishiwi River watershed through Kawishiwi River and several lakes into BWCAW. Water from the Stony River and the Isabella River watersheds flows into the Birch Lake watershed.

The BWCAW's Natural Environment

The SNF provides abundant and diverse habitat for thousands of breeding, wintering, and migratory species of terrestrial and aquatic wildlife, including over 100 species of migratory breeding birds in a zone with North America's greatest diversity of songbirds and forest-dependent warblers. The SNF also has one of the largest populations of gray wolves outside of Alaska, common loons, and moose. It has popular game species such as walleye, trout, deer, ruffed grouse, fisher, and beaver; and numerous rare species such as great gray owl, black-backed woodpecker, ram's-head ladyslipper and other orchids, and lake sturgeon. The SNF also has a great diversity and abundance of species common to the boreal forest biome, including three-toed woodpecker, boreal owl, boreal chickadee, lynx, moose, and grizzled skipper butterfly. All these species provide a wide array of crucial ecological, social and economic benefits and uses - from big game hunting and fishing to wildlife watching and research.

The BWCAW is also home to three threatened or endangered species: Canada lynx, northern long-eared bat, and gray wolf. Over the decades the BWCAW has been protected, it has provided refugia for species under stress or with declining populations, such as moose. In the face of climate change, the BWCAW may be critical to the continued existence of these species within Minnesota.

Cultural Resources and Treaty Rights Associated with the BWCAW

The BWCAW region has been home to Native Americans for millennia. The Minnesota Chippewa Tribe and three associated Bands – the Grand Portage Band, the Fond du Lac Band,

and the Bois Forte Band -- retain hunting, fishing, and other usufructuary rights throughout the entire northeast portion of the State of Minnesota under the 1854 Treaty of LaPointe. In the Ceded Territory all Bands have a legal interest in protecting natural resources, and the FS shares in federal trust responsibility to maintain treaty resources. Many resident Ojibwe, who ceded lands that became the BWCAW, continue to visit ancestral sites and traditional gathering and fishing locations within the wilderness. Tribes rely on natural resources like fish, wildlife and wild plants such as wild rice for subsistence and to support them spiritually, culturally, medicinally, and economically.

The northern border of the BWCAW is situated along a winding, 120-mile canoe route known locally as the Border Route, or Voyageurs Highway. This historic canoe route, bordered on the north by Ontario's Quetico Provincial Park, on the east by Grand Portage National Monument, and on the west by Voyageurs National Park, was utilized extensively by pre-contact Native Americans, European fur traders, and tribal groups such as the Dakota, Cree, and Ojibwe.

There are approximately 1,500 cultural resource sites identified on National Forest System (NFS) lands within the BWCAW. Many more cultural resources are believed to exist within the wilderness; as of 2015 only about 3 percent of the landscape has been intensively surveyed. Cultural resource sites include historic Ojibwe village sites, French and British period fur trade sites dating from 1730-1830, Woodland period village sites (2,000-500 years old) situated on wild rice lakes, Native American pictograph panel sites, Archaic period (8,000-3,000 years old) sites with copper tools, and large Paleoindian quarry sites such as those recently discovered on Knife Lake where Native Americans shaped stone tools up to 10,000 years ago.

Wilderness Designation

The irreplaceable natural qualities of the BWCAW were recognized nearly a century ago in 1926 when the Department of Agriculture first set aside the area to preserve its primitive character. The Wilderness Act of 1964 officially designated land inside today's BWCAW as part of the National Wilderness Preservation System. The Boundary Waters Canoe Area Wilderness Act of 1978 expanded the wilderness area to 1,090,000 acres. The 1978 Act also established a separate Boundary Waters Canoe Area Mining Protection Area (MPA) to protect existing natural values and high standards of environmental quality from the adverse impacts associated with mineral development. Sec. 9, Pub. L. 95-495, 92 Stat. 1649, 1655 (1978). Congress provided very clear direction regarding the purposes of the BWCAW and MPA:

(1) provide for the protection and management of the fish and wildlife of the wilderness so as to enhance public enjoyment and appreciation of the unique biotic resources of the region,

(2) protect and enhance the natural values and environmental quality of the lakes, streams, shorelines and associated forest areas of the wilderness,

(3) maintain high water quality in such areas,

(4) minimize to the maximum extent possible, the environmental impacts associated with mineral development affecting such areas.... Sec. 2, Pub. L. 95-495, 92 Stat. 1649 (1978).

The BWCAW Act bans authorization of federal mineral development within the BWCAW and MPA. However, the BWCAW Act does not govern federal mineral development on other NFS lands. Instead, the authorities governing federal mineral development on SNF lands outside the BWCAW and MPA are 16 U.S.C. § 508b and Section 402 of Reorganization Plan No. 3 of 1946, 60 Stat. 1097, 1099-1100. A decision withholding FS consent to the lease renewals is fully consistent with this statutory framework.

World Renowned Research Laboratory

Because of its unique quality and character, the BWCAW is a living laboratory supporting dozens of research projects each year. Scientists of all disciplines rely on scarce areas like the BWCAW to support scientific inquiry and serve as control areas in the study of water quality, climate change effects, and natural ecological processes. The BWCAW is internationally known as a laboratory for ground-breaking research on forest fires, landscape patterns, biodiversity, wildlife, soils, nutrient cycles, other ecosystem processes, lakes, climate change, and recreational use of wilderness. This body of work is widely cited by scientists around the world. As an example, Miron Hinselman's work on forest fires in BWCAW, published during the 1970s-1990s, has been cited in more than 1,700 published studies. More recent BWCAW-related studies by Frelich and Reich have already been cited in 1,300 studies in 70 peer-reviewed science journals published in 20 countries on 4 continents. New results from BWCAW research are regularly presented at prestigious international meetings on scientific study.

Recreation Values of the BWCAW

The BWCAW is one of the most visited areas in the entire National Wilderness Preservation System, and the System's only large lake-land wilderness. It provides an experience unique within the continental United States. The BWCAW's thousands of lakes and hundreds of miles of streams comprise about 190,000 acres (20 percent) of the BWCAW's surface area and provide for long distance travel by watercraft. The opportunity to pursue and experience expansive solitude, challenge and personal immersion in nature are integral to the BWCAW experience. Winter BWCAW visitors enjoy opportunities for skiing, dog-sledding, camping and ice fishing. Fishing is one of the most popular BWCAW activities throughout the year due to the range of species found in its waters, including smallmouth bass, northern pike, walleye, and lake trout.

Social and Economic Environment

TMM's leases are located near Ely, in St. Louis and Lake Counties. The population of St. Louis County is concentrated in and around the City of Duluth, approximately 100 miles south of the lease area. The Iron Range communities of Ely, Hibbing, and Virginia are smaller secondary population centers. The 2010 U.S. Census shows area population has declined by nearly 10 percent since 1980, while Minnesota's population as a whole has increased by more than 30 percent. At least some of this population decline may be attributable to a loss of iron industry jobs. The Fond du Lac, Grand Portage, and Bois Forte reservations are exceptions to the regional trend - populations there have increased since 1990.

The median income of area communities is significantly lower than that of the State as a whole. It is also the case that the median income of the area's secondary population centers is generally lower than that of St. Louis County as a whole. In some of these communities, such as Ely and Tower, the median household income is slightly more than half of the state median. In many individual communities, poverty rates are as high as or higher than statewide (with the exceptions of the secondary population centers of Hoyt Lakes, Soudan, and Tower).

Mining employment in St. Louis County declined from more than 12,000 jobs in 1980 to approximately 3,000 jobs in 2009. However, since mining employment can vary greatly from one year to the next, this decline does not represent a steady reduction. Mining-related employment is volatile and fluctuates due to changes in the market price of commodities being extracted. During the same time period, service-related employment (which includes the North American Industry Classification System categories for professional services, management, health care, education, arts/entertainment, and accommodation/food) in the study area has increased substantially, mirroring broader state and national trends.

Tourism is rooted in the region's unique recreation opportunities such as the BWCAW, and is broadly dependent on hunting, fishing, boating, sightseeing, and wilderness experiences provided by the region's high-quality natural environment. Industries associated with tourism (arts, entertainment, recreation, accommodation, and food services) account for nearly 13 percent of all employment in St. Louis County. The landscape and recreational opportunities attracts retirees and new residents.

Fishing in Minnesota lakes and rivers generates $2.8 billion in direct annual expenditures and contributes more than $640 million a year in tax revenues to the treasuries of the state and federal governments. The BWCAW itself has provided millions of visitors with a unique water-based recreation experience and provided an economic driver to local communities and the state of Minnesota. Leases MNES-01352 and MNES-01353 are surrounded by 29 resorts, outfitters, campgrounds and hundreds of homes and cabins. Similarly, Voyageurs National Park and Quetico Provincial Park both support vibrant tourism industries.

In 2015, 150,000 people visited the BWCAW. Economic benefits generated from recreation in the BWCAW average approximately $44.5 million annually. Continued economic returns rely on sustaining BWCAW's natural resource quality and wilderness character.

The FS's Role with Respect to Hardrock Mineral Leases

TMM's two leases include a mixture of NFS lands reserved from the public domain and acquired NFS lands, with the vast majority being reserved lands. 16 U.S.C. § 508b applies to reserved NFS lands and provides in pertinent part:

"the Secretary of the Interior is authorized … to permit the prospecting for and the development and utilization of [hard rock] mineral resources: provided, that the development and utilization of such mineral deposits shall not be permitted by the Secretary of the Interior except with the consent of the Secretary of Agriculture."

Section 402 of Reorganization Plan No. 3 of 1946, 60 Stat. 1097, 1099, applies to acquired NFS lands and provides in pertinent part:

"The functions of the Secretary of Agriculture and the Department of Agriculture with respect to the uses of mineral deposits in certain lands pursuant to … 16 U.S.C. § 520 … are hereby transferred to the Secretary of the Interior and shall be performed by him or … by such officers and agencies of the Department of the Interior as he may designate: Provided, That mineral development on [lands acquired pursuant to the Weeks Act] shall be authorized by the Secretary of the Interior only when he is advised by the Secretary of Agriculture that such development will not interfere with the primary purposes for which the land was acquired and only in accordance with such conditions as may be specified by the Secretary of Agriculture in order to protect such purposes."

In pertinent part, 16 U.S.C. § 520 provides:

The Secretary of Agriculture is authorized, under general regulations to be prescribed by him, to permit the prospecting, development, and utilization of the mineral resources of the lands acquired under the Act of March first, nineteen hundred and eleven, known as the Weeks law, upon such terms and for specified periods or otherwise, as he may deem to be for the best interests of the United States….

Under the Weeks Act, 16 U.S.C. § 515, the Secretary of Agriculture is authorized to purchase lands for the purposes of "the regulation of the flow of navigable streams or … the production of timber."

The Department of the Interior adopted regulations providing for disposal of mineral resources pursuant to 16 U.S.C. § 508b and Section 402 of Reorganization Plan No. 3 of 1946, 60 Stat. 1097, 1099, by means of a leasing system governed by 43 C.F.R. part 3500. 43 C.F.R. § 3501.1(b)(1) & (3). The Department of the Interior's regulations provide that BLM's issuance of leases for hard rock minerals, including deposits of copper, nickel and associated minerals, on lands administered by another surface managing agency is "[s]ubject to the consent of the surface managing agency," 43 C.F.R. § 3503.13(a) & (c), which in the case of NFS lands is the United States Department of Agriculture, Forest Service. 16 U.S.C. § 1609(a). Specifically, 43 C.F.R. § 3503.13(a) relates to lands acquired under the Weeks Act while 43 C.F.R. § 3503.13(c) relates to the reserved lands.

On March 8, 2016, Department of Interior Solicitor Hilary Tompkins issued memorandum M-37036 (M-Opinion) in response to a BLM request asking "whether it has the discretion to grant or deny Twin Metals Minnesota's pending application for renewal of two hardrock preference right leases in northern Minnesota." The M-Opinion advises the BLM determining that, "Neither of the statutory authorities under which [MNES-01352 and MNES-01353] are issued–section 402 of Reorganization Plan No. 3 of 1946, 60 Stat. 1097, 1099-1100, and 16 U.S.C. § 508b–creates an entitlement to a lease or otherwise mandates the issuance of leases" and "[t]o the contrary, both authorities expressly condition leasing on surface owner consent (in this instance the Forest Service) and thus are discretionary." Therefore, on June 3, 2016, the BLM advised the Forest Service:

"[i]n light of the legal determination that the government has discretion in granting or denying the TMM lease renewal application, in accordance with 43 CFR 3503.20, 16 U.S.C. 508b, Section 402 of Reorganization Plan No. 3 of 1946, 60 Stat. 1097, 1099, and 16 USC 520, the

BLM requests that the USDA Forest Service provide, in writing, a decision on whether it consents or does not consent to the renewal of the leases."

Irrespective of the M-Opinion, the FS's consent to any hardrock lease renewal is mandated by 16 U.S.C. § 508b and Section 402 of Reorganization Plan No. 3 of 1946, 60 Stat. 1097, 1099. Pursuant to 16 U.S.C. § 508b, the Secretary of Agriculture's right to consent to "the development and utilization of [hardrock] mineral resources" is coextensive with the Secretary of the Interior's authority to permit "the development and utilization of [hardrock] mineral resources." The fact that the Secretary of the Interior has implemented the authority 16 U.S.C. § 508b confers to permit "the development and utilization of [hardrock] mineral resources" by means of a regulatory scheme containing a number of decision points simply means that the Secretary of Agriculture's statutory consent authority with respect to hardrock mineral development and utilization – authority expressed in terms identical to the Department of Interior's authority – similarly extends to the same universe of decision points providing those decisions have the potential to affect NFS surface resources.

Whereas pursuant to Section 402 of Reorganization Plan No. 3 of 1946, 60 Stat. 1097, 1099, the Secretary of the Interior's authority per 16 U.S.C. § 520 "to permit the … development … of the [hardrock] mineral resources of the lands acquired under … the Weeks law…" is contingent upon the Secretary of Agriculture's determination that "such development will not interfere with the primary purposes for which the land was acquired…." It is well established that mineral "development" is authorized by a lease, whether it is one issued in the first instance or a subsequent renewal. Indeed, the M-Opinion explicitly recognizes that "the entire purpose" of a mineral lease is "for the lessee to develop the minerals…." Another M-Opinion finds that since the 1970s hardrock prospecting permits for NFS lands, which are the precursor for the issuance of hardrock mineral leases including MNES-01352 and MNES-01353, have uniformly included the condition that "no mineral development of any type is authorized hereby." M-36993, Options Regarding Applications for Hardrock Mineral Prospecting Permits on Acquired Lands Near a Unit of the National Park System (1998 WL 35152797 (April 16, 1998)). *Missouri Coalition for the Environment,* 124 IBLA 211, 217 (1992) ("mineral development … may only be authorized upon issuance of a [hardrock] lease*); John A. Nejedly Contra Costa Youth Association, 80 IBLA 14, 26 (1984)* (concurring opinion) (development under a hardrock lease "is a logically foreseen result of successful prospecting"). So again, the fact that the Secretary of the Interior has implemented the authority Section 402 of Reorganization Plan No. 3 of 1946, 60 Stat. 1097, 1099, confers to permit the development of hardrock mineral resources on lands acquired pursuant to the Weeks Act by means of a regulatory scheme containing a number of decision points simply means that the Secretary of Agriculture's consent authority with respect to hardrock mineral development – authority expressed in terms identical to Interior's authority – similarly extends to the same universe of decision points providing those decisions have the potential to affect NFS surface resources.

Of course, under Section 402 of Reorganization Plan No. 3 of 1946, 60 Stat. 1097, 1099, the Secretary of Agriculture cannot block mineral development absent a finding that "such development will … interfere with the primary purposes for which the land was acquired…." Here, since the small percentage of acquired lands subject to TMM's two leases were purchased in accordance with the Weeks Act, those primary purposes were "the regulation of the flow of navigable streams or … the production of timber." As discussed below, TMM hopes to construct and operate an underground mine on its two leases – not a strip mine. At this juncture the FS consequently cannot definitively say that the mineral development which TMM hopes to conduct on its leases will interfere with those purposes. Uncertainty about this question is of little import, however, since the lands subject to TMM's leases are an admixture of lands reserved from the public domain and acquired lands with the reserved lands being in excess of 90% of the acreage included in both leases. Further, there is no reason to believe that TMM's mineral development exclusively could be confined to the acquired lands. The FS's conclusion that the agency should exercise the absolute discretion that 16 U.S.C. § 508b confers upon it to withhold consent to the renewal of TMM's leases insofar as the reserved lands are concerned accordingly has preclusive effect with respect to the lands acquired pursuant to the Weeks Act.

The Role of Forest Plans

The FS develops land and resource management plans to provide a framework that protects renewable surface resources. This framework balances both economic and environmental considerations to provide for multiple uses and sustained yield of NFS renewable surface resources.

The 2004 SNF Plan at D-MN-1 states: "Exploration and development of mineral and mineral material resources is allowed on NFS land, except for federally owned minerals in designated wilderness and the Mining Protection Area." The Plan also provides that the FS will manage the BWCAW in a manner that perpetuates and protects its unique natural ecosystems, provides an enduring wilderness resource for future generations, and provides opportunities for a primitive and unconfined recreation experience.

Although forest plans provide a framework, they do "not authorize projects or activities or commit the Forest Service to take action" (36 C.F.R. § 219.2(b)(2)). Instead, forest plans provide broad management guidance and ensure all program elements and legal requirements are considered prior to critical project level decisions, such as a decision to authorize timber harvesting, grazing or mining operations. As the Supreme Court has determined, forest plans:

"…do not command anyone to do anything or to refrain from doing anything; they do not grant, withhold, or modify any formal legal license, power, or authority; they do not subject anyone to any civil or criminal liability; they create no legal rights or obligations. Thus, for example, the Plan does not give anyone a legal right to cut trees, nor does it abolish anyone's legal authority to object to trees being cut. *Ohio Forestry Ass'n. v. Sierra Club*, 523 U.S. 726, 733 (1998)."

Following Forest Plan approval, proposals are evaluated on a case-by-case basis. Proposals inconsistent with Plan direction may not be authorized (16 U.S.C. §1604(i)). However, a proposal might reveal the need to amend plan direction that would otherwise stand as an impediment to a proposal. Yet a proposal's consistency with applicable Plan standards and guidelines is not an assurance that the proposal will be authorized. The FS retains discretionary judgment concerning overall multiple use, sustained yield management of NFS lands. Further, denial of a proposal consistent with applicable Plan standards and guidelines does not require alteration of the applicable direction.

The SNF Plan does not prohibit mineral development within the management area where TMM's leases are located. But the FS is not bound to approve TMM's application for renewal of its leases either. Neither the statute nor regulations governing forest plans mandate the approval of proposals consistent with a Forest plan. Moreover, as discussed above, pursuant to the express terms of 16 U.S.C. § 508b and Section 402 of Reorganization Plan No. 3 of 1946, 60 Stat. 1097, 1099, the FS retains discretion to withhold consent to TMM's lease renewals given the leases' purpose is mineral development, as recognized by the M-Opinion. Specifically, the FS denial of consent to TMM's lease renewals is warranted for the reasons set out in the M-Opinion and also because the bar in both 16 U.S.C. § 508b and Section 402 of Reorganization Plan No. 3 of 1946, 60 Stat. 1097, 1099, against mineral development absent the consent of the Secretary of Agriculture applies with equal force to the initial issuance of the lease and any renewal of that lease. Accordingly, the FS may consider any potential negative environmental impacts that might flow from mineral development on those leases and their effect on future national forest conditions.

National Environmental Policy Act (NEPA) Applicability

NEPA ensures federal agencies take into account significant environmental matters in their decision making, and that they disclose to the public that the agency has considered environmental concerns. An environmental impact statement (EIS) must be prepared when an agency proposes to undertake a major federal action that may significantly affect the quality of the human environment. In summary, NEPA tasks agencies to assess changes in the physical environment caused by the action it proposes to authorize.

Council on Environmental Quality (CEQ) regulations implementing NEPA are clear that a proposal "exists at that stage in the development of an action when an agency subject to the Act has a goal and is actively preparing to make a decision on one or more alternative means of accomplishing that goal and the effects can be meaningfully evaluated." 40 C.F.R. § 1508.23. This provision is reinforced by CEQ's instruction that major federal actions "includes actions with effects…." 40 C.F.R. § 1508.18. FS NEPA regulations establish a four part test for determining when NEPA obligations arise, including whether "[t]he Forest Service has a goal and is actively preparing to make a decision on one or more alternative means of accomplishing that goal and the effects can be meaningfully evaluated…." 36 C.F.R. § 220.4(a)(1). Thus, when the FS declines to authorize a private application, the mere contemplation of that application does not constitute a federal proposal and the FS is not required to conduct an environmental analysis under NEPA.

As it is my determination not to consent to issuance of lease renewals based on the application before the agency at this time, preparation of an environmental analysis is not required. As further explained below, no significant environmental effects will occur as a result of the agency's no-consent determination.

This outcome is entirely in keeping with NEPA and its implementing regulations. Situations like this pose the unusual question of whether NEPA requires consideration of environmental effects of federal actions that foreclose development or use of natural resources. NEPA does not require a federal agency to consider effects arising from an action it has declined to allow third parties to undertake when that does not represent change in the physical environment caused by the federal action itself. In other words, only federal actions with significant environmental effects trigger NEPA's detailed statement requirement. Actions which do nothing to alter the natural physical environment and maintain the environmental status quo are not subject to NEPA.

The FS routinely prescreens non-mineral, special use authorization applications and agency regulations direct that non-conforming uses do not need to receive further evaluation and processing. See 36 C.F.R. § 251.54(e) (2). The FS does not have regulations governing consideration of discretionary mineral leasing applications, but agency practice is consistent.

As recently as 2014, Regional Forester Atkinson rejected a request for consent to a prospecting permit on the Hiawatha National Forest without preparing a NEPA document. Diverting scarce budgetary resources to prepare NEPA documents for proposals that will not move forward trivializes NEPA and diminishes its utility in providing useful environmental analysis for actions that the agency accepts and actively evaluates for approval.

In these circumstances, the Court of Appeals' Eighth Circuit holding that a FS decision to refrain from using herbicides as a method of vegetation control is not a "proposal or action to which NEPA can apply" pertains. *Minnesota Pesticide Information and Educ., Inc. v. Espy,* 29 F.3d 442, 443 (8th Cir. 1994).

NFS Land Management Perspectives

Half of a century has passed since TMM's leases were issued in 1966. The original leases were issued prior to statutes such as the National Historic Preservation Act of 1966, National Environmental Policy Act of 1969, Clean Water Act of 1972, Endangered Species Act of 1973, National Forest Management Act of 1976, and Boundary Waters Canoe Area Wilderness Act of 1978. Without these laws in place the environmental consequences of potential "commercial development [of the nickel and copper deposit] by a large-scale mining operation" originally envisioned by BLM in 1956 on what are now TMM's leases received markedly less consideration in comparison with current requirements. Given changes in policy and information availability, it is not unreasonable to anticipate a higher level of interest and concern regarding these consequences than when TMM's leases were originally issued, as demonstrated in the examples to follow.

In 1991 the Minnesota Department of Natural Resources recognized the value of the BWCAW for its scenic beauty and solitude by establishing a State Mineral Management Corridor. In light of surface water flow and recreational uses, no surface disturbance or state leases may be offered in the Corridor. The State Mineral Management Corridor overlaps with federal lease MNES-1353.

The federal relationship with Native American tribes has also evolved significantly over the 50 years since the TMM leases were issued. The FS has a legal obligation to acknowledge rights of Tribes and tribal members, including off-reservation rights to hunt, fish, gather and continue cultural and spiritual practices. Such recognition did not occur until the late 1970s when Indians began to assert their rights to off-reservation resources in federal court, including those rights to fish and gather wild rice. (E.g.: *Lac Courte Oreilles Band of Lake Superior Chippewa Indians v. State of Wis.*, 653 F. Supp. 1420 (W.D. Wis. 1987) (LCO III), *Lac Courte Orelles Band of Lake*

Superior Chippewa Indians v. State of Wis., 668 F. Supp. 1233 (W.D. Wis. 1987) (LCO IV)). No documentation suggests that consultation occurred or treaty rights were considered in the 1966 decision to grant the two leases.

Finally, since the last renewal of TMM's leases in 2004, we have gained experience with copper sulfide ore mining in different parts of the country. It is clear that these types of mines pose substantial risk of failure and environmental mitigation and remediation technologies are limited, and often ineffective, as discussed later in this letter. Awareness of the environmental effects of mining, specifically those from copper-nickel mining, has increased since 2004. While economic values are important to area communities and the nation, preserving Wilderness Areas and their associated qualities also have national and local support and precedent.

Evaluation of the Present Lease Application

In light of the M-Opinion's legal conclusion that TMM does not have the right to automatic renewal of its leases MNES-01352 and MNES-01353, on March 8, 2016 the BLM notified TMM that the agency would review the company's lease renewal application using the same criteria that are employed in deciding whether to grant initial hardrock mineral leases. The BLM's letter also specified that as part of its consideration of TMM's lease renewal application, the BLM would ask the FS whether it consents to the leases' renewal. In response to the BLM's June 3, 2016 letter making that request of the FS, the agency began considering whether to consent to the renewal of TMM's leases based upon the agency's recognition that it has full discretion to consent or withhold consent to the renewal of TMM's two leases.

As noted above, CEQ and FS NEPA regulations make clear that an application must be accepted by the agency as a proposal before NEPA obligations are triggered. At this time, the FS will not consent to lease renewal based on the submitted application and therefore does not have a goal that it is actively pursuing to authorize such activities. For this reason, no NEPA analysis is required.

Acid Mine Drainage

Bedrock geochemistry in northeastern Minnesota plays a large role in the low buffering capacity of the lakes and streams in the region. Both the Minnesota Pollution Control Agency and the Environmental Protection Agency (EPA) have identified the surface waters of northeastern Minnesota as sensitive to changes in pH, acid deposition, and acid runoff. Unlike surface waters bounded by carbonate bedrock, or relatively thick carbonate rich glacial till where neutralization of acid runoff occurs through dissolution of limestone and exsolution of carbon dioxide from water, the waters of northeastern Minnesota are largely underlain by igneous and metamorphic bedrock with thin overlying soils and surficial deposits with little acid neutralization capacity.

A risk of mining development is acid mine drainage (AMD). AMD generally occurs when sulfide minerals present in ore bodies and rock overburden are exposed to air and water. The exposure to air (oxidation) and water (hydrolysis) creates sulfuric acid, which subsequently increases water pH and leaches harmful metals such as copper, zinc, lead, cadmium, iron and nickel. FS data indicates between 20,000 and 50,000 mines currently generate acid on lands managed by the agency. Negative impacts from these mines affect 8,000 to 16,000 km of streams. While AMD can originate naturally from the ore body itself, its likelihood is dramatically increased by the generation of any mining product (stockpiles, overburden, and tailings) exposed to air and water, and can continue for decades.

Hardrock mines in sulfide bearing mineralization are known worldwide for producing AMD that requires continuous management and perpetual water treatment. Production of AMD is prevalent in all mining operation elements: construction, waste rock, tailings, and mine structures such as pits and underground workings. Acid drainage is one of the most significant potential environmental impacts at hardrock mine sites.

Water from a mine site could potentially enter streams and lakes through wastewater treatment plant discharges, uncollected runoff and leakage, concentrate spills, pipeline spills, truck accidents, spillway releases, tailings dam failures, water collection and treatment operation failures, and post-closure failures. All carry some risk to the environment. The magnitude and setting of a failure would drive the significance of the environmental risk and its potential impact.

The AMD increases lake and stream acidity, with potential risks to aquatic life including sport fisheries. A decline in water quality and aquatic species would have a negative effect on recreational visitors to the BWCAW. For example, the USGS estimated that in 2010 approximately 3,000 miles of Pennsylvania streams degraded by acid mine drainage led to approximately $67 million in lost sport fishing revenue each year.

Mining accidents are inherently unpredictable and can result from geotechnical failures or human error. Other circumstances that can affect the likelihood of mining failures or discharges include changing metals markets, financial crises, political events, and climate change. In addition, climatic trends affecting the frequency and magnitude of storm events and seasonal temperatures could lead to unpredicted environmental changes in vegetative composition, water quality and quantity, and wildlife habitat making the environment more susceptible to damage resulting from mining operations.

There is a direct flow of water from the lands subject to TMM's leases to the BWCAW. Specifically, the leases are located within the South Kawishiwi River Watershed and the Birch Lake Watershed which both are catchments of the Rainy River Watershed. Water flows from the lands embraced by the northern lease into the South Kawishiwi River which in turn flows into Birch Lake. Water from the lands embraced by the southern lease also flows into Birch Lake and Birch Lake empties into the main Kawishiwi River and then into the BWCAW.

TMM's leases overlay the Duluth Complex known for nickel-copper-platinum group element ore deposits. Due to the inherent sulfide chemistry of this ore type, mining facilities and byproducts can produce significant amounts of acid. Consistent with the footprint and infrastructure of similar mines, as well as publically available preliminary information from TMM about this specific site, TMM's potential project area could include underground mine(s) producing mainly copper and nickel, plus smaller amounts of other metals. TMM's project would require a concentrator facility (potentially 1-2 miles west of the mine(s)), a tailing storage facility (potentially 13 miles southwest of concentrator), and connecting utility corridors. The utility corridors would include roads, rail lines, power transmission lines, natural gas pipelines, tailing and concentrate pipelines, and water pipelines. TMM's Pre-Feasibility Study also reveals that its project would involve four delineated ore bodies – Maturi, Maturi Southwest, Birch Lake, and Spruce Road – all of which are north and east of the Laurentian Divide and thus in the watershed draining towards BWCAW.

TMM's mining operations are expected to dispose of some waste rock and tailings underground. Other waste rock and tailings would be disposed of using surface facilities. All of the waste rock and tailings derived from the sulfide ore bodies on the leases would have a high likelihood of oxidizing and becoming sources of AMD. TMM's Technical Report on Pre-Feasibility Study shows that TMM's subsurface mining operations would occur north of the Divide and present BWCAW contamination risks. That is also true of TMM's ore processing concentrator facilities. But TMM's Technical Report on Pre-Feasibility Study shows that TMM's tailings disposal facilities potentially would be south of Laurentian Divide in the Superior Watershed, which drains away from the BWCAW.

There are limitations in understanding the full contours of the mineral operations that ultimately might occur on TMM's leases, including the location of important features such as its tailings disposal facilities. The pre-feasibility study is an economic feasibility analysis, not TMM's final proposal to mine the hardrock mineral deposits. But pursuant to the terms of both 16 U.S.C. § 508b and Section 402 of Reorganization Plan No. 3 of 1946, 60 Stat. 1097, 1099-1100, the FS's consent is required for hardrock mineral development and the purpose of any lease, whether it is one issued in the first instance or a subsequent renewal, is mineral development. Indeed, the M-Opinion explicitly recognizes that "the entire purpose" of a mineral lease is "for the lessee to develop the minerals…." Another M-Opinion reports that since the 1970s hardrock prospecting permits for NFS lands, which are the precursor for the issuance of Preference Right hardrock mineral leases including MNES-01352 and MNES-01353, have been issued subject to the condition that "no mineral development of any type is authorized hereby." M-36993, Options Regarding Applications for Hardrock Mineral Prospecting Permits on Acquired Lands Near A Unit Of The National Park System (1998 WL 35152797 (April 16, 1998)). *See also John A. Nejedly Contra Costa Youth Association*, 80 IBLA 14, 26 (1984) (concurring opinion) (development under a preference right lease "is a logically foreseen result of successful prospecting").

Another factor relevant to assessing the likelihood of AMD if TMM develops a mine on the lands subject to the two leases it seeks to renew is that the waters in the Rainy River watershed flow largely through bedrock fractures with limited carbonate rock surface area. Therefore the watershed has low capacity to buffer AMD.

In sum, given the hydrology and hydrogeology of this area, the likelihood of these ore bodies being exposed to water is very high, and given these particular ore bodies' composition, resulting drainage from the mine workings and mining wastes are likely to be highly acidic.

Lessons from Similar Copper Sulfide Mines

Contamination from mining operations can also occur instantaneously via catastrophic failure of the type that occurred in 2014 at the Mount Polley Mine in British Columbia, Canada and at other copper mines. A review of water quality impacts from 14 operating U.S. copper sulfide mines found: 100% of the mines experienced pipeline spills or accidental releases; 13 of 14 mines' water collection and treatment systems failed to control contaminated mine seepage resulting in significant water quality impacts; tailings spills occurred at 9 operations; and a partial failure of tailing impoundments occurred at 4 mines. The inherent risks of mining hardrock mineral deposits on the lands leased to TMM set a high bar for potential mineral development within this watershed due to potentially severe consequences for the BWCAW resulting from such failures. Because of the hydrology and hydrogeology of this particular area, should contamination occur, it could cover a very broad region.

Recent reviews of similar mining proposals in Minnesota and Alaska highlight inherent risks of metal mining to natural resources, and provide examples of risks associated with long term effectiveness of planned containment strategies. In Minnesota, the Final Environmental Impact Statement for nearby NorthMet Mining Project and Land Exchange recognizes that no matter the depth of analysis and planned containment strategies there remain uncertainties associated with mine development, operation and long-term water and waste rock treatment.

Similarly, the EPA, in a Proposal Determination Pursuant to Section 404(c) of the Clean Water Act for the Pebble Mine in Alaska, warns that, "There is also real uncertainty as to whether severe accidents or failures, such as a complete wastewater treatment plant failure or a tailings dam failure, could be adequately prevented over a management horizon of centuries, or even in perpetuity, particularly in such a geographically remote area subject to climate extremes. If such events were to occur, they would have profound ecological ramifications." While the ramifications of these risks are possibly greater in the case of the Pebble Mine, due to its location, the BWCAW shares many similarities in terms of hydrogeology, extreme weather and remoteness.

Unique Attributes of Copper Sulfide Ore Mining in the BWCAW Region

Many operating copper mines in the United States are situated in the arid southwest or other drier areas of the Nation. Northern Minnesota has an established history of taconite mining - indeed, the region to the west of the lease sites is known as the "Iron Range." However, taconite is an iron-bearing oxide ore. Mining of the copper-nickel sulfide ore found on TMM's leases is untested in Northern Minnesota. This lack of experience with copper-nickel sulfide ore mines in environments with the complex hydrogeology of northern Minnesota complicates assessment of the consequences of mining operations on TMM's leases, which could occur if those leases are renewed.

Another variable in assessing the consequences of these operations is climate change. In Minnesota, mean annual temperatures are expected to continue rising and precipitation is expected to increase, along with the size and magnitude of weather events. An increase in precipitation and water supply in association with significant events could exacerbate the likelihood of AMD and water resource contamination. The projected changes in climate and associated impacts and vulnerabilities would have important implications for economically important timber species, forest dependent wildlife and plants, recreation, and long-range planning. The combined impacts of contaminants from mineral development and climate change could impact the ecosystem resilience of the BWCAW and the Superior National Forest outside of the wilderness.

The NorthMet Mining Project and Land Exchange, the first copper-nickel mine proposed in Minnesota, has similar concerns regarding AMD, climate change, and water quality. These concerns were addressed in NorthMet's final EIS through engineering, permitting, and monitoring requirements. Significantly, the NorthMet project is located in an area either previously disturbed and/or surrounded by brown-field taconite open pit mines and waste piles in the Laurentian Watershed, which drains away from the BWCAW. In contrast, TMM's leases are in close proximity to the BWCAW and within its high quality watershed resource of outstanding value. The inherent and legislated wilderness values and untrammeled qualities of the BWCAW contrast with the extensively disturbed surroundings of NorthMet's location. Additionally, if there is any potential for NorthMet's copper-nickel mining project to affect the BWCAW and MPA, this potential would be far less than that associated with any copper-nickel mining operations TMM might ultimately conduct.

If TMM ultimately conducts mining operations on lands subject to its two leases and they result in AMD, metal leaching, and water contamination, very few of the available containment and remediation strategies would be compatible with maintaining the BWCAW's quality and character. Available containment and remediation strategies such as sediment basins, water diversions, or construction and long-term operation of water treatment plants have the potential to deleteriously affect the BWCAW. Of particular concern, given the location of TMM's leases, is the effectiveness of available methods to counteract AMD in the case of seepage, spills, or facility failures. Water is the basic transport medium for contaminants. Consequently, all measures aimed at controlling AMD generation and migration involve controlling water flow. To reduce the generation and release of AMD, the infiltration of meteoric water (rain and snow) can be retarded through the use of sealing layers and the installation of under-drains, respectively. Diversion of contaminated water most commonly requires installation of ditches or sedimentation ponds. But even with the use of these measures successful long-term isolation of intercepted contaminated groundwater is, at best, very difficult to achieve.

Moreover, even if available remediation techniques to handle contaminated water, such as flushing, containment and evaporation, discharge through wetlands, neutralization and precipitation, desalination, water treatment plant construction and operation, utilization of ditches or sedimentation ponds, and installation of cut-off walls, trenches or wells, are effective, very few, if any, of them are compatible with maintaining the quality and character of BWCAW and MPA, as required by the Boundary Water Canoe Area Wilderness Act. Given the TMM's leases' proximity to the BWCAW's boundary (adjacent to in one case and less than 3 miles distant in the other) and the direct transport route of surface water from Birch Lake and the Kawishiwi River, it is reasonable to expect direct effects of any mining operations on those leases to the BWCAW and MPA.

Potential Impacts to Water, Fish, and Wildlife

As noted above, the potential for environmental harm is inherent to copper-nickel and other sulfide-bearing ore mining operations. This potential exists during all phases of mine development, mineral extraction and processing, and long-term mine closure and remediation. Expected environmental harm could encompass damage to both surface and ground water resources, including changes in water quantity, quality, and flow direction, contamination with acid and leached metals resulting from AMD and tailings disposal facility failures, and more. It is also well established that this environmental damage can adversely affect fish populations and aquatic ecosystems directly and by indirect effects on food supplies and habitat. Recognizing this potential harm, the second edition Rainy-Lake of the Woods State of the Basin Report (2014) recommends scientifically examining the effect of new mining proposals on water quality in the Rainy River Watershed.

TMM's leaseholds lie within the Rainy River's Birch Lake Sub-Watershed (HUC 10) which the SNF has identified as a priority watershed per the FS's Watershed Condition Framework. The Framework is a comprehensive approach for: 1) evaluating the condition of watersheds, 2) strategically implementing integrated restoration, and 3) tracking and monitoring outcome based program accomplishments. According to the Watershed Restoration Action Plan for Birch Lake the watershed is currently functioning at risk, based on fair ratings for aquatic biotic condition, water quality condition, aquatic habitat condition, soil condition, and fire effects/fire regime condition. The Action Plan recognizes that further development in the watershed has the potential to move the watershed from its suboptimal level of functioning at risk to the worst level of impaired functioning.

As noted previously, the BWCAW and SNF are home to dozens of sensitive species. Three species, the Canada Lynx, gray wolf and northern long-eared bat, are listed as threatened. Crucially, the BWCAW and SNF are considered critical habitat for the threatened Canada Lynx, which requires spruce-fir boreal forest with dense understory. Canada Lynx cover large areas, traveling extensively throughout the year, meaning that development and habitat fragmentation can affect the viability of lynx populations.

The threatened northern long-eared bat lives in both Lake and St. Louis County, where TMM's leases are located. The northern long-eared bat spends its winter hibernating in caves. In summer it roosts in both live and dead trees, as well as caves. Northern long-eared bat populations are under significant stress from White-nose Syndrome, which has caused drastic declines in bat populations across the country. Increased impacts to their habitat could exacerbate population decline.

The gray wolf population in the western Great Lakes, including the BWCAW, was re-listed as threatened in 2014 by the Fish and Wildlife Service. Gray wolves also cover large areas to hunt, so wolf populations can be impacted by development and habitat fragmentation. Other animals benefit from wolves living in northern Minnesota as carcasses wolves leave behind feed many other animals.

Northern Minnesota is one of the few places in the continental U.S. where visitors can see moose. However, the state's iconic moose population continues to decline – decreasing by approximately 60 percent in the last decade, according to Minnesota's State Department of Natural Resources. Given this population decline, the U.S. Fish and Wildlife Service (FWS) initiated a status review for the U.S. population of northwestern moose (i.e., those in Michigan and Minnesota). The status review was initiated as a result of a positive 90-day finding on a petition to list moose under the Endangered Species Act. FWS determined information in the petition provided substantial scientific or commercial information indicating that species listing may be warranted.

Moose often gather around ponds, lake shores, bogs and streams where they feed on aquatic vegetation. They are under stress from climatic change, likely due to a greatly increased number of ticks brought about by warmer summers. Therefore they are ever more dependent on the extensive, high quality habitat available in the BWCAW. Additional development, such as mining activity and associated road building, in the vicinity of the BWCAW could lead to habitat fragmentation that may further stress the moose population. While contamination of BWCAW waters by acid and leached metals could lead to habitat degradation that would also add to the moose population's stress.

The potential impacts of mining activities also could affect other species dependent upon forested areas through habitat fragmentation, increased dispersal of invasive plant and animal species, and alterations to wildlife migration and residence patterns.

Social and Economic Considerations

The State of Minnesota has primary responsibility under the Clean Water Act of 1972 to protect the water quality of the BWCAW and identifies the wilderness area as an "outstanding resource value water" under Minnesota Rules (Minn. R. 7050.0180). That section also provides that "[n]o person may cause or allow a new or expanded discharge of any sewage, industrial waste, or other waste to waters within the Boundary Waters Canoe Area Wilderness."

On March 6, 2016, Minnesota Governor Mark Dayton sent, and publicly released, a letter to TMM stating that he had directed the State's Department of Natural Resources "not to authorize or enter into any new state access or lease agreements for mining operations on those state lands" near the BWCAW. The Governor stated he has grave concerns about the use of state surface lands for mining near the BWCAW:

"[M]y concern is for the inherent risks associated with any mining operation in close proximity to the BWCAW and … about the State of Minnesota's actively promoting advancement of such operations by permitting access to state lands."

"As you know the BWCAW is a crown jewel in Minnesota and a national treasure. It is the most visited wilderness in the eastern US, and a magnificently unique assemblage of forest and waterbodies, an extraordinary legacy of wilderness adventure, and the home to iconic species like moose and wolves. I have an obligation to ensure it is not diminished in any way. Its uniqueness and fragility require that we exercise special care when we evaluate significant land use changes in the area, and I am unwilling to take risks with that Minnesota environmental icon."

As a partner in managing and conserving natural resources within the State of Minnesota, the FS takes Governor Dayton's statements seriously. The FS shares many of the Governor's concerns. These shared concerns also support the decision to withhold consent to renewal of leases MNES-01352 and MNES-0153.

The FS was aware of negative public sentiment regarding other mineral related projects on nearby SNF lands and many people's concern about the possible renewal of leases MNES-01352 and MNES-01353. Consequently, on June 13, 2016 the FS announced it would provide a 30-day public input period commencing June 20, 2016 and including a listening session on July 13, 2016 to better understand public views about renewal of TMM's two leases. A second listening session on July 19, 2016 was subsequently announced.

Individuals and organizations expressed passionate views both in support of and opposition to renewing the leases during the input period and listening sessions. In addition, TMM submitted comments for the record during the public input period. Overall the FS received over 30,000 separate communications is response to the listening sessions. In total, this input provided FS decision makers the fullest possible understanding of public views and concerns regarding the proposed lease renewals.

Local sentiment is similarly mixed regarding the desirability of TMM developing a mine on the lands subject to its two leases. Northeastern Minnesota has a long history of mining, and much of the local economy along the Iron Range remains dependent on iron mining. Ely, Virginia, and other local communities, have a long-standing social identity associated with mining. During the two listening sessions, elected officials, union representatives, and miners expressed their concerns regarding the future of these communities, mining-associated tax revenues that support schools and local services, and high-paying jobs for future generations. These mining proponents often cited the potential economic benefits of mining, should TMM develop a mine on its leases. They also stated that young people and families are leaving the area due to a depressed local economy. Mining proponents also referred to the need for strategic metals for American industry and national defense, including their use in sustainable technologies such as wind turbines and hybrid cars.

Those who oppose TMM's development of a mine on the lands subject to its two leases emphasize the copper-nickel mining industry's history of causing serious environmental harm, the potential mine's proximity to the BWCAW, the interconnected hydrology of the leased lands and the BWCAW, and the probable negative impacts to water quality, quantity and aquatic ecosystems downstream from any mine TMM establishes. These mining opponents often stated that mining has created a boom-bust economy that only now has stabilized with the creation of sustainable recreation-based jobs reliant on an unspoiled environment. They also raised concerns about the probable negative impacts any TMM mine would have on the quality of individuals' future recreational experiences in the BWCAW, maintenance of the BWCAW's wilderness character, and preservation of the BWCAW for future generations.

In its Technical Report on Pre-Feasibility Study, TMM estimates the company's initial capital investment for mine construction will be $2.77 billion while over the projected 30-year life of the mine its total capital investment will be $5.41 billion. TMM also estimates the potential economic contributions of mining the copper-nickel deposits underlying its two leases could include the need for close to 12 million labor hours during the estimated three-year mine construction period and approximately 850 full-time jobs when the mine becomes operational.

Based on accepted multipliers of direct and indirect economic contribution, TMM's mining operations predicated upon its two leases might generate approximately 1,700-1,900 additional indirect jobs in the region's economy.

Conversely, across the country, counties with designated wilderness areas are associated with rapid population growth, greater employment, and enhanced personal income growth, relative to counties lacking wilderness areas. This is attributable to the increasing mobility of service jobs, and many entrepreneurs' preference to locate their businesses in areas offering a high quality of life. Specifically, up to 150,000 visitors visit the BWCAW annually. Economic benefits generated by BWCAW-related recreation have been estimated at approximately $44.5 million annually. The wilderness recreation-based tourism and any derivative economic return is dependent upon preserving the BWCAW's natural quality and wilderness character.

With passage of the Boundary Waters Canoe Area Wilderness Act in 1978, the business model of industries and communities associated with the BWCAW shifted. Timber production was halted. Many resorts located within the wilderness were bought out by the federal government and others received financial assistance to shift to a wilderness based business model. Gateway communities such as Ely, Tofte and Grand Marais have also shifted to wilderness based economies. While the transition has been long and often difficult these communities are now highly dependent on revenue generated by the BWCAW for economic sustainability. Potential unforeseen impacts to natural resources and water quality within the BWCAW would likely result in substantial economic impacts to established local businesses and communities now dependent upon a wilderness based business model.

On April 15, 2015, Congresswoman Betty McCollum (D-MN) introduced the National Park and Wilderness Waters Protection Act (H.R. 1796). The Act would withdraw all federal lands in the Rainy River Watershed from the mining laws, the mineral leasing laws, and the laws governing the disposal of mineral materials, subject to valid existing rights. The Act also would impose additional restrictions on the issuance of any lease or permit for mineral related activities. In a February 2, 2016, letter to the Secretaries of Agriculture and the Interior and the Director of CEQ, Congresswoman McCollum urged them "to immediately take action to protect two of America's natural treasures – the BWCAW and Voyageurs National Park." Specifically, Congresswoman McCollum requested the denial of TMM's requested lease renewals and administrative withdrawal of the Rainy River watershed.

Former Vice President—and former Minnesota Senator—Walter Mondale also has advocated that the Department of the Interior deny the renewal of TMM's leases and withdraw all federal minerals in the BWCAW's watershed. On April 1, 2016, he wrote that "Arizona has its Grand Canyon, Wyoming its Yellowstone, California, its Yosemite. These wonders come to mind unbidden as images of a place when those states are named. The Boundary Waters is such an image for Minnesota." Vice President Mondale goes on to say:

"Vice President Hubert Humphrey and I were deeply committed to protection of the Boundary Waters and its precious waters. Although we were mindful of the need for jobs, we knew that it was important to protect the magnificence of the Boundary Waters. The Twin Metals mining proposal lacks this balance. That means that today I join Minnesota's Gov. Mark Dayton and urge the federal land management agencies to continue the work of nearly 100 years and to ensure that the Boundary Waters wilderness remains the place it is today."

Then in a July 1, 2016 letter characterizing the BWCWA as pristine and irreplaceable wilderness, Vice President Mondale warned that the kind of heavy-metal mining that TMM proposes:

"…is in a destructive class all its own. Enormous amounts of unusable waste rock containing sulfides are left behind on the surface. A byproduct of this kind of mining is sulfuric acid, which often finds its way into nearby waterways. Similar mines around the country have already poisoned lakes and thousands of miles of streams. The consequence of acid mine drainage polluting the pristine Boundary Waters would be catastrophic. It is a risk we simply can't take."

Conclusion

The FS understands the important economic and national security benefits provided by mineral extraction and supports mining as a legitimate activity on NFS lands. However, mining is not appropriate on all places within the NFS or on every acre of NFS lands. When evaluating whether to consent to issuance of an initial lease or the lease's renewal, the FS may consider the unique ecological and cultural attributes of all NFS lands that might be adversely affected by mineral development on the leasehold along with the social and economic consequences that could flow from both a decision to consent and to withhold consent. The FS also has an affirmative responsibility to protect and maintain the character and quality of the BWCAW and MPA for present and future generations. Sec. 2, Pub. L. 95-495, 92 Stat. 1649 (1978). Thus the agency may weigh the possible benefits of TMM's potential mineral development against the possible harm TMM's potential mineral development might do to the BWCAW's uniquely valuable landscape.

TMM's potential mineral development on its two leaseholds might contribute markedly to employment and economic growth in St. Louis County, Lake County, and nearby areas. Copper-nickel mining conducted by TMM also would furnish metals important to U.S. industries and modern technology. Deposits of copper are relatively abundant in the United States and many operating copper mines in the United States are situated in arid or drier areas of the Nation where their potential for environmental harm may be reduced. The United States Geological Survey reported that as of 2015 there was only one operating nickel mine in the United States but nonetheless nickel was in oversupply and three other U.S. mining projects that would supply nickel were in development.

The BWCAW contributes to the cultural and economic sustainability of communities within the State of Minnesota, the Nation and beyond and to the ecological sustainability of unique landscapes and rare species dependent upon those landscapes that are valued within the State of Minnesota, the Nation and beyond. The BWCAW is irreplaceable, but likely irreparable in the event of its significant degradation.

Based on information provided by TMM to date (e.g., its Technical Pre-Feasibility Report), existing science, and examination of similar proposals, there is no reason to doubt that the mining operations TMM hopes to eventually conduct could result in AMD and concomitant metal leaching both during and after mineral development given the sought after copper-nickel ore is sulfidic. This fact is very significant given TMM's two leases are adjacent or proximate to the BWCAW and within the same watershed as the wilderness. It might be possible for TMM to develop a mine which employs mitigation and containment strategies that reduce the mine's potential to cause AMD and leached metals that could harm the wilderness. However, at the very least it is equally possible that available water treatment technologies would be unable to prevent the spread of any AMD and leached metals in the watershed. Further, there appears to be even less likelihood that any contamination of the BWCAW resulting from TMM's mining operations could later be remediated, especially not in a manner compatible with the BWCAW's wilderness character. Moreover, any degree of contamination of the BWCAW by AMD and leached metals has the potential to seriously degrade the wilderness area's character and quality. Thus, even if the probability that TMM's mining operations might generate and release of AMD and leached metals was very low, which the FS does not believe to be the case, the environmental harm to the BWCAW that could result from any contamination of the area with AMD and leached metals might be extreme. Failing to prevent such damage also is contrary to Congress' determination that it is necessary to "protect the special qualities of the [BWCAW] as a natural forest-lakeland wilderness ecosystem of major esthetic, cultural, scientific, recreational and educational value to the Nation." Sec. 1, Pub. L. 95-495, 92 Stat. 1649 (1978).

Balancing what are primarily economic benefits of the mining operations that TMM hopes to conduct in connection with the renewal of its two leases against even a remote possibility of damaging the BWCAW—a unique ecosystem that Minnesota elected officials have fittingly called irreplaceable and a national treasure—makes it clear that it is incumbent upon the FS to withhold consent to the renewal of TMM's leases MNES-01352 and MNES-01353.

This decision withholding consent to the renewal of TMM's leases is subject to discretionary review by the Under Secretary for Natural Resources and Environment pursuant to 36 C.F.R. § 214.7(b), but not appeal pursuant to 36 C.F.R. part 214 (36 C.F.R. § 214.7(a)(2)). No additional information may be considered by the Under Secretary for Natural Resources and Environment in connection with the discretionary review of this decision (36 C.F.R. § 214.19(b) & (e)).

Sincerely,

Thomas L. Tidwell

THOMAS L. TIDWELL
Chief

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September 30, 2017

To: Kevin Lee, Minnesota Center for Environmental Advocacy

From: Jim Kuipers PE, Kuipers & Associates

Re: **Review of Northmet Mining Project Dam Safety Permit**

Please find the following comments related to the Northmet Mining Project Dam Safety Permit. The comments address four specific areas, all of which relate to engineering best practice as currently recognized by the mining industry for tailings storage facilities (TSFs). The first area of comment is with respect to Minnesota's current regulations for dam safety and their adequacy with respect to tailings storage facilities; with the exception of Montana which has enacted more current regulations in response to the Mount Polley Independent Expert Review Panel (IERP) recommendations most states in part or in whole rely on regulations specific to water storage dams which differ significantly from tailings storage facilities. The second area of comment is with respect to the current criteria which the Minnesota Department of Natural Resources (DNR) considers in approving a dam safety permit to address public safety; the requirement for clearly defined and specified design performance criteria is evident in most regulatory approaches. And the third area is with respect to how DNR considered what happened at Mount Polley in its review. Finally, the fourth area is with respect to the actual engineering analysis and reviews that have been performed for the Northmet TSFs.

The comments provided are based on more than 35 years of professional experience which has included significant levels of involvement in tailings storage facility design, permitting, operations, reclamation and closure, and long-term monitoring, maintenance and operations, as well as financial assurance for tailings storage facilities and associated requirements such as supernatant and seepage water capture and treatment. I was the government appointed Technical Liaison for the Mount Polley IERP and the tailings dam re-stabilization engineering effort on behalf of the primary affected First Nations, and was responsible for assisting them in understanding and responding to the reports and recommendations related to those activities. I was also involved in the development of the 2015 Montana tailings dam safety regulations and have spent significant time assessing and directly participating at a number of mine sites in current industry best practice for tailings storage facilities. The overall objective of these comments is to lend my professional expertise to the determination by the State of Minnesota and its citizen's in how best to address public safety and environmental issues with respect to the storage of tailings from mining activities.

The comments provided are intentionally thorough and include numerous citations and references to professional and governmental regulations and guidance identifying current best practice for TSFs. In many cases the references are not available to the public and regulators, in particular the Canadian Dam Association reports cited herein. In addition, clarifying information is provided as to the views of the Mount Polley IERP that should be acknowledged if their work is to be cited relative to Dr. van Zyl's involvement at a TSF evaluation. The objective of including this level of information is to ensure that both regulators and the public are informed as to current best practice as well as ongoing controversy as to in particular the application of Best Available Technology (BAT).

1. Dam Safety and Tailings Storage Facilities

In most jurisdictions nationally and internationally, mining TSFs are regulated in terms of stability and hydrology based on requirements designed and intended for water storage dams. In some cases, those regulations may include practices applicable to mining TSFs, however only one U.S. state, Montana, has enacted regulations specific to mine TSFs. Mining TSFs are different from water storage dams in the following ways (sources, Martin et al 2002, CDA 2014):

- TSFs are constructed, operated and closed by mine owners focused on extraction for a profit and not necessarily for public benefit or on TSF safety.
- TSFs are designed to retain solids (that may or may not be contaminated) and/or process solutions (that may or may not be contaminated).
- TSFs can contain large quantities of fluids and solids that if released can cause significant environmental damage and result in loss of human life.
- TSFs are built during the development and operation of mines and remain as part of the landscape becoming a permanent feature that must perform as designed after closure of the mine indefinitely (e.g. in perpetuity).
- TSFs, if they contain contaminated substances (fluids and/or solids), have no minimum size where the consequences of failure would be generally acceptable.
- Many TSFs are built in stages over the mine life, rather than built in a single stage prior to decommissioning.
- The condition of TSFs is continually changing so safety must be continually re-evaluated rendering TSF management more onerous as a steady-state condition is only achieved some time after the mine operations cease.
- TSF decommissioning cannot be accomplished by breaching and removal but instead typically requires a transition period and long-term monitoring and maintenance.
- TSFs are not generally viewed as an asset but instead as a liability and thus may warrant a lower standard of care from their owners.
- TSF owners typically rely on consultants rather than in-house expertise leading to the potential for poor communication and project continuity.

As discussed further in these comments, Minnesota's existing dam safety statutes, RSM 6115.0410, are intended for water storage dams and do not specifically address tailings storage facilities. In fact, because the requirements rely on "current, prudent engineering practice" and "prudent, current environmental practice" rather than on current accepted industry engineering standards it is questionable as to whether Minnesota's existing dam safety statutes are consistent with accepted standards for water storage facilities, much less TSFs.

2. Dam Safety Performance Standards

a. Hazard Classification

Minnesota classifies existing and proposed dams into three hazard classes (RSM 6115.0340):

those dams where failure, misoperation, or other occurrences or conditions would probably result in:

A. Class I: any loss of life or serious hazard, or damage to health, main highways, high-value industrial or commercial properties, major public utilities, or serious direct or indirect, economic loss to the public;

B. Class II: possible health hazard or probable loss of high-value property, damage to secondary highways, railroads or other public utilities, or limited direct or indirect economic loss to the public other than that described in Class III; and

C. Class III: property losses restricted mainly to rural buildings and local county and township roads which are an essential part of the rural transportation system serving the area involved.

Minnesota's classification system is consistent with the recommendations of the Federal Emergency Management Agency (FEMA), which is responsible for federal dam safety requirements. FEMA (2013a) notes that significant variations of the dam classification system are in use which is problematic and therefore suggests the following hazard potential classes for dams in Table 1.

Table 1: Recommended Dam Classification System Based on Hazard Potential (FEMA 2013a)

Hazard Potential Classification	Loss of Human Life	Economic Loss, Environmental Loss, and/or Disruption of Lifeline Facilities
High	Probable (one or more expected)	Yes (but not necessary for this classification)
Significant	None expected	Yes
Low	None expected	Low and generally limited to owner

Although the Minnesota regulations do not specifically require it, in practice, dam break and inundation studies are used to support assessment of the consequences of potential failure of dams in order to use a dam classification system. However, the Canadian Dam Association (CDA 2014) notes that the science of predicting tailings dam breaches and flows is relatively new, and techniques therefore limited. Additionally, the lethality of failures from TSFs may be different than for conventional dam breach flooding. For example, because the supernatant fluid and/or solids in a TSF are frequently contaminated, the incremental environmental consequences may be worse for a sunny day failure than a flood induced failure.

The CDA (2014) recommends a more detailed approach to dam classification for dams and mentions the following specifically for TSFs:

- Since many mining dams are remote from population centers, the potential for loss of life is often not as prevalent as it is for conventional dams. There could be occasions where there are people in the area downstream of the dam temporarily due to seasonal cottages, roads and highways, rail corridors, and recreational activities.
- Mining dams can also have the special case where the failure could threaten employees of the mine working downstream of the mining dam, such as in an open pit mine. In this instance, the training of the mine staff can be considered with respect to evacuation procedures and the potential for reducing the potential for loss of life.
- Environmental losses are often the most significant aspect of a mining dam failure. Specific studies may be required to predict the degree of environmental loss.
- The economic losses to a mining company can be substantial and may be much larger than the direct financial burden associated with a failure. Failures of mining dams can result in lost production, have a negative impact on the market capitalization of a company, and limit the ability of the company to engage in other mining projects.

- Mining dam failures can result in loss of site infrastructure such as roads, pump stations, power lines, and pipelines.

The CDA recommends a dam classification approach that also can be used to provide guidance on the standard of care expected of dam owners and designers. As loss of life is difficult to predict, it considers both population at risk as well as loss of life. It also separately considers environmental and cultural values from infrastructure and economics. The approach is presented in Table 2.

The CDA notes that:

- Because of the difficulty in predicting the environmental and ecosystem effects from accidental releases, it is often necessary to be on the conservative side when applying dam classifications.
- The owner must also consider the other consequences, as described above that the dam presents to their operation when establishing the risk profile and although this may not change the classification, the risk profile could have a bearing on the surveillance activities and design criteria

By definition, because the catastrophic failure of TSFs can result in the loss of life and is considered a serious hazard, TSFs are generally considered to be in the highest hazard category, or Class I in the case of Minnesota's classification system. Minnesota should consider revising its requirements to reflect current practice for TSF by adopting a more detailed and specific hazard classification system such as that recommended by the CDA, including a requirement for inundation studies using methods consistent with, and in consideration of further aspects specific to TSFs as provided in CDA (2014) guidance.

b. Performance Standards

Minnesota addresses dam safety in subpart 8. Performance standards which follows:

Subp. 8. Permit standards. Approval or denial shall be based on the potential hazards to the health, safety, and welfare of the public and the environment including probable future development of the area downstream or upstream. The applicant may be required to take measures to reduce risks, and the commissioner shall furnish information and recommendations to local governments for present and future land use controls to minimize risks to downstream areas.

The commissioner shall determine if the proposal is adequate with respect to:

A. For Class I, a showing of lack of other suitable feasible and practical alternative sites, and economic hardship which would have a major adverse effect on population and socioeconomic base of the area affected.

B. For Class II, a showing of lack of other suitable feasible and practical alternative sites and that the dam will benefit the population or socioeconomic base of the area involved.

C. The need in terms of quantifiable benefits.

D. The stability of the dam, foundation, abutments, and impoundment under all conditions of construction and operation, including consideration of liquefaction, shear, or seepage failure, overturning, sliding, overstressing and excessive deformation, under all loading conditions including earthquake. This determination must be based on current, prudent engineering practice, and the degree of conservatism employed must depend on hazards.

E. Discharge and/or storage capacity capable of handling the design flood based on current, prudent engineering practice and the hazard classification.

F. Compliance with prudent, current environmental practice throughout its existence.

Table 2: CDA TSF Dam Classification (Source: CDA 2013, 2014)

Dam class	Population at risk [note 1]	Incremental losses		
		Loss of life [note 2]	Environmental and cultural values	Infrastructure and economics
Low	None	0	Minimal short-term loss No long-term loss	Low economic losses; area contains limited infrastructure or services
Significant	Temporary only	Unspecified	No significant loss or deterioration of fish or wildlife habitat Loss of marginal habitat only Restoration or compensation in kind highly possible	Losses to recreational facilities, seasonal workplaces, and infrequently used transportation routes
High	Permanent	10 or fewer	Significant loss or deterioration of *important* fish or wildlife habitat Restoration or compensation in kind highly possible	High economic losses affecting infrastructure, public transportation, and commercial facilities
Very high	Permanent	100 or fewer	Significant loss or deterioration of *critical* fish or wildlife habitat Restoration or compensation in kind possible but impractical	Very high economic losses affecting important infrastructure or services (e.g., highway, industrial facility, storage facilities for dangerous substances)
Extreme	Permanent	More than 100	Major loss of *critical* fish or wildlife habitat Restoration or compensation in kind impossible	Extreme losses affecting critical infrastructure or services (e.g., hospital, major industrial complex, major storage facilities for dangerous substances)

Note 1. Definitions for population at risk:

None—There is no identifiable population at risk, so there is no possibility of loss of life other than through unforeseeable misadventure.

Temporary—People are only temporarily in the dam-breach inundation zone (e.g., seasonal cottage use, passing through on transportation routes, participating in recreational activities).

Permanent—The population at risk is ordinarily located in the dam-breach inundation zone (e.g., as permanent residents); three consequence classes (high, very high, extreme) are proposed to allow for more detailed estimates of potential loss of life (to assist in decision-making if the appropriate analysis is carried out).

Note 2. Implications for loss of life:

Unspecified—The appropriate level of safety required at a dam where people are temporarily at risk depends on the number of people, the exposure time, the nature of their activity, and other conditions. A higher class could be appropriate, depending on the requirements. However, the design flood requirement, for example, might not be higher if the temporary population is not likely to be present during the flood season.

In our experience the federal agencies and most state agencies use a more prescriptive, standards based approach to dam safety performance standards. The CDA (2013) describes the traditional standards-based approach as follows:

"Established practice in safety assessment of dams relies mainly on a standards-based approach deterministic concept, largely because it is computationally straightforward; provides the reassurance of a well-known method; and uses numerical measures, such as safety factors. The deterministic approach requires the determination of stability or stress state for a critical region the dam or its foundation. These states are typically analyzed for a set of usual, unusual, and extreme load combinations. The deterministic loads and resulting stresses are then related to the deterministic ultimate stability and failure criteria. The quantitative definitions of the factors of safety are determined primarily by empirical evidence, experience, and engineering judgment.

A deterministic design or assessment of unique structures is typically based on either (i) worst case values for the input variables or (ii) nominal values with a safety factor applied to the results Thus, the approach accounts for uncertainty by

- Assuming conservative (extreme) values for the loads
- Assuming conservative (safe) values for resistance variables
- Applying conservative safety factors

The usual (normal), unusual, and extreme cases can be considered from the perspective of exceedance probability. The most critical loads-seismic and hydrotechnical-are to some extent characterized on the basis of statistics, reliability theory, and probability. In this way, the deterministic approach has been gradually transformed to a semi-probabilistic concept. The calibration and numerous simplifications introduced in the final format of a standards-based procedure are often hidden in the background, and thus the deterministic method may be called prescriptive.

It should be noted that a particular factor of safety is physically meaningful only with respect to given design assumptions and equations. Engineering guidelines or regulations may provide precise instructions for calculation of the factor of safety. This ensures a certain uniformity of approach on the part of different designers. However, practising engineers must have a full understanding of the actual reliability assessment methods and meanings of factors used to express the safety, durability, and serviceability of structural components."

i. Seismic Criteria

Table 3, from CDA's *Application of CDA Dam Safety Guidelines to Mining Dams* provides suggested target levels that can generally be applied to the Construction, Operation, and Transition Phases of a TSF. CDA suggests that these are intended for consideration and consultation between the owner and regulator, and that the owner may adopt, or regulations may require, more stringent criteria. The CDA also notes that for TSFs, crest deformations could be much larger compared to conventional dams, and result in release of contents. They suggest that "criteria should be established for suitable deformations of a mining dam and the appropriate analyses undertaken to demonstrate the effect of an earthquake on the dam and determine if the deformation criteria is met."

Table 3: Target Levels for Earthquake Hazards, Standards-Based Assessments, for Construction, Operation, and Transition Phases (For Initial Consideration and Consultation Between Owner and Regulator) (From CDA 2014)

Dam Classification	**Annual Exceedance Probability – Earthquakes** (note 1)
Low	1/100 AEP
Significant	Between 1/100 and 1/1,000
High	1/2,475 (note 2)
Very High	1/2 Between 1/2,475 (note 2) and 1/10,000 or MCE (note 3)
Extreme	1/10,000 or MCE (note 3)

Notes:

Acronyms: MCE, Maximum Credible Earthquake; AEP, annual exceedance probability

1. Mean values of the estimated range in AEP levels for earthquakes should be used. The earthquake(s) with the AEP as defined above is(are) then input as the contributory earthquake(s) to develop the Earthquake Design Ground Motion (EDGM) parameters as described in Section 6.5 of the *Dam Safety Guidelines* (CDA 2013).
2. This level has been selected for consistency with seismic design levels given in the National Building Code of Canada.
3. MCE has no associated AEP.

ii. Geotechnical

CDA (2014) recommends target levels related to slope stability for static and seismic conditions in Tables 4 and 5.

Table 4: Target Factors of Safety for Slope Stability in Construction, Operation, and Transition Phases - Static Assessment (From CDA 2014)

Loading Condition	**Minimum Factor of Safety**	**Slope**
During or at end of construction	> 1.3 depending on risk assessment during construction	Typically downstream
Long term (steady state seepage, normal reservoir level)	1.5	Downstream
Full or partial rapid drawdown	1.2 to 1.3	Upstream slope where applicable

Table 5: Target Factors of Safety for Slope Stability in Construction, Operation, and Transition Phases - Seismic Assessment (From CDA 2014)

Loading Condition	**Minimum Factor of Safety**
Pseudo-static	1.0
Post-earthquake	1.2

The CDA (2014) notes that "A factor of safety of 1.3 may be acceptable during construction of a dam where the consequences could be minor and measures are taken during construction to manage the risk such as detailed inspection, instrumentation, etc. But, the factor of safety of 1.3 should not simply be adopted because it is "End of construction." A factor of safety of 1.5 has typically been adopted for tailings dams because of the potential consequences of failure. Therefore, when setting the design criteria for the dam, these target levels can be considered, but the risks associated with instability of the dam also need to be considered."

The CDA (2014) also notes the following unique aspects of TSFs with respect to geotechnical design:

- The design, construction, and operation of tailings dams often use the observational method due to the long construction period and opportunities to review actual conditions.
- Loading on a dam shell from an upstream tailings beach needs to be accounted for in stability assessments.
- Liquefaction of tailings upstream of the dam needs to be considered in stability assessments.
- Mine waste is often used in the structural portion of a mining dam and this requires special care with respect to the design of filters and transition zones to protect the seepage control elements of the dam.
- Geochemical processes (often acid rock drainage or metal leaching) can clog filters and drains through precipitate accumulation. While this can also occur with conventional dams, it is more prevalent in mining dams that contain materials with acid rock drainage generation potential. The rate of clogging and the time that the drains are required to operate may greatly exceed those typical of conventional water storage dams. Cementing of soil into a "hard pan" can affect the seepage conditions in a dam.
- Decant structures and/or pipes embedded in embankments in general are potential pathways for seepage. The deterioration of pipes through dams is a well-known cause of several mining dam failures. Development of preferential seepage paths and arching zone(s) are also notable safety hazards. For decant pipes with intermittent discharge, frost action can also create seepage pathways around the pipes. Hence, these structures need to either be avoided or designed and constructed with a high level of care, including redundant protective measures.
- Mining dams are often located near other infrastructure such as open pits and underground workings. The consequences of failure of such mining dams require careful consideration. Also, the potential interaction of the mining operations (i.e. blasting or large waste rock dumps) on the mining dams must be assessed.
- Subsidence of ground beneath a mining dam can occur due to underground workings that may not have been detected prior to the design and construction of the dam.
- Piping can occur into underground workings with caving occurring upward into the tailings.
- Design for thickened tailings discharge facilities.
- Geosynthetics are often considered for mining dams because of limited construction materials, but these must be used judiciously when considering structures that will have to last a long time.
- Design should be flexible to accommodate variability and availability of construction materials throughout the life of a tailings dam.
- Instrumentation monitoring, recording between raises, damage to instrumentation during construction or mine operations.
- The use of impervious membranes for lined ponds that also require measures to prevent wildlife from getting trapped in the ponds and causing damage to the liners.
- Vandalism, particularly recreational vehicles that can cause damage to closed site dams.

iii. State and Other Approaches

Montana and New Mexico provide two examples of prescriptive state requirements for seismic and geotechnical performance standards for TSFs, as do requirements for the province of British Columbia. The requirements for Montana and British Columbia were formulated based in part on the recommendations of the Mount Polley Independent Engineering Review Panel (IERP). The requirements are shown in Table 6.

Table 6: Seismic and Geotechnical Requirements – Montana[1], New Mexico[2], British Columbia[3]

State	Design Criteria	
	Minimum Seismic Event	Factor of Safety – Static
Montana	1-in-10,000-year event, or the maximum credible earthquake, whichever is larger	During Construction, 1.3 Normal Operating, 1.5 Post-Earthquake, 1.2 (loss of containment) Post-Earthquake, 1.0 (no loss of containment)
New Mexico	Dams classified as high hazard potential other than flood control structures shall be designed for the maximum credible earthquake or for a 1% probability of exceedance in 50 years (approximately 5000-year return frequency).	End of Construction, 1.3 Operational Drawdown, 1.5 Rapid Drawdown, 1.3 Steady-state Long-term, 1.5
British Columbia, CAN	minimum seismic design criteria shall be a return period of 1 in 2475 years for dam classification of low-high, ½ between 1/2475 and 1/10,000 or MCE for dam classification of very high to extreme.	End of Construction, 1.5 Long-term, 1.5 Full or Partial Drawdown, 1.5

iv. Hydrology

1. Inflow Design Flood

Surface water estimation related to flood flows (e.g. most extreme event) is presently performed using either deterministic approaches or based on risk analysis. The deterministic, or traditional standards-based approach is based on historic meteorological data and computational approaches to establish precipitation estimates, including Probable Maximum Precipitation (PMP) and Probable Maximum Flood (PMF) estimates, as well as estimates based on various return intervals (5, 10, 25, 100, 500 year). The

[1] Montana's Metal Mine Reclamation Act (MMRA) amended 2015. §82-4. Section 5. http://leg.mt.gov/bills/mca_toc/82_4_3.htm
[2] New Mexico Office of State Engineer, Part 12 Dam Design, Construction & Dam Safety (2010). §19.25.12.11 http://www.ose.state.nm.us/DS/Regs/19-25-12-NMAC-2010%202016-05-27.pdf
[3] British Columbia Health, Safety and Reclamation Code, Part 10 Revisions (2016). §10.1.8 http://www2.gov.bc.ca/gov/content/industry/mineral-exploration-mining/health-safety/health-safety-and-reclamation-code-for-mines-in-british-columbia

risk analysis approach uses either a quantitative or qualitative or risk assessment technique. Quantitative risk assessment mathematically calculates estimates of risk, but it requires data upon which to estimate risk, is highly complex, and the current state-of-knowledge is limited (CDA 2013). Qualitative risk assessment characterizes uncertainty in more general terms and uses methods for indexing, scoring and ranking risk factors. The most common example used in TSF design is the MAA process. Robertson (2012) and others have long advocated for the use of various alternatives analysis assessment methods which has resulted in the development and widespread adoption of the Multiple Accounts Analysis (MAA) process for TSF sites, and more recently TSF technology and design selection. Today the process is both well-defined and in common use, particularly in Canada where it is required. Environment Canada's *Guidelines for the Assessment of Alternatives for Mine Waste Disposal*[4] summarizes the MAA process as it would apply to TSF selection. The deterministic approach is the most accepted approach for dam design and assessment, however the use of a risk-based approach is becoming more widely used and is recommended where appropriate as noted in the following section.

The Inflow Design Flood (IDF) is the most severe inflow flood for which a TSF or associated facilities are designed and should be considered applicable to the construction, operation, and transition phases (CDA 2014). In selecting an IDF the risks of hydrologic failure of a TSF should be balanced with the potential downstream consequences. Current practice is described in FEMA's Federal Guidelines for Selecting and Accommodating Inflow Design Floods for Dams which was first published in 1986 and most recently updated in 2013. FEMA (2013) notes that no single approach to selection of an IDF is adequate given the unique situations of each site, and therefore recommends the following approaches:

Prescriptive Approach – In this initial phase, a planned dam is designed or an existing dam is evaluated for a prescribed standard based on the hazard potential classification of the dam. This approach is intended to be conservative to allow for efficiency of resource utilization while providing reasonable assurance of the safety of the public. It is not intended to assure that there is an economical marginal benefit from designing for a conservative IDF.

Site-specific PMP Studies (Refinement of the Prescriptive Approach) – The prescriptive approach relies upon determination of a PMF for high hazard dams which requires assessment of the PMP. The most common sources of the PMP information are the regional HMRs published by the NWS. These reports provide generalized rainfall values that are not basin-specific and tend to represent the largest PMP values across broad regions. Most of these reports have not been updated to reflect current state-of-the-art knowledge and technology. A site-specific study of the PMP/PMF using current techniques can result in a more appropriate estimate of the PMF for consideration as the IDF.

Incremental Consequence Analysis – The volume of many reservoirs may be small in comparison to the volume of the hydrologic events to which they may be subjected. In these cases, the IDF can be established by identifying the flood for which the downstream consequences with and without failure are not significantly different.

Risk-informed Decision Making – This method allows a dam owner or regulator to consider the risk associated with hydrologic performance of dams relative to other dam safety risks at the same dam, across a portfolio of dams, or in comparison to societal risks in general. In this method, the IDF is selected as the design flood which assures that a given level of "tolerable risk" is not exceeded. The strengths of this method include providing dam owners and regulators the ability to assess the marginal

[4] https://ec.gc.ca/pollution/default.asp?lang=En&n=125349F7-1&offset=2&toc=show

value of increasing levels of flood protection, balancing capital investment in risk reduction across a number of different failure modes, and prioritizing risk reduction actions across a portfolio of dams.

2. Prescriptive Approach

FEMA (2013a) recommends prescriptive IDF criteria corresponding to the hazard potential classification described in Table 1 in Table 7.

FMEA (2013a) notes that, "When selecting an IDF based on either probabilistic or PMP concepts, it should be recognized that these values are derived using limited information. Accordingly, such estimates are not fixed but inherently have a margin of uncertainty. As science evolves and additional data is collected, precipitation estimates and frequency of floods can change. The occurrence of events greater in magnitude than had been previously recorded in a specific location or region can cause such estimates to increase. This uncertainty in the foundation data is combined with the uncertainty inherent in modeling a postulated, rather than actual, event. Practitioners should be aware of this and, when possible, select IDFs that consider this reality. In all cases, an appropriate IDF selection should be performed, or directed and reviewed by a registered professional engineer experienced in hydrology and hydraulics."

Table 7: IDF Requirements for Dams Using a Prescriptive Approach (FEMA 2013a)

Hazard Potential Classification	Definition of Hazard Potential Classification	Inflow Design Flood
High	Probable loss of life due to dam failure or mis-operation (economic loss, environmental damage, or disruption of lifeline facilities may also be probable, but are not necessary for this classification)	PMF[1]
Significant	No probable loss of human life but can cause economic loss, environmental damage, or disruption of lifeline facilities due to dam failure or mis-operation	0.1% Annual Chance Exceedance Flood (1,000-year Flood)[2]
Low	No probable loss of human life and low economic and/or environmental losses due to dam failure or mis-operation	1% Annual Chance Exceedance Flood (100-year Flood) or a smaller flood justified by rationale

(1) Incremental consequence analysis or risk-informed decision making may be used to evaluate the potential for selecting an IDF lower than the prescribed standard. An IDF less than the 0.2% annual chance exceedance flood (500-year flood) is not recommended.

(2) Incremental consequence analysis or risk-informed decision-making studies may be used to evaluate the potential for selecting an IDF lower than the prescribed standard. An IDF less than the 1% annual chance exceedance flood (100-year flood) is not recommended.

Based on CDA's use of five hazard potential classifications as shown in Table 2, the CDA (2014) recommends target levels for the inflow design of TSFs described in Table 8. CDA also suggests that "…the dam classification and the associated target levels shown…" in Table 2 "…should be considered when developing the design criteria. In addition, the mining dam owner will want to factor in other risks and may choose to adopt more stringent design criteria than suggested by the classification alone."

Table 8: Target Levels for Flood Hazards, Standards-Based Assessments, for Construction, Operation, and Transition Phases (CDA 2014)

(For Initial Consideration and Consultation Between Owner and Regulator)

Dam Classification	**Annual Exceedance Probability – Floods** (note 1)
Low	1/100
Significant	Between 1/100 and 1/1,000 (note 2)
High	1/3 Between 1/1,000 and PMF (note 3)
Very High	2/3 Between 1/1,000 and PMF (note 3)
Extreme	PMF (note 3)

Notes:

Acronyms: PMF, Probable Maximum Flood; AEP, annual exceedance probability

1. Simple extrapolation of flood statistics beyond 10^{-3} AEP is not acceptable.
2. Selected on basis of incremental flood analysis, exposure, and consequences of failure.
3. PMF has no associated AEP.

3. Mount Polley Independent Expert Review Panel Findings

In August 2014, the Mount Polley Mine tailings facility breached, resulting in a catastrophic release of tailings that was previously considered unlikely due to the circumstances of it occurring in what is touted as one of the more progressively regulated jurisdictions (British Columbia - BC) at a mine operated by a rising and supposedly highly capable Canadian based mining company (Imperial Metals) and designed and inspected by leading engineering firms (Knight Piésold and AMEC). The event was considered by the industry and associated engineering consultants as a highly significant event. The need for conservative and proactive measures for the design, operation and closure of tailings facilities has since been further reinforced by the even more catastrophic failure that occurred at the Samarco tailings facility in Brazil in November 2015.

The Mount Polley Independent Expert Review Panel (IERP), consisting of three leading experts in the geotechnical stability of mine tailings facilities, was convened by the BC Government to address the minimization and elimination of the risk of similar failures from tailings facilities. The Panel Report was issued in January 2015 and included recommendations that can be grouped into the following seven areas:

1. Implement Best Available Practices (BAP) and Best Available Technologies (BAT) using a phased approach,
2. Improve corporate governance,
3. Expand corporate design commitments,
4. Enhance validation of safety and regulation of all phases of a TSF,
5. Strengthen current regulatory operations,
6. Improve professional practice, and
7. Improve dam safety guidelines

Table 9 summarizes the Panel recommendations and the British Columbia regulatory revisions. For comparison purposes, Table 1 also includes the revisions made to Montana's Metal Mine Reclamation Act (MMRA) in 2015 intended to address the Panel recommendations, and the existing Minnesota regulations.

a. Implement Best Available Practices (BAP) and Best Available Technologies (BAT) using a phased approach

The Panel recommended using Best Available Practices (BAP) to address existing TSFs, and recommended using Best Available Technology (BAT). They further recommended applying BAT principles to closure of active impoundments to eliminate risk. The Panel identified the three principles of BAT as: no surface water; unsaturated conditions, and; achieve dilatant conditions by compaction. The Panel further identified backfilling of mined out pits or underground workings as being the most direct method, but otherwise identified "filtered tailings" technology as the primary BAT. In doing so, the Panel suggested that "There are no overriding technical impediments to more widespread adoption of filtered tailings technology" and "While economic factors cannot be neglected, neither can they continue to pre-empt best technology."

The BC Revisions define BAT as "the site-specific combination of technologies and techniques that most effectively reduce the physical, geochemical, ecological and social risks associated with tailings storage during all stages of operation and closure." The BC Revisions incorporate a "combination of technologies" to "reduce" risk during all stages of the TSF life-cycle. The BC revisions do not include or identify the BAT principles identified by the Panel, or filtered tailings as the prime BAT with cost as a secondary factor. The BC Revisions are not consistent with the Panel recommendations. They do not provide the underlying BAT principles or identify BAT technology to "prevent" or achieve zero risk of TSF failures, but instead the approach uses site specific technologies and techniques to "reduce" the risk of TSF failures.

It is important to note that subsequent to the Panel report, BC regulators had engaged in additional discussions with Dirk van Zyl, one of the three Panel members, whom has issued a letter suggesting he favors the approach being taken by BC regulators consistent with industry recommendations. In response, Steve Vick, another Panel member, has provided comments suggesting that the Panel recommendations were to achieve zero risk by the use of primary BAT and that any compromise will result in further avoidable TSF failures. Those communications are attached as Appendix A to these comments in the interest of ensuring that the views of the IERP and Dr. van Zyl are available for consideration by the public and the regulators that may otherwise depend on them to be representative of the IERPs views.[5]

The MT MMRA Revision requires "an evaluation indicating that the proposed tailings storage facility will be designed, operated, monitored, and closed using the most applicable, appropriate, and current technologies and techniques practicable given site-specific conditions and concerns" and defines "practicable" as "available and capable of being implemented after taking into consideration cost, existing technology, and logistics in light of overall project purposes." The MMRA revisions do not include or identify the BAT principles identified by the Panel, or filtered tailings as the prime BAT. The MT MMRA Revisions do not appear to be consistent with the Panel recommendations in that they do not provide the underlying BAT principles or identify BAT technology to "prevent" or achieve zero risk of

[5] Communications can be provided.

TSF failures, but instead present the approach favored by industry which is to use site specific technologies and techniques to “reduce” the risk of TSF failures.

Minnesota’s regulations, typical to most if not all other U.S. State regulations with the exception of Montana’s recent revisions, do not address either BAP or BAT or the need to evaluate them to either reduce or prevent risk of catastrophic failures.

b. Improve corporate governance

The Panel recommended that corporations operating TSFs should be required to be a member of the Mining Association of Canada (MAC) or be obliged to commit to an equivalent program for tailings management, including the audit function.

The MAC, in response to issues presented by TSFs worldwide owned by Canadian based corporations, developed guidelines for tailings management that are considered worldwide as best management practice (BMP). This includes: A Guide to the Management of Tailings Facilities; Developing an Operation, Maintenance and Surveillance Manual for Tailings and Water Management Facilities, and; A Guide to the Audit and Assessment of Tailings Facility Management.[6] The Tailings Management Protocol was updated in 2015 and an additional update is expected in 2016, in part implementing Panel recommendations for corporate governance.

The BC Revisions fall short of the Panel recommendations in that while they require the mine manager to “consider” the HSRC Guidance Document, it does not require they be a member of MAC or be obliged to commit to an equivalent program.

The MT MMRA Revisions require a description of proposed risk management measures. They fall far short of the Panel recommendation and require no obligation to a program equivalent to those required of MAC members. There are no equivalent U.S. based industry or professional groups that have developed equivalent tailings management guidance or that similarly oblige their members to commit to an equivalent program.

The Minnesota regulations address “measures to reduce risk” however typical to most if not all other U.S. State regulations including Montana’s recent revisions, do not address or provide stringent and current requirements for tailings management similar to those contained in MAC guidance and member obligations.

c. Expand corporate design commitments

The Panel recommended that new TSFs “should be based on a bankable feasibility study and consider all technical, environmental, social and economic aspects of the project in sufficient detail to support an investment decision” and should contain a failure modes and effects analysis, cost/benefit analysis of BAT tailings and closure options with the caveat the cost/benefit should not super-cede safety considerations, and detailed and declared Quantitative Performance Objectives (QPOs).

The BC Revisions are for the most part consistent with the Panel’s recommendations. They require risk assessment and management, an alternatives assessment of best available technology, and QPO’s. The

[6] http://mining.ca/towards-sustainable-mining/protocols-frameworks/tailings-management

primary difference with the Panel recommendations is that the alternatives assessment does not specifically require that safety considerations must not super-cede cost/benefit considerations.

The MT MMRA revisions are for the most part consistent with the Panel's recommendations. They require a failure modes effects analysis, QPO's and risk management measures. The primary difference with the Panel recommendations is that the MMRA revisions do not specifically require that safety considerations must not super-cede cost/benefit considerations.

The Minnesota regulations do not require a failure modes effects analysis, BAT cost-benefit analysis, or QPOs, similar to most if not all other U.S. State regulations with the exception of Montana's recent revisions. Typical to water reservoirs, they do require analysis of a dam break flood as a result of dam failure.

d. Enhance validation of safety and regulation of all phases of a TSF

The Panel recommended that Independent Tailings Review Boards (ITRBs) be utilized together with QPOs to improve safety and regulation of all phases of TSFs.

The BC Revisions require an ITRB and the submission of the terms of reference and qualifications for board members for approval. The BC revisions also requires a report of the activities of the ITRB, confirmation and incorporation of ITRB recommendations, and assurance that the report is a true and accurate representation of their reviews. The BC Revisions do not address the use of QPOs to improve regulator evaluation of TSFs. The BC Revisions do not address the requirements for ITRB members to be independent of the proponent.

The MT MMRA Revisions require an ITRB and the submission and approval of board members. The MT MMRA Revisions do not require the submission and approval of the terms of reference for the ITRB. The MT MMRA revisions require that "The panel shall review the design document, underlying analysis, and assumptions for consistency with this part. The panel shall assess the practicable application of current technology in the proposed design. (9) The panel shall submit its review and any recommended modifications to the operator or permit applicant and the department. The panel's determination is conclusive. The report must be signed by each panel member." The MT MMRA Revisions do not address the use of QPOs to improve regulator evaluation of TSFs.

The Minnesota regulations do not address either ITRBs or use of QPOs in regulator evaluation.

e. Strengthen current regulatory operations

The Panel recommended that inspections be performed at all existing TSFs to ascertain whether they may be a risk and require appropriate actions due to specific failure modes: filter adequacy; water balance adequacy; undrained shear failure of silt and clay foundations.

The BC government required inspections to be completed and submitted by June 30, 2015 to comply with the Panel's recommendations.

The Montana MMRA Revisions do not require inspections for this purpose although the requirement for both annual EOR and independent audit inspections can be construed as requiring these failure modes be addressed. The Minnesota regulations do not require inspections specific to these failure modes.

f. Improve professional practice

The Panel encouraged the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC) to develop guidelines that would lead to improved site characterization for tailings dams with respect to the geological, geomorphological, hydrogeological and possibly seismotectonic characteristics.

The APEGBC developed and published Site Characterization for Dam Foundations in BC in August 2016, including a section on seismotectonic conditions.

There are no equivalent U.S. or Minnesota based industry or professional groups that have developed equivalent site characterization guidance for either dams or TSFs.

g. Improve dam safety guidelines

The Panel, recognizing limitations of current Canadian Dam Association guidelines, recommended that dam safety guidance be developed specific to the conditions encountered with TSFs in British Columbia and incorporated as a statutory requirement. The Montana and BC dam safety regulations include prescriptive and specific design criteria requirements for TSFs. The Minnesota regulations rely on the consideration of alternative sites and the determination of dam safety by "current, prudent engineering practice."

4. Northmet TSF Engineering Analysis and Reviews

The Northmet Dam Safety Permit Application (NDSPA) prepared by Barr and dated May 2017 addresses the classification of the TSF, dam break analysis, and performance standards.

a. TSF Classification

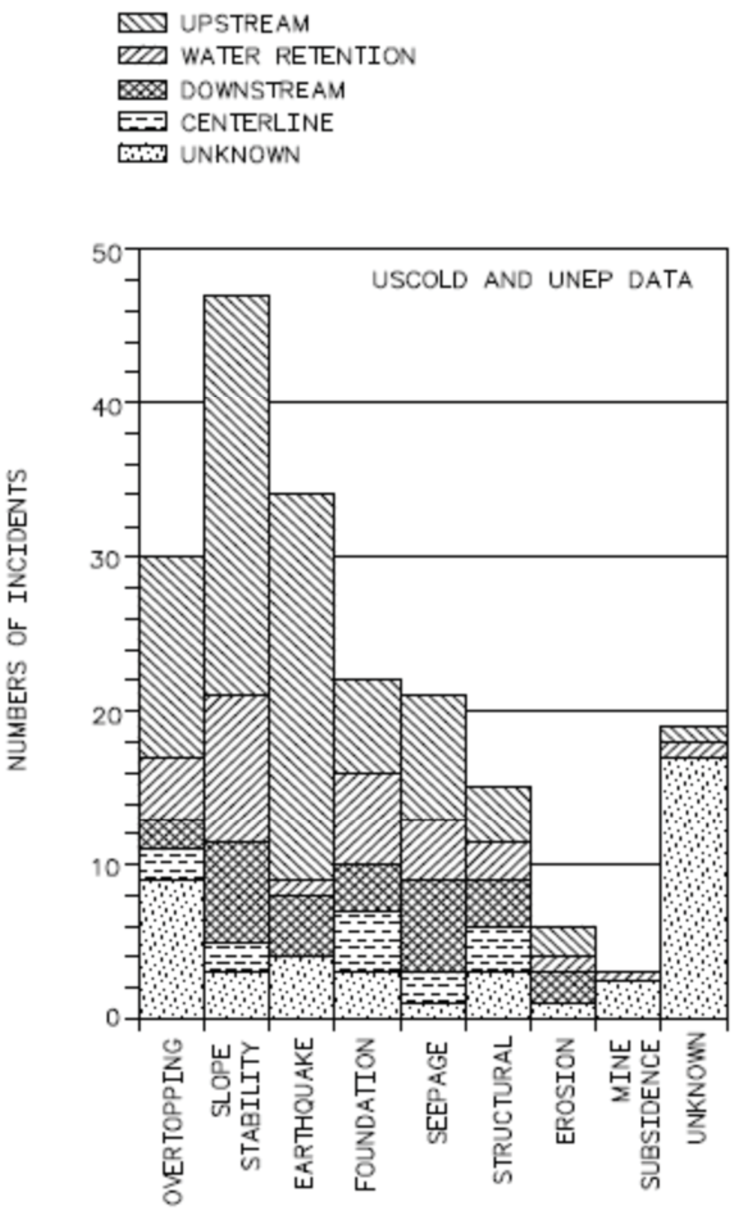
According to the NDSPA (p. 9), "The FTB dams have been designed to achieve necessary factors of safety, so a dam break is unlikely." It goes on to say that "A dam break analysis was completed to understand the potential extent of flood inundation between the FTB and the Embarrass River in the unlikely event of a failure at the dam." Based on these premises, the report concludes "The FTB dams can be categorized as Class I or Class II dams."

We find this approach, particularly as it speaks for the assumptions used by the design engineer, to be highly concerning as it suggests the design engineer has not taken into account all failure modes that can cause a catastrophic dam breach, many of which are independent of design factors of safety. Similarly, the recent Mount Polley and Samarco (Brazil) TSF catastrophic failures were considered to be "unlikely," if not impossible, until they occurred. In our experience and professional judgment, a more accurate portrayal would be to consider all potential failure modes and identify TSF failure as "possible" and would likely lead to highly significant safety, environmental and economic consequences, and for that reason the TSF should be classified as Class I.

Potential failure modes related to TSFs can include both structural (geotechnical) failure modes (sliding, overtopping, internal erosion, etc.) and other modes that are non-structural in nature and are related to operations and/or environmental protection. UNEP's, 2001, Tailings Dams Risk of Dangerous

Occurrences, Bulletin 121 is frequently cited as a reference with respect to TSF failure modes. They identified (see Figure 1) slope stability as being the primary failure mode of TSFs, followed by earthquake (seismic) and overtopping. They also identify foundation failures, seepage, structural, erosion and mine subsidence as failure modes.

Figure 1: TSF Failure Incidents by Failure Mode and Design Type (UNEP 2001)



LePoudre (XXXX) provides a more comprehensive list of Failure Modes and Contributing Factors for TSFs, noting they are partial, which includes the following:

Physical / Structural Failure Modes

Slope Failure
- Raising of Dyke
- Placement of tailings
- Undercutting
- Poor construction materials
- Over-steepening
- Direct loading
- Seismic

Foundation Failure
- Undrained loading
- Sensitivity clays
- Seepage forces
- Strength loss
- Weak layers

Surface Erosion
- Overtopping
- Runoff
- Excessive inflow
- Insufficient outflow conveyance
- Inadequate rip-rap
- Landslide into impoundment

Internal Erosion
- Piping
- Lack of adequate filter
- Zoned dams
- Sinkholes
- Unprotected conduits
- Joints/seepage in foundation/abutments

Contributing Factors for Mode of Failure

Design /Construction
- Dam type
- Materials
- Hydrology/hydrogeology
- Construction
- Outlet Structures
- Freeboard
- Foundation/ Abutments
- Chemical processes
- Biological Processes

Operation /Maintenance
- Rate of deposition
- Water Management
- Inspection / Monitoring
- Maintenance

- Chemical processes
- Biological processes

External Factors

- Human Activity
- Climate/weather
- Seismic activity
- Earth movement
- Unforeseen

The CDA (2014) identifies the following Other Failure Modes for TSFs:

- Unplanned release of contaminated water via an emergency overflow spillway.
- Release of excessive contaminated seepage down gradient of the dam.
- Contamination of groundwater.
- Excessive seepage causing the loss of water cover required over a tailings deposit to inhibit sulphide oxidation.
- Excessive erosion by wind resulting in dust releases (in the case of mining dams constructed of tailings).

It appears that the design engineer has not conducted a formal Failure Modes Effects Analysis (FMEA) to assist in the TSF design and identification of other key aspects such as operational and closure requirements to ensure TSF safety. FMEA is a form of risk analysis and risk management that has become widely used for both typical water retaining dams and for TSFs. Figure 2 shows how risk analysis, risk assessment, and risk management relate to each other.

The U.S. ACOE (2014) notes that it has moved from a solely standards-based approach for its dam safety program to a dam safety risk management approach for dams within its portfolio. They provide an extensive example of application of the approach in their policies and procedures document.

Robertson (2012) describes risk assessment and management for TSFs as "The Balance Between Experience, Judgement and Science" and suggests that an effective risk management program must include the following elements:

- Identification of all failure modes and the factors that contribute to the likelihood of occurrence of that failure mode.
- A realistic assessment of the probability and consequences – yielding a risk rating.
- A program that mitigates the risks to reduce either probability (likelihood) or consequences to tolerable levels.
- An Action Plan and Management that implements the Action Plan.

He goes on to suggest the following stages in the performance of the risk management process.

STAGE 1 – PERFORM A FMEA

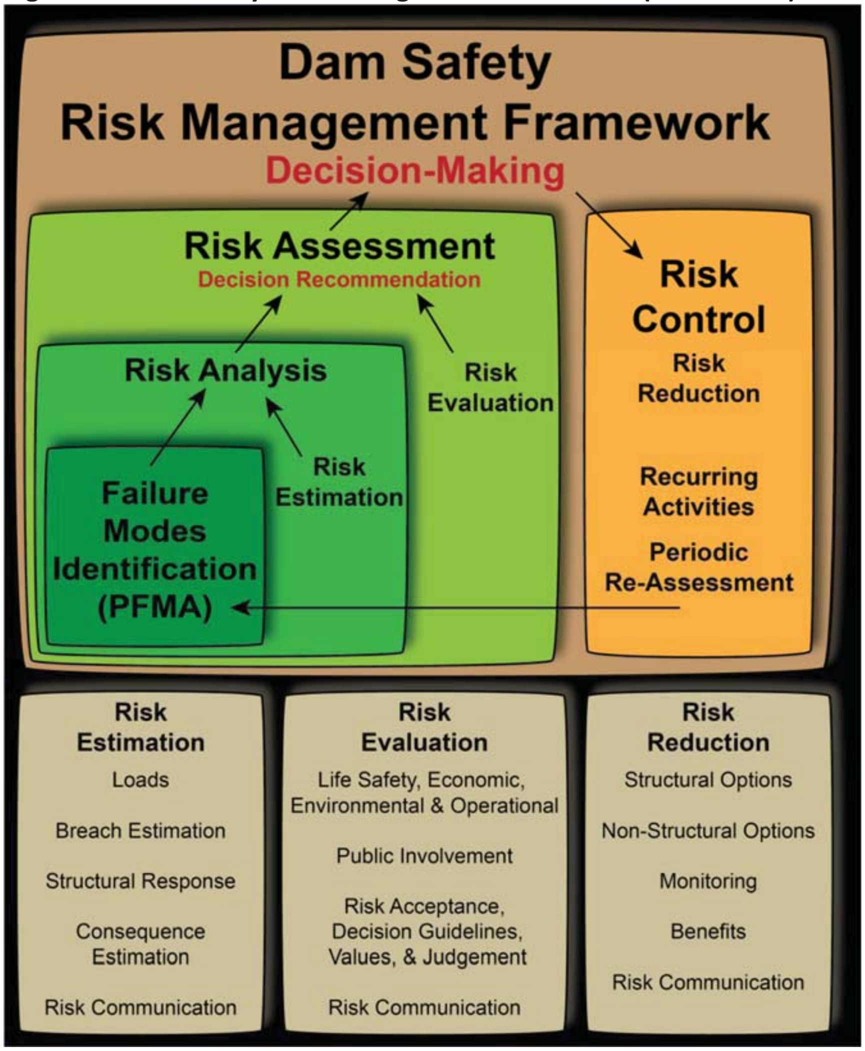
- Identify all significant failure modes
- For each: assess likelihood and consequences
- Determine Risk - see the following matrix
- Identify tolerable risk levels and failure modes with excessive risk – prioritize need for mitigation

- Identify mitigation measures that will reduce risks to tolerable limits

STAGE 2 – PERFORM RISK MITIGATION
- Develop an Action Plan, including schedule
- Implement Action Plan – Mitigate

STAGE 3 – PERFORM PERIODIC FMEA REASSESSMENTS
- Determine if Risks remain tolerable, and implement additional mitigation as required

Figure 2: Dam Safety Risk Management Framework (FEMA 2015)



b. Dam Break Analysis

According to the NDSPA (p. 9), "A dam break analysis was completed to understand the potential extent of flood inundation between the FTB and the Embarrass River in the unlikely event of a failure at the dam." The report refers to the analysis in Appendix H but does not provide further information in support of the dam classification.

Appendix H of Attachment A of the NDSPA Flotation Tailings Basin Dam Break Analysis identifies the methodology. According to Appendix H (p. 4), "The dam break analysis focused on the north side of the FTB, because this is the section of the dam where a break would result in the shortest warning time for potentially affected downstream properties. A breach was not considered to the east or south of the FTB because a large portion of the perimeter ties into natural ground and/or no homes are within the respective downstream flow path." The identification of "downstream properties" where a break was considered and "homes" where a breach was not considered, when it can be assumed that the downstream properties included residential homes where a breach was considered, appears to underplay the consequences of loss of life in the analysis. Appendix H (p. 5) goes on to identify piping as the selected cause of the dam break, suggesting that "Failure resulting from overtopping was not considered because the dam is designed not to be overtopped even with the volume of the 72-hour PMP event." The Appendix (p. 6), while noting "Time to failure is a sensitive parameter for dam failure analysis," chose to use a time to failure of three hours, suggesting that FERC's recommendation of less than one hour seemed unrealistic based on the size of the dam and final configuration." The Appendix (p. 8) concluded that "a dam break could increase flood elevations approximately 15 feet at the upstream end of Trimble Creek (near the FTB) and approximately 9 feet at the downstream end of Trimble Creek (at the Embarrass River)" and "that there are 34 properties along Trimble Creek or the breakout paths that could potentially be affected by a FTB dam break." The analysis does not identify the actual number of homes on those properties or the number of lives that could be lost due to a TSF breach.

We appreciate the elements that were included in the analysis that were conservative as noted in the report (p. 7) and agree that for the type of failure analyzed "The actual extent of inundation and risk to residents and infrastructure can reasonably be anticipated to be lower than suggested by this analysis." However, the report (p. 7) also suggests that additional analysis is not warranted "given the objective of this dam break analysis, which is to serve as an aid in development of the facility Emergency Action Plan." An additional objective of the break analysis should be to assist in determination of the dam classification, as well as consideration of the consequence of failure in a FMEA as suggested in the preceding section.

For the dam break analysis to be truly conservative, current industry guidance and experience suggests additional consideration should be given to the analysis. The CDA (2013) recommends the evaluation address initial hydrologic conditions for the following:

- Sunny day failure – A sudden failure that occur during normal operations such as may be caused by internal erosion, piping, earthquakes, mis-operation leading to overtopping, or another event.
- Flood induced failure – A TSF failure resulting from a natural flood of a magnitude that is greater than what the dam can safely pass.

The incremental environmental consequences are often worse for a sunny day failure than a flood induced failure because of the large amount of process water and solids that are contained by TSFs (CDA 2014). The CDA (2013) recommends that simple and conservative procedures be applied to obtain a first approximation and that if necessary more detailed analysis should be conducted. The CDA (2013) suggests that TSF failure consequences should be evaluated for the following:

- Loss of Life

- Economic Losses
- Environmental Losses
- Cultural Losses
- Incremental and Total Consequences

As noted by Morgenstern et al (2015) the Mount Polley TSF failure was a blue sky or sunny day failure, and it occurred in a matter of minutes if not seconds. The failure was also compounded by process water levels that exceeded freeboard requirements and contributed significantly to the extent of the failure. And while it did not result in the loss of human life, that was only by coincidence. The Samarco TSF failure, which was also sudden, did result in significant loss of human life.

The Northmet TSFs breach analysis should include consideration of a sunny day failure that occurs within a short amount of time (minutes). It should further identify the number of homes and corresponding population and estimate the potential loss of life in the event of a worst-case TSF failure. This information should be used not only to inform the ERP, but also to inform the TSF design process, including a FMEA.

c. Permit Standards

According to the NDSPA (p. 10) the permit standards were previously submitted to DNR and a "Large Table" is referenced. The Large Table shows that ARM 6115.0410 (8)(D) is addressed in Geotechnical Data Package Volume 1 (Appendix B) Section 7.3, pages 102-115 and 6115.0410 (8)(E) is addressed in Flotation Tailings Management Plan (Appendix A) Section 3.3, page 20 and Attachment H.

i. Geotechnical

Geotechnical Data Package Volume 1 (Appendix B) Section 7.3 does not identify the source for the geotechnical design standards that were used for the Flotation TSF. Appendix B (p. 10) identifies the seismic design event as using a 2,475-year return period. Also, according to Appendix B (p. 93) "The proposed FTB dams have been configured to have safety factors equal to or greater than 1.5 for drained (ESSA) conditions, equal to or greater than 1.3 for undrained ($USSA_{yield}$) conditions, and equal to or greater than 1.1 for liquefied ($USSA_{liq}$) conditions.

The seismic design event using a 2,475-year return period does not reflect current best practice. As previously noted (see Table 6), Montana's MMRA requires a minimum seismic event for a 1-in-10,000-year event, or the maximum credible earthquake, whichever is larger and British Columbia requires that the minimum seismic design criteria shall be a return period of 1 in 2475 years for dam classification of low-high, ½ between 1/2475 and 1/10,000 or MCE for dam classification of very high to extreme. Given the significant potential for loss of human life without additional information we would suggest the TSF classification and corresponding seismic design event should be considered highly conservatively and consistent with the practice of those jurisdictions that have considered the recommendations of the Mount Polley IERP.

Notwithstanding the inadequacy of the seismic design criteria, the analysis does appear to consider Target FOSs equivalent to those recommended by the CDA and regulatory requirements such as for Montana and British Columbia. Review of the stability modeling results suggests that with the exception of the operations modeled FOS of 1.10, application of the recommended MCE is unlikely to result in any of the other FOS being below the Target FOSs.

ii. Hydrology

According to the Flotation Tailings Management Plan (Appendix A) (p. 7, 13, 20) the design incorporated the Probable Maximum Precipitation (PMP), the stability analysis considered the effects of the PMP event, and the dam break analysis was based on a 72-hour PMP event, which is consistent with current best practice.

5. EOR Review Team

EOR, Minnesota DNR's contractor, assembled a Review Team to supplement the review process, which is described in a memo titled PolyMet Dam Safety Permit Application Review and dated May 15, 2017. According to the memo (p. 1) "The review approach focused on key elements similar to tailings basin review panels required by law in Montana and other western states." The memo identifies Dirk van Zyl and Steve Gale, both professional engineers (PEs), as members of the review team but does not identify other participants either from EOR or DNR. The Review Team's scope consisted of review of documents, site visit and discussion with Polymet and TSF designers, Review meetings with DNR, and presentation of a Draft Report and preparation of a Final Report.

The EOR Review Team comments included the following:

- The EOR Team concluded that the permit application lacks the detail and description of contingencies for the Observational Method to be effective. If monitoring data indicate a potentially unsafe condition during construction, then the alternate construction methods and designs (contingencies) must be already in place so that they can be implemented immediately.

- 0The former LTV tailings basin was constructed over layers of peat in some areas. Layers of slimes (very fine-grained taconite tailings) were also included in the construction of the tailings basin dam. Both peat layers and slimes layers have very low shear strength, which could potentially contribute to a dam failure. The tailings basin can be designed to safely mitigate for these conditions, but the areas with peat and slimes must be well-defined and tested. The EOR Team commented that additional data should be gathered on the peat layers and slime layers, and that the design may need to be modified in the future in accordance with the Observational Method.

- As currently designed, a pond of water will be maintained on top of the tailings basin in perpetuity. The EOR Review Team recommended that a water pocket distance of less than 625 feet (or in direct contact with the tailings dam) be analyzed as an event/condition of the Observational Method approach.

- To minimize water seepage from the tailings basin, bentonite will be added to the soils at the top of the basin during the closure and reclamation process. The permit application only lists alternatives for placing the bentonite that will be pilot tested and field tested later. The EOR Review Team commented on specific elements that should be included in the field testing that would impact the permeability of the bentonite amended tailings. Once the preferred bentonite application method is selected, the EOR Review Team recommended developing material and installation specifications and a detailed protocol for both a laboratory and a field pilot study.

- EOR Review Team commented that some of the geotechnical test results (i.e. low coarse tailings friction angles) were excluded from the statistical analyses. Because of their importance in the overall stability of the basin, the EOR Review Team recommended that coarse tailings friction angles be considered as a variable condition in the Observational Method process. This would also provide a consistent and proper procedure for future analyses.

- Wet closure has ongoing costs like; maintaining water levels to prevent flooding and drying out, erosion repair, treatment of discharged water and on-going monitoring. Dry closure (no water ponding) requires a greater initial investment, but has much lower ongoing maintenance costs and less long-term environmental risk. The EOR Review Team did not proposed dry closure as a permit requirement at this time. The EOR Review Team recommended that if the wet closure is permitted, the DNR should require PolyMet to continually review the current state-of-the-practice for dry closure techniques prior to starting any tailings basin closure activities.

- HydroMet Residue Facility - The soft ground beneath the proposed residue facility consists of up to 30 feet of slimes, peat and tailings concentrate. This will not be an adequate foundation for the 80-foot-high basin. Three potential remediation alternatives have been considered:
  - Pre-loading the existing material with 50 feet of rock and soil to compress and consolidate the underlying material. This is the method currently proposed by PolyMet.
  - Installing wick drains that will allow water to flow out of the existing material, thereby increasing its shear strength.
  - Removing the existing material and any soft soils before constructing the basin.
  - The basin will have a geomembrane or geosynthetic liner. The liner could deform and fail if the existing underlying material cannot support the material added to the basin.

- The EOR Review Team commented that the proposed pre-load design should be re-evaluated to determine if it will adequately surcharge and compress the existing material.

The EOR Review Team recommended that the comments and issues be addressed pre-permit, post-permit and made a condition of the permit, or addressed pre-construction.

The inclusion of Dirk van Zyl as an EOR review team member is notable. He was one of the three Mount Polley IERP members, but as noted previously in Section 3.a., he has also chosen to distance himself from the findings of the IERP by advocating for a less conservative approach to TSF's than the IERP, particularly with respect to the recommendation for filtered dry stack tailings as BAT for new TSFs. In this case, he also advocates for a wet closure approach which also contradicts the recommendations of the Mount Polley IERP which advocated for dry closure - for existing TSFs the Panel identified the three principles of BAT as: no surface water; unsaturated conditions, and; achieve dilatant conditions by compaction. While we respect van Zyl's professional expertise and opinions and work with him in numerous forums, we find it necessary to note his difference of opinion from that of the IERP. It is our opinion that in order to ensure a balanced review and for it to be considered valid by the public in particular, a truly independent review process must be undertaken that includes the participation of additional TSF expertise including nominees from public stakeholders.

6. Summary of Conclusions and Recommendations

- Minnesota's existing dam safety statutes, RSM 6115.0410, are intended for water storage dams and do not specifically address tailings storage facilities. In addition, the requirements rely on "current, prudent engineering practice" and "prudent, current environmental practice" rather than on current accepted industry engineering performance standards as is common to nearly all other regulatory jurisdictions in the U.S. and internationally. We would encourage consideration by all parties as to the critical need to revise or modify Minnesota's approach to dam safety and to incorporate statutes, rules and guidance specifically for TSFs that are consistent if not better than those recently enacted and/or developed by Montana and British Columbia.

- Performance standards for TSFs should be specifically required and should include a hazard classification system based on TSFs similar to that recommended by the CDA (Table 2), seismic criteria requiring 1/10,000 year or Maximum Credible Earthquake (MCE), geotechnical minimum Factors of Safety (FOS) and Inflow Design Flood (IDF) consistent with CDA and other guidance.

- If Minnesota DNR's intention is to consider the actual recommendations of the Mount Polley IERP, then at a minimum additional consideration must be given to the IERPs recommendations for BAT. Our recommendation would be for a Multiple Accounts Analysis (MAA) (see Section 2.b.iv.1.) to evaluate BAT for both operation and closure of the proposed TSF and alternative approaches to the TSF including filtered dry stack tailings and closure of the TSF to achieve dilatant conditions.

- Minnesota should also consider the IERPs recommendations and undertake to:
  - Require TSF operators to commit to an equivalent program of tailings management, including the audit function, as are required by member of the Mining Association of Canada;
  - Expand corporate design commitments and require a failure modes effects analysis, BAT cost-benefit analysis, and QPOs;
  - Require the use of formal Independent Review Boards (IRBs) and use QPOs in regulator evaluations of TSF safety;
  - Require that inspections be performed at all existing TSFs to ascertain whether they may be a risk and require appropriate actions due to specific failure modes: filter adequacy; water balance adequacy; undrained shear failure of silt and clay foundations; and,
  - Develop guidelines that would lead to improved site characterization for tailings dams with respect to the geological, geomorphological, hydrogeological and possibly seismotectonic characteristics.

- Northmet should conduct further TSF analysis including a multi-stakeholder FMEA to consider all potential failure modes and their consequences as well as mitigating measures together with the development of an AMP to ensure that means to mitigate potential failures are developed and triggered appropriately.

- Northmet should also appoint a formal IRB for the TSF and involve them in the final design, construction, operation and reclamation through final closure and during post-closure if necessary. The IRB should be robust and include at least three representatives. The IRB process should complement the Engineer of Record's process but at the same time be transparent and involve public representatives in a capacity that would allow them to ensure and report on the outcome of the overall process. We have been involved at several other sites as a technical representative to IRBs or have served on similar panels and would be glad to advise DNR and the NGO community on processes to achieve this recommendation.

- It is our professional opinion that the ultimate determination of acceptability of risk, if a wet tailings approach such as the Northmet TSF is proposed, should lie with the public members whose lives would be at risk in the event of a catastrophic breach. Northmet's analysis shows that the proposed TSF represents significant risk of loss of life in the event, however unlikely, of a catastrophic failure. For that reason, we recommend that the inundation analysis together with the proposed emergency response plan be presented to both the responding regulatory agencies but also to the potentially affected public, through a very intentional process to engage and take their opinions wholly into account, prior to approval of the dam safety permit. The DNR otherwise would be making a decision to put those persons at risk without their input or potentially even their knowledge.

- Our recommendations and opinions should not be seen as exclusive of the possibility that the proposed wet tailings approach, at least during the operational period, might be considered as a reasonable risk by the parties most at risk and otherwise involved. For that reason, we also take the opportunity at this time to recommend in that event, in addition to the involvement of a formal IRB and transparent technical process, that Northmet, DNR and the EOR undertake to:

    - Develop a corporate TSF management strategy similar to that recommended by Mining Association of Canada (MAC) (2011).
    - Develop a TSF operations, maintenance and surveillance (TOMs) manual similar to that recommended by the MAC (2012).
    - Conduct a technology development program together with modifications to the TSF reclamation and closure design to achieve a final stable landform design.

- In the absence of any of our above recommendations and in particular informed public consent, it is our professional opinion that the Mount Polley IERP recommendations for BAT for new tailings (e.g. filtered dry stack tailings) should be required for the Northmet TSF and as a requirement of the dam safety permit. If the decision is to allow for a wet tailings facility during operations, then at the least the dam safety permit should specify closure to meet dilatant landform conditions so as to avoid the threat of a catastrophic failure in perpetuity.

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Table 9 - Comparison of Mt Polley Expert Panel, Montana SB409 Revisions, British Columbia Part 10 HSRC Revisions, Minnesota RSM 6115.0410

Mount Polley Expert Panel Recommendations (2014)	Montana Metal Mine Reclamation Act Section 82-4 Revisions 7(SB 409) (2015)	British Columbia Health, Safety and Reclamation Code Part 10 Revisions (2016)	Minnesota Department of Natural Resources Section RSM 6115.0410 New Dams or Enlargements (2003)
Implement Best Available Technologies (BAT) using a phased approach: • Existing TSFs. Rely on best practices for the remaining active life. • New TSFs. BAT (filtered tailings) should be actively encouraged for new tailings facilities at existing and proposed mines. • At closure. BAT principles (no surface water, unsaturated conditions, achieve dilatant conditions) should be applied to closure of active impoundments so that they are progressively removed from the inventory by attrition.	**82-4-303. Definitions.** (25) "Practicable" means available and capable of being implemented after taking into consideration cost, existing technology, and logistics in light of overall project purposes. **Section 5**. **Tailings Storage facility - design document - fee.** (2) The design document must contain: (e) an evaluation indicating that the proposed tailings storage facility will be designed, operated, monitored, and closed using the most applicable, appropriate, and current technologies and techniques practicable given site-specific conditions and concerns;	**Definitions.** "best available technology" means the site specific combination of technologies and techniques that most effectively reduce the physical, geochemical, ecological and social risks associated with tailings storage during all stages of operation and closure. **Application Requirements. 10.1.3** The application shall include the following unless otherwise authorized by the chief inspector: (f) an alternatives assessment for the proposed tailings storage facilities that assesses best available technology,	*Not addressed.*
Improve corporate governance: Corporations proposing to operate a TSF should be required to be a member of the Mining Association of Canada (MAC) or be obliged to commit to an equivalent program for tailings management, including the audit function.	**Section 5**. **Tailings Storage facility - design document - fee.** (2) The design document must contain: (x) a description of proposed risk management measures for each facility life-cycle stage, including construction, operation, and closure;	**Governance. 10.4.2** (1) The manager of a mine with one or more tailings storage facilities shall: (a) develop and maintain a Tailings Management System that considers the HSRC Guidance Document and includes regular system audits	**Subp. 8. Permit standards.** The applicant may be required to take measures to reduce risks, and the commissioner shall furnish information and recommendations to local governments for present and future land use controls to minimize risks to downstream areas.

Table 9 - Comparison of Mt Polley Expert Panel, Montana SB409 Revisions, British Columbia Part 10 HSRC Revisions, Minnesota RSM 6115.0410 (Continued)

Mount Polley Expert Panel Recommendations (2014)	Montana Metal Mine Reclamation Act Section 82-4 Revisions (SB 409) (2015)	British Columbia Health, Safety and Reclamation Code Part 10 Revisions (2016)	Minnesota Department of Natural Resources Section RSM 6115.0410 New Dams or Enlargements (2003)
Expand corporate design commitments: Future permit applications for a new TSF should be based on a bankable feasibility that would have considered all technical, environmental, social and economic aspects of the project in sufficient detail to support an investment decision, which might have an accuracy of +/- 10-15%. More explicitly it should contain the following: • A detailed evaluation of all potential failure modes and a management scheme for all residual risk • Detailed cost/benefit analyses of BAT tailings and closure options so that economic effects can be understood, recognizing that the results of the cost/benefit analyses should not supersede BAT safety considerations • A detailed declaration of Quantitative Performance Objectives (QPOs).	**Section 5. Tailings Storage facility - design document - fee.** (2) The design document must contain: (n) a dam breach analysis, a failure modes and effects analysis or other appropriate detailed risk assessment, and an observational method plan addressing residual risk; (t) a list of quantitative performance parameters for construction, operation, and closure of the tailings storage facility. The quantitative performance parameters may be expressed as minimums or maximums for embankment crest width, embankment slopes, beach width, operating pool volume, phreatic surface elevation in the embankment and foundation, pore pressures, or other parameters appropriate for the facility and location. (x) a description of proposed risk management measures for each facility life-cycle stage, including construction, operation, and closure;	**Application Requirements. 10.1.3** The application shall include the following unless otherwise authorized by the chief inspector: (d) a mine plan including: (vii) designs and details for tailings storage and a description of proposed quantifiable performance objectives, (e) a program for the environmental protection of land and watercourses during the construction and operational phases of the mining operation, including plans for (i) prediction, identification and management of physical, chemical, and other risks associated with tailings storage facilities and dams, (f) an alternatives assessment for the proposed tailings storage facilities that assesses best available technology,[i] **Governance. 10.4.2** (1) The manager of a mine with one or more tailings storage facilities shall: (d) review annually the tailings storage facility risk assessment to ensure that the quantifiable performance objectives and operating controls are current and manage the facility risks,	**Subp. 6. Final design requirements.** Upon acceptance and agreement by the commissioner of the preliminary report, the applicant shall submit to the commissioner, for approval, a final design report, together with plans and specifications and the initial inspection fee. The final design report shall include, but is not limited to, the following: A. general description of the project, such as its service life, production rates, required storage and area(s); geological considerations such as physiography, topography, geology, seismicity, groundwater conditions, and maps; hydrologic studies such as physical features, climatology, design, storm and design flood characteristics, flood routing, water-material balance, free-board requirements, dam-break flood; geotechnical information, such as rock-soil sampling and logging, geophysical investigations, field and lab testing, instrumentation data; considerations of construction materials and their properties, such as quantities required, borrow and aggregate locations and volumes, field and lab work and investigations, concrete, waste materials generation and placement techniques, investigation of the stored waste materials such as generations, transportation, mechanical/chemical/special testing, disposal practice; B. analytical determinations, such as seepage and underseepage studies, stability, deformation and settlement analysis; analytical and design details of facilities, such as dam, foundation, impoundment, abutments, spillways (for the purpose of these rules, spillway means any facility appurtenant to the dam available to discharge excess water and/or waste from the impoundment) or decant facilities, diversions, outlet works, instrumentation; operational aspects, such as impoundment operating criteria, initial filling criteria, responsibility and coordination, emergency procedures and warning systems: air, water, and solid pollution controls, sedimentation, and erosion controls: operational and postoperational maintenance and abandonment considerations; surveillance and inspection programs; and C. a detailed cost estimate.

Table 9 - Comparison of Mt Polley Expert Panel, Montana SB409 Revisions, British Columbia Part 10 HSRC Revisions, Minnesota RSM 6115.0410 (Continued)

Mount Polley Expert Panel Recommendations (2014)	Montana Metal Mine Reclamation Act Section 82-4 Revisions (SB 409) (2015)	British Columbia Health, Safety and Reclamation Code Part 10 Revisions (2016)	Minnesota Department of Natural Resources Section RSM 6115.0410 New Dams or Enlargements (2003)
Enhance validation of safety and regulation of all phases of a TSF: • Increase utilization of Independent Tailings Review Boards. • Utilize the concept of Quantitative Performance Objectives (QPOs) to improve regulator evaluation of ongoing facilities.	**Section 6. Independent review panel - selection - duties.** (1) An independent review panel shall review the design document required by [section 5]. (2) The operator or permit applicant shall select three independent review engineers to serve on the panel and shall submit those names to the department. The department may reject any proposed panelists. If the department rejects a proposed panelist, the operator or permit applicant shall continue to select independent review engineers as panelists until three panelists are approved by the department. (3) An independent review engineer may not be an employee of: (a) an operator or permit applicant; or (b) the design consultant, the engineer of record, or the constructor. (4) The operator or permit applicant shall contract with panel members, process invoices, and pay costs. (5) A representative of the department and a representative of the operator or permit applicant may participate on the panel, but they are not members of the panel and their participation is nonbinding on the review. (6) The engineer of record is not a member of the panel but shall participate in the panel review. (7) The operator or permit applicant shall provide each panel member with a hard copy and an electronic copy of the design document and other information requested by the panel. (8) The panel shall review the design document, underlying analysis, and assumptions for consistency with this part. The panel shall assess the practicable application of current technology in the proposed design. (9) The panel shall submit its review and any recommended modifications to the operator or permit applicant and the department. The panel's determination is conclusive. The report must be signed by each panel member. (10) The engineer of record shall modify the design document to address the recommendations of the panel and shall certify the completed design document. The operator or permit applicant shall submit the final design document to the department pursuant to [section 5]. (11) For an expansion of a tailings storage facility for which the original design document was approved by the department, the operator shall make a reasonable effort to retain the previous panel members. To replace a panel member, the process in subsection (2) must be followed.	**Governance. 10.4.2** (1) The manager of a mine with one or more tailings storage facilities shall: (c) establish an Independent Tailings Review Board, unless exempted by the chief inspector, (2) The composition of an Independent Tailings Review Board established under subsection (1) (c) shall be commensurate with the complexity of the tailings storage facility in consideration of the HSRC Guidance Document. (d) review annually the tailings storage facility risk assessment to ensure that the quantifiable performance objectives and operating controls are current and manage the facility risks, (3) The manager shall submit the terms of reference for the Independent Tailings Review Board including the qualifications of the board members to the chief inspector for approval. (4) The terms of reference for the Independent Tailings Review Board shall be developed or updated as required in consideration of the review under subsection (1) (d). **Annual Reporting. 10.4.4** The owner, agent or manager shall submit one or more annual reports in a summary form specified by the chief inspector or by the conditions of the permit by March 31 of the following year on the following: (c) a report of the activities of the Independent Tailings Review Board established under section 10.4.2 (1) (c) of this code that describes the following: (i) a summary of the reviews conducted that year, including the number of meetings and attendees; (ii) whether the work reviewed that year meets the Board's expectations of reasonably good practice; (iii) any conditions that compromise tailings storage facility integrity or occurrences of non-compliance with recommendations from the engineer of record; (iv) signed acknowledgement by the members of the Board, confirming that the report is a true and accurate representation of their reviews	*Not addressed.*

Table 9 - Comparison of Mt Polley Expert Panel, Montana SB409 Revisions, British Columbia Part 10 HSRC Revisions, Minnesota RSM 6115.0410 (Continued)

Mount Polley Expert Panel Recommendations (2014)	Montana Metal Mine Reclamation Act Section 82-4 Revisions (SB 409) (2015)	British Columbia Health, Safety and Reclamation Code Part 10 Revisions (2016)	Minnesota Department of Natural Resources Section RSM 6115.0410 New Dams or Enlargements (2003)
Strengthen current regulatory operations: Utilize the recent inspections of TSFs in the province to ascertain whether they may be at risk due to the following potential failure modes and take appropriate actions: • Filter adequacy • Water balance adequacy • Undrained shear failure of silt and clay foundations	*No additional requirements for existing TSFs.*	Inspections required and completed. Final submissions received June 30, 2015. More information available at: http://www2.gov.bc.ca/gov/content/industry/mineralexploration-mining/dam-safety-inspections-2014	*No additional requirements for existing TSFs.*
Improve professional practice: Encourage the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC) to develop guidelines that would lead to improved site characterization for tailings dams with respect to the geological, geomorphological, hydrogeological and possibly seismotectonic characteristics.	*No equivalent action has yet been performed by a professional organization located in the U.S.*	APEGBC developed and published Site Characterization for Dam Foundations in BC, August 2016. https://www.apeg.bc.ca/getmedia/34e1bb3f-cd39-450d-800e-614ac3850bc5/APEG_2016_Site-Characterization-for-Dam-Foundations_WEB_2.pdf.aspx	*No equivalent action has yet been performed by a professional organization located in the U.S.*

Table 9 - Comparison of Mt Polley Expert Panel, Montana SB409 Revisions, British Columbia Part 10 HSRC Revisions, Minnesota RSM 6115.0410 (Continued)

Mount Polley Expert Panel Recommendations (2014)	Montana Metal Mine Reclamation Act Section 82-4 Revisions (SB 409) (2015)	British Columbia Health, Safety and Reclamation Code Part 10 Revisions (2016)	Minnesota Department of Natural Resources Section RSM 6115.0410 New Dams or Enlargements (2003)
Improve dam safety guidelines: Recognizing the limitations of the current Canadian Dam Association (CDA) guidelines incorporated as a statutory requirement, develop improved guidelines that are tailored to the conditions encountered with TSFs in British Columbia and that emphasize protecting public safety.	**Section 5. Tailings Storage facility - design document - fee.** (g) a demonstration through site investigation, laboratory testing, geotechnical analyses, and other appropriate means that the tailings, embankment, and foundation materials controlling slope stability are not susceptible to liquefaction or to significant strain-weakening under the anticipated static or cyclic loading conditions, to the extent that the amount of estimated deformation under the loading conditions would result in loss of containment; (h) for a new tailings storage facility, design factors of safety against slope instability not less than: (i) 1.5 for static loading under normal operating conditions, with appropriate use of undrained shear strength analysis for saturated, contractive materials; (ii) 1.3 for static loading under construction conditions if the independent review panel created pursuant to [section 6] agrees that site-specific conditions justify the reduced factor of safety and that the extent and duration of the reduced factor of safety are acceptable; and (iii) 1.2 for postearthquake, static loading conditions with appropriate use of undrained analysis and selection of shear strength parameters. Under these conditions, a postearthquake factor of safety less than 1.2 but greater than 1.0 may be accepted if the amount of estimated deformation does not result in loss of containment. (i) for a new tailings storage facility, an analysis showing that the seismic response of the tailings storage facility does not result in the uncontrolled release of impounded materials or other undesirable consequences when subject to the ground motion associated with the 1-in-10,000-year event, or the maximum credible earthquake, whichever is larger. Any numeric analysis of the seismic response must be calculated for the normal maximum loading condition with steady-state seepage. The analysis must include, without limitation, consideration of: (i) anticipated ground motion frequency content; (ii) fundamental period and dynamic response; (iii) potential liquefaction; (iv) loss of material strength; (v) settlement; (vi) ground displacement; (vii) deformation; and (viii) the potential for secondary failure modes.	**10.1.8 (**1) Seismic and flood design criteria for tailings storage facilities and dams shall be determined by the engineer of record based on the consequence classification determined under section 10.1.7 of this code in consideration of the HSRC Guidance Document, subject to the following criteria: (a) for tailings storage facilities that store water or saturated tailings, (i) the minimum seismic design criteria shall be a return period of 1 in 2475 years, (ii) the minimum flood design criteria shall be a return period1/3rd of the way between the 1 in 975-year event and the probable maximum flood, and (iii) a facility that stores the inflow design flood shall use a minimum design event duration of 72 hours; (b) for tailings storage facilities that cannot retain water or saturated tailings, (i) the minimum seismic design criteria shall be a return period of 1 in 975 years, and (ii) the water management design shall include an assessment of tailings facility erosion and surface water diversions as well as measures to prevent impounded tailings from becoming saturated that consider the consequence classification as determined under section 10.1.7 of this code. (2) The environmental design flood criteria shall be determined by a Professional Engineer in consultation with other qualified professionals. **10.1.9** For a tailings storage facility design that has an overall downstream slope steeper than 2H:1V, the manager shall submit justification by the engineer of record for the selected design slope and receive authorization by the chief inspector prior to construction. **10.1.10** For a tailings storage facility design that has a calculated static factor of safety of less than 1.5, the manager shall submit justification by the engineer of record for the selected factor of safety and receive authorization by the chief inspector prior to construction.	**Subp. 8. Permit standards.** The commissioner shall determine if the proposal is adequate with respect to: A. For Class I, a showing of lack of other suitable feasible and practical alternative sites, and economic hardship which would have a major adverse effect on population and socioeconomic base of the area affected. B. For Class II, a showing of lack of other suitable feasible and practical alternative sites and that the dam will benefit the population or socioeconomic base of the area involved. C. The need in terms of quantifiable benefits. D. The stability of the dam, foundation, abutments, and impoundment under all conditions of construction and operation, including consideration of liquefaction, shear, or seepage failure, overturning, sliding, overstressing and excessive deformation, under all loading conditions including earthquake. This determination must be based on current, prudent engineering practice, and the degree of conservatism employed must depend on hazards. E. Discharge and/or storage capacity capable of handling the design flood based on current, prudent engineering practice and the hazard classification. F. Compliance with prudent, current environmental practice throughout its existence.

	(j) if a pseudo-static stability analysis is performed to support the design, a justification for the use of the method with respect to the anticipated response to cyclic loading of the tailings facility structure and constituent materials. The calculations must be accompanied by a description of the assumptions used in deriving the seismic coefficient.		

[i] BC HSRC for Mines Version 1.0, July 2016. The alternatives assessment for TSFs will consider BAT and will provide a comparative analysis of options considering the following sustainability factors: Environment; Society; Economics.

Appendix A – van Zyl and Vick Letters

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Honourable Bill Bennett
Minister of Energy and Mines
PO BOX 9060, STN PROV GOVT
Victoria, BC V8W9E2

August 18, 2015

Dear Minister Bennett,

I have observed with interest the public dialogue around tailings management options following the Mount Polley tailings storage facility failure and would like to take this opportunity to provide you with some of my thoughts with respect to tailings management facility (TMF) design related to best available technology (BAT).

The Independent Expert Engineering Investigation and Review Panel identified three components to accomplish BAT and listed filtered tailings ("dry stack"), underground backfill and mined out pits as examples of potential management options.

The geology, topography and climate of British Columbia are diverse and varied. Considerations for selecting a tailings management option requires site specific measures to result in a stable and resilient tailings deposit; one size does not fit all. Structural integrity must be the number one priority of the design and management of every TMF.

Following the release of the Panel report, the Panel met in Prince George on March 20, 2015, as part of a discussion on tailings management involving government, First Nations and industry. There was further discussion about this topic, specifically risk management practices, including redundancies in design. It was emphasized that, like most technologies, there are variations for site specific tailings management that could satisfy BAT and there are opportunities for industry to explore new innovations with respect to tailings management.

In my opinion, BAT is not a single technology; its selection is based on a site-specific risk management process with the outcome of a stable and resilient tailings deposit. Thank you for the opportunity to provide you with my thoughts. Should you be interested, I would be pleased to discuss these ideas with you further.

Sincerely,

Dirk van Zyl.

Dirk van Zyl, Ph.D., P.Eng.
Professor, Chair of Mining and the Environment

Steven G. Vick

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April 22, 2016

MEND Secretariat
555 Booth Street
Ottawa, ON
K1A 0G1
attn: Gilles A. Tremblay
Manager, Mine Closure and Ecosystem Risk Management

Review comments
Draft Report for the State of Practice Assessment of Tailings Management Technologies

Dear Mr. Tremblay:

1.0 INTRODUCTION

This letter forwards the writers' review comments on the March 4, 2016 Draft Report for the State of Practice Assessment of Tailings Management Technologies by KCB. These comments represent exclusively the views of the writer. They have been prepared at your request and without compensation by or consultation with MEND or any interested party. In the remarks that follow, *report* and *study* refer to the above-referenced KCB document unless otherwise indicated. Similarly, page numbers, tables, or figures in brackets [ ] refer to the KCB report.

The writer has long advocated the need for better integration of dam safety and geochemical aspects of tailings management, so it is encouraging to see MEND sponsor this work. Both MEND and the report's authors should be complimented for undertaking this effort. The end product will serve as a primary resource for tailings management.

MEND's Terms of Reference[1] make it clear that the study is an outgrowth of the failure of the Mount Polley tailings dam in August, 2014 and consequent recommendations of the Mount Polley report[2] for preventing such failures elsewhere. But regrettably, events have overtaken this incident. November, 2015 saw the failure of the Fundao tailings dam owned by Samarco in Brazil,

1 MEND PROJECT - TERMS OF REFERENCE, Study of tailings management technologies
2 Independent Expert Engineering Investigation and Review Panel, 2015, Report on Mount Polley Tailings Storage Facility Breach, Province of British Columbia.

a company with corporate connections to Canada. As of this date, the failure has resulted in more than $40 billion in direct costs to Samarco and its parent companies, 22 fatalities, destruction of the town of Bento Rodriguez, and criminal homicide indictments of Samarco executives, engineers, and consultants. Samarco has changed the complexion of tailings dam failures, elevating their consequences to an entirely new level in contemporary experience. Both the report and these comments should be read in this light.

Even more regrettably, the Samarco experience is not unique. Tailings dam failures have cost more than 1800 lives worldwide since 1960, despite improvements in tailings management practices over this period[3]. This too is necessary for proper perspective on the report and the remarks forwarded here.

The statistical implications of these incidents can be understood from Appendix I of the Mount Polley report, using the subpopulation of tailings dams in British Columbia as a sample of tailings dams in Canada more broadly. The failure frequency for BC tailings dams is about $1.7x10^{-3}$/yr. At the time of the Mount Polley report, there were 120 active tailings impoundments at 60 mines in BC, or a ratio of 2:1. This same ratio would imply that the 177 mines in Canada [Table 1.1] have 354 tailings facilities of some kind. Of these, 20 use alternative tailings technology, leaving 334 conventional impoundments. Applying the BC failure frequency, it is easily shown that the annual probability of at least one failure of a conventional tailings facility in Canada is 0.43, or almost a 50/50 chance, and the average failure recurrence interval is about two years. Thus, absent substantive changes in current tailings management technology, tailings dam failures are and will continue to be an expected and statistically predictable occurrence in Canada. Only by appreciating this can informed decisions regarding tailings technology be made.

2.0 BAT

2.1 Definitions

The Mount Polley report defined BAT for physical stability according to the following three criteria:

1. Eliminate surface water from the impoundment.
2. Promote unsaturated conditions in the tailings with drainage provisions.
3. Achieve dilatant conditions throughout the tailings deposit by compaction.

This definition is cited in the report. Notwithstanding, the report advances its own definition of BAT, in the process traveling far afield from its Terms of Reference. It proposes that BAT should be enlarged to encompass not just physical stability, but geochemical stability as well:

3 Vick, S., 2011, The Consequences of Tailings Dam Failures, Cross Canada Lecture, Canadian Geotechnical Society

The chief goal of applying BAT is to achieve physical and geochemical stability for operations and closure [5].

It goes on to expand BAT's scope still further:

Therefore, BAT should be defined as processes and designs that are most suitable for a project and climate that enhance (reduce risk of) physical, geochemical, biophysical and social stability [8].

The reader then learns that BAT is even more broad:

Therefore BAT cannot be "one-size-fits-all" as it needs to be specific to the climate, geology, geomorphology and sensitivity of the downstream receptors of the tailings facility site, the tailings characteristics, and the social situation of the project [8].

And finally, BAT emerges as an all-encompassing array of technologies, management strategies, risks, and activities over every stage of the life cycle:

For purposes of this study the definition of Best Available Technology (BAT) means the combination of technologies and management strategies that most effectively reduce the economic, physical, geochemical, ecological and social risks associated with tailings during all stages of operation and closure. BAT includes site selection considerations, technologies and design features that provide a resilient and robust tailings facility during operations and post-closure. BAT should be implemented at every stage of the tailings life cycle [8].

Unlike those advanced in the Mount Polley report, these definitions contain no objective criteria by which they can be judged. As such, BAT is whatever you want it to be. And if everything is BAT, then nothing is BAT.

It would have been better had the report more faithfully adhered to its Terms of Reference.

2.2 Multiple Objectives, Tradeoffs, And Multiple Accounts Analysis

In its attempts to define BAT, the report becomes entangled in another problem. BAT now has multiple attributes, objectives it seeks to achieve. These include [iii]:

1. physical stability
2. geochemical stability
3. ecological stability
4. biophysical stability
5. social stability

and added to these [vi] is:

6. life cycle costs

While these are all worthy goals, the report recognizes that they may conflict and allows that risk tradeoffs among them will be necessary (i.e., geochemical risk for physical risk or vice-versa). But having raised the issue, it then begs the question of just how these tradeoffs should be made, leaving this task for government and industry [iii]. Later it allows that Multiple Accounts Analysis is the right tool for the job [66] but again does not elaborate.

The report frames physical stability as merely one desirable attribute among many. This is a false premise. No other objective can be achieved without first assuring physical stability. Restated in terms of the report's BAT objectives:

1. there can be no geochemical stability if the dam fails
2. there can be no ecological stability if the dam fails
3. there can be no biophysical stability if the dam fails
4. there can be no social stability if the dam fails
5. and there can be virtually immeasurable life-cycle costs if the dam fails

Hence, dam safety is not one among many competing objectives, but prerequisite to all of them. This is contrary to Multiple Accounts Analysis, which is predicated on a zero-sum decision rule. That is, any nonzero weighting factor assigned to one objective necessarily reduces the influence of some other objective in the decision outcome. And the more such objectives there are, the greater the dilution becomes.

But dam safety is a decision constraint, not a decision outcome. If tradeoffs of dam safety are acceptable, the result is to accept dam failures. It is inconceivable that failures like Mount Polley or Samarco could be rationalized by having traded off safety for something else.

3.0 DEWATERED TAILINGS VERSUS WATER COVERS

A continuing theme throughout the report is that physical risks are in conflict with geochemical risks, and the corollary that water covers eliminate ARD while dewatered tailings promote it [iii, 2, 21, 34, 39]. While this dichotomy exists at some level, the report tends to oversimplify and overstate it. For example:

... to limit oxidation and prevent Acid Rock Drainage (ARD) the best practice is to keep Potentially Acid Generating (PAG) tailings submerged with an appropriate water cover.[iii]

Standard practice with conventional PAG tailings is to limit oxidation and reduce the risk of ARD by maintaining greater than 85% saturation in the tailings with an appropriate pond or water cover (INAP 2014).[2]

...saturation of sulphidic tailings is the most successful method of controlling ARD...[20]

...to limit oxidation and prevent ARD, keeping PAG tailings submerged with an appropriate water cover, often greater than 1 m deep is recommended (INAP 2014) [21]

The writer can find no such references to best practice, standard practice, or recommended practice in Chapter 6 of INAP (2014)[4]. What Section 6.6.7 INAP actually says is:

Disposal of acid generating materials below a water cover is ***one of*** *the most effective methods for limiting ARD* ***generation***. [emphasis added]

In fact, water covers are only one of more than 28 control methods described in INAP (2014) and one of 14 such methods enumerated in the report itself [Table 1.3]. But as far as the writer can determine, the report devotes only two sentences to the entire category of dry covers [41]. And while sulphide flotation does receive some discussion [37], it later falls off the list of case histories without explanation. Water covers aside, the entire topic of geochemical control occupies only three pages of the 81 page report.

The above citation from INAP alludes to another overlooked factor. It is not simply generation, but transport of ARD reaction products that produces ARD consequences and risks. Water covers produce saturation and flow gradients that enhance transport of anionic constituents like SO_4 and Se that are very difficult and costly to treat, even in the absence of ARD reaction products. Indeed, the entire matter of contaminant transport that constitutes fully half the ARD problem receives no discussion at all. For example, the alternating cycles of oxidation and flushing that occur in cyclone sand dams are well known but receive no mention [49]. Neither is it noted that dewatered tailings are typically nonsegregated, with reduced conductivity to water and oxygen that retard both oxidation and transport. Dewatered tailings may also offer opportunities for sequential “cell” deposition and covering that serve the same ends, but this is not explored.

The Terms of Reference intend that the report promote informed decisionmaking about tailings technology. If so, the report needs to spend more time explaining how physical and chemical stability can be reconciled, and less time insisting that they cannot be.

4.0 FEASIBILITY AND COSTS

In a number of instances the report takes note of the limited tonnage for dewatered tailings applications to date. It is commonly claimed that scaleup difficulties make high-tonnage operations unfeasible, and the writer had looked forward to learning why. The report hints [Figure 6.4] that, to the contrary, there may be economies of scale for larger operations, but this important question remains unanswered.

As the Mount Polley report discussed, the cost of alternative technologies has been the chief factor in their adoption to date. The cost estimates provided in the report, while crude, are a first step in illuminating this factor. The more important question, however, concerns not so much

4 The International Network for Acid Prevention, 2014, Global Acid Rock Drainage Guide

the relative costs of these technologies compared to conventional methods, but their impact on the overall cost of the mine over the life of the operation. A followup study to evaluate the cost of alternative tailings technologies at actual operating mines—say a high tonnage open-pit operation, low-tonnage open pit, and an underground mine—would be a worthwhile undertaking for MEND to consider.

4.0 OTHER TOPICS

Most readers are likely to find, as did the writer, Section 5 on Case History Review of Tailings Management Technologies and Practices to be the most useful part of the report. It is surprising, however, that cyclone sand dams are introduced at this stage as an alternative technology when they are actually a well-established aspect of conventional technology that do not seem to warrant separate status.

And lastly, the report included a clarifying comment on the Mount Polley report by Professor Dirk van Zyl [6]. Having done so, it is obligated to also acknowledge both of the items in Dr. van Zyl's subsequent correspondence attached to this letter.

Thank you for the opportunity to share these ideas, and I trust you will find them useful.

Yours very truly,

Steven Vick

attachment: letter from Dirk van Zyl to Bill Bennett dated September 14, 2015

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NORMAN B. KEEVIL Institute of Mining Engineering

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Honourable Bill Bennett
Minister of Energy and Mines
PO BOX 9060, STN PROV GOVT
Victoria, BC V8W9E2

September 14, 2015

Dear Minister Bennett,

This correspondence is intended to provide two important clarifications with respect to my letter of August 18, 2015 which presented some of my thoughts with respect to tailings management facility (TMF) design related to best available technology (BAT). The August 18, 2015 letter does not suggest that the overall goal of moving to zero TMF failures, as stated in the Independent Expert Engineering Investigation and Review Panel Report, can be reconsidered. It must remain the overall target.

Lastly, the August 18, 2015 letter represents my observations and comments and does not represent the opinions of the other Panel members.

Sincerely,

Dirk van Zyl.

Dirk van Zyl, Ph.D., P.Eng.
Professor, Chair of Mining and the Environment

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"Technical Support for Grassroots Public Interest Groups"

October 16, 2017

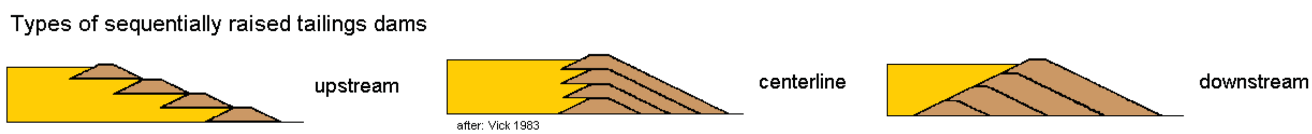
To: Kevin Lee
Senior Staff Attorney
Minnesota Center for Environmental Advocacy
klee@mncenter.org

Re: Draft Dam Safety Permit Numbers 2016-1380 and 2016-1383

COMMENTS ON DRAFT DAM SAFETY PERMIT NUMBER 2016-1380, FLOTATION TAILINGS BASIN

1. The permit allows upstream-type construction to continue.

Safety should be the prime consideration in the design, construction, operation, and closure of a dam, whether this be a water supply reservoir or a tailings dam. However, unlike water supply reservoir dams, which are typically of concrete arch-type or downstream-type construction, tailings dams can use centerline-type and upstream-type construction, each of which is inherently less safe than downstream-type dam construction.



As can be seen from the illustrations, upstream-type dam construction uses the tailings themselves for support for most of dam construction stages. Centerline and downstream-type construction, even though it is also done in stages like upstream, depends only on materials that are sized, placed, compacted, and subsequently tested for support of the sequential stages. When tailings are hydraulically spigotted into the impoundment, their placement and water content are not uniform. There is no practical way to test the characteristics of the tailings material to assure that it is subsequently drained of excess water after hydraulic placement, and that is has the consistency and density assumed by the design modeling.

This lack of control of the underlying tailings introduces a level of uncertainty into upstream-type construction that does not exist with centerline and downstream-type dam construction. This does not mean that upstream-type dams cannot be safely designed and constructed, but it does mean there is more risk inherent in the upstream approach. Moreover, it turns out that upstream-type tailings dam construction has proven to be the most risky and problematic type of dam construction.

The only reason to use both centerline and upstream construction, over a conventional downstream-type approach, is to save money. At best, it might be argued that safety and cost carry equal weight in tailings dam considerations, but today for most design, operation, and closure tailings dam considerations cost carries more weight than safety. The impoundment proposed by PolyMet is a good example of this imbalance.

The Mt Polley Expert Panel,[1] which was convened by the Province of British Columbia after the Mt Polley tailings dam failure, was asked to analyze the failure mechanisms at Mt Polley and to make recommendations on preventing such failures in the future. The Panel noted:

> *"Mount Polley illustrates that dam safety guidelines intended to be protective of public safety, environmental and cultural values cannot presume that the designer will act correctly in every case."* (Expert Panel 2015, p. 133)

In tailings dam accidents we do not see a preponderance of one or two failure causes dominating. What we see is that the number and distribution of failure type is remarkably similar. That is, overtopping, seismic failure, foundation issues, internal seepage, slope instability, and structural failure all have similar number-of-failure profiles for both active and inactive tailings dam failures (Bowker & Chambers 2016).[2] This strongly suggests there is something more fundamental than the inability to deal with the causes of one or two failure types. It suggests we have failed to recognize and address something that is affecting all of these failure types. I suggest one fundamental problem is that safety is not being given clear priority over cost in the design, construction, operation, and closure of tailings dams. This affects not only the design of tailings dams, where cost plays a dominant role, but also operational management of dams, where there it too much incentive to cut corners when times get tough. Because of these factors, we are not seeing a decrease in the rate of failures for catastrophic tailings dam failures (Bowker & Chambers 2016).

2. The permits allow water to be permanently impounded.

One of the key recommendations of the Mt Polley Expert Panel is:

> *"The goal of BAT (*Best Available Technology*) for tailings management is to assure physical stability of the tailings deposit. This is achieved by preventing release of impoundment contents, independent of the integrity of any containment structures. In accomplishing this objective, BAT has three components that derive from first principles of soil mechanics:*
>
> *1. Eliminate surface water from the impoundment.*
>
> *2. Promote unsaturated conditions in the tailings with drainage provisions.*
>
> *3. Achieve dilatant conditions throughout the tailings deposit by compaction."* (Expert Panel 2015, p. 121)

By building on top of an existing upstream-type impoundment, and/or by allowing water to pond on the tailings post-closure, the tailings facility proposed by PolyMet violates all of the recommendations listed above. The danger is leaving water on top of the tailings is 2-fold.

First, water means partial or full saturation of the tailings below. If the tailings are saturated, they have essentially no weight-bearing capacity under seismic loading. This means that any structure built on top of saturated tailings, like an upstream-type tailings dam, is susceptible to failure under seismic shaking. The permit requires further testing of the in-palace tailings to confirm assumptions used in the modeling upstream dam safety calculations, but at best, these are only interrupted samples, as opposed to an engineered structure that is required of downstream-type and centerline-type dams. It is too expensive to sample on a density that would truly provide enough data to prove that the model assumptions are

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[1] Expert Panel 2015. Report on Mount Polley Tailings Storage Facility Breach, Independent Expert Engineering Investigation and Review Panel, Province of British Columbia, January 30, 2015

[2] Root Causes of Tailings Dam Overtopping: The Economics of Risk & Consequence, Lindsay Newland Bowker and David M. Chambers, International Seminar on Dam Protection Against Overtopping, Protections 2016, Fort Collins, CO, September 2016.

accurate. In addition, if the sampling shows that the modeling assumptions are incorrect, from a cost and technical perspective it is probably too late to mitigate these issues with post-deposit drains or rock buttresses.

Second, water remaining on and in the tailings acts as a deadly mobilizing agent should a catastrophic failure occur. Dry tailings can be mobilized if support is removed, but the distance they will move is orders of magnitude less than tailings saturated with water. The Mt Polley Expert Panel recognized this by remarking:

> *"Mount Polley failure shows why physical stability must remain foremost and cannot be compromised. ... No method for achieving chemical stability can succeed without first ensuring physical stability: chemical stability requires above all else that the tailings stay in one place."* (Expert Panel 2015, p. 124)

3. Does not specify dam hazard classification.

The permit specifies:

> *"The Permittee understands the hazard classification of this dam could change …"*

Not only do we not know the hazard classification being assigned to the tailings dam, but the wording also suggests that the anticipated hazard classification will not be that of the highest risk.

Because of the size of this dam, and environmental and economic destruction it could cause if it failed catastrophically, it should be classified at the highest hazard category of risk.

4. The permit does not require an Independent Tailings Review Board.

The permit does not require the use of an Independent Tailings Review Board. The use of an Independent Tailings Review Board oversight board is a recommendation of the Mt Polley Expert Review Panel.

Independent review also the recommendation of virtually every major post-Mt Polley review conducted by regulatory bodies (e.g. British Columbia, Montana, and the IFC/World Bank) and professional organizations (e.g. the Mining Association of Canada and the International Council on Mining and Metals[3]).

COMMENTS ON DRAFT DAM SAFETY PERMIT NUMBER 2016-1383, HYDROMETALLURGICAL RESIDUE FACILITY

1. The permit does not require an Independent Tailings Review Board.

As noted for the flotation tailings basin, there is no Independent Tailings Review Board oversight required. It would be simple to require an ITRB that would review both facilities.

Thank you for the opportunity to comment.

Sincerely;

David M Chambers

David M Chambers, Ph.D., P. Geop.

---

[3] Position Statement on Preventing Catastrophic Failure of Tailings Storage Facilities, International Council on Mining & Metals, December, 2016

October 12, 2017

Mr. Kevin Lee
Senior Staff Attorney
Minnesota Center for Environmental Advocacy
26 East Exchange Street, Suite 206
Saint Paul, MN 55101

Subject: Comments on Draft Dam Safety Permit 2016-1380 (Flotation Tailings Basin), Updated Permit Application Documents, and Outstanding Permit Issues

Dear Mr. Lee:

I am writing to provide my comments to the Minnesota Center for Environmental Advocacy (MCEA) regarding the subject draft permit for the NorthMet project. My comments, presented below, are based on my review of the draft permit, the NorthMet Dam Safety Permit Application for the Flotation Tailings Basin (PolyMet, May 2017), and the Template for Pilot/Field Testing of Bentonite Amendment of Tailings (PolyMet, April 2017). Please note that I also reviewed the Draft Dam Safety Permit 2016-1383 for the Hydrometallurgical Residue Facility, but I have no comments on this draft permit.

Comments on Draft Dam Safety Permit 2016-1380

1. <u>Condition 31: Bentonite Testing</u>:

   As stated in the Template for Pilot/Field Testing of Bentonite Amendment of Tailings (PolyMet, April 2017), the objective of bentonite amendment of the dams, beaches, and pond bottom are to limit oxygen infiltration into the tailings by reducing water infiltration and maintaining a continuous areal zone of saturation in the bentonite-amended layers. However, neither the permit application nor the pilot/field testing template specifies a requirement for the degree of saturation that must be maintained in the bentonite-amended dams and beaches. Also, the pilot/field testing template does not describe any laboratory QC testing that will be performed to verify the efficacy of the field mixing or the adequacy of the proposed 3 % granular bentonite amendment for meeting the design objective. According to Specification 03100 in Version 7 of the Flotation Tailings Management Plan (PolyMet, May 2017), laboratory test requirements should be part of the pilot testing plan. Finally, although the pilot/field testing template presents a list of considerations for field testing and describes various field testing and monitoring methods that “could” be used, the template provides only a conceptual level of detail for how the field tests may be carried out and does not specify the performance metrics that will be used to determine success or failure. Thus, the template, in its current form, falls short of being a field/pilot testing plan.

Based on the above, the DNR should require PolyMet to produce a field/pilot testing plan prior to initiation of field work. I recommend the following revision to Permit Condition 31:

> *"Prior to dam construction, Permittee shall prepare a pilot/field testing plan for the bentonite amendment that represents an expanded and more detailed version of the April 2017 Template for Pilot/Field-Testing of Bentonite Amendment of Tailings. This plan should clarify the design criteria for the layers, specify the laboratory and field testing methods that will be used to verify adequate mixing, placement, and performance in accordance with the design, specify how the field tests will be carried out (including testing/monitoring frequencies, locations, and installation details), and establish the performance metrics that will be used to determine success or failure of the pilot tests. Permittee shall obtain written approval from the DNR Dam Safety Engineer of both the pilot/field testing plan and a subsequent report of the results of the pilot/field testing. Construction may not commence until such written approval of both the plan and the report is obtained."*

The EOR Review Team expressed concerns over the adequacy of a 3 % bentonite addition and the effectiveness of injecting bentonite into the pond bottom for creating a reliable infiltration barrier. While I share these concerns, a more fundamental problem is the lack of a design basis to support the feasibility of these layers for meeting the project objective. For example, the primary objective of the bentonite-amended layers in the dams and beaches is to provide a barrier to oxygen migration into the tailings by maintaining a "continuous areal zone of saturation." However, the permit documents do not specify a required degree of saturation, no mix design work has been completed to justify the proposed mixture, and no moisture retention testing or unsaturated flow modeling has been conducted to assess what level of saturation can be realistically expected to be maintained in the field. It is not possible to achieve fully saturated conditions in these layers at the time of placement (even wet of optimum compaction is not likely to yield a degree of saturation greater than about 90 %), and the degree of saturation will be prone to decrease, rather than increase, over time. While I fully support the use of pilot/field testing to establish means and methods and demonstrate performance, PolyMet still needs to establish appropriate performance criteria and design the layers accordingly before conducting field trials.

This introduces another concern: it appears that PolyMet is basing the use of 3 % granular bentonite on the results of a single laboratory hydraulic conductivity test conducted on a trial mixture of 3 % bentonite-amended tailings, which are provided in Attachment D of the Flotation Tailings Basin Dam Safety Permit Application. Replicate tests should be performed to demonstrate reproducibility. Also, no backup documentation for this test (e.g., no table of head measurements versus time, no plot of hydraulic conductivity as a function of time or pore volumes of flow, no measurements of inflow/outflow balance, etc.) is provided. As a result, I was not able to examine the test data or check the accuracy of the calculations. The reported hydraulic conductivity ($\sim 1.5 \times 10^{-7}$ cm/s) is much lower than expected based on comparison with data available for similar mixtures in the geoenvironmental engineering literature (e.g., see Abichou et al. 2000). I remain concerned that a 3 % granular bentonite amendment will be too low to create a homogenous layer in the field that is free of zones containing no bentonite. Bench-scale tests need to be performed using materials with gradations representative of those anticipated for the bentonite-amended layers to determine the percentage of bentonite required to meet the design criteria.

PolyMet proposes to conduct monitoring and mini-experiments during the pilot/field tests to assess factors that may interfere with or degrade the maintenance of a continuous zone of areal saturation in the bentonite-amended tailings layers, including desiccation, freeze-thaw degradation, root penetration, and incompatibility with pond water. Although I support this approach, the template for the pilot/field testing provides only a conceptual level of detail for how the mini-experiments may be carried out. Also, most of these factors (desiccation, freeze-thaw, and pond water incompatibility, in particular) can and should be investigated first by laboratory testing as part of mix design work conducted prior to field testing.

PolyMet provides no evidence to support the claim that the proposed bentonite-amended tailings layers, over the long term, will not be susceptible to root penetration, or that placing these layers beneath a 30-inch vegetated layer will provide adequate protection against wet-dry or freeze-thaw cycling. These processes can create macropores (i.e., large scale features such as cracks and fissures) that alter the network of pores controlling retention and movement of water (and air) in barrier layers. These types of problems are well documented in a recent, peer-reviewed study by Benson et al. (2011).

Likewise, proof of concept for the bentonite pond bottom remains inadequate. PolyMet has proposed three possible subaqueous placement methods (i.e., broadcasting of bentonite granules or pellets, bentonite injection into the existing bottom, or placement of a geosynthetic clay liner over the existing bottom), none of which are supported by laboratory studies, field case studies of successful use on other projects, or any other type of feasibility assessment. What is the contingency plan if none of the three proposed methods prove to be feasible based on the field test results?

Lastly, there does not appear to be a sound technical basis for the specified maximum hydraulic conductivity of 10^{-6} cm/s for the bentonite-amended tailings layers to be placed on the FTB dam side slopes and beach areas. According to PolyMet, the bentonite-amended tailings are meant to act as an oxygen barrier. However, no evidence is provided to demonstrate that bentonite-amended tailings with a hydraulic conductivity of 10^{-6} cm/s will be effective as an oxygen barrier. Moisture retention testing and unsaturated flow modeling are needed to assess the performance of these layers.

2. <u>Condition 45: Future Closure Considerations</u>:

Subpart B(2) of Part 6132.2200 of the Minnesota Rules states, in part, that storage of reactive mine waste, at closure, must "permanently prevent substantially all water from moving through or over the mine waste." The proposed wet closure for the Flotation Tailings Basin, is designed to allow 6.5 inches per year of percolation (i.e., approximately one-fourth of the average annual precipitation rate) to pass through the tailings. In contrast, dry closure generally achieves much lower percolation rates into the waste, typically less than 5 percent of the average annual precipitation rate and often on the order of a few millimeters per year or less (e.g., see Wilson et al. 1995, Woyshner and Yanful 1995, Ayres et al. 2003, Keller et al. 2010). Although Permit Condition 45 requires the Permittee to continue exploring future closure options (including a dry cap), the DNR should consider making dry closure a permit condition rather than an option for PolyMet to explore at their discretion. Dry closure would be a much better approach for meeting the intent of Part 6132.2200 Subpart B(2).

References:

Abichou, T. et al. (2000). Foundry Green Sands as Hydraulic Barriers: Laboratory Study. *Journal of Geotechnical and Geoenvironmental Engineering*, 126(12), 1174-1183.

Ayres, B., Silveira, C., Ellice, E., Christensen, D., and O'Kane, M. (2003). Development of a cover system design for potentially acid-forming tailings and Peak Gold Mines. *Proceedings, 6th International Conference on Acid Rock Drainage (ICARD)*, 957-963.

Benson, C.H., Albright, W.H., Fratta, D.O., Tinjum, J.M., Kucukkirca, E., Lee, S.H., Scalia, J., Schlicht, P.D, and Wang, X. (2011). Engineered Covers for Waste Containment: Changes in Engineering Properties and Implications for Long-Term Performance Assessment. NUREG/CR-7028 Volume 1, U.S. Nuclear Regulatory Commission, Nuclear Regulatory Research (http://www.nrc.gov/reading-rm/doc-collections/nuregs/contract/cr7028/).

Keller, J., Milczarek, M., Yao, T.M., and Buchanan, M. (2010). The effect of tailings characteristics on cover system success. *Tailings and Mine Waste 2010. CRC Press*, 121-130.

Wilson, G.W., Barbour, S.L., Swanson, D., and O'Kane, M. (1995). Instrumentation and modeling for saturated /unsaturated performance of soil covers for acid generating waste rock, *Hydrogéologie*, 4, 99-108.

Woyshner, M.R. and Yanful, E.K. (1995). Modelling and field measurements of water percolation through an experimental soil cover on mine tailings. *Canadian Geotechnical Journal*, 32, 601-609.

If you have any questions or concerns regarding these comments, please do not hesitate to contact me at 570-412-2069 or michael.malusis@bucknell.edu.

Sincerely,

Michael A. Malusis, Ph.D., P.E.
Consulting Engineer

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File No.:VA101-1/29-A.01
Cont. No.:VA11-00298

February 10, 2011

Mr. Brian Kynoch
Mount Polley Mining Corporation
Suite 200 - 580 Hornby Street
Vancouver, BC V6C 3B6

Dear Brian,

Re: Mount Polley Tailings Storage Facility Engineer of Record

We have completed all assignments and on January 25, 2011 issued to Mount Polley Mining Corporation (MPMC) the final versions of the 'Tailings Storage Facility - Report on the 2010 Annual Inspection' and 'Tailings Storage Facility – Report on Stage 6B Construction'.

We are currently assuming that MPMC will be retaining the services of a separate individual or organization to take over as the Engineer of Record for the tailings storage facility, as a result of Knight Piésold's decision to opt out of the bidding process implemented by MPMC late last year. We would like to facilitate a formal handover to the new individual/group, as it is essential that it be recognized that Knight Piésold will not have any responsibility for any aspects of the on-going operations, or of any modifications to the facilities that are undertaken from now onwards. To date, the tailings impoundment has been developed using the observational approach, wherein the design is modified as appropriate depending on actual performance and conditions. It must be understood that Knight Piésold will no longer have any responsibility for the performance of the tailings storage facility.

The embankments and the overall tailings impoundment are getting large and it is extremely important that they be monitored, constructed and operated properly to prevent problems in the future. Knight Piésold would be happy to assist in the formal handover to the new Engineer of Record.

As we have a long relationship with the Mines Branch and the Ministry of Energy, Mines and Petroleum Resources, we consider that it is prudent to notify them of the change in status. Therefore, we have copied them on this correspondence.

We would like to thank you for our long and constructive association at the Mount Polley Mine and look forward to working together again in the future.

Signed:
Ken Brouwer, P.Eng.
Managing Director

Approved:
Jeremy Haile, P.Eng.
President

Copy To: Don Parsons (IMC), Ron Martel (MPMC), Tim Fisch (MPMC)
Al Hoffman, Chief Inspector of Mines
/kjb

ISO 9001, ISO 14001
OHSAS 18001

Independent Expert Engineering Investigation and Review Panel

Report on Mount Polley Tailings Storage Facility Breach

January 30, 2015

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Report on Mount Polley Tailings Storage Facility Breach

Independent Expert Engineering Investigation and Review Panel

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EXECUTIVE SUMMARY

THE EMBANKMENTS

Three contiguous embankments confine the Mount Polley tailings storage facility (TSF). Of these, the Perimeter Embankment, where the breach occurred, was the northern flank of the TSF.

The embankments are composed of a core with the function of acting as an impervious element. Downstream of the core, a filter zone restrains material in the core from outward migration. The core and filter are then supported by a rockfill zone. In the upstream direction, the core is supported by an upstream fill zone composed of rockfill and/or tailings.

THE BREACH

The breach occurred within the Perimeter Embankment. At the time of the breach, the TSF was permitted under the Ministry of Energy and Mines, Permit M-200, with approval to raise the crest by 2.5 metres. The breach occurred early on August 4, 2014 at a crest elevation 1 metre short of its permitted elevation. Loss of containment was sudden, with no warning. The recorded pond elevation at 6:30 pm on August 3, 2014 was 2.3 metres below the crest.

THE MANDATE

Following the breach of the tailings storage facility at the Mount Polley Mine, the Government of British Columbia, through the Ministry of Energy and Mines, together with the Williams Lake Indian Band and the Soda Creek Indian Band, established an independent expert investigation and review panel (the Panel) to investigate and report on that breach. The Panel was required to submit a final report to the Ministry of Energy and Mines and the Williams Lake Indian Band and the Soda Creek Indian Band on or before January 31, 2015.

The purpose of the investigation has been as follows:

- To investigate and report on the cause of the failure of the tailings storage facility that occurred on August 4, 2014 at the Mount Polley Mine (the Mine) in B.C.
- In addition, the Panel may make recommendations to government on actions that could be taken to ensure that a similar failure does not occur at other mine sites in B.C.
- The Panel is authorized, as part of its investigations and report, to comment on what actions could have been taken to prevent this failure and to identify practices or successes in other jurisdictions that could be considered for implementation in B.C.

EXECUTIVE SUMMARY

Further, it was expected that the Panel would:

- Identify any mechanism(s) of failure of the tailings storage facility.
- Identify any technical, management or other practices that may have enabled or contributed to the mechanism(s) of failure. This may include an independent review of the design, construction, operation, maintenance, surveillance and regulation of the tailings storage facility.
- Identify any changes that could be considered to reduce the potential for future such occurrences.

PANEL ACTIVITIES

The Panel began its inquiry with multiple hypotheses for failure:

- Human intervention
- Overtopping
- Piping and cracking
- Foundation failure

The Panel found no evidence of failure due to either human intervention or failure due to overtopping, notwithstanding the fact that an episode of overtopping over portions of the Perimeter Embankment occurred in May 2014. The question of piping and cracking, which is a common cause of failure of earth dams, received corresponding attention. Although factors of concern were identified by the Panel, it did not find evidence that piping and/or cracking caused the breach.

This reduced the focus of the Panel to failure in the foundation of the embankment. Visual evidence of bodily outward displacement and rotation of the embankment remnants were consistent with foundation failure. A foundation can be weak and fail in a number of ways. One is the presence of a weak layer that had been undetected during design. Another is the presence of a brittle stratum that loses strength as it comes under load and becomes too weak to support the load applied by the embankment and TSF contents, so that failure ensues. Yet another possibility is the presence of a layer that is compressible under the applied load and, when stressed, develops high pore pressure that results in weakening of an otherwise much stronger material. This is termed undrained failure. It was the object of the site studies undertaken by the Panel to determine which of these foundation failure mechanisms prevailed.

EXECUTIVE SUMMARY

The Panel undertook comprehensive Surface Investigations that provided detailed, observable information on the sliding mechanism that had occurred. A challenging and complex Subsurface Investigation was also undertaken, partly in collaboration with the site investigation program initiated by the both the Mines Inspector and Mount Polley Mining Corporation (MPMC), and in addition, by the Panel alone.

The Subsurface Investigation was particularly valuable in defining the controlling stratigraphy in the breach area and identifying that the failure occurred in a glaciolacustrine layer, called Upper GLU. No indication of pre-shearing or the presence of markedly strain-weakening materials was detected, leaving undrained failure in the Upper GLU as the only viable hypothesis. The type and extent of pre-failure site investigations were not sufficient to detect this stratum or to identify its critical nature. The Panel's Subsurface Investigation was structured to obtain undisrupted samples of the Upper GLU and subsequently determine its properties.

The Upper GLU was found to be preconsolidated prior to embankment construction, but became normally consolidated under the loads applied by construction of the Perimeter Embankment. That is, it had experienced prior consolidation and strengthening under loads in its geological past, but not under the loads associated with the Perimeter Embankment, which created the normally consolidated state. Under these conditions, the Upper GLU was compressible and susceptible to undrained failure. This condition had not been recognized in the design of the TSF.

Laboratory tests were performed to determine the undrained strength of the Upper GLU and these parameters were utilized in computer analyses to calculate whether failure should have occurred under the applied load. The results were confirmatory.

MCEA Comments Ex. 06

EXECUTIVE SUMMARY

CONCLUSIONS

The Panel concluded that the dominant contribution to the failure resides in the design. The design did not take into account the complexity of the sub-glacial and pre-glacial geological environment associated with the Perimeter Embankment foundation. As a result, foundation investigations and associated site characterization failed to identify a continuous GLU layer in the vicinity of the breach and to recognize that it was susceptible to undrained failure when subject to the stresses associated with the embankment.

The specifics of the failure were triggered by the construction of the downstream rockfill zone at a steep slope of 1.3 horizontal to 1.0 vertical. Had the downstream slope in recent years been flattened to 2.0 horizontal to 1.0 vertical, as proposed in the original design, failure would have been avoided. The slope was on the way to being flattened to meet its ultimate design criteria at the time of the incident.

REGULATORY OVERSIGHT

The Panel reviewed the roles and responsibilities of the B.C. Ministry of Energy and Mines (the Regulator) and its interactions related to the MPMC TSF. The Panel found that inspections of the TSF would not have prevented failure and that the regulatory staff are well qualified to perform their responsibilities. The Panel found that the performance of the Regulator was as expected.

THE FUTURE

The Panel has examined the historical risk profile of the current portfolio of tailings dams in B.C. and concluded that the future requires not only an improved adoption of best applicable practices (BAP), but also a migration to best available technology (BAT). Examples of BAT are filtered, unsaturated, compacted tailings and reduction in the use of water covers in a closure setting. Examples of BAP bear on improvements in corporate design responsibilities, and adoption of Independent Tailings Review Boards. Specific recommendations are made in the body of the report.

1 | Introduction

Following the breach of the tailings storage facility at the Mount Polley Mine on August 4, 2014, the Government of British Columbia, through the Minister of Energy and Mines, together with the Williams Lake Indian Band and the Soda Creek Indian Band, established an independent expert engineering investigation and review panel (the Panel) to investigate and report on that breach.

1.1 PURPOSE OF THE PANEL

The purpose of the Panel is as follows:

- To investigate into and report on the cause of the failure of the tailings storage facility (TSF) that occurred on August 4, 2014 at the Mount Polley Mine (the Mine) in B.C.
- In addition, the Panel may make recommendations to government on actions that could be taken to ensure that a similar failure does not occur at other mine sites in B.C.
- The Panel is authorized, as part of its investigation and report, to comment on what actions could have been taken to prevent this failure and to identify practices or successes in other jurisdictions that could be considered for implementation in B.C.

Under its Terms of Reference, it is expected that the Panel will:

- Identify any mechanism(s) of failure of the TSF.
- Identify any technical, management or other practices that may have enabled or contributed to the mechanism(s) of failure. This may include an independent review of the design, construction, operation, maintenance, surveillance and regulation of the TSF.
- Identify any changes that could be considered to reduce the potential for future such occurrences.

1.2 PANEL MEMBERS

The members of the Panel are:

- Dr. Norbert R. Morgenstern (Chair), CM, AOE, FRSC, FCAE, Ph.D., P.Eng.
- Mr. Steven G. Vick, M.Sc., P.E.
- Dr. Dirk Van Zyl, Ph.D., P.E., P.Eng.

The detailed Terms of Reference are included as Appendix A.

2 | What Did the Panel Do?

2.1 PANEL ACTIVITIES

In furtherance of its mandate, the Panel undertook the following:

- It retained Thurber Engineering Ltd. (Thurber) to conduct field investigations, data compilation, laboratory testing, and analyses. All of this work proceeded under the direction of the Panel.
- It assembled and inspected related documents in the files of the Mine, its consultants who have acted as the Engineer of Record (EOR), and the Ministry of Energy and Mines (MEM).
- It solicited and collected relevant information from the public at large.
- It conducted a number of personal interviews to clarify information recorded in documents.
- It convened regular formal meetings with recorded minutes.
- It interpreted all of the above to arrive at conclusions and recommendations.

2.2 SUPPORTING INFORMATION

As directed by the Terms of Reference, the Panel has provided this final report, along with the appendices, to the Minister of Energy and Mines, the Williams Lake Indian Band and the Soda Creek Indian Band. The background reports and information used by the Panel for the preparation of this report were also made available to these parties through an online data room. The Panel considers the supporting information and substantiating documentation to be an integral part of its report that is necessary for a proper understanding of its findings.

The B.C. Ministry of Energy and Mines and Ministry of Environment, Conservation Officer Services, have directed the Panel to withhold some of the documents, and redact portions of other documents, so that the Panel's inquiry does not compromise any other investigations and to ensure it is in compliance with the privacy protection provisions of *Freedom of Information and Protection of Privacy Act* (FIPPA) (see Appendix A). The redaction of personal information was completed by Shared Services BC. As a result, these documents, which may have been cited in this report are not available at this time.

The background information was provided to the Panel by many different sources. Appendix B contains further details on background reports and information, and how these were organized and provided to the Minister of Energy and Mines, the Williams Lake Indian Band and the Soda Creek Indian Band.

Within the text of the report and appendices, specific documents are referenced by an endnote, which contains the document number as it relates to where it can be found in the data room. See Appendix B for more details.

Additional technical references are also cited by endnote directly within the body of this report. Endnotes can be found at end of each section of the report. The collected technical references can be found at the end of the report.

The observations of the Panel are supported by referenced documents where possible. The findings of the Panel are outlined in the sections of the report that follow. Conclusions and recommendations are presented in detail at the end of the report.

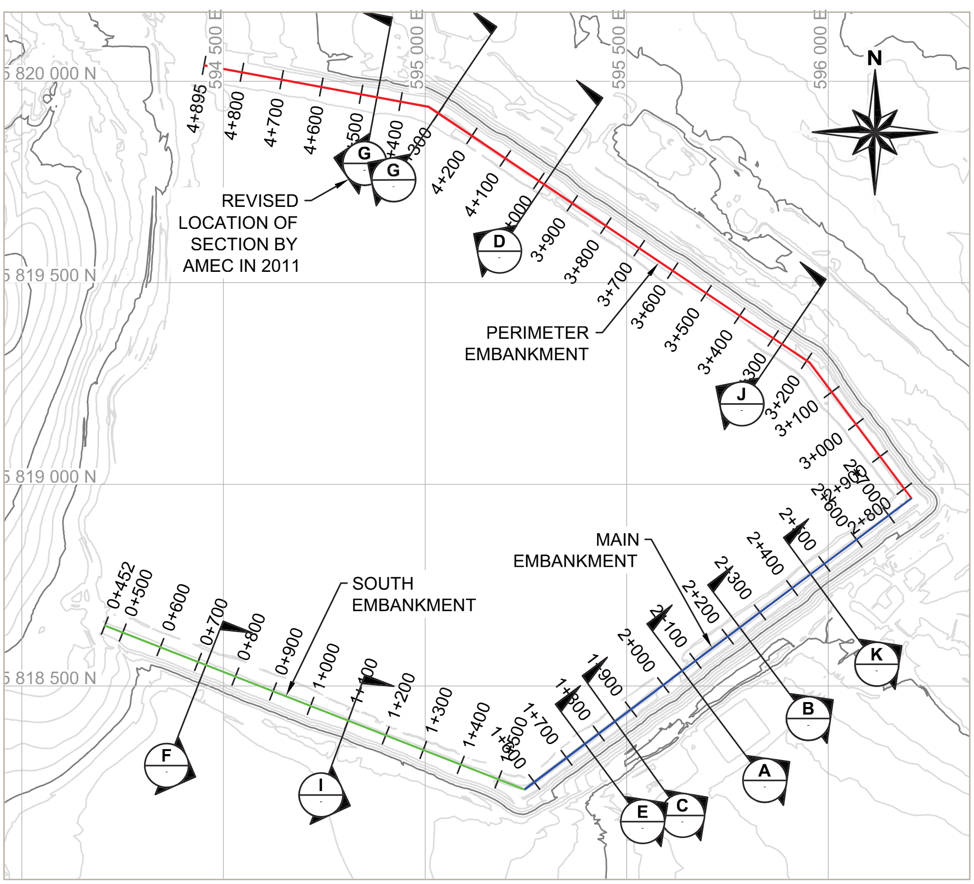
3 | Tailings Storage Facility (TSF)

3.1 DESCRIPTION OF TSF

Mount Polley Mining Corporation (MPMC) operates the Mine, and the British Columbia Ministry of Energy and Mines (MEM) is the Regulator. From the first approved and constructed portion of the TSF, in 1995, to early 2011, Knight Piésold (KP) was the Engineer of Record (EOR). Subsequently, AMEC assumed the responsibility as EOR and had that role at the time of the breach. BGC were to assume that responsibility after the 2014 construction season.

Figure 3.1.1 is a plan of the TSF adapted from the last As-Built Construction Report.[1] It indicates that the TSF was composed of three embankments: the Main Embankment, the Perimeter Embankment, and the South Embankment. The TSF is closed to the west by rising natural ground. The figure also indicates the location of instrumented control sections utilized by the succession of EORs. The breach occurred in the Perimeter Embankment near Section G.

FIGURE 3.1.1: TAILINGS STORAGE FACILITY PLAN



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3 | Tailings Storage Facility (TSF)

3.2 CONSTRUCTION OF TSF AND POND ELEVATION

The Main Embankment and the Perimeter Embankment were the first to go into construction, with the Starter Dam completed in 1996. The South Embankment followed in later years. **Figures 3.2.1** and **3.2.2** present the sections of the Main and Perimeter Embankments at the end of the 2013 construction season. **Figure 3.2.1** is for the Main Embankment at Section A, approximately the highest section. **Figure 3.2.2** is for the Perimeter Embankment and represents the closest instrumented section (Section D) to the breach zone at the time of failure.

FIGURE 3.2.1: MAIN EMBANKMENT AT SECTION A

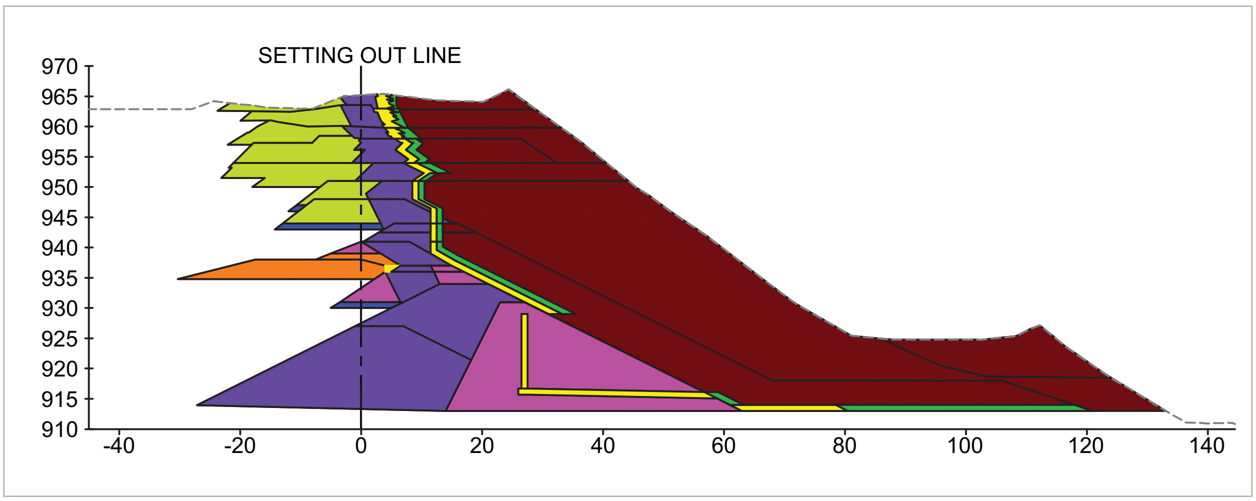
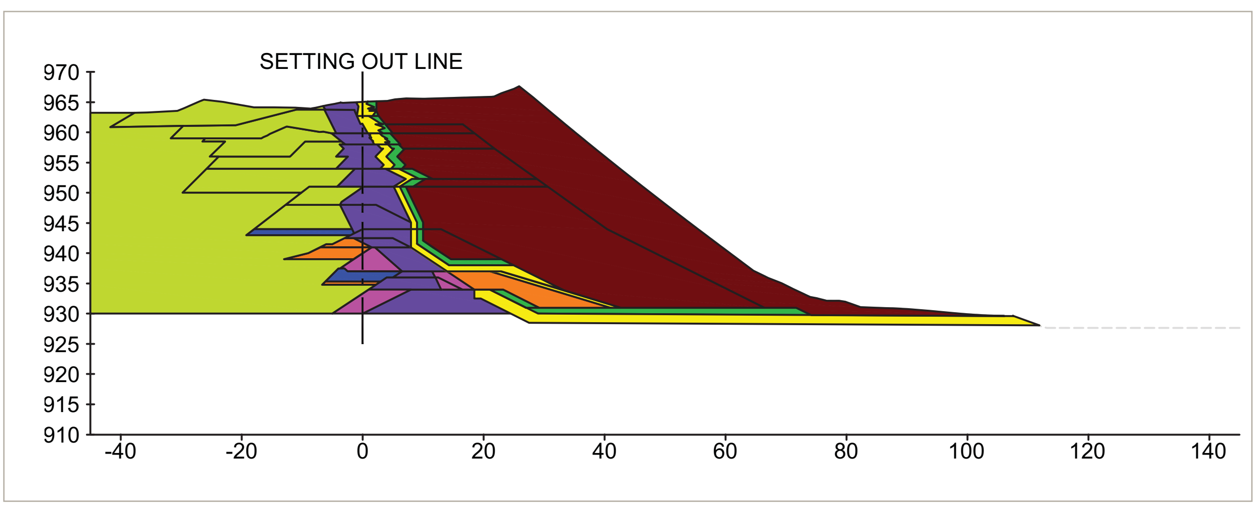


FIGURE 3.2.2: PERIMETER EMBANKMENT AT SECTION D



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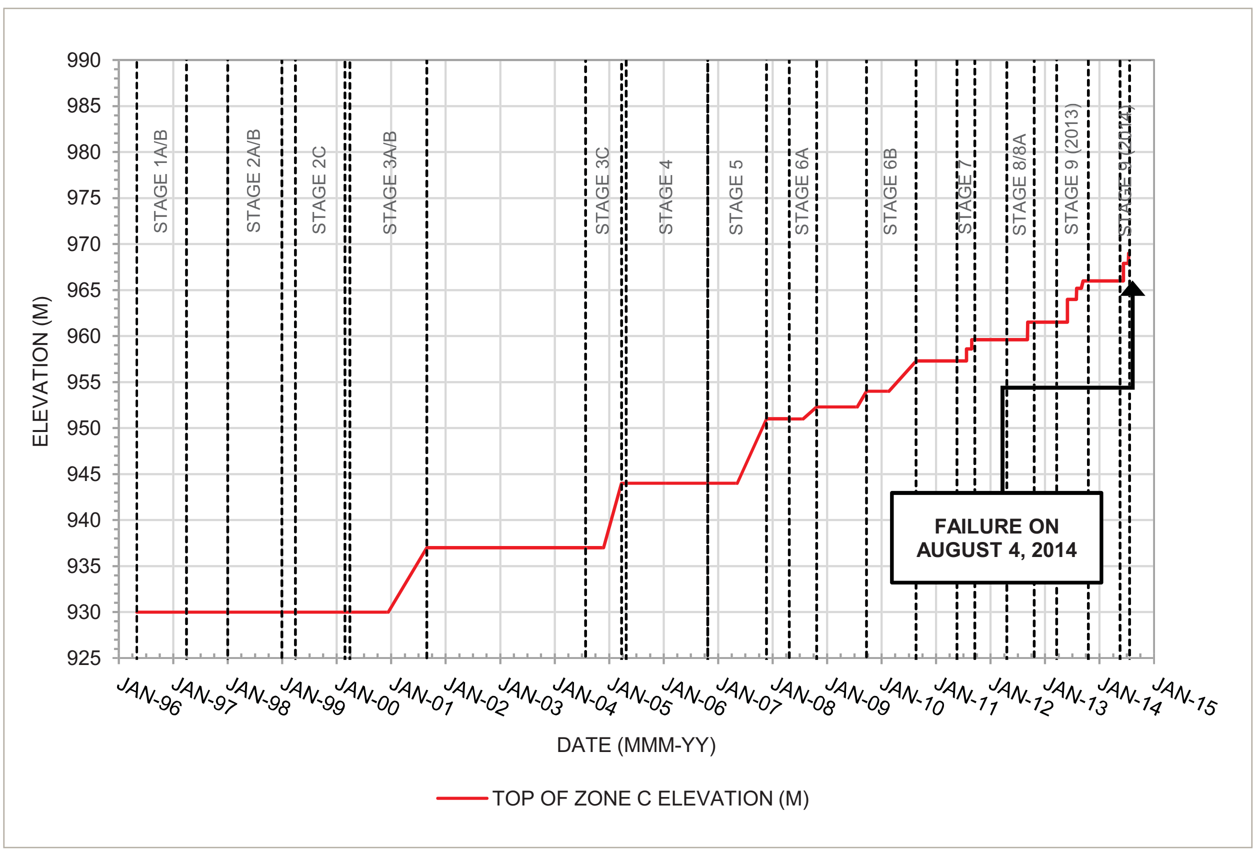
3 | Tailings Storage Facility (TSF)

The history of the construction of the embankments is summarized in **Figure 3.2.3,** which indicates each stage of dam raising up to the occurrence of the breach. The Starter Dam for the embankment was constructed in 1996 to a crest elevation of 927.0 metres (m). The embankments were subsequently raised together in stages as shown. Construction of the Stage 9 raise from approximately elevation (El.) 967.5 m to El. 970.0 m was started at the end of April 2014. Following completion of Stage 9, Stage 10 was planned to raise the crest to El. 972.5 m, which would have provided adequate storage to the end of September 2015. Stage 10 was under review for approval at the time of the breach.

FIGURE 3.2.3: MOUNT POLLEY TSF AND ZONE C (SHELL) TOP ELEVATIONS VERSUS TIME



MCEA Comments Ex. 06

3 | Tailings Storage Facility (TSF)

3.3 POND ELEVATION AT TIME OF BREACH

At the time of the breach, the TSF was permitted under MEM Permit M-200 with approval to raise the crest to El. 970 m. The breach occurred early on August 4, 2014 at a core elevation of El. 969.1 m. Loss of containment was sudden, with no identified precursors. The recorded pond elevation at 6:30 pm on August 3, 2014 was El. 966.83 m.

Loss of containment was sudden, with no identified precursors.

ENDNOTE

1) MP00044

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4 | How Did This Dam Fail?

4.1 CLASSIFICATION OF DAM FAILURE

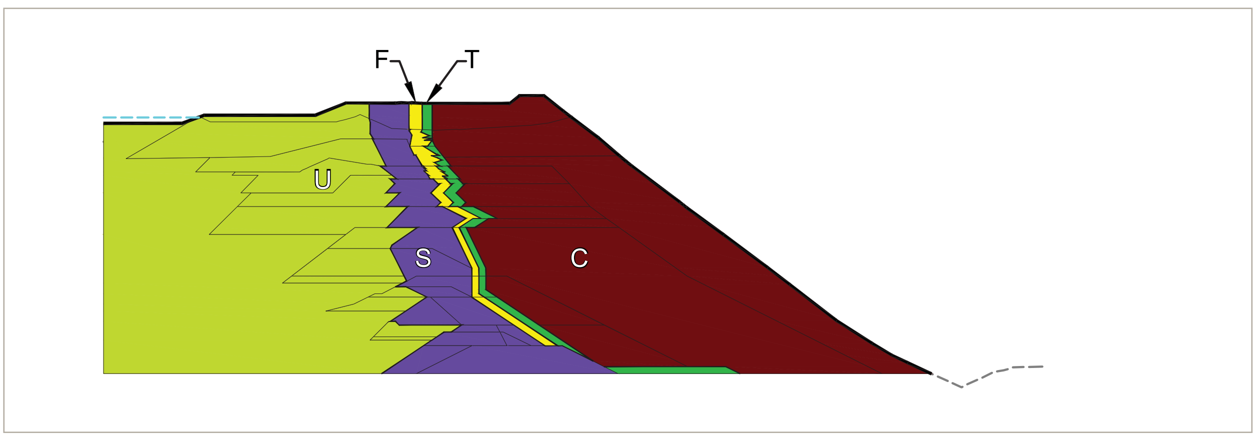
The following directional conventions are adopted throughout, with the dam as the frame of reference:

- Upstream — toward the impoundment interior
- Downstream — away from the impoundment interior
- Right — to the right, looking downstream
- Left — to the left, looking downstream

Figure 4.1.1 shows a simplified cross-section of the dam. It is constructed of both earth and rockfill and would be classified as a zoned earth and rockfill dam. The specific zones are:

U Zone	Upstream fill
C Zone	Rockfill
S Zone	Till core
F Zone	Filter
T Zone	Transition

FIGURE 4.1.1: SIMPLIFIED CROSS-SECTION OF THE MOUNT POLLEY DAM



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The impervious element of the dam is the till core (Zone S), composed of glacial deposits (till) excavated from selected borrow areas. The duty of the rockfill zone (Zone C) is to support the core, without which the core would not be stable. When seepage flows through the core or if it became cracked, its relatively fine-grained material might erode into the rockfill. The duty of the filter (Zone F) and transition (Zone T) is to collect any seepage coming through the core and to prevent fines from migrating out of it. Zone T is a transition to Zone C and it is intended to stop migration of filter material into Zone C. While Zone C is somewhat compacted to improve its density, this is not sufficient to preclude migration of Zone F into Zone C. Hence, a transition Zone T is needed.

Zone U is composed either of tailings or rockfill if tailings beach material cannot be delivered in time. It should be noted that the core of the dam is inclined slightly in an upstream direction. This configuration is known as modified centreline construction. Zone U provides support on the upstream side of the core. It also has other functions such as keeping clear water away from the core. Zone U tailings would tend to migrate into and fill any cracks that might develop in the core, preserving its function.

4.2 POTENTIAL FAILURE MODES

Assessing potential failure modes should be consistent with the characteristics of the breach; it was relatively sudden and with no apparent warning. In addition, by outward comparison, the Perimeter Embankment appears less vulnerable than the Main Embankment design section (**Figure 3.2.1**). The Main Embankment is higher, is designed on the same principles as the Perimeter Embankment, and Zone U has a less developed beach. Moreover, by the time of the breach, small movements had previously been detected in the Main Embankment foundation, and they were being managed by design and construction changes in response to observations. The Panel concluded that, in order to account for such an abrupt event in the Perimeter Embankment, local features likely prevailed.

The Panel concluded that, in order to account for such an abrupt event in the Perimeter Embankment, local features likely prevailed.

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4.3 FOUR CLASSES OF FAILURE MECHANISMS CONSIDERED

Based on the experience of the Panel with both water and tailings dams, the Panel determined that the following four classes of failure mechanisms required consideration:

- Human intervention
- Overtopping
- Piping and cracking
- Foundation failure

Before considering each in turn, it is necessary to understand the timeline of activities at the site prior to the failure.

The timeline constructed from construction and personal reports is presented in **Table 4.3.1**. The breach section extends approximately from survey station (Sta.) 4+200 to 4+300. (refer to **Figure 3.1.1**). Key observations are:

- Last construction ending at 6:30 pm, August 3, 2014
- Site observation indicating no issues at 10:30 pm, August 3, 2014
- Operations at perimeter seepage pond, no issues at 11:45 pm, August 3, 2014
- Perimeter seepage pond water fluctuation beginning at 12:45 am, August 4, 2014
- Power lost (likely due to breach) at 1:15 am, August 4, 2014
- Breach identified at 2:05 am, August 4, 2014

TABLE 4.3.1: TIMELINE OF EVENTS AND ACTIVITIES AT BREACH SECTION AND ADJACENT AREAS

DATE	ACTIVITY	POND El.	SOURCE
7/10	Zone F (filter) trenching from 4+300 to 4+750, El. 967.0	966.55	Construction Daily Report (MPMC)
7/14	Zone S (till) placement from 4+305 to 4+925, El. 968.5	966.55	Construction Daily Report [1]
7/15	Zone S (till) placement from 3+980 to 4+305, El. 968.5 Zone S (till) placement from 4+420 to 4+770, El. 968.8	966.55	Construction Daily Report [1] *
7/16	Zone S (till) placement from 3+990 to 4+768 @ El. 968.5 (completed to PE pipe) [i] Zone S (till) placement from 4+395 to 4+757 @ El. 968.8	966.53	Construction Daily Report [1] *
7/17	Zone S (till) placement from 3+995 to 4+395 @ El. 969.1 (completed to PE pipe)	966.60	Construction Daily Report [1] *
7/24	Zone C (rock) placement from 4+525 to 4+650, El. 968.8	966.68	Construction Daily Report [1] *
	Extreme rainfall, Perimeter overflowing [ii]		TSF Leadhand Report
7/25	Zone C (rock) placement from 4+335 to 4+525, El. 968.8	966.73	Construction Daily Report [1]*
	Perimeter still in "Red" with all pumps running, only 1.7 metres left in perimeter overflow		TSF Leadhand Report [2]

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TABLE 4.3.1: TIMELINE OF EVENTS AND ACTIVITIES AT BREACH SECTION AND ADJACENT AREAS *continued*

DATE	ACTIVITY		POND El.	SOURCE
7/26	Perimeter held @ "6" on scale all day but still overflowing Till pit level went up 20 cm overnight Retrieved piezo below corner 1			TSF Leadhand Report [2]
7/28	Zone C (rock) placement from 4+180 to 4+335, El. 968.8		966.70	Construction Daily Report [1]*
8/01	Zone C (rock) placement (grading down near Corner 1), El. 969.0 Raising of PE pipe in the C zone		966.80	Construction Daily Report [1]*
	Last placement of Zone C (rock) in breach area with four 733s (60T) and one D-8R			Panel Interview 10/22/14
8/02	Placing Zone C on the PE pipe after raising it		966.82	Construction Daily Report [1]*
8/03	6:30 am to 6:30 pm	Placing C zone on the PE pipe after raising it (completed) Grading C Zone (rock) from corner 5 to the PE pipe that has been recently placed by Peterson	966.83	Construction Daily Report [1]*
	10:30 pm	Good berm, good slope, no visible cracks		Shifter Dump Logbook (Mine Ops), B-crew night shift [3]
	11:00 pm	East Perimeter Pond going to alarm in high level within the hour		Dam Breach Report [4]
	11:30 pm	Second pump (perimeter pond) started, nothing unusual noticed		Dam Breach Report [4]
	11:45 pm	Drove from perimeter pond across dam crest to PE pipe and back to Corner 5 (across breach area), nothing unusual noticed		Panel Interview 10/22/14
	12:00 midnight	Second pump drawing down perimeter pond water level (recollection of control instrumentation)		Panel Interview 10/22/14
8/04	12:15 am	Pond water starts to level out (recollection of control instrumentation)		Panel Interview 10/22/14
	12:45 am	Pond water level starts to slightly increase (recollection of control instrumentation)		Panel Interview 10/22/14
	1:00 am	Pond water level rising sharply (recollection of control instrumentation)		Panel Interview 10/22/14
	1:15 am	Lights went out in electrical shop, mill shut down, pond water level spikes sharply (recollection of control instrumentation)		Dam Breach Report [4] Panel Interview 10/22/14
	2:05 am	Dewatering operator discovers that tailings dam had breached		Dam Breach Report [4]

* PHOTO AVAILABLE IN CONSTRUCTION DAILY REPORT.

[i] "PE PIPE" IS THE RETURN-WATER HDPE LINE FROM THE SEEPAGE RECYCLE PUMP THAT CROSSES THE DAM CREST AT THE LOCATION OF SECTION D, APPROX STA. 3+960 (SEE PHOTO IN CONSTRUCTION DAILY REPORT OF 8/02/14).

[ii] "PERIMETER" REFERS TO PERIMETER SEEPAGE POND. "OVERFLOW" REFERS TO OVERFLOW FROM PERIMETER SEEPAGE POND INTO TILL BORROW PIT (PANEL INTERVIEW, 10/22/14).

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4.3.1 HUMAN INTERVENTION

Human intervention may be accidental, such as discharge from a tailings line eroding the structure in an uncontrolled manner, or wilful destruction. A tailings pipeline on the Perimeter Embankment crest was not in service at the time of the breach, and the Panel has found no other evidence of failure due to human intervention.

4.3.2 OVERTOPPING

The Panel has found no evidence of failure due to overtopping prior to breach development.

Although water management had been challenging in later years, and an episode of overtopping over portions of the Perimeter Embankment had occurred in May 2014, freeboard was being carefully monitored around the time of the breach as a result of prior insistence on the part of the Ministry of Energy and Mines (MEM). The freeboard with respect to the core at the time of the failure was 2.3 metres (m).[5]

The Panel has found no evidence of failure due to overtopping prior to breach development.

4.3.3 PIPING AND CRACKING

Piping and cracking of the core of an earth-rockfill dam can lead to internal erosion and ultimately loss of containment. This is one of the most common causes of failure of earth dams and has been much studied. The failure of the Omai Tailings Dam[6] provides an example of a failure by piping and internal erosion.

The following factors were of concern to the Panel:

1) Modified centreline tailings dams, while within precedent, are disposed to longitudinal cracking.
2) Following Stage 5, the core width was reduced to 5 m, which is thin for the planned hydraulic head; again, this has precedent but requires careful filter and transition design and construction.
3) The filter and transition were particularly thin and required meticulous care to be constructed as intended.
4) Details of filter and transition construction in as-built drawings indicated departure from intended design.[7]
5) Much of the as-placed filter material failed to meet applicable filter criteria and requirements for internal stability of its grading.
6) The core had been overtopped in one location for a brief period in 2014, resulting in softening and enhanced deformability.
7) The core was not contained by the steep rockfill shell in as stiff a manner as might have been possible.
8) A cavity was detected in the core remnant of the left abutment of the breach that was the result of internal erosion, see Appendix C.
9) Observed flow to the seepage collection system exhibited a transient spike on April 22, 2013, of the kind

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sometimes characteristic of internal erosion (see Appendix F for details).

Notwithstanding these concerns, the Panel notes:

1) No abnormal seepage observations were detected except for the spike on April 22, 2013 (see Appendix F).
2) Sonic drillholes were located as close to the abutments of the breach as safely possible. They did not detect any suspicious piping pathways.
3) Excavation of the right abutment of the breach did not find any piping pathways through the core.
4) Grading of samples of filter material recovered from the breach area indicate that internal erosion did not produce large flows or overall loss of core integrity (see Appendix C).

Accordingly, and despite the concerns identified by the Panel, it did not find evidence that the breach was caused by piping and/or cracking resulting in uncontrolled internal erosion.

4.3.4 FOUNDATION FAILURES

Observations from the Surface Investigations provided clear evidence for shearing, bodily lateral displacement, and rotation of the embankment that resulted in the breach.

Observations from the Surface Investigations (see section 5.1 and Appendix C for details) provided clear evidence for shearing, bodily lateral displacement, and rotation of the embankment that resulted in the breach. The Panel concluded that the primary cause of the breach was dislocation of the embankment due to foundation failure. This resulted in loss of containment of both the clear water contained in the tailings storage facility (TSF), and tailings, which flowed out of the breach. Clearly, the foundation has behaved in a weaker manner than anticipated in the design. A major focus of this investigation was therefore to determine the foundation characteristics that account for the observed failure mode and to compare the outcome with the design basis.

A number of circumstances can contribute to such weak behaviour, and all require careful assessment.

It is well-known that glaciated terrain can be exposed to glacial drag forces and leave in the underlying sediments and bedrock, if relatively soft, continuous weak surfaces at the residual strength of the material. The residual strength is the weakest resistance that the material can offer and arises from preferred orientation of platy clay particles. Valley rebound folding and expansion in soft bedrock can also result in weak residual strength materials, but these processes have not acted at the TSF. Examples of large tailings facilities on glacially sheared material at

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low strengths are the Mildred Lake Settling Basin at the Syncrude Canada Ltd. site [8] and the large TSF at the Zelazny Most Copper Mine. [9] In both cases, movements are slow and are managed by adaptive response to observations.

Another source of unanticipated behaviour can be a deposit within the foundation of a dam that exhibits pronounced strain-weakening behaviour; that is, it loses considerable resistance once its peak resistance is attained. This type of behaviour was discovered during the forensic investigation into the Aznalcollar (Los Frailes) Tailings Dam failure in Spain. [10] The movements in that case were sudden, without any observable precursors. The instrumentation at the time was minimal.

A further source of unanticipated behaviour arises when a structure is being built in stages on a soft substrate that is contractant; that is, it tends to contract, or densify. When such a soil is subjected to shearing due to loading that occurs slowly, the resulting volume changes strengthen the soil as it densifies. This is known as drained loading. However, if the contractant soil were to be loaded too quickly for water to be expelled and permit volume change, pore pressures develop that weaken the soil. This is known as undrained loading, and the resistance is less than its drained equivalent.

Undrained response can also be initiated if the soil displays a rapid reduction in resistance as yielding is initiated, even under drained conditions. This has sometimes been called spontaneous liquefaction when flowslides develop. While the Mount Polley failure was not sufficiently mobile to be regarded as a flow, the concept of spontaneous undrained response cannot be disregarded in a broadly based inquiry such as this. The implications for stability of the undrained response of soft, contractant soils to staged construction were presented at length in a classic paper by Ladd [11] and are discussed in a widely accepted graduate-level text by Duncan and Wright. [12] The Kingston fly ash slurry spill is an example of a TSF that failed by this undrained mechanism (http://www.tva.gov/kingston/rca/).

All of the above hypotheses regarding the characteristics of the ground conditions in the breach zone were considered in the Technical Commentary that follows. The Technical Commentary presents the findings arising from both surface and subsurface investigations, laboratory studies, computational analyses, and design, construction, and monitoring reviews, in support of the explanation of the cause of failure.

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ENDNOTES

1) MP10000

2) MP10021

3) MP10022

4) MP10013

5) MP00188

6) Vick, S.G., 1996. The failure of the Omai Tailings Dam. *Geotechnical News*, September, pp. 34-39.

7) MP00044

8) Alencar, J., Morgenstern, N.R. and Chan, D.H., 1994. Analysis of foundation deformations beneath the Syncrude tailings dyke. *Canadian Geotechnical Journal*, Vol. 31, pp. 868-884.

9) Jamiolkowski, M., 2014. Soil mechanisms and the Observational Method: Challenges at the Zelazny Most copper tailings disposal facility. *Geotechnique*, Vol. 64, pp. 590-618.

10) Gens, A. and Alonso, E.E., 2006. Aznalcollar dam failure – Part 2: Stability conditions and failure mechanism. *Geotechnique*, Vol. 56, pp. 185-201.

11) Ladd, C.C., 1991. Stability evaluations during staged construction. *Journal of Geotechnical Engineering*, American Society of Civil Engineers, Vol. 117, pp. 540-615.

12) Duncan, J.M. and Wright, S.G., 2005. *Soil Strength and Slope Stability*. John Wiley and Sons, Inc., New Jersey, 298 p.

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5.1 SURFACE INVESTIGATIONS

Surface investigations of the breach and adjacent areas were conducted to gather evidence about the cause of the failure, to document this evidence, and to provide necessary context for related Panel activities. Appendix C provides a comprehensive account of the surface investigations from which this summary has been compiled. The electronic version of Appendix C also contains a virtual three-dimension (3-D) flyover to help orient the reader to features and interrelationships described here.

5.1.1 DATA COLLECTION

The surface investigations made use of imagery from a variety of sources, field mapping on the ground, and exploratory excavation of key features. Data sources and collection activities included the following:

- Review of pre-failure satellite imagery
- Review of a helicopter video made by the Cariboo Regional District during failure
- Review of post-failure helicopter photos by the Panel and airphoto stereopairs
- Review and preprocessing of Panel ground photos
- Preparation of topographical base maps and cross-sections
- Field mapping of ground features and exposures
- Excavation and logging of exploratory works in a remnant section of the dam core

5.1.2 KEY FEATURES

An oblique view of the breach area looking upstream is provided in **Figure 5.1.1**. This and the following image were obtained from the Cariboo Regional District's video. The video was taken on the morning of August 4, 2014, about 8 hours after breach initiation with breach outflow still in progress. It provides a unique opportunity to observe how many of the key post-breach features were formed.

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FIGURE 5.1.1: VIEW LOOKING UPSTREAM THROUGH THE BREACH (ARROW SHOWS DIRECTION OF OUTFLOW)

The labelled features in **Figure 5.1.1** can be interpreted with reference to the internal zoning of the dam previously provided as **Figure 4.1.1**. On the upstream side of the breach, remnant projections of the dam core (S) can be seen on the left and right abutments. The projection on the right abutment acts like a jetty in directing flow toward the left abutment.

Zone C rockfill (C) is exposed on the left abutment, where it has been eroded by these redirected breach outflows. On the right abutment, the surface of the displaced rockfill (D) was subject to erosional overflow during earlier stages of breach development. It was subsequently protected from erosional undercutting in the main channel by the projecting core remnant and eddies that developed downstream.

The whaleback feature (W) is a linear, uplifted ridge of foundation till that extends across the entire width of the breach. Highly erosion-resistant in both native and compacted forms, the upthrusted till here acts as the control section for breach outflow.

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A reverse-angle perspective looking downstream through the breach is provided in **Figure 5.1.2**. It shows the damage to the upstream side of the dam and the processes that caused it.

FIGURE 5.1.2: VIEW LOOKING DOWNSTREAM SHOWING UPSTREAM SIDE OF DAM AND REMAINING TAILINGS

At this point, two major flow channels have developed within the impoundment that converge at the upstream entry to the breach. In the distance (centre left), one of these can be seen flowing along the left side of the dam. As is did so, it eroded away the supporting tailings. This caused the upstream side of the dam to collapse, leaving the prominent near-vertical face. These structural effects are again best appreciated with reference to **Figure 4.1.1**.

Conditions adjacent to the right side of the dam illustrate how the combined action of fluvial erosion and tailings flowsliding produced similar effects. The active flowslide (B) has left a semicircular headscarp that is progressing back and undermining the Zone U tailings supporting the upstream side of the dam core. Arcuate headscarps of earlier flowslides (A) have captured and concentrated overland flows from surface water remaining in the impoundment. The resulting cascades readily transport flowslide debris, while at the same time causing backward erosion of the headscarps by scour within their terraced plunge pools.

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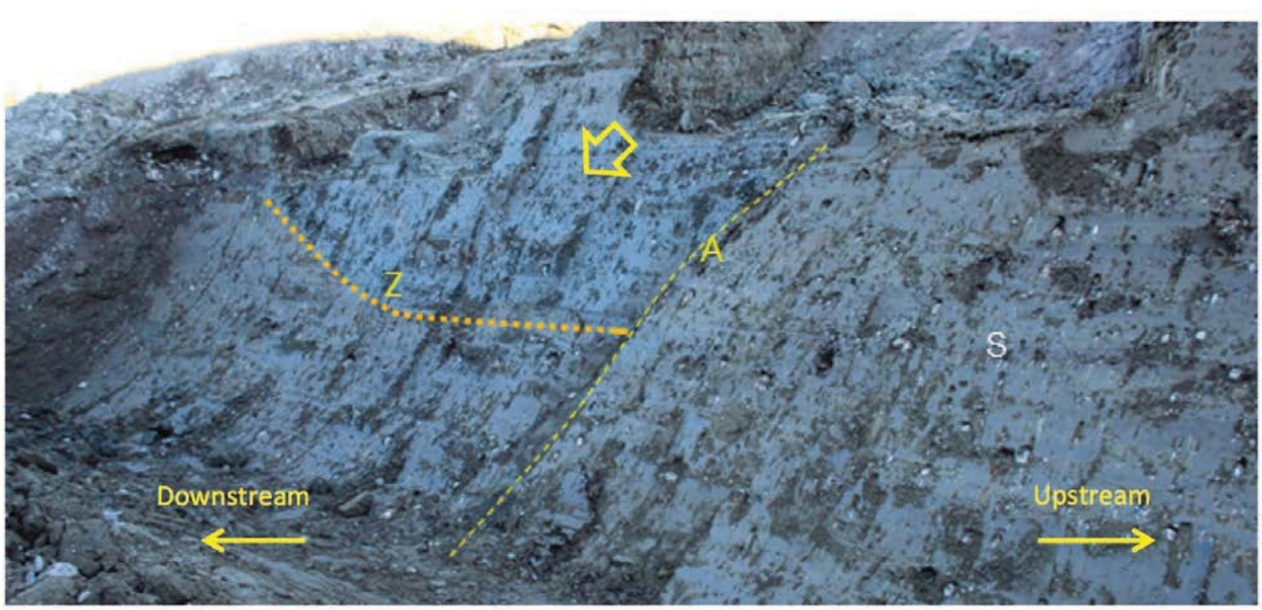
5.1.3 FOUNDATION SLIDING

The surface investigations produced direct evidence for foundation sliding as the initiating mechanism for the breach. This is most clearly demonstrated by a shear surface observed within the remnant core projection on the right abutment at the location shown previously in **Figure 5.1.1**.

Figure 5.1.3 shows an excavated exposure of the shear surface (A) through the Zone S core material (S). The marker bed (Z) was not present on the upstream footwall side (right in photo), indicating at least 3.3 metres (m) of downthrow on the downstream hanging wall. Appendix C, section 3.7, contains more detail on the orientation and configuration of the shear surface.

FIGURE 5.1.3: SHEAR SURFACE THROUGH REMNANT DAM CORE (ARROW INDICATES DIRECTION OF DOWNDROP)



The surface trace and 3-D orientation of the shear are shown on the core remnant in Figure 5.1.4, with a dip angle and direction as indicated.

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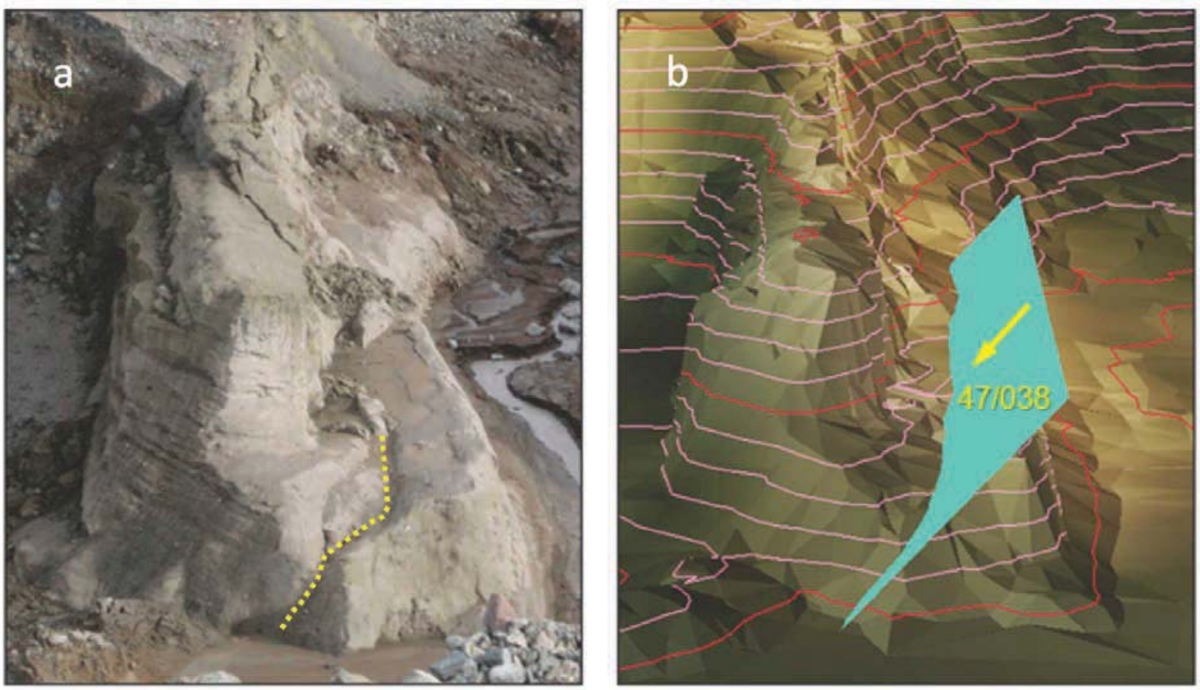
FIGURE 5.1.4: PHOTO (a) AND SURFACE MODEL (b) SHOWING SHEAR SURFACE ORIENTATION



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Other artifacts of foundation sliding are evident elsewhere. As indicated in **Figure 5.1.5**, lift lines in the Zone C rockfill on the left abutment (C) are tilted at an inclination of 7° to 10°, with corollary inclinations on the right abutment of 5° to 14°. Also shown in **Figure 5.1.5** are the Zone S core (S), with a containment dike (K) under construction in the background. Scarps higher on the left abutment were produced by post-sliding downdrop of large slump blocks into the breach due to erosional undercutting at foundation level.

FIGURE 5.1.5: APPARENT BEDDING ROTATION ON LEFT ABUTMENT OF BREACH (SEPT. 4, 2014 PHOTO)

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Figure 5.1.6 shows the right abutment, with the left abutment in the background. Noteworthy features include open cracks (O) and headscarps (H) at higher elevations. At the downstream toe, upthrust of foundation till (L) ranging from 2.8 m to 3.5 m has occurred along an alignment collinear with the whaleback (W).

The mass of rockfill (D) from the upper headscarps to the lower upthrusted till was rotated and displaced by foundation sliding. It was preserved when surface sheet flow was terminated by breach downcutting on the left side. From displacement of surface lineations, lateral (downstream) translation of 11 m occurred in the vicinity of the arrows in the figure.

FIGURE 5.1.6: SLIDING-RELATED FEATURES AT RIGHT ABUTMENT (SEPT. 4, 2014 PHOTO)

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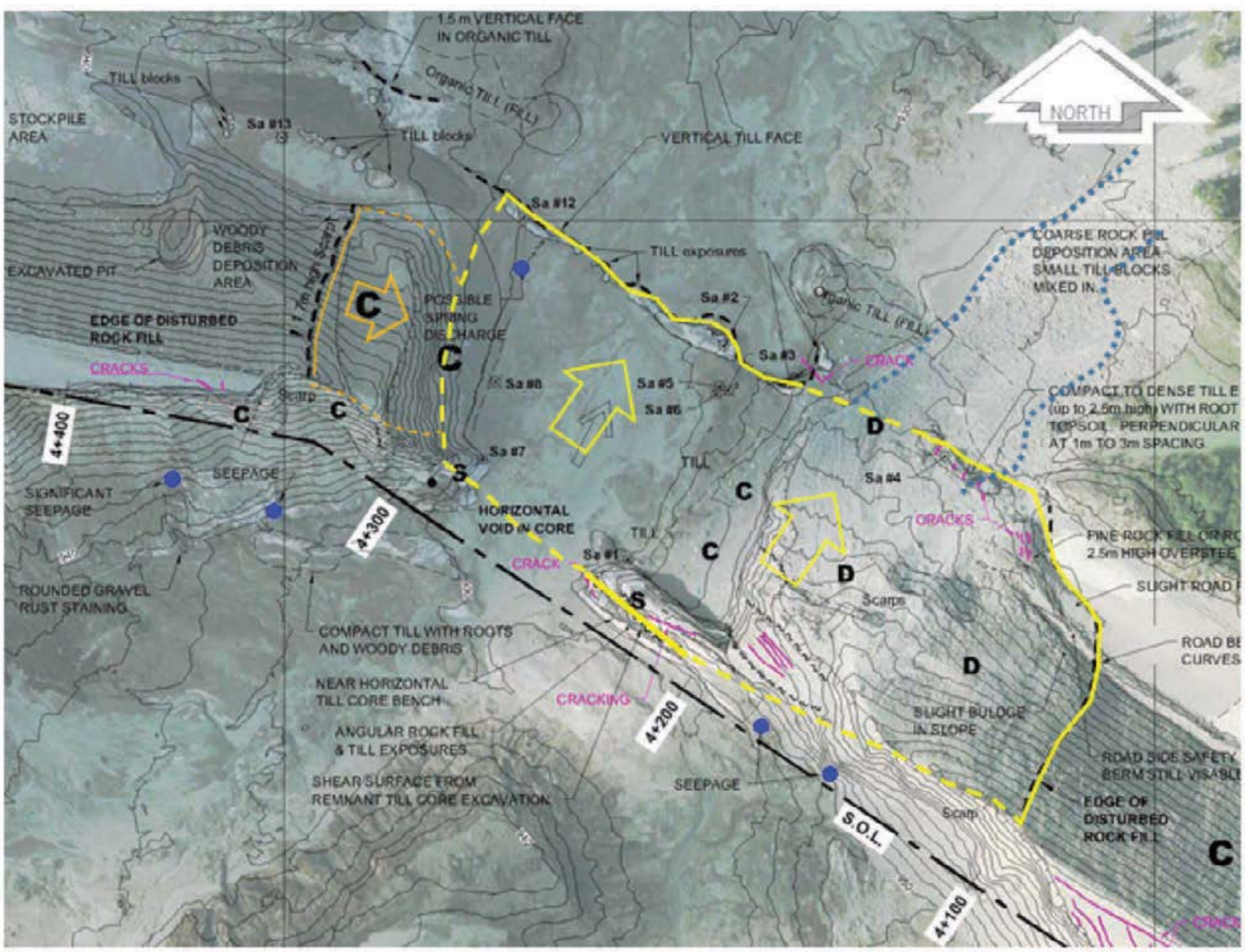
Surface investigations delineate the limits of foundation sliding given in **Figure 5.1.7**, with the solid and dashed yellow lines indicating observed and inferred boundaries, respectively. These are superimposed on the post-failure orthophoto and contours to show the extent of mass movement in relation to the breach. Arrows indicate directions of movement from surface observations.

FIGURE 5.1.7: PLAN SHOWING DIRECTION AND EXTENT OF MASS MOVEMENTS



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Movements are summarized in **Table 5.1.1** below.

TABLE 5.1.1: MEASURED AND INFERRED SLIDE MOVEMENTS

LOCATION	DISPLACEMENTS AND ORIENTATIONS	
DOWNSTREAM TOE	Vertical: Horizontal:	2.8 to 3.5 m upward 11 m downstream
UPSTREAM SHEAR SURFACE	Vertical: Dip:	>3.3 m downward 47 degrees
RIGHT ABUTMENT	Rotation:	5 to 14 degrees
LEFT ABUTMENT	Rotation:	7 to 10 degrees

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5.1.4 INTERNAL EROSION

A void shown in **Figure 5.1.8** was observed on the upstream side of the left abutment, measuring 0.7 m by 0.3 m and extending back 1.1 m into the Zone S core. The angular corners and abrupt transitions at the opening are distinct from the smoother, more rounded surfaces produced by surface erosion at other locations.

FIGURE 5.1.8: VOID ON UPSTREAM SIDE OF LEFT ABUTMENT

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Additionally, Zone F filter material immediately downstream from the core was sampled in the right abutment excavation. Gradation data show that none of these samples met filter criteria that would have enabled them to prevent transport of fines from the till. Together, these factors suggest internal erosion as the likely cause of the left abutment void.

This filter material also has an internally unstable gradation, such that its finer fraction is free to pass through the voids in the coarser fraction under sufficient flow velocity. However, the fact that this finer fraction is still present means that high discharge through the filter, and therefore through the core, did not occur. Moreover, painstaking excavation and thorough logging found no evidence of other such voids in the excavated core. Nor were continuous cracks or softened zones indicative of hydraulic fracturing discovered. These factors suggest that internal erosion was not pervasive over the breach area or sufficiently severe to have compromised core integrity overall.

These factors suggest that internal erosion was not pervasive over the breach area or sufficiently severe to have compromised core integrity overall.

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5.2 SUBSURFACE INVESTIGATIONS

5.2.1 INTRODUCTION

Following the breach of the Mount Polley Perimeter Embankment, site investigation programs were initiated by the Mines Inspector team with Klohn Crippen Berger (KCB) as the geotechnical lead, Mount Polley Mining Corporation (MPMC) with Golder Associates (Golder) as the geotechnical lead, and the Panel, with the support of Thurber Engineering Limited (Thurber). This section summarizes the Panel's program, its outcome, and key findings that affect other aspects of the Panel's activities. Appendix D provides further details of Thurber's work, results and interpretations. Results of the KCB work are available in Appendix B.[1] The Panel relied on factual data collected by both Thurber and KCB, but made its own interpretations of these data.

5.2.2 JOINT SITE INVESTIGATION

In early September 2014, MPMC invited the Panel to participate in a coordination meeting with KCB and Golder to review proposed joint site investigation plans consisting of:

- Geophysics — Direct current resistivity, induced polarity and seismic refraction surveys.
- Drilling and coring — Sonic drilling to allow initial foundation characterization at the dam breach and adjacent areas, and mud rotary drilling and sampling with focus on clays and silts designated the Upper Glaciolacustrine Unit (Upper GLU).
- In situ testing and instrumentation — cone penetration test (CPT) and piezometer and inclinometer installation at selected locations.

Safe work plans had to be implemented to establish access limits with respect to the remaining breach abutments. These limits influenced the locations of the final geophysics lines and the drillhole locations.

KCB field engineers took large numbers of samples from the sonic cores for routine or index testing in the KCB laboratory. Index test results were shared with the Panel. Thurber also collected samples for index testing from a number of locations during the site mapping and other activities for testing in the Thurber laboratories. All index test results are presented in Appendix D, Attachment 7.

Daily reports describing fieldwork progress were shared with all parties. Weekly conference calls, in which the Panel participated, served to further coordinate the fieldwork and provide updates. Adjustments were made to the detailed locations of the geophysics lines as well as drillholes as the program was implemented. **Figure 5.2.1** shows the final locations of the joint site investigation holes in and around the breach area.

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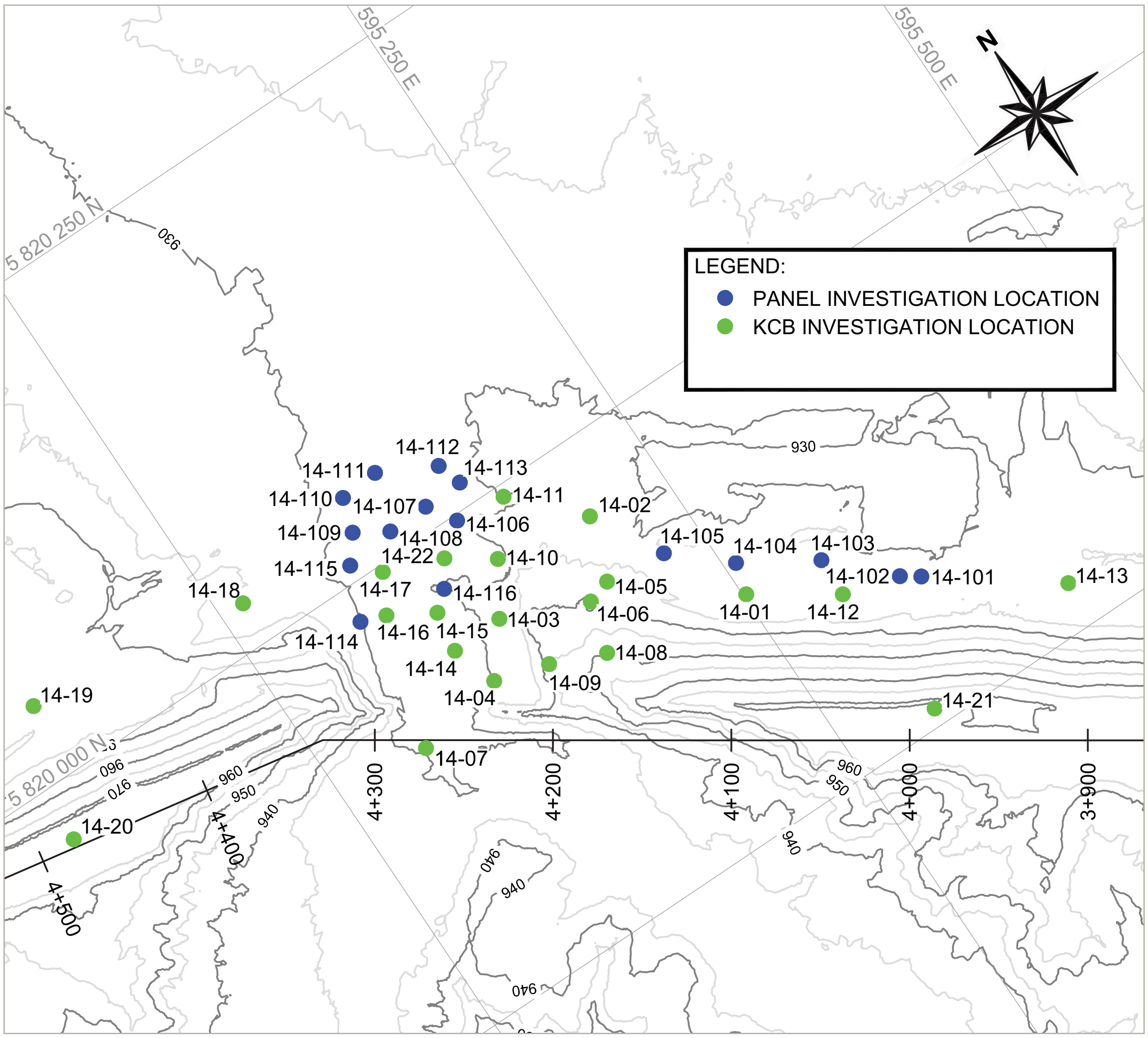
FIGURE 5.2.1: JOINT AND PANEL SITE INVESTIGATION DRILLHOLE LOCATIONS



Thurber's field engineer observed the sonic drilling by KCB and logged all the sonic holes in parallel with the KCB personnel. The Thurber logs of the KCB sonic holes are provided in Appendix D, Attachment 1. Related seismic and resistivity surveys are also described in Appendix D, Attachment 1.

5.2.3 PANEL SITE INVESTIGATION

About half of the locations for the KCB investigation were located in the failed section footprint as well as through the remaining embankment into the underlying foundation. The Panel developed a separate field program that allowed in situ testing and sampling at a larger number of locations where foundation materials had not been preloaded by the embankment. The Panel site investigation consisted of:

- CPT and vane testing to characterize the foundation stratigraphy and to identify sampling locations for advanced shear and consolidation testing (refer to section 5.3 for further details of the laboratory testing).
- Mud rotary drilling to obtain disturbed and undisturbed samples of the Upper GLU and other units for laboratory testing.
- Pressuremeter testing in the till to obtain shear strength and shear modulus values.
- Drilling and sampling using Large Penetration Testing (LPT) in selected areas of foundation till.

Details of drilling methods, in situ testing and related information are included in Appendix D. Excavation, sampling and related laboratory testing are described in Appendix C.

5.2.4 PRE-FAILURE SITE INVESTIGATIONS IN BREACH AREA

Appendix D provides a summary of pre-failure site investigations for the Mount Polley tailings storage facility (TSF). Knight Piésold (KP) performed site investigations in the early to mid-1990s for the design of the facility. Additional site investigations and laboratory testing were done during operations, notably a sonic drilling and instrumentation program implemented by AMEC in 2011. **Figure 5.2.2** shows all the geotechnical drillhole locations for pre-failure investigations in the breach area.

While a large number of locations are shown, many were condemnation holes or shallow test pits of limited usefulness for embankment design purposes. As subsequently discussed, the Panel found the critical soils in the breach area at depths of about 8 m. There are only four locations where the holes were deeper than 8 m and where in situ or laboratory testing was done. None of these locations were in the area where the breach occurred.

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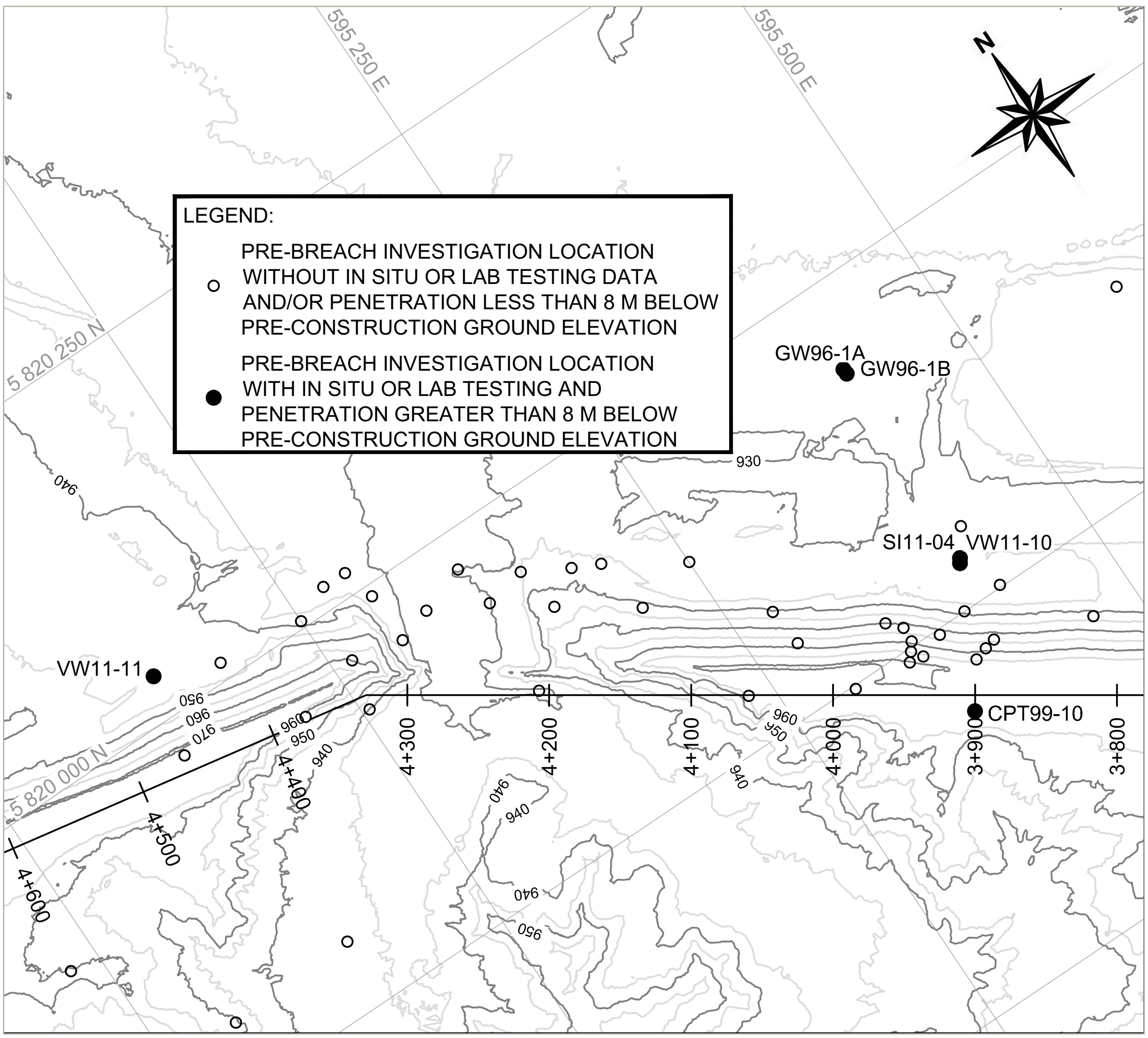
FIGURE 5.2.2 PRE-FAILURE SITE INVESTIGATION DRILLHOLE LOCATIONS IN BREACH AREA



Based on the subsurface investigations and related laboratory data, the Panel derived several key findings that informed its larger efforts. These are discussed in the following sections.

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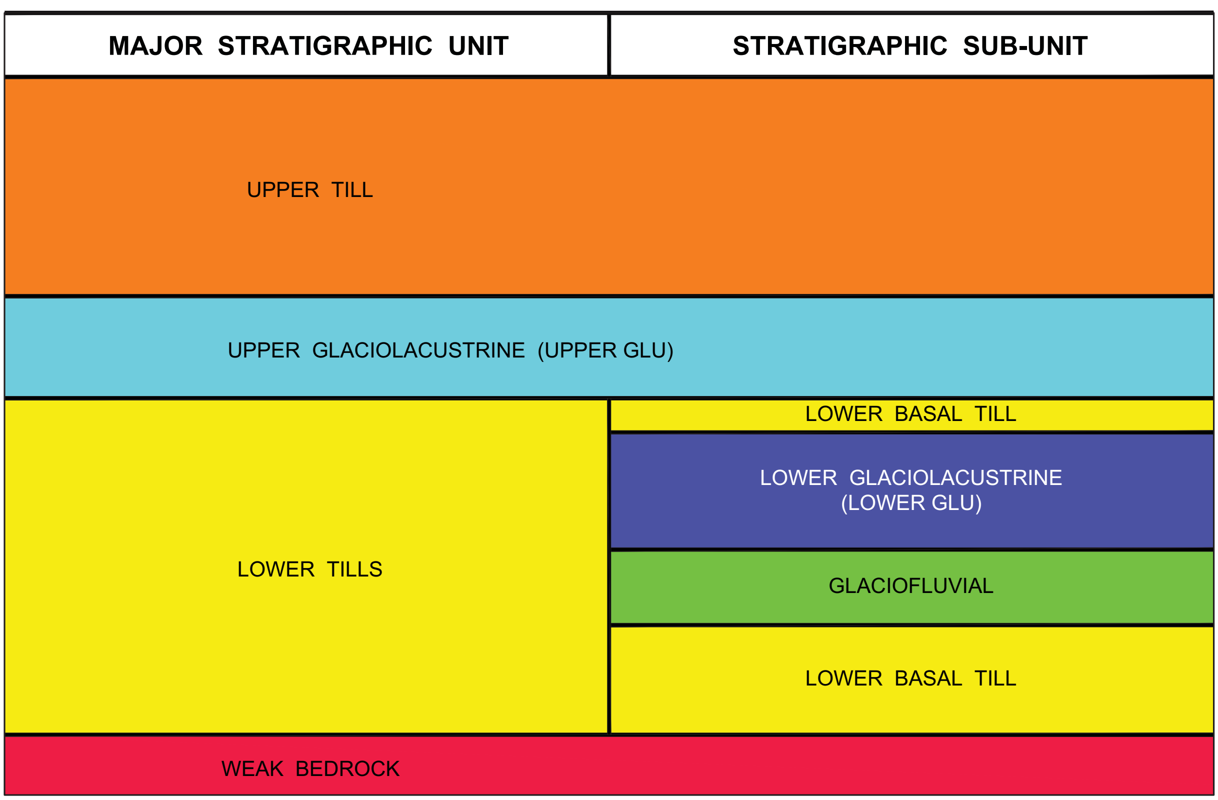
5.2.5 CONTROLLING STRATIGRAPHY

Soils in the breach area are of three main types, all glacially deposited. These include:

- Glaciolacustrine soils (designated GLU) deposited in standing water.
- Glaciofluvial, or streamchannel, deposits.
- Glacial tills produced by glacial transport and reworking.

Appendix D describes the interpreted depositional environment that results in the generalized sequence shown below.

FIGURE 5.2.3: GENERALIZED SOIL STRATIGRAPHY IN BREACH AREA



Of special significance are the two glaciolacustrine units designated Upper and Lower GLU, shown in **Figure 5.2.3**, in turquoise and blue, respectively. Both consist of thinly laminated, or varved, silts and clays, and both classify predominantly as low- to high-plasticity clay (CL to CH). They can be distinguished by differences in their pre-failure water content, CPT tip resistance, and overconsolidation ratio (OCR). Establishing these differences requires looking to areas outside the embankment footprint, or those covered by slide debris, in order to eliminate preloading effects. The resulting comparisons therefore reflect initial pre-construction conditions.

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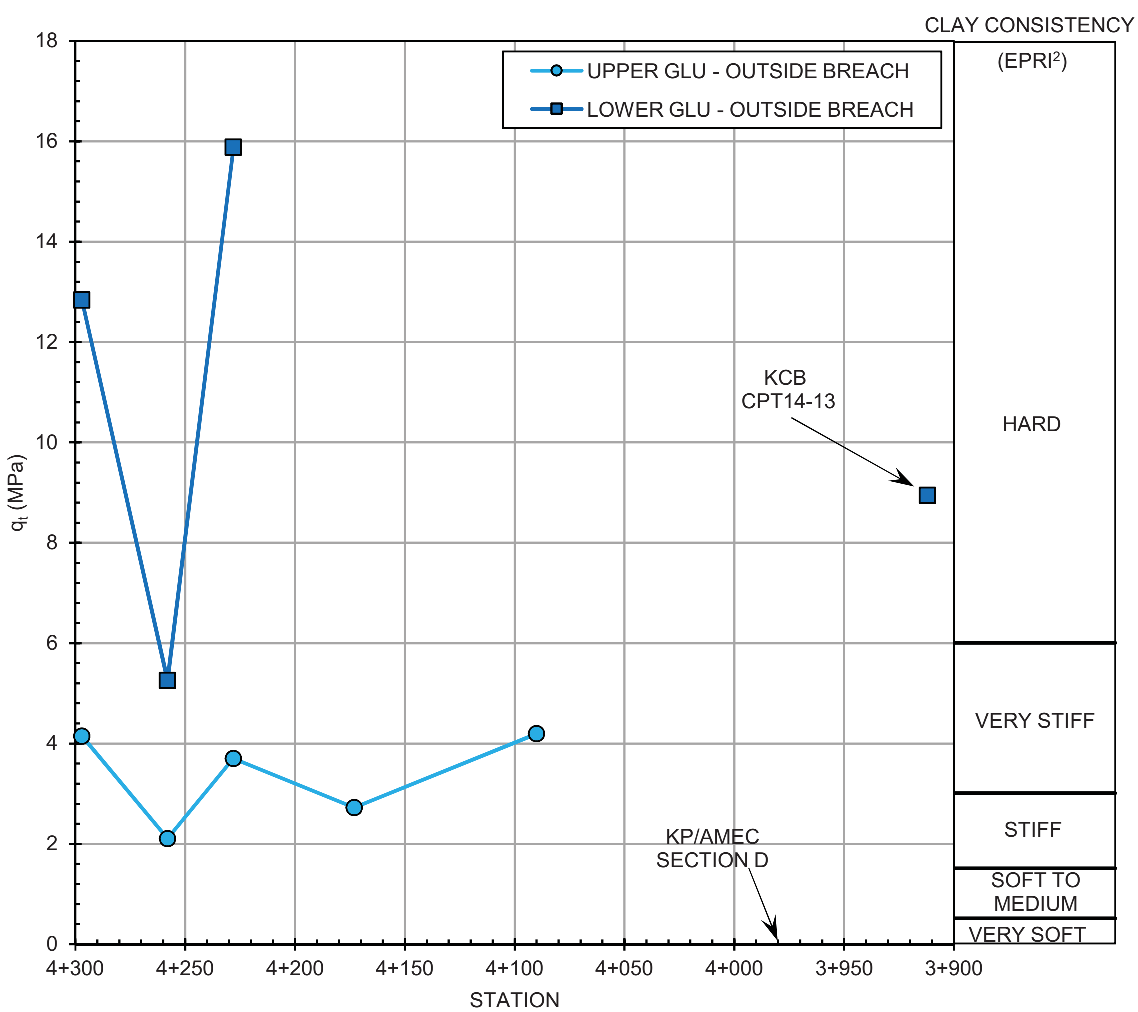
Firstly, the difference in water content is substantial. The average of mean values from individual borings for the Upper GLU is 32%, compared to 24% for the Lower GLU.

Secondly, CPT tip resistance in the two units is distinctly different. **Figure 5.2.4** shows that tip resistance q_t for the Upper GLU is less than one-half of that for the Lower GLU across the breach area. Using q_t as a measure of clay consistency, the Upper GLU classifies as stiff to very stiff, while the Lower GLU classifies as very stiff to hard.

FIGURE 5.2.4: LONGITUDINAL VARIATION IN CPT TIP RESISTANCE



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During their depositional history, glaciolacustrine deposits can experience episodes of drying, freezing, glacial overriding, or other factors that consolidate them to varying degrees. The result is to induce an effective preconsolidation pressure, designated σ'_p, that has a substantial influence on undrained strength properties. This effect can also be expressed as the ratio of the preconsolidation pressure to the effective stress in the ground, termed OCR. In general, higher OCR and higher σ'_p correlate with higher undrained strength.

But when the applied stress increases, for example, due to placement of overlying dam fill, OCR decreases. If the higher stress reaches or exceeds σ'_p, the beneficial effects of preconsolidation no longer pertain, and the clay is said to be normally consolidated with OCR = 1.0. Together, the preconsolidation of a clay, the stresses it experiences, and the changes in these stresses are called its stress history, which has a major influence on its undrained strength.

These factors are reflected in the Upper and Lower GLU, where the initial pre-construction σ'_p and OCR are compiled on a composite plot in **Figure 5.2.5** using CPT data from RCPT14-107 and all available oedometer data.

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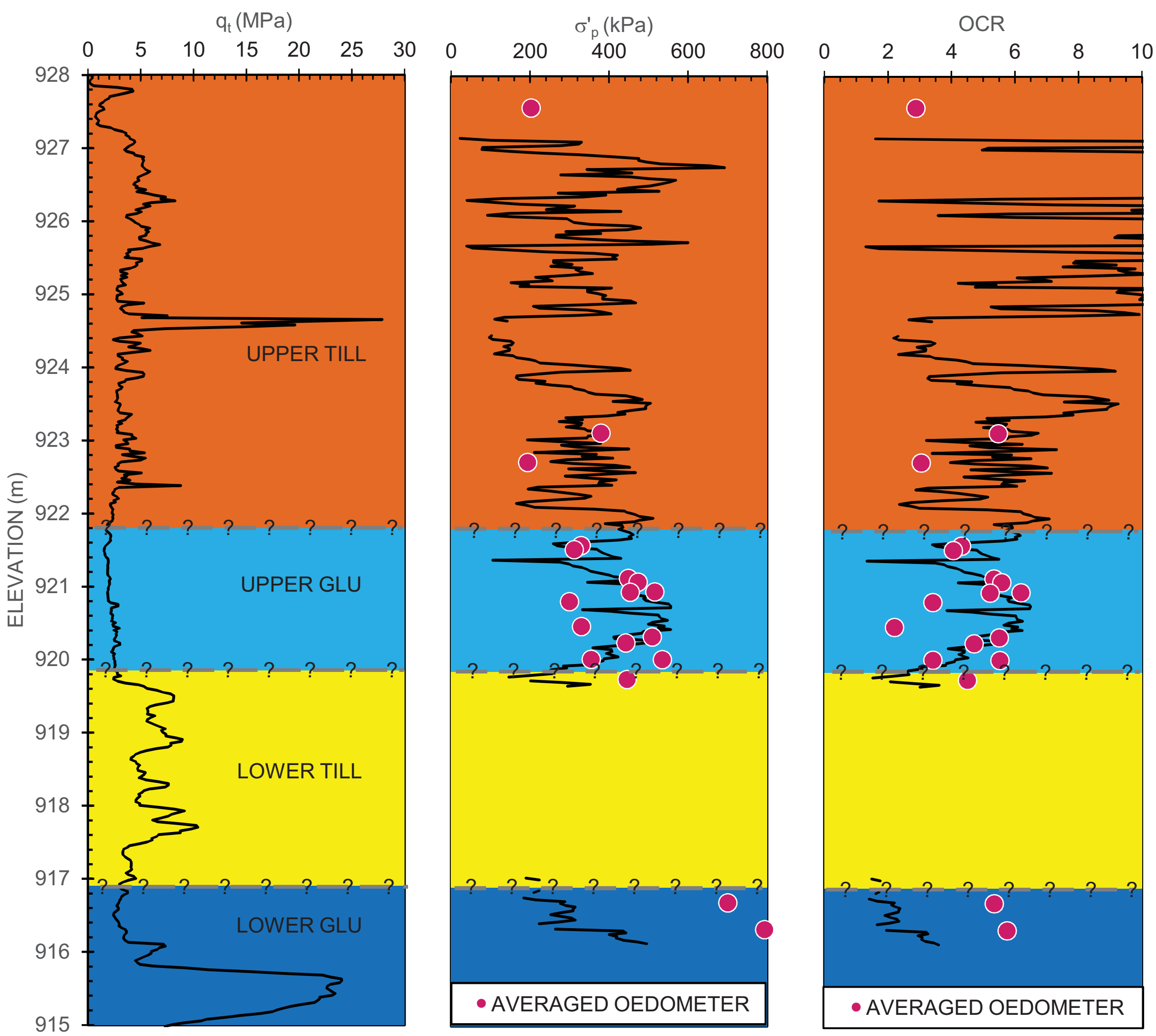
FIGURE 5.2.5: INITIAL (PRE-CONSTRUCTION) STRESS HISTORY



The continuous plots of σ′p and OCR adopt published CPT correlations,[2] while the laboratory-derived data points are taken from Appendix E. From laboratory data, the average σ′p for the Upper and Lower GLU units is 433 kilopascals (kPa) and 748 kPa respectively, with corresponding OCRs of 6.0 and 6.9.

The differences in properties are summarized in **Table 5.2.1.** Drawing D18 in Appendix D provides further details of the properties.

TABLE 5.2.1 PRE-CONSTRUCTION PROPERTIES OF UPPER AND LOWER GLU IN BREACH AREA

STRATIGRAPHIC UNIT	WATER CONTENT, AVG. AND RANGE	CPT TIP RESISTANCE, q_t AVG., RANGE, AND CONSISTENCY	PRECONSOLIDATION PRESSURE, σ'_p AVG. AND RANGE	OVERCONSOLIDATION RATIO, OCR AVG. AND RANGE
Upper GLU	32% (19 — 53)	3.4 MPa (2.1 — 4.2) (stiff to v. stiff)	433 kPa (312 — 535)	6.0 (4.1 — 7.7)
Lower GLU	24% (19 — 29)	11.4 MPa (5.6 — 16) (v. stiff to hard)	748 kPa (701 — 794)	6.9 (6.7 — 7.2)

Taken together, these properties show that the Upper GLU is the weaker of the two units.

5.2.6 EXTENT AND CONTINUITY OF UPPER GLU

Having targeted the Upper GLU as the controlling stratum, it is of further interest to determine its extent. These results are also highly significant. **Figure 5.2.6** demonstrates that the Upper GLU is not pervasive throughout this entire section of the Perimeter Embankment. But is present in the area beneath the footprint between Sta. 4+050 and Sta. 4+300. Moreover, the greatest thickness directly underlies the remaining slide debris on the right side of the breach, thinning toward the left but still extending across the entire width. The maximum thickness also directly underlies the location of the downstream toe of the embankment at the time of failure.

Having targeted the Upper GLU as the controlling stratum, it is of further interest to determine its extent. These results are also highly significant.

There are smaller-scale variations even within this area. **Figure 5.2.6** shows a localized thickening to the east, just at the limit of the slide from **Figure 5.1.7**.

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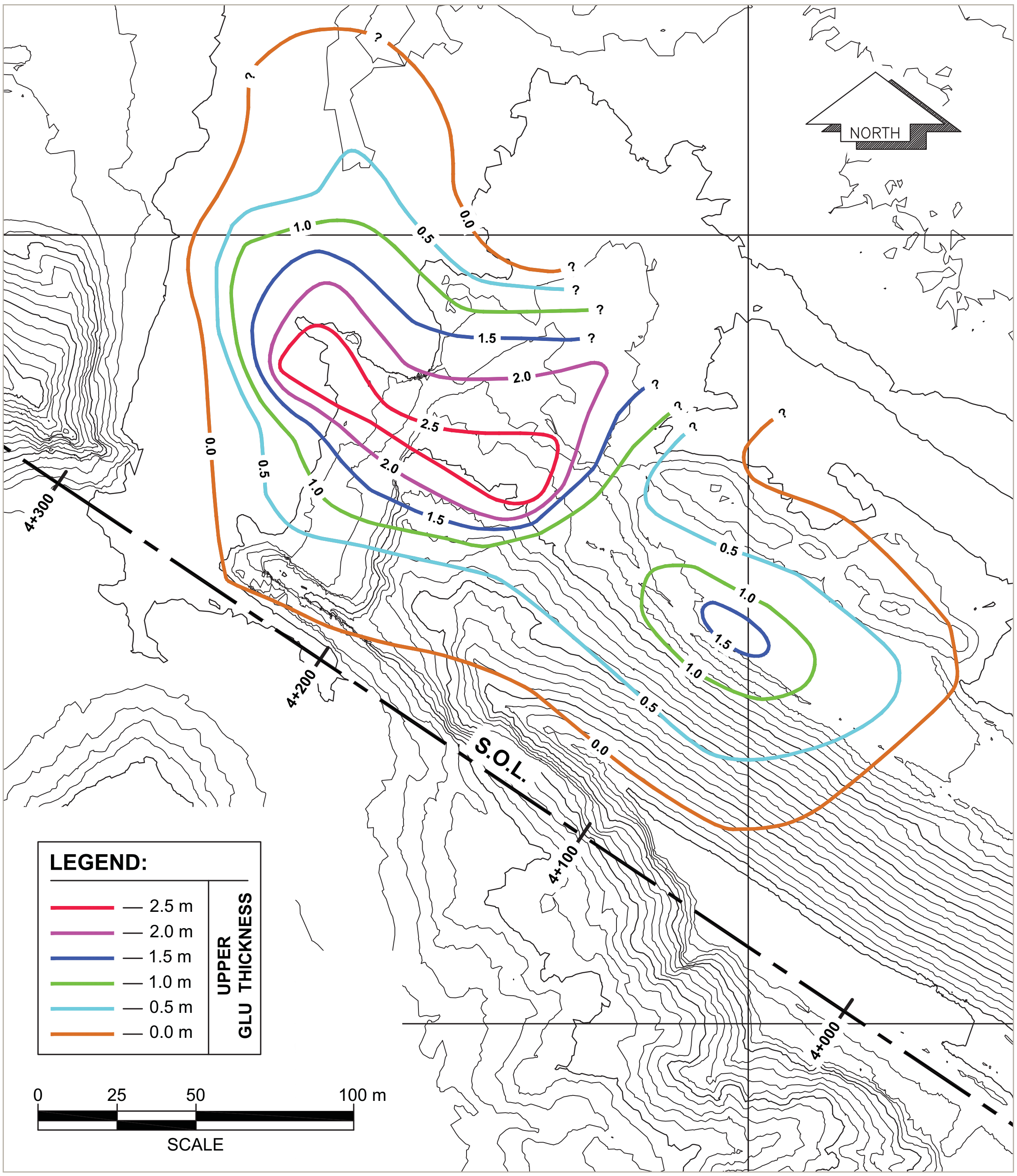
FIGURE 5.2.6: CONTOURS OF UPPER GLU THICKNESS IN BREACH AREA



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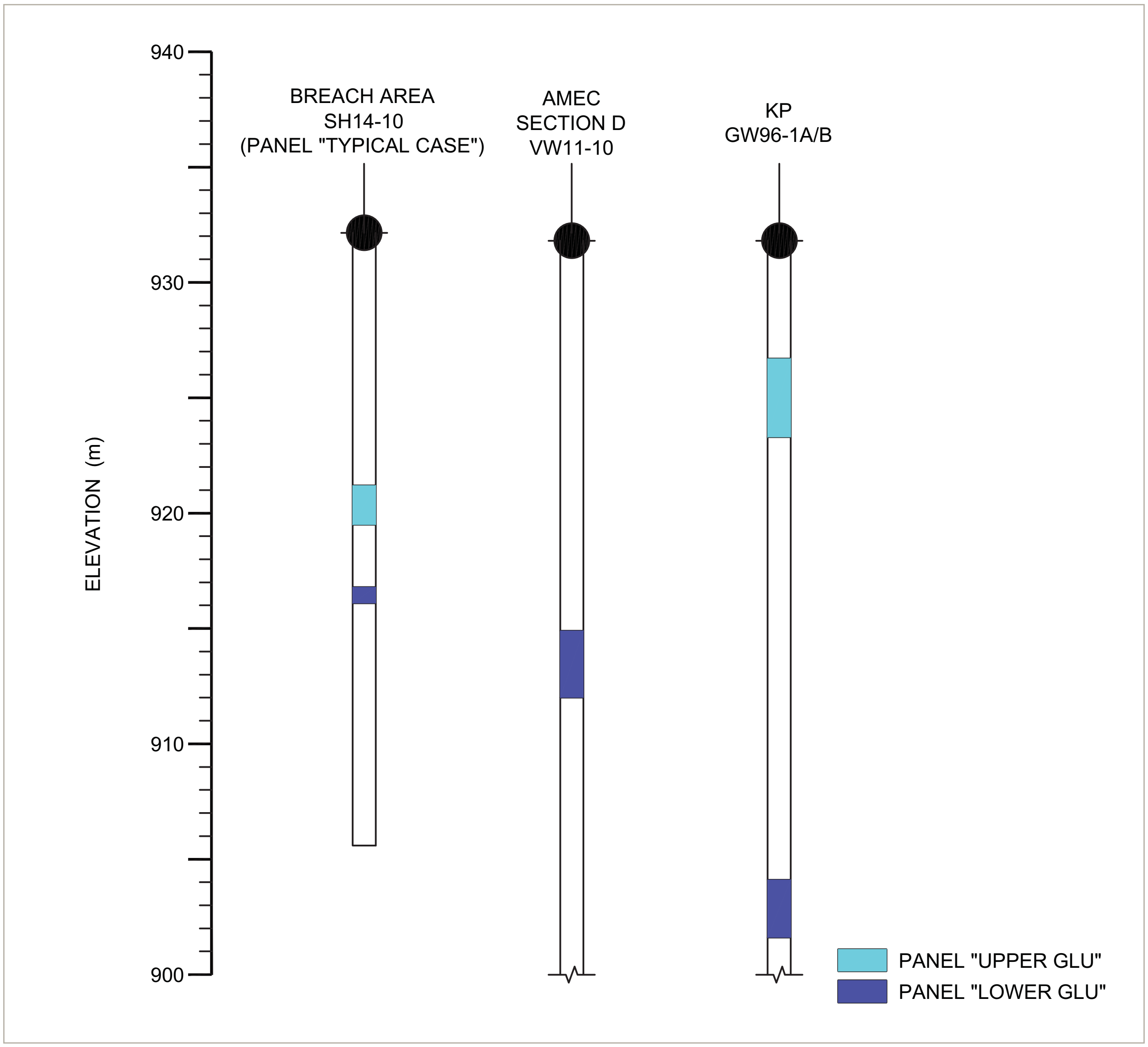
Stratigraphic variations on a larger scale are also apparent from **Figure 5.2.7**, which relates the Upper and Lower GLU units at the breach to glaciolacustrine soils elsewhere at boring locations presented previously in **Figure 5.2.2**. The Upper GLU at the breach shows apparent similarities to glaciolacustrine soils at similar elevation in GW96-1A that would be characterized as soft to medium-stiff according to their standard penetration test (SPT) blow count of 6. On the other hand, the uppermost GLU layer encountered in VW11-10 has an average water content of 23%, which corresponds closely to that of the Lower GLU shown in **Table 5.2.1**. Details are provided in Appendix D.

FIGURE 5.2.7: COMPARISON OF GLU UNITS IN BREACH TO OTHER AREAS



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These illustrations of both small-scale and large-scale variation in stratigraphy and properties of the GLU materials serve to highlight the complexity that their depositional environment produced. This degree of geologic complexity discourages attempts at broader generalization beyond the immediate areas where subsurface data have been obtained.

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5.2.7 LOCATION AND CHARACTERISTICS OF FAILURE SURFACE

Section 5.1 previously identified the entry of the failure surface through the surviving core remnant and into the upper foundation till. The subsurface investigations described here reveal the nature and location of the failure surface at depth.

As will be chronicled in section 5.4, the presence of a glacially pre-sheared surface in the dam foundation posed significant uncertainty throughout the design process. This type of pre-shearing, with the residual strength it produces, was also hypothesized as a potential failure mechanism by the Panel in section 4.3.4. Commensurate effort was devoted to detecting the presence of pre-shearing in foundation soils within the breach.

Pre-shearing in stiff, clayey soils manifests as a thin (a few millimetres to a few centimetres) zone with slickensides—shiny surfaces polished by shearing—on both sides of the zone or within it. These surfaces are continuous and traceable between borings, often along bedding. While detailed logging did show some small, discontinuous slickensided surfaces at random orientations, an expected condition in stiff clays, no continuous surfaces common to multiple borings were found. In this respect, the Panel's investigation at the breach corroborated the more general conclusion of the 2011 site investigation.[3]

Even so, the Upper GLU exhibited other signs of shearing inside but not outside the breach. For example, **Figure 5.2.8** compares the Upper GLU for these two locations.

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FIGURE 5.2.8 UPPER GLU (a) INSIDE THE BREACH, (b) OUTSIDE THE BREACH

The thinly laminated, planar varving outside the breach contrasts sharply with the contorted and folded laminations within it. This is consistent with shearing of the Upper GLU having occurred within the breach.

Additionally, CPT tip resistance q_t in the Upper GLU varies systematically. Drawings D19 and D20 in Appendix D show that average q_t inside the breach is only about one-third to two-thirds of that outside of it (inferred sensitivity of 1.0 to 3.0), reflecting the effects of remoulding attributable to shearing. Hence, both visual inspection and CPT data indicate that the failure produced shearing in the Upper GLU. This, together with its less favourable properties summarized in **Table 5.2.1**, identifies the Upper GLU as the location of the failure surface in the analyses to be presented in section 6.

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5.2.8 COMMENTARY

The key findings of the subsurface investigations with regard to the failure mechanism can be summarized as follows:

- The Upper GLU can be distinguished as a distinct foundation unit based on its water content and other properties.
- The failure occurred within varved silts and clays of the Upper GLU.
- There is no indication of pre-shearing in these or other foundation soils.
- Stratigraphic variability reflects a complex geologic environment and depositional history.

Beyond these immediate findings lie other insights that concern characterization of the GLU during the design process. These are summarized below:

- The discontinuous Upper GLU stratum, the seat of the failure, was infelicitously situated at the worst possible place in the dam foundation.
- The type and extent of pre-failure site investigations were not sufficient to detect this stratum or identify its critical nature.
- The strength behaviour of the GLU was misinterpreted.

The first two of these points are evident from the material presented above. The third requires explanation.

From the outset, the stiffness of GLU materials in the Main Embankment foundation was recognized and attributed to overconsolidation. In response to a review comment by the Ministry of Energy and Mines (MEM), KP obtained samples of GLU materials at the Main Embankment in 1995.[4] Commenting on the characteristics of these soils, KP made the following observations:

Two additional Shelby samples were recently collected (May 16, 1995) during the soil investigation survey. These samples were obtained from the glaciolacustrine sediments and have confirmed that the foundation materials consist of dense, overconsolidated materials. In fact, it was extremely difficult to insert the Shelby tubes in the field and it was not possible to extract the undisturbed samples from the tubes in the laboratory... It is unlikely that any significant pore pressure development will occur in these materials during construction of the embankment.

KP concluded that no significant pore pressures would develop, but did not directly relate this to the effective-stress strength properties its stability analyses adopted. This connection was made explicit much later in AMEC's 2011 Geotechnical Site Investigation.[5]

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Among other things, the AMEC report compiled all available data for Liquidity Index (LI) at the Main Embankment, a laboratory parameter that can be correlated to preconsolidation pressure σ'_p. The report noted that a number of GLU samples had low LI values near zero, some of them even negative, pointing again to the overconsolidated condition of the GLU. Elaborating on the strength interpretation this supported, the report went on to say:

Moreover, for heavily overconsolidated soils with high fines contents (such as the GLU) that will shear in an undrained manner due to low hydraulic conductivity, the undrained shear strength will typically exceed the drained shear strength, owing to negative shear-induced pore pressure.

Thus, undrained strength could be disregarded for the GLU, with drained (effective-stress) strength applicable instead.

Review of the AMEC data calls into question the premise of this conclusion. While most of the LI values were indeed low, fully one-third of them were equal to or greater than 0.5. This means that significant portions of the GLU beneath the Main Embankment were not so heavily overconsolidated. From published correlations,[6] the Panel estimates that σ'_p for these higher LI values ranged from about 250 to 575 kPa, quite similar to the range for the Upper GLU at the breach from **Table 5.2.1**. These σ'_p values correspond to an average OCR of only about 3, given the loading conditions of the catalogued samples, insufficiently high to warrant neglecting undrained strength.

But more than this, the assessment did not account for stress history—how these loading conditions varied at different locations beneath the dam or how they would change over time. Stage 7 of the Main Embankment had just been completed at the time of 2011 site investigation. The Panel estimates that normally consolidated conditions (OCR=1) had already been reached beneath the crest of Stage 2 years before and would continue to propagate outward beneath the slope as the dam grew higher. The key factor that went unrecognized was that undrained strength behaviour would unequivocally control for these normally consolidated conditions.

The key factor that went unrecognized was that undrained strength behaviour would unequivocally control for these normally consolidated conditions.

The same effect, equally unrecognized, would occur at the Perimeter Embankment breach section. Normally consolidated conditions, and the governing undrained strength accompanying them, would first develop beneath the crest during Stage 5 and continue to spread thereafter. This would set the stage for much that followed.

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5.3 ADVANCED LABORATORY STUDIES

5.3.1 INTRODUCTION

A distinction can be made between routine and advanced laboratory studies. Routine laboratory studies are performed as part of the description and classification of materials encountered in a site characterization study. Routine laboratory studies undertaken in this investigation have been reported as part of the description of materials identified in the site characterization investigation (see Appendix D). Advanced laboratory studies are undertaken to aid in the explanation of the physical response of a soil to loading. Important responses are the reduction in volume of a soil when loaded, reflected by consolidation testing, and the ultimate resistance of a soil specimen, as measured by a variety of shear strength tests.

5.3.2 SAMPLING

The joint site investigation, summarized in section 5.2, concentrated on the ground conditions adjacent to the breach that would have been affected by the ground movements. The Panel-directed investigation concentrated on the ground conditions adjacent to the disturbed zone in order to provide the opportunity to inspect soil conditions that would not have been affected by the ground movements. Obtaining undisturbed samples for both inspection and advanced laboratory studies was an integral objective of this investigation.

Obtaining undisturbed samples requires pushing a thin-walled sampler into the ground. Given the conditions encountered, this was not a straightforward exercise. The till contains numerous rocks, and even the fine-grained glaciolacustrine (GLU) deposits contain gravel-sized pieces, most likely deposited during melt of ice rafts.

The inventory of samples that were potentially useful for undisturbed sample testing is tabulated in Appendix E. All samples were subject to scanning at FP Innovations at the University of British Columbia (UBC). This facility can undertake both X-ray and CT scanning on large items. Both digital radiography and CT scans were completed on all sample tubes. Observation of internal disturbance, voids or natural structure aided in the quality control. The horizontal CT scans ultimately proved best to determine complex interlayering and to detect voids. **Figure 5.3.1** displays a sample of till (MR14-104-SA8) that exhibits significant sample disturbance, together with a sample of the GLU (MR14-106E-SA3) that shows internal structure with minimal disturbance. Scans performed on the inventory of samples obtained are included in Appendix E.

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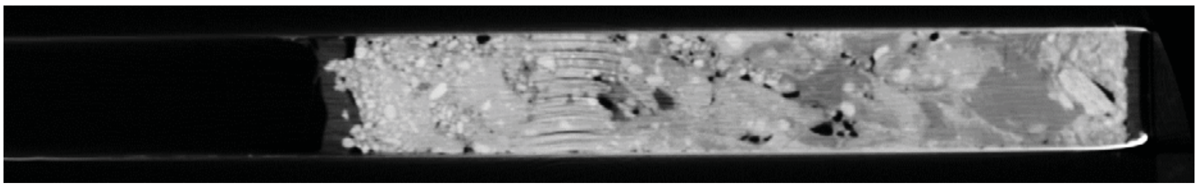
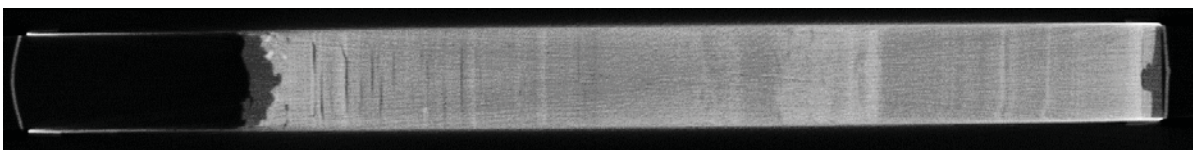
FIGURE 5.3.1: CT SCAN/TILL/GLU



TILL SAMPLE (MR14-104-SA8)

DEPTH: 11.4 TO 12.0 M / EL. 920.3 TO 919.7 M



UPPER GLU SAMPLE (MR14-106E-SA3)

DEPTH: 8.2 TO 8.8 M / EL. 920.5 TO 919.9 M

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5.3.3 OEDOMETER TESTS

Oedometer tests are used to study the reduction in void ratio (porosity), with applied load simulating the construction of the embankment in stages. The change in curvature of the settlement response provides a base for estimating the preconsolidation pressure of the deposit, which is the maximum pressure experienced by the deposit in its geological past. The technique is illustrated in **Figure 5.3.2** for both a till specimen and a GLU specimen. Till is fundamentally less compressible than the GLU, and the technique to estimate preconsolidation stress has greater uncertainty. The data reveal that these deposits are not highly overconsolidated and that the pressure to be applied by the embankment will exceed the preconsolidation pressure, creating normally consolidated conditions. Normally consolidated conditions are conducive for the soil to behave in a contractive manner when subjected to both vertical pressure and shear.

The data reveal that these deposits are not highly overconsolidated and that the pressure to be applied by the embankment will exceed the preconsolidation pressure, creating normally consolidated conditions.

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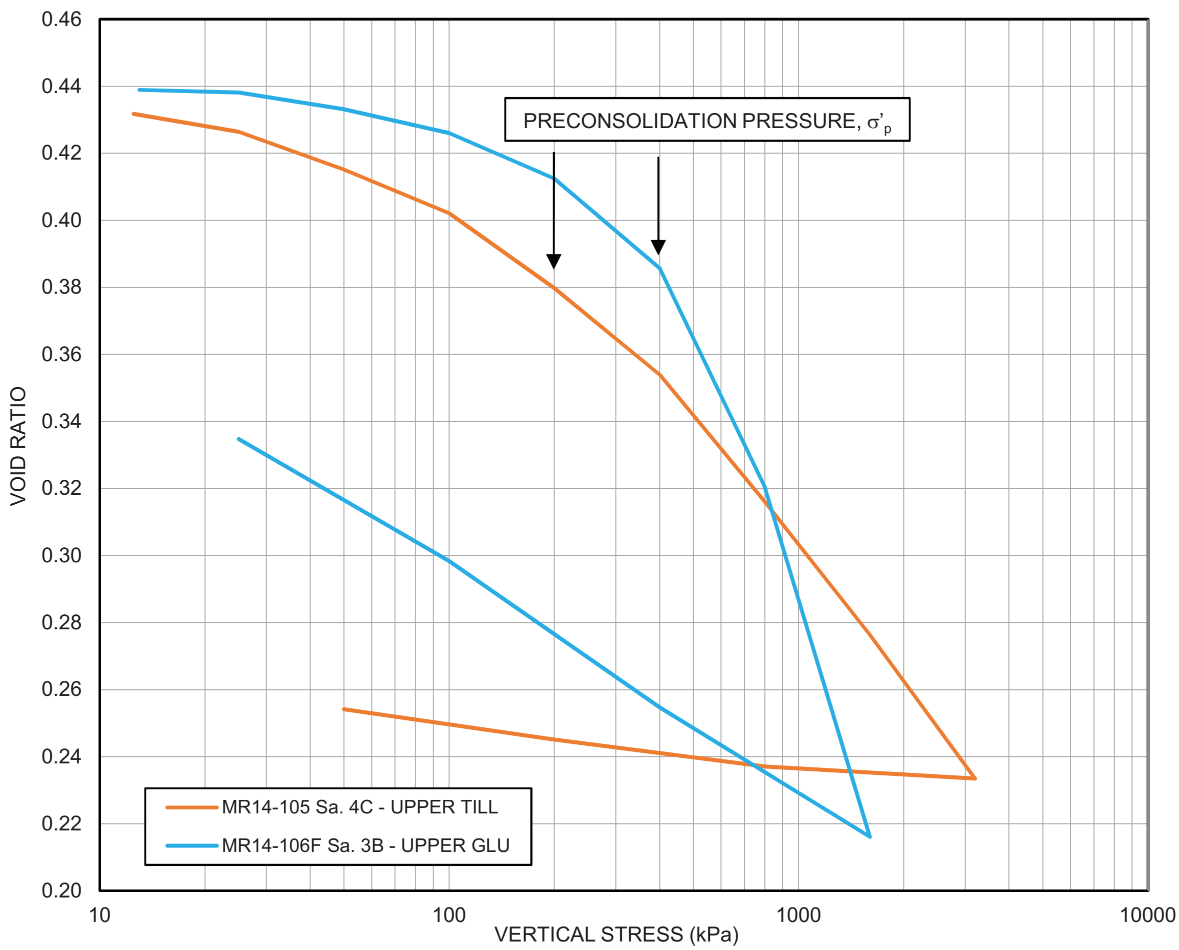
FIGURE 5.3.2: PRECONSOLIDATION STRESS EFFECT FOR BOTH TILL AND GLU



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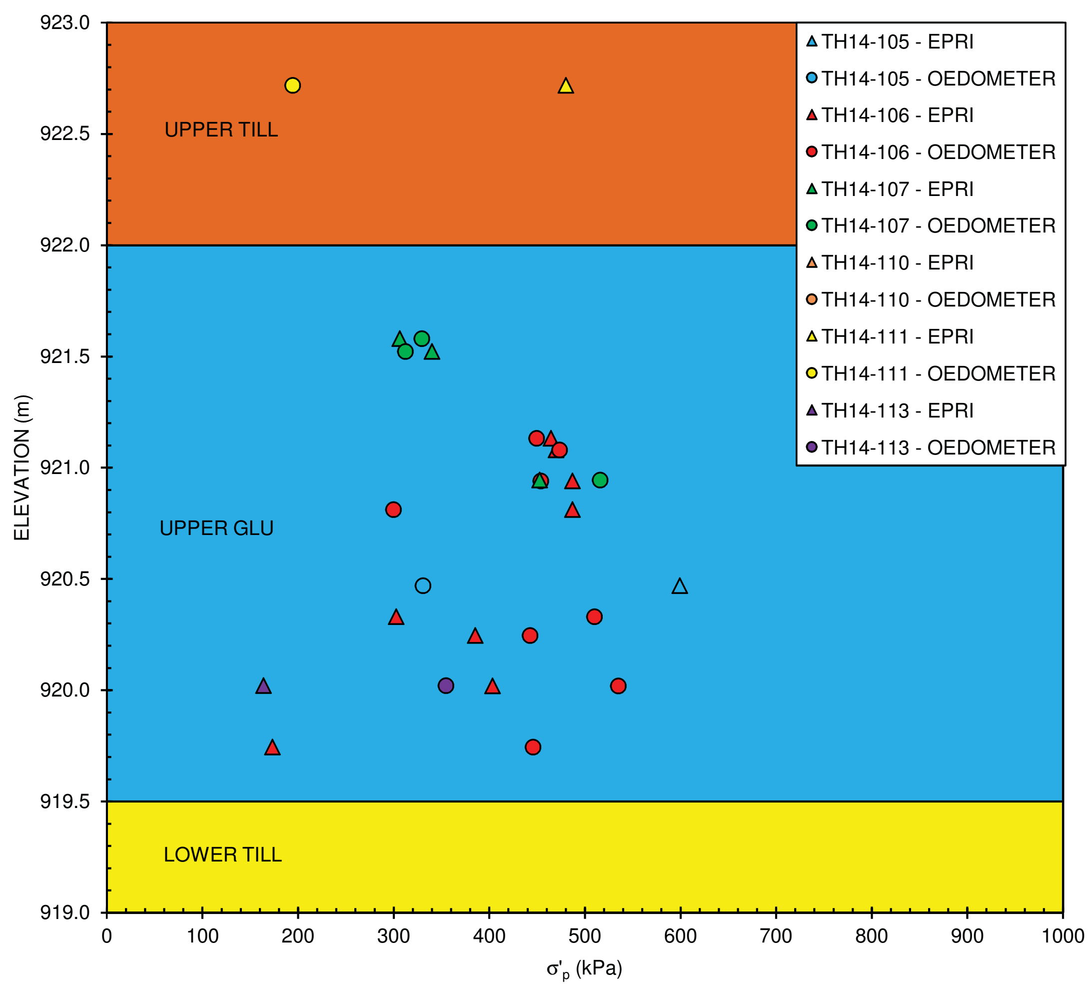
Preconsolidation pressure can also be inferred from the CPT testing conducted as part of the Panel's site investigation. Again, only modest preconsolidation stresses have been determined. A comparison between the results obtained from the field tests with those obtained from the oedometer tests is shown in **Figure 5.3.3**, and the agreement is acceptable. As the embankment was raised to a stress level beyond the preconsolidation stresses, the underlying GLU reverted to normally consolidated behaviour.

As the embankment was raised to a stress level beyond the preconsolidation stresses, the underlying GLU reverted to normally consolidated behaviour.

FIGURE 5.3.3: PRECONSOLIDATION PRESSURE WITH ELEVATION



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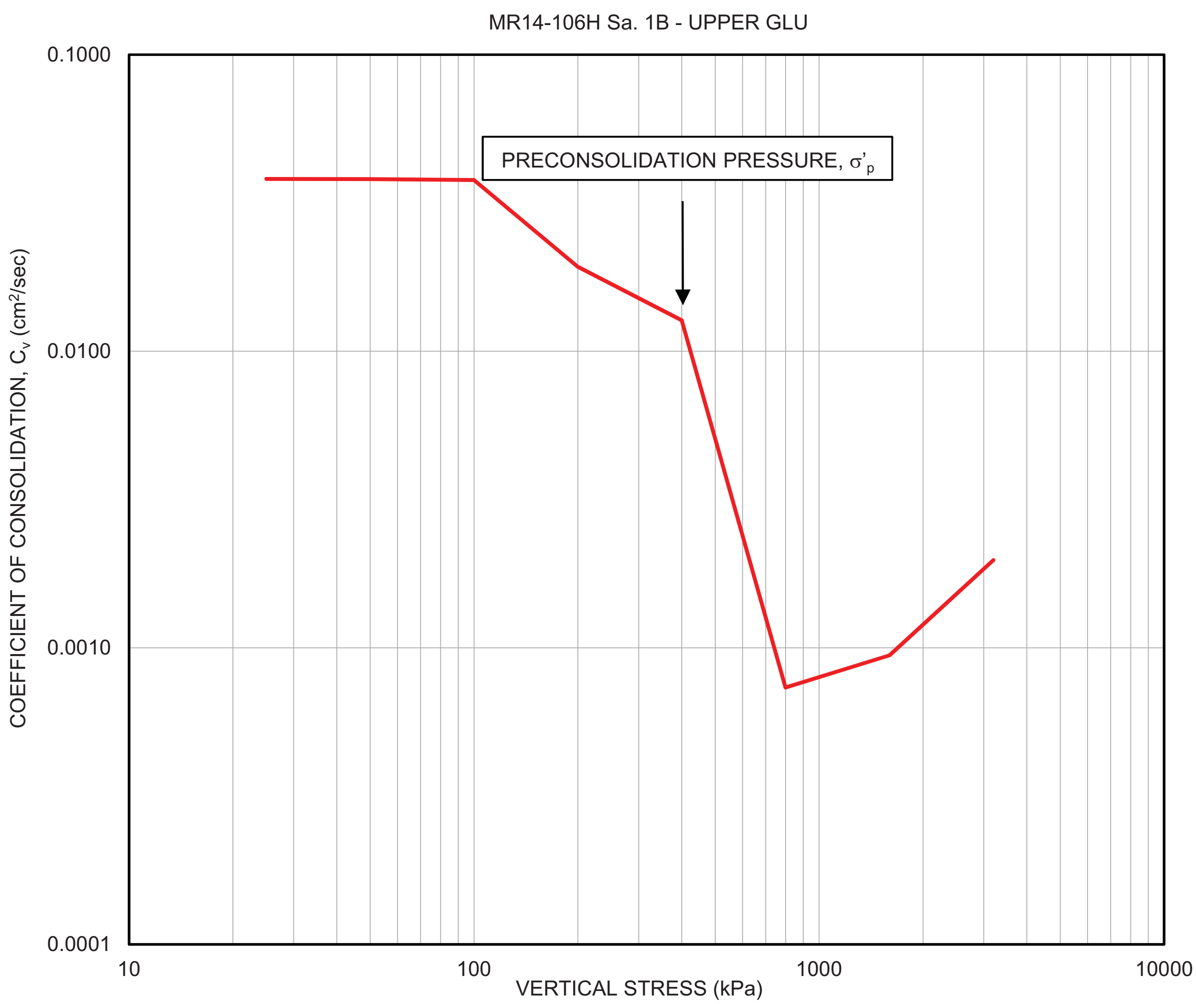
The data from oedometer tests are presented in Appendix E, including information on the coefficient of consolidation that reflects the rate of pore pressure dissipation on loading. **Figure 5.3.4** provides an example of this response. The significant reduction in this value in the GLU at pressures in excess of the preconsolidation stress is noteworthy.

FIGURE 5.3.4: VARIATION OF COEFFICIENT OF CONSOLIDATION WITH APPLIED VERTICAL STRESS



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5.3.4 DIRECT SIMPLE SHEAR (DSS) TESTS

The subsurface characterization has inferred a sub-horizontal shear zone at about El. 920 m. The strength along this zone is best evaluated by DSS tests, which provide the ratio of undrained strength S_u to effective vertical consolidation stress σ_V', or simply the undrained strength ratio. Tests on specimens from the GLU unit that reflect the shear zone at about El. 920–921 m are particularly relevant to the stability analyses that are discussed in section 6. Accordingly, the test program has been extensive, varying initial confining stress and initial shear stress. Testing with an initial shear stress (i.e., stress bias) is intended to explore the influence of a stage-constructed embankment that induces shear stresses in the ground prior to failure.

Another important feature exhibited by this test program on GLU specimens is a decline in resistance following its peak. This is called strain weakening. An example of a test exhibiting strain weakening is shown in **Figure 5.3.5**. As will be discussed, the presence of strain weakening contributes to understanding of the sudden nature of the breach mechanism. **Table 5.3.1** summarizes test characteristics and results from the DSS test program. Complete test results, with a brief description of test methodology, are presented in Appendix E.

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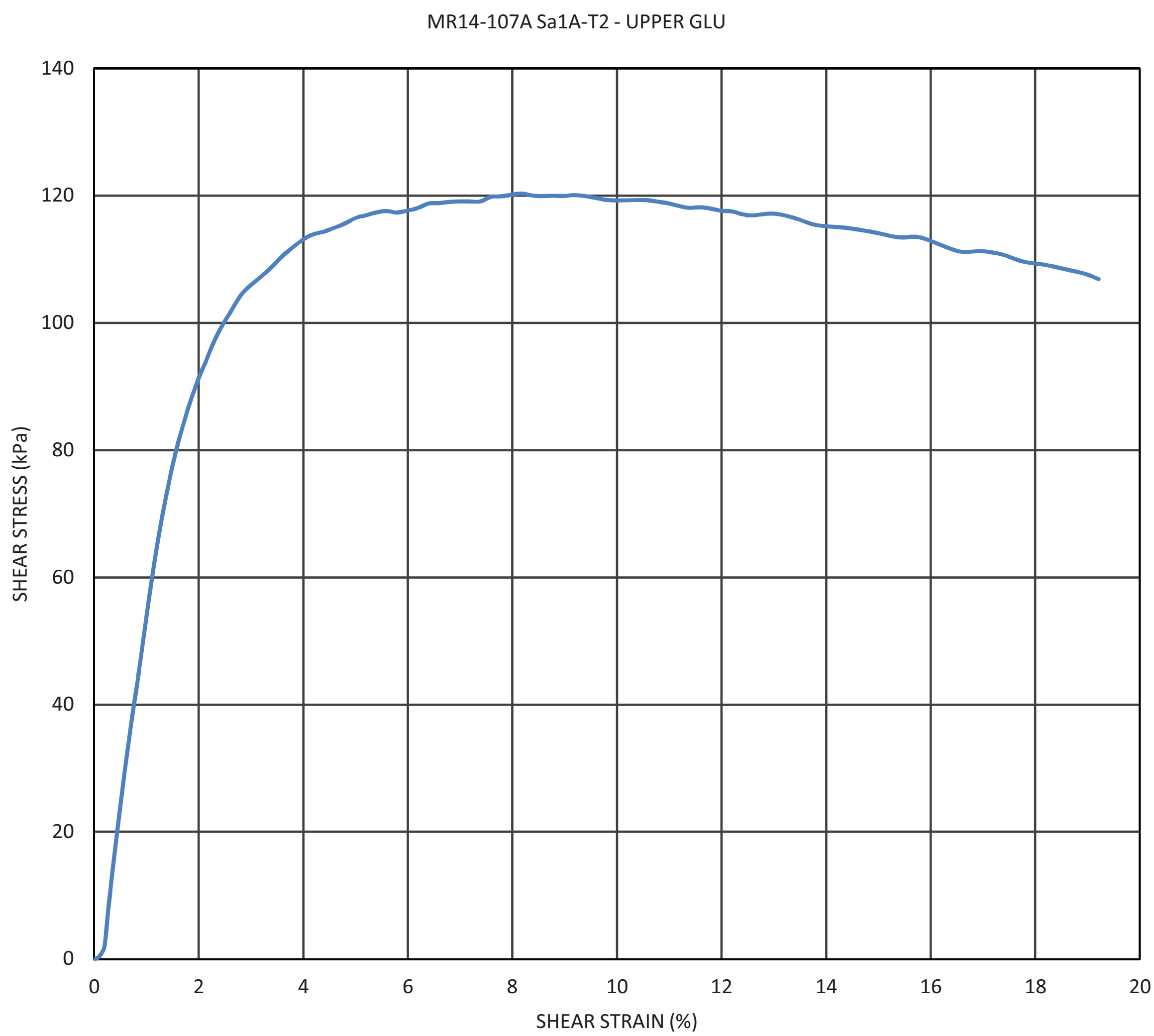
TABLE 5.3.1: SUMMARY OF DIRECT SIMPLE SHEAR (DSS) TEST RESULTS

SAMPLE	ELEVATION (m)	INFERRED PANEL SOIL UNIT	WATER CONTENT (%)	PLASTICITY INDEX	VERTICAL STRESS (kPa)	SHEAR BIAS	PEAK UNDRAINED STRENGTH RATIO
14-106A Sa1C-T1		UPPER GLU	43	31	600	0%	0.22
14-106A Sa1C-T2	921.1	UPPER GLU	37	31	600	10%	0.23
14-106A Sa1C-T3	921.1	UPPER GLU	38	31	600	20%	0.26
14-106A Sa1C-T4	921.1	UPPER GLU	33	31	300	20%	0.28
14-106C Sa1B-T1	921.2	UPPER GLU	44	33	600	30%	N/A
14-106C Sa1B-T2	921.2	UPPER GLU	43	33	600	25%	0.27
14-106C Sa1B-T3	921.2	UPPER GLU	39	33	600	10%	0.21
14-106G SA2B-T1	920.9	UPPER GLU	44	21	600	10%	0.21
14-106G SaB-T2	920.6	UPPER GLU	38	21	600	20%	0.26
14-107 Sa6C-T1	921.5	UPPER GLU	44	23	300	10%	0.27
14-107 Sa6C-T2	921.5	UPPER GLU	43	23	300	10%	0.25
14-107A Sa1A-T1	920.9	UPPER GLU	43	34	600	0%	0.21
14-107A Sa1A-T2	921.0	UPPER GLU	42	34	600	0%	0.20
14-107A Sa7	916.3	LOWER GLU	26	15	600	0%	0.30
14-109 Sa6B	916.6	LOWER GLU	22	13	300	10%	0.42
14-110 Sa6C	916.3	LOWER GLU	21	15	300	10%	0.27
14-113 Sa4B	922.9	UPPER TILL	13	7	300	10%	0.43

Average Peak Undrained Strength Ratio in Upper GLU	(no shear bias)	0.21
	(10% shear bias)	0.23
	(≥ = 20% shear bias)	0.27
	overall	0.24

FIGURE 5.3.5: DSS TEST: GLU WITH STRAIN WEAKENING



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5.3.5 TRIAXIAL COMPRESSION TESTS

Triaxial compression tests with pore pressure measurements have also been conducted, in part for the record, and in part for use in stability analysis. The triaxial test data on till, together with the results from in situ pressuremeter tests, were used to inform the judgment of the Panel on an appropriate value to be used in the stability analyses.

Details of all triaxial tests are tabulated and presented in Appendix E.

5.3.6 DIRECT SHEAR TEST

While not used directly on any of its analyses, the Panel undertook a direct shear test on a pre-cut specimen of the GLU. This was primarily for the record, but afforded an opportunity for comparison with magnitudes adopted in some phases of the design. The Panel's measured residual strength of 16 degrees is at the lower end of the range used by others. The data are found in Appendix E.

5.3.7 DESIGN BASIS TESTING

The design of the Perimeter Embankment did not rely on any deep sampling of its foundation. Hence, no undisturbed samples were obtained, and no advanced laboratory tests were performed to provide data for purposes of comparison.

5.3.8 JOINT INVESTIGATION

Advanced laboratory studies were also performed on samples procured during the joint site investigation. The tests were not performed under the direction of the Panel, but are also included in a separate identifiable section within Appendix E.

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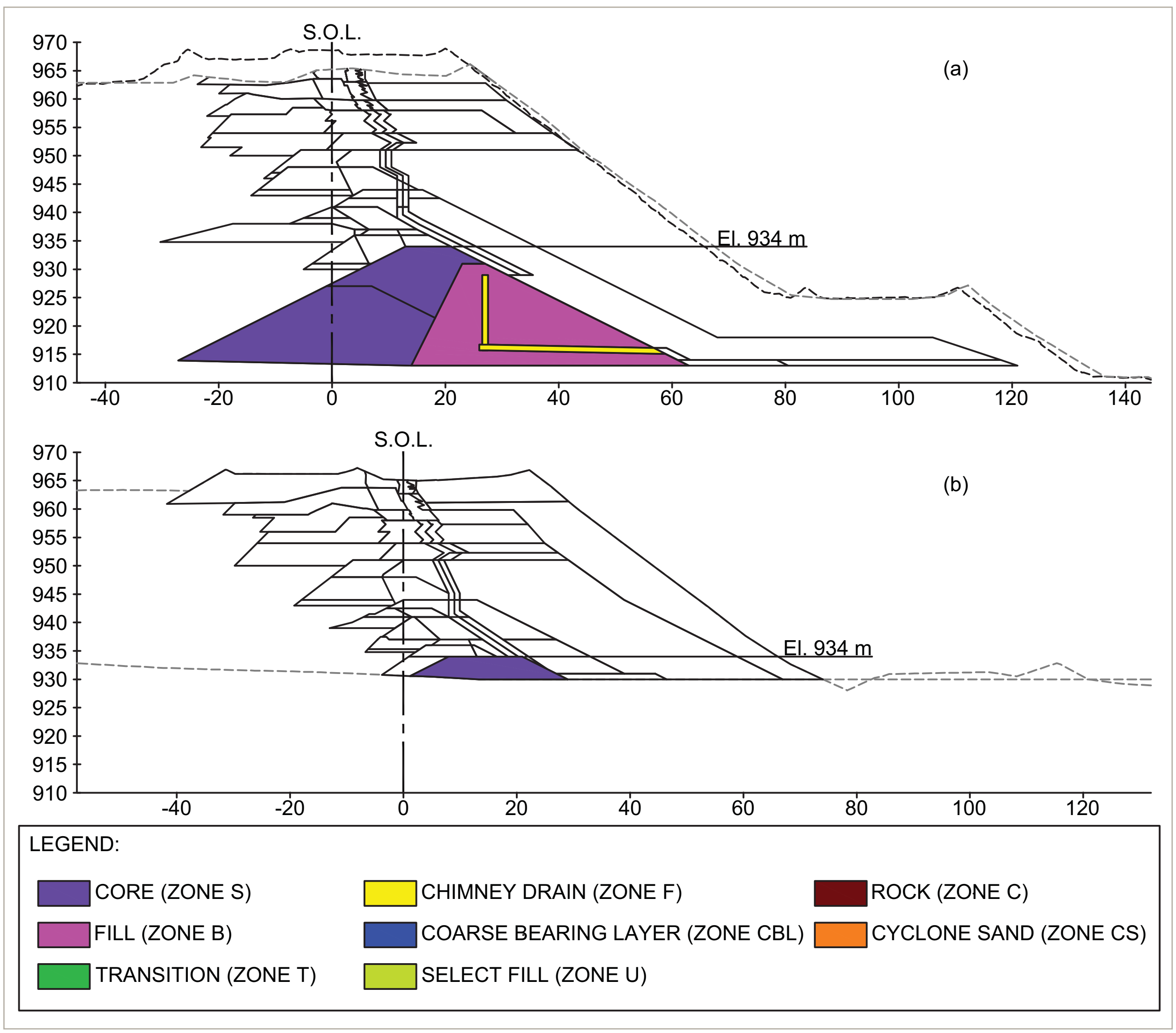
5.4 DESIGN, CONSTRUCTION AND OPERATION

This section describes the historical and sequential development of the Mount Polley Tailings Dam. The Main Embankment is included here along with the Perimeter Embankment to explain salient features and milestones related to design, construction and operation. The dam was developed in stages designated 1 through 9 that are treated in turn in the following discussion. At each stage, as-built cross-sections for the Main Embankment and for the Perimeter Embankment at the breach location are used to portray the dam's progressive expansion.

5.4.1 STAGE 1: 1997 — 1998

FIGURE 5.4.1: STAGE 1 (a) MAIN EMBANKMENT (b) PERIMETER EMBANKMENT



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Design of the Main Embankment in May 1995[7] by KP established the direction for subsequent events. The overall plan incorporated dam raises to El. 960, of which Stage 1 would be the first, as illustrated in **Figure 5.4.1**. With planned raising by the "modified centreline" method,[8] the ultimate dam would rely on deposited tailings to provide structural support for the core. Fill would consist primarily of glacial till borrow soils, with sand tailings obtained by cycloning placed upstream of the core. The setting out line (S.O.L) provided the reference for dimensioning the dam's fill zones and for stationing along its length.

Seismic criteria were based on a "low" consequence classification as defined by the Canadian Dam Association (CDA). The minimum factor of safety (FS) for the downstream dam slope was taken as 1.3 during impoundment operation and 1.5 at closure, design criteria that remained in effect for all subsequent raises. For the 2H:1V Stage 1 downstream slope, an effective-stress analysis (ESA) showed FS = 1.43, thereby satisfying the operational requirement.

Glaciolacustrine (GLU) fine sands, silts and clays were recognized from the outset to be present in the Main Embankment foundation. They were described as "typically dense to very dense and have been heavily overconsolidated by glaciers," with two samples confirming that they consisted of "stiff, overconsolidated materials."[9] In a crucial interpretation of their behaviour that would be relied upon throughout, a Ministry of Energy and Mines (MEM) query prompted KP to respond that "it is unlikely that any significant pore pressure development will occur in these materials during construction of the embankment."[10]

Refinement of the Stage 1 design and its component Stages 1A and 1B continued as construction approached. With encouragement from MEM's regulatory precursor, the Ministry of Employment and Investment, a narrow (1 m wide) chimney drain was added,[11] and four relief wells were installed in the foundation of the Main Embankment to reduce uplift pressures acting on GLU layers.[12]

In addition, the detailed Stage 1 design included a small dam only a few metres high to close off a topographic depression west of the Main Embankment. Designated the Perimeter Embankment and shown in **Figure 5.4.1(b)**, it would grow with subsequent stages to itself become a substantial structure contiguous with the Main Embankment. It would also host the site of the breach.

Construction of Stage 1 was completed in March 1997. Glacial till (Zone S and Zone B) was sourced from borrow excavations within the impoundment interior. The chimney drain materials (Zone F) were obtained by crushing, as would remain the case for subsequent raises. Both materials were subject to Construction Quality Assurance (CQA) testing. Vibrating wire piezometers were installed in both the embankment fill and foundation, with four of the six foundation instruments indicating elevated pressures. In response, a new operational stability criterion was established—an allowable ESA factor of safety of 1.1 at a trigger (action) level of 6 m of measured pressure head above the ground surface.[13, 14]

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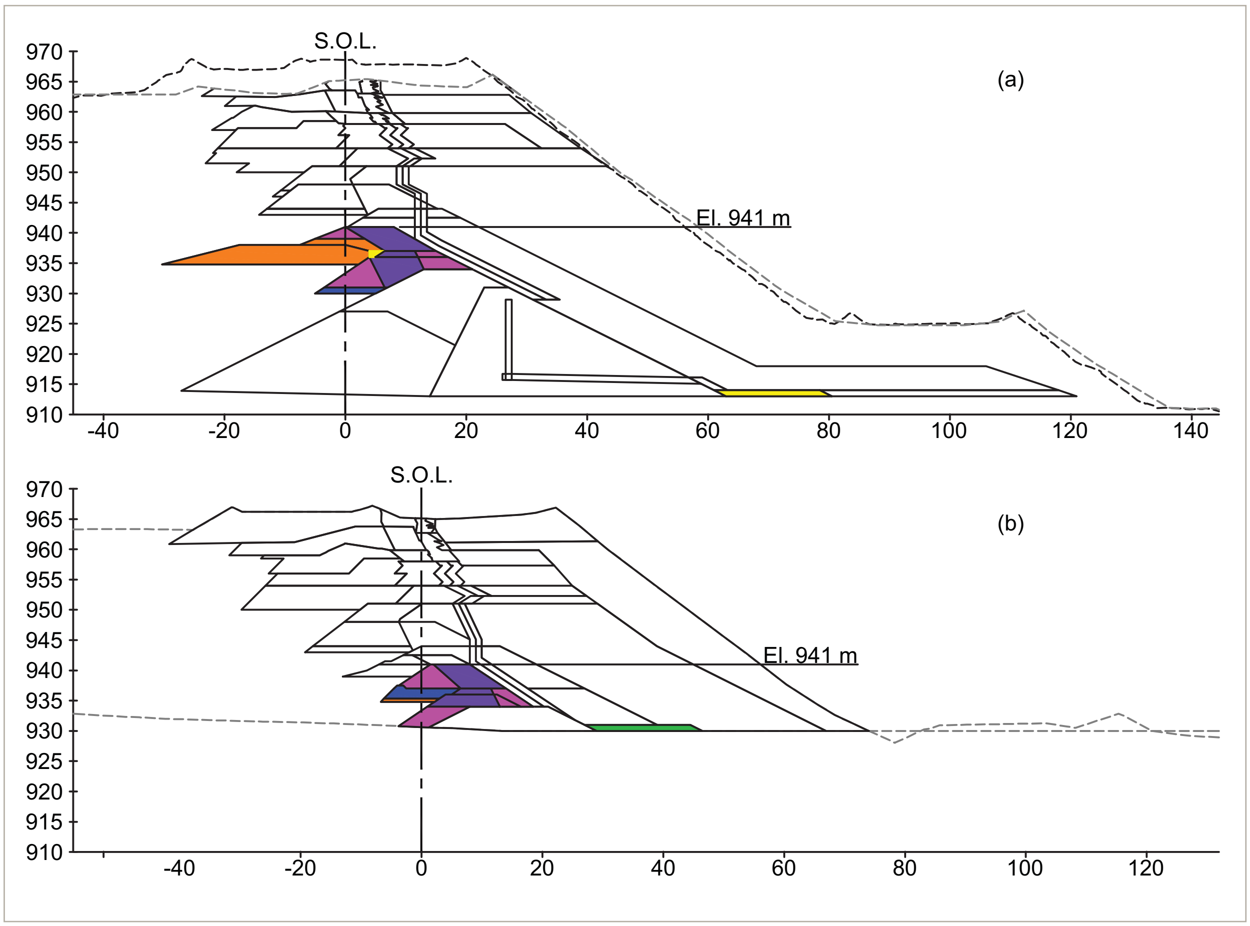
5.4.2 STAGE 2: 1998 — 2000

FIGURE 5.4.2: STAGE 2 (a) MAIN EMBANKMENT (b) PERIMETER EMBANKMENT



Stage 2 was designed to be the first "modified centreline" raise. In the design, the core (Zone S) and chimney drain (Zone F) were extended upward, while adding a new zone of what was intended to be mine waste rock (Zone C) on the downstream slope and outward as a berm along the Main Embankment.[15] Another new feature for the Main Embankment was a longitudinal drain, designated the "upstream toe drain," on the upstream side of the core near the crest of the raise. Its purpose was to allow drainage of the deposited tailings and reduce the embankment phreatic surface. An additional seven relief wells[16] and a relief trench were also included to reduce elevated foundation pore pressures. For the design configuration of the Main Embankment, FS = 1.67 was computed for the downstream slope, exceeding the minimum required value of 1.3.[17]

The as-built configuration of the Stage 2 Main Embankment shown in **Figure 5.4.2(a)** differed from the design in several important respects. The intended Zone C mine waste fill was not added to the downstream slope, and the berm along the toe was not constructed. Rather than adhering to a "centreline" configuration, raise 2 utilized entirely "upstream" construction.[18] The same conditions prevailed for the Perimeter Embankment shown in **Figure 5.4.2(b)**. These as-built conditions were never reconciled with the Stage 2 stability analyses, which had been predicated on the original design configuration.

Operational trials and test fills established the feasibility of using cyclone sand underflow upstream of the core (Zone CS), and Stage 2 was the first to do so. A limited trial zone of cyclone sand would remain in the downstream shell of the Perimeter Embankment at design Section D, but cycloning would later be abandoned for both operational and economic reasons.[19, 20]

In other operational matters, problems with the tailings pipeline system produced difficulties in maintaining the required tailings beach, with water directly contacting the embankments in some places.[21]

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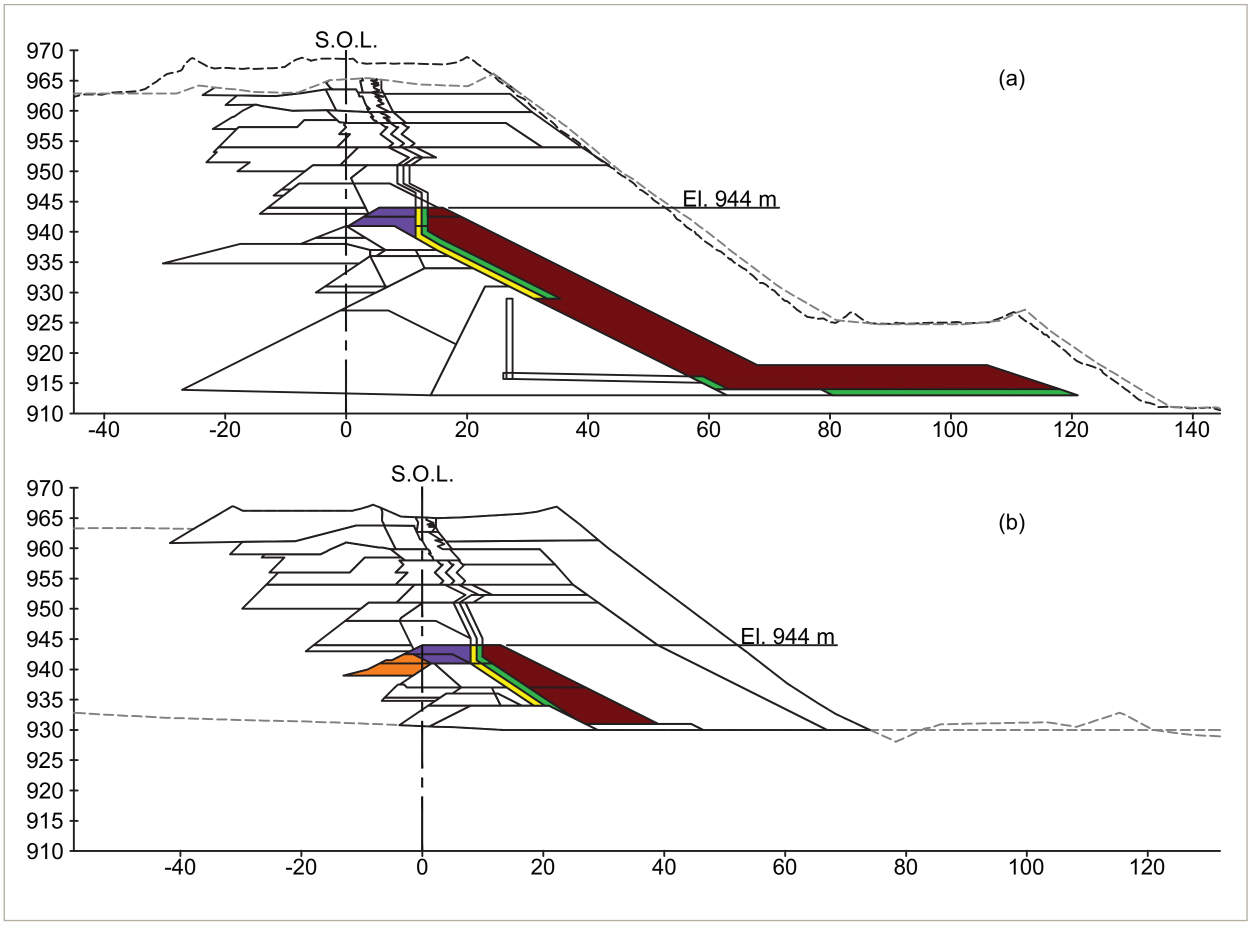
5.4.3 STAGE 3: 2000 — 2001

FIGURE 5.4.3: STAGE 3 (a) MAIN EMBANKMENT (b) PERIMETER EMBANKMENT



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As Stage 2 was being constructed, efforts were underway to select materials for the upcoming Stage 3 and the remainder of the dam. A series of design studies in 1999 and 2000 developed a variety of configurations and options using cyclone sand [22, 23] as well as a rockfill alternative. [24] Various combinations were considered for the Main Embankment, the Perimeter Embankment, and the newly added South Embankment that would confine the third side of the impoundment beginning with Stage 3.

In May 2000, MPMC requested approval from MEM for a Stage 3 design using only cyclone sand for the Perimeter Embankment, with the Main Embankment raised using rockfill and the South Embankment with glacial till. [25] This was changed, however, in April 2001, when MPMC requested MEM approval for yet a different Stage 3 design using rockfill for the downstream Zone C in all three embankments. As shown by the as-built configuration in Figure 5.4.3, this plan was ultimately adopted for Stage 3 using rockfill sourced from a quarry. [26]

Despite the convoluted nature of the Stage 3 design process, an important milestone was that the Observational Method was formally invoked as the basis for design. [27] Thenceforward, each incremental raise was to be continually re-evaluated during operations, based on measured data from the piezometers and two inclinometers installed in July 2001. Putting this into effect, however, would have to wait. Not long thereafter, Mine operations were suspended for economic reasons on October 13, 2001. [28]

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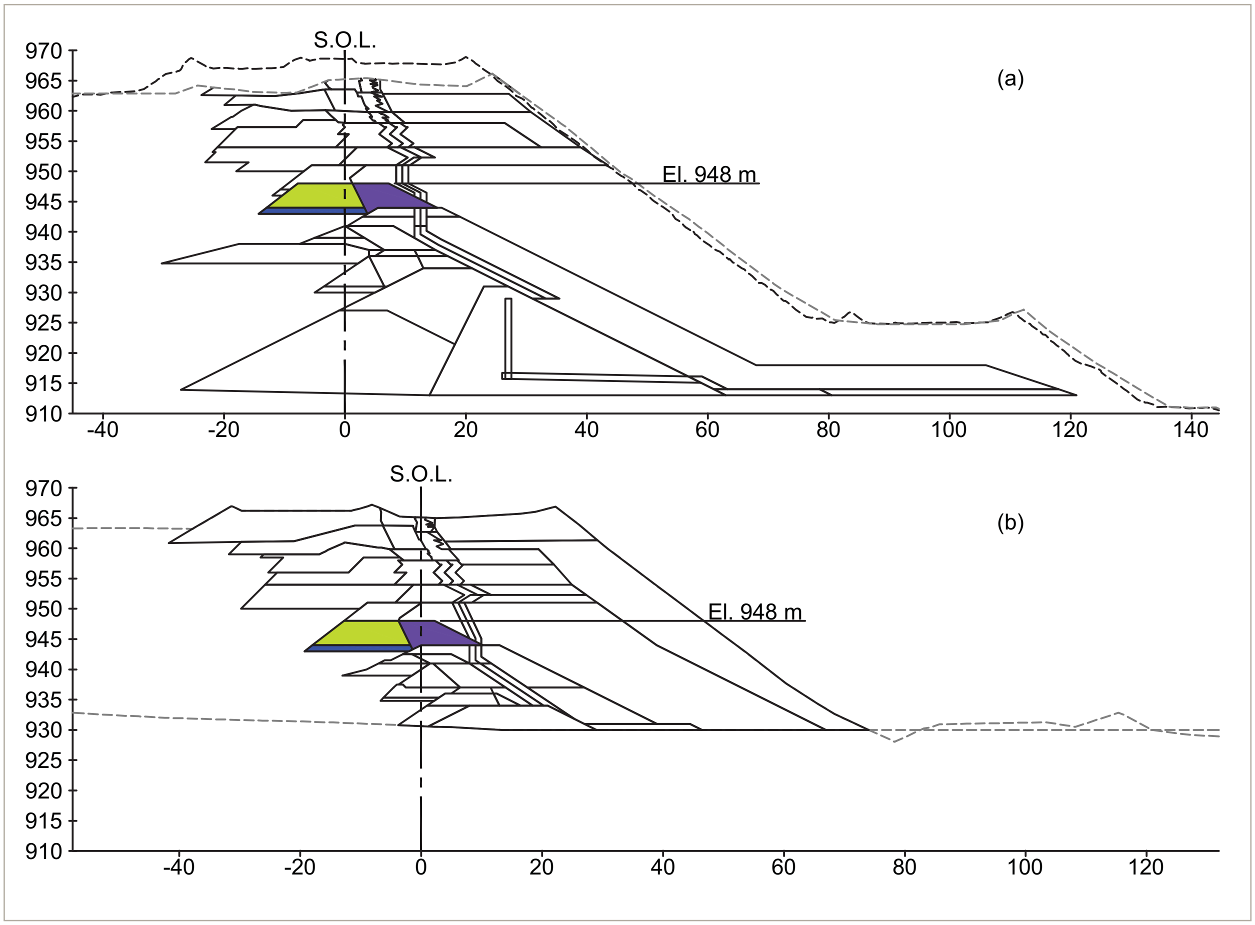
5.4.4 STAGE 4: 2005 — 2006

FIGURE 5.4.4: STAGE 4 (a) MAIN EMBANKMENT (b) PERIMETER EMBANKMENT



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After a hiatus of over 3 years, Mine operations resumed in February 2005. Design of the Stage 4 raise called for a small cap on the Stage 3 crest extending over the tailings in an "upstream" configuration, together with Zone C rockfill on the downstream slope. Also included in the design was a rockfill buttress on the downstream slope of the Main Embankment to increase the factor of safety to 1.5 in anticipation of closure requirements. [29]

As illustrated in **Figure 5.4.4**, only the cap was constructed in Stage 4 without any additional rockfill on the downstream slope, resulting in another "upstream"-type raise. [30] In constructing this raise, trial programs pioneered the use of hydraulic-cell deposition of tailings for the upstream Zone U, a practice that continued throughout construction.

Separately, operational problems in maintaining the required tailings beach continued, with water directly against the embankment in several areas. [31]

Renewed operation brought renewed queries from MEM concerning the glaciolacustrine foundation materials. One concerned the characteristics and effects on dam stability of softer GLU deposits at groundwater well GW96-1A downstream from the Perimeter Embankment. [32] In response, KP cited borrow area test pits and auger borings as confirming that "the glaciolacustrine deposit encountered in GW96-1A is a discontinuous unit and will not adversely affect the dam stability." [33]

The breach subsequently occurred 300 m due west of GW96-1A.

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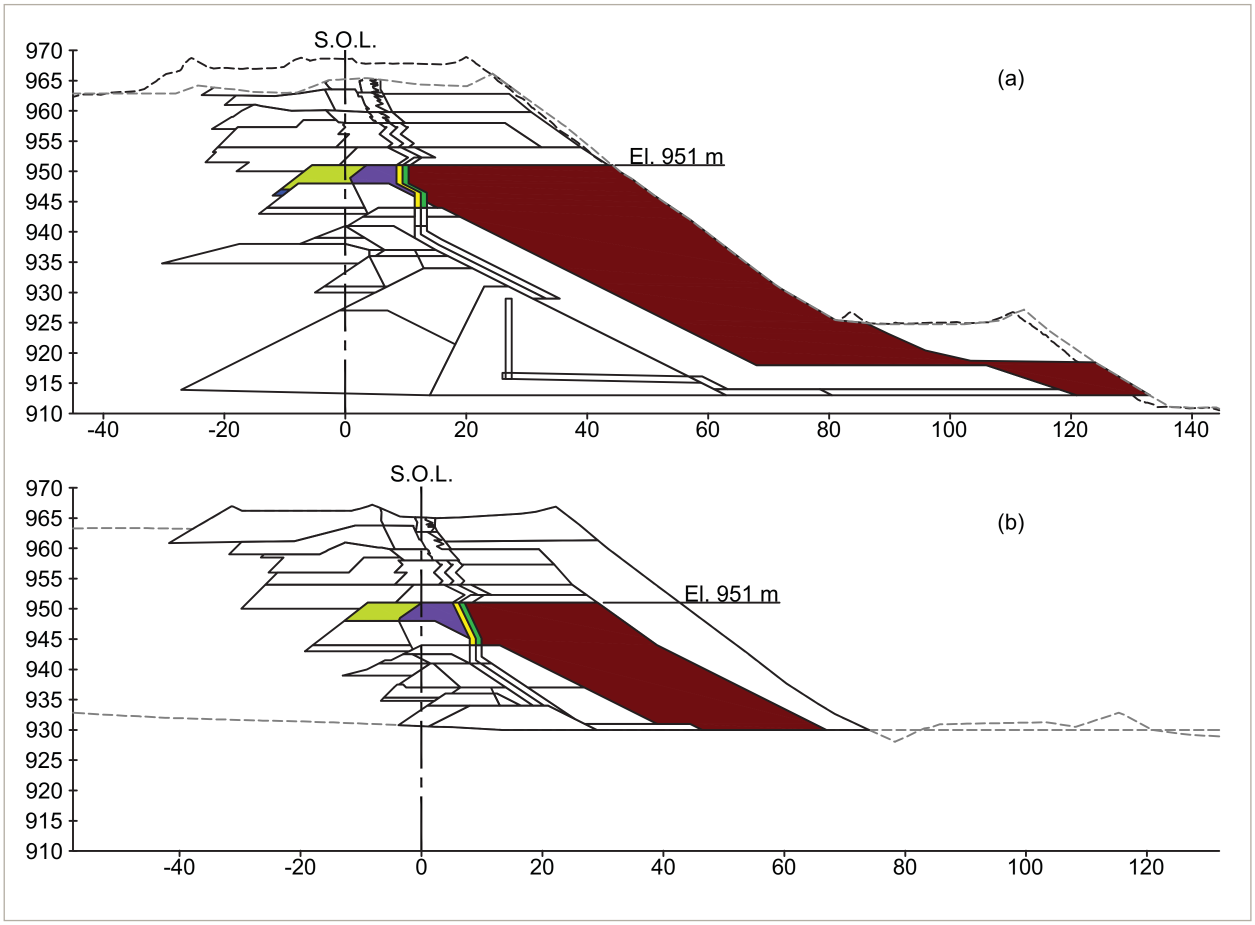
5.4.5 STAGE 5: 2006 — 2007

FIGURE 5.4.5: STAGE 5 (a) MAIN EMBANKMENT (b) PERIMETER EMBANKMENT



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The Stage 5 design once again incorporated the downstream Zone C rockfill that had been deferred in previous stages, and this time it was built. But since the material would now be sourced from mine waste rather than quarried, mine production and delivery had to be accommodated.[34] Due to related restrictions, it was planned to place the Zone C outslope to an "interim" 1.4H:1V inclination—rather than the design basis 2.0H:1V—as a temporary expedient until mine waste delivery could catch up with construction. The steeper slope would be expanded and flattened to 2.0H:1V "once the embankments have reached the Stage 5 design elevation."[35] An ESA factor of safety of 1.5 was reported for the steeper interim slopes of the Main Embankment and 1.9 for the Perimeter Embankment.

It was planned to place the Zone C outslope to an "interim" 1.4H:1V inclination—rather than the design basis 2.0H:1V—as a temporary expedient until mine waste delivery could catch up with construction.

Stage 5 construction proceeded from Stage 4 in a continuous, uninterrupted campaign and was completed in November 2007. But instead of rectifying the interim steep slopes at this time as had been intended, such measures were left to future stages of embankment raising.[36]

Stage 5 saw the first substantial enlargement of the Perimeter Embankment, with widening of the crest and expansion of the downstream Zone C rockfill as shown in **Figure 5.4.5(b)**. At the same time, an upstream toe drain was added to complement the companion drain already installed at the Main Embankment.

Operationally, chronic problems with maintaining the tailings beach continued, with procurement of enough tailings pipe to traverse the entire embankment perimeter now the anticipated solution.[37]

The year 2006 marked the 10-year interval for the mandatory third-party Dam Safety Review (DSR), which was prepared by AMEC.[38] The most salient aspects of this report concern its assessment of foundation strength and related dam stability. Shear failure of the dam slope, including failure through the foundation, was first on a list of potential failure modes applicable to the Mount Polley dam in relation to "excessive loading at or near the crest or a weakness in the foundation." Noting the apparent overconsolidation of the glaciolacustrine materials, the report identified two conditions of particular interest: the possible presence of pre-sheared planes of weakness, and the potential for "brittle" response involving strength loss at small strains. The DSR contained no mention of the behaviour of foundation materials in undrained shear.

The Dam Safety Review contained no mention of the behaviour of foundation materials in undrained shear.

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The DSR also remarked on the lack of a tailings placement strategy that had impeded systematic development of a tailings beach for so long, calling lack of such a beach a “deficiency” and noting that the dam had not been designed as a water dam.

Shortly after the DSR was submitted, at MPMC’s request, AMEC produced a follow-up report that reviewed several possible optimization measures for the TSF.[39] One measure was to reduce the width of the core to as little as 3 m to 4 m. Another was to eliminate the uppermost 1 m of the dam core, since this part of the crest “only provides freeboard.”

The optimization report also questioned the need for the Main Embankment buttress first proposed for Stage 4 and partially constructed for Stage 5. It concluded that foundation strengths used previously would result in adequate stability without a buttress. The only proviso was the potential for pre-sheared planes of weakness in the foundation, a question that remained outstanding from the DSR.

With the water balance “fine tuned to an accuracy that is in the range of centimeters” in terms of impoundment water elevation, the report proposed that the wave runup allowance, and therefore freeboard requirements, could be reduced. Remarking on beach development, it further stated that unless water was deep enough to affect stability of the Zone U tailings, there was “no rush” in developing a beach along the Main Embankment to correct the deficiency identified in the DSR.

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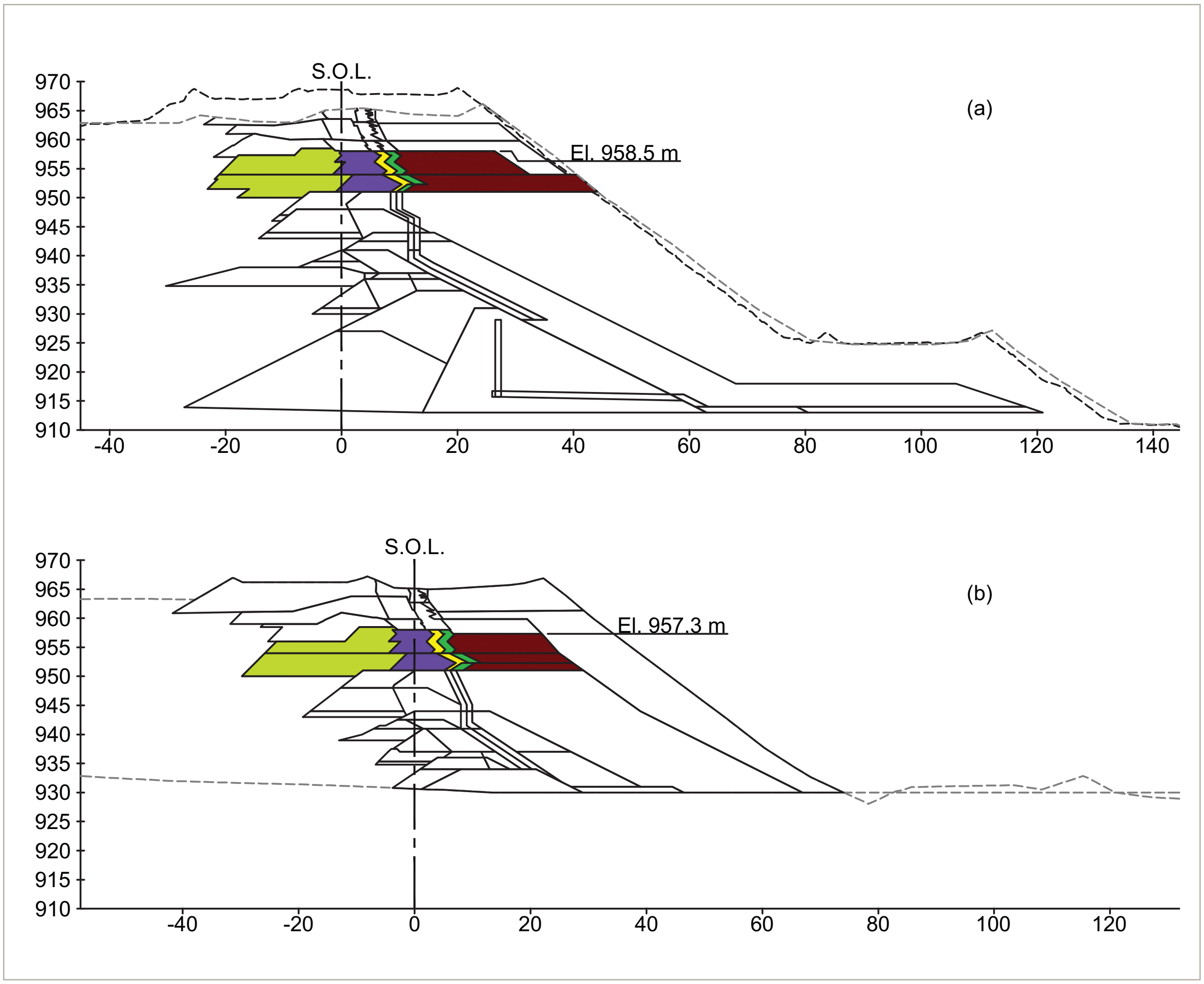
5.4.6 STAGE 6: 2007 — 2011

FIGURE 5.4.6: STAGE 6 (a) MAIN EMBANKMENT (b) PERIMETER EMBANKMENT



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The Stage 6 design for the Main Embankment incorporated two components: an additional 7 m of fill on the crest, and a Zone C rockfill buttress at the downstream toe. The Zone S core was reduced from its former 8 m width to 5 m on the basis of the effectiveness of the upstream toe drains in lowering the phreatic surface and gradients within the core.[40] Stage 6 also introduced the practice of raising the Zone S core, the thin Zone F filter, and the equally thin Zone T transition in an intricate zigzag configuration.

The Main Embankment buttress, first included in the Stage 4 design but never fully constructed, was an outgrowth of two factors. First was the effect on stability of the "interim" 1.4H:1V slopes that had persisted since Stage 5. Second were the foundation strength interpretations put forward in the DSR. Stage 6 stability analyses adopted an estimated residual strength of 24° for the GLU foundation materials at the Main Embankment to account for the possible presence of pre-shearing. The resulting buttress produced an ESA factor of safety of 1.4, satisfying the FS = 1.3 design requirement for operation.

But once again, the Stage 6 buttress was not constructed as designed, turning out to be about 5 m below its design height and short of its design extent.

The Stage 6 design sought to accommodate the limited mine waste delivery rates experienced in Stage 5—and the consequent slope oversteepening—by extending construction over a 2-year period. Even so, the calculated FS = 1.4 for the Stage 6 Main Embankment indicated that the buttress would need to continue being raised in future dam stages, requiring more material. To make matters worse, KP noted that only non-reactive mine waste could be used, further constraining available quantities and confirming buttress construction as a continuing proposition. But once again, the Stage 6 buttress was not constructed as designed, turning out to be about 5 m below its design height and short of its design extent.[41]

None of these buttressing considerations pertained to the Perimeter Embankment. Residual strength parameters were not applied to its foundation, and the resulting factor of safety of 1.7 required no enhancement according to the FS = 1.3 criterion.

Elsewhere, beach deposition from the extended tailings discharge line had not been successful in preventing water accumulation against the Main Embankment. This increased flows in the same upstream toe drain whose effectiveness had been cited as justification for reducing the width of the Stage 6 core.[42]

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Meanwhile, follow-up related to the DSR continued. MEM requested that KP provide the results of its recommended direct shear testing, which essentially confirmed the Stage 6 design ESA factor of safety.[43, 44] But beyond this was another item in KP's response that marked a milestone in two fundamental respects. For the first and only time during the design process, an undrained strength analysis (USA) was performed. This was also the only instance that the foundation clay behaviour would be taken as other than that of stiff and highly overconsolidated material. Using a typical S_u/σ_V', of 0.25 for soft, normally consolidated clays, KP found a USA factor of safety of 1.1 for the Stage 6 configuration. Not recognizing that this strength might indeed be the operational strength under static loading conditions, KP concluded that "there is also sufficient undrained strength in the lacustrine unit for the embankment to remain stable." This conclusion would henceforth never be called into question.

Not recognizing that this strength might indeed be the operational strength under static loading conditions, KP concluded that "there is also sufficient undrained strength in the lacustrine unit for the embankment to remain stable." This conclusion would henceforth never be called into question.

Operation of Stage 6 throughout 2009 and 2010 highlighted other matters. In 2009, movements in the GLU recorded at Inclinometer SI01-02 resulted in expanding the Main Embankment buttress in the immediate area. This proved to be effective in arresting further displacements.[45] By 2010, the buttress had been extended along the west side of the Main Embankment, but still remained to be completed along its entire length.[46] In another development, a tension crack appeared at the downstream edge of Zone C at Sta. 3+400 of the Perimeter Embankment. Although interpreted to be an artifact of near-surface movement, a follow-up stability assessment was nonetheless recommended.

Inadequate tailings beach development along the Main and South Embankments was flagged yet again, this time in an MEM inspection. Noting that an above-water beach was a requirement of the design, the inspector considered its absence at the southeast corner of the Main Embankment to be a "Departure from Approval" and ordered that a beach be "re-established as soon as possible in this area to meet the design objectives."[47]

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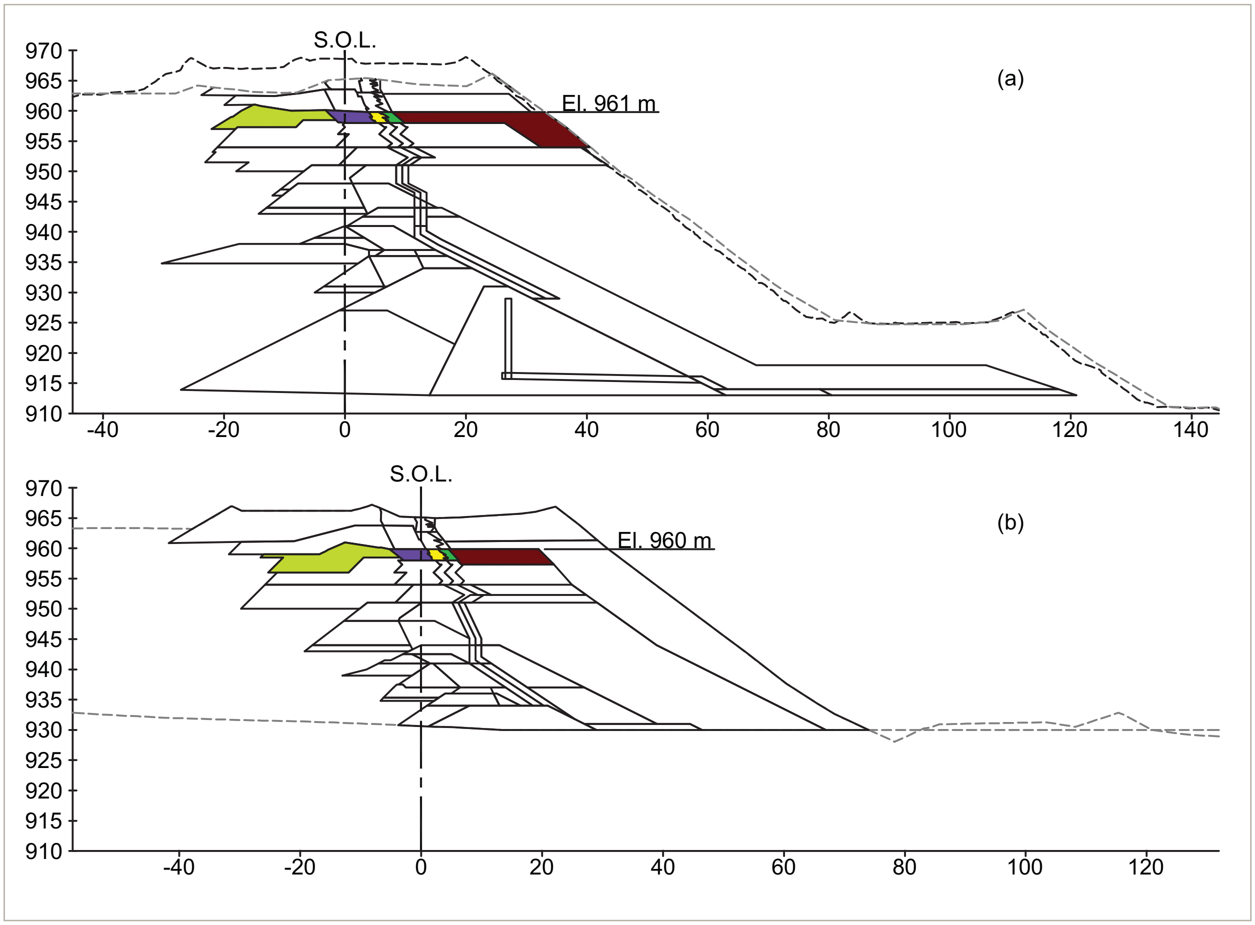
5.4.7 STAGE 7: 2011 — 2012

FIGURE 5.4.7: STAGE 7 (a) MAIN EMBANKMENT (b) PERIMETER EMBANKMENT



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In 2011, the Engineer of Record (EOR) responsibilites were transferred from KP to AMEC, and with them the design of Stage 7 for a height increase of 2.5 m. No new Zone C fill would be added to flatten the downstream slope, and no buttress expansion would be conducted. Continuing the stability analysis protocols from Stage 6 and the 2006 DSR, AMEC found that the ESA factor of safety using residual strength for the foundation GLU was unchanged from the Stage 6 value of 1.4 [48] for the Main Embankment. Similar conclusions applied to the factor of safety for the Perimeter Embankment.

The same year also saw the completion of the 2011 Geotechnical Site Investigation, the first major foundation exploration program since Stage 1. It consisted of 11 sonic drillholes, with piezometers installed in each, plus three new inclinometers. [49] Emphasis was on definitively evaluating the DSR hypothesis that the glaciolacustrine foundation soils might contain pre-sheared planes of weakness and the operative residual strengths that would accompany them.

Careful inspection of recovered core revealed no indications of slickenside features and no evidence of pre-shearing. Thus, residual strengths need no longer be considered. Neither, it was concluded, did these conditions indicate that the 2010 crack in the Perimeter Embankment was attributable to weak soil conditions in the area.

With respect to stress history, the 2011 report further concluded that the GLU was overconsolidated, consistent with previous interpretations. The softer conditions in monitor well GW96-1A adjacent to the Perimeter Embankment that MEM had questioned in 2005 were said to be "not of significant concern in this instance as the drillhole location is approximately 140 m further downstream from the current toe of the dam." In fact, the report said, "based upon available information, foundation conditions along the Perimeter Embankment appear more favourable than those along the Main Embankment" in terms of the presence and extent of clay-rich zones within the GLU.

At the Main Embankment, piezometers were installed generally beneath the buttress where measured pore pressures would be reflective of additional fill. This was not the case at the Perimeter Embankment, where all of the new piezometers were located 15–20 m from the downstream toe. Similarly, Inclinometer SI11-04 installed during the program was 15 m away.

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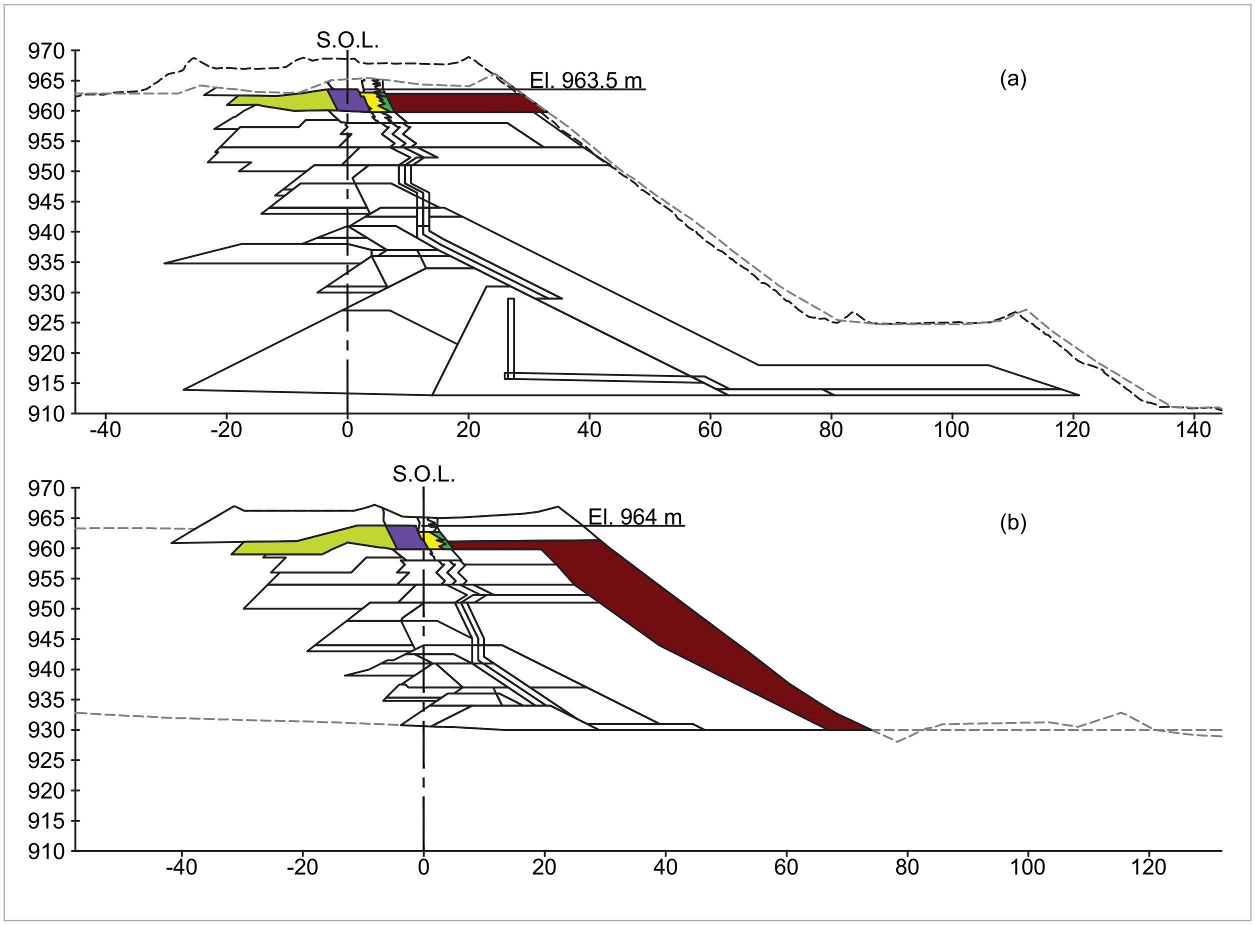
5.4.8 STAGE 8: 2012 — 2013

FIGURE 5.4.8: STAGE 8 (a) MAIN EMBANKMENT (b) PERIMETER EMBANKMENT



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> **Flattening of the Stage 5 "interim" oversteepened slope to 2H:1V was deferred yet again—not until completion of Stage 5 raise as first proposed, but this time until completion of the entire dam.**

Stage 8 was initially designed as a 3.5 m raise, then increased to 5 m with an accelerated construction program. It would change to conventional "centreline" raising from the previous "modified centreline" that had progressively shifted the raises upstream. Stage 8 fill was added only to the crest of Stage 7 Main Embankment,[50, 51] while the Perimeter Embankment was widened as well. But in both cases, flattening of the Stage 5 "interim" oversteepened slope to 2H:1V was deferred yet again—not until completion of Stage 5 raise as first proposed, but this time until completion of the entire dam.

In evaluating the stability of the steepened slope, AMEC returned to the peak-strength interpretation for the GLU materials based on the findings of its 2011 field program. For a peak effective-stress friction angle of 28°, the ESA factor of safety was found to be a barely adequate 1.31[52] for the Main Embankment.

The 2011 investigation showed the GLU materials at Section D of the Perimeter Embankment to be deeper than at the Main Embankment. In stability analyses at Section D near the breach, the critical failure surface did not reach the GLU and remained within the overlying foundation till, producing a much higher factor of safety of 1.77.

The larger issue of what minimum factor of safety should be required was addressed in a September 19, 2012 communication from MEM to MPMC that deserves to be quoted at length:

The factor of safety for the main embankment is only marginally above the short-term design criteria of 1.3... AMEC has interpreted Table 6-2 from the 2007 Dam Safety Guidelines somewhat differently than I have seen in the past. This table recommends a minimum factor of safety of 1.3 at the end of construction and 'before reservoir filling' and a factor of safety of 1.5 at the 'normal reservoir level.' AMEC has interpreted the construction period as the entire pre-closure period, and this is open to debate. However, I consider that sufficient mitigation measures are in place (i.e., piezometer trigger thresholds) to support this more liberal interpretation in this instance.[53]

Although questioning AMEC's interpretation of the Dam Safety Guidelines, MEM was prepared to accept FS = 1.3, but only in conjunction with the Observational Method.

In other matters, the recurring problem of tailings beach development was not directly addressed in the 2012 inspection report, but an airphoto showed no tailings beach over approximately 40% of the impoundment perimeter.[54] The report also noted that seepage had been present at the toe of the Perimeter Embankment near the breach section and that it had moved from previous years. Based on interviews with MPMC personnel, the Panel believes that the likely source of the apparent seepage was actually a buried outlet of the upstream toe drain.[55, 56]

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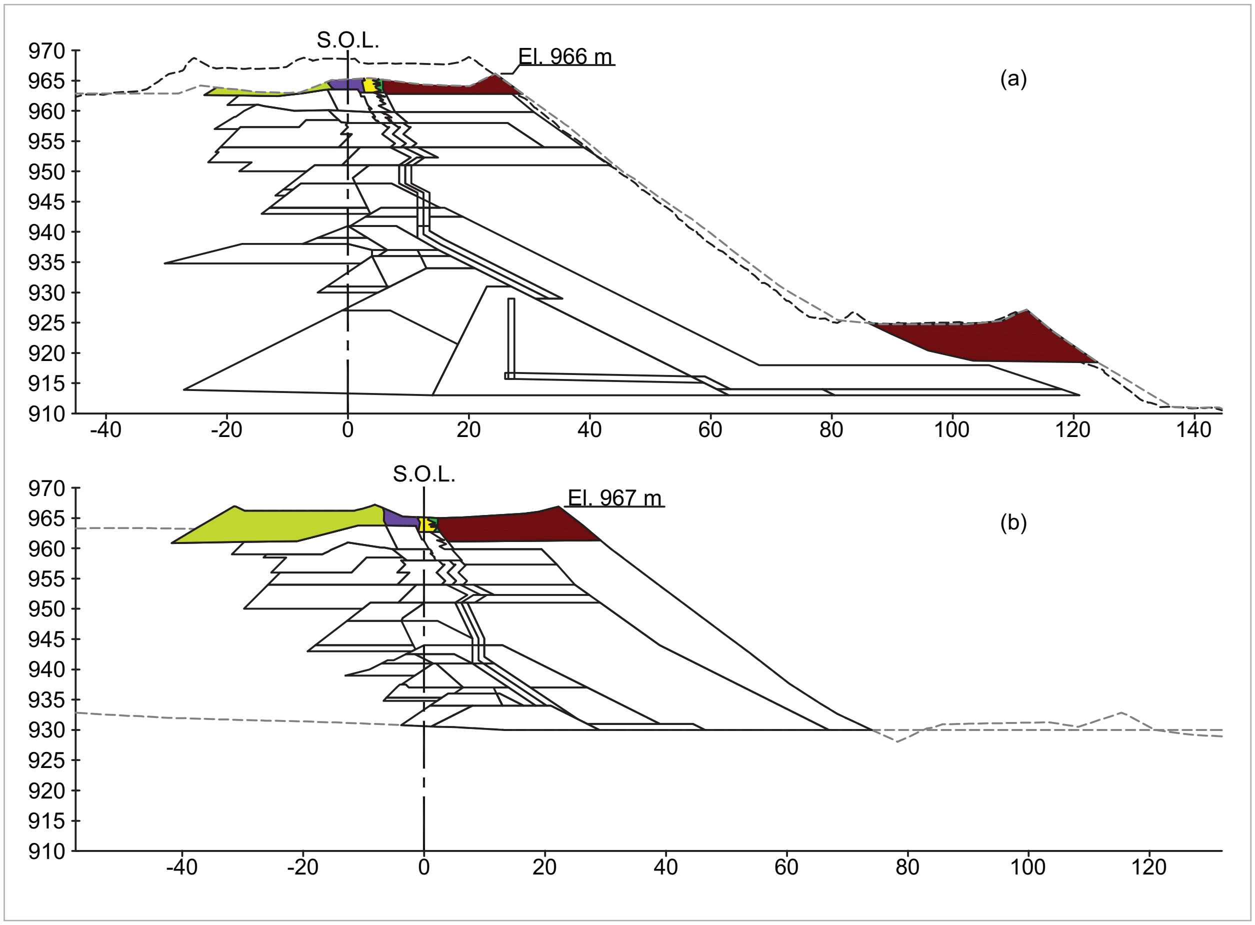
5.4.9 STAGE 9: 2013 — 2014

FIGURE 5.4.9: STAGE 9 (a) MAIN EMBANKMENT (b) PERIMETER EMBANKMENT



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Stage 9, whose construction was being completed when the breach occurred, encompassed a period of intense activity with a number of seminal events in the months, weeks and days preceding the failure. While AMEC remained the EOR until the planned completion of Stage 9 to El. 970, BGC would officially become the EOR beginning with construction of the planned Raise 10. Consequently, 2013 to 2014 was also a period of transition, with overlap in activities, if not responsibilities.

AMEC's April 11, 2013 design for Stage 9 planned a substantial 6.5 m height increase by adding fill to the crest of Stage 8. Retaining the peak-strength interpretation for the GLU foundation materials, AMEC found that raising the Main Embankment buttress to El. 925 m would be needed to nominally achieve a minimum ESA factor of safety of 1.3.[57]

Commenting on the implications of this value, MEM's remarks on July 29, 2013, echoed its previous concerns:

The stability analyses indicate that the FOS for the 'Main Embankment' only marginally achieves the short term CDA design criteria of 1.3. ... Previous correspondence from MEM has highlighted the difference in interpretation of the CDA Guidelines. AMEC has considered the construction period to be the entire 'pre-closure' period while CDA Guidelines, Table 6-2 recommends a minimum FOS of 1.3 'before reservoir filling,' and a FOS of 1.5 at the 'normal reservoir level.'

MEM requires a commitment from Mount Polley that they are moving toward increasing these FOS for the main embankment as part of subsequent dam raises in an effort to move toward achieving a long term FOS equal to 1.5. It is expected that Mount Polley will continue their transition to centerline construction and provide additional buttressing with time.[58]

This marked a major change in direction. A factor of safety of 1.5, not 1.3, would become the governing criterion. Moreover, buttressing could no longer be deferred for either embankment. A factor of safety of 1.58 had been calculated for Section D of the Perimeter Embankment, once more unaffected by the GLU foundation materials. But even this value was approaching the new minimum of 1.5 that MEM was now aiming to enforce, and buttress preparation needed to begin.

This marked a major change in direction. A factor of safety of 1.5, not 1.3, would become the governing criterion.

By the end of the 2013 construction season, pre-stripping for a buttress around the Perimeter Embankment had been completed, including the area of the breach section.[59] In a Panel interview, the contractor who performed the work stated that portions of this area remained open at the time of the breach,[60] an assessment confirmed by MPMC.[61, 62]

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Meanwhile, attention was turning to longer-term prospects for continued dam raising, and the outlook was not good. BGC made explicit the connection between the structural limitations of the dam and the ever-growing volumes of surplus water it was being called upon to contain. In a June 18, 2013 memorandum, it stated:

A continuous beach along the complete upstream length of the dam is the design requirement necessary for dam stability and needs to be achieved moving forward regardless of the final targeted crest elevation. The current water pond surplus does not allow for the development/maintenance of above-water beaches. [63]

It elaborated on this topic a month later, on July 25, 2013:

An above-water tailings beach separating the till core from the reclaim water pond constitutes a fundamental design element of the dam. Without a wide above-water beach, the MPMC tailings dam is effectively being operated as a water-retaining dam, with the water pond effectively in direct contact with the till core, separated by only a narrow zone of tailings or waste rock. [64]

During the ensuing months, this chronic water-surplus problem would become acute. For years, dam raising had managed to stay one step ahead of the rising water. But on May 24, 2014, the water caught up. With Stage 9 nearing completion, what was described as "seepage flow" was observed over the dam core.[65] Intensive surveillance and construction activity over the following days and weeks succeeded in raising low areas around the embankment perimeter, restoring containment integrity, and saving the dam from overtopping failure.

For years, dam raising had managed to stay one step ahead of the rising water. But on May 24, 2014, the water caught up.

As the gravity of the water problem was becoming apparent, so was the consequent necessity of dam raising beyond Stage 9. MPMC required some estimate of future dam footprint so that prerequisite stripping of additional areas could commence immediately. BGC responded on October 22, 2013, with a memorandum that outlined an approach to dam raising that resurrected the residual-strength interpretation for GLU, while at the same time establishing new factor of safety criteria conforming to MEM's 2013 directive.[66]

This approach was formalized in BGC's design report for Stage 10 issued on July 25, 2014, just eight days before the breach.[67] The proposed raise would achieve a minimum FS = 1.5 for the Main Embankment using peak effective-stress strength for the GLU and full dissipation of load-induced pore pressures. But this new design philosophy would go one step further.

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Notwithstanding AMEC's 2011 subsurface investigation, a "more conservative" approach would be taken by allowing for the possibility of brittle behaviour or pre-shearing in the GLU. This would apply an additional criterion of FS = 1.1 using residual strength in the GLU for what was characterized as a "reasonable worst-case scenario." So the residual-strength interpretation was now reintroduced after first being suggested in the 2006 DSR, adopted in design of Stages 6 and 7, then abandoned in design of Stage 8.

The BGC report also commented on the application of the Observational Method to these conditions. Citing its chief progenitor Ralph Peck, the report recognized that this design strategy requires preplanned actions to deal with "every unfavourable situation that might be disclosed by the observations."[68] But it also acknowledged that any brittle behaviour detected by the instrumentation would result in strength reduction too rapid to recognize and respond to. Hence the need, it said, for the minimum FS = 1.1 and its associated residual strength interpretation as a contingency. As a result, the existing buttress on the Main Embankment would be raised, and a new buttress about 8 m high would be added to the Perimeter Embankment. This was to include what would become the area of the breach.

In a final irony, the Stage 10 buttress was scheduled for construction on the Perimeter Embankment in late 2014 or early 2015. Had it been in place on August 3, 2014, the dam would have survived.

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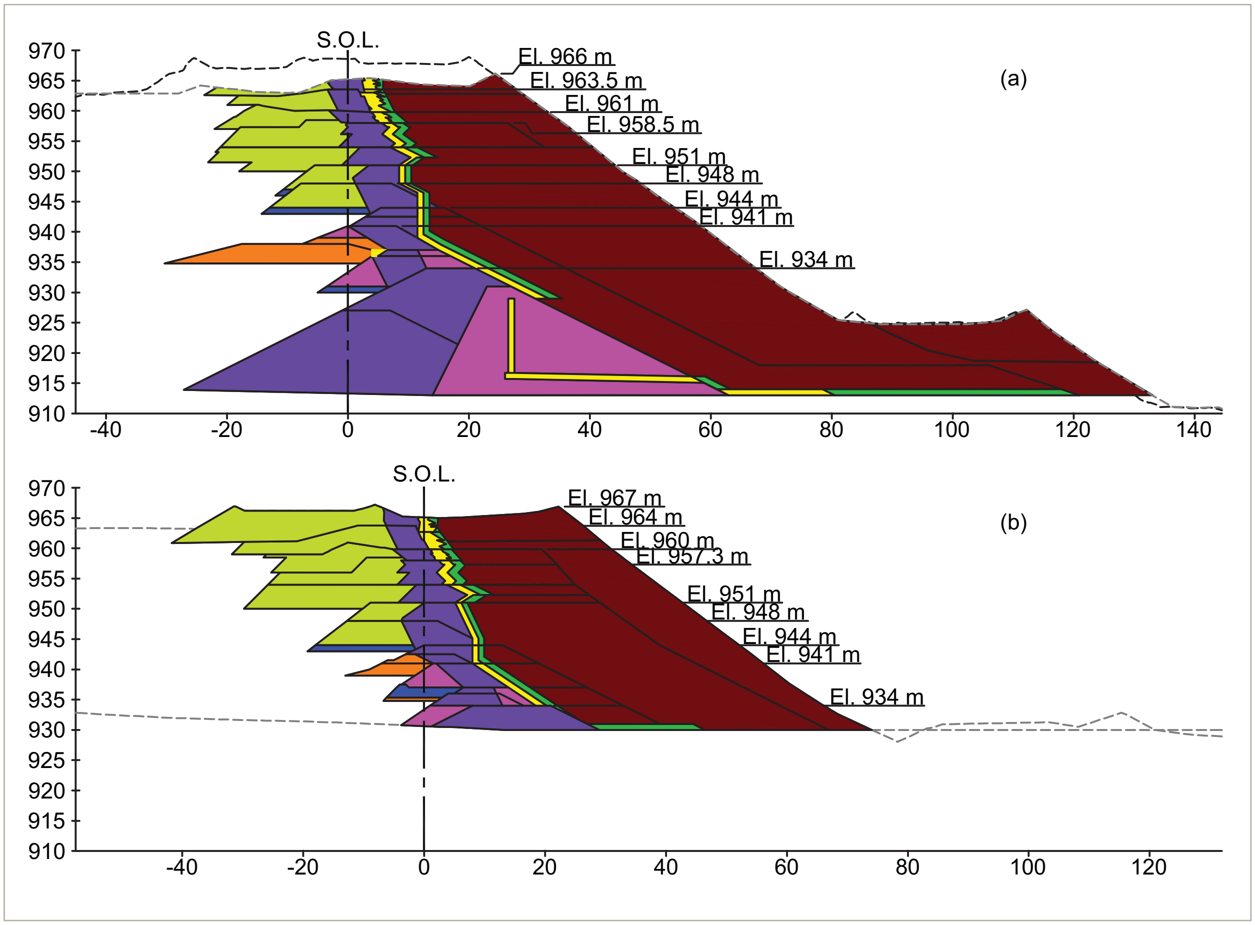
FIGURE 5.4.10: DAM CONFIGURATION ON AUGUST 3, 2014. (a) MAIN EMBANKMENT (b) PERIMETER EMBANKMENT AT BREACH SECTION



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5.4.10 COMMENTARY

The preceding account is in many ways a story of too little, too late. From the beginning, dam raising proceeded incrementally, one year at a time, driven by impoundment storage requirements for only the next year ahead. More reactive than anticipatory, there was little in the way of long-term planning or execution. This was most clearly displayed by the absence of an adequate water balance or water treatment strategy, and the overtopping failure that nearly resulted. Moreover, the related absence of a well-developed tailings beach violated the fundamental premise of the design as a tailings dam, not a water-storage dam.

The preceding account is in many ways a story of too little, too late. From the beginning, dam raising proceeded incrementally, one year at a time, driven by impoundment storage requirements for only the next year ahead.

The same problem was apparent in production and scheduling for mine waste used in dam construction. The design was caught between the rising water and the Mine plan, between the imperative of raising the dam and the scarcity of materials for building it. Something had to give, and the result was oversteepened dam slopes, deferred buttressing, and the seemingly ad hoc nature of dam expansion that so often ended up constructing something different from what had originally been designed.

Ultimately, the tortuous, incremental nature of this process, and the constraints under which it was conducted, caused it to lose sight of basic precedent. With a slope steepness ordinarily reserved exclusively for rockfill dams on sound rock foundations, the Perimeter Embankment at the breach section was allowed to reach a height of almost 40 m with an unbuttressed downstream slope of 1.3H:1V.

Not just the design process but also the design itself had shortcomings. Even if not contributing directly to the failure, some design details were problematic. Already thin to begin with, reducing the core width from 8 m to 5 m made it even more vulnerable to differential settlement and cracking. Both the filter and transition zones were just 1 m wide, placing great demands on their performance. Yet in a sampling of as-placed Zone S filter gradations, the Panel found that 30% were too coarse to meet the $D_{15}<0.7$ mm filter criterion and 70% had internally unstable grading,[69, 70] with only about 25% satisfying both filter and internal stability requirements.

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There were ambiguities in the governing factor of safety, adapted from CDA Guidelines never intended for tailings dams. An FS = 1.3 design criterion using peak effective-stress strength left little margin for error, and trigger-level factors of safety for critical piezometric conditions were even lower at 1.1. Such values may have made it easier to rationalize the departure from slope precedent, but harder to gauge just how closely dam raising was approaching the edge of the cliff.

There was an oversimplified conception of the complex stratigraphy of the glacial deposits described in section 5.2. An Upper GLU unit had been encountered in groundwater well GW96-1A and a lower unit in sonic borehole VW11-10. But only the lower unit was included in stability analysis of the Perimeter Embankment, and it had no influence on calculated factors of safety. The possibility that the upper unit might be present beneath the Perimeter Embankment was not accounted for in conceptualization of geologic conditions. More than this, its stress history was much less favourable.

Yet the overarching problem, and the one the Panel finds most troubling, is the failure throughout to adopt the appropriate undrained strength interpretation for the glaciolacustrine silts and clays in the foundation. These materials were assumed everywhere to be stiff, and therefore overconsolidated, although there was never any attempt to quantify their degree of overconsolidation or stress history. And even if they were overconsolidated to begin with, it was not recognized that the increasing loads imposed by the dam as it grew higher would eventually cause them to reach a normally consolidated state.

The overarching problem, and the one the Panel finds most troubling, is the failure throughout to adopt the appropriate undrained strength interpretation for the glaciolacustrine silts and clays in the foundation.

There is a fundamental difference in pore pressure behaviour between these two conditions and the undrained strengths they produce. Overconsolidated clays are dilatant during undrained shearing. That is, they tend to increase in volume, producing no positive pore water pressures. By contrast, normally consolidated clays are contractive and do develop positive pore pressures. This difference in pore pressure response during shearing makes the undrained strength of a normally consolidated clay lower than the same material in an overconsolidated state. But the design did not account for the undrained strength that would pertain if the dam were to fail rapidly—which proved in the end to be the case.

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Rather, the design was based exclusively on ESA in various forms using peak and residual strengths, all of which neglected pore pressures that would develop in normally consolidated GLU during rapid, undrained shearing. The design never incorporated an undrained strength analysis (USA), except in one instance. A USA performed for Stage 6 using an undrained strength typical of normally consolidated clays produced a factor of safety of only 1.1. But this was not seen to be the operative strength and was not considered further here or in subsequent stages. If undrained strength behaviour had been properly understood and applied throughout, the outcome could have been much different.

The Observational Method was invoked early on as the basis for design. This commonly accepted approach uses observed performance from instrumentation data for implementing preplanned design features or actions in response.

But there were a number of problems in applying this strategy to the Mount Polley dam that are treated in the following section. The first was simple geometry. The Observational Method relies on measuring the right things in the right places. While this was comparatively easy over the 1,000 m length of the Stage 1 dam, it became increasingly difficult as the length grew to 5 kilometres (km) by Stage 9. Nor could foundation instrumentation be installed beneath the dam crest and slopes where piezometric data mattered most. The slopes were too steep to be accessible, and few instruments installed on the crest could survive the near-constant construction there for very long. As a result, the few piezometers and inclinometers at the Perimeter Embankment were too far beyond the dam toe to produce critical data, and too far between to cover the area where the breach occurred.

Even more fundamentally, the piezometers as installed were only capable of measuring static ("water table") pore pressures and, if properly located, those induced by applied loads. But piezometers cannot measure pore pressures induced by undrained shearing because the location of the failure surface on which to measure them cannot be known in advance.

The remaining problem is that the Observational Method is useless without a way to respond to the observations. Constructing buttresses and obtaining the necessary mine waste had been hard enough under ordinary circumstances. Were the instruments to warn somehow of a rapidly developing failure, there would be no way to respond in time to avert it. Hence, the Observational Method could not be relied on to determine the need for buttressing, so the buttress would be required regardless.

This fact was belatedly recognized in the Stage 10 design just days before the breach—the final fateful instance of too little, too late.

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5.5 INSTRUMENTATION AND MONITORING

5.5.1 PRE-BREACH MONITORING OF TSF

Geotechnical instrumentation was installed beginning with Stage 1 of Main Embankment construction in 1996 and early 1997. During the initial phase, the focus was on vibrating wire piezometers, survey monuments, drain flow monitoring, and monitoring wells.[71] The first inclinometers on the Main Embankment were installed in July 2001. During the pre-breach period, instrumentation was installed at a total of 12 sections for the three embankments, Main, Perimeter and South (see Appendix F, Drawing F1). Further details of the inclinometers, piezometers, and drain flow monitoring during pre-breach monitoring are presented in Appendix F, Attachment 1.

A total of 10 inclinometers were installed after the start of operations. Of these, nine were still operating when the failure occurred: six at the toe of the Main Embankment and three along the toe of the Perimeter Embankment. One of the inclinometers along the Perimeter Embankment (SI11-04) was still being read, but was not reliable due to "a compression failure"[72] and had been replaced by Inclinometer SI12-04. Therefore, the Perimeter Embankment had two reliable inclinometers.

Vibrating wire piezometers were installed during ongoing construction activities at the 11 sections shown in Drawing F1. The last two sections (J and K) were added in 2011. As of August 2014, there were a total of 64 operating piezometers and 52 non-operating piezometers, of which 47 in the Main Embankment operated and 34 did not (see Appendix F, Attachment 1). Piezometers can fail not only due to instrumentation defects but also due to construction damage to piezometer cables. For example, during Stage 4 construction from May 2005 to October 2006, "22 piezometers were accidentally destroyed," of which five were repaired.[73] In contrast, a number of the piezometers installed in 1996 and 1998 were still operating in 2014.

Piezometers were installed in the dam foundation, in various embankment components, such as the upstream fill, core, and downstream transition zone, in drains located in the embankment and foundation, and in the tailings upstream from the embankment.

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The majority of the piezometers maintained steady pore pressures during 2014. Typical observations of piezometer pore pressure readings during construction were:

- Pore pressures in foundation piezometers typically increased due to fill placement and dissipated readily following construction.
- Pore pressures in piezometers located in embankment components (core and other downstream layers) and drains were stable.
- Pore pressures in tailings and upstream fill increased in response to the rising pool level.
- Piezometers located near the upstream toe drains experienced less pore pressure increases than those near the pond elevation.[74]

During the first phase of construction in 1996–1997, artesian pressures were observed in three of the six foundation piezometers in the Main Embankment. This prompted the development of trigger levels, or action levels, for many of the piezometers in the foundation and drains.[75]

As part of their annual construction manual in 2012, AMEC developed the instrumentation trigger framework shown in **Table F.1.1**,[76] Appendix F. This framework is for all the inclinometers and the Main Embankment foundation piezometers. The AMEC construction manual states that "embankment construction will be suspended if the inclinometers or piezometers fall under the yellow or red condition described in the Table, and/or if embankment foundation piezometer data indicates a significant increasing trend." No corresponding trigger levels were established for the Perimeter Embankment piezometers because "factor of safety values...are sufficiently high that monitoring of piezometric trends, without defined trigger levels, is deemed sufficient ."[77]

Drain flow of the foundation drains and chimney drain was measured for the Main Embankment during the first phase of construction. Flow measurements were also initiated when similar drains were installed in the Perimeter and South Embankments. Upstream drains were installed in the tailings (also referred to as "upstream toe drains"[78]) at all the dams as they progressed in height, and these flows were also measured starting in 1996. Flows from these drains report to the seepage collection ponds constructed downstream of each dam. These flows were measured monthly (weather permitting) in a manifold for the Main Embankment and across ditch profiles close to the ends of the outlet pipe for the Perimeter and South Embankments.[79] In Appendix F, drain flow readings are shown in **Figure F.1.2**, and these results are further discussed.

Survey monuments were used from Stage 1 construction until about 2010 to measure surface movements of the embankments. These were installed after completing the raise construction.

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5.5.2 PRE-BREACH MONITORING IN BREACH AREA

Locations of the inclinometers and piezometers in the breach area are shown in Appendix F, Drawing F2. In this area, one reliable inclinometer was located about 300 m east of the breach. It was about 15 m to 44 m from the toe of the embankment at the time of the failure. There were nine operating and 13 non-operating piezometers along this section of the Perimeter Embankment. Locations of all the piezometers are shown in the sections in Appendix F, Drawings F3 and F4.

The upstream toe drain in the tailings shown in Drawings F3 and F4 was located at El. 946.3 m. Seepage collection elements for the upstream toe drain are shown in Drawing F4. Flows were conveyed along a drainage ditch to the Perimeter Embankment seepage collection pond at the time of the breach.

5.5.3 PANEL KEY OBSERVATIONS

Section 5.2 clearly demonstrates that foundation conditions in the area of the breach were complex and that the Upper GLU layer was not continuous along the full length of the Perimeter Embankment. The foundation conditions assumed for the initial and ongoing design were based on only four drillholes deeper than 8 m, none directly in the area of the breach. A sentinel control section was therefore not identified, and instrumentation could not be installed to monitor this sentinel section.

Foundation piezometers could not be installed after the downstream slope was constructed at an angle of repose slope (1.3H:1V). Access to the slope was impossible, and piezometers installed from the crest into the foundation would not have been at the correct locations to measure increased pore pressure below the advancing downstream slope. Piezometers downstream from the dam toe (as at Section D) were too far away from the slope to provide any useful information, as was also the case for the inclinometer.

Foundation piezometers could not be installed after the downstream slope was constructed at an angle of repose slope (1.3H:1V).

The complex configuration of the internal embankment zoning made it very difficult, if not impossible, to install replacement piezometers in a specific fill zone at a specific depth.

The complex configuration of the internal embankment zoning made it very difficult, if not impossible, to install replacement piezometers in a specific fill zone at a specific depth. Most piezometers were installed during the construction phases, and many were damaged during those stages.

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While some piezometers provided very useful information (e.g., the tailings piezometers provided pore pressure values that could be applied to slope stability analyses), the Perimeter Embankment instrumentation overall could not have provided any warning of the looming failure. Nor did it provide any monitoring relevant to the critical failure mode.

It should be noted that if failure were to occur suddenly, deformation monitoring could not provide timely warning and a more defensive design would be appropriate. The failure mode encountered here was sudden without any surface evidence and is an example of this behaviour. In their design for the proposed Stage 10, BGC anticipated this issue and recognized that a berm would be required for the Perimeter Embankment.[80]

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5.6 WATER BALANCE

5.6.1 INTRODUCTION

A clear distinction can be made between the water balance actions and outcomes during the Phase 1 Active Mining, the Care and Maintenance period and the Phase 2 Active Mining. **Table 5.6.1** provides a summary of the mining activities, the Mine areas and the water management operating conditions. Appendix G describes in more detail the design objectives, water balance models and their implementation as well as observations found in the TSF Annual Inspection Reports. The consequences of the operational conditions are presented in this section.

TABLE 5.6.1: MOUNT POLLEY MINE LIFE

YEAR	ACTIVITY	MINE PITS	WATER MANAGEMENT OPERATING CONDITIONS
1997 – 2001	Phase 1 Active Mining	Cariboo and Bell	Deficit
2001 – 2005	Care and Maintenance		Neutral
2005 – 2014	Phase 2 Active Mining	Wight, Springer, Southeast Zone, Pond Zone	Surplus

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5.6.2 PHASE 1 ACTIVE MINING

From 1997 to 2001 MPMC mined the Cariboo and Bell pits. The area of disturbance in the mining area was quite small and the overall TSF water balance was in a deficit. Water from Polley Lake and surface runoff on-site helped to provide the annual operating requirements.

5.6.3 CARE AND MAINTENANCE

As a result of low copper prices, the Mine suspended operations from October 2001 to February 2005. A small staff was maintained at the Mine and they managed the TSF water balance carefully, making sure that sufficient freeboard was maintained. Towards the end of the Care and Maintenance period, mine development in preparation for start-up was underway and surface water accumulated in the TSF. It was recognized at this time that plans would have to be developed to discharge water to the environment.

5.6.4 PHASE 2 ACTIVE MINING

During the second phase of Active Mining, the footprint of the Mine was expanded to a total of four additional pits and associated infrastructure and waste rock piles. MPMC and the designers knew that there was a surplus of water in the TSF and that strategies had to be developed to discharge water. MPMC also understood the need for permitted discharge from the TSF.

In 2009 MPMC prepared a report entitled *Mount Polley Mine Technical Assessment Report for a Proposed Discharge of Mine Effluent.*[81] In this report, alternative discharge approaches were evaluated. The approach selected was to apply for a permit to discharge water to Hazeltine Creek. A permit amendment was granted on November 7, 2012 that allowed the discharge of up to 1.4 million cubic metres (m^3) per year of filtered water to Hazeltine Creek. The maximum discharge is 35% of flow in the Creek and the window is April to October. In April of 2014 it was estimated that only 170,500 m^3 total discharge was possible, due to constraints of permit requirements.

Discharging small amounts of extra water to Hazeltine Creek did not have a significant impact on the water surplus. Permitting of a water treatment plant was pursued in late 2013 and the Terms of Reference for Discharge was issued by the Ministry of Environment on March 26, 2014. Completion of treatment plant construction was expected in September 2014 or later. This plant would allow total annual discharge of 3 million m^3.

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5.6.5 WATER BALANCE AND TSF CONSTRUCTION

During the life of the Mine, two water balance models were used. The first was compiled by KP and was used from start-up until about 2005. The second was based on a model modified by MPMC to account for the expanded footprint of Phase 2 Active Mining. MPMC updated the water balance regularly with site-specific climatic and operating data as well as bathymetric surveys of the TSF pool. The EOR reviewed the water balances throughout operations except from 2010 to 2014. The Panel could not find any documentation explaining the reason for this change in procedures.

The embankment of the TSF was raised on a regular basis, typically on an annual basis. The design engineers used the outcome of the water balance calculations by MPMC to select the height of the increase. The overall approach was well summarized by KP in 2005 in their report entitled *Design of the Tailings Storage Facility to Ultimate Elevation*: [82]

Each embankment raise will provide incremental storage capacity for approximately one-year of production. The filling schedule incorporates sufficient live storage capacity for containment of runoff from the 24-hour PMP volume of 679,000 m^3 at all times, which would result in an incremental raise in the tailings pond level of about 0.39 m, with an additional allowance of 1 m for freeboard for wave run-up.

The water balance model included the site-specific information to the date of analysis, and future conditions were based on average climatic conditions. They did not account for specific wet year conditions.

Figure 5.6.1 shows the accumulation of water in the TSF as determined from bathymetric surveys. The figure also shows approximate volumes reported in the records for three dates (refer to Appendix G).

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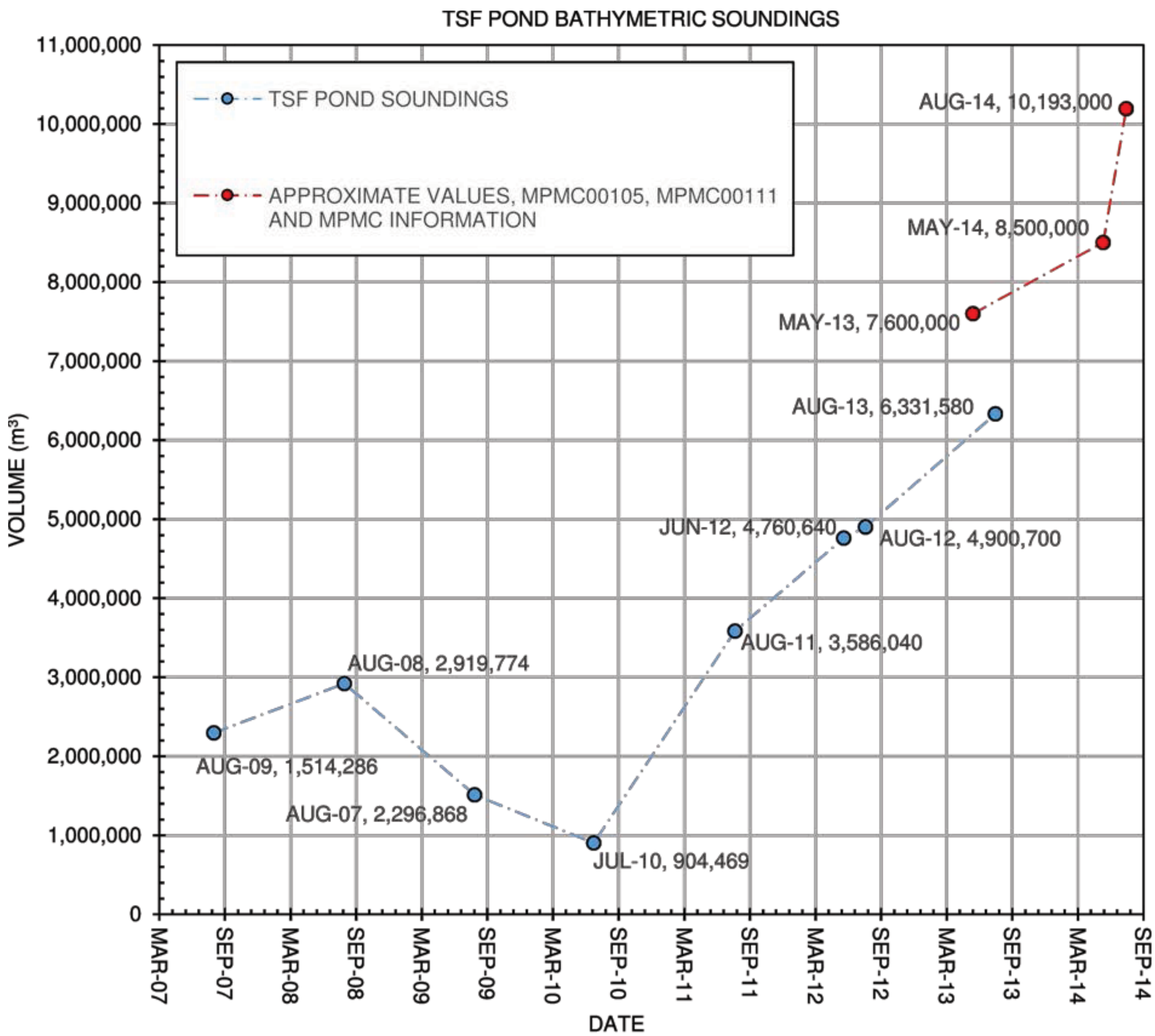
FIGURE 5.6.1: WATER ACCUMULATION IN TSF



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5.6.6 OVERTOPPING IN MAY 2014

On Saturday May 24, 2014, a potential "dam breach" event occurred at the TSF as a result of a large rainfall, approximately 24 mm in 24 hours, followed by ongoing rain. On Monday May 26 the water level was at El. 966.3 m, which resulted in a freeboard of 0.7 m to the top of the constructed core at El. 967.0 m, as stated in the 2013 Annual Construction Report (refer to Appendix G). The core was found to have a few low spots at 966.3 m (Corner 3), 966.4 m (Corner 2), 965.5 m (Corner 5) and 966.2 m (at the pipe crossing on the Perimeter Embankment). Wet spots and standing water were observed at Corner 3 and the pipe crossing, but no major erosion due to large flows or direct seepage. All the low areas were addressed through emergency construction measures by Thursday, May 29 when the pool water level increased to El. 966.45 m. The top of the Perimeter Embankment was increased to El. 967.3 m. All water collection systems were diverted from the TSF and water was routed for storage in the Cariboo Pit.

The pond elevation was monitored on a daily basis from the end of May until the time of the breach. During that time, construction proceeded to increase the embankment height. On August 3, 2014, the day before the breach, the freeboard was 2.3 m.

5.6.7 COMMENTARY

The way in which the water balance was utilized with annual raises had significant limitations.

The way in which the water balance was utilized with annual raises had significant limitations. Construction of annual embankment raises was based on water balance evaluations using average climatic conditions at the site. This does not provide a reliable approach to establishing adequate capacity for tailings and water storage. Uncertainties in the water balance input parameters combined with uncertainties in climatic conditions and construction schedules cannot provide a robust design for water containment. Construction delays due to site climate or availability of construction materials could impact the targeted capacity. Overtopping of the embankment occurred at selected locations in May 2014.

As indicated in section 4, the Perimeter Embankment did not fail due to overtopping; however, storing large volumes of water in the TSF had other implications.

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Throughout most of its term as the EOR, KP emphasized the importance of maintaining a beach width of at least 10 m. The Panel does not consider this to be a beach. Nevertheless, the principle was clear: the Mount Polley TSF embankments were not designed as water-retaining dams, and a beach would provide some stabilizing function. It was impossible to maintain beaches against all the embankments throughout the year during the last years of operation because of the large volumes of water stored in the TSF. Section 5.4 summarizes the chronic problems experienced in beach development.

MPMC was aware of the water surplus conditions at the start of Phase 2 operations. The pond volumes in **Figure 5.6.1** show that the last number of embankment raises were necessary to store water and not necessarily much higher tailings production. It is not clear to the Panel why it took so long to design and implement a water treatment strategy that would provide for a significant reduction in the amount of surplus water stored on the TSF.

It is not clear to the Panel why it took so long to design and implement a water treatment strategy that would provide for a significant reduction in the amount of surplus water stored on the TSF.

The pore pressure in the tailings piezometer at the breach location increased as a result of the higher pool elevation (refer to section 5.5). This happened despite the presence of the upstream toe drain. The higher pore pressure had a secondary effect on the overall slope stability.

Finally, the volume of water in the pool at failure, about 10 million m^3, resulted in a much larger loss of solids from the TSF due to erosion than might have occurred if there was a smaller pool (refer to Appendix C). And a wider beach of unsaturated tailings might have delayed breach development long enough for emergency actions to have been taken.

ENDNOTES

1) Klohn Crippen Berger, 2014, Site Investigation Progress Report No. 1, Report to BC Ministry of Energy and Mines.
2) Kulhawy, F.H. and Mayne, P.W., 1990, "Manual on Estimating Soil Properties for Foundation Design", Electric Power Research Institute (EPRI), EL-6800, 308 p.
3) MP10012
4) MP00054
5) MP10012
6) U.S. Dept. of the Navy, 1982, Soil Mechanics Design Manual 7.1, NAVFAC DM-7.1, 205 p.
7) MP00001
8) MP00005
9) MP00001
10) MP00054
11) MP00083
12) MP00069
13) MP00019
14) MP10009
15) MP00008
16) MP00012
17) MP00008
18) MP10032
19) Panel Interview, 12/12/14.
20) SUB00023
21) MP00011
22) MP00013
23) MP00014
24) MP00021
25) MP00118
26) MP00038
27) MP00021
28) MP00126
29) MP00026
30) MP00031
31) MP00076
32) MP00137
33) MP00139
34) Panel Interview, 12/12/14.
35) MP00028
36) MP00033
37) MP00077
38) MP00104
39) MP10035
40) MP00032
41) MP00036
42) MP00035
43) MP00222
44) MP00225
45) MP10034
46) MP00037
47) MP00181
48) MP00039
49) MP10012
50) MP00040
51) MP00217

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52) MP00042

53) MP00224

54) MP00043

55) Panel Interview, 10/22/14.

56) MPMC00048

57) MP00045

58) MP000187

59) MP00044

60) Panel Interview, 10/22/14.

61) Panel Interview, 10/22/14.

62) MPMC00048

63) BGC00002

64) BGC00003

65) MP00192

66) BGC00004

67) MP00208

68) Peck, R.B., 1969, Advantages and limitations of the observational method in applied soil mechanics. *Geotechnique*, v. 19, no. 2, pp. 171-187.

69) Kenney, T.C. and Lau, D., 1985, Internal stability of granular filters. *Canadian Geotechnical Journal*, v.22, pp. 215-225.

70) Kenney, T.C. and Lau, D., 1986, Internal stability of granular filters: reply. *Canadian Geotechnical Journal,* v.23, pp. 429-423.

71) MP00019, note that the monitoring wells were for water level and quality measurements outside the TSF; these will not be further discussed in this section.

72) AMEC00164

73) MP00031

74) MP00044

75) MP00019

76) MP00040

77) AMEC00132

78) MP00037

79) MP00044

80) BGC 00007

81) MOE00001

82) MP00026

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6 | Analysis of Breach Mechanics

6.1 INTRODUCTION

As demonstrated in sections 5.1 and 5.2, the breach of the Mount Polley Tailings Dam (the Dam) arose because of failure in the foundation of the Perimeter Embankment. According to Ministry of Energy and Mines (MEM) requirements, design with respect to overall stability must be compliant with CDA Guidelines. The specific guideline for a dam under construction and before reservoir filling requires a factor of safety (FS) of 1.3 where:

$$FS = \frac{\text{Available Strength}}{\text{Strength Required for Equilibrium}}$$

That is, the design requires a reserve resistance over and above that required to maintain equilibrium, and with this reserve resistance, it is expected that the structure will perform in a safe manner. This criterion has been accepted for tailings dams during construction, with a higher FS required if the dam has a long service life after it has been filled.

Many potential failure modes have to be considered to meet the requirements that FS = 1.3. The CDA Guidelines are not prescriptive with respect to potential failure modes. It is the obligation of the designer, as EOR, to recognize the potential failure modes, to characterize the operational strength of the materials associated with these potential failure modes, to adopt an appropriate method of analysis to calculate the FS, and to ensure that the FS is equal to or greater than 1.3 during the construction of the dam.

The Perimeter Embankment failed during construction, and hence the FS = 1. It moved sufficiently to lose containment of the impounded water and tailings that flowed out and eroded most of the displaced embankment.

In the following analyses, calculations will show that shear strengths determined by the Panel to reflect undrained failure of the Upper GLU beneath the Upper Till of the foundation are consistent with the strength required for limiting equilibrium, i.e., FS = 1.0.

A detailed explanation of the process leading to failure of the Dam will be presented, and comparisons will be made with the assumptions that underpin the design in order to highlight the deficiencies associated with it.

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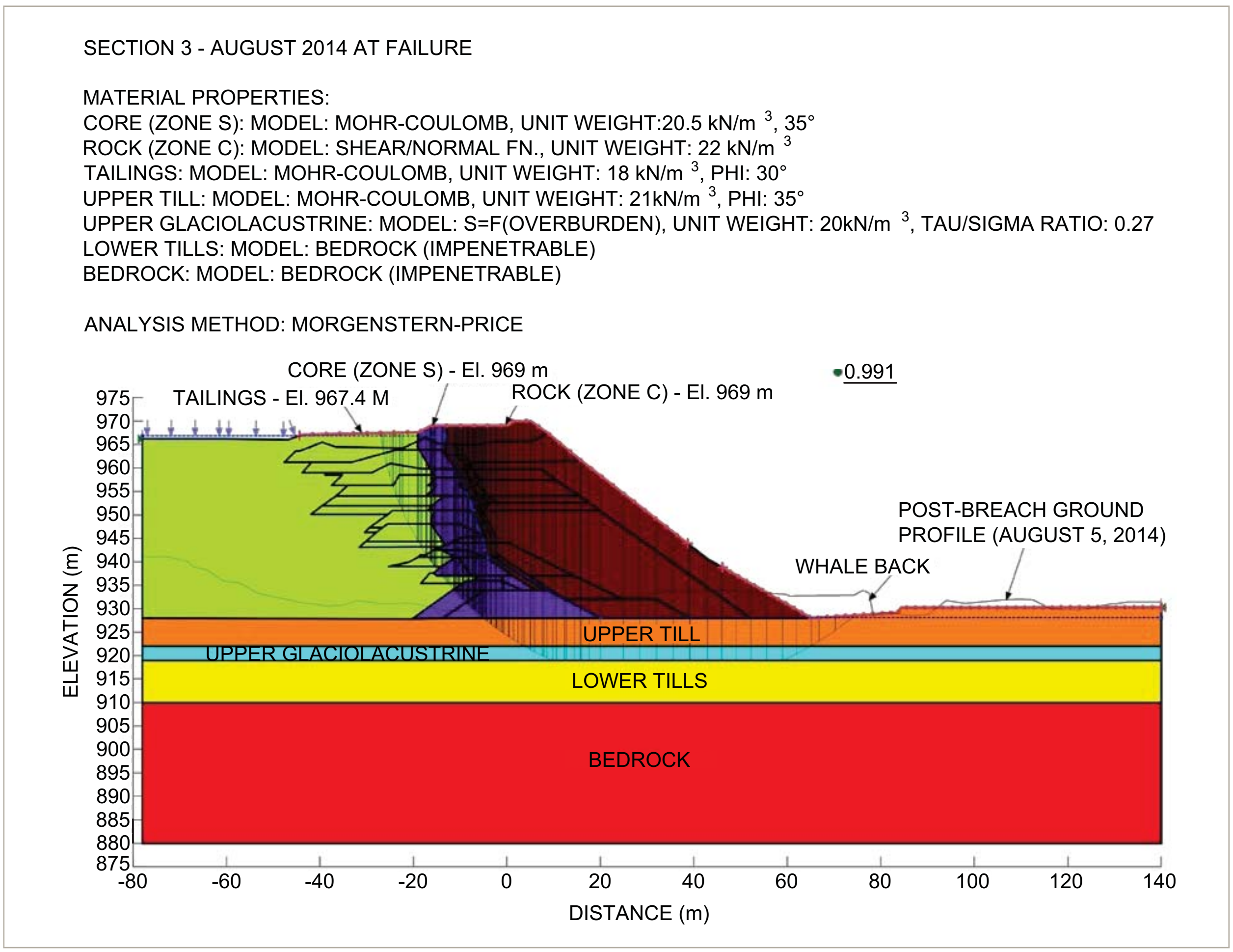
6.2 ANALYSES

6.2.1 LIMIT EQUILIBRIUM ANALYSES (2-D)

Analyses for purposes of designs are conventionally performed on two-dimensional (2-D) sections. Cross-section 3 (see section 5.2; Appendix D) was assumed to represent the stratigraphy more or less in the middle of the displaced mass.

Figure 6.2.1 presents the detailed section. It is based on the last LiDAR survey of the embankment prior to failure, a detailed reconstruction of the top of the structure and pond elevation based on construction inspector reports, and it includes a shallow excavation at the toe of the embankment as reported to the Panel. More details associated with the compilation of this and related sections are presented in Appendix H.

FIGURE 6.2.1: DETAILED SECTION USED FOR LIMIT EQUILIBRIUM ANALYSIS (HIGH WATER TABLE, UNDRAINED STRENGTH RATIO 0.27)



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6 | Analysis of Breach Mechanics

The Morgenstern-Price method of stability analysis and the SLOPE-W computer program were used for the computations. Both are recognized standard tools and were also used for the design of the structure at various stages.

Strength properties and densities for each stratum must be defined in order to calculate the FS. The values assumed are also displayed in **Figure 6.2.1**. Only the upper of the two GLUs defined in section 5.2 is included due to its high water content, its lower cone penetration testing (CPT) tip resistance, and overconsolidation ratio (OCR). Except for the strength of the Upper Till unit and the GLU, all strengths and densities are the same as those used in design studies. Based on pressuremeter testing, experience of the Panel members, and the limited pore pressure response during undrained triaxial testing, a frictional resistance of 35° was adopted for the Upper Till. Fully drained conditions are assumed up to failure. The magnitude of the undrained strength ratio in the GLU is then varied until FS=1 is obtained. For the case illustrated in **Figure 6.2.1**, this ratio is 0.27. In this case, the observed level of the pond is carried horizontally through the beach, which would, in most circumstances, be the design basis case (High Water Table case).

However, the installation of drainage at the upstream face of the core creates downward flow that will reduce the water pressure acting on the core of the Dam. A potential limiting case is shown in **Figure 6.2.2**, and the calculated strength ratio is 0.22 (Low Water Table case). The likely case is between these limits, with the Panel favouring a result above the average, say 0.25.

The calculated value represents the average resistance mobilized by the GLU at the instant of failure. It should be noted that this value lies sensibly in the middle of the range of the measured undrained strength (see **Table 5.3.1**), consistent with the hypothesis that the breach resulted from undrained failure of the GLU at an elevation of about 920 metres (m).

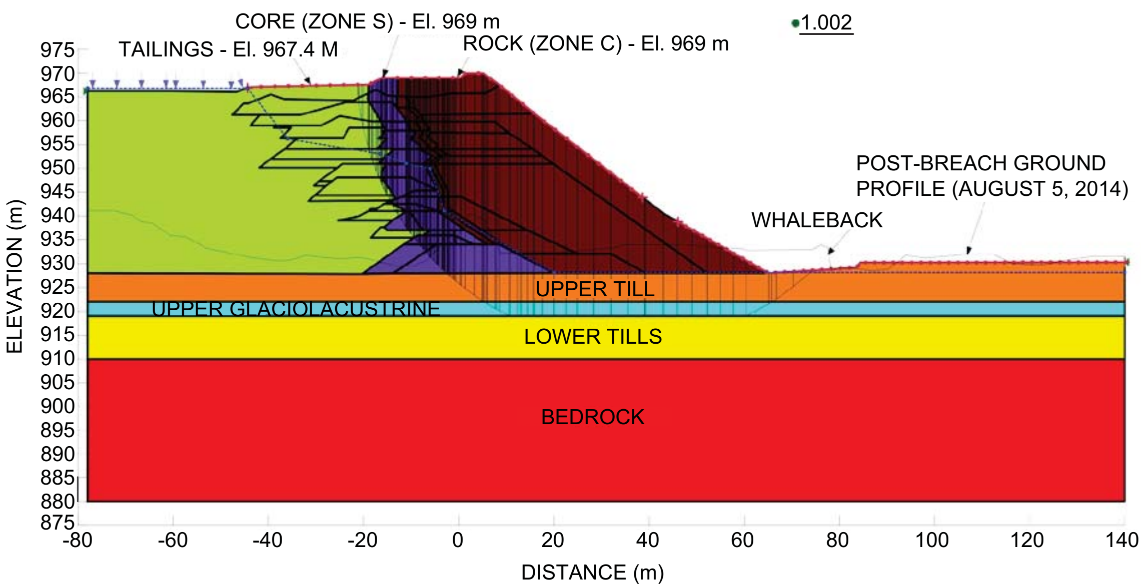
The actual available shear strength will vary with consolidation history as the Upper GLU responds to the stresses imposed by the embankment and the lateral loads transmitted by the impounded tailings and water. This will induce both normal and shear stresses in the Upper GLU. The Panel has not calculated these stresses in any detail. However, it is evident that the maximum applied stresses will substantially exceed the preconsolidation stress level associated with the Upper GLU. This response will reduce towards the breakout zone of the calculated slip surface and beyond, where the influence of applied stresses diminishes. Where the preconsolidation pressure has been overcome, the available shear strength will be that of a normally consolidated soil. Beneath the toe of the embankment and beyond, available strength will be higher, depending upon the local stresses and the preconsolidation stresses. The calculated average resistance reflects this distribution. Appendix E, Attachment 2 shows a comparison between vertical overburden stress and preconsolidation stress.

FIGURE 6.2.2: DETAILED SECTION USED FOR LIMIT EQUILIBRIUM ANALYSIS (LOW WATER TABLE, UNDRAINED STRENGTH RATIO 0.22)

SECTION 3 - AUGUST 2014 AT FAILURE

MATERIAL PROPERTIES:
CORE (ZONE S): MODEL: MOHR-COULOMB, UNIT WEIGHT:20.5 kN/m^3, 35°
ROCK (ZONE C): MODEL: SHEAR/NORMAL FN., UNIT WEIGHT: 22 kN/m^3
TAILINGS: MODEL: MOHR-COULOMB, UNIT WEIGHT: 18 kN/m^3, PHI: 30°
UPPER TILL: MODEL: MOHR-COULOMB, UNIT WEIGHT: 21kN/m^3, PHI: 35°
UPPER GLACIOLACUSTRINE: MODEL: S=F(OVERBURDEN), UNIT WEIGHT: 20kN/m^3, TAU/SIGMA RATIO: 0.22
LOWER TILLS: MODEL: BEDROCK (IMPENETRABLE)
BEDROCK: MODEL: BEDROCK (IMPENETRABLE)

ANALYSIS METHOD: MORGENSTERN-PRICE



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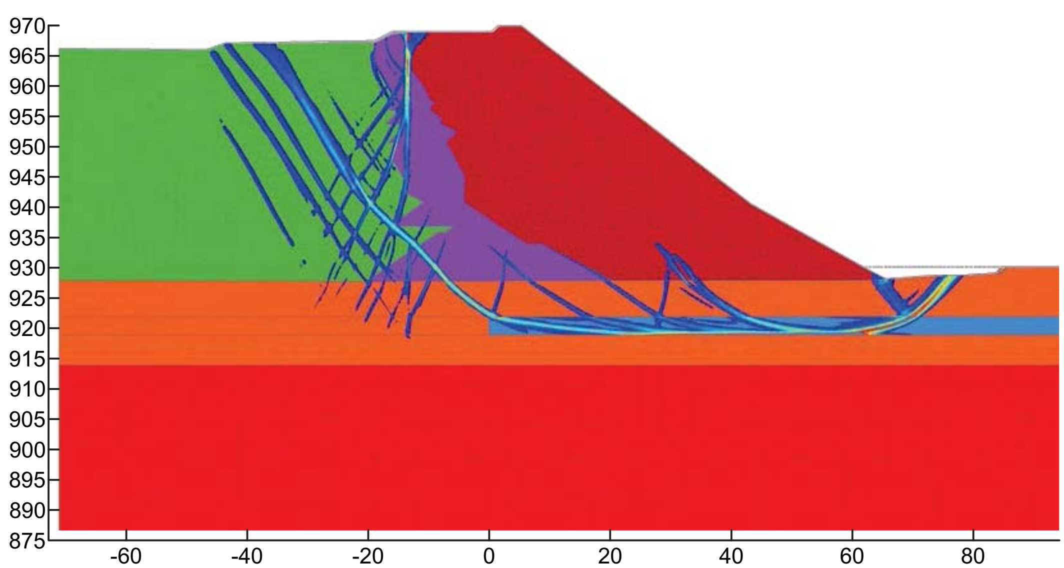
6.2.2 DEFORMATION ANALYSES (2-D)

An alternate way of assessing the undrained failure mechanism is to calculate the deformation patterns that develop at failure. While not a routine design procedure, the means for conducting such analyses are facilitated by powerful numerical simulation tools. In this case, PLAXIS, a well-recognized computer program developed specifically to model soil deformations, was adopted.

Figure 6.2.3 portrays the PLAXIS model at collapse. Prior to creating collapse, the model was constructed with essentially the same input parameters as used in the limit equilibrium analyses, except for the Upper GLU that is given a high strength to avoid yielding. The strength of the Upper GLU is then reduced until a deformation mechanism forms and the embankment collapses. This provides not only a measure of the strength of the Upper GLU at which failure occurs, but also an indication of the deformed shape arising from failure. In the model presented in **Figure 6.2.3**, collapse occurred at an undrained strength ratio of 0.29, which is to be compared with 0.27 calculated from the limit equilibrium analysis that incorporates the same boundary conditions. Lower undrained strength conditions would indicate significantly larger deformations. The figure also indicates the zones of localized strain that develop to facilitate motion. Variations of continuity of the Upper GLU with respect to this case yielded similar results.

FIGURE 6.2.3: PLAXIS MODEL AT COLLAPSE (UNDRAINED STRENGTH RATIO 0.29)



Both the entry and exit of the failure surface in the foundation correspond closely with field observations summarized in **Figure C4.2.2** in Appendix C.

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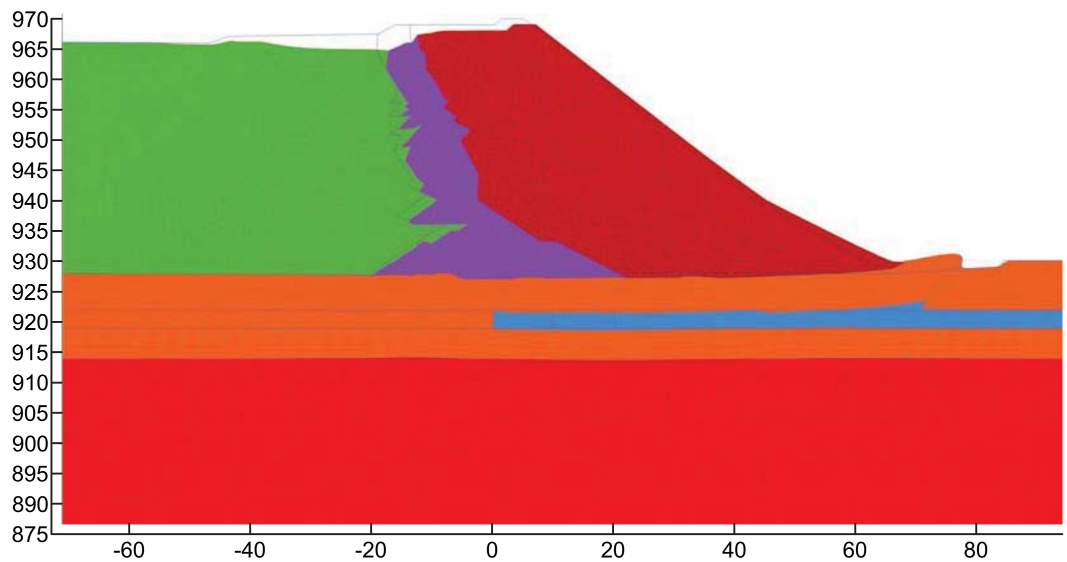
Figure 6.2.4 is a scaled-up display to illustrate the calculated deformations. The rotational movement with a lesser lateral displacement are evident. Particularly striking is the thrust feature that occurs very close to the whaleback feature identified in section 5.1. Also significant is subsidence of the crest that allowed overflow to begin, initiating the breach process as described in section 5.1 and Appendix C.

The PLAXIS analyses provide compelling support for the hypothesis that the movements of the Perimeter Embankment arose due to the undrained failure of the Upper GLU.

FIGURE 6.2.4: SCALED-UP FIGURE 6.2.3 TO ILLUSTRATE DEFORMATIONS



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6.2.3 LIMIT EQUILIBRIUM ANALYSES (3-D)

The length of the breach is relatively short compared to the height of the Perimeter Embankment at failure (~ 40 m). This is expressed as an Aspect Ratio (length/height) and is calculated to be 2.6. At this Aspect Ratio, three-dimensional restraints might be a significant factor influencing the analysis of the breach mechanism. At small Aspect Ratios, the side resistance acting on the potential moving mass increases in significance. This is ignored in the 2-D analyses described above, which are used routinely in design. Nevertheless, the Panel regarded it of value to assess three-dimensional considerations in order to fully explore the factors affecting the breach mechanism.

Three-dimensional limit equilibrium analyses have been conducted using the computer program SVSlope 3D, a widely accepted program for conducting such analyses. The geometry and boundary conditions are a three-dimensional extension of the case illustrated in **Figure 6.2.1**. All soil properties used in the 3-D analysis are the same as those employed in **Figure 6.2.1**.

Figure 6.2.5 presents the 3-D case. The FS with an undrained strength ratio of 0.27 and an Aspect Ratio of 2.6 is calculated to be 1.3. This is a significant increase over the 2-D case, and it merits interpretation.

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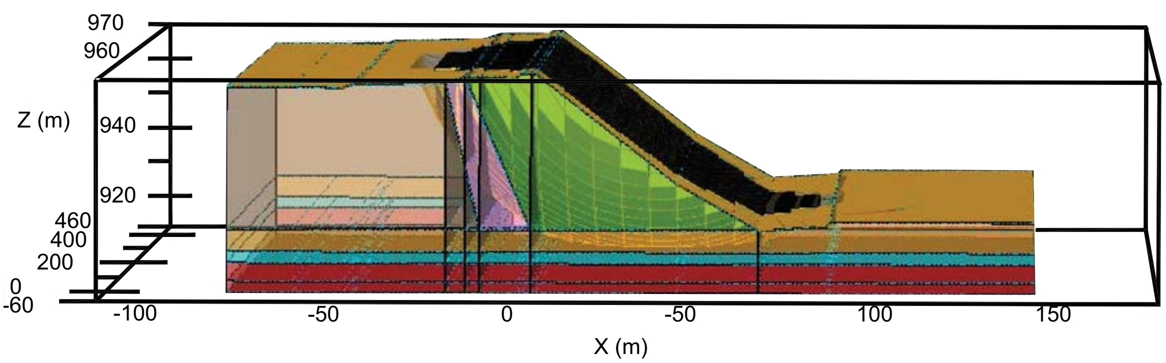
FIGURE 6.2.5: 3-D LIMIT EQUILIBRIUM ANALYSIS (UNDRAINED STRENGTH RATIO 0.27, ASPECT RATIO 2.6, FS IS ABOUT 1.3)

CALCULATION METHOD: M-P
SEARCH METHOD: ENTRY AND EXIT
FS: 1.288
CENTRE POINT: X: 37.591 Y: 200.000 Z: 981.233
ELLIPSOID ASPECT RATIO: 2.600, RX:70.168

MODEL



MATERIAL		MATERIAL PROPERTIES	
TAILINGS	MOHR COULOMB	UNIT WEIGHT = 18 (kN/m^3)	PHI=30°
UPPER TILL	MOHR COULOMB	UNIT WEIGHT = 21 (kN/m^3)	PHI=35°
UPP GLU	VERTICAL STRESS RATIO	UNIT WEIGHT = 20 (kN/m^3)	RATIO=0.27
LOWER TILLS	BEDROCK		
ZONE S	MOHR COULOMB	UNIT WEIGHT = 20.5 (kN/m^3)	PHI=35°
ZONE C	SHEAR NORMAL FUNCTION	UNIT WEIGHT = 22 (kN/m^3)	

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As deformations developed in the Upper GLU, the available resistance reduced due to strain weakening and soil remoulding.

As shown in Appendix D, Drawing D19, the sensitivities of the Upper GLU in the failure zone is about 1–3, based on CPT-measured tip resistances. Hence, as deformations developed in the Upper GLU, the available resistance reduced due to strain weakening and soil remoulding. Based on the observed sensitivity, it could have dropped to an undrained strength ratio of perhaps 0.13. Repeating 3-D limit analyses with these values yields an FS of about 1.1, which is close to collapse. Hence, as movements developed, the available resistance of the Upper GLU was reduced due to strain weakening to a degree that the three-dimensional restraints to movements at an Aspect Ratio of 2.6 were overcome. The idealizations involved in these 3-D analyses do not permit greater accuracy than expressed here. Going forward, a review of some of the assumed strength parameters that influenced the 3-D modelling and a more detailed representation of local geology that influence the 3-D results would be warranted. **Figure 6.2.6** displays visual evidence of the remolding processes that have occurred due to shearing of the GLU.

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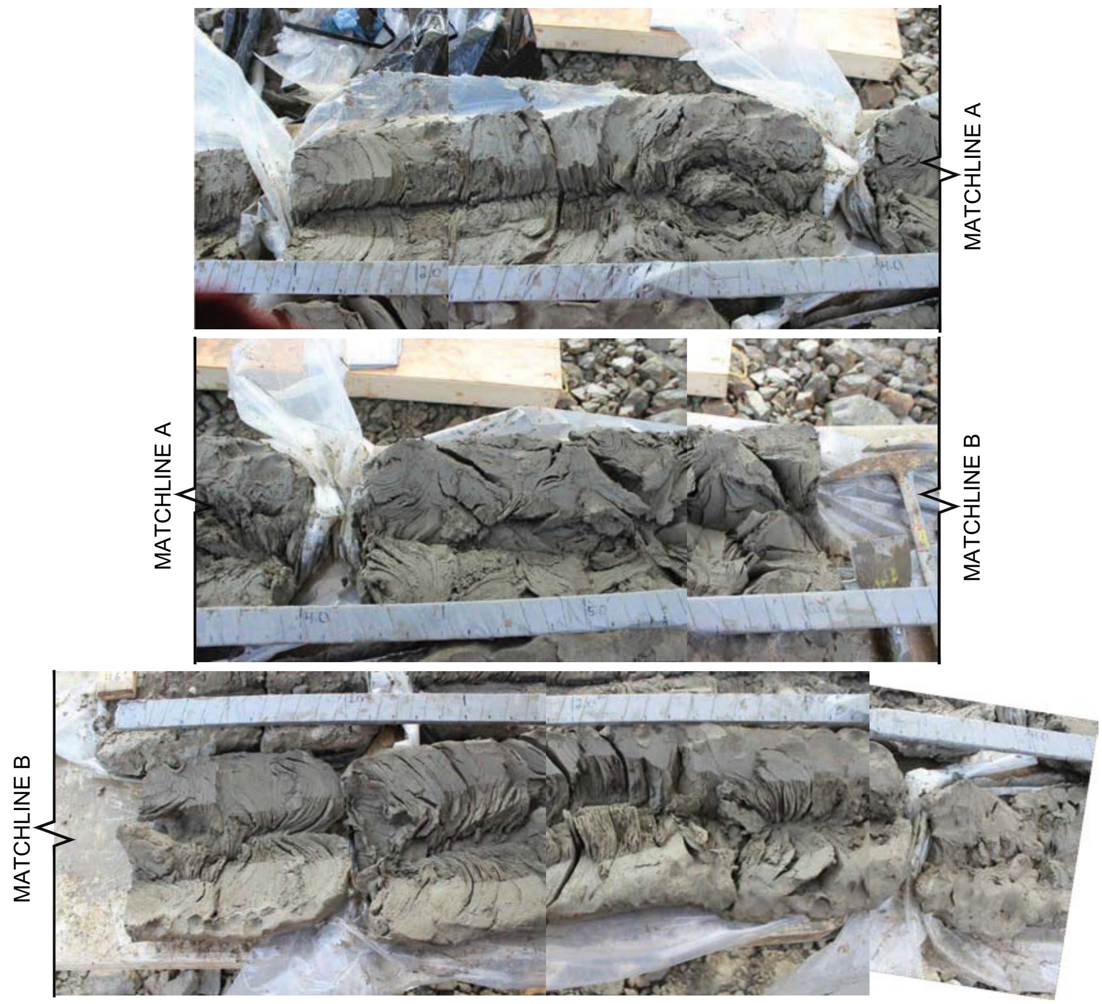
FIGURE 6.2.6: TYPICAL SHEARING IN THE UPPER GLU



UPPER GLU DEPOSIT, EXHIBITING HEAVILY DEFORMED AND CHAOTIC BEDDING INCLUDING OVERTURNED FOLDS.
NOTE: GREY SCALE BAR UNITS ARE DECIMAL FEET.

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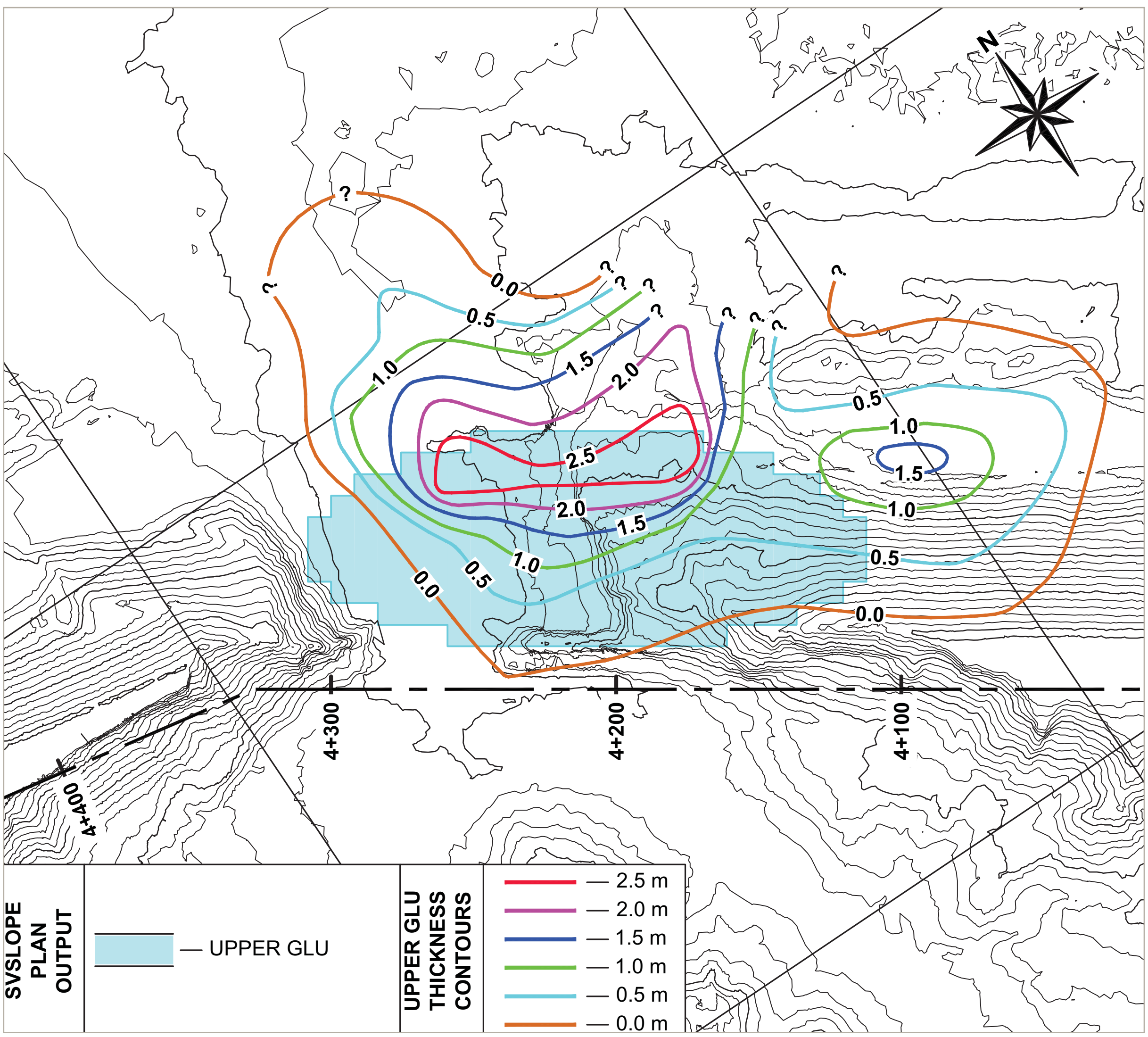
Additional support for the insight provided by the 3-D interpretation can be found by comparing the footprint of the 3-D analysis with the Aspect Ratio of 2.6 where it intersects the Upper GLU. The distribution of the thickness contours of Upper GLU is presented in **Figure 5.2.6**. This comparison is shown in **Figure 6.2.7**, which indicates a striking fit between the extent of the Upper GLU mobilized in the 3-D analysis (shown in cyan) with the extent of the deepest portion of the Upper GLU.

FIGURE 6.2.7: COMPARISON OF THE 3-D ANALYSIS WITH THE THICKNESS CONTOURS OF THE UPPER GLU



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6.3 TRIGGER ANALYSIS

6.3.1 INTRODUCTION

Both the 2-D and 3-D analyses discussed above reflect a simplified interpretation of how failure began and subsequently progressed. They indicate that the foundation was brought to failure under fully drained conditions until the undrained strength was reached and the collapse of the embankment subsequently mobilized the undrained shear strength. After initial failure, the Upper GLU behaved in a strain-weakening manner, reducing its resistance as reflected by the observed sensitivity of the deposit. Ultimately, the increased load associated with the weakening material overcame the residual resistance of the stronger zones, allowing the unconstrained 3-D mechanism to develop. The calculations presented provide average undrained strength ratios at failure that are generally consistent with the magnitudes observed in the laboratory.

In order to understand the failure mechanism in more detail, it is of value to address two questions:

1) **Was the loading path to failure fully drained?**
2) **Was the shear strength at failure mobilized uniformly?**

6.3.2 PORE PRESSURE HISTORY

To address the first question, it is possible to calculate the pore pressure development and dissipation during embankment construction. If the pore pressures remain high, the available shear strength is reduced accordingly. This type of evaluation is an integral part of any stability assessment involving stage construction on soft constructed soils, such as are present beneath the breach zone.

Calculations involve the estimates of stresses on a structure, the magnitude of pore pressure reaction, and its subsequent dissipation with time as construction proceeds through the various stages to completion. The data obtained from consolidation testing (see Appendix E) are used to calculate the rate of pore pressure dissipation. Pore pressures dissipate as a result of water flow to drainage boundaries, and in the case of Upper GLU, dissipation will be enhanced by horizontal flow reflecting the laminated structure of the Upper GLU. Details of the pore pressure predictions for both one-dimensional (vertical only) and two-dimensional (vertical and horizontal) water flow are presented in Appendix H. In the latter case, some estimates of anisotropy of the flow parameters have also been made.

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The calculated values at the time of failure suggest that an average excess pore pressure of about 50 kPa might exist in the potential shear zone. This is a small percentage of the applied load and, if it does exist, is not particularly consequential. Moreover, in the experience of the Panel, laboratory tests tend to underestimate the coefficients of consolidation in place due to scale effects, and it is likely that the potential for lateral drainage in the analyses due to stratigraphic variations has been underestimated. The Panel concludes that the loading path to failure has been essentially drained with transient episodes of undrained loading. The small peak of pore pressure development beneath the crest of the embankment in 2014 may have had some impact on the ultimate trigger, as the embankment was close to failure at this time.

Loading the Upper GLU to failure under predominantly drained conditions also implies the imposition of shear stresses as well as vertical stresses. As shown in Appendix E, Attachment 5, consolidating specimens under a shear stress not only has an effect on available resistance, but also reduces the subsequent tolerable strain to failure. Given the high stresses that acted on the Upper GLU prior to the final construction campaign in 2014, it would have taken only a small additional load to initiate undrained failure, and little incremental deformation. This is consistent with the collapse of the embankment without any apparent warning.

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6.3.3 PROGRESSIVE FAILURE

The strength at failure will only be mobilized uniformly if it does not vary with deformation. Failure will begin initially at a position where the local stresses equal the strength. As additional load is applied, yielding spreads to adjacent locations because resistance is limited at locations that have already yielded. This spreading of the yield zone migrates until a failure mechanism develops and unrestrained movement occurs with mobilization of a uniform shear strength.

However, as emphasized in section 5.3, the Upper GLU exhibits strain-weakening behaviour. After yielding has been initiated, the local resistance reduces with increasing load, requiring stress transfer to accommodate not only the influence of additional externally applied load, but also the influence of the reduced capacity of already failed material to resist the applied stresses. The transfer process proceeds to ultimate failure, but the average resistance at ultimate failure is less than the peak resistance.

This process is known as progressive failure. Once progressive failure has been initiated, the development of ultimate collapse can be sudden, depending on the shape of the whole stress-strain relation. As noted in section 6.2.3, the observed sensitivity of the Upper GLU indicates that it might display an ultimate resistance of one-half to one-third of its peak value.

While the mechanics of progressive failure are generally understood, the ability to calculate it is a complex undertaking and is generally reserved for research endeavours or other special studies. Progressive failure analyses have not been undertaken in this study, but Lobbestael et al. (2013) provide a useful overview and example of how progressive failure calculations might be performed. The Panel is of the view that progressive failure was involved in the initiation of collapse of the Perimeter Embankment and subsequent motion. Its influence is embedded in the back-calculated average resistance.

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6.4 FAILURE MECHANISM

The Panel's Terms of Reference require it to: "report on the cause of the failure of the tailings storage facility at the Mount Polley Mine."

The failure of the tailings storage facility (TSF) was caused by deformation of the Perimeter Embankment that allowed the containment to be breached between survey stations 4+200 and 4+300. The deformation arose because of inadequate resistance of a continuous layer of glaciolacustrine clays (Upper GLU) that existed at about El. 920 m, beneath the overlying till. The GLU deposit had properties that became increasingly contractive when sheared, following consolidation under the applied embankment loads to a normally consolidated state. This made the Upper GLU disposed to undrained failure. Moreover, the Upper GLU exhibited strain-weakening properties when sheared, such that overall resistance of the formation reduced as deformation developed, ultimately overcoming all of the resistance of the stabilizing elements in the section. Hence, the root cause of the breach was the undrained failure of the Upper GLU under the imposed load of the Perimeter Embankment on August 4, 2014.

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6.5 CAUSES OF FAILURE

As outlined in the Terms of Reference, it is expected that the Panel will "identify any technical, management, or other practices that may have enabled or contributed to the mechanism(s) of failure. This may include design, construction, maintenance surveillance and regulation of the facility."

The dominant contribution to the failure resides in its design. The design did not take into account the complexity of the sub-glacial and pre-glacial geological environment associated with the Perimeter Embankment foundation. As a result, foundation investigations and associated site characterization failed to identify a continuous GLU layer in the vicinity of the breach and to recognize that it would be disposed to undrained failure when subjected to the stresses associated with the Dam.

The design did not take into account the complexity of the sub-glacial and pre-glacial geological environment associated with the Perimeter Embankment foundation.

At the time of Stage 4 (2006 – 2007), Knight Piésold (KP) had proposed a design for the Perimeter Embankment with a 2H:1V downstream slope and raises of the core and filter with a parallel inclined alignment to El. 965 m. This design has been projected in **Figure 6.5.1** to the core elevation at the time of failure (El. 969 m), and adopting an undrained strength ratio of 0.27 and a high water table, the calculated FS is 1.02. At El. 965 m, the FS is 1.04, much less than the design target of 1.3. Based on the back-calculated undrained strength ratio, the design was doomed to fail.

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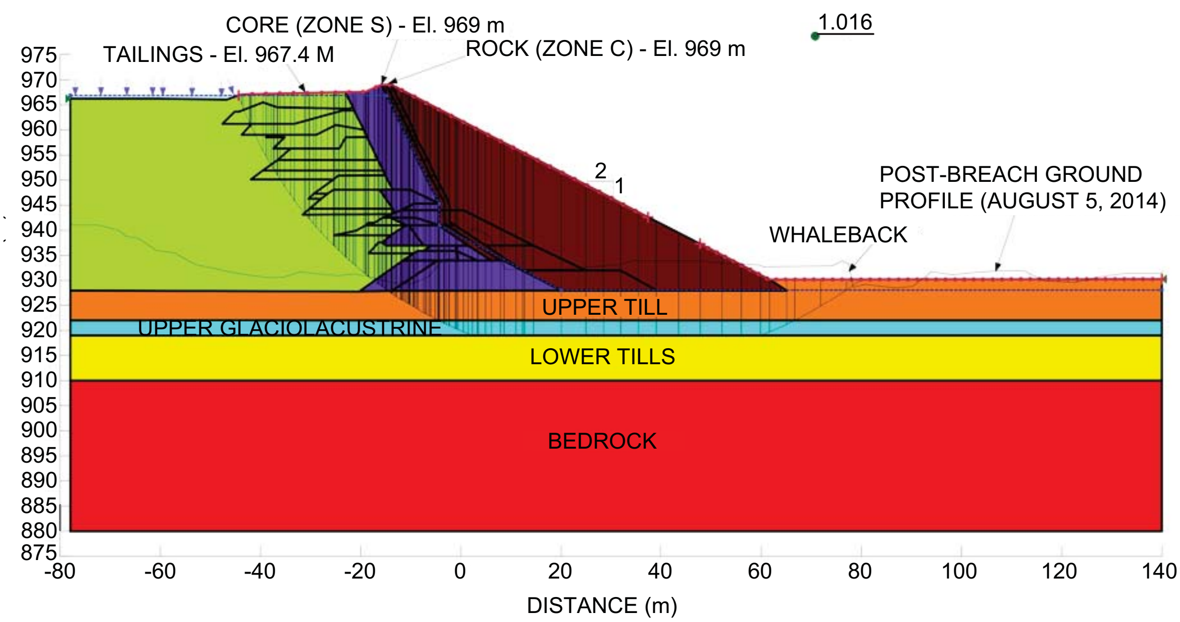
FIGURE 6.5.1: 2-D LIMIT EQUILIBRIUM 2H:1V SLOPE TO ELEVATION 969 M (UNDRAINED STRENGTH RATIO 0.27, HIGH WATER TABLE, FS 1.02)

SECTION 3 - AUGUST 2014 AT FAILURE

MATERIAL PROPERTIES:
CORE (ZONE S): MODEL: MOHR-COULOMB, UNIT WEIGHT:20.5 kN/m^3, 35°
ROCK (ZONE C): MODEL: SHEAR/NORMAL FN., UNIT WEIGHT: 22 kN/m^3
TAILINGS: MODEL: MOHR-COULOMB, UNIT WEIGHT: 18 kN/m^3, PHI: 30°
UPPER TILL: MODEL: MOHR-COULOMB, UNIT WEIGHT: 21kN/m^3, PHI: 35°
UPPER GLACIOLACUSTRINE: MODEL: S=F(OVERBURDEN), UNIT WEIGHT: 20kN/m^3, TAU/SIGMA RATIO: 0.27
LOWER TILLS: MODEL: BEDROCK (IMPENETRABLE)
BEDROCK: MODEL: BEDROCK (IMPENETRABLE)

ANALYSIS METHOD: MORGENSTERN-PRICE



Hence, the omissions associated with site characterization may be likened to creating a loaded gun. Notwithstanding the large number of experienced geotechnical engineers associated with the TSF over the years, the existence of this loaded gun remained undetected.

The omissions associated with site characterization may be likened to creating a loaded gun.

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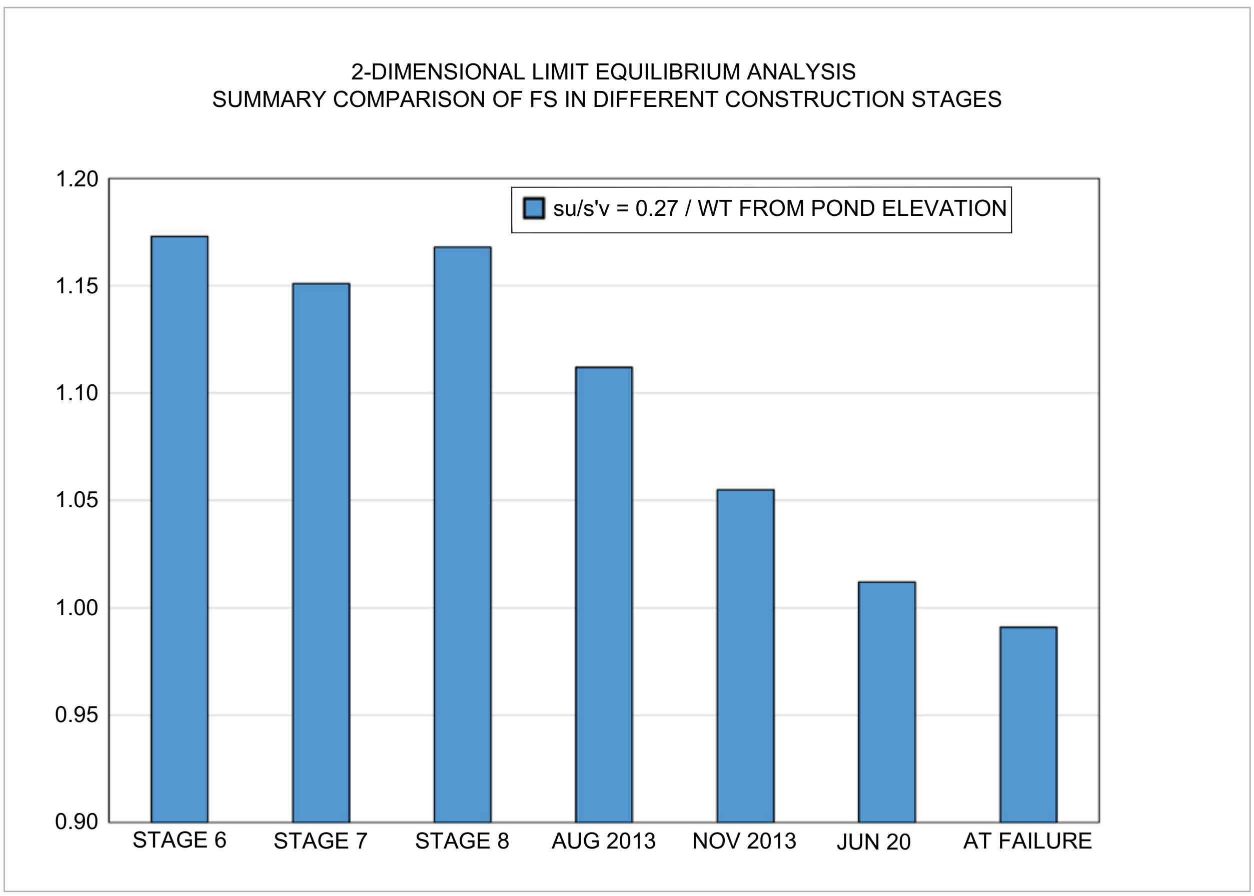
The lack of recognition of a critical potential failure mode resulted in a misapplication of the Observational Method and, therefore, a false appreciation that the structure was performing as intended during stages of raising. The Observational Method is a powerful tool to manage uncertainty in geotechnical practice. However, it relies on recognition of the potential failure modes, an acceptable design to deal with them, and practical contingency plans to execute in the event observations lead to conditions that require mitigation. The lack of recognition of the critical undrained failure mode that prevailed reduced the Observational Method to mere trial and error.

Figure 6.5.2 shows the variation of the calculated FS with each stage, from Stage 6 to failure, based on the as-built section for each stage. El. 965 m corresponds approximately to the height of the structure at the end of the 2013 construction season. At this stage, the FS is calculated to be only about 1.05, which is similar to the FS for the original design with a 2H:1V slope.

FIGURE 6.5.2: VARIATION IN FS FOR EACH STAGE FROM STAGE 6 TO FAILURE



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6 | Analysis of Breach Mechanics

> **If constructing unknowingly on the Upper GLU...constituted loading the gun, building with a 1.3H:1V angle of repose slope over this stratum pulled the trigger.**

Prior to 2014, the Zone C fill began to be constructed as an angle of repose slope of 1.3H:1V. This appeared to have been an expedient measure and, as illustrated in **Figure 6.5.2**, ultimately resulted in failure of the Perimeter Embankment on August 4, 2014. If constructing unknowingly on the Upper GLU stratum, and not recognizing the potential undrained failure constituted loading the gun, building with a 1.3H:1V angle of repose slope over this stratum pulled the trigger. It appears that the 1.3H:1V slope began as an expedient temporary measure to facilitate construction during Stage 5. It became more or less permanent for subsequent phases, although concerns had been raised before the failure. The circumstances associated with the relative permanency of the 1.3H:1V slope are not well understood by the Panel. The complex issues that prevailed during construction are summarized in section 5.4. **Figure 6.5.3** indicates that, had the downstream slope incorporating the widened crest been flattened to 2H:1V, the FS would have been 1.28. This was close to the required value of 1.3, and the embankment would not have failed. Moreover, the slope of 2H:1V was required, in any case, to support reclamation and closure criteria.

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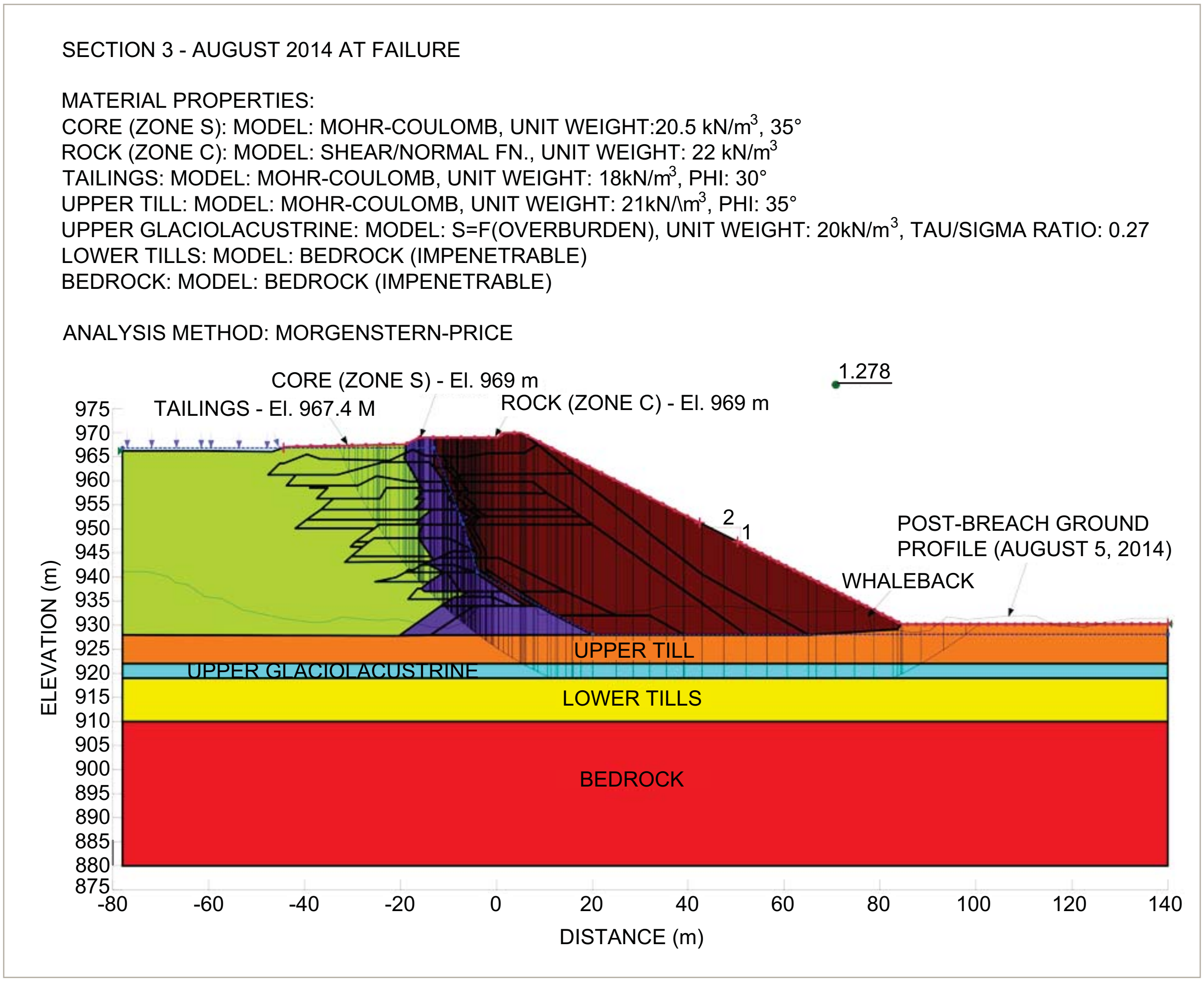
FIGURE 6.5.3: PANEL'S STABILITY ANALYSIS FOR 2H:1V SLOPE WITH WIDENED CREST (FS = 1.28)



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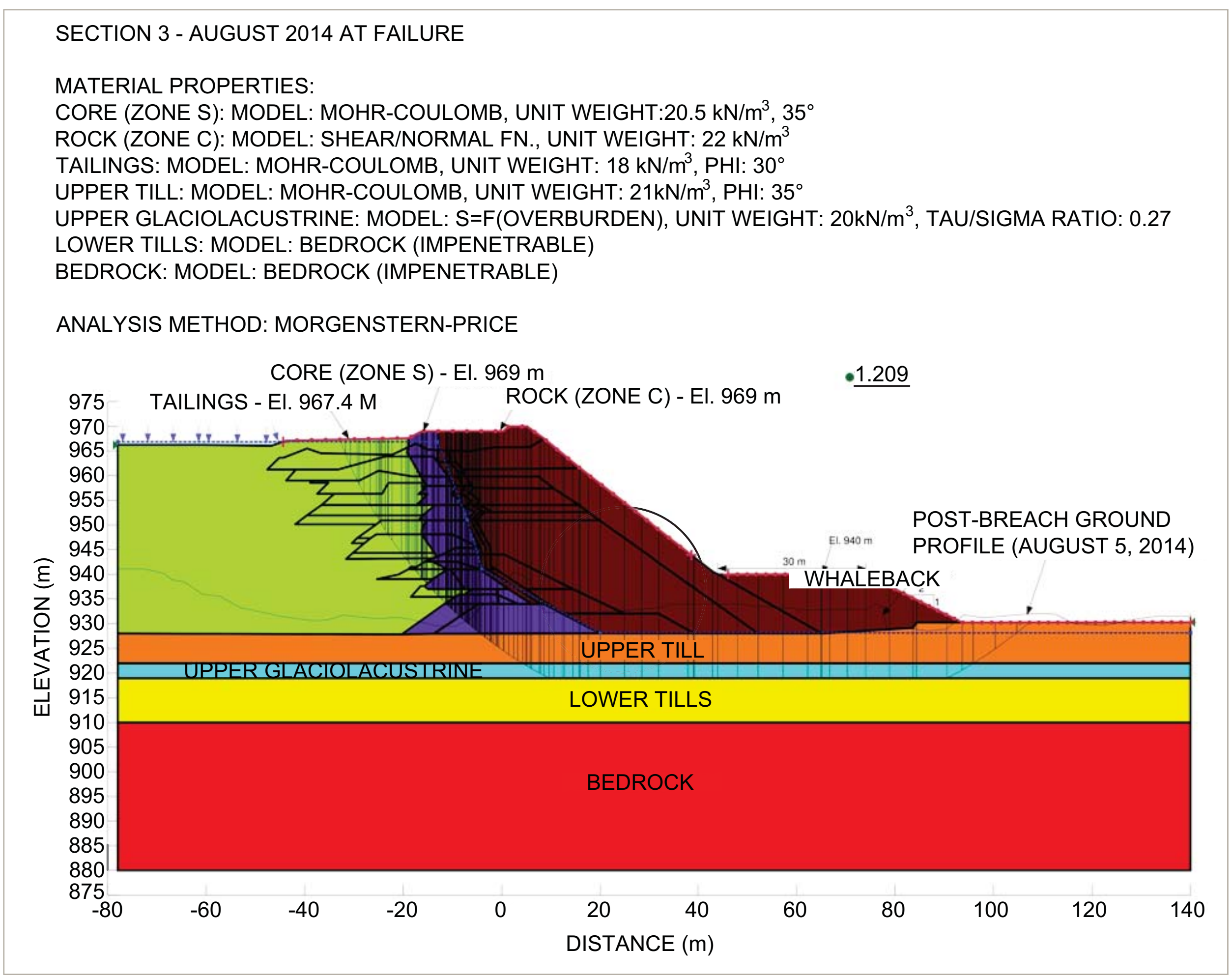
6.6 PREVENTION OF FAILURE

The Terms of Reference (TOR) authorize the Panel to comment on "what actions could have been taken to prevent this failure."

Looking specifically at the failure as documented in section 5.4, it was deemed desirable to increase the target FS to 1.5 since the TSF was operating more or less continually at full capacity. No significant progress to this end was made in Stage 9 before failure occurred. BGC's design report for Stage 10, issued on July 25, 2014, indicated the buttress required to meet the new design objectives that they identified. Had it been in place as shown on **Figure 6.6.1**, the FS would have been 1.2 and the failure would have been prevented.

FIGURE 6.6.1: PANEL STABILITY ANALYSIS FOR BGC BUTTRESS ON STAGE 9



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7 | Management Practices

The Panel is cognizant that management practices have had a significant influence on the design, construction and operation of the tailings storage facility (TSF). For example, the Panel has already drawn attention to water balance protocols and the growth of water inventory in the TSF due to the timing associated with the implementation of water treatment and discharge. It has pointed out that the recurrent adoption of a 1.3H:1V downstream slope for the Perimeter Embankment may have been due to limited material availability or other aspects related to mine planning. The details are not clear. What is clear is that multiple changes were made in the section of the dam in response to the limited time horizons adopted in mine and water planning.

The Panel has been advised that Mount Polley Mining Corporation (MPMC) were in the midst of becoming Mining Association of Canada (MAC) compliant and that tailings management issues were reported to the Board of Directors. It has not identified any flaws in this reporting structure.

However, in conducting its inquiry, the Panel limited itself to relying on interviews and on the documents that it received from the various stakeholders, which were sufficient to determine root cause of the breach. The Panel did not conduct its process according to formal legal procedures. To do so would have extended the length of this investigation and would have entered into an assessment of roles and responsibilities, which is beyond the Panel's authorization. As a result, the Panel is not able to offer an adequate assessment of the role of management and oversight in its contribution to the cause of the failure. In particular, the Panel has not explored the relationship between the designers and owner, contractual or otherwise. Accordingly, the Panel is unable to ascertain the circumstances that contributed to key decisions.

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8 | Regulatory Oversight

8.1 ROLES AND RESPONSIBILITIES

This section describes the regulatory roles and responsibilities for impoundments and diversions at mines in B.C. A Memorandum of Understanding (MOU) is in place between the Ministry of Energy and Mines (MEM), the Ministry of Forests, Lands and Natural Resource Operations (MFLNRO) and the Ministry of the Environment (MoE) to clarify the regulation of these facilities. This MOU and other documents related to Mine Tailings are available on the Geotechnical page of the MEM website:

http://www.empr.gov.bc.ca/MINING/PERMITTING-RECLAMATION/GEOTECH/Pages/default.aspx

The MOU clearly places the responsibility for the engineering aspects of the Mount Polley tailings storage facility (TSF), seepage collection ponds and diversions on the shoulders of MEM, while the water quality of any discharges is the responsibility of MoE. Two permits are in place for the TSF and associated facilities: Permit M-200 from MEM and Permit 11678 from MoE.

MEM permits are issued by the Chief Inspector of Mines of B.C. The Manager of Geotechnical Engineering and the Manager Environmental report to the Deputy Chief Inspector of Mines, Permitting. The Manager Geotechnical Engineering has a staff of two geotechnical engineers and one reclamation specialist, while the Manager Environmental has a staff of three geoscientists. This staff of eight is responsible for inspection of operating mines and permitting of new mines in B.C. Apart from TSF-related activities, they also have regulatory responsibility for open pits, underground workings, and mined rock and overburden piles. The Geotechnical Manager and staff also participate in secondary activities including the Canadian Dam Association (CDA) Regulatory Committee and coordination with the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC) in development of the *Professional Practice Guidelines for Dam Safety Reviews* for mining dams in B.C.[1] The latter publication is available on the above-mentioned website.

The ongoing activities of the geotechnical staff include:

- Review of geotechnical aspects of proposed mining projects in the Environmental Assessment process.
- Review of geotechnical aspects of *Mines Act* Permit applications during the approval and permit conditions development process.
- Review of permit amendment applications for dam raises, mine expansions, etc.
- Geotechnical site inspections of operating and closed mines.
- Review of geotechnical reports submitted under the Code, including annual dam safety inspections for mining dams and diversions.

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Filling the positions at MEM has been challenging at times. The present Manager of Geotechnical Engineering joined MEM in October 2011 following a period of over 3 years when the position was open. A senior geotechnical position was made redundant in 2003, but a new geotechnical inspector position was created in 2007. This position was also vacant for about 2 years until filled in September 2012. A third position was posted in May 2014 and filled in early October 2014. To attract qualified personnel, MEM has to compete with industry salaries, which is a challenge, especially during a booming mining cycle. To help accomplish these tasks, MEM has appointed four consulting professional engineers as Contract Inspectors to inspect tailings dams and other mining facilities.

An annual inspection schedule is developed for all the inspectors. The target is to inspect about 30 mines on an annual basis. Mount Polley is one of these mines.

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8.2 REGULATORY INTERACTIONS RELATED TO MOUNT POLLEY MINING CORPORATION (MPMC) TAILINGS STORAGE FACILITY (TSF)

Table 8.2.1 lists the dates of the geotechnical inspections completed at Mount Polley from 1995 to 2014. Annual inspections were completed during the Phase 1 operations and were resumed after start-up of Phase 2 operations until 2008. There were no geotechnical inspections during 2009, 2010 and 2011, which is the same period as the vacancy of the Geotechnical Manager's position.

TABLE 8.2.1: GEOTECHNICAL INSPECTIONS AT MOUNT POLLEY

DATE OF INSPECTION	TYPE OF INSPECTION PERFORMED	INSPECTOR
Sept 20, 1995	Geotechnical	G. Headley
Oct 19, 1995	Geotechnical	G. Headley
Oct 19, 1995	Geotechnical	C. Brawner
July 9 and 13, 1996	Geotechnical	G. Headley
Aug 26, 1996	Geotechnical	G. Headley
Sep 27–28, 1996	Geotechnical	G. Headley
May 27, 1997	Geotechnical	G. Headley
Jun 4, 1998	Geotechnical	G. Headley
Jun 17, 1999	Geotechnical	G. Headley
Aug 17, 2000	Geotechnical	G. Headley
April 25, 2001	Geotechnical	C. Carr
Feb 3, 2005	Geotechnical	C. Carr
Oct 13, 2005	Geotechnical	N. Rose
Aug 30, 2006	Geotechnical	N. Rose
July 31, 2007	Geotechnical	N. Rose
Jun 7, 2008	Geotechnical	D. Apel
Apr 12, 2012	Geotechnical – site visit	G. Warnock
Sept 24, 2012	Geotechnical	M. Cullen
Sept 13, 2013	Geotechnical	M. Cullen
Dec 4, 2014	Geotechnical	M. Cullen

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Most of the inspection reports did not identify any concerns with the TSF, except in the following cases. Based on the inspection of April 25, 2001 the inspector observed: *"The Ministry would strongly support the installation of two slope inclinometers at the downstream toe buttress to monitor potential dam and/or foundation movement. The slope inclinometers should extend through the underlying glaciolacustrine sediments."* [2] MPMC responded that this matter was forwarded to Knight Piésold (KP). [3] These inclinometers were installed in July 2001 (refer to Appendix F).

On October 13, 2005, [4] narrow beach widths were observed on the southwest side of the pond. On August 30, 2006, [5] wide beach widths were observed and MEM requested a specific specification for beach width. MPMC responded, quoting KP: [6] *"The tailings embankments have been designed to remain stable for any condition and therefore there is not a 'requirement' for a minimum beach width in terms of embankment performance."*

On July 31, 2007, [7] the inspection found two concerns that were Departures from Approval. First, Zone S material contained particles as large as 12 inches, which had to be removed to satisfy the specification of 4 inches. In addition, there was no beach in the vicinity of the southeast corner and MPMC was told that the beach must be re-established and that more frequent monitoring of the piezometers must be conducted in that area.

While the examples above illustrate the role of the Regulator in matters of construction and performance, the Regulator also reviewed design. The following design-related issues were brought up by the Regulator:

- The shear strength associated with the lacustrine materials in the well log GW96-1A. [8]
- Testing on residual strength. [9, 10]
- Need to migrate factor of safety (FS) from 1.3 to 1.5. [11]

In each case, the Engineer of Record (EOR) responded to the inquiries and these instances illustrate the limited ability of the Regulator to influence the design issues.

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8.3 PANEL ASSESSMENT

The roles and responsibilities of the Ministry of Energy and Mines (MEM) to regulate impoundments and diversions at mines are well defined and agreed upon with other Ministries. Within MEM, the roles and responsibilities of the geotechnical engineering group responsible for regulating the design, construction and operational aspects of TSFs are also clearly defined. This small group of professionals covers a large portfolio of existing facilities, permitting of new facilities and environmental assessments for proposed projects.

The Panel finds that the MEM Geotechnical Staff and the Contract Inspectors are well qualified to perform their responsibilities. The team is well organized and has clear targets and schedules for annual inspections. The Panel considers the technical qualifications of the MEM Geotechnical Staff as among the best that it has encountered among agencies with similar duties.

MEM geotechnical engineers addressed significant issues during the reviews and inspections of the Mount Polley TSF. They had insightful questions for the designers at many instances during their review of the design documents, as noted above. The EOR responded to these questions based on their observations and understanding of site conditions. The EOR is responsible for the overall performance of the structure as well as the interpretation of site conditions. The Regulator has to rely on the expertise and the professionalism of the EOR as the Regulator is not the designer.

Despite having a strong regulatory process and personnel, the Perimeter Embankment of the Mount Polley TSF still failed. As indicated in earlier sections, it was a sudden failure without precursors. Additional inspections of the TSF would not have prevented the failure.

However, the question remains as to the expectations from the Regulator in the future. The relationship between the Regulator and the EOR can result in different opinions being expressed that are not easy to resolve without independent input. In such circumstances, independent external advice could be sought as further described in section 9.0. There is a difference between regulating construction and regulating design after it has been approved. The Regulator by observation and experience has the capacity to regulate construction but does not have the capacity to modify the design. Regulators are not normally recruited with specific dam design experience and are limited by statute in their capacity to take on design responsibilities. This role resides with the EOR.

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8 | Regulatory Oversight

It is difficult to review the adequacy of a constructed facility without having limits of measurable indicators that define its performance. Measurable indicators of safe and orderly design and construction are needed for all existing and future tailings facilities that can be monitored and interpreted to evaluate this performance. Section 9.0 provides further elaboration of Quantitative Performance Objectives (QPOs) as a means of accomplishing this.

ENDNOTES

1) Panel Interview, 12/12/14.
2) MP00170
3) MP00218
4) MP00174
5) MP00175
6) MP00216
7) MP00177
8) MO00137
9) MP00139
10) MO00222
11) MP00187

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9 | Where Do We Go From Here?

9.1 PERFORMANCE OF B.C. TAILINGS DAMS

Central to the Panel's Terms of Reference (Appendix A) is to recommend actions for preventing future tailings dam failures:

"... *the Panel may make recommendations to government on actions that could be taken to ensure that a similar failure does not occur at other mine sites in B.C.*"

Fulfilling this mandate starts by considering the tailings dams that currently exist in the province. In particular, this involves how many there are and how they have performed. Appendix I describes the Panel's efforts in this respect. It found that there are currently 123 active tailings dams, those that contain surface water in their impoundments along with tailings.

Active tailings dams were tracked through the years from Ministry of Energy and Mines (MEM) records. In the 46-year period since 1969, there was a total of 4,095 years of active operation and 7 failures, where failure is considered to be breach of the dam resulting in release of tailings and/or water. This corresponds to a failure frequency of 1.7×10^{-3} per dam per year. In other words, statistically there is approximately a 1-in-600 chance of a tailings dam failure in any given year, based on historical performance over the period of record.

While these numbers may seem small, their implications are not. If the inventory of active tailings dams in the province remains unchanged, and performance in the future reflects that in the past, then on average there will be two failures every 10 years and six every 30. In the face of these prospects, the Panel firmly rejects any notion that business as usual can continue.

The Panel firmly rejects any notion that business as usual can continue.

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9.2 GETTING TO ZERO

In risk-based dam safety practice for conventional water dams, some particular level of tolerable risk is often specified that, in turn, implies some tolerable failure rate. The Panel does not accept the concept of a tolerable failure rate for tailings dams. To do so, no matter how small, would institutionalize failure. First Nations will not accept this, the public will not permit it, government will not allow it, and the mining industry will not survive it.

Clearly, improvements to current practice provide an essential starting point on the path to zero failures. But the Panel's evaluation of portfolio risk shows that incremental changes will not be sufficient to achieve this objective.

The path to zero needs an added dimension, and that dimension is technology.

Appendix I explains why. Ultimately, the problem stems from how many active tailings dams there are in the province. To ensure against future failures for all of them would require roughly a hundredfold reduction or more in the current failure frequency. While advances in practices, procedures and policies are imperative, the Panel does not expect these measures by themselves to achieve this degree of improvement. The path to zero needs an added dimension, and that dimension is technology.

Tailings dams are complex systems that have evolved over the years. They are also unforgiving systems, in terms of the number of things that have to go right. Their reliability is contingent on consistently flawless execution in planning, in subsurface investigation, in analysis and design, in construction quality, in operational diligence, in monitoring, in regulatory actions, and in risk management at every level. All of these activities are subject to human error.

Failures invariably bring about improvements in technology that help compensate for human error.

Human error is often, if not always, found to play a key role in technological failures. And human error will always be with us, as much as we might wish it to be otherwise. This is why failures invariably bring about improvements in technology that help compensate for human error. In perhaps the most notorious containment failure, double-hulled tankers were mandated after the *Exxon Valdez* oil spill. Similarly, improvements to rail tank cars are being adopted in the wake of the Lac-Mégantic tragedy. But tailings dams have no such redundancies. Without exception, dam breaches produce tailings releases. This is why best practices can only go so far in improving the safety of tailings technology that has not fundamentally changed in the past hundred years.

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Improving technology to ensure against failures requires eliminating water both on and in the tailings: water on the surface, and water contained in the interparticle voids. Only this can provide the kind of failsafe redundancy that prevents releases no matter what. In terms of portfolio risk, Appendix I shows that this works by reducing the inventory of active tailings dams subject to failure in the first place. Simply put, dam failures are reduced by reducing the number of dams that can fail.

Thus, the path to zero leads to best practices, then continues on to best technology.

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9.3 BEST AVAILABLE TAILINGS TECHNOLOGY

9.3.1 BAT PRINCIPLES

While best practices focus on the performance of the tailings dam, best available technology (BAT) concerns the tailings deposit itself. The goal of BAT for tailings management is to assure physical stability of the tailings deposit. This is achieved by preventing release of impoundment contents, independent of the integrity of any containment structures. In accomplishing this objective, BAT has three components that derive from first principles of soil mechanics:

1. **Eliminate surface water from the impoundment.**
2. **Promote unsaturated conditions in the tailings with drainage provisions.**
3. **Achieve dilatant conditions throughout the tailings deposit by compaction.**

The first of these, eliminating surface water, not only precludes release of water itself, but also eliminates fluvial tailings transport mechanisms like those illustrated in Appendix C during the Mount Polley breach. The second, promoting unsaturated conditions by drainage, reduces the possibility for, and the quantity of, high-mobility flowslide release of tailings. And the third, achieving dilatant conditions by compaction, further reduces flowslide potential by improving the properties of the tailings mass. Thus, underpinning these principles are multiple redundancies that provide defence in depth.

The Panel recognizes that eliminating water from the tailings deposit will not eliminate the need for storage of mine and processing water elsewhere. But Mount Polley has shown the intrinsic hazards associated with dual-purpose impoundments storing both water and tailings. The Panel considers that security can be more readily assured for conventional water dams that are designed and constructed for their own purpose and that preventing tailings release is the overriding imperative.

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9.3.2 BAT METHODS

The overarching goal of BAT is to reduce the number of tailings dams subject to failure. This can be achieved most directly by storing the majority of the tailings below ground—in mined-out pits for surface mining operations or as backfill for underground mines. Both methods require integrating tailings planning into mine planning. This has not been common practice in the industry to date, as the Mount Polley case has shown, and the synergies to be achieved are mostly unexplored. Apart from this, surface storage using filtered tailings technology is a prime candidate for BAT.

The overarching goal of BAT is to reduce the number of tailings dams subject to failure.

Demonstrated technology for producing and placing filtered tailings (sometimes termed "dry stack" tailings) is well-known in the industry. Its adoption and design practices are documented in the literature.[1, 2] Using various kinds of equipment, the water content of the tailings is reduced before they leave the mill. The specified degree of water removal can vary, but is sufficient to allow transport by truck or conveyor to the tailings facility and compaction. Compaction is necessary to prevent liquefaction flowslides that can and have occurred in loosely placed dewatered materials due to infiltration of ponded surface runoff. The Panel recognizes that creating dry tailings may increase the amount of water requiring treatment or storage.

Filtered tailings technology embodies all three BAT components described in section 9.3.1. Most commonly used in dry climates where economy in water consumption is important, it has also been adapted to cold regions.[3] This method has been used since start-up of the Greens Creek mine in Alaska under conditions not unlike coastal B.C.[4] The Greens Creek facility is shown in **Figure 9.1.1**.

Variations on this technology are easily envisioned, for example separation, dewatering, and gravity drainage of sand tailings by cycloning to reduce quantities requiring filtration dewatering. The Panel believes that additional enhancements are ripe for development if there is incentive to do so.

In some cases, clayey ore may pose difficulties in dewatering. And most filtered tailings operations to date have been relatively small. But some new operations will be producing filtered tailings at a rate of 68,000 tonnes per day—almost three times the production of Mount Polley—in facilities that will reach heights of 150 metres (m). As demonstrated by the Greens Creek case and others, there are no overriding technical impediments to more widespread adoption of filtered tailings technology.

There are no overriding technical impediments to more widespread adoption of filtered tailings technology.

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FIGURE 9.1.1 FILTERED TAILINGS FACILITY, GREENS CREEK, ALASKA

The chief reason for the limited industry adoption of filtered tailings to date is economic. Comparisons of capital and operating costs alone invariably favour conventional methods. But this takes a limited view. Cost estimates for conventional tailings dams do not include the risk costs, either direct or indirect, associated with failure potential. The Mount Polley case underscores the magnitude of direct costs for cleanup, but indirect losses—notably in market capitalization—can be even larger.[5] Nor do standard costing procedures consider externalities, like added costs that accrue to the industry as a whole, some of them difficult or impossible to quantify. Full consideration of life cycle costs including closure, environmental liabilities, and other externalities will provide a more complete economic picture. While economic factors cannot be neglected, neither can they continue to pre-empt best technology.

While economic factors cannot be neglected, neither can they continue to pre-empt best technology.

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9.3.3 BAT FOR CLOSURE

Closure of tailings deposits is subject to two fundamental considerations: physical stability and chemical stability. Although the former is the object of the Panel's investigation, no treatment of tailings technology can ignore the latter. Matters related to physical and chemical stability reside in different domains and have developed independently, each with their own goals and methods. These two aspects converge in the context of BAT.

In short, the most serious chemical stability problem concerns tailings that contain sulfide minerals, particularly in metal and coal mining. In the presence of oxygen, these sulfides react to produce acid that then mobilizes a variety of metals in solution. There are a number of ways to arrest this reaction, and one is to saturate the tailings so that water replaces oxygen in the void spaces. This saturation is most conveniently achieved by maintaining water over the surface of the tailings. Hence, so-called water covers have sometimes been adopted for reactive tailings during operation and for closure.

It can be quickly recognized that water covers run counter to the BAT principles defined in section 9.3.1. But the Mount Polley failure shows why physical stability must remain foremost and cannot be compromised. Although the tailings released at Mount Polley were not highly reactive, it is sobering to contemplate the chemical effects had they been. No method for achieving chemical stability can succeed without first ensuring physical stability: chemical stability requires above all else that the tailings stay in one place.

Filtered tailings technology adopts a different approach to chemical stability. Rather than arresting the reaction, it retards the transport of reaction products. Seepage gradients are greatly diminished by eliminating surface water. This has a beneficial effect not only on sulfide reaction products; it also equally reduces transport of soluble constituents such as arsenic, sulfates and selenium, if present in the tailings.

Moreover, the technology for alternative dry covers is well advanced. Using different cover designs for different climatic conditions, soil covers placed over the tailings deposit further reduce infiltration, retard oxygen entry, or both. Cover placement and reclamation can proceed concurrently with operation, as shown in the foreground in **Figure 9.1.1** at Greens Creek.

Yet other technologies attack the chemical effects of sulfide minerals by removing them from the tailings. Doing so using conventional metallurgical processes has been shown to be technically and economically feasible.[6] These same techniques can be used, in effect, to manufacture clean tailings cover material free from sulfides.[7]

This shows that the physical stability objectives of BAT are not incompatible with chemical stability. A variety of complementary technologies are available for achieving both.

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9.3.4 BAT RECOMMENDATIONS

Implementation of BAT is best carried out using a phased approach that applies differently to tailings impoundments in various stages of their life cycle.

- **For existing tailings impoundments**. Constructing filtered tailings facilities on existing conventional impoundments poses several technical hurdles. Chief among them is undrained shear failure in the underlying saturated tailings, similar to what caused the Mount Polley incident. Attempting to retrofit existing conventional tailings impoundments is therefore not recommended, with reliance instead on best practices during their remaining active life.
- **For new tailings facilities**. BAT should be actively encouraged for new tailings facilities at existing and proposed mines. Safety attributes should be evaluated separately from economic considerations, and cost should not be the determining factor.
- **For closure**. BAT principles should be applied to closure of active impoundments so that they are progressively removed from the inventory by attrition. Where applicable, alternatives to water covers should be aggressively pursued.

As discussed in section 9.2, best technology is only one of the two components necessary for safety improvement. The complementary aspects of best practices are presented in the following sections.

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9.4 BEST APPLICABLE PRACTICES (BAP)

The safety of any dam, water or tailings, relies on multiple levels of defence. The Panel was disconcerted to find that, while the Mount Polley Tailings Dam failed because of an undetected weakness in the foundation, it could have failed by overtopping, which it almost did in May 2014. Or it could have failed by internal erosion, for which some evidence was discovered. Clearly, multiple failure modes were in progress, and they differed mainly in how far they had progressed down their respective failure pathways.

Accordingly, recommendations for future BAP require considerations that go beyond stability calculations. It is important that safety be enhanced by providing for robust outcomes in dam design, construction and operations. As discussed below, this has implications for corporate responsibility, enhanced regulatory capacity, expanded technical review, and improvements in professional practice.

It is important that safety be enhanced by providing for robust outcomes in dam design, construction and operations.

9.4.1 CORPORATE GOVERNANCE

In response to several international tailings dam failure incidents in the 1990s, the Mining Association of Canada (MAC) established a task force in 1996 to promote safe, environmentally responsible management of tailings and mine waste. The task force concluded that the main priority should focus on improvement of tailings management, which resulted in the establishment of the MAC Tailings Working Group. The outcome of this initiative were several guides related to the management of tailings facilities; the development of operations, maintenance and surveillance manuals; and auditing and assessment of tailings management facilities.[8] The guides themselves are available from the MAC.[9] They are now embraced by the Towards Sustainable Mining (TSM) initiative launched by MAC in 2004.

Compliance with the TSM initiative is an element of BAP for the mining industry today. Accordingly, mining operations in B.C. proposing to operate a tailings storage facility (TSF) should either be required to be a member of MAC—ensuring adherence to the TSM—or be obliged to commit to an equivalent program, including the audit function. Tailings management is often not a core skill in many mining organizations. Embracing MAC's TSM initiative will ensure awareness of responsibilities at the highest corporate levels.

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At the same time, many in the industry have reacted to the Mount Polley failure with incredulity, asking how it could have happened with programs such as MAC's in place. This serves as a reminder that these programs should not instill a sense of overconfidence and cannot themselves be seen as a substitute for more fundamental changes in technology.

9.4.2 CORPORATE TSF DESIGN RESPONSIBILITIES

In the experience of the Panel, TSF design studies submitted to Regulators are often lacking in detail regarding the factors that need to be considered in assuring safety of the facility. This applies equally to appropriate tailings technology and to performance metrics for confirming orderly construction and operations.

At Mount Polley, the only quantitative performance objectives were those implied in its design criteria. A list of potential failure modes was compiled in the 2006 Dam Safety Report, but these were generic and not tied to specific site conditions. One of the lessons learned here is that future permit applications for TSFs must provide a more comprehensive assessment of potential geotechnical problems associated with the selected site. In addition, BAT for both tailings storage and closure considerations also needs to be incorporated in such proposals.

The Panel is of the view that the inclusion of these considerations and the declaration of Quantitative Performance Objectives (QPOs) are best incorporated early in project commitment at the bankable feasibility level. QPOs are intended to constrain the type of ad hoc design practices that characterized Mount Polley and strengthen regulatory capacity.

The Panel would require a bankable feasibility study and related permit application to have considered all technical, environmental, social and economic aspects of the project. Resolution of technical and environmental considerations would usually be supported by proven methods, although technology development studies would not be precluded if they have advanced far enough to warrant implementation in practice. The bankable feasibility study would be of sufficient detail to support an investment decision that might have an accuracy of ±10%–15%.

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More explicitly, the bankable feasibility document would be required to contain the following:

1) A detailed evaluation of all potential failure modes associated with:

- The geological conditions of the site
- The uncertainties associated with this evaluation
- The role of the Observational Method to manage residual risk
- Mitigation measures in case worse than anticipated conditions are encountered.

This evaluation should be updated and incorporated into MEM requirements for annual inspection and construction review. This is to ensure that the evaluation would become a living document maintained throughout the life of the facility. It should be sufficiently well documented to survive changes in mine personnel, mine ownership or Engineers of Record (EORs), and it should be referenced as part of the Operations Maintenance and Surveillance (OMS) manual. The Panel anticipates that as-built reports would provide the basic information recording departures from what had been anticipated. An ongoing compilation should be maintained by the EOR as a separate document.

2) Detailed cost analyses of BAT tailings and closure options, so that alternative means of achieving BAT can be understood and accommodated. As discussed in section 9.3.2, this assessment should recognize that indirect and unquantifiable costs cannot be fully incorporated and hence the results of the cost analyses should not supersede BAT safety considerations.

3) A detailed declaration of QPOs, beyond those associated with regulatory compliance and ordinary design criteria. Examples of QPOs are numerical values and limits associated with:

- Beach widths
- Calibration of impoundment filling schedule
- Water balance audits and calibration
- Construction material availability and scheduling to ultimate height of structure
- Instrumentation adequacy and reliability
- Trigger levels for response to instrumentation
- Performance data gathering, interpretation, and reporting intervals

The Panel recognizes the need for a regulatory process that is responsive to changed conditions arising from market forces, reserves, regulatory revisions and technical issues. It is envisaged that such changes can be accommodated by staged approval for construction, as occurs at present. However, the stage applications should honour the declared QPOs or present a basis for their modification.

The Panel recognizes the need for a regulatory process that is responsive to changed conditions arising from market forces, reserves, regulatory revisions and technical issues.

9.4.3 INDEPENDENT TAILINGS REVIEW BOARD (ITRB)

The appointment of ITRBs to provide third-party advice on the design, construction, operation and closure has become increasingly common and is recognized to provide value.[10] The World Bank and other lenders groups are requiring the formation of an ITRB. International Finance Corporation/World Bank guidance and operating principles OP4.01 and OPR.37 establish the requirement to review the development of tailings dam design, construction and initial dam filling. Maintaining an ITRB through operations and closure will depend upon the scale and complexity of the facility. Some large corporations retain a third-party review board for ongoing advice on tailings operations to complement their internal technical audit systems.

The appointment of ITRBs to provide third-party advice on the design, construction, operation and closure has become increasingly common and is recognized to provide value.

ITRBs are not unique to the mining industry. They have a long history in water dam design and safety assessments. In British Columbia, BC Hydro has considerable experience with such Boards for safety assessment of both existing and new dam projects. In a mining context, an ITRB could be asked to provide opinions on the following:

- Whether the design, construction and operation of the TSF are consistent with satisfactory long-term performance.
- Whether design and construction have been performed in accordance with the Board's expectation of good practice.
- Whether safety and operation of the TSF conform to the Board's expectation of good practice.
- Whether there are weaknesses that would reasonably be expected to have a material adverse effect on the integrity of the TSF, human health, safety, and successful operation of the facility for its intended purpose.

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Experience has shown that the effectiveness of an ITRB in specific circumstances depends on the following:

- That it not be used exclusively as a means for obtaining regulatory approval.
- That it not be used for transfer of corporate liability by requesting indemnification from Board members.
- That it be free from external influence or conflict of interest.
- That there be means to assure that its recommendations are acted upon.

No ITRB can function successfully without unqualified support and commitment at the highest corporate levels. While it is essential that the Board be organized by Mine Operations, it is equally essential that its reports go to senior corporate management and Regulators. To establish and strengthen credibility, Board reports should also be open to other stakeholders. An important mechanism for accountability in response to Board recommendations is the creation of an Action Log that reviews corporate response to Board recommendations at each successive meeting.

No ITRB can function successfully without unqualified support and commitment at the highest corporate levels.

It is evident that the establishment of Independent Tailings Review Boards is an element of BAP, and the Panel is of the view that they have a role in improving current practice. But they should not be necessary for all tailings undertakings and MEM should consider, based on their current portfolio of operating and proposed TSFs, the conditions related to complexity and failure consequence that warrant an ITRB.

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9.4.4 MINISTRY OF ENERGY AND MINES (MEM)

As noted in section 7, the Panel was favourably impressed by the skill and commitment of MEM's geotechnical staff in carrying out their responsibilities. Nevertheless, it also considered what measures could be taken to improve regulatory operations.

With recent inspections of TSFs in the province in hand, the short-term need is to evaluate these facilities with respect to the following potential failure modes, in order of importance:

1. **Undrained shear failure** for dams with silt and clay foundation soils.
2. **Water balance adequacy,** including provisions and contingencies for wet years.
3. **Filter adequacy,** especially for dams containing broadly graded soils or mine waste.

The Regulator is not the designer.

One issue identified in section 8.0 is the ultimate reliance of the Regulator on the EOR to confirm that the facility is safe and is operating as intended. The Regulator is not the designer, and this limits the degree of inquiry that is manageable. If Regulators were provided with more information in an ongoing manner, they would be better versed to engage the EOR. This is one of the benefits of having declared QPOs that can be monitored, as discussed in section 9.4.2. To this end, MEM should evaluate how to determine the QPOs associated with ongoing facilities and begin to apply them in practice.

Additionally, the Panel's compilation of the province's tailings dam inventory revealed limitations in MEM's capacity for information retrieval, especially for timely response to unexpected occurrences. Tailings dam data for each mine and each structure needs to be scanned electronically, compiled separately from permit files, and maintained in a readily accessible database.

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9.4.5 PROFESSIONAL PRACTICE

The Panel found it disconcerting that, notwithstanding the large number of experienced geotechnical engineers associated with the Mount Polley TSF, the overall adequacy of the site investigation and characterization of ground conditions beneath the Perimeter Embankment went unquestioned. This may reflect a regional issue, or possibly one of wider extent. Regardless, it calls for a concerted effort to improve professional practice in this area. The situation is reminiscent of the conditions that prevailed in B.C. that resulted in the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC) Guidelines for *Legislated Landslide Assessment for Proposed Residential Developments in B.C.*

In the view of the Panel, the fundamental need is to improve the geological, geomorphological , hydrogeological and possibly seismotectonic understanding of sites proposed for tailings dams in B.C. This improved understanding should account for the likely scale associated with variability so that site investigations can be planned with enhanced reliability.

APEGBC appears to be well-suited for this task.

9.4.6 CANADIAN DAM ASSOCIATION (CDA) GUIDELINES

From its inception in 1995, the Mount Polley TSF adopted a minimum factor of safety (FS) of 1.3 during operations and 1.5 for closure. As chronicled in section 5.4, these FS criteria drove key decisions throughout the design process, and so the Panel is of the view that it would be helpful to comment on them.

CDA dam safety guidelines originally developed for water dams were subsequently adapted to tailings dams, with target factors of safety as indicated in **Table 9.4.1**.[11]

TABLE 9.4.1 TARGET FACTORS OF SAFETY FOR SLOPE STABILITY IN CONSTRUCTION, OPERATION AND TRANSITION PHASES – STATIC ASSESSMENT (AFTER CDA, 2014)

LOADING CONDITIONS	MINIMUM FACTOR OF SAFETY	SLOPE
During or at end of construction	>1.3 depending on risk assessment during construction	typically downstream
Long-term (steady state seepage, normal reservoir level)	1.5	downstream

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These 2014 guidelines vest responsibility for establishing appropriate FS criteria solely with the designer, subject to the designer's consideration of the following:

- The consequences of failure
- The loading conditions
- The strength parameters used

Hence, the CDA Guidelines are premised on proper evaluation of these factors. But in the case of Mount Polley, this premise was flawed. Few would argue that the failure consequences were anything less than catastrophic to those affected. The loading conditions did not account for the development of normally consolidated conditions in the foundation. And the strength parameters neglected undrained shearing. Furthermore, selection of FS criteria using risk analysis, as specified in **Table 9.4.1**, could not have succeeded because the operative failure mode went unrecognized.

Mount Polley illustrates that dam safety guidelines intended to be protective of public safety, environmental and cultural values cannot presume that the designer will act correctly in every case. To do so defeats the purpose of FS criteria as a safety net. In this, the CDA Guidelines are unable to achieve their intended purpose. Neither is the Province well served, to the extent that MEM has incorporated compliance with these guidelines as a statutory requirement.[13]

Tailings dam guidelines and criteria tailored to conditions in B.C. would more effectively meet the needs of the Province in protecting public safety.

The Panel considers that tailings dam guidelines and criteria tailored to conditions in B.C. would more effectively meet the needs of the Province in protecting public safety. Those developed by the U.S. Army Corps of Engineers for water dams provide one example, among others, that might be used as a starting point.[12] This does not preclude adopting parts of the CDA Guidelines where appropriate as well as the CDA technical bulletin *Geotechnical Considerations for Dam Safety*.[14] The Panel anticipates that this will result in more prescriptive requirements for site investigation, failure mode recognition, selection of design properties, and specification of factors of safety.

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Based on the activities described and interpretations advanced in the preceding sections of the report, the Panel has developed the findings summarized below.

10.1 MECHANISM OF FAILURE

The breach of the Perimeter Embankment on August 4, 2014 was caused by shear failure of dam foundation materials when the loading imposed by the dam exceeded the capacity of these materials to sustain it. The failure occurred rapidly and without precursors.

Direct evidence of this failure mechanism is provided by an identified shear surface in surviving remnants of the dam core and by deformations consistent with shearing in a weaker glacially-deposited layer of silt and clay about 8–10 metres (m) below the original ground surface. This layer, its properties, and its extent received intense scrutiny during this investigation, and analyses using representative parameters provide indirect evidence that further supports this failure mechanism.

Deposited in a complex geologic environment, the weaker glaciolacustrine layer was localized to the breach area. It went undetected, in part because the subsurface investigations were not tailored to the degree of this complexity. But neither was it ever targeted for investigation because the nature of its strength behaviour was not appreciated.

Throughout, the design investigations took note of the stiff, dense character of foundation soils and used corresponding strength properties in stability analyses. But it was not recognized that this character would change, with a corresponding change in strength behaviour under the increased loading as the dam grew higher. Specifically, it was never recognized that the glaciolacustrine soils that were initially overconsolidated would become normally consolidated, requiring undrained shear strengths for stability analyses. This is the process that affected the weaker glaciolacustrine layer in the breach area that was not accounted for in the design of the dam.

Adding to the antecedent foundation conditions was the unprecedented steepness of the 1.3H:1V Perimeter Embankment slope. This was justified by design analyses without questioning its reasonableness. The higher Main Embankment had glaciolacustrine foundation soils with properties broadly comparable to those at the breach section. But here, the steep slopes were effectively flattened by the addition of a buttress, which explains why the failure did not occur at the highest part of the dam.

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10.2 CONTRIBUTING FACTORS

10.2.1 LONG-TERM PLANNING

A lack of foresight in planning for dam raising contributed to the failure. Successfully executing the raising plan required intimate coordination of impoundment water-level projections, production and transport of mine waste for raising, and seasonal constraints on construction. This made the tailings dam contingent at the same time on the water balance, the Mine plan, and the weather. But instead of projecting these interactions into the future, they were evaluated a year at a time, with dam raising often bordering on ad hoc and only responding to events as they occurred. The effects were twofold: a near overtopping failure in May of 2014, and restrictions on mine waste availability that produced the oversteepened slopes and deferred buttress expansion.

10.2.2 OBSERVATIONAL METHOD

The Observational Method was adopted as a design philosophy, but misapplied. For reasons not unrelated to planning shortcomings, instrumentation was relied upon to substitute for definitive input parameters and design projections. But the Mount Polley dam was ill-suited to this approach, for both practical and strategic reasons. The steep slopes and constant construction activity on the Perimeter Embankment prevented installation of instruments at optimal locations. More importantly, the instrumentation program was incapable of detecting critical conditions because, once again, the critical materials and their critical mode of undrained behaviour were not recognized.

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10.3 ROLE OF WATER

In light of its importance in planning and the near-overtopping incident, the role of water contained in the tailings storage facility (TSF) deserves special mention. First of all, overtopping did not cause the breach of August 4, 2014. However, the high water level acted in other ways that influenced both the failure and its effects.

High impoundment water levels were a major cause of chronic problems in maintaining a tailings beach around the perimeter of the dam. At the breach section, water was in direct contact with the upstream zone of tailings fill when failure occurred. This increased the piezometric level in the upstream zone above what it would have been had a wide tailings beach been present. The Panel's analyses show that this had some influence on dam stability, although it was not the dominant factor.

The high water level was the final link in the chain of failure events. Immediately before the failure, the water was about 2.3 m below the dam core. The Panel's excavation of the failure surface showed that the crest dropped at least 3.3 m, which allowed overflow to begin and breaching to initiate. Had the water level been even a metre lower and the tailings beach commensurately wider, this last link might have held until dawn the next morning, allowing timely intervention and potentially turning a fatal condition into something survivable.

Finally, the quantity of water had a great deal to do with the quantity of tailings released after the breach developed. It was water erosion that transported the bulk of the tailings, and these fluvial processes ended when the supply of water was exhausted. Had there been less water to sustain them, the proportion of the tailings released from the TSF would have been less than the one-third that was actually lost.

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10.4 REGULATORY FACTORS

The Panel examined regulatory activities by the Ministry of Energy and Mines (MEM) in relation to the failure and whether different actions on MEM's part might have prevented it. In particular, the Panel's attention was drawn to the period from 2009 to 2011 when no government inspections of the Mount Polley dam were performed. The Panel concludes that this lack of inspection was immaterial to the failure because there were no precursors that could have been detected, even on the eve of the breach. By definition, no amount of inspection can discover a hidden flaw.

The Panel also examined MEM's actions concerning factors that did have a material relationship to the failure. In this regard, MEM queried the designer about softer conditions in glaciolacustrine soils encountered in a groundwater well that were similar to those at the breach. Its inspector issued a "Departure from Approval" notice concerning the absence of an adequate tailings beach. The inspector questioned the designer's factor of safety FS = 1.3 criterion, subsequently requiring its increase. The Panel found these actions to be appropriate and within the expected conduct of regulatory responsibilities.

It is not unreasonable to ask whether MEM could have acted sooner or more aggressively in these matters or even intervened in the design process, and perhaps this might have been warranted under the harsh illumination of hindsight. Yet the Panel considers that a bright line must be maintained between designer and Regulator. It is axiomatic that a Regulator cannot regulate its own activities. Were it to usurp the role of the designer, it would also usurp its own role.

10.5 POSSIBLE FAILURE PREVENTION

In fulfilling its Terms of Reference, the Panel considered what actions could have been taken to prevent the failure. From a purely technical perspective, apart from rectifying the deficiencies reviewed here, there is one that stands out.

The design for the next raise of the dam had been submitted only days before the failure. In it was a buttress that would have extended along the Perimeter Embankment, including the breach section. Although this buttress was still not designed using the appropriate stratigraphy or undrained strengths, the Panel determined that had it been in place, the failure would have been averted. The solution would have been correct, even if for the wrong reasons.

In keeping with its Terms of Reference, the Panel has developed these conclusions on the basis of technical factors specific to the Mount Polley failure. It must be left to others to determine how they might translate more broadly to legislative, administrative process, and policy areas.

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11 | Recommendations

Recognizing that the path to zero failures involves a combination of best available technology (BAT) and best applicable practices (BAP), the Panel recommends the following:

1) **To implement BAT using a phased approach:**
   - **a**. **For existing tailings impoundments**. Rely on best practices for the remaining active life.
   - **b**. **For new tailings facilities**. BAT should be actively encouraged for new tailings facilities at existing and proposed mines.
   - **c**. **For closure**. BAT principles should be applied to closure of active impoundments so that they are progressively removed from the inventory by attrition.

   See section 9.3.

2) **To improve corporate governance:**

   Corporations proposing to operate a tailings storage facility (TSF) should be required to be a member of the Mining Association of Canada (MAC) or be obliged to commit to an equivalent program for tailings management, including the audit function.

   See section 9.4.1.

3) **To expand corporate design commitments:**

   Future permit applications for a new TSF should be based on a bankable feasibility that would have considered all technical, environmental, social and economic aspects of the project in sufficient detail to support an investment decision, which might have an accuracy of ±10%–15%. More explicitly, it should contain the following:

   - **a**. A detailed evaluation of all potential failure modes and a management scheme for all residual risk.
   - **b**. Detailed cost/benefit analyses of BAT tailings and closure options so that economic effects can be understood, recognizing that the results of the cost/benefit analyses should not supersede BAT safety considerations.
   - **c**. A detailed declaration of Quantitative Performance Objectives (QPOs).

   See section 9.4.2.

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11 | Recommendations

4) To enhance validation of safety and regulation of all phases of a TSF:

Increase utilization of Independent Tailings Review Boards.

See section 9.4.3.

5) To strengthen current regulatory operations:

a. Utilize the recent inspections of TSFs in the province to ascertain whether they may be at risk due to the following potential failure modes and take appropriate actions:

i. Undrained shear failure of silt and clay foundations

ii. Water balance adequacy

iii. Filter adequacy

b. Utilize the concept of QPOs to improve Regulator evaluation of ongoing facilities.

See section 9.4.4.

6) To improve professional practice:

Encourage the APEGBC to develop guidelines that would lead to improved site characterization for tailings dams with respect to the geological, geomorphological, hydrogeological and possibly seismotectonic characteristics.

See section 9.4.5.

7) To improve dam safety guidelines:

Recognizing the limitations of the current Canadian Dam Association (CDA) Guidelines incorporated as a statutory requirement, develop improved guidelines that are tailored to the conditions encountered with TSFs in British Columbia and that emphasize protecting public safety.

See section 9.4.6.

12 | Postscript and Acknowledgements

The Panel has been acutely aware of its responsibilities in conducting this investigation. It set out to be thorough, focusing on the technical issues, and to report its findings in an independent, open, transparent and timely manner. It is content that it has fulfilled its mandate. To do so required the digestion of thousands of pages of technical documents; field investigations involving mapping, drilling and sampling; complex laboratory tests; various theoretical analyses; and consolidation of its findings, conclusions and recommendations in a manner intended to be accessible to a variety of stakeholders. The Panel could not have met its objectives without the assistance of a number of dedicated and skilled individuals. The Panel wishes to acknowledge this assistance here.

First, and possibly foremost, the Panel expresses its gratitude to Mr. Kevin Richter, Assistant Deputy Minister, Ministry of Transportation and Infrastructure, who was appointed to lead the Secretariat for the investigation and his assistants, Stacy Scriver and Rupinder Prihar. Mr. Richter managed the business of the investigation with enormous skill, diplomacy and good grace. This allowed the Panel to focus on its main task and hence the Secretariat made a most valuable contribution to the collective effort.

The Panel retained Thurber Engineering Limited (Thurber) to undertake a wide variety of technical tasks acting under the direction of the Panel. These tasks involved site mapping, drilling and sampling, a wide suite of laboratory tests, a variety of analyses, and preparing material for inclusion in the report. The Thurber team was outstanding in its technical contributions and dedication to this assignment. The Panel was extremely pleased to work with such a skilled team that included the following:

- In Vancouver – David Regehr (Project Manager), Paul Wilson, Caleb Scott, Ben Singleton-Polster, Denny Ma, Andrea Lougheed, Paul Evans
- In Victoria – Stephen Bean, Warren Wunderlick, Suzanne Powell
- In Calgary – John Sobkowicz
- And others too numerous to mention

Deborah Lovett, QC, of Lovett & Westmacott was retained as legal advisor to the Panel and provided wise counsel throughout the period of the investigation.

Judith Brand provided senior editorial advice and Shawn Robins, Robins Communications, assisted the Panel in organizing its outreach activities.

While many have contributed to this report, the Panel retains sole responsibility for its content.

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13 | List of Abbreviations

ABBREVIATIONS

APEGBC:	Association of Professional Engineers and Geoscientists of British Columbia
BAP:	best applicable practices
BAT:	best available technology
CDA:	Canadian Dam Association
CPT:	cone penetration test
CQA:	Construction Quality Assurance
DSR:	Dam Safety Review
DSS:	direct simple shear
EOR:	Engineer of Record
EPRI:	Electric Power Research Institute
ESA:	effective-stress analysis
FOIPPA:	*Freedom of Information and Protection of Privacy Act*
FS:	factor of safety
GLU:	Upper Glasciolacustrine Unit
GSA:	grain size analyses
ITRB:	Independent Tailings Review Board
KCB:	Klohn Crippen Berger
KP:	Knight Piésold
LiDAR:	Light Detection And Ranging
LI:	Liquidity Index
LPT:	Large Penetration Testing
MAC:	Mining Association of Canada
MEM:	Ministry of Energy and Mines
MFLNRO:	Ministry of Forests, Lands and Natural Resource Operations
MoE:	Ministry of the Environment
MOU:	Memorandum of Understanding
MPMC:	Mount Polley Mining Corporation
OCR:	overconsolidation ratio
OMS:	Operations Maintenance and Surveillance
PMP:	Probable Maximum Precipitation
QPO:	Quantitative Performance Objective
RCPT:	Resistivity Cone Penetration Test
S.O.L.:	setting out line
SPT:	standard penetration test
Thurber:	Thurber Engineering Limited
TSF:	tailings storage facility
TSM:	Towards Sustainable Mining
USA:	undrained strength analysis
USBR:	U.S. Bureau of Reclamation
VST:	vane shear test

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14 | Glossary of Technical Terms

GLOSSARY

Angle of repose: the maximum slope steepness that dry granular material can sustain

Anisotropy: directional differences in properties, typically horizontal and vertical

Anticlinal structure: dome-shaped folding

Applied load: usually gravity stresses imposed by a structure; simplistically, its weight

Arcuate headscarps: semicircular and nearly perpendicular slopes delineating the upper end of a slope movement

Artesian pressure: water pressure sufficient to cause water to flow upwards out of the ground

Bankable feasibility study: a level of design sufficient for detailed cost estimates

Bathymetric survey: survey of underwater surfaces

Beach: a gently sloping surface of deposited tailings

Bedding: layering, commonly horizontal

Blow count: the number of drops of a heavy weight required to advance a sampler 30 cm into the ground

Buttress: a berm constructed at the bottom of a slope to increase its stability

Chimney drain: a zone of sand or gravel within a dam for collecting and conveying water

Coefficient of consolidation: a parameter used to calculate change of pore pressure with loading

Crest: the top of a dam or slope

Critical failure surface in stability analysis: the failure surface with the lowest factor of safety

Cycloning: separation of tailings into coarser and finer fractions

Dendritic drainages: branching stream channels

Dip direction: the direction in which a geologic structure slopes downward

Direct shear: type of test used to determine drained shear strength

Direct simple shear: type of test used to determine undrained shear strength

Downcutting: a natural process of excavation, usually by erosion

Downthrow: downward movement

Effective stress: the stress experienced by soil particles after the known pore pressure is subtracted

Effective-stress strength: the strength of a soil expressed only in terms of the effective stress

En echelon scarps: parallel steep slopes produced by ground movement

Factor of safety: the ratio of available strength to the strength required for equilibrium; a measure of stability

Fines: fine particles smaller than visible with the naked eye, typically less than 0.074 mm diameter

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Flowslide: high-velocity earth movement; mudflow

Fluvial processes: processes caused by or associated with rivers or streams

Freeboard: reservoir capacity reserved for storage of flood inflows, including wave height

Grab samples: disturbed samples

Graben: downdropped block within the ground

Headscarp: steep slope at the upper end of a landslide

Hydraulic-cell deposition: controlled discharge of tailings into a small, confined area

Hydraulic fracturing: cracking of soil caused by water pressure

Inclinometer: a device for measuring horizontal subsurface movements

Internal erosion: subsurface transport of soil particles by water

Interparticle voids: open spaces between soil particles

Glaciofluvial: associated with or deposited in a glacial stream

Glaciolacustrine: associated with or deposited in a glacial lake

Lift lines: boundaries between successive layers of compacted fill

Loading: the imposition of stresses or weight; see applied load

Marker bed: a prominent layer of soil or rock used as a reference

Normally consolidated: a state or condition of soil that is experiencing pressures equal to or exceeding the pressures that it has experienced in the past

Oedometer test: a test for measuring compression of soil under load

Offtake: a drain or pipe that discharges flow

Orthophoto imagery: aerial photograph looking directly down on the terrain

Overconsolidation: a state or condition of soil produced by past stresses greater than those that currently exist

Overtopping: water flowing over the crest of a retaining dam or structure

Phreatic surface: water table

Piezometer: a device for measuring subsurface water pressure

Piping: see internal erosion

Pore pressure: the pressure of water that exists within the voids of a soil mass; see interparticle voids

Preconsolidation pressure: the maximum pressure experienced by the soil in its past

Pre-shearing: the process or condition of having been previously sheared

Relic erosional surface: ground surface remaining after previous erosion

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14 | Glossary of Technical Terms

Residual strength: strength of a soil after having been sheared; see also pre-shearing

Rills: small-scale gullies

Runup: the height of breaking waves on a slope

Sand tailings: coarser fraction of tailings

Scarp: a very steep, near-perpendicular slope at the head of a landslide; see also headscarp

Scour: erosion by surface water

Seepage flow: flow of subterranean water

Sentinel section: an instrumented section providing preliminary information; see also inclinometer, piezometer

Shear: a) the act or process of one surface sliding across another; b) a state of stress in the ground

Shell: a zone of material that supports the core of a dam

Slickenside: polished surface resulting from shearing

Slimes: finer fraction of tailings

Slump blocks: large masses subject to or transported by downslope movement

Stereopairs: aerial photographs producing a three-dimensional image

Stratigraphy: systematic or characteristic layering exhibited by soil or rock at a particular locale

Substrate: underlying soil

Survey monuments: fixed reference points for measuring relative movements

Tailings: finely ground rock particles remaining after extraction of valuable minerals

Tailings beach: see beach

Till: unsorted glacial sediment moved or deposited directly by the glacier

Tip resistance: the pressure measured at the tip of the cone during CPT testing

Toe: bottom of a slope

Triaxial test: type of test used here to determine drained and undrained strength

Undrained strength: the strength of a soil that incorporates the effect of pore pressures generated by shearing

Undrained strength ratio: the ratio of undrained strength to effective stress

Vane testing: an in situ test for measuring undrained strength of clays

Varving: thinly laminated layering

Water balance: an accounting of water inputs and outputs for determining water accumulation or deficit

Whaleback: a linear bulge or uplift

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Keynote Address
Tailings and Mine
Waste Practices
SME, AGM
Phoenix, 2002

Design of Tailings Dams and Impoundments

Michael P. Davies, Peter C. Lighthall, Steve Rice and Todd E. Martin

ABSTRACT
The state of practice for tailings dam and impoundment design is summarized. The design process, which embraces construction, operational and closure issues together with requisite technical aspects, has evolved over the past several decades though the engineering principles have remained the same. The design process has evolved to meet the demands of a regulatory environment that has become increasingly stringent together with more challenging mining economic conditions, as well as alternative tailings management technologies have been developed; including filtered dewatered, stacked deposits; thickened dewatered systems; frozen tailings deposits; and paste disposal. In response to a number of highly publicized tailings impoundment failures, tailings management systems, dam safety programs, and risk assessment techniques have been created and are now part of standard practice and appropriate dam/impoundment stewardship. The concept of environmental sustainability is now an integral component of the tailings dam/tailings impoundment design process and appropriate project conceptualization followed by stewardship of the design are essential to this sustainability.

INTRODUCTION
Environmental factors have increasingly driven tailings dam designs in recent years, but not necessarily to the benefit of engineering safety of tailings dam structures. For while design tools have improved and designs may have become more rigorous with technological improvements, the safety record of tailings dams has not markedly improved. Highly publicised failures continue to occur, resulting in a negative image for the mining industry, and particularly for Canadian mining companies, which have been involved in several of the more prominent failures. Unfortunately, the failure statistics do not tell the real story, which is that design, construction, operation and management of tailings facilities has advanced tremendously over the past thirty years. The security of tailings facilities is now a recognized priority at a corporate level in most large mining companies and the concept of sustainable mining, which clearly involves appropriate mine waste stewardship, is an accepted part of the modern industry.

Tailings, and waste rock, are the waste products of the mining industry. Their disposal adds to the cost of production, and consequently, it is desirable to accomplish their disposal as economically as possible. This requirement for low cost led to development the upstream method of tailings dam construction, which was the standard for tailings disposal up to the mid-1900's, irrespective of site conditions. With the advent of sound engineering practice, it became recognised that there are significant weaknesses and risks in the upstream method of construction under many site conditions. To augment the upstream method, embankment designs were developed using downstream and centreline construction methods. Sound civil engineering designs for embankment slopes, transition zones and filters were applied to tailings dams on an industry wide basis for the first time, beginning in the 1960's. Once designed and constructed based on an empirical and experience-based approach, tailings dam design has since evolved into a formal specialist engineering discipline.

Over the past 30 years, the most significant change in tailings disposal technology has been the recognition of the long-term geochemical risks, particularly the potential for acid drainage and metal leaching. The tailings dam designer of 30 years ago was not tasked with understanding and addressing geochemical issues. Today, geochemical characterisation is one of the most important aspects of tailings disposal planning, and designs for operation and closure are focussed on geochemical issues. These issues often govern not only the type of tailings dam, but also may govern tailings dam site selection. Contaminant loading analysis is now required on practically all

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tailings dam design projects. Closure strategies to prevent long-term geochemical impact to receiving environments are often the driving factors in design of tailings impoundments.

The other significant trend over the last 30 years has been the considerable number of highly publicised failures of tailings dams that have continued to occur with alarming frequency over the past three decades. In response to these large numbers of failures, mining corporations, financial institutions, environmental groups, government regulators and even the general public have instituted much more rigorous scrutiny of tailings management systems. This paper attempts to describe how the state of practice of tailing impoundment design, construction and operation has changed, and to outline some of the technologies that have been developed in recent years.

MINE TAILINGS

Tailings are the finely ground barren minerals left following ore extraction processes. They are typically a product of milling, although in several industries, e.g. the oil sands mining activity in Northern Alberta, large volumes of tailings can be produced without mechanical crushing. In milling, the process begins with crushing mine-run ore to particle sizes generally in the range of millimetres to centimetres. Crushed ore is then further reduced by grinding mills to sizes less than 1 mm in ball mills, rod mills, and semi-autogenous (SAG) mills. Water is added to the ground ore, and the material remains in slurry form throughout the remainder of the extraction process.

The grain size distribution of tailings depends upon the characteristics of the ore and the mill processes used to concentrate and extract the metal values. A wide range of tailings gradation curves exist for various mining operations and consequently, tailings may range from sand to clay-sized particles. For most base metal mines 40% to 70% of the tailings will pass a No. 200 sieve (74 μm). However, some milling processes such as gold extraction may grind the ore so that 90% or more of the tailings pass the No. 200 sieve. Tailings may also include metal precipitates from neutralisation sludges or residues from pressure leaching processes. Such materials may exhibit long-term chemical stability concerns and need to be disposed in secure, lined facilities.

BACKGROUND AND HISTORY OF TAILINGS DAMS

Mining has been carried out in some form for at least 5 000 years. In forms more similar to modern mining, crude millstone crushing and grinding of ore were initially practised in the New World in the 1500's, and continued through the mid-1800's. The largest change over those centuries was the introduction of steam power, which greatly increased the capacity of grinding mills and hence, the amount of barren by-product (tailings) produced.

Minerals of economic interest were initially separated from crushed rock according to differences in specific gravity. The remaining tailings were traditionally routed to some convenient location. The location of greatest convenience was often the nearest stream or river where the tailings were then removed from the deposition area by flow and storage concerns were largely eliminated. Later in the 1800s, two significant developments, which changed mining dramatically, were the development of froth flotation and the introduction of cyanide for gold extraction.

Flotation and cyanidation greatly increased the world's ability to mine low-grade ore bodies, and resulted in the production of still larger quantities of tailings with even finer gradation (i.e., more minus 74 μm material). However, tailings disposal practices remained largely unchanged and, as a result, more tailings were being placed and transported over greater distances into receiving streams, lakes and oceans.

Around 1900, remote-mining districts began to develop, and attract supporting industries and community development. Conflicts developed over land and water use, particularly with agricultural interests. Accumulated tailings regularly plugged irrigation ditches and "contaminated" downstream growing areas. Farmers began to notice lesser crop yields from tailings-impacted lands. Issues with land and water use that led to the initial conflicts then led to litigation in both North America and Europe. Legal precedents gradually brought an end to uncontrolled disposal of tailings in most of the western world, with a complete cessation of such practices occurring by about 1930.

To retain the ability to mine, industry fostered construction of some of the first dams to retain tailings. Early dams were often built across a stream channel with only limited provisions for passing statistically infrequent floods. Consequently, as larger rainfalls or freshet periods occurred, few of these early in-stream dams survived. Very little, if any, engineering or regulatory input was involved in the construction or operation of early dams.

Mechanized earth-moving equipment was not available to the early dam builders. As a result, a hand-labour construction procedure (the initial upstream method) was developed. A low, dyked impoundment was initially filled with hydraulically-deposited tailings, and then incrementally raised by constructing low berms above and behind the dyke of the previous level. This construction procedure, now almost always mechanized, remains in use at many mines today.

The first departure from traditional upstream dam construction likely followed the failure of the Barahona tailings dam in Chile. During a large earthquake in 1928, the Barahona upstream-constructed dam failed, killing more than 50 people in the ensuing, catastrophic flowslide. The Barahona dam was replaced by a more stable downstream dam, which used cyclones to procure coarser-sized material for dam construction from the overall tailings stream. By the 1940's, the availability of high-capacity earthmoving equipment, especially at open-pit mines, made it possible to construct tailings dams of compacted earthfill in a manner similar to conventional water dam construction practice (and with a corresponding higher degree of safety).

The development of tailings dam technology proceeded on an empirical basis, geared largely to the construction practices and equipment available at the time. This development was largely without the benefit of engineering design in the contemporary sense. Nonetheless, by the 1950's many fundamental dam engineering principles were understood and applied to tailings dams at a number of mines in North America. It was not until the 1960's, however, that geotechnical engineering and related disciplines adopted, refined, and widely applied these empirical design rules. The 1965 earthquake-induced failures of several tailings dams in Chile received considerable attention and proved to be a key factor in early research into the phenomenon of liquefaction. Earthquake-induced liquefaction remains a key design consideration in tailings dam design.

Issues related to the environmental impacts from tailings dams were first seriously introduced in the 1970's in relation to uranium tailings. However, environmental issues related to mining had received attention for centuries. Public concerns about the effects of acid rock drainage (ARD) have existed for roughly 1,000 years in Norway. Public concerns were similarly expressed hundreds of years ago in Spain and in Greece.

In the early 1970's, most of the tailings dam structural technical issues (e.g. static and earthquake induced liquefaction of tailings, seepage phenomena and foundation stability) were fairly well understood and handled in designs. Probably the only significant geotechnical issue not recognised by most designers was the static load-induced liquefaction (e.g. the reason for many previously "unexplained" sudden failures). However, issues related to geochemical stability were not as well recognised, and tailings impoundments were rarely designed with reclamation and closure in mind.

Over the past 30 years, environmental issues have grown in importance, as attention has largely turned from mine economics and physical stability of tailings dam to their potential chemical effects and contaminant transport mechanisms. Physical stability have remained at the forefront, as recent tailings dam failures have drawn unfortunate publicity to the mining industry, with severe financial implications in many cases. In response, a great many mining companies, at a corporate level, have identified safe tailings management as a priority, and have made resources available to address that priority. A significant tailings impoundment failure will almost certainly have a direct cost in the tens of millions of dollars and indirect costs, including devaluation of share equity, often many times the direct costs. In all of the tailings dam failure cases, a few examples of which are noted later in this paper, relatively simple, well-understood structural failure mechanisms were found to be at fault in causing the incidents.

THE ROLE OF GOVERNMENT REGULATIONS AND WORLDWIDE STANDARDS

As a new century unfolds, regulators and non-government organisations worldwide are becoming increasingly educated about tailings dam design and stewardship requirements. Lending agencies have also dramatically increased their technical requirements prior to funding new or expanded projects. This trend in education is a welcome development as candid discussions on risk levels for any given technical issue can be carried out with good understanding from all stakeholders. However, an unwelcome development has been the significant amount of non-technical and often misguided opposition and it is this latter situation that causes the most grief for the mining industry. No longer can a mine development proponent simply agree to meet the criteria of the senior governing authority and provide evidence of credible design. Often, several levels of government and non-government organisations must be satisfied with the proposed mining development. The tailings impoundment is often the most critical component of a mine development in the eyes of regulators and third party interest groups.

Many developing countries, where the international mining industry is focussing considerable attention, have only recently enacted regulations pertaining to tailings disposal. These regulations typically are based on those in place in more developed jurisdictions. Regulators in developing countries, however, all too often lack the resources and the expertise to fully implement these regulations. As a result, regulations in developing countries are often technically prescriptive. Such technically prescriptive regulations provide a false sense of security, since failures are so often the result of a combination of design flaws and improper stewardship. Many failures have occurred at facilities that conformed with all regulations, except for the most important of all (the dam failed).

The problem of limited resources for regulators is by no means unique to developing jurisdictions. Government budget cutbacks in jurisdictions such as Canada and North America mean that the mining industry is striving to a condition of "co-regulation", in partnership with regulators. This is a welcome trend, placing the initiative for continuous improvement and safety of tailings disposal facilities squarely with the mining industry.

Although visible exceptions continue to arise, as noted later in this paper, the modern tailings facility is typified by a well-designed and constructed facility that has met several levels of regulatory and non-regulatory scrutiny and has received corporate attention to the highest level. Most of the world's mining companies have multi-national operations, and to continue operation, and to attract share capital, must be seen to have exemplary environmental and safety records. Regulators, mining companies and international environmental organisations have developed numerous programs to ensure a high level of security of tailings disposal systems. Examples of some of these programs include:

- The Mining Association of Canada (MAC), has recently published a document entitled "A Guide to the Management of Tailings Facilities" (MAC, 1998);
- The Canadian Dam Association (CDA) recently updated its dam safety guidelines (CDA, 1999). The update focussed in large part on incorporating elements specific to the safety of tailings dams;
- The International Committee on Large Dams (ICOLD), and related organizations, have published numerous materials with regards to tailings dams;
- The United Nations Environment Programme, Industry and Environment (UNEP), and the International Council on Metals and the Environment (ICME) have been active in recent years in sponsorship of seminars, and publication of case studies (UNEP-ICME, 1997 & 1998), related to tailings management;
- Several major Canadian-based mining companies have established corporate policies and procedures to ensure that all personnel involved in stewardship of tailings facilities, from the corporate level to the operators, clearly understand their roles and responsibilities (e.g., Siwik, 1997);
- Numerous mining companies, including Syncrude, Kennecott Utah Copper, and Inco, retain a board of eminent geotechnical consultants to provide independent review and advice in terms of the design, operation, and management of their tailings facilities. These programs are described in McKenna (1998), Dunne (1997) and (McCann, 1998); and
- Many mining companies have regular third party risk assessment programs for their tailings facilities, in which experienced consultants, usually teamed with the owner's personnel, carry out audits of tailings facilities.

Of all of the above measures, the authors consider the last two to be of most importance.

SITE CHARACTERIZATION

A major factor in managing tailings impoundment failure risk is to carry out adequate site characterisation for siting and designing tailings impoundments. Experience in tailings impoundment design and construction has served to emphasise the critical importance of a proper understanding of the geology of impoundment sites, and of an appreciation of how geology will affect the design, construction, and performance of the tailings facility. Many of the catastrophic structural failures have occurred as a direct result of inadequate site characterization.

Site characterization has always relied on carrying out thoughtful site investigations to develop a thorough understanding of site geology. The judgement of experienced engineering geologists should play a major role in site characterization. Traditional tools of geological mapping, air photo interpretation, test pitting, geotechnical drilling continue to form the basics of site investigation. However, significant changes and technological improvements have been made, so that tailings designers have a much wider range of tools from which to choose. As well, the needs of site characterization have added new demands. Some significant advances/changers in recent years include:

- Generally improved technology in site investigation techniques;
- Increased emphasis on water management and environmental characterization of the site, particularly with regards to hydrogeology;
- Greater importance being stressed on recognising geochemical issues, such as acid rock drainage (ARD), and in terms of attenuation of groundwater contaminant transport;
- Greater emphasis on closure and reclamation considerations and the site investigation required to support that design; and
- Development of site investigation methodologies for characterization of liquefaction potential that have evolved significantly.

Technological Advances

The technology available for use in execution of site investigation programs has advanced greatly in recent decades. There are more and better-equipped geotechnical drilling contractors, with more powerful and efficient drilling equipment. Examination of aerial photographs represents an essential method of site assessment, and the quality of aerial photography has improved considerably. Remote sensing, and satellite imagery techniques are now available and prove invaluable on many projects. Geographic Information Systems (GIS) have been developed and represent a major advance in the collation and ultimate usefulness of site data. The world earthquake database, and the means for using it in probabilistic characterisation of site seismicity, a significant consideration for tailings impoundment design, has also advanced. Geophysical methods have also become more useful as techniques have become more reliable and analysing the data within complex solutions more readily available with the dramatic increase in computer processing power.

Perhaps the most significant advance has been the development and widespread application of electronic piezocone technology for geotechnical and environmental characterization of clay, silt, and sand soils. The piezocone provides end bearing, friction, and porewater pressure data on a near-continuous basis as the probe is advanced. From these data, such geotechnical information as soil type, shear strength, in situ state, sensitivity, relative density/consistency, and liquefaction susceptibility can be determined using semi-quantitative relationships. Resistivity measurements are also possible, enabling contaminant plumes in groundwater to be delineated. The piezocone can be used effectively as a piezometer to characterize porewater pressure gradients, and for in situ estimation of hydraulic conductivity and consolidation parameters. Piezocone technology is particularly well suited to geotechnical and environmental profiling of mine tailings deposits.

Emphasis on Environmental Site Characterisation

The increasing emphasis on environmental protection in siting, design, construction, operation, and closure of tailings impoundments has placed increased emphasis on environmental aspects of site investigations.

In terms of siting and design of a tailings impoundment, groundwater quality protection is perhaps the most significant environmental protection aspect requiring investigation. To evaluate potential groundwater quality impacts, the following are essential:

- Establish baseline (pre-development) groundwater quality conditions, by collecting surface water and groundwater samples (monitoring wells).
- Identify principal hydrogeologic units (overburden and bedrock), and develop a hydrogeologic model of the site. In most cases, a larger, regional hydrogeologic model is also required.
- Model tailings impoundment development and estimate contaminant loadings. This may require characterization of attenuation capacity in the hydrogeologic units. Based on that model, and on parametric (sensitivity) analyses, determine compliance versus non-compliance at appropriate locations.

Seepage modelling, and contaminant transport modelling, can now be carried out using very powerful yet simple to use finite element and finite difference computer models. These models include both saturated and unsaturated flow regimes, which allow better prediction of behaviour at the important interface between the tailings and the atmosphere, where much of the geochemical activity is occurring. The development of these tools has to some degree driven the need to obtain the data that allows their effective use. The foundation of these models is a reasonable hydrogeologic model, and the foundation of a hydrogeologic model is an understanding of local and regional geology.

Geochemical Characterization

The role of geochemical issues in tailings impoundment design, and particularly closure, is equally as important as geotechnical issues driven primarily by the critical issue of acid rock drainage (ARD). Tailings that are potentially acid generating require closure strategies, and therefore impoundment designs, that will prevent/control acid generation. This issue is a major component of initial mine studies involve addressing the acid generation potential of tailings and/or waste rock in a comprehensive manner.

Susceptibility of dam fill and foundation materials to structural change due to the effects of ARD generation in the tailings deposit must also be considered. For example, if an impervious core dam is being considered, then the mineralogy of the core material should be checked, as dissolution of carbonates within the material could greatly increase the permeability of the core. Similarly, geochemical effects on materials being considered as a clay liner for the impoundment must also be considered. As another example of the importance of this issue, there have been many documented case histories of ARD resulting in clogging of internal drainage zones within tailings dams, requiring in many cases extensive remedial measures.

NEW DEVELOPMENTS IN TAILINGS DISPOSAL

A number of improvements have been made in tailings disposal technology and tailings dam design, both to improve on the weaknesses of previous practices and also to take advantage of tailings processing technologies. Geochemical aspects now largely drive the siting, of a tailings impoundment, the design of retention structures, and tailings disposal technology. There have been technologies put forward as panaceas for tailings disposal problems, which have turned out to be flawed in practice. These improvements can be categorized as changes in basic management practices and changes in tailings characteristics through pre-discharge dewatering.

Designing for Geochemical Issues

Geochemical issues have become highly prominent as severe acid generation problems became apparent at a number of mature mines around the world. Some of these mines, which had been operated by smaller mining companies, became orphan sites, leaving significant legacies for future generations. The majority of the acid drainage mine sites have become very expensive legacies for the major mining companies that owned them. It has been necessary to develop and operate acid drainage collection and treatment systems for continued operation and closure of numerous mines. Capital costs for ARD collection and treatment systems have been in the several tens of millions of dollars, with ongoing operating costs up to several millions of dollars annually. As a result, companies developing new mines have focussed on methods to predict and prevent or reduce acid generation from tailings.

Considerable research, for example CANMET's Mine Effluent Neutral Drainage (MEND) program, was carried out in the 1980’s and 1990’s, to assess viable methods of acid drainage control. The most significant conclusion of the past 20 years is that it is far easier (economic) to prevent ARD in the first place than to control it. From a number of existing sites where tailings had been placed in lakes in northern Canada, it was concluded that long-term submergence of acidic wastes was probably the most effective means of ARD control. Considerable work has also been done on placement of impervious closure covers over tailings to prevent ingress of air and water. Sophisticated designs of multiple-layer covers, incorporating impervious zones, pervious capillary barriers and topsoil for vegetation growth, have been developed. Covers have been found to present the risk of long term cracking or erosion, and to be ineffective in excluding air, so are less favoured solutions than submergence from the geochemical standpoint. Some of the main technologies for reduction of ARD potential from sulphide bearing tailings are the following:

1. **Design for submergence by flooding the tailings at closure.** This is a solution, which is being increasingly encouraged and accepted by regulators. However, the authors are concerned that flooded impoundments may create a risky legacy. The more traditional closure configuration for tailings impoundments has been to draw down water ponds as completely as possible, to reduce the potential for dam failure by overtopping or erosion. To raise water levels in impoundments formed by high dams could present considerable long-term risk. One of the reasons that closed tailings impoundments have traditionally proven to be generally more safe, from the physical stability perspective, than operating impoundments is the relatively more “drained” condition of closed impoundments that do not include a large water pond. The flooded closure scenario represents an “undrained” condition that does not allow this improvement in physical stability to develop, so the risk does not decrease with time.
2. **Treatment of tailings to create non-acid generating covers.** To avoid the necessity of flooding impoundments, non-reactive covers of tailings can be placed on the top of the impoundment on the last few years of operation. It has been shown in several mining operations, for example at the Inco Ontario Division central milling operation in Copper Cliff, Ontario, that by the relatively inexpensive installation of some additional flotation capacity, pyrite can be removed to the level that the tailings can be made non-acid generating. The upper non-acid generating tailings placed on top can be left as a wide beach for dam safety, while the underlying mass of potentially acid generating tailings remains saturated below the long-term water table in the impoundment. Normally, the small amount of pyrite removed by flotation can be disposed as a separate tailings stream, placed in the deepest part of the impoundment where it can be left flooded.
3. **Lake or ocean subaqueous disposal.** The surest, safest and most cost-effective solution to prevent ARD is sub-aqueous disposal in a lake or the ocean. Tailings will remain permanently submerged and have shown to be non-reactive under water and to have few permanent environmental impacts. The challenge for this solution is that regulators have become reluctant to permit lake or ocean disposal, and there are not always appropriate sites available. In addition, the public often reacts emotionally and negatively to the concept of such disposal, despite the considerable benefits of these approaches. The authors are aware of at least two examples where public pressure incited regulators to demand that existing operations switch from ocean and lake disposal to on-land impoundments, with the result that environmental problems actually increased. The authors do note a slight trend to re-acceptance of subaqueous disposal, particularly in the marine environment, as the true environmental impact of the technique can be demonstrated to be almost negligible in certain instances. Moreover, the corporate risks and environmental liabilities associated with surface tailings storage on many projects grows to the point where project viability is threatened without looking to environmentally acceptable alternatives including subaqueous disposal.

Improved Basic Design Concepts

Improved Upstream Construction. Considerable attention has been given to improving traditional upstream dam construction to make the technique not only economical but also stable under both static and dynamic conditions. Numerous failures of upstream constructed dams have occurred. The failures have been the results of earthquakes, high saturation levels, steep slopes, poor water control in the pond, poor construction techniques incorporating fines in the dam shell, static liquefaction, and failures of embedded decant structures. Most failures have involved some combination of the above weaknesses.

Based on the above experiences, and through the use of improved analytical tools (computer programs for stability, seepage, and deformation under both static and seismic conditions), safe, optimised designs have been developed. Some of the key design features that have been added include:

- Underdrainage, either as finger drains or blanket drains, to lower the phreatic level in the dam shell;
- Beaches compacted to some minimum width to provide a stable dam shell. Beaches are compacted by tracking with bulldozers, which are also used for pushing up berms for support of spigot lines;
- Slopes designed to a lower angle than was used for many failed tailings dams. Slopes are generally set at 3 horizontal to 1 vertical or flatter, depending on the other measures incorporated into the designs. Steeper slopes, without an adequate drained and/or compacted beach, create the potential for spontaneous static liquefaction - a phenomenon not widely recognized in 1972 but one responsible for a number of major tailings dam failures.

Figure 1 below shows a typical section of an improved upstream design.

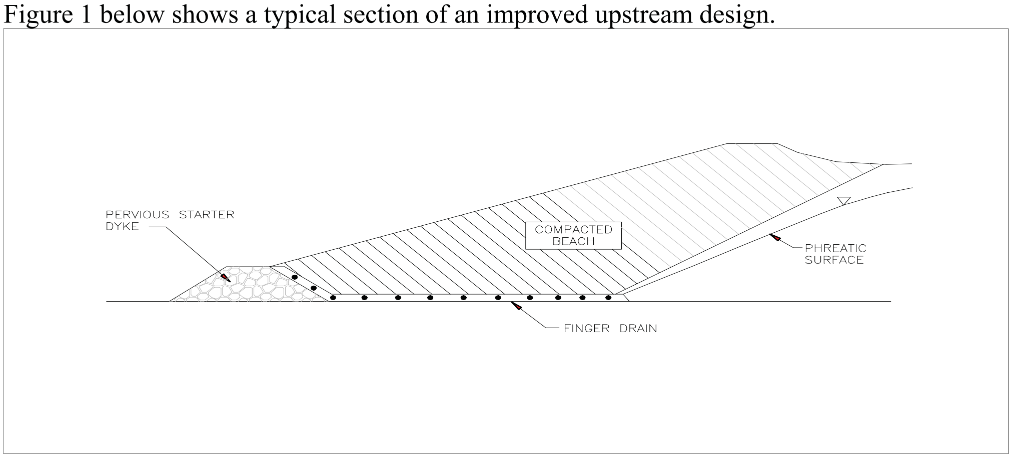

Figure 1 Typical section of improved upstream tailings dam design

Lined Tailings Impoundments. With the advent of larger gold mining operations, and the almost universal use of sodium cyanide as an essential part of gold extraction, the need came about to develop impervious impoundments to contain cyanide solutions. Although cyanide is in most forms an unstable compound that naturally breaks down on exposure to air, it can be very persistent and migrate long distances in groundwater. As well as cyanided gold tailings, other types of tailings may also be considered potentially contaminating. For protection of aquifers, where tailings impoundments are not sited over impervious soils or bedrock and embankment cut-offs are not sufficient to reduce seepage, it is often necessary to design and construct a liner over the base of a tailings impoundment. Great progress has been made in liner design and construction practise.

Liners may be as simple as selective placement of impervious soil to cover outcrops of pervious bedrock or granular soils, or may need to be a composite liner system constructed over the entire impoundment. Where geomembrane liners are used, it is normal practise to incorporate a drainage layer above the geomembrane, to reduce the pressure head on the liner and minimise leakage through imperfections in the liner. Another benefit of such under-drainage is that a low pore pressure condition is achieved in the tailings, giving them a higher strength than would exist without such under-drainage. The drainage layer typically consists of at least 300 mm of granular material, with perforated pipes at intervals within the drainage layer. The pipes are laid to drain water extracted from the base of the tailings deposit and to discharge to a seepage recovery pond. Figure 2 below shows two typical configurations of lined impoundments. Figure 2a shows a liner extending up the face of the embankment, requiring special detailing of drainage pipe penetrations through the liner. In Figure 2b, the liner extends beneath the embankment. In the latter case, care must be taken to design for lower foundation shear strength for the downstream slope of the embankment, as the liner may form a plane of weakness.

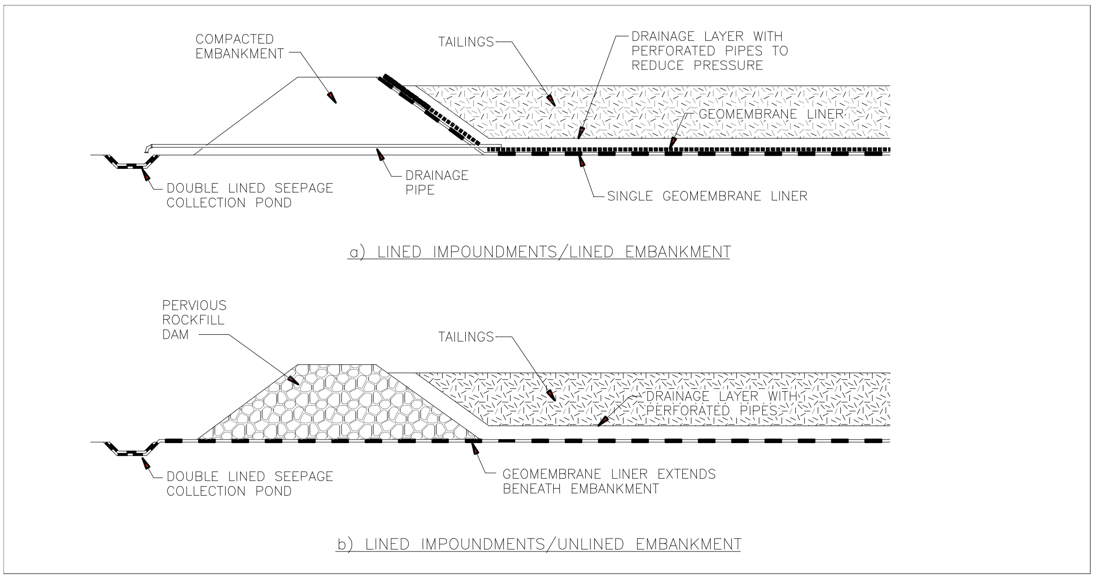

Figure 2 Conceptual sections of lined impoundments with underdrains

Dewatering Technologies. As shown on Figure 3 below, the basic segregating slurry is part of a continuum of water contents available to the tailings designer in 2000. Although tailings dewatering was previously practised for other purposes in the mining process, until recently the only form of tailings for most tailings facilities was a segregating, pumpable slurry with geotechnical water contents of well over 100%.

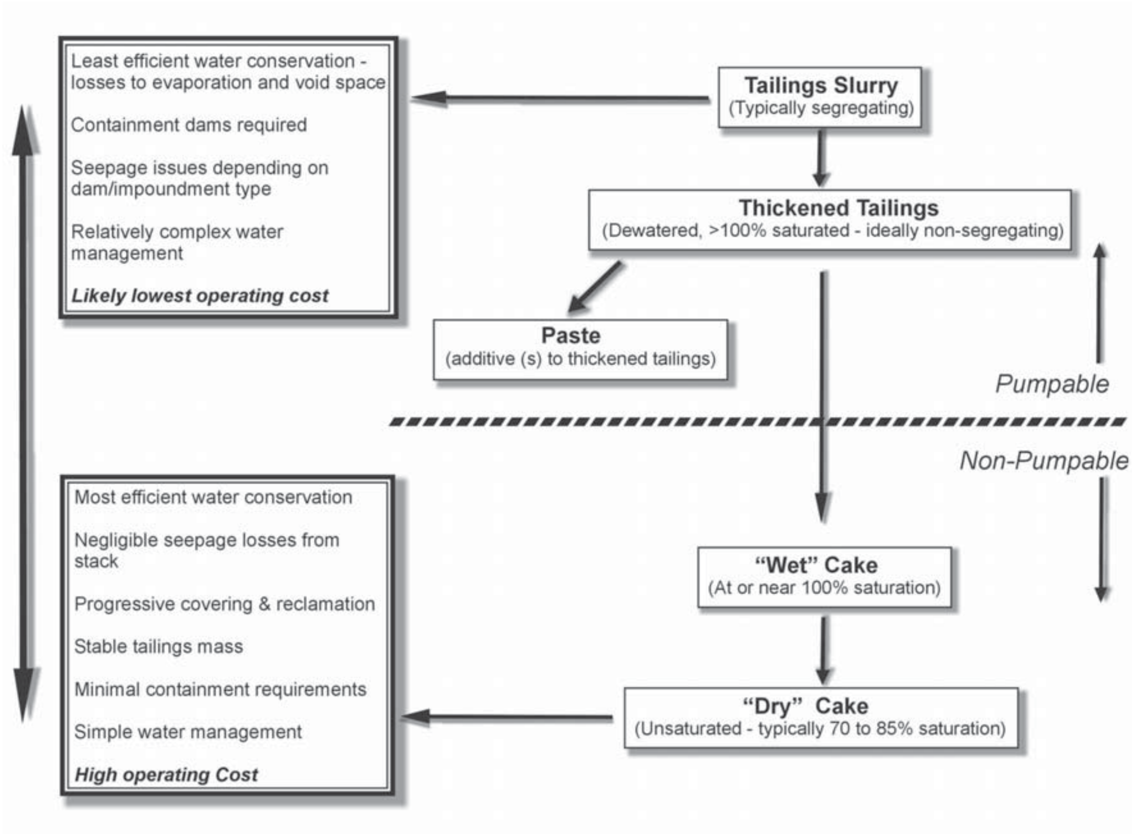

Figure 3 Classification of Tailings by Degree of Dewatering (after Davies and Rice, 2001)

There are several candidate scenarios where dewatered tailings systems would be of advantage to the mining operation. However, dewatered tailings systems have less application for larger operations for which tailings ponds must serve dual roles as water storage reservoirs, particularly where water balances must be managed to store annual snowmelt runoff to provide water for year round operation.

"Dry" Cake filtered tailings disposal. Development of large capacity, vacuum and pressure belt filter technology has presented the opportunity for disposing tailings in a dewatered state, rather than as a conventional slurry. Tailings can be dewatered to less than 20% moisture content (using soil mechanics convention, in which moisture content is defined as weight of water divided by the dry weight of solids). At these moisture contents, the material can be transported by conveyor or truck, and placed, spread and compacted to form an unsaturated, dense and stable tailings stack (often termed a "dry stack") requiring no dam for retention. While the technology is currently considerably more expensive per tonne of tailings stored than conventional slurry systems, and would be prohibitively expensive for very large tonnage applications, it has particular advantages in the following applications:

- **In very arid regions, where water conservation is an important issue.** The prime example of such system is at the La Coipa silver/gold operation in the Atacama region of Chile. A daily tailings production of 18,000 t is dewatered by belt filters, conveyed to the tailings site and stacked with a radial, mobile conveyor system. The vacuum filter system was selected for this site because of the need to recover dissolved gold from solution, but is also advantageous for water conservation and also for stability of the tailings deposit in this high seismicity location; and

- **In very cold regions, where water handling is very difficult in winter.** A dewatered tailings system, using truck transport, is in operation at Falconbridge's Raglan nickel operation in the arctic region of northern Quebec. The system is also intended to provide a solution for potential acid generation, as the tailings stack will become permanently frozen. A dry stack tailings system is also being planned for a new gold project in central Alaska.
- **Relatively low tonnage operations.** A separate tailings impoundment can be avoided all together by having a tailings/waste rock co-disposal facility.
- **Regions where a "dry landscape" upon closure is required.** The tailings area can be developed and managed more like a waste dump and therefore avoids many of the operation and closure challenges of a conventional impoundment.

Moreover, filtered tailings stacks have regulatory attraction, require a smaller footprint for tailings storage (much lower bulking factor), are easier to reclaim and close, and have much lower long-term liability in terms of structural integrity and potential environmental impact. Figure 4 below shows a photograph of a large dry-stack tailings system. Davies and Rice (2001) present a state-of-practice overview of dry stack filtered tailings facilities.

Figure 4 Example Dry-Stack Tailings Facility

Thickened/paste technologies. It is critical before basing mining operations on new technologies to carry out adequate engineering studies to demonstrate feasibility. Several tailings disposal technologies have been introduced to the mining industry that, over the past 30 years, have not proven out to be as effective as may have been hoped. While all have contained good ideas, they have often been wholly or partially unsuccessful, or have not found extensive application to date. However, two developments will likely see renewed emphasis over the coming years.

1. **Paste disposal.** The development of improved thickener technology has led to tremendous advances in paste tailings for underground backfill operations. Paste tailings are essentially the whole tailings stream, thickened to a dense slurry (previously only the coarse fraction of tailings was separated from tailings for use as backfill). Cement is added to the paste and the material is pumped underground to use as ground support in mined out stopes. Advocates of paste technology have promoted its use for surface tailings disposal, claiming that it can be placed in stable configurations with the cement

providing adequate strength. However, the technology has not been shown to be economically pragmatic at a large scale for many surface applications.

2. **Thickened disposal.** Thickened tailings are paste without the additives. Thickened disposal is a technique that has been proposed for over 25 years and has been implemented in a few operations. The main premise of thickened disposal is that tailings may be thickened to a degree that they may be discharged from one or several discharge points to form a non-segregating tailings mass with little or no water pond. In the most classical connotation, thickened tailings is assumed to form a conical mass with the tailings surface sloping downwards from the centre of the cone. A thickened tailings system, if successful, should require lower retaining dykes, as storage is gained by raising the centre of the impoundment. It had been proposed that, at the start of operations, the tailings could be thickened to a lesser degree, when flatter slopes on the impoundment would suffice, then later thickened to a higher degree as it became necessary to raise the central point of the cone. In most instances where thickened tailings was implemented, thickening technology was not capable of producing a consistent non-segregating material, so fines would form a very flat slope and require additional dyking at the toe. As well, flatter than projected slopes were experienced, and it was not possible to steepen these slopes to avoid extensive land use impact. From the above experiences, the thickened disposal did not become accepted. It has, however, been very successful in very arid regions, such as the gold mining districts of Australia. In recent years, high density thickening technology has been developed which make it useful to re-examine thickened disposal. The authors are aware of several major mining projects considering thickened tailings as an alternative management practice.

INSTRUMENTATION AND MONITORING

General

The reasons for instrumentation and monitoring of tailings dams are as set out by Klohn (1972). Reworded, these reasons are as follows:

1. To check that the environmental performance of the facility is meeting design intent, with no downstream out-of-compliance impacts on surface water quality and groundwater quality.
2. To confirm that the dam and impoundment are safe from the physical standpoint, from construction through operation and closure.
3. To provide the data required for confirmation and/or optimisation of design and construction through successive stages of impoundment construction and development.

The key consideration to be accounted for in instrumentation and monitoring of tailings impoundments is that they are dynamic structures, changing nearly continuously, constructed over periods of several to many years, often under the stewardship of changing personnel. An important consideration that has come into focus in recent years is the realisation that monitoring must continue into the closure phase, and that only in rare and favourable circumstances can a truly “walk-away” closure scenario, requiring no ongoing monitoring or maintenance, be achieved.

Technology and Communication Improvements

Technology improvements in instrumentation and monitoring, and in the ability to communicate the results, have been remarkable over the last 30 years. Instrumentation has become more advanced, robust, accurate, and reliable. It can be read more quickly and efficiently due to improved data logging equipment. It is now possible to monitor instrumentation remotely, using automatic data loggers and telemetry to transmit data to the office. Personal computers make plotting of the data in graphical form simple and rapid. Results can be emailed to the design engineer’s office for quick review. Threshold (alert) levels can be included on these plots to visually define where the data plots in relation to safe versus unsafe conditions. Photographs can likewise be emailed for quick review by other mine personnel and/or the designer. Video surveillance cameras may even have some application at very remote sites, particularly for monitoring during the closure phase.

Despite these favourable trends, there is a significant caveat: technology advances are not yet at the point where they can replace visual inspection of the structure by a qualified, experienced engineer. Nor can technology serve as a substitute for application of imagination and judgement to the interpretation to monitoring data.

Training of Operations Personnel and Documentation of Monitoring Program

The international mining industry, and the mining industry in Canada in particular, has become increasingly focused on tailings dam safety issues. Instrumentation and monitoring represent important tools in tailings dam safety programs. It is becoming increasingly common for mining companies to require that their employees responsible for tailings dam operation receive training in dam monitoring.

The instrumentation and monitoring program for a tailings dam is normally included in a comprehensive Operations Manual (a document required by legislation in an increasing number of jurisdictions). Having an Operations Manual in place is now considered to be state of practice for tailings facilities management, and provides the following benefits:

1. It provides a concise, practical document that can be used by site operating personnel for guidelines on operation and monitoring of the tailings facilities.
2. It serves as a useful training document for new personnel involved in tailings management and operations.
3. Its existence provides reassurance to senior level management, and to regulators, that formalised practices are in place for the safe operation of the facility.
4. It demonstrates due diligence on the part of the owner.

Environmental Performance

Requirements for environmental performance have become more stringent, paralleling environmental regulations. Environmental performance monitoring can include the following:

- Surface water quality, downstream and upstream of the tailings impoundment.
- Groundwater quality, downstream and upstream of the tailings impoundment.
- Tailings pond supernatant water quality.
- Acid Base Accounting (ABA) testing of waste rock and tailings to determine susceptibility to acid rock drainage generation.
- Air quality (dust).
- Fisheries resources, and maintenance of minimum flows to fish bearing watercourses.
- Progress of revegetation where test plots and/or progressive reclamation are underway.

Most mining operations have on staff an Environmental Superintendent or Coordinator, who typically takes a keen interest in the operation and monitoring of the tailings impoundment.

Dam Safety Monitoring

The increasing focus on tailings dam safety brings with it an increasing awareness of the importance of a good monitoring and instrumentation program to confirm that the tailings dam is in a safe condition. Guidelines in terms of monitoring of tailings dams are provided by ICOLD (1994) and the Canadian Dam Association (1999), among others.

Monitoring to confirm tailings dam safety involves the following components:

- Periodic, detailed visual inspections of the tailings dam and its associated appurtenant structures (spillway, decants, diversion ditches). These inspections are carried out and documented by mine personnel who have received training in potential modes of dam failure and their warning signs. Any unusual conditions or concerns must be immediately reported.

- Definition of “green light” (safe), versus “yellow light” (caution) and “red light” (unsafe) conditions, and a pre-determined course of action (who to contact, what to do, increased frequency of monitoring, etc.) if yellow or red light conditions are noted.
- Reading of instrumentation (piezometers, survey monuments, inclinometers, seepage weirs, etc.) according to a set schedule, presentation of the results in graphical form, and interpretation and reporting of the results to the appropriate personnel.

The scope and frequency of dam safety monitoring will change through the life of a tailings impoundment, depending on the phase (construction versus operation versus closure) and on the consistency of monitoring results.

Confirmation and Optimisation of Design and Construction

Tailings dam engineering practice places considerable reliance on monitoring of the structure performance to confirm satisfactory performance, and to confirm design assumptions. Tailings dam construction, because it happens on a near-continuous basis, provides the opportunity to optimise design and construction over the life of the facility. This is the basic tenet of the observational method (Peck, 1969), a risk management method accepted and used in geotechnical engineering to avoid initial designs that may be overly conservative and overly expensive. As tailings disposal represents a cost rather than a profit centre to mining operations, the advantages of this method to optimise design and construction are obvious.

The elements of the observational method are illustrated schematically on Figure 5. Monitoring and instrumentation represent an integral component of the method. The observational method has a number of limitations, as follows:

- The method is not suitable for failure modes that can develop very quickly, with little or no warning, examples being static liquefaction of loose tailings, or a brittle, over-consolidated clay deposit that undergoes significant loss in strength with minimal straining. Such failure modes can only be properly addressed by good design.
- Once unfavourable conditions are noted, there must be sufficient time, and resources available, to react, putting in place measures that are pre-determined, an essential requirement of the method.
- The method cannot compensate for an inadequate site investigation program.
- The monitoring program in support of the observational method must be properly designed, not just to confirm anticipated conditions, but, more importantly, to detect unanticipated, unfavourable conditions.

Failure to recognise these limitations represents an abuse of the process, which then goes from being the observational approach to the “hope for the best” approach.

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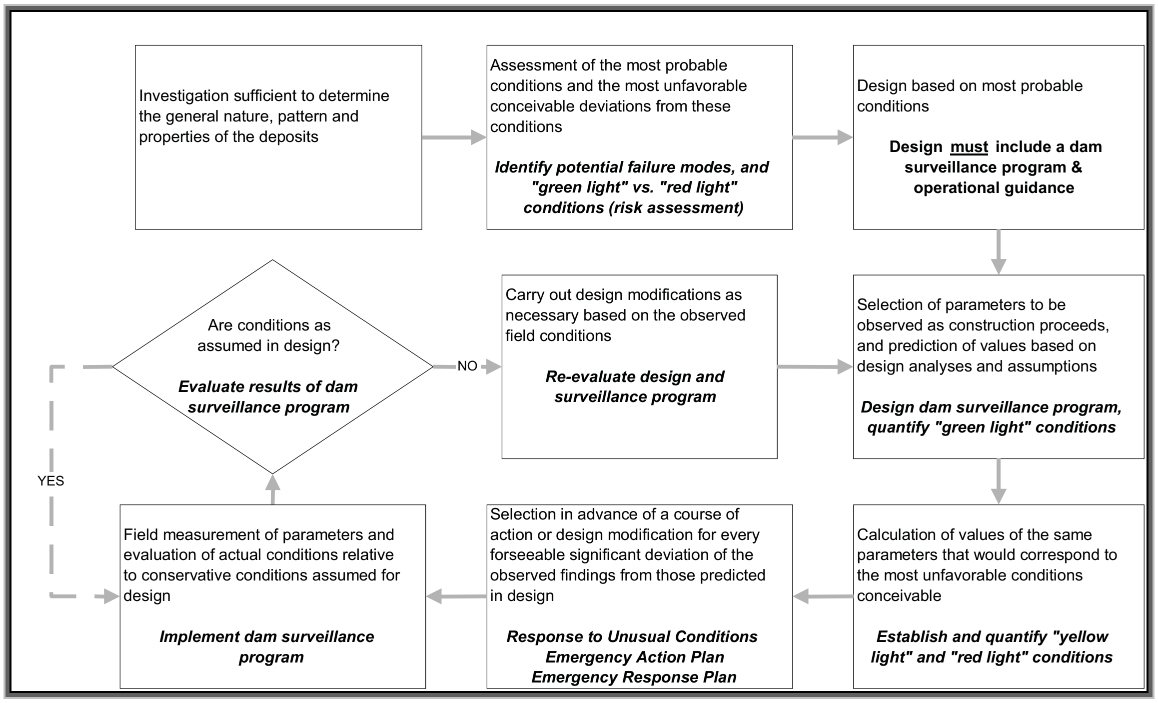

Figure 5 Elements of the Observational Method (adapted from Peck, 1969)

Operational Monitoring

Tailings and water management plan must be formulated for any tailings impoundment, and these must be monitored regularly. Elements of such plans include:

- Tailings deposition schedule;
- Storage versus elevation relationship for the impoundment;
- Operation of diversion structures;
- A mass balance model; and
- Pond filling and dam raising schedule.

Tailings and water management plans are projections, and require updating and calibration against actual conditions on a regular basis, usually no less frequently than annually. Operational monitoring data required for this purpose are as follows:

1. Measured precipitation, evaporation, runoff and snowpack data (mass balance models typically assume average annual conditions, broken down to a monthly basis). Runoff data is particularly useful in confirmation and adjustment of assumed runoff coefficients.
2. Regular tailings beach surveys and soundings to determine above and below water tailings slopes, and to allow the elevation versus storage volume curve for the impoundment to be updated.
3. Recording of tailings discharge points, elevations, and tonnages of tailings discharged from each point.
4. Pond level measurements, no less frequently than monthly.
5. Operation of reclaim barge, decants, spillways, etc.
6. Other water inflows to the impoundment (e.g. mine water) and outflows (e.g. water discharged directly, or discharged following treatment).

The Operations Manual should describe the data required, the frequency with which it is to be collected, and the manner in which it is to be collated and reported.

CONSTRUCTION AND OPERATION PROBLEMS

Davies et al. (2000) note that if one becomes a student of tailings impoundment case histories, an interesting conclusion arises. Tailings dam failures, each and every one, are entirely explainable in hindsight. These failures cannot be described as unpredictable accidents. There are no unknown loading causes, no mysterious soil mechanics, no "substantially different material behaviour" and definitely no acceptable failures. There is lack of design ability, poor stewardship (construction, operating or closure) or a combination of the two, in each and every case history. Tailings impoundment operational "upsets" or more catastrophic failures are a result of design and/or construction/operational management flaws - not "acts of god".

Should an severe upset or breach failure occur, several ramifications can be expected including, but not limited to:

- Extended production interruption;
- Possible injury and, in extreme cases, loss of life (there are more than 1100 documented fatalities attributable to tailings impoundment failures, Davies and Martin, 2000);
- Environmental damage;
- Damage to company and industry image;
- Financial impact to mine, corporate body and shareholders; and
- Legal responsibility for company officers

For perspective, a few recent failures are presented with their summary characteristics and impacts.

Stava, Italy, 1985

Static liquefaction collapse of two dykes, with release of 190,000 m^3 mud wave, travelling 10 km down a valley and killing 268. Legal ramifications for mine owners.

Bafokeng, 1974 and Merriespruit, 1994, South Africa

Tailings paddocks overtopped as a result of high rainfalls on ponds with excess water storage. Release of 3 million m^3 and 600,000 m^3 respectively, with 12 and 17 deaths and immense property damage.

Omai Gold Mine, 1995, Guyana

Dam failure due to inadequate internal filter design/construction led to release of cyanide solution to river. Although minimal short-term environmental impact was indicated, extended production loss, loss of share value and worldwide outrage resulted.

Los Frailes, 1998, Spain

Shallow foundation failure led to translation of dam shell and subsequent breach of one "cell" of tailings impoundment resulting in the release of 4 to 5 million m^{3}of water and tailings slurry. Slurry inundated some significant areas of agricultural land.

DESIGN FOR CLOSURE AND RECLAMATION OF TAILINGS IMPOUNDMENTS

General

Many old mining operations, and the environmental problems associated with them, are now in the care of governments. Regulators, the public, and lending agencies are no longer willing to accept mining operations resulting in long term environmental legacies for governments. Therefore, closure and reclamation considerations have become perhaps the most important driving factors in siting, design, and permitting of mining projects. Successful closure and reclamation of tailings impoundments to a secure condition are an obvious necessity. Keeping closure in mind from day one is very much in the best interest of a mine operator, as it facilitates the most effective closure and reclamation of the site, and also avoids high costs at the end of the mine life. Regulators increasingly demand that mining companies post reclamation bonds for existing or proposed new mining developments. Impounded tailings so often represent the most significant closure risk, and so it follows that these issues are best dealt with when a project is in its conceptual stages.

Design Criteria for Closure

The design criteria (flood, earthquake, static stability) to which tailings dams must be designed depend on the time of exposure to the hazard. Because the exposure period of closed impoundments is perpetuity, the tailings dam must in most cases be designed to endure the most extreme events, such as the Maximum Credible Earthquake (MCE) and the Probable Maximum Flood (PMF). Modern tailings dam design must account for these requirements at the outset, so that expensive retrofit measures are not necessary at the end of the mine life. Design must also account for the cumulative effects of major floods (e.g. damage to riprap and/or spillways) and multiple earthquakes in high seismic zones.

Other design aspects that must be considered for closure are, for example, the erosion resistance of the dam, and the durabilility (weathering resistance) of the materials of which the dam is constructed.

"Dry" Closure

Where closure does not involve submergence of tailings below a water cover, modern practice requires either direct revegetation of the tailings surface, and/or covering of the tailings surface with a material that is more erosion resistant than tailings. Mining companies now typically carry out extensive reclamation studies (appropriate species, topsoil availability and requirements, etc.) during the feasibility study phase of projects.

The "dry" closure scenario can be used for closure of tailings with potential for generation of ARD, in concert with collection and treatment of seepage and (if necessary) surface runoff from the impoundment. Eventually, the sulphides would be depleted, and/or the contaminant generation rate would become sufficiently low as to eliminate the need for continuing collection and treatment. The "short term" disadvantage of having to collect and treat must be weighed against the longer term advantage of improved dam safety and reduced failure consequences associated with now "inert" tailings.

Submerged Closure of ARD Tailings Impoundments

Permanent submergence of tailings behind a water retaining dam to prevent ARD resolves the key geochemical issue, but places more stringent requirements on the safety of the tailings dam. The safety of a dam will not increase over time because the "drained" closure condition is not achieved. The closure spillway(s) now represents a particularly critical component of the closed impoundment, and, like the dam, must be inspected and maintained on a regular basis. The dam safety program implemented through the closure phase should be no less comprehensive than one that would be implemented for a conventional water retaining dam with similar failure consequences. In fact, the failure consequence of a tailings dam impounding sulphide tailings would be higher than that for a conventional water retaining dam. Failure of the tailings dam would, besides uncontrolled release of water, also represent an environmental failure in the release of sulphide tailings to the environment. Submerged closure of ARD tailings impoundments, therefore, is not a "walk-away" closure scenario.

Other ARD Control Alternatives

Other ARD control alternatives include:

- Permanent neutralization of the ARD source by adding/enhancing neutralization potential within the tailings mass, such as could be achieved by mixing in finely crushed limestone with the tailings; or
- Sulphide removal and separate storage towards the latter stages of the mine life, such that the portion of the tailings containing sulphides remains saturated and/or under very low oxygen flux.

As both understanding of the ARD problem and the technologies available for mitigation of the problem continue to advance (e.g. improving water treatment technologies), it is likely that closure options less adverse to dam safety than permanent submergence will continue to be developed and become increasingly cost effective.

CONCLUSIONS

A review of the state-of-practice in tailings dam/impoundment design and construction shows that great technical progress has been made. Better investigation and design tools are available. New technologies in thickening and filtration of tailings have provided the opportunities for alternatives to disposing of tailings as conventional slurries. New concepts include "dry-stack" tailings, thickened tailings and paste tailings. Geomembrane liners are commonly used where tailings may present a risk of groundwater contamination, and design and construction methods for lined impoundments have been developed. Improvements have been made to the traditional upstream construction method to reduce stability risks.

Environmental considerations have become increasingly more important in tailings dam design and permitting. Closure planning has become an integral part of initial design and permitting. Designs must address ARD and include measures for long term control and/or prevention of ARD.

In spite of the improvements in tailings disposal practices, a number of highly publicised failures have overshadowed the advances. These failures have led to increased scrutiny of mining projects by regulators, environmental groups, and financial institutions. Numerous guidelines and risk assessment programs have been developed by the industry to reduce tailings dam failure risks.

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An alternative to conventional tailing management – "dry stack" filtered tailings

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ABSTRACT: Development of large capacity vacuum and pressure belt filter technology presents the opportunity for storing tailings in a dewatered state, rather than as conventional slurry and/or in the "paste like" consistency associated with thickened tailings. Filtered tailings are dewatered to moisture contents that are no longer pumpable and need to be transported by conveyor or truck. Filtered tailings are placed, spread and compacted to form an unsaturated, dense and stable tailings stack (termed a "dry stack") requiring no dam for water or slurried tailings retention. This paper presents the basics of dry stack tailings management including design criteria and site selection considerations. Examples of several operations using dry stack technology are presented. Approximate operating costs for dry stack facilities are also included. Dry stack tailings are not a panacea for tailings management but present, under certain circumstances, an option to the tailings planner.

1 INTRODUCTION

Tailings management for the past several decades has largely involved the design, construction and stewardship of tailings impoundments. These impoundments are developed to store tailings slurry that typically arrives at the impoundment with solids contents of about 25% to 40%. The management of the traditional tailings impoundment is therefore a combination of maintenance of structural integrity and managing immense quantities of water.

The basic segregating slurry that has been used for conventional tailings management is only part of a continuum of products available to the modern tailings designer. Development of large capacity vacuum and pressure filter technology has presented the opportunity for storing tailings in a dewatered state, rather than as conventional slurry and/or in the "paste like" consistency associated with thickened tailings. Tailings are dewatered to moisture contents that are no longer pumpable. The filtered tailings are transported by conveyor or truck, and placed, spread and compacted to form an unsaturated, dense and stable tailings stack (a "dry stack") requiring no dam for retention. While the technology is currently (considerably) more expensive per tonne of tailings stored than conventional slurry systems, it has particular advantages in:

a) arid regions, where water conservation is an important issue
b) situations where economic recovery is enhanced by tailings filtration
c) where very high seismicity contraindicates some forms of conventional tailings impoundments
d) cold regions, where water handling is very difficult in winter

Moreover, "dry-stacks" have regulatory attraction, require a smaller footprint for tailings storage (much lower bulking factor), are easier to reclaim, and have much lower long-term liability in terms of structural integrity and potential environmental impact.

This paper will utilize the most common terminology in the industry. This includes:

- slurry tailings – the typically segregating mass of tailings that are in a fluidized state for transport by conventional distribution systems

• thickened tailings –partially dewatered but still a slurry that has a higher solids content by weight than the basic tailings slurry but is still pumpable. Chemical additives are often used to enhance slurry tailings thickening

• paste tailings – thickened tailings with some form of chemical additive (typically a hydrating agent such as Portland cement)

• wet cake tailings – a non-pumpable tailings material that is at, or near, saturation

• dry cake tailings – an unsaturated (e.g. not truly dry) tailings product that cannot be pumped.

The terms "dry stacked" or "dry stack" tailings have been adopted by many regulators and designers for filtered tailings. As long as the designers, owners and regulators understand that the tailings are not truly dry but have a moisture content several percent below saturation, there is nothing wrong with continuing the use of this terminology.

2 CONTINUUM OF TAILINGS

Figure 1 shows the continuum of water contents available for tailings management today and includes the standard industry nomenclature.

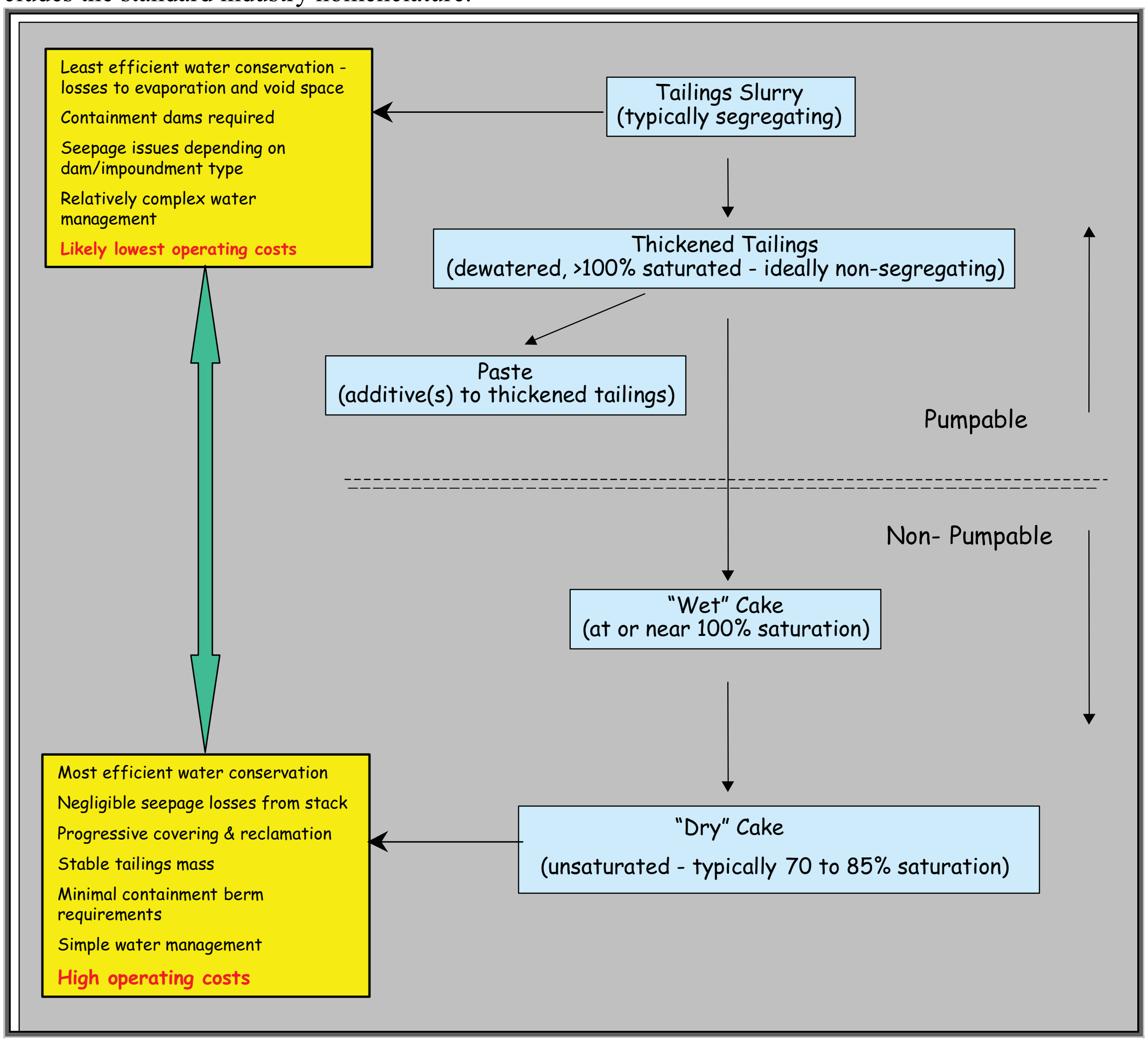

Figure 1. Tailings Continuum

Filtered tailings are typically taken to be the dry cake material shown in Figure 1. This material has enough moisture to allow the majority of pore spaces to be water filled but not so much as to preclude optimal compaction of the material.

Filtering can take place using pressure or vacuum force. Drums, horizontally or vertically stacked plates and horizontal belts are the most common filtration plant configurations. Figure 2 shows a typical filter press. Pressure filtration can be carried out on a much wider spectrum of materials though vacuum belt filtration is probably the most logical for larger scale operations.

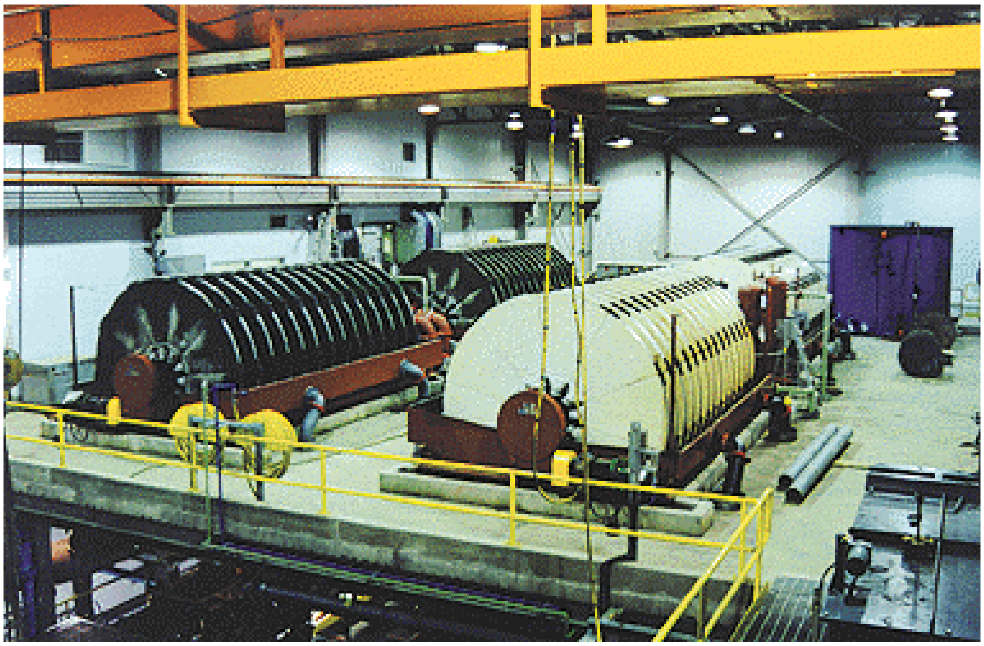

Figure 2. Example of a Filter Plant

The nature of the tailings material is important when considering filtration. Not only is the gradation of the tailings important, but the mineralogy is as well. In particular, high percentages of <74 µm clay minerals (i.e.. not just clay-sized but also with clay mineralogy) tend to contra-indicate effective filtration. Furthermore, substances such as residual bitumen (e.g. oil sands tailings) can create special difficulties for a filtration plant.

3 CONSIDERATIONS FOR FILTERED TAILINGS

As for any other form of tailings management, there are a number of issues that require careful consideration prior to selecting filtered tailings for a given project. Ultimately, these considerations are all about economics (capital, operating and closure liability) but require individual attention during the prefeasibility and feasibility stages of the project.

3.1 *General*

Whether filtered tailings are a candidate for a given project depends on the motivation to consider alternatives to a conventional slurried tailings impoundment. The motivation could include a more favourable, or timely, regulatory process or perhaps one of several technical issues presented by the site.

As noted in the introduction, filtered tailings could have application to meet technically challenging sites in:

a) arid regions

b) mines where dissolved metal recovery is enhanced by tailings filtration
c) high seismicity regions
d) cold regions
e) mine sites where space is limited as filtered tailings result in a lesser footprint than for slurried tailings.

In addition, site legacy issues are also part of the selection criteria as dry stacked tailings facilities are substantially easier to reclaim for mine closure, in most circumstances, when compared to conventional impoundments. Following is some elaboration on key issues to be considered.

3.2 *Water Management*

Where water is relatively scarce, either year round or seasonally due to extreme cold, sending immense quantities of water to quasi-permanent storage in the voids of a conventional impoundment can severely hamper project feasibility. By reclaiming the bulk of the process water in or near the mill, far more efficient recycle is achieved. Moreover, the amount of water "stored" in a dry stack facility will be typically >25% less than that in a conventional slurried impoundment even if 100% pond reclaim efficiency is achieved with the impoundment.

3.3 *Commodity Extraction*

Many mines, particularly those dealing with precious metals, can improve the bottom line by maximizing the amount of tailings water that can be reclaimed. Both the economic commodity (e.g. dissolved gold) and process chemicals (e.g. cyanide) can be recovered from the filtration water and one or more rinse cycles.

3.4 *Storage Availability*

Filtered tailings can be placed in a relatively dense state meaning that more solids per unit volume can be achieved. Furthermore, more aggressive use of available land (e.g. valley slopes) can be used with filtered tailings. As discussed in 3.5, lesser foundation conditions can also be considered in comparison to conventional impoundments.

3.5 *Geotechnical Issues*

The questionable manner in how some conventional impoundments are designed and/or operated provides support to considering the geotechnical advantages of filtered tailings. By objectively reviewing an instability database for conventional slurry tailings impoundments, over the past 30 years there have been approximately 2 to 5 "major" tailings dam failure incidents per year (Davies and Martin, 2000). There have been at least two events each year (1970-1999, inclusive). If one assumes a worldwide inventory of 3500 conventional tailings impoundments (a tenuous extrapolation at best), then 2 to 5 failures per year equates to an annual probability of between 1 in 700 to 1 in 1750. This rate of failure does not offer a favorable comparison with the 1 in 10,000 figure that appears representative for conventional water dams. The comparison is even more unfavorable if less "spectacular" tailings impoundment failures are considered. These impoundment failures, often equally economically damaging, are not just of older facilities constructed without formal designs, but include facilities designed and commissioned in the past 5 to 20 years - supposedly the "modern age" of tailings dam engineering.

The most common failure modes for slurry tailings impoundments are physical instability (including static and dynamic liquefaction) and water mismanagement issues (including lack of freeboard and seepage phenomena like piping). Filtered tailings placed in dry stacks are essentially immune to catastrophic geotechnical "failure" and can be readily designed to withstand static and seismic forces. A case can also be made for a reduction in the seismic design criteria based on failure consequence. This can significantly reduce operating costs. The unsaturated tailings mass is extremely resistant to saturation and seepage is governed by unsaturated hydraulic conductivities. Moreover, far less is required of foundation conditions as the unsaturated, largely dilatant tailings within a dry stack are not susceptible to static liquefaction or catastrophic breaching by an impounded pond should the foundation move creating substantive shear strains in the tailings mass.

3.6 *Reclamation/Closure Issues*

Dry stack facilities can be developed to consist of, or closely approximate, their desired closure configuration. Some form of assured surface runoff management plan is required. The tailings can be progressively reclaimed in many instances. In all cases, a closure cover material is required to resist runoff erosion, prevent dusting and to create an appropriate growth media for project reclamation.

The lack of a tailings pond, very low (if any) appreciable seepage from the unsaturated tailings mass and general high degree of structural integrity allows dry stacks to present the owner/operator with a comparably straight forward and predictable facility closure in comparison with most conventional impoundments.

3.7 *Environmental Stewardship*

Issues related to the environmental impacts from tailings dams were first seriously introduced in the 1970's in relation to uranium tailings. However, environmental issues related to mining have received attention for centuries. For example, public concerns about the effects of acid rock drainage (ARD) has existed for roughly 1,000 years in Norway. Today, environmental issues are growing in importance as attention has largely turned from mine economics and physical stability of tailings dam to their potential chemical effects and contaminant transport mechanisms. Recent physical failures such as Merriespruit, South Africa in 1994 and Omai, Guyana in 1995 and Los Frailes in Spain in 1998 illustrates this issue with most of the media reports highlighting the real or perceived environmental impacts of the failures.

Dry stacked tailings facilities have some tremendous potential environmental advantages over impounded slurried tailings largely because the catastrophic physical failures that define tailings management to non-supporters of the industry cannot occur. Moreover, leachate development is extremely limited due to the very low seepage rates possible. Oxidation processes are possible though the very slow rates for such, coupled with the limited seepage potential, limits or eliminates the concern of significant metallic drainage. Clearly, industry/regulatory standards of testing for potential operating and long-term impacts are essential. However, if the stack is operated to maintain its unsaturated character, any potential impacts should be predicted as acceptable except under unusual conditions.

Fugitive dusting, both during operation and upon closure, is a very real concern with dry stacks; particularly in arid environments. Progressive reclamation is the only effective method to address this concern.

3.8 *Regulatory Environment*

The regulatory environment worldwide is generally becoming less tolerant of one of humanities essential industries. Mining cannot exist without the creation of some form of tailings so the availability of a management strategy that is viewed (and correctly so) as both less invasive and less difficult to decommission as well as one that does not conjure up "massive" failure scenarios is a positive to the industry. As discussed elsewhere in this paper, the challenge is to get this regulatory friendly tailings management system to become cost-effective for those operations that would benefit (eg. in terms of the permitting process) from its consideration.

4 DESIGN CRITERIA

4.1 *General*

There are four main design criteria for filtered tailings:

1. filtering characteristics
2. geomechanical characteristics
3. tailings management
4. water management

In addition, the design must be compatible with an optimal closure condition (designing for closure). Implicit to the overall design criteria is project economics.

4.2 *Filtering Characteristics*

Determine the most cost-effective manner to obtain a dewatered product consistent with the other three design criteria (geomechanics, placement management and water management). Filter suppliers are both knowledgeable and helpful in this regard but some form of pilot test(s) is essential as every tailings product will exhibit its own unique filtering character. It is important to anticipate mineralogical and grind changes that could occur over the life of the project. The candidate filtering system(s) must be able to readily expand/contract with future changes at the mine with the least economical impact.

4.3 *Geomechanical Characteristics*

The strength, moisture retention and hydraulic conductivity characteristics of the tailings need to be established. The saturated tailings should be determined to "anchor" the results and tests as variable moisture contents are required to demonstrate the impact of the inevitable range of operating products. The other important geomechanical characteristic to determine is the moisture-density nature of the tailings. The unsaturated moisture-density relationship indicates insitu density expectation as well as the sensitivity of the available degree of compaction for a given moisture content. From a compaction perspective, the filtered tailings should neither be too moist nor too dry. The optimal degree of saturation is usually between 60 and 80%.

4.4 *Tailings Management*

The design needs to be compatible with how the stack can be practically constructed using conventional haulage and placement equipment. Other than the capital and operating costs of the filtering process, the economics of dry stack management is the most important component of filtered tailings viability. Haul distance, placement strategy and compactive effort and additional works for closure and reclamation can make a larger incremental difference to the unit cost of a dry stack facility in comparison with a slurried impoundment.

The design should also clearly identify what contingency(s) will be in place if the filtering process experiences short-term disruptions. A temporary storage area or vessel is sound strategy. It is, however, the authors' experience that the filters should become part of the process plant under the management of the mill superintendent. The tailings processing then becomes integrated with the metal recovery functions and consequently down time is minimized because operation of the tailings system becomes critical to the overall mill performance.

4.5 *Water Management*

Surface water, particularly concentrated runoff, should not be permitted to be routed towards a dry stack. As important, the catchment and routing of precipitation (and any snow melt in colder climates) on the stack itself must be appropriately designed for. For the surface runoff within the overall catchment containing the dry stack, one (or more) of perimeter ditches, binds or under-stack flow through drains designed for an appropriate hydrological event(s) should be included in the design. For on stack water management, routing of flows to armoured channels and limiting slope lengths/gradients to keep erosion potential at a minimum are the best design criteria.

5 CONSTRUCTION AND OPERATIONAL CONSIDERATIONS

5.1 *General*

There are a number of construction and operational considerations that need to be accounted for in the design and planning of a dry stack. These considerations are very different from the construction and operational considerations normally associated with slurry tailings facilities. The main considerations are usually:

1. Site development
2. tailings transport and placement
3. water conservation and supply
4. reclamation and closure

In addition, there are often other considerations that need to be addressed on a site-specific basis for example co-disposal of waste rock in a combined mine waste management facility, storage of water treatment plant sludges etc.

5.2 *Site Development*

Site development for a dry stack normally consists of the construction of surface and groundwater control systems. There are normally two systems:

1. A collection and diversion system for non-contact water (i.e. natural surface water and groundwater from the surrounding catchment area that has not yet come into contact with the tailings). This system usually consists of ditches to divert surface runoff around the site and if necessary a groundwater cut-off and drainage system usually combined with surface water diversion. The cut-off system can range from simple ditches to sophisticated cut-off walls depending upon site conditions.
2. An interception and collection system for contact surface water, impacted groundwater, and seepage from the dry stack. This system usually consists of an underdrainage system of finger drains, toe drains, drainage blankets and french drains; collection sumps and ponds. Water collected in the ponds and sumps is usually used in process or pumped to a water treatment plant depending upon the site water balance. Liners for the facilities can also be components of the interception and collection system depending upon predicted impacts and regulatory requirements.

5.3 *Tailings Transport and Placement*

There are two methods in common use for transport of the filtered tailings to the tailings storage facility. These are conveyors or trucks and the equipment selection is a function of cost. Placement in the facility can be by a conveyor radial stacker system or trucks depending upon the application and the design criteria. Conveyor transport of tailings to the disposal site can be combined with placement by truck, so conveyor transport does not automatically result in placement by radial stacker.

The main issue associated with the placement of the filtered tailings by truck is usually trafficability. The filtered tailings are generally produced at or slightly above the optimum moisture content for compaction as determined in laboratory compaction tests (Proctor Tests). This means that a construction/operating plan is required to avoid trafficability problems. This is especially true in wetter environments since trafficability drops as moisture content rises and if the tailings surface is not managed effectively it can quickly become un-trafficable resulting in significant placement problems and increased operating costs. In addition, in high seismic areas there is often a design requirement to compact the tailings to a higher density in at least the perimeter “structural” component of the facility. This requirement increases the need for construction quality control. It is the authors’ experience that the degree of compaction required for assured and efficient trafficability is often higher than the compaction required to achieve design densities.

Dry stack designs often incorporate placement zones for “summer/good weather” placement (dry, non-freezing conditions) and “winter/bad weather” placement (wet, or freezing conditions) with summer placement being focused on the structural zones. Again, this is especially true for facilities planned for wetter or colder climates were seasonal fluctuations are significant and predictable.

The key is to consider the environment and the design criteria and develop a flexible operating plan to achieve them.

5.4 *Water Conservation*

Often one of the main reasons to select dry stacked filtered tailings as a management option is the recovery of water for process water supply. This is particularly important in arid environments were water is an extremely valuable resource and the water supply is regulated (e.g. Northern Chile and Mexico). Filtering the tailings removes the most water from the tailings for recycle when compared with other tailings technologies as discussed earlier. This recovery of water has a cost benefit to the project, which offsets the capital and operating cost of the tailings system. It should be noted, that water surcharge storage needs to be factored in to the design of a filtered tailings system. Depending upon the application this can be a small water supply reservoir or tank.

5.5 *Reclamation and Closure*

One of the main advantages of dry stack tailings is the ease of progressive reclamation and closure of the facility. The facility can often be developed to start reclamation very early in the project life cycle. This can have many advantages in the control of fugitive dust, in the use of reclamation materials as they become available, and in the short and long term environmental impacts of the project. Progressive reclamation often includes the construction of at least temporary covers and re-vegetation of the tailings slopes and surface as part of the annual operating cycle.

6 ECONOMICS

6.1 *General*

It is hard to compare the economics of dry stack filtered tailings with other tailings options particularly conventional slurry tailings. This is mainly because of the difficulty of estimating the cost of closure and the potential costs associated with the long-term risk environmental liability associated with mine waste facilities. Therefore, the following discussion on economics is very subjective with a focus on perception.

6.2 *Capital Cost*

The capital costs are clearly a function of the size of the operation. Dry stack, filtered tailings currently appears to be limited to operations of 15000 tpd or less depending upon financial credits e.g. water recovery for use in process. Capital costs normally shift from the construction of engineered tailings containment structures to the dewatering (filter) plant. The capital costs may be further mitigated if the application is considered for small tonnage (less than say 4000 tpd) where the mine plan calls for paste backfill underground. Paste backfill requires a tailings processing plant with dewatering so incremental dewatering to produce filtered tailings make the economics more attractive. The capital cost appears to be much more attractive for operations under approximately 2000 tpd.

Other costs that should be factored into the equation are reduced costs associated with the smaller footprint, site development costs, and regulatory acceptance associated with dry stack tailings. These costs are often difficult to estimate accurately.

6.3 *Operating Cost*

The operating costs associated with the transport and placement of dry stack, filtered tailings are higher when compared with conventional slurry tailings, transported hydraulically and deposited in a tailings pond. The operating costs for a dry stack are difficult to summarize as every operation accounts for the costs differently. For example, if a mine uses a surface crew who do both tailings stack development as well as other duties, the cost/tonne will be much lower than a dedicated dry stack work force. Under the range of conditions for the presently operating dry stacks, the cost per tonne ranges from \$1 to \$10 but the average is more like \$1.50 to \$3. All costs are \$US and include filtering, transport, placement and compaction in the facility.

6.4 *Reclamation and Closure*

Reclamation and closure costs are significantly reduced for dry stack tailings when compared with conventional tailings. This cost reduction is due to a reduced footprint and constructability. Other issues that need to be somehow factored into the "cost" of closure are the reduction in long-term risk and liability associated with dry stacks.

7 EXAMPLES

There are a growing number of dry stack facilities. At the same time, it would be fair to say that there is likely not any one of those operations who can point to an overall operating economic advantage to the practice. However, for at least three of the operating dry stack projects, the increased operating cost was sufficiently negated by other factors including regulatory issues and closure/liability costs.

The majority of the dry stacks are either in colder climates (e.g. Greens Creek, Alaska, Raglan, Quebec) or in arid environments (e.g. La Coipa, Chile). The La Coipa facility, developing at more than 15,000 tons/day, is one of the largest operating dry stacks. The La Coipa facility is located in a high seismic region with designed, and confirmed, structural integrity. Figure 3 shows the La Coipa facility a few years ago.

Figure 3. Dry-Stack Tailings Facility - Chile

8 SUMMARY AND CONCLUSIONS

There are several candidate scenarios where dewatered tailings systems would be of advantage to the mining operation. However, dewatered tailings systems may have less application for larger operations for which tailings ponds must serve dual roles as water storage reservoirs, particularly where water balances must be managed to store annual snowmelt runoff to provide water for year round operation.

Filtered tailings, a form of dewatered tailings, are not a panacea for the mining industry for its management of tailings materials. Purely economic considerations rarely indicate a preference for dry stacked tailings facilities over conventional slurry impoundments. However, under a growing number of site and regulatory conditions, filtered tailings offer a real alternative for tailings management that is consistent with the expectations of the mining industry, its regulators and the public in general.

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Proceedings Tailings and Mine Waste 2011
Vancouver, BC, November 6 to 9, 2011

Filtered Dry Stacked Tailings – The Fundamentals

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Abstract

Filtered tailings are becoming an increasingly common consideration for tailings management at many mines. There are more filtered dry stack tailings storage facilities than there are surface paste facilities yet the amount of guidance documentation on filtered tailings is virtually non-existent in compare to those same paste tailings facilities. The reason for this lack of guidance materials is uncertain but it has led to some unfortunate tailings management decisions based on misinformation about dry stacked tailings facilities in general.

This paper provides practical guidelines for the design and development of filtered dry stack tailings facilities. These guidelines are based upon the successful conceptualization, design, and operating experience at a number of these facilities. Issues related to target moisture content, appropriate testing methods and criterion, geotechnical conditions and placement considerations are included. The guidelines include specific reference to "lessons learned" from existing operations that will benefit designers and owners alike.

Filtration – End Member of the Tailings Continuum

The vast majority of the world's tailings facilities involve tailings impoundments. These impoundments are developed to store tailings slurry that typically arrives at the impoundment at solids contents of about 25% to 60% depending upon whether any thickening is carried out prior to deposition. These impoundments require construction and maintenance of structural integrity for the retention structures as well as management for what are typically immense quantities of water. Following operating these complex entities, closure of these impoundments can represent significant challenges in terms of both physiochemical reclamation as well as geotechnical considerations.

As the future of mining includes increasing scrutiny on the industry's stewardship of the natural environment, including use of water in most regions in the world, a commitment to alternatives beyond impoundments is often sought. The amount of water that is "lost" to the voids in the stored tailings, seeps or evaporates from the tailings impoundments is something being increasingly viewed by critical regulatory and public eyes that insist on evaluating whether there are viable alternatives for any given proposed mining development. This pressure to seek alternative tailings management approaches exists today and the future will likely only see these pressures intensified.

Conventional tailings impoundments remain the best alternative for the majority of operating and proposed mines around the world. These facilities are developed using tailings slurries that are the end waste product of the milling process. However, with advances in dewatering technologies over the past few decades, that tailings slurry is actually being only part of a continuum of tailings "states" available to the modern tailings designer. Development of large capacity vacuum and pressure filter technology has presented the opportunity for storing tailings in an unsaturated state, rather than as conventional slurry and/or in the "paste like" consistency associated with thickened tailings. For the minority set of projects that can find a non-slurried tailings alternative advantageous to optimal permitting and/or operating conditions, filtered tailings are often an excellent alternative.

Figure 1 shows the continuum of water contents available for tailings management and includes the standard industry nomenclature. With decreasing water content comes increased expense at hauling the tailings (e.g. pumping costs increase and then, upon becoming a wet cake, the tailing are no longer pumpable and other transport methods are required). However, as the water content decreases, which means increased water recovery within the process, the tailings are far more readily able to be used in self-supporting structural situations such as stacks.

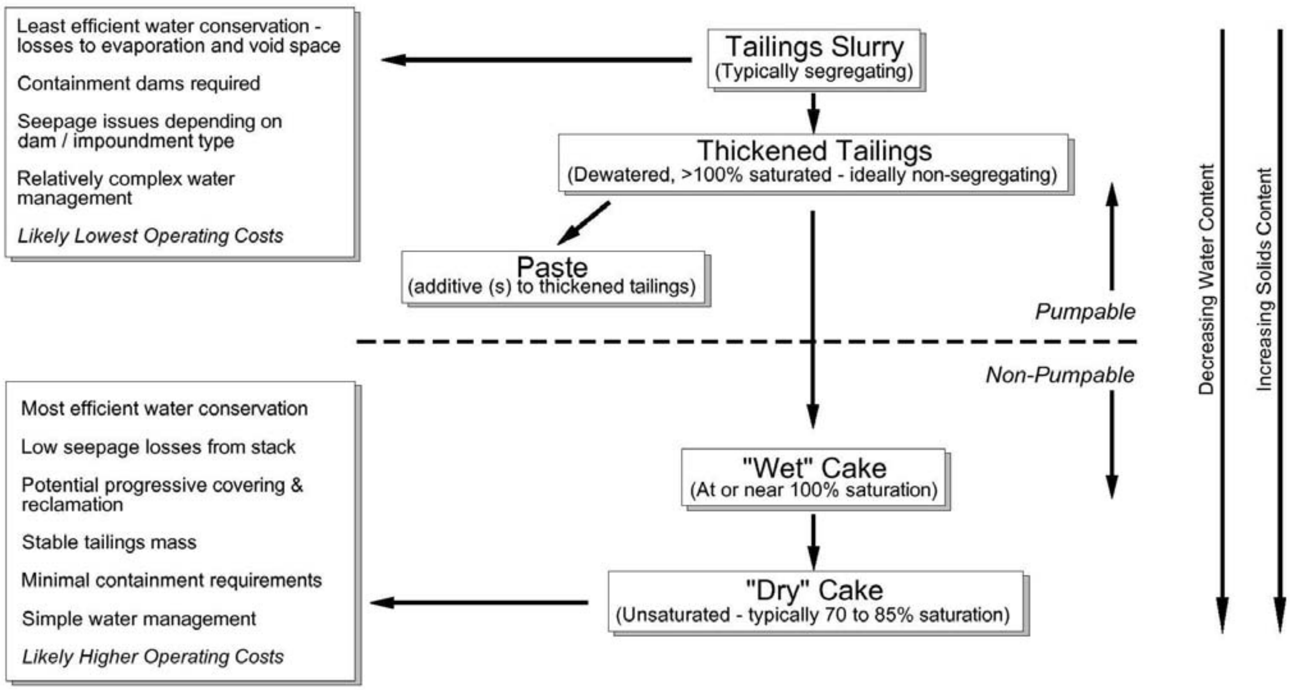

Figure 1: Tailings Continuum

Filtered tailings are typically taken to be the dry cake material shown in Figure 1. This material has enough moisture to allow the majority of pore spaces to be water filled but not so much as to preclude optimal compaction of the material.

Filtering and Dry Stacking

The Basics

Filtering of tailings can take place using pressure or vacuum force. Drums, horizontally or vertically stacked plates and horizontal belts are the most common filtration plant configurations. Pressure filtration can be carried out on a much wider spectrum of materials though vacuum belt filtration is probably the most logical for larger scale operations.

The nature of the tailings material is important when considering filtration. Not only is the gradation of the tailings important, but the mineralogy is as well. In particular, high percentages of <74 µm clay minerals (i.e. not just clay-sized but also with clay mineralogy) tend to contraindicate effective filtration. Furthermore, substances such as residual bitumen (e.g. oil sands tailings) can create special difficulties for a filtration plant.

Determining the most cost-effective manner to obtain a filtered product consistent with the geomechanical requirements of the tailings can be a challenge. Filter suppliers are both knowledgeable and helpful in this regard but some form of pilot test(s) is essential as every tailings product will exhibit its own unique filtering behaviour. It is important to anticipate mineralogical and grind changes that could occur over the life of the project. The candidate filtering system(s) must be able to readily expand/contract with future changes at the mine with the least economical impact.

Filtered tailings emerge from the process facility within a prescribed range of moisture contents discussed later. The tailings are then transported by conveyor or truck and then placed, spread and compacted to form an unsaturated, dense and stable tailings “stack” (often termed a "dry stack") requiring no dam for retention with no associated tailings pond. The filtered tailings are not “dry” but are unsaturated so the early nomenclature referring to them as dry is incorrect. However, it is doubtful this mislabeling has led to any misunderstandings amongst experienced designers, operators and regulators.

Each project needs to assess the potential applicability for filtered tailings based upon technical, economical and regulatory constraints. Experience shows the most applicable projects are those that have one or more of the following attributes:

1. Reside in arid regions, where water conservation is crucial (e.g. Western Australia, Southwest United States, much of Africa, many regions of South America, arctic regions of Canada and Russia)
2. Have flow sheets where economic recovery (commodity or process agent(s)) is enhanced by tailings filtration
3. Reside in areas where very high seismicity contraindicates some forms of conventional tailings impoundments
4. Reside in cold regions, where water handling is very difficult in winter
5. Have topographic considerations that exclude conventional dam construction and/or viable storage to dam material volume ratios
6. The operating and/or closure liability of a conventional tailings impoundment are in excess of the incremental increase to develop a dry stack.

To date, the two most common reasons to select dry stacked filtered tailings as a management option have been to recover water for process water supply and where terrain/foundation conditions contraindicate conventional impoundments. The recovery of water is particularly important in arid environments were water is an extremely valuable resource and the water supply is regulated (e.g. Chile, Western Australia, and Mexico). This recovery of water has a cost benefit to the project, which offsets the capital and operating cost of the tailings system. It should be noted that water surcharge storage needs to be factored in to the design of a filtered tailings system. Depending upon the application this can be a small water supply reservoir or tank. Where water is relatively scarce, either year round or seasonally due to extreme cold, sending immense quantities of water to quasi-permanent storage in the voids of a conventional impoundment can severely hamper project feasibility. By reclaiming the bulk of the process water in or near the mill, far more efficient recycle is achieved. Moreover, the amount of water “stored” in a dry stack facility will be typically >25 to 50% less than that in a conventional slurried impoundment even if 100% pond reclaim efficiency is achieved with the impoundment.

One of the main advantages of dry stack tailings over other tailings management options is the ease of progressive reclamation and closure of the facility. The facility can often be developed to start reclamation very early in the project life cycle. This can have many advantages in the control of fugitive dust, in the use of reclamation materials as they become available, and in the short and long term environmental impacts of the project. Progressive reclamation often includes the construction of at least temporary covers and re-vegetation of the tailings slopes and surface as part of the annual operating cycle.

How Common is Dry Stacking?

On a global basis, conventional tailings facilities (e.g. slurry tailings direct from mill into a tailings impoundment) make up by far the majority all existing tailings facilities. In terms of dewatered tailings, meaning those that are "lower" on Figure 1 than slurried tailings, there are a similar number of thickened/surface paste tailings facilities to filtered tailings facilities in terms of number of worldwide operations. There is, however, an intriguing dichotomy between available information about paste/thickened tailings and filtered tailings.

For paste/thickened tailings there has been a steady stream of publications (far outnumbering actual projects where the methods have been applied) and even annual specialty conferences. For example, each year since the late 1990s, there is an international conference on paste and thickened tailings where the presentations focus has necessarily been on potential advances and such more than actual case studies simply as there have not been sufficient projects to write about. Including the papers from these annual, and other, conferences, there are more than 200 publications on paste/thickened tailings including several guidebooks.

Filtered tailings, on the other hand, have simply not had the attention other dewatered tailings have had yet, as noted above, there are a similar number of actual operating mines using filtered tailings in comparison to, for example, thickened/paste tailings surface storage. There are but a handful of publications on filtered tailings/dry stacks and rare mention in conference proceedings. This is a curious development when the comparative number of actual projects using the various methods of tailings management is considered.

Figure 2, taken from a recent evaluation of global trends in dewatered tailings practice (Davies et al, 2010) provides a summary of the relative number of dewatered facilities on a global scale.

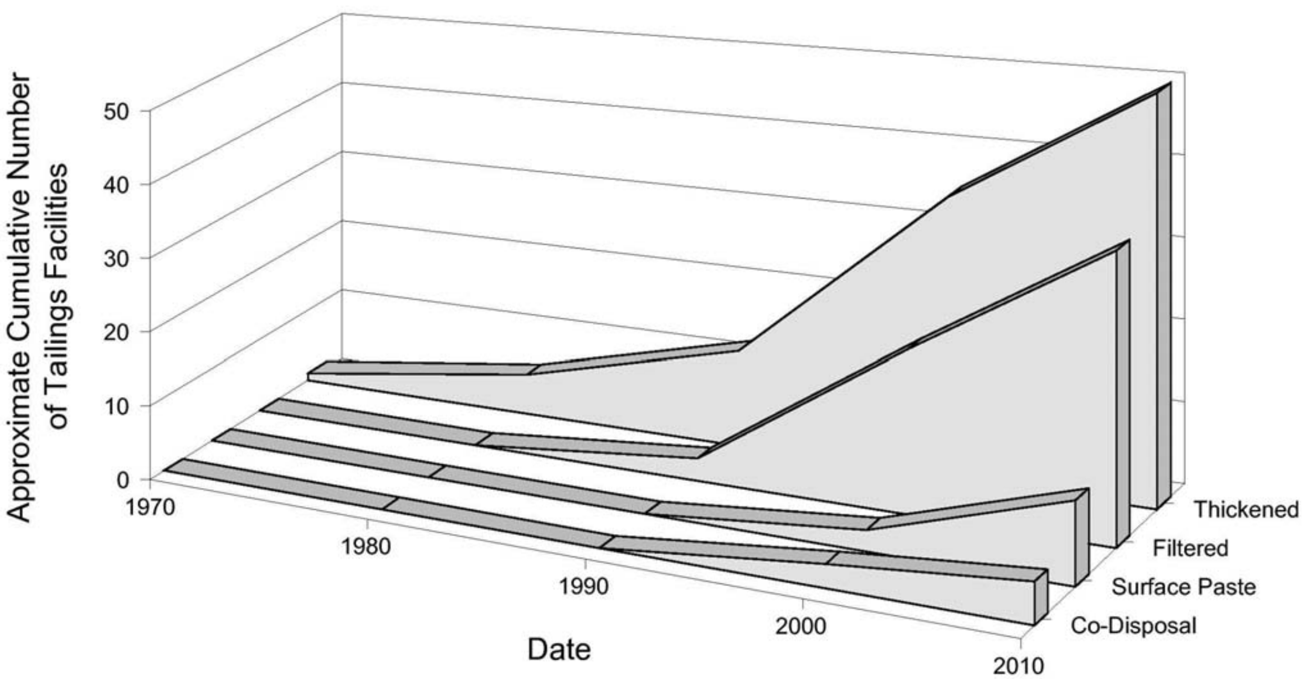

Figure 2: Trends in Use of Dewatered Tailings in Mining (after Davies et al, 2010)

Filtered Tailings - Design Guidelines

Overview

The strength, moisture retention and hydraulic conductivity characteristics of the tailings need to be established for any given project considering the technology. The strength and hydraulic parameters from saturated tailings should be determined to "anchor" the results and tests as variable moisture contents are required to demonstrate the impact of the inevitable range of operating products. The other important geomechanical characteristic to determine is the moisture-density nature of the tailings. The unsaturated moisture-density relationship indicates in-situ density expectation as well as the sensitivity of the available degree of compaction for any given moisture content. From a compaction perspective, the filtered tailings should neither be too moist nor too dry. The optimal degree of saturation is usually between 60 and 80%.

Filtered tailings can be placed in a relatively dense state meaning that more solids per unit volume can be achieved. Furthermore, more aggressive use of available land (e.g. valley slopes) can be used with filtered tailings. Lesser foundation conditions can also be considered in comparison to conventional impoundments.

Siting Considerations

While a filtered tailings dry stack will still require a foundation consistent with acceptable deformation criteria provided the loading conditions that the stack would be projected to be subjected to, static and dynamic, the range of topographic settings and foundation conditions where dry stacking will work is substantially wider than for conventional tailings impoundments. Avoidance of concentrated runoff water flows directed at the stack is one essential siting consideration. Other key siting considerations include:

- Placing the stack to avoid fugitive dusting from prevailing winds
- Avoiding placing where "blinding" off groundwater discharge areas (unless a sufficiently robust underdrainage system is designed, constructed and maintained)
- Optimizing the haulage and/or conveyance from the filtration plant; the tailings are no longer a slurry and a common "error" with those not familiar is dry stacks is to site the facility in same way one would a conventional slurry impoundment
- Potential ability to co-dispose with and/or abut waste rock dumps.

Tailings Testwork

The testwork required to provide sufficiently detailed engineering decisions at all project stages is relatively modest with filtered tailings. Minimum testing requirements are provided based upon project stage as follows:

Conceptual – Prefeasibility Project Stage(s)

- Approximate tailings gradation and mineralogy
- Flask or similar filtrate testing
- Standard Proctor (moisture-density)
- Vendor engagement – filtration and transportation

Feasibility Stage

- Tempe Cell laboratory testing
- Geochemical testwork
- Bench scale filtration testing
- Extended moisture density work
- Transport behavior evaluation

Detailed Engineering Stage

- Variable moisture testwork
- Possible field compaction trial

More detailed strength testing (e.g. triaxial) is an option and is only typically required for the largest of stacks as the range of strength parameters for the majority of tailings is within the margin of accuracy of the stability estimation programs used by designers. Strength testing that includes an ability to obtain key deformation moduli for the tailing is important, at the feasibility level, where deformation of the facility will govern performance (due, for example, to a weaker foundation scenario). Again, such considerations are only typically of relevance for the larger dry stacks being considered.

Target Moistures

Likely one of the most misunderstood design parameters for any filtered dry stack is the target moisture content for the filtrate. The degree of dewatering readily achievable depends upon the filtering technology adopted, the application rate of tailings into that technology and the tailings physical characteristics. However, what should be the more driving discriminator is what is required to develop the stack itself in a manner that expedites construction, maintains structural integrity post-compaction and provides all of the water management advantages that an appropriately developed dry stack exhibits.

From experience of developing more than ten dry stacks and testwork on many more, a very useful rule of thumb is to have the target moisture content be equivalent to the tailings Standard Proctor optimum moisture content as described by ASTM D-698 (ASTM 2011). While this target can vary as much as 1 or 2% under (wetter climates) to 1% over (extremely dry climates), the target has worked extremely well on all facilities presently existing that include those up to, and including, throughputs to 20,000 tpd. As filtered dry stacks increase in size, and appropriately the size of compaction equipment, it is probably that target moistures more consistent with the Modified Proctor may become more appropriate.

Facility Zonation

One of the most consistent “challenges” that operators of filtered dry stacks have is that no ore body is entirely consistent let alone the mechanical and human variability elements involved in transporting and placing/compacting those tailings. As a result, the filtrate’s character will vary and occasionally not meet the target moisture contents. Moreover, there can be extreme cold seasons in a year and/or infrequent but intense rainfall/snow events throughout a year that can all impact abilities to achieve consistent compaction of the filtered tailings.

The best solution for addressing filtrate and climatic variation is to design and operate the dry stack with “zones”. The facility can have, for example, a “shell” that is reserved for only filtrate that meets all specifications and is placed in optimal conditions during a day/week/year. The shell can then surround an interior of tailings that are provided the same/similar compactive effort but there is, and appropriately so, less expectation of these materials in global stability and otherwise evaluations.

Zonation can also exist for placement of waste rock within the dry stack. There are not fewer than five operating dry stacks that are provide encapsulation of mineralized waste rock that is provided the excellent oxygen barrier than a considerable thickness of unsaturated compacted tailings provides.

Water Management

Surface water, particularly concentrated runoff, should not be permitted to be routed towards a dry stack. As important, the catchment and routing of precipitation (and any snow melt in colder climates) on the stack itself must be appropriately designed for. For the surface runoff within the overall catchment containing the dry stack, one (or more) of perimeter ditches, binds or under-stack flow through drains designed for an appropriate hydrological event(s) should be included in the design. For on stack water management, routing of flows to armored channels and limiting slope lengths/gradients to keep erosion potential at a minimum are the best design criteria.

Site development for a dry stack normally consists of the construction of surface and groundwater control systems. There are normally two systems:

1. A collection and diversion system for non-contact water (i.e. natural surface water and groundwater from the surrounding catchment area that has not yet come into contact with the tailings). This system usually consists of ditches to divert surface runoff around the site and if necessary a groundwater cut-off and drainage system usually combined with surface water diversion. The cut-off system can range from simple ditches to sophisticated cut-off walls depending upon site conditions.
2. An interception and collection system for contact surface water, impacted groundwater, and seepage from the dry stack. This system usually consists of an under-drainage system of finger drains, toe drains, drainage blankets and French drains; collection sumps and ponds. Water collected in the ponds and sumps is usually used in process or pumped to a water treatment plant depending upon the site water balance. Liners for the facilities can also be components of the interception and collection system depending upon predicted impacts and regulatory requirements.

Finally, the subject of facility lining is a prevalent topic and bound to arise on most every project where tailings are involved whether dry stacked or not. There is no hard set rule for lining versus no lining as, for the most part, lining with an appropriately designed and operated dry stack is more for political purposes than technical ones. Well-compacted filtered tailings at/near “optimum” moisture will have an equivalent hydraulic conductivity in a similar range to a typical liner element with average installation and other defects. The moisture content specified for optimal compaction is often very similar to the residual moisture content for the material and “drain down” is both slow and very limited in actual quantity of flow in most cases.

Tailings Transport/Placement

The design of any tailings dry stack needs to be compatible with how the stack can be practically constructed using the selected haulage and placement equipment. Haul distance, placement strategy and compactive effort and additional works for closure and reclamation make a larger incremental difference to the unit cost of a dry stack facility.

There are two methods in common use for transport of the filtered tailings to the tailings storage facility. These are conveyors or trucks and the equipment selection is a function of cost. Placement in the facility can be by a conveyor radial stacker system or trucks depending upon the application and the design criteria. Conveyor transport of tailings to the disposal site can be combined with placement by truck, so conveyor transport does not automatically result in placement by radial stacker.

The main issue associated with the placement of the filtered tailings by truck is usually trafficability. The filtered tailings are generally produced at or slightly above the optimum moisture content for compaction. This means that a construction/operating plan is required to avoid trafficability problems. This is especially true in wetter environments since trafficability drops as moisture content rises and if the tailings surface is not managed effectively it can quickly become un-trafficable resulting in significant placement problems and increased operating costs. In addition, in high seismic areas there is often a design requirement to compact the tailings to a higher density in at least the perimeter "structural" component of the facility. This requirement increases the need for construction quality control. It is the authors' experience that the degree of compaction required for assured and efficient trafficability is often higher than the compaction required to achieve design densities to meet geotechnical considerations.

Reclamation/Closure

Dry stack facilities can be developed to consist of, or closely approximate, their desired closure configuration. There is negligible facility deformation post-placement versus the considerable consolidation settlement conventional tailings undergo over what can be a very long period. Commensurately, the tailings can be progressively reclaimed in many instances.

The most important closure element is an assured surface runoff management plan with redundancy. In all cases, a closure cover material is required to resist runoff erosion, prevent dusting and to create an appropriate growth media for project reclamation.

The lack of a tailings pond, very low (if any) appreciable seepage from the unsaturated tailings mass and general high degree of structural integrity allows dry stacks to present the owner/operator with a comparably straight forward and predictable facility closure in comparison with most conventional impoundments.

Key Lessons Learned from Operating Dry Stacks

From design, operating and review knowledge of a majority of the world's dry stack tailings facilities, there are a number of "lessons learned" that should assist in any new facility being considered and/or in optimizing an existing facility. There are presented in no particular order of importance:

- Zonation is essential to a pragmatic and efficient tailings dry stack. Having an ability to deal with slightly off-specification material and/or still place in any weather condition removes many of the constraints that some have placed on dry stack development. It would be an extremely rare/unique situation that would not benefit and/or allow for a zoned approach to managing a given dry stack. Davies and Veillette (2007) describe the zonation approach adopted for the Pogo Mine in Alaska.
- If there is proper compaction and maintenance of target moisture contents, seepage is negligible. Instead of creating a complex system to capture seepage that will likely never appear, spend those resources more appropriately on surface water management measures that include a collection pond downgradient of the dry stack.
- Resaturation of properly placed and compacted filtered tailings is extremely difficult and not the concern many presume.

- Diversion ditches should be appropriately lined and the water routed in such a way that erosion of the tailings surface is not permitted to occur.
- Compaction specifications can be achieved in sub-freezing conditions if tailings windrows are compacted within a few hours of being transported from the plant.
- Heated bed liners are essential in colder climates.
- Tarps are excellent, though not elegant, way to provide short-term erosion protection in areas of intense rainfall where tailings windrows cannot be compacted prior to such rainfall events occurring (e.g. where they are daily events).
- Carrying on from the point above, dry stacks can be effectively developed in very wet conditions.
- Fugitive dust generation can be considerable in colder months (in cold climates) due to freeze drying of surface of the tailings stack.
- Filtration plants have occasional challenges and a temporary storage area(s) for one to three days of storage of material unsuitable for the dry stack is of great value to provide operational flexibility. This storage area should be close to the filtration plant so that the material can be readily reintroduced to the filtration process for permanent storage in the dry stack. In the case of lower tonnage operations, this storage can be achieved in large vessels/tanks whereas for larger operations, a lined impoundment is usually required.

Finally, filtered tailings dry stacks are not a panacea for mine waste management. They should be appropriately viewed as an alternative form of tailings placement and a part of the overall tailings continuum of options for today's designer/operator. There are site conditions, including regulatory regime, that make a tailings dry stack the best choice for certain projects. Where that is the case, the guidelines offered in this paper should provide a sufficient point to avoid the pitfalls that earlier dry stacks met and attain the successes that many current dry stacks demonstrate.

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Geotechnical/Geochemical Questions Related to Polymet Tailings[1]
Dana Dostert - January 18, 2012

PolyMet Mining proposes building a new tailings storage facility on top of the existing LTV tailings basin. PolyMet plans on mining old LTV coarse tailings for construction materials to be used in the perimeter dams that will contain the PolyMet tailings between the perimeter dams and above the lower lying LTV tailings. PolyMet further proposes to place a bentonite cap on top of the PolyMet tailings and maintain a wet cover to reduce or prevent oxygen from entering the PolyMet tailings.

PolyMet plans on using the upstream construction method, which will ultimately result in the perimeter dams being constructed over the PolyMet tailings. Dam Safety has experience with tailings dams that are constructed from the residue from the taconite industry, but has no experience dealing with tailings that will be derived from minerals in the Duluth Complex.

Furthermore, Dam Safety has numerous concerns with this project because the tailings dams must function properly for an extended period of time - we've heard on the order of 900 years. Our first concern is whether the PolyMet tailings will form a structurally sound base to support the perimeter dams. Our second concern is that the proposed wet cap will significantly increase the potential for a dam failure, and will result in costly monitoring and maintenance over the life of the project. (Including monitoring costs to DNR for 900 years)

Questions:
1. Geochemical: Will the PolyMet tailings be geochemically stable over the life of the tailings basin (900 years), or will they degrade into secondary minerals? Will those minerals migrate? Can we expect deformations within the basin due to the chemical breakdown and possible migration of the PolyMet tailings? What happens if the bentonite cap leaks and water enters the tailings?

- LAM Technical Input: The PolyMet tailings are projected to have the following primary mineral percentage ranges: 50-80% plagioclase, 10-15% olivine, 4-5% clinopyroxene and < 0.5% sulfides. Natural weathering processes will lead to breakdown of the primary minerals into less competent secondary clay and carbonate minerals. However, the extent and rate to which these reaction processes would occur under this setting is not clear. Leaking through the bentonite cap would likely increase the extent and rate of these weathering processes.

2. Physical: Will the PolyMet tailings be physically stable over the life of the tailings basin? Will they be prone to rolling, crushing and/or collapse? Deteriorate due to freeze/ thaw? Fracture due to moisture? It is our understanding that the PolyMet tailings will be rounder, less jagged and less likely to interlock - Is this correct?

[1] Supplemented by LAM technical input provided by Zach Wenz; EIS Guidance proposed by Stuart Arkley

- COE Technical Input: Slope stability of the TB dam is the principle mode of failure associated with these questions. From that standpoint there are two main properties that are considered with particle crushing or particle deterioration; shear strength and permeability. The follow on question is whether these physical changes from crushing or deterioration will affect the material structure leading to a greater potential for collapse from a looser state to a denser state. Lade, et.al., discusses factors affecting particle breakage in an ASCE paper entitled "Significance of Particle Crushing in Granular Materials." Some of these include stress level, where the higher the stress (especially shear stress) the greater the breakage; particle size as the larger the size the greater the number of flaws or defects (probably not an issue on Polymet); particle angularity since the more angular the more fracturing at particle points or breakage along narrow dimensions; uniformity of materials as the more uniform the less the number of contracts and therefore higher contact stresses; and mineral hardness where the harder and stronger particles have lesser breakage for the same stress. Assuming the hardness of Plagioclase is about that of Orthoclase, the hardness is about 70% of quartz. Taleb Al-Rousan, et.al. reports in the paper "Effect of Inherent Anisotropy on Shear Strength Following Crushing of Natural Aqaba Subgrade Sand" that coarse granitic sand particles with 2.8 mm diameter experience breakage at a pressure of 2MPa (41,770 psf). Using a moist density of 100 pcf and no groundwater conditions this would equate to a fill height of 418 ft. This is a compressive stress and says nothing about breakage under shearing but based on information reviewed it is felt that particle crushing and degradation isn't a significant concern. Instead, the structure or fabric of the tailings and their potential to liquefy under dynamic or static loading is the governing behavior that should be investigated. Freeze-thaw degradation might have an effect on the outer band of the TB (say the outer 7-10 ft) but since processing has eliminated bedding plane and fracture defects we are really only talking about degrading minerals and from judgment it is expected that this occur very slowly. According to Lade and others, less angular particles are subject to less breakage and therefore less potential change in shear strength and decrease in permeability from design conditions.

3. Alterations: Will Polymet tailings congeal over time (900 years) due to pressure and the presence of Ca and Mg, or will they remain loose with characteristics similar to a fine sugar or flour.

COE Technical Input: Aging of soils will likely have some impact on shear strength. As water levels drop and effective stresses increase it may be expected that the grains will move into a denser, more stable position. This doesn't mean that the soils are no longer prone to liquefaction. It is felt that once steady state conditions are reached and no further consolidation occurs the aging process can continue and include cementation effects. Cementation can take place over hundreds or thousands of years and it seems to be unconservative to count on this cementation or congealing to allow the use of higher shear strengths for design.

-

4. Is there a geochemical concern if there is a dam failure and PolyMet tailings spill into the surrounding wetlands, so long as no one gets hurt, or are there no concerns? US EPA has stated they would require the tailings to be picked up.

- LAM Technical Input: PolyMet has projected the sulfur content of the tailings to be in the neighborhood of 0.1% to 0.2%. Given the small particle size of the tailings and their likely widespread deposition resulting from a dam failure it is likely that water quality violations regarding sulfate, metals, or suspended solids could arise.

5. Are there any other geochemical, or other concerns related to the PolyMet tailings that could impact dam stability over the life (900 years) of the project that we should be aware of? Are the concerns we have expressed valid?

- LAM Technical Input: The potential for PolyMet tailings to breakdown into less competent minerals (clays and carbonates) overtime could negatively impact the stability of the tailings basin. However, the probability that this would lead to failure of the tailings basin is not clear. Therefore, requesting a projection of the effect of mineral weathering to the stability of the tailings basin is valid.

Message

From:	Olson, Michael (DNR) [/O=MMS/OU=EXCHANGE ADMINISTRATIVE GROUP (FYDIBOHF23SPDLT)/CN=RECIPIENTS/CN=OLSON, MICHAEL (DNR4C96FCD1-0B4B-4AF7-BAAA-FFB6973C9B80]
Sent:	2/28/2013 10:01:05 AM
To:	Kunz, Michael (DNR) [michael.kunz@state.mn.us]
Subject:	FW: PolyMet Tailings Wet Closure

FYI

From: Donald Sutton [mailto:sutton@spectrum-eng.com]
Sent: Monday, January 23, 2012 11:52 AM
To: Dostert, Dana M (DNR)
Cc: Arkley, Stuart (DNR); Engstrom, Jennifer N (DNR); Olson, Michael (DNR); Wenz, Zach (DNR); Boyle, Jason (DNR); 'Ross Vellacott'; 'Jeff Coffin'; 'Schwanz, Neil T MVP'; 'Bryan Ulrich'
Subject: RE: PolyMet Tailings Wet Closure

Dana
I reviewed the documents you and Stuart provided. I share your wet closure concern and have additional concerns related to the long term tailings wet closure uncertainties and risks. I don't have many comments on the Barr documents other than thinking that the bentonite amended dam face and interior slopes will be subject to faster erosion if more precipitation runs off and less infiltrates. This could lead to other erosion problems, especially on the outside, because the slope geometry is geomorphologically unstable and the sandy matrix invites erosion. Can the soil cover become saturated and slide off the bentonite? I think the bentonite cover will eventually deteriorate due to erosion and plant roots and become ineffective, and that the erosion will weaken and destroy the embankments. If air is permitted to enter the tailings, they will oxidize and the purpose of the wet closure with bentonite seals will be negated. In my opinion, the reclamation plan is not a stable permanent closure.
Regarding the stockpile foundation settlement, my only concern is that they make sure that leachate from the piles flows to manageable collection points and doesn't leak. It would be helpful if any leakage reported to the open pits, because then it could be collected and treated with the other mine water discharge.

I've been thinking about alternative closures for the tailings basin, how closures might be designed, and how surface water and leachates would be permanently managed. I believe the purpose of the wet cover is to inhibit or prevent sulfide oxidation and the generation of sulfates, but as I discuss later, PolyMet's proposal may not completely prevent oxidation and infiltration if the bentonite seal is not installed properly or after it erodes away, so water collection and treatment may still be needed depending on the sulfide/sulfate content. If there is a reasonable risk that the wet closure won't prevent oxidation or sulfates for 900 years and if perpetual water collection and treatment will be needed, then why not investigate some dry closure options and compare the long term O&M costs and long term risks of each alternative? Perhaps there is a dry closure alternative that is more economical and less risky when perpetual maintenance O&M are considered. At some point, the cost of the risk will need to be assessed.

The challenge is to predict the amount and rate of sulfate generation for each alternative, how it will be collected and treated, and estimate how long sulfates will be generated depending on the oxidation rate, the infiltration rate, and the initial sulfide type and concentration. What is the maximum acceptable rate sulfate can be permitted to escape the site? This might tell us how long PolyMet or the State needs to collect and treat before the source is consumed and the pile of tailings becomes geochemically benign. If we are considering a 900 year life, I suspect that if the sulfides oxidize and flush out that eventually the source will dissipate and there will be no more contaminants. After the sulfides/sulfates have dissipated to an acceptable level, what is the remaining risk? The PolyMet plan attempts to minimize the sulfate generation rate, which will require a longer time before the sulfides and sulfates are flushed out. What about flipping the logic around and maximizing the sulfate generation rate (oxidation rate) to minimize the length of time this problem will persist? If water must be collected and treated for both cases, then which one will be more economical?

In its simplest form, the proposed tailings basin will be a big pile of highly erosive loose sand and silt. The wet closure will include a pond of water on top that saturates the sand/silt making it less stable and more likely to fail than the dry option. For a wet closure to remain geotechnically and geochemically stable until 2900, the water level and infiltration rates must be perpetually regulated, and the embankments and water management controls must be perpetually inspected and repaired. In the Tailings Basin Reclamation summary in NorthMet Project Description, p73/115, the anticipated perpetual maintenance items include: maintaining the pond no closer than 625 feet from the interior edge of the dams, pumping water from Colby Lake to maintain the pool level, recycling seepage back to the pond or sending it to a water treatment plant to reduce the sulfates to 10 ppm. It is assumed that eventually the pond water level will stabilize and the sulfate level will become less than 10ppm, but the document doesn't explain how they come to this conclusion. Surplus storm water will be directed through "clog resistant" lined/revegetated or riprapped channels or pipe outfall structures. Oxidation of the tailings will be prevented by placing a layer of bentonite 30 inches below the surface of the dams and beaches. Bentonite will be placed on the pond bottom to minimize infiltration.

I envision that PolyMet's reclamation plan could work for a while, but don't see how it will function forever without falling apart unless it is continuously maintained; which is a major leap of faith. The wet closure will require perpetual maintenance and water management to control the water level in the pond, to collect and treat seepage, and to maintain the dam embankments and the flood control structures, pipes, etc.

We are assuming that the geotechnical modeling reasonably reflects expected conditions into perpetuity, and that the model input variables are representative and reliable. We are trusting PolyMet's lab scale data for the tailings size consist and properties. If this were a dry closure, and the tailings were allowed to drain, then I wouldn't be so concerned, because mill tailings tend to be very rough and angular and can gain strength as they dewater and compact. Wet closure is different. I believe that the MSHA guidance assumes that at some point in time there will be a dry closure, and that the saturated conditions are only temporary.

In addition to seismic and geotechnical failure events, dam failures can be triggered by water related issues such as piping or erosion that weaken the structure over time, or catastrophic storm events that overwhelm the structures or gradually weaken them. What happens to these structures after decades or centuries of wet/dry, freeze/thaw cycles? Can they crumble or crack? Roots and vegetation can plug them. Water can leak under or around them and cause piping.

When we drove around the embankment perimeter in November, I could see that portions of the embankment were already eroding. The loose sandy embankment is not very resistant to erosion. What happens to the stability over time if the embankment becomes thinner due to erosion? Has anyone calculated the erosion rate of the outside embankment? If it loses 1 inch per year, then in 900 years it could be 75 feet thinner. How might that change the stability? What if it erodes 2 or 3 inches per year? I noticed some animal burrows. How far do they burrow? How far do they need to burrow before encountering the saturated zone? If there is a concentrated flow during a major run-off event, or if piping occurs there can be rapid and major erosion of the embankment that might allow the ponded water to escape. If the out-slopes are made impermeable (with a layer of bentonite), then run-off will be greater and erosion will more rapidly attack the embankment because less water will infiltrate and more will run off. During a thunderstorm there will be a lot of water coming down the outside that will cause erosion and saturate the 30" of fill above the bentonite, possibly resulting in a sliding failure, rills and gullies. Once the erosion begins, it can continue head cutting into the impounded water unless it is regularly repaired.

The climate is changing in unpredictable ways that makes it difficult to predict the water balance. For example, at the Zortman and Landusky sites in north central Montana, we have experienced three plus 100 year events in the last 10 years. In 2011, we experienced an event believed to be the 200 to 500 year event that overwhelmed all our water management and water treatment facilities and caused a waste dump to collapse and destroy a water capture and pump back station. The average precipitation for the last 2 years is 100 % above average. I don't know how the climate change will affect MN, but it is getting wetter in some areas and drier in other areas. If it becomes wetter or drier than anticipated how will it affect the closure water balance? How will it affect the saturation of the embankments or the erosion rates? How will it affect the storm water controls? What about wave erosion on the inside combined with wind and water and varmit erosion on the outside? More thunderstorms would cause more rilling and gullying. All the materials are highly susceptible to erosion, so more intense rainfall will cause faster erosion.

What happens if the bentonite doesn't seal the infiltration and the pond water drains out faster than anticipated, or if the climate is drier? How long does PolyMet intend to pump water from Colby Lake? If the Colby Lake pumping funding is used up, and the pond dries out, it essentially becomes a dry closure. I suspect their wet closure plan will only function within a narrow range of precipitation and leakage assumptions without perpetual pumping from the lake during dry periods and decanting surplus water during wet periods.

PolyMet's plan to recover as much sulfide as possible (including pyrite) to minimize the sulfide content in the tailings is a good idea. I know they can never get the sulfide level to zero, but is there some level that would be low enough that would meet the same standards as the low sulfur waste stockpile? If PolyMet spent more to clean up the tails, then perhaps there could be long term closure savings if the sulfide level was no longer an issue.

The environmental consequence of a failure depends on the amount of sulfate the local ecosystem can tolerate and the length of time the problem will persist. The amount of sulfate available after a failure depends on the initial sulfide content of the tailings, the rate the sulfide oxidizes to sulfates, the rate the sulfate is released from the tailings, and the elapsed time. All combine to control the remaining sulfide/sulfate that would be released after a failure. (PolyMet is proposing to recirculate the seeps – if they do that, then I assume the sulfate concentration would increase, which would create worse contamination if it all escaped at once).

If the sulfides oxidize and are gradually flushed out of the bottom of the tailings basin or into a seepage collection system and treated or discharged, then eventually the tailings will become relatively inert. How long will this take? Probably a long time if they remain sub-aqueous beneath a bentonite seal, but if the basin doesn't hold water and the tailings are allowed to rapidly oxidize or are forced to oxidize rapidly, it might become inert fairly rapidly.

Perhaps, rather than trying to prevent oxidation, there might be a way to accelerate the oxidation to minimize the collection and treatment time. Consider installing a leachate collection system in the taconite tailings basins before the new tailings are added. This might entail putting the bentonite on the existing surface and installing some type of under drain collection system before the tailings are placed on top. This alternative might be more economical than placing the bentonite on top of the tailings and maintaining everything forever. In this alternative, there would be no need for any type of low permeability cap – all the precipitation would be allowed to infiltrate. Some additional water management would need to be engineered so 100% of the precipitation doesn't report to the leachate, but there would be less need for elaborate storm control features other than some decants similar to those already installed in the taconite ponds to drain away any large storm events.

I was trying to imagine a dry closure design that would be geomorphologically stable. The challenge is to get the precipitation off the top without causing erosion. If the tailings are graded so that water runs off the interior, there will be a big risk of erosion and gullying in the sandy material unless the velocities can be kept low, which will be very difficult to achieve without substantial regrading or some type of a drop inlet decant structure. Lined channels might work for a while, but are likely to fail after several years if water gets under the liner. I imagine that the plus 100 year event flow rates will be considerable and the velocities quite erosive if the run-off is uncontrolled. Any type of impermeable cover system will be very expensive and will have trouble dissipating the run-off energy. The cover will eventually erode and unravel. A stable configuration may be to allow the precipitation to infiltrate and evaporate or evapo-transpire as it is doing today, but how do you deal with the leachate? Will the pond actually drain sufficiently to reduce the saturation levels, or will some type of permanent decant or central drain system be needed, similar to the existing system, to minimize the pool, reduce the area that is saturated, and reduce the quantity of leachate that must be captured and treated?

Geochemically, a wet closure is only practical if it permanently inhibits sulfate generation to a rate slow enough that no water treatment or collection is needed. If the wet closure does not prevent oxidation or if the oxidation of the fine grains occurs during milling or when the fine grained sulfides are exposed in the pond before being buried, then leachate will need to be collected and treated. In this case, wet closure may act geochemically similar to the uncovered dry closure, but may take longer for the sulfates to dissipate and be flushed out. If the wet closure successfully prevents oxidation and sulfate production, then a structural failure will release all the sulfides that will rapidly oxidize to sulfates, so the contamination event is only delayed, not prevented. When a wet closure failure occurs, the environmental damage my occur before the spill can be managed, especially if the sulfate concentration is high.

If seepage collection or treatment is or might be necessary for an indefinite time with a wet closure, then what is the benefit of wet closure? The wet closure is riskier, has more uncertainties, and may be more expensive because it will require more perpetual care and maintenance than a dry closure. I suggest that PolyMet investigate some alternatives. My preference is to look into installing an underdrain collection on the taconite so that only tailings leachate is collected and treated. I assume PolyMet already knows the sulfide content of the tailings under various recovery scenarios. They may need to run some additional grinding and flotation tests, but it would be useful if they could investigate the oxidation rate of the finely ground tailings when exposed to air, and to explore the amount of oxidation when treated with various oxidizers to determine if my idea about accelerating the oxidation has any merit. This might be something the geochemists should comment on first. I don't like the wet closure, because it is not a permanent closure. I believe it will eventually fail and release the sulfates.

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From: Dostert, Dana M (DNR) [mailto:dana.dostert@state.mn.us]
Sent: Thursday, January 19, 2012 10:11 AM
To: Donald Sutton
Cc: Arkley, Stuart (DNR)
Subject: RE: PolyMet

Hi Don,

Sorry that you couldn't get into the meetings as your expertise would have been valuable.

I sent out a list of questions prior to the meeting. Do you have any comments on those. Our Geochemists don't think there is a geochemical concern but there may be a physical one. Neil is going to provide some comments on my questions. If you could provide any comments to my questions and any of your own, that would be appreciated.

Thanks

Dana D.

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From: Donald Sutton [mailto:sutton@spectrum-eng.com]
Sent: Thursday, January 19, 2012 11:03 AM
To: Dostert, Dana M (DNR)
Subject: PolyMet

Dana
I couldn't get into the meeting conference call. I kept getting a message that the meeting hasn't started.
Did you need anything from me?

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The RISK, PUBLIC LIABILITY, & ECONOMICS of TAILINGS STORAGE FACILITY FAILURES

Lindsay Newland Bowker[1] & David M. Chambers[2]
July 21, 2015

1. Introduction

Prior works interpreting the history of Tailings Storage Facility (TSF) failures, 1910-2010, have concluded that the lower numbers of failures and incidents in the two most recent decades evidence the success of modern mining regulation, improved industry practices and modern technology. When examined more closely the 100 years of TSF failures shows an emerging and pronounced trend since 1960 toward a higher incidence of "Serious"[3] and "Very Serious"[4] failures. That is, the consequence of loss is becoming increasingly greater.

In a keynote address at a 2011 tailings conference Dr. A. Mac G Robertson described this trend and its implications going forward as elevating risk potential by a factor of 20 every 1/3 century. His address called a "red flag" on the current "Mining Metric" which results in ever larger and higher TSFs (Robertson 2011).

Risk potential has increased by a factor of 20 every 1/3 century. (Robertson 2011)

The Mining Metric creating this exponentially increasing consequence in the event of a tailings dam failure, is driven by continuously lower grades in identified resources and continuously falling real prices of most metals. The costs to excavate more material for a ton of end product at a lower price has been made possible through technology improvements in milling and concentration processes, bulk mining and economies of scale. There have been some new technologies e.g. dry stack and paste tailings and the more prevalent use of center line over upstream dam designs which offer the potential for lower consequence in the event of failure, and perhaps a lower overall risk of failure. However, many of the same features of modern mining that create economic feasibility in lower grades of ore also pose greater challenges for the management of mine waste and waste water. One of the manifestations of these challenges overall is a greater frequency of Very Serious tailings dam failures with significant levels of social and economic consequence, sometimes non remediable.

The modern "Mining Metric" is well mapped: higher mine production necessitated by lower grades of ore, a century of declining prices offset by declining costs per ton. The metric is to continuously develop the resource through economies of scale, larger and deeper footprints, more efficient operations, bigger and better bulk mining technology.

49% (33/67) of all recorded Serious and Very Serious failures from 1940-2010 have occurred since 1990. Of all 52[5] recorded incidents cited, 1990-2010, 17 (33%) were Serious failures, i.e. large enough to cause significant impacts or involved loss of life. Another 16 (31%), were Very Serious failures, i.e. catastrophic dam failures that released more than 1

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[3] We defined Serious failures as having a release of greater than 100,000 cubic meters and/or loss of life.

[4] We defined Very Serious failures as having a release of at least 1 million cubic meters, and/or a release that travelled 20 Km or more, and/or multiple deaths (generally ≥ 20).

[5] Our study included authoritatively documented TSF failures that were not in the WISE or ICOLD inventories. See Appendix 1, TSF Failure Data Table, for a complete list of TSF incidents & failures included in our study and the basis on which they were classified.

million cubic meters of tailings and in some instances resulted in multiple loss of life. 63% of all incidents and failures since 1990 were Serious or Very Serious. The total costs for just 7 of these 16 large failures was $3.8 billion, at an average cost of $543 million per failure (See Appendix 3). These losses, according to dam committee reports and government accounts are almost all the result of failure to follow accepted practice. These failures are a direct result of the increasing prevalence of TSF's with greater than a 5 million cubic meter total capacity necessitated by lower grades of ore and the higher volumes of ore production required to attain or expand a given tonnage of finished product. We project 11 Very Serious failures 2010-2020 at total unfunded unfundable public cost of $6 billion. We estimate an additional $1 billion for 12 Serious failures this decade. These losses are uninsurable. Very few miners can simply absorb a loss at this scale without risking bankruptcy and permanent closure of a resource that has not yet been "mined out". There is no organized industry attempt to pool these losses in the context of a risk management loss prevention program, and no political jurisdiction issuing permits is large enough to prefund a low frequency high consequence loss of this scale. The inevitable result is either government pays or the damages go unremediated.

Much of our data on cost of large scale failures was sourced from court cases or proceedings where government sought unsuccessfully to recover what had been spent on remediation, compensation for damages or assigned as value for actual socio economic and natural resources loss. Shielded via wholly owned subsidiaries who can legally declare bankruptcy when liabilities exceed assets of the subsidiary (not the parent), the parent companies paid little or nothing toward most of these large losses. In countries founded on the common law tradition that all are responsible for the consequence of their actions, this gap between outcome and expectation for the most serious local impacts violates the terms and conditions of a "social license to operate" and fails to meet a standard of "polluter pays".

Miners Must Move Forward or Perish (Jones 2014)

As we have seen with Mt. Polley, very large releases do not just occur at very large mines. In comparison to the scale envisioned by mines like Pebble or KSM, the Mt. Polley TSF was relatively small, only about 35 meters high at failure with a total capacity of about 74 million cubic meters (Independent Panel 2015). In fact this is the pattern we see on close examination of Very Serious and Serious failures; older TSFs with smaller footprints are pushed to unplanned heights to accommodate additional production that was not anticipated when the tailings dams were originally designed and the permits originally issued.. Capital markets and investors don't finance clean ups. They finance production that is profitable. Smaller companies operate on tighter margins within the same overall metric affecting all miners but are less able to take advantage of and finance optimizations or achieve economies of scale that will keep production costs low enough to maintain a specific mine site as economically feasible.

Our sense of the data, and the case histories we have looked to for a deeper understanding of the data, is that "mining economics" plays a significant role in TSF failures. It is important in permitting, and in the checks and balances built into the regulatory process over the life of a TSF, to look beyond "mechanisms of failure" to the fundamental financials of the miner, the mine, and mega trends that shape decisions and realities at the level of miner and individual mine.

Our sense of the data and the case histories we have looked to for a deeper understanding of the data is that "mining economics" plays a significant role in TSF failures and that it is important in permitting and in the regulatory process to look beyond the "mechanisms of failure" to the fundamental financials of the miner, the mine, and mega trends that shape decisions and realities at the individual mine.

Taking our study of the relationship between "mining economics" and TSF failures 1910-2010 into account, it is our expectation that large failures in the near term (through 2020) will continue to come from operating mines under ownership of smaller miners first commissioned from the late 60's to the early or late 80's. These smaller older mines are producing within the Mining Metric of lower grades and now steeply rising production costs against the continuous possibility of a sharp adverse price swing but with much less capital, as compared with larger mines, to buffer contingencies or provide required levels of stewardship for TSFs from design through closure. For a mega mine like the 100 year old Bingham Canyon mine it was possible to respond to an identified threat of failure and the growing environmental problems of age. It is not clear how smaller old mines will find the funds to identify or respond in a timely fashion to threats at their facilities, or whether regulatory structures now in place will serve well enough to identify such "at risk" facilities.

If they are identified in time, it is not clear how smaller miners skating on thin balance sheets will finance the closure or improvements at TSFs and carve out the funds for new TSFs where necessary. Larger mining companies, however, are better positioned financially to manage and mitigate these threats.

This study anticipates the future trend of Serious and Very Serious TSF failures over the next decade, through 2020, and estimates the total public economic consequence of those failures, which are presently unfunded and un-fundable. We borrow the applicable elements of "loss development" in insurance rate making utilizing 100 years of data on loss and consequence and on the production levels of the mining metric producing TSF waste volumes to project an expected number of failures and an average expected loss per failure from which global estimates of expected public loss can be reasonably estimated.

Having something more like "actuarial data" to refer to is important in understanding the potential magnitude of loss from an individual dam or a permitting districts portfolio of dams and TSFs. With such low frequency high severity losses we can never assign risk to an individual TSF based on its design and receiving environment parameters. Unless it has an identified flaw that puts it at near certain risk of imminent failure, we can't say whether a given dam "will" fail. We can only say what the consequence would be in economic terms if it failed.

Satellite imagery has lead us to the realization that tailings facilities are probably the largest man-made structures on earth. Their safety, for the protection of life, the environment and property is an essential need in today's mining operations. These factors, and the relatively poor safety record revealed by the numbers of failures in tailings dams have led to an increasing awareness of the need for enhanced safety provisions in the design and operation of tailings dams. (ICOLD 2001)

2. INCREASING CONSEQUENCE OF FAILURES

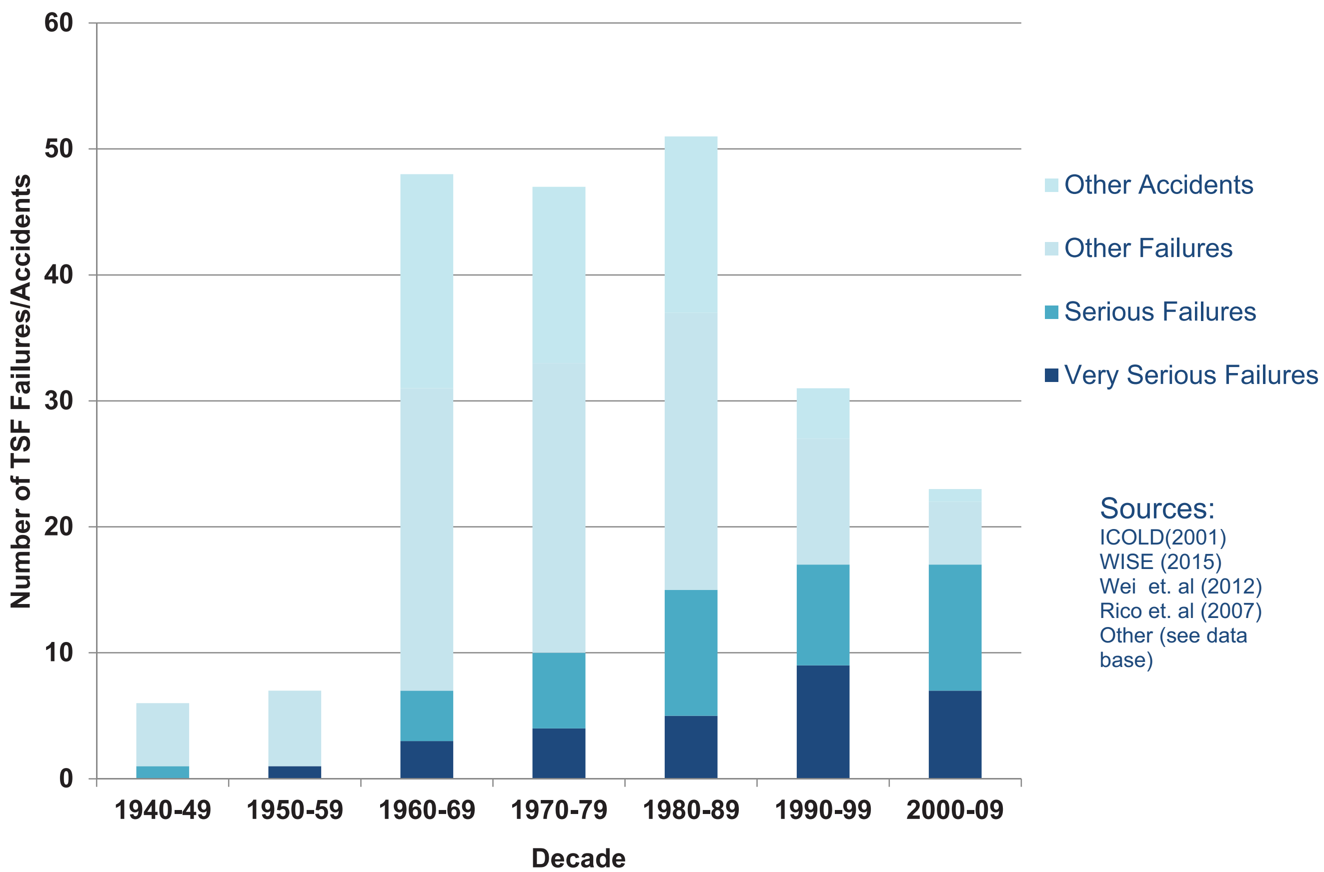
For this study we are interested primarily in the history and trend of Serious and Very Serious Failures rather than all incidents in the International Commission on Large Dams (ICOLD) or the World Information Service on Energy (WISE) compilations. These are the failures that cause consequential compromise of environmental security beyond the mine site. Serious and Very Serious failures accounted for 31% (67) of the 214 TSF failures and accidents 1940-2010, but comprise 63% (33/52) of the 52 total incidents, 1990-2010, with sufficient data for meaningful analysis.

We defined Serious failures as having a release of greater than 100,000 cubic meters and/or loss of life. 38 recorded incidents out of the 214 failures and accidents in the period 1940 to 2010 (18%) that had sufficient data for analysis met that criteria. 17 of those (45%) occurred in the last two decades.

We defined Very Serious failures as having a release of at least 1 million cubic meters, and/or a release that travelled 20 Km or more, and/or multiple deaths (generally ≥ 20). Very Serious failures comprised 14% of total historic events (29/214), but 31% (16/52) of all incidents and events in the past two decades (1990-2010). The complete list and criteria is presented in Appendix 1, TSF Failure Data Table.

This very clear trend to larger and more consequential losses is apparent in Figure 2.1 below. The clear aqua and paler blue is the distribution of incidents other than failures, most of which are very small with little or no release or consequential damage. Prior to 1980 Other Failures and Accidents (pale and aqua blue) were most prevalent. Post-1990 Serious and Very Serious failures (deep and dark blue) dominate.

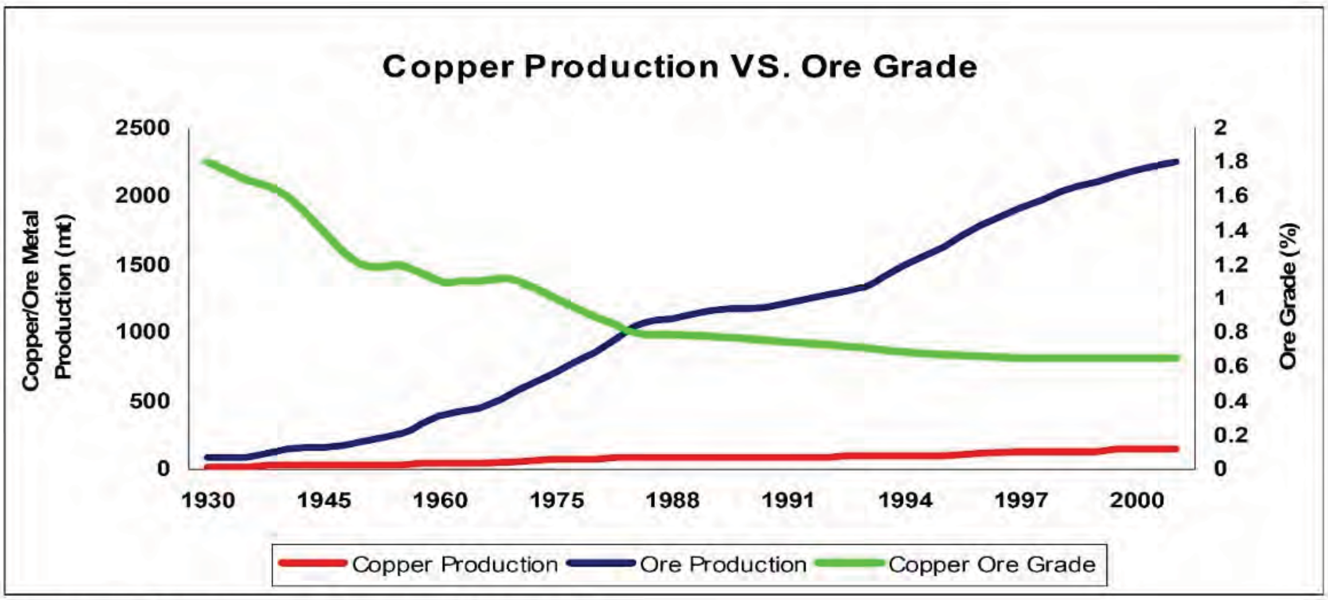
Figure 2.1 Increasing Severity of TSF Failures Globally 1940-2010



3.0 RELATIONSHIP BETWEEN LARGE FAILURES & THE MINING METRIC

Our aim was to explore the relationship between economic factors not explicitly accounted for in the permitting and regulatory oversight of mines and the observed trend toward failure incidents of greater consequence. Our data base included a count by decade of failures (Serious failures, Very Serious failures, Other failures, and Other Accidents) and a data set of variables describing the main economic trends driving mine production: price, costs to produce and grade. The following chart for copper prepared by the Raw Materials Group for the World Bank (World Bank 2006) describes the generic fundamental elements of the Mining Metric affecting all primary metals and most precious metals.

Figure 3.1. Copper Production & Ore Grade



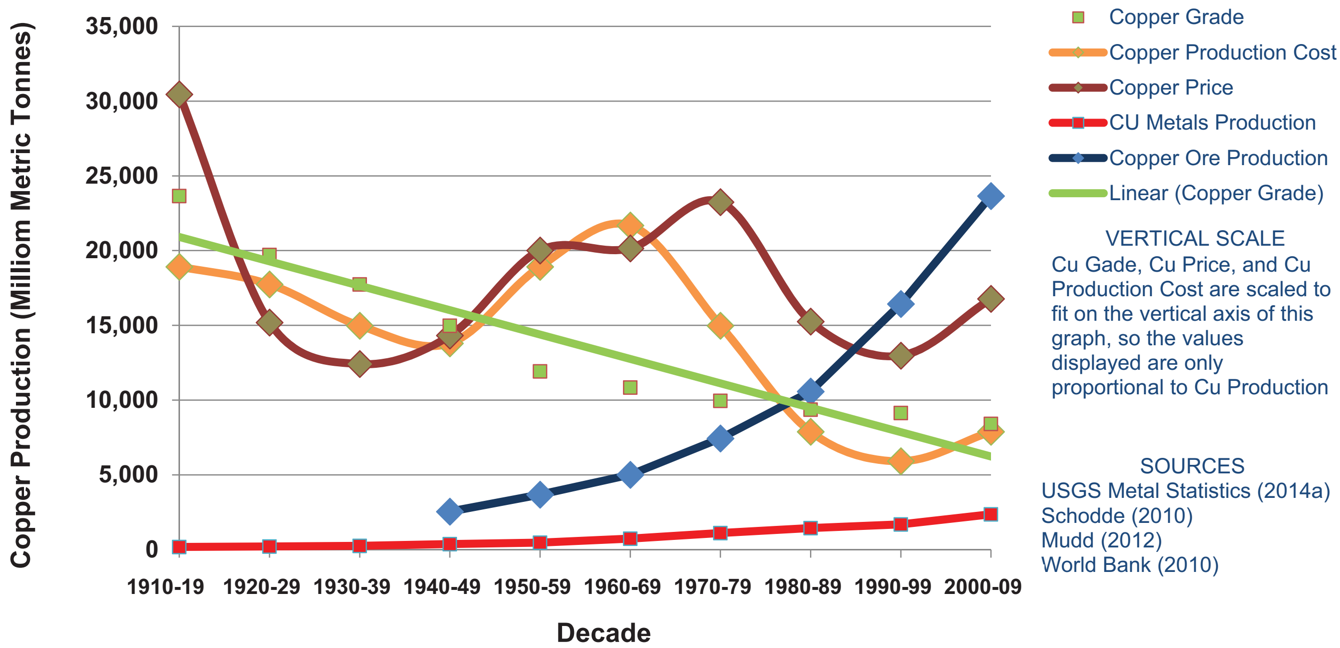
Source: *Raw Materials Group, Stockholm*

The chart is highlighting the very dramatic change in the relationship between metals output (the red line) which increased only 17% over the decade 1990-2000 and ore production[6] which increased 63% as grades continued to decline. The two key elements missing from this chart that explain how it was possible to "grow the resource" against a long trend of falling prices and falling grades the economic viability of these trends are the market price of the red line (the final refined product) and the costs to produce are highlighted by Richard Schodde, who noted that the declining costs to produce more than offset a century of falling prices. (Schodde 2010)

This fuller context is shown in Figure 3.2 below. That production costs have offset price is apparent through 1990.

[6] In our analysis we have used copper ore production data taken from the World Bank/Raw Materials Group graph because it is the only available published data for copper ore production. We have also done a comparison by using average copper ore grade and metal production to back-calculate to ore produced. For the back-calculation we used metal production data from Kelly & Matos (USGS 2014a), Schmitz/ABARE (Mudd 2012), the International Copper Study Group (ICSG 2014), and copper grade data from Mudd (2012). These data compared very favorably with the World Bank/Raw Materials Group data. We made several attempts to contact the Raw Materials Group through their corporate parent, SNL Metals & Mining, in an attempt to both verify the data (World Bank 2006) and the method(s) they used to develop it, but did not receive a response to these inquiries.

Figure 3.2 Mining Metric 1910-2010
Declining Prices Offset By Lower Production Costs



In correlation analysis, Table 3.1, price had a lower correlation than production cost with all failure classes. The most significant correlations with the four failure variables were with Cu Production Cost, Cu Grade and annual Cu Ore Production volume and Cu Metal Production. The correlations were only notable with the two highest failure severity categories. Cu Metal Production had higher correlations with both Very Serious failures (0.881) and Serious failures (0.826) as compared with Cu Ore Production. Cu Ore Production is more closely related, however, to TSF waste volume and also seems to distinguish between the two highest severity classes. This small difference also occurs with Cu Grade (greater negative for Serious) and Cu Production Cost (greater negative for Very Serious).

Table 3.1 Correlation Between Failure Severity and Mining Metric Indicators

	Cu Ore Production	Cu Metal Production	Cu Grade	Cu Prod Cost	Cu Price
Very Serious Failures	0.860	0.881	-0.794	-0.788	-0.427
Serious Failures	0.720	0.826	-0.884	-0.682	-0.126
Other Failures	-0.265	-0.099	0.298	0.300	0.489
Other Accidents	-0.216	-0.050	-0.312	0.281	0.485

Abbreviations:

Cu Prod Cost = Cost to produce copper concentrate from copper ore, including waste disposal

Cu Grade = grade of copper in the ore

Cu Prod = copper ore production

Other Failures = tailings dam failures and incidents other than Serious or Very Serious Failures

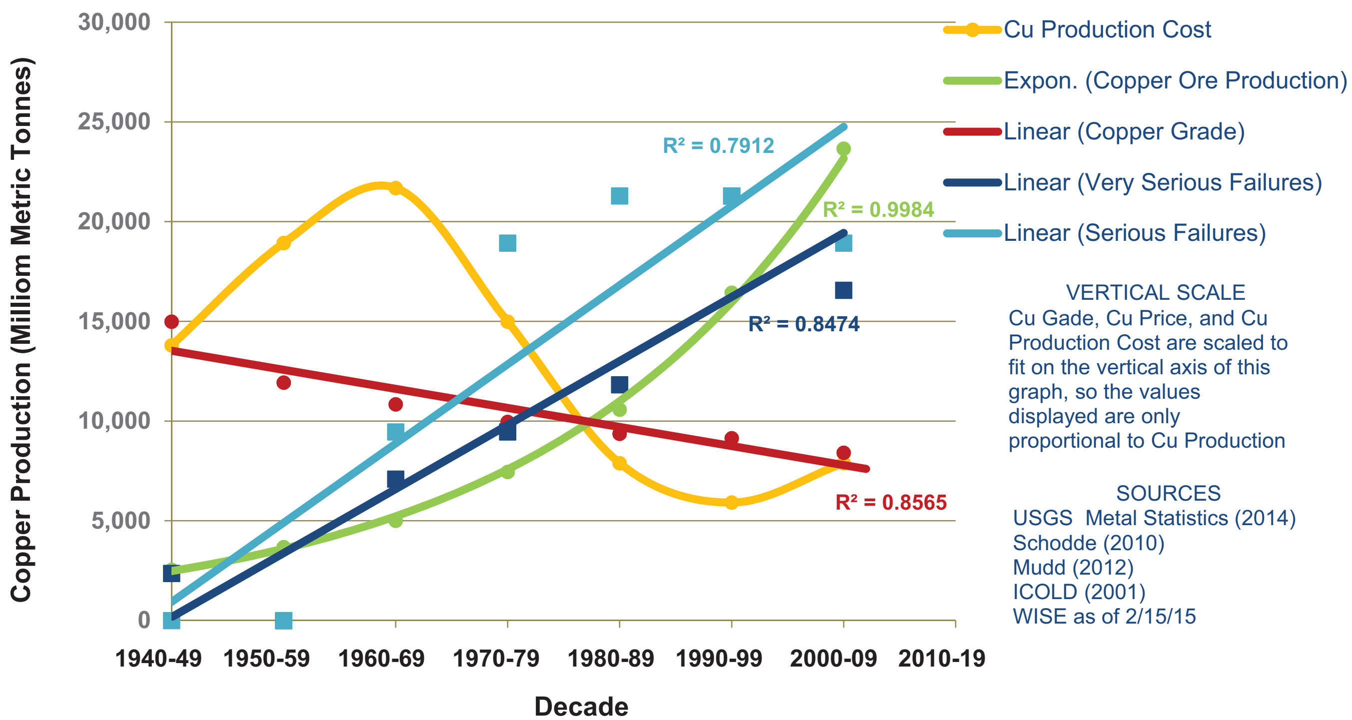
Serious Failures = Serious tailings dam failures

Very Serious Failures = Very Serious tailings dam failures

Sources: USGS Metal Statistics (2014a), Schodde (2010), ICOLD (2001), WISE (2015) & additional

Therefore, we chose Cu Ore Production, Cu Grade and Cu Production Cost to produce for further analysis. We did not include, or have a basis for deeper consideration, of copper price. These relationships are graphically presented in Figure 3.3 below.

Figure 3.3 Relationship Between Mining Metric and TSF Very Serious and Serious Failures 1940-2010



The key mining metric variable, Copper Production Cost to produce, dropped from $85/tonne in 1900 to only $15/tonne in 2000. Over this same period price dropped from $7,723/tonne to $3,292 per tonne. The largest cluster of Serious and Very Serious failures of TSFs, 88% (59/67), occurred in the long downward price trend from 1970 to 2000. 86% (25/29) of Very Serious failures and 89% (34/38) of Serious failures occurred during this period. 2000 marked the beginning of an upward trend in price but also a 33% increase in costs to produce, from $15/tonne in 2000 to $20/tonne by 2010 but with Serious and Very Serious failures still representing 71% (15/21) of all failures for the decade 2000-2010.

The dramatic shift emphasized in the World Bank/Raw Metals charts (Figure 3.1) co- occurs with an upward swing in costs to produce while grade continues to fall (Figure 3.3). This suggests a higher level of financial risk beginning in 1990, which co-occurs with the emergence of Very Serious TSF failures.

Our data suggests that the many smaller mines and miners that became part of global production of all primary and precious metals post-1950 were not as able to take full advantage of as many of the technologies and economies of scale as larger miners, and therefore remained more sensitive to price changes than larger miners, with frequent shutdowns in a small portfolio of investments as price changes made continued production unviable. Smaller miners run on thinner balance sheets with more price vulnerability in comparison to the larger miners.

Another major factor affecting stewardship for TSFs and other mining environmental liabilities, which was not mapped sufficiently for inclusion in our database, is access to capital markets. Smaller mines have always had access only to more risk tolerant markets, such as the Toronto Stock Exchange, and sometimes, as in the case of Mt. Polley, with one or two specific backers. The top miners are financed through markets with tight, well defined credit standards and an increasing underwriting emphasis on full disclosure and accounting of environmental liabilities. Smaller miners have almost no meaningful access to insurance for their environmental liabilities, whereas larger miners have more integral relationships with insurance and reinsurance markets (even though the types of risks that are insurable are no different between large and small insurers). These large market relationships create more external accountability to environmental risk management and to financial risk management for larger miners than exists for small miners, and a more rigorous ongoing process of review and reckoning. Regulatory structures don't include enough structure on assessment of financial capacity to balance that difference creating an "apparent norm" of higher financial risk in smaller mines that translates into the higher losses we see in the historical data.

Two significant changes in financial risk also weigh more heavily for smaller mines than for larger mines: a radical contraction of all capital markets for mining (Jones 2014); and, a 30% increase in costs to produce. The increase in costs to produce is across the board and attributable, according to informed market analysts, to both an increase in energy costs and also in foreign exchange rates. Chile, a major producer of copper globally, has had to commit to a major capital program to improve its mining infrastructure to maintain grade and hold its place in world concentrate markets.

While each principal base metal (iron, aluminum, copper, zinc, etc.) has its own version of the Mining Metric, the basic "shape" and slope of trend lines for production and price for all base metals are the same. The basic bottom line, vis-a-vis manifest environmental loss across all metals, is the same. All operate on close margins. Those with larger budgets, better quality assets, lower production costs and uniform corporate policies on optimization and efficiency at each site, and who can also achieve economies of scale, will generally fare better than smaller miners with tighter budgets and less access to global capital markets. The global capital markets are able to provide external checks and balances on financial/risk management relationships that hold miners to account on environmental liability management, even when regulatory structures don't – but only if the miner in question is working in the global capital market.

Copper is widely recognized as a bellwether base metal for the mining industry. Most works on mining economics use copper as the "index metal". Beyond that, the greater quality and detail of regularly produced copper commodity information over the entire last century led us to explore its use as the index metal for TSF failures, i.e. expressing TSF failures per million tons of copper production. The USGS publishes metal statistics on two of Mining Metric elements, price and mine production, but no historical data on costs to produce or grade. So copper is the only metal for which it was possible to establish a full century long "actuarial" data base on the relationship between the economics of mining and environmental loss attributable to TSF failures. Going forward it will be possible to build the data base for other metals from current and data and short term projections. In the next section we present the statistical correlation between mining economics and TSF failures.

4.0 The Statistical Correlation between Mining Economics & Environmental Loss from TSF Failures

We chose Canonical Correlation Analysis (CCA) as a way of further exploring the relationship between the failure severity categories we created for this research and the main elements of the mining metric that affect all miners and all mine sites. We were interested in knowing whether there is a significant relationship and if so, whether it warrants greater attention in permitting standards and oversight of mine permits. We know from past study of TSF failures that there are many physical attributes of a TSF that influence severity as well as other often noted but so far unstudied factors such as the structure of the regulatory framework and the technical capacity available to oversight.

Canonical Correlation is a multivariate technique that aims at identifying the degree of influence of one data set with another (rather than causality). We had no pre conceived notion of what the degree of influence might be, nor did we have the data set we would like to have had. Nevertheless, the results of this exploration strongly suggest that the influence of the mining metric on frequency and severity of TSF failure is unexpectedly strong.

The First Canonical variant F1 explained 95% of the variability between the two data sets (failures v mining metrics elements). The correlations between F1 and both high severity variables are strong: Very Serious (-0.922); and, Serious (-0.995). The Wilks Lambda on F1 was 0.046 indicating a high degree of certainty that the two data sets (Failures and Mining Metric are not independent of one another). The Eigenvalue for F1, 0.903, suggests a very strong linear relationship between the two data sets (See Appendix 2, Technical Documentation on Canonical Correlation Analysis, for the data set and complete technical documentation on the Canonical Correlation).

Table 4.1 Canonical Correlation Values

	F1
Canonical Correlation	0.950
Eigenvalue	0.903
Wilks' Lambda	0.046
Correlation between:	
Very Serious failures & F1	-0.922
Serious failures & F1	-0.995

Because no other research team that we could find had explored the dimensionality of this relationship, we began with a larger set of mining metric variables beyond the 4 basic variables (Cu Production, Cu Production Cost, Cu Price and Cu Grade), and also attempted to create variables indicating the characteristics of TSF's so that the degree of influence of the mining metric variables could be compared with dam characteristics. We integrated all ICOLD/WISE recorded incidents from 1910 to 2010 into a single reconciled data set, and in the course of our research on consequence of those incidents discovered several compilations that added to WISE/ICOLD, and which also filled in gaps on our main indicators of consequence (total TSF release and release run out). We used both correlation matrix analysis and canonical correlations to find the strongest set of mining metric variables, which turned out to be tons of Cu Ore Production, Cu Production Cost, and Cu Grade. As there was only one recorded Serious failure prior to 1940 and very little information on all incidents, our final data set and analysis focused on the period 1940-2010.

Initially, none of our created synthetic variables for the Mining Metric were as strong the four main variables (copper price, production cost, grade, and copper ore production). One variable, Risk Factor, which combined cost and production volume into a single indicator actually had higher correlations with each of the two most Serious failure categories and also in linear regressions on each of the two highest severity categories. It did not perform as well in lieu of production and cost, though, in a canonical correlation. Further work is needed to evaluate Risk Factor so we are not presenting it here. Within the 4 basic variables price and cost canceled each other out, and cost was the stronger correlation, so the final data set for the Canonical Correlation was only cost, production and grade.

We were not able to develop a meaningful data set on dam characteristics for comparisons of degree of influence as between the variables of the Mining Metric and various dam characteristics (dam height, volume, etc.).

Even though these results are not conclusive, because the number of observations is very small for a CCA, they are persuasive evidence of a greater than expected and very significant influence of Mining Metric mega trends on the frequency and severity of TSF failures. Further, it is important to note that these are not "individual measurements" in the usual sense, but rather aggregations by decade of over 200 observations, and so should be afforded more consideration and weight than would normally attend such a small set of observations. The data set and the full CCA output are at Appendix 2, Technical Documentation on Canonical Correlation Analysis, along with additional technical annotation.

Although further research would be useful to shed more light on how these mega trend variables interact to affect failure, these results in our opinion support a conclusion that financial feasibility of the mine and financial capacity of the miner require greater specific consideration on permit issuance and permit oversight.

Strength of Influence of Copper Ore Production

Among the variables in the Mining Metric data set we were especially interested in the relative degree of influence/connection between copper ore volumes and the TSF failure categories especially whether it could be a reliable denominator for TSF failure rates. The conventional one to one correlations, which are a standard output of CCA in XLSTAT©, showed that both Very Serious and Serious failures were strongly correlated with copper ore production, 0.860 and 0.720 respectively. We had both production and price data on all metals 1900-2010 from the USGS metal statistics (USGS 2014a), but the correlations with aggregate all metals production and the failure variables were not nearly as strong. So the CCA output also lent support to copper ore production as the most reliable and meaningful denominator for TSF failure rates.

Although we did reasonably form an expectation that the mega trends would have a measureable and significant effect on the failure categories established (i.e. that the mega trends contribute to severity), we also know from dam committee reports and other research that many other dam specific elements have a known effect on severity of failure. The final output of a canonical correlation is a set of synthetic variables which maximize the accounting for mutual variability between the two sets of variables. Thus it is an approach which inherently recognizes that all of the information needed to explain the output of interest, the severity of failures over time, are not contained in the analysis, and further that the influence that may exist within in the expected determinant set (the mega trend variables) may result from complex interactions among the determinant data set.

While Canonical Correlation Analysis, and its focus on dimensionality rather than causality, may be the perfect tool for exploring the effect of mega trends of the Mining Metric on the trends in severity of TSF failures, many key variables that would shed more light were not available. We would hope in the future to have a more rich and complete data set, including standing TSFs that didn't fail with the same geographic distribution as those that did.

At present there is no comprehensive compilation of recent or historic tailings dam failures. This is partly understandable given the multi-national nature of the mining industry, but given the severity of the problem, coupled with the fact that it is probably not realistic to think that the problem can be solved without a full analysis of the nature of the problem, it is disappointing that someone has not stepped forward to perform this service.

5.0 FREQUENCIES & PROJECTIONS FROM COPPER PRODUCTION VOLUMES

The results of the correlation analyses give strong support that copper production volumes are a meaningful denominator for TSF failures. Even if there were a centrally professionally maintained inventory of TSFs it would, in our opinion, still be preferable to express TSF failures on the basis of mine production.

Copper metal production is the only reliably managed data element we have available globally that correlates directly with TSF risk potential. The analysis shows us, however, that copper ore production distinguishes more clearly between the two high severity failure categories and is a better descriptor of risk. While it is not routinely and authoritatively compiled and reported as metal production is, the World Bank/Raw Materials Group data (Figure 3.1) did give us an authoritative and reliable historical compilation. As ore production volume is more directly related to TSF waste, in our opinion Cu Ore Production is the better predictor to use. We don't have a global census inventory of standing TSFs. To be meaningful any denominator must be available for all TSFs globally as it is only through data on the global whole that meaningful expectations and comparisons can be made at the level of a nation, province or state.

Secondly, we know there is a great deal of variation in the standing operating TSFs at any point in time. Size and therefore possible maximum consequence of failure varies from small mines with a total capacity of less than 10^5 cubic meters to those over 10^7 cubic meters. Therefore, failure frequency per TSF isn't meaningful without enough attending globally available data to adjust for size and other known risk factors. Post failure it is possible to reexamine the losses more closely, taking account of the specific characteristics of the particular TSF (and eventually to recompile findings if enough new information is developed or if there is more systematic capture of these elements in WISE or other data sources).

Thirdly, we know that the risk profile of TSFs is constantly changing based on production volumes, and how the waste volumes generated from that production are managed. We know that 90% of all TSF failures in Europe (Rico et. al. 2008), to 95% in China (Wei et. al. 2012), occur during operations, as opposed to being in standby or in closure. Cu Ore Production provides an equalized basis for looking across an inventory of TSFs with highly varying size, and it is more directly tied to the phase of active life for the TSFs in which most failures occur (Rico 2008).

Table 5.1, below, shows the failure incidents data for Very Serious failures, Serious failures and Other failures by decade, expressed per million tons of copper ore production. For example, a 0.0020 rate for Other failures in 1940-1949 on 2,545 million tons of ore production describes 1 event. A 0.0006 rate on 16,437 million tons (16.44 billion) of ore production in 1980 describes 10 Other failure events.

Table 5.1 Failures per Million Tonnes Copper Mine Production 1940-2011

Decade	Cu Ore Prod (MMt)	Very Serious failures (#)	Very Serious failures rate	Serious failures (#)	Serious failures rate	Other Failures (#)	Other Failures rate	Other Accidents (#)	Other Accidents rate
1940-49	2,545	1	0.0004	0	0.0000	5	0.0020	0	0.0000
1950-59	3,680	0	0.0000	0	0.0000	7	0.0019	0	0.0000
1960-69	5,004	3	0.0006	4	0.0008	25	0.0050	17	0.0034
1970-79	7,445	4	0.0005	8	0.0011	23	0.0031	15	0.0020
1980-89	10,575	5	0.0005	9	0.0009	22	0.0021	14	0.0013
1990-99	16,437	9	0.0005	9	0.0005	10	0.0006	3	0.0002
2000-09	23,658	7	0.0003	8	0.0003	5	0.0002	1	0.0000
Total/Ave	69,344	29	0.0004	38	0.0005	97	0.0021	50	0.0010

Abbreviations:

Cu Prod = copper ore production in the decade noted in millions of metric tonnes

Very Serious failure = multiple loss of life (~20) and/or release of ≥ 1,000,000 m^3 semi-solids discharge, and/or release travel of 20 km or more.

Serious failure = loss of life and/or release of ≥ 100,000 m^3 semi-solids discharge

Other failures = ICOLD Category 1 failures other than those classified as Very Serious or Serious

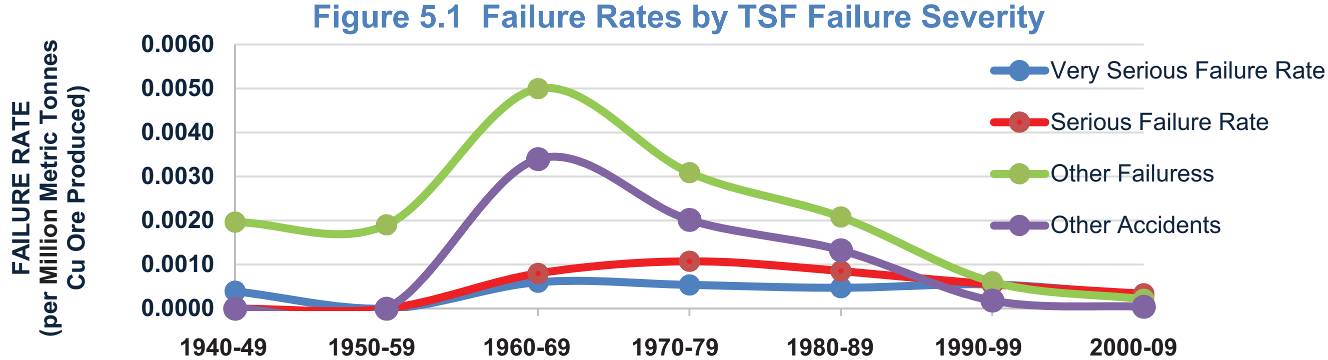
Other Accidents = ICOLD Category 2 accidents other than those classified as Very Serious or Serious

Failure Rate = number of failures per million metric tonnes (MMt) Cu Ore Produced

The overall rate of Very Serious failures and Serious failures 1940-2010 were comparable, 0 00004 and 0.0005 respectively. As expected, the higher the severity the lower the frequency. The frequency rates for all the lower severity loss categories were much lower; 0.021 Other Failures, 0.0010 for Other Accidents.

As shown in Figure 5.1 below the most dramatic change occurred with the shift from predominantly Other Failures (less Serious failure events) to predominantly more Serious failures post 1970. Across the board for each failure category, the rate of failure per ton of copper production has decreased. However, as noted in the introductory section, the severity of failures has steadily increased. More of the failures that occur are Serious or Very Serious). Our data is incomplete (we don't have actual loss data for every Serious and Very Serious failure), however it is certain that that the absolute consequence of all TSF failures has increased and is increasing substantially. This is obvious in that 55% (16/29) of all catastrophic (Very Serious failures) over the past 100 years have occurred since 1990, and that 74% (17/23) of all failure events post-2000 are Serious or Very Serious.

Figure 5.1 Failure Rates by TSF Failure Severity

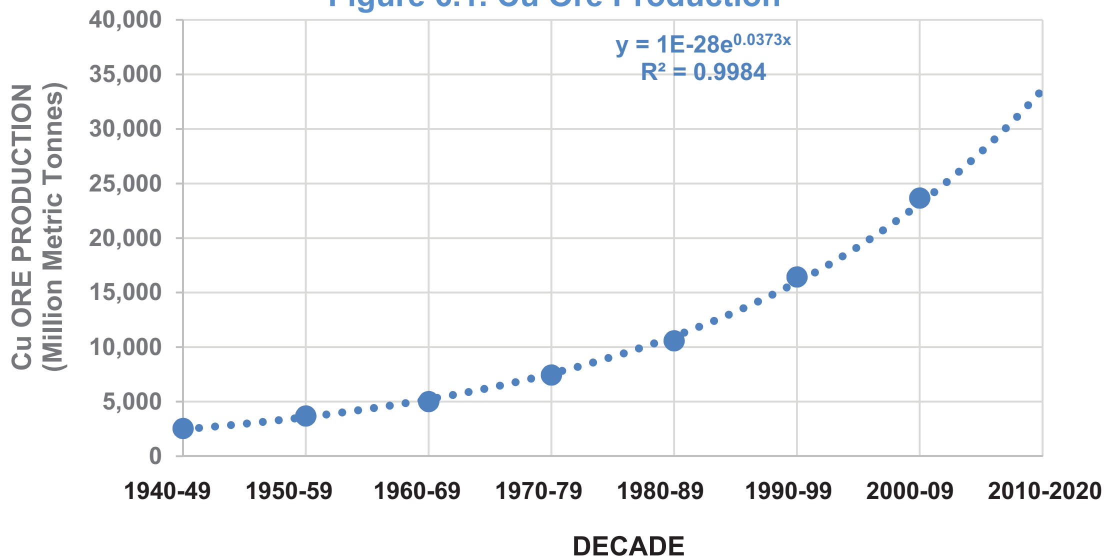


6.0 PROJECTIONS FROM COPPER MINE PRODUCTION V. FAILURE TRENDS

The heart of risk analysis is to reliably measure and forecast expected losses that are beyond control (and to hopefully finance these losses via third party transfers, i.e. insurance or risk pool). We know that will not apply to TSF failure losses, as almost without exception all losses were subject to control and prevention. The basic techniques for forecasting future losses, based on past loss experience, are nevertheless applicable to anticipating the future consequences of continuing the Mining Metric without some new forms of regulatory control and oversight which takes more adequate account of the financial viability of the deposit and the miner.

The Copper ore production estimate for this decade (2010-2019) is advanced from the equation associated with the trend line which had an extremely high R square, 0.9984. The result is 36,338 million metric tonnes, a projected increase of 54%.

Figure 6.1. Cu Ore Production



In insurance rate making the normal procedure for estimating future losses is to combine the last four years of loss data. For this data, though, each cell represents 10 years of experience data not 1, and we can see from analysis of the variables over 100 years that the events that shape loss and failure are unique to each decade, i.e. that each decade has its own pattern of determinant/loss-affecting characteristics.

Table 6.1 below compares three estimates of next decade failures based on three approaches to uses of copper production based frequencies: (1) average of last three decades; (2) last decade only; and, (3) "50-50" weighting between most recent decade and last three decades. The trended values based on failure data alone are presented in Table 6.1 in the last row of the table.

The chart values in Table 6.1 are computed from the trend line equations as they appear in Figure 6.2 (The trend lines in Figure 6.2 are linear data projections, rounded to the nearest whole number).

Very Serious failures 2020 = 0.1393*2020-271.64 = 9.746

Serious failures = 0.1643*2020-3189.6 = 12.026

Table 6.1 Predictions 2010-2020 From Historic Failure Rates

Basis	Very Serious failures		Serious failures		Other Failures		Other Accidents	
	Rate	Pred.	Rate	Pred.	Rate	Pred.	Rate	Pred.
Last 3 Decade Ave	0.0004	15.9	0.0006	21.0	0.0010	35.1	0.0005	18.8
Last Decade	0.0003	10.8	0.0003	12.3	0.0002	7.7	0.0000	1.5
50-50 Weighting	0.0004	13.3	0.0005	16.7	0.0006	21.4	0.0003	10.1
Chart		9.5		12.0				

Rate = number of failures per million metric tonnes (MMt) ore mined

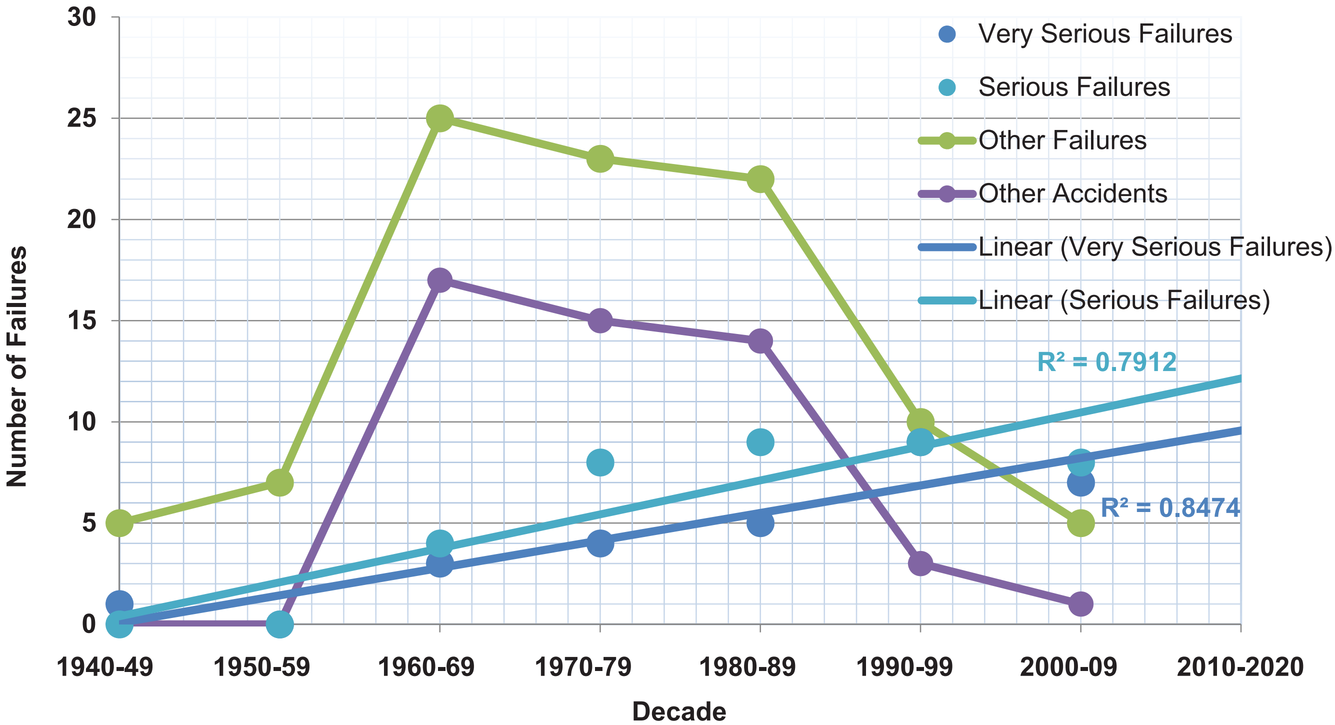
Pred = number of predicted failures in the period 2010 - 2019

The high R-squared values on the trend lines for both Serious failures and Very Serious failures indicate a "goodness of fit" that is apparent on visual inspection alone (i.e. the markers closely track the trend line). The calculated predictions by chart trend line equation most closely matches the prediction based on the most recent decade failure rates.

The canonical correlation demonstrates that the trends in the high severity failures are shaped by the entire metric (as represented in grade, cost and production). Inspection of the data set shows that the main elements of the metric as of 2009 were very different than those of either of the prior two decades. It is not likely costs will return to as low as $15 or that prices will fall to as low as they were in either of the two most recent decades. Therefore we have greater confidence in the most recent failure rate by class than we do in the either the average of the last three decades, or a 50-50 weighting between the average of the last three decades and current decade. Still there are already clear indications that this decade involves uncertainty about the direction of cost to produce, price, and perhaps even production volumes. The previous two decades both had constant costs of production against failing prices, a very different pattern with an expected higher rate of failure. Mid-decade 2010-2019 the overall environment seems to be trending toward higher financial risk, and therefore higher potential environmental liability than the 2000-2009 decade.

We are though projecting 12 Serious failures and 11 Very Serious failures for the present decade (2010–2019) relying on the failure rates of the most current decade (see Table 6.1).

Figure 6.1 Failure Predictions By Trend Line



Our dataset included 5 failures 1910-2010 that met our criteria for Very Serious that were not listed in WISE or ICOLD data bases, from a compilation of Chinese major failures and a compilation of Philippine significant tailings incidents. The frequency rate 2000-2009 was essentially the same with or without these five failures. We cannot say that whatever undercount actually exists in WISE/ICOLD data would have no bearing, however, in our view this is a conservative projection quite apart from the possible undercount issue. It makes no allowances for the possibly higher risks of price jitters on many metals (e.g. molybdenum, iron, zinc, gold), of rising production costs mostly from energy and foreign exchange rates, and the uncertainty about the roles China, Chile, and Peru (as producers, and China and India (as consumers) will play, and how that could elevate financial risks for smaller mines and smaller miners.

7.0 Projected Cost of Remediation and Non Remediable Uncontrolled releases from TSFs

We searched the historic record for what local authorities had deemed the costs of public damages from the major releases in our database, and found sufficient authoritative documentation on a total of 6 of the 14 post-1990 Very Serious uncontrolled TSF originating release incidents. Our process was to translate from foreign currency to US in the year of the incident and then to convert those $US to 2014-$US. The average cost of the 7 incidents for which we found authoritative data was $543 million (Figure 7.1). That translates to a projected public liability for remediation of 11 Very Serious releases from TSFs at cost of approximately $5 billion globally before the end of this decade (2020). We did not attempt any estimates for the expected 12 Serious failures by 2020 but a guess of an additional $1 billion is probably not unreasonable.

Usually losses are forecast from a record of homogeneous data maintained by one source over time by the entity which has actually incurred or paid out those losses (i.e. an insurer or a rating bureau like the Insurance Services Office), or a company's or agency's risk manager. That is not true of our loss history data for TSF failures. Although WISE has followed with some detail on a few cases involving litigation for recovery of outlays (e.g. for Los Frailes), descriptions of consequence are brief and narrative. There are few links to more in-depth authoritative analysis on consequence. Losses are not systematically or uniformly captured or developed as part of either the WISE or ICOLD databases. The costs data we present here is all we could find for Very Serious post-1990 failures which pertained to environmental losses, and which were cited or developed by authoritative or credible sources.

We aimed for as much homogeneity as possible in choosing amounts documented for inclusion in our loss history (i.e. to include only natural resources/environmental losses whether or not cleanup was ordered or undertaken. In one case, Omai, we used a token amount to acknowledge what farmers, fisherman, and NGOs attempted to recover, and to acknowledge what is widely agreed was environmental damage notwithstanding the governments judgments to the contrary. The token amount allocated to Omai actually lowers the overall average cost estimate but, given all the litigation and controversy that has attended, simply admitting to the extent of environmental damage we felt Omai could not simply be left off the list, even though we could not find documentation on what part of $2 billion joint damage claim was attributable to documented environmental damages to lands and waters.

While sketchily sourced and documented, the few failures which are systematically and authoritatively developed give us a high level of confidence that our average natural resource loss of $543 million for a catastrophic failure is not overstated. For example, the estimated costs to clean up the Los Frailes spill was borne primarily by the Andalusian Government as a non-remediable loss. We think that situations like this, where the actual costs are so high or cleanup costs so astronomical that losses from Very Serious TSF failures will more and more be permanent non-recoverable losses. Mt Polley is a possible example of a tailings spill into a creek and lake that will not be retrieved. Such losses will, hopefully, still have a complete accounting of value whether or not remediation is ordered, undertaken, or possible.

The data on the 7 failures forming the basis of our average loss amount of $543 million and its sources are presented in Table 7.1, below. See Appendix 3 for more detail on this chart.

Apply this to our projections of the number of Very Serious failures, 11 results in a projected unfunded unfundable public liability loss of $6.0 billion from Very Serious TSF failures for the decade 2010-2019.

Our sense of the data and case histories is that this decades' TSF failures will continue to arise mostly from standing operating TSFs, pushing older TSFs up to and past their original designs, or stretching the limits of TSFs that were not built or managed to best practices in the first place. We expect most to arise from smaller mines and miners. We see in the record an indication that in many instances releases and events suggesting fundamental problems with the structure of the TSF preceded a final catastrophe by two to four years. In the cases of Golden Cross (New Zealand), Bingham Canyon (Utah), and Mike Horse (Montana) long term issues with dam stability led to closures in time to avert catastrophe at costs that were significantly lower than the remediation costs or assessed damages would have been for a structural failure.

Table 7.1 Documented TSF Very Serious Natural Resource Losses 1990 – 2010

TSF Failure	Year	Original Currency (Millions)	Failure Year M US$	2014 M US$	Ore	Release (M m^3)	Run Out (km)	Deaths
Kingston Fossil Plant, Harriman, Tennessee, USA	2008	US 1,200	$1,200	$1,300		5.4	4.1	
Taoshi, Linfen City, Xiangfen, Shanxi Province, China	2008	US 1,300	$1,300	$1,429	Fe	0.19	2.5	277
Baia Mare, Romania	2000	US 179	$179	$246	Au	0.1	5.2	
Los Frailes, Spain	1998	EU 275	$301	$437	Zn/Cu /Pb	4.6	5	
Marinduque Island, Philippines	1996	P 180 + US 114	$123	$185	Cu	1.6	27	
Omai, Guyana	1995	US 100	$100	$156	Au	4.2	80	
Merriespruit, South Africa	1994	R 100	$29	$46	Au	0.6	2	17
				======				
		Average US$2014: $543		$3,799				

Reviewing their own role in creating and perpetuating the environment in which we have allowed TSFs at risk of consequential failure to proliferate, the International Bank for Reconstruction & Development and the International Development Association put it well:

> *"Governance should be strengthened until it is able to withstand the risks of developing major extractions. Once that has happened, the International Bank for Reconstruction and Development (IBRD) and the International Development Association (IDA) can add support for the promotion of a well-governed extractive sector. Similarly, when the International Finance Corporation and the Multilateral Investment Guarantee Agency (MIGA) consider investing in an oil, gas, or mining project, they need to specifically assess the governance adequacy of the country as well as the anticipated impacts of the project and then only support projects when a country's government is prepared and able to withstand the inherent social, environmental, and governance challenges." (IFC 2003)*

Our study has provided a very conservative estimate of future unfunded public liabilities for standing, already operating, and permitted TSFs globally. We know globally that every one of those failures can be prevented for a cost much less than $6.0 billion for just the 11 Very Serious failures we are predicting by 2020.

We know globally, and in Canada and the US, the regulatory structure is not presently in place to identify and correct these at-risk TSFs before they fail, and we know many of them are operated by companies whose balance sheets are too thin to fund repairs and closure where necessary.

We hope our work will begin a collaborative and highly focused multi-disciplinary dialogue to prevent the materialization of these $6.0 billion in public losses by 2020.

8.0 SUMMARY & CONCLUSIONS

The advances in mining technology over the past 100 years which have made it economically feasible to mine lower grades of ore against a century of declining prices have not been counterbalanced with advances in economically efficient means of managing the exponentially expanding volume of associated environmental liabilities in waste rock, tailings and waste waters. In fact those new technologies which do offer better management of mine wastes usually add significant cost and are often detrimental to bottom line financial feasibility. This is evidenced in a post-1990 trend toward un-fundable environmental losses of greater consequence. This interdisciplinary review of TSF failures 1910-2010 establishes a clear and irrefutable relationship between the mega trends that squeeze cash flows for all miners at all locations, and this indisputably clear trend toward failures of ever greater environmental consequence.

The implication of our findings is that a continuation of the present Mining Metric is not environmentally or economically sustainable, and that regulatory systems must begin to understand and address financial capacity of the miner, and the financial feasibility of mining itself, both in permitting criteria and in oversight of mine water management over the life of the mine.

Our findings point toward undocumented and unstudied risks of failure in the standing operating already permitted mines of smaller miners globally where cash flow pressures have led to an avoidance of best practices in waste management, and where political pressures have led to avoided close scrutiny of decades of neglect and shortfalls.

We have not identified an existing statutory or regulatory system anywhere that has the authority and capacity to identify and prevent the $6 billion in losses we estimate the public globally will be liable for by the end of this decade.

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APPENDIX 1

TSF Failure Data Table

TSF FAILURE DATA TABLE LEGEND

Category	Count	Definition
Very Serious	30	Very Serious = multiple loss of life (~20) and/or release of ≥ 1,000,000 m3 semi-solids discharge, and/or release travel of 20 km or more
Serious	38	Serious = loss of life and/or release of ≥ 100,000 m3 semi-solids discharge
Other Failures	98	Other Failures = ICOLD Category 1 failures other than those classified as Very Serious or Serious
Other Accidents	50	Other Accidents = ICOLD Category 2 accidents other than those classified as Very Serious or Serious
Non-Dam Failure	10	Non-Dam Failures = groundwater, waste rock, etc.
	=======	
Total	226	

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DAM TYPE Key		DAM FILL MATERIAL Key		INCIDENT TYPE Key			INCIDENT CAUSE Key	
US	Upstream	**T**	Tailings	**1A**	Failure	Active Impoundment	**SI**	Slope instability
DS	Downstream	**CST**	Cycloned sand ta	**1B**	Failure	Inactive Impoundment	**SE**	Seepage
CL	Centerline	**MW**	Mine waste	**2A**	Accident	Active Impoundment	**FN**	Foundation
WR	Water retention	**E**	Earthfill	**2B**	Accident	Inactive Impoundment	**OT**	Overtopping
NR	Not reported	**R**	Rockfill	**3**	Groundwater		**ST**	Structural
							EQ	Earthquake
							MS	Mine subsidence
							ER	Erosion
							U	Unknown, or
							NR	Not Reported

GENERAL NOTE

We found small variations source to source on total release, run out, deaths and other details, but we found no ambiguities or inconsistencies that precluded a clear classification as "Serious" or "Very Serious".

Overall we found much more detailed accounts of "consequence" in local compilations or regional or national studies. WISE & ICOLD occasionally including details on consequence, or linked to sources detailing consequence. Our bibliogtaphy includes a more extensive list of materials related to the consequence of TSF failures

COLOR CODE	MINE/PROJECT & LOCATION	DAM TYPE	DAM FILL MATERIAL	DAM HEIGHT (meters)	STORAGE VOLUME (cu. meters)	ICOLD TYPE	INCIDENT DATE	RELEASE VOLUME (cu. meters)	RUNOUT (km)	DEATHS	Source Color Code	SOURCES	NOTES
	Karamken, Magadan Region, Russia					1A	29-Aug-09	1,200,000		1		WISE, MACE	11 houses lost, 1 death (Karamken Update - MACE 2012-02-10)
	Huayuan County, Xiangxi Autonomous Prefecture, Hunan Province, China					1A	14-May-09	50,000		3		WISE	3 killed, 4 injured
	Kingston fossil plant, Harriman, Tennessee, USA					1A	22-Dec-08	5,400,000	4.1			WISE	5.4 million cubic yards (1.09 billion gallons) of fly ash was released (http://www.sourcewatch.org/index.php?title=TVA_Kingston_Fossil_Plant_coal_ash_spill#TVA_Reaction)
	Taoshi, Linfen City, Xiangfen county, Shanxi province, China	US		50.7	290,000	1A	8-Sep-08	190,000	2.5	277		WISE	At least 254 dead and 35 injured.
	Glebe Mines, UK		E			1B-OT	22-Jan-07	20,000				HSE Report	Initial Report of the HSE investigation into the Glebe Mines Stony Middleton dam failure 2007, HSE Central Division - Nottingham, UK, 23Feb07
	Miliang, Zhen'an County, Shangluo, Shaanxi Province, China					1A	30-Apr-06		5	17		WISE	17 missing
	Pinchi Lake, BC, Canada	WR	E	12		2A-ER	30-Nov-04	6,000-8,000				WISE	Mercury contaminated tailings into Pinchi Lake
	Riverview, Florida, USA					1A	5-Sep-04	227,000				WISE	
	Partizansk, Primorski Krai, Russia					1A	22-May-04	166,000				WISE	
	Malvési, Aude, France					1A	20-Mar-04	30,000				WISE	Uranium slurries elevated nitrate in river
	Cerro Negro, near Santiago, Chile, (5 of 5)	US	T			1A-ER	3-Oct-03	80,000	20			WISE	
	El Cobre, Chile, 2, 3, 4, 5	US	T			1B-OT	22-Sep-02	8,000				Villavicencio (2014)	
	San Marcelino Zambales, Philippines, Bayarong dam (9/11/02)				47,000,000	1B	11-Sep-02					WISE, Piplinks	Sep. 11: low lying villages flooded with mine waste; 250 families evacuated;
	San Marcelino Zambales, Philippines, Camalca dam (8/27/02)					1B	27-Aug-02					WISE, Piplinks	Aug. 27: some tailings spilled into Mapanuepe Lake and eventually into the St. Tomas River.
	El Cobre, Chile	US	T			1B-OT	11-Aug-02	4,500				Villavicencio (2014)	
	Sebastião das Águas Claras, Nova Lima district, Minas Gerais, Brazil					1A	22-Jun-01		8	2		WISE	2 killed, 3 missing. Tailings 8 km downstream the Córrego Taquaras stream, mud affected an area of 30 hectares
	Nandan Tin mine, Dachang, Guangxi					1A	18-Oct-00			28		WISE, Wei	WISE:15 killed, 100 missing, 100 houses destroyed

COLOR CODE	MINE/PROJECT & LOCATION	DAM TYPE	DAM FILL MATERIAL	DAM HEIGHT (meters)	STORAGE VOLUME (cu. meters)	ICOLD TYPE	INCIDENT DATE	RELEASE VOLUME (cu. meters)	RUNOUT (km)	DEATHS	Source Color Code	SOURCES	NOTES
	Inez, Martin County, Kentucky, USA					1A	11-Oct-00	950,000	120		Table 1	ICOLD, WISE	
	Aitik mine, near Gällivare, Sweden	DS	MW & E	15	15,000,000	1A-ER	8-Sep-00	1,800,000	5.2		Table 1	ICOLD, WISE	
	Baia Mare, Romania Esmerelda Exploration	DS then US	T	A few m	800,000	1A-ST	30-Jan-00	100,000	>100		221	ICOLD, WISE, Rico	Killed tonnes of fish and poisoned drinking water of more than 2 million people in Hungary
	Borsa, Romania					1A	2000	22,000t			Table 1	ICOLD, WISE	Company: Remin SA
	Surigao Del Norte Placer, Philippines (#3 of 3)					1A	26-Apr-99	700,000 t	12	4	Table 1	ICOLD, Piplinks	
	Toledo City (Philippines)					1B	1999	5,700,000				Piplinks	Drainage tunnel blowout
	Huelva, Spain					1A	31-Dec-98				Table 1	ICOLD, WISE	Fertiberia phosphate mine
	Los Frailes, near Seville, Spain	WR	R	27	15,000,000	1A-FN	25-Apr-98	6,800,000	41		209	ICOLD, WISE, Rico	
	Zamboanga Del Norte, Sibutad Gold Project					1A-OT	6-Nov-97					Piplinks	
	Pinto Valley, Arizona, USA					1B	22-Oct-97	230,000			Table 1	ICOLD, WISE	
	Amatista, Nazca, Peru					1A-EQ	12-Nov-96	300,000				WISE	due to M6.4 earthquake
	El Porco, Bolivia					1A	29-Aug-96	400,000	300		Table 1	ICOLD, WISE	300 km of Pilcomayo river contaminated
	Marcopper, Marinduque Island, Philippines(3/24) (#2 of 2)					1A-ST	24-Mar-96	1,600,000	26		208	ICOLD, WISE, Piplinks	Drainage tunnel plug failed. 26 km of the Makulaquit and Boac river systems filled with tailings rendering them unusable; US$ 80 million in damage
	Sgurigrad, Bulgaria	US	T	45	1,520,000	1A-SI	1996	220,000	6		220	ICOLD, Rico	
	Negros Occidental, Bulawan Mine Sipalay River					1A	8-Dec-95					Piplinks	
	Golden Cross, Waitekauri Valley, New Zealand		R	25-30	3,000,000	1A-FN	Dec-95	9,999			207	ICOLD	
	Surigao del Norte Placer, Philippines (#2 of 3)	WR	E	17		1B-SI	2-Sep-95	50,000		12	206	ICOLD, WISE	
	Omai Mine, Tailings dam No 1, 2, Guyana	WR	R	44	5,250,000	1A-ER	19-Aug-95	4,200,000	80		205	ICOLD, WISE, Rico	80 km of Essequibo River declared environmental disaster zone
	Middle Arm, Launceston, Tasmania	CL	E	4	25,000	1A-OT	25-Jun-95	5,000			204	ICOLD	
	Riltec, Mathinna, Tasmania	CL	E	7	120,000	2A-SE	Jun-95	40,000			203	ICOLD	
	Hopewell Mine, Hillsborough County, Florida, USA					1A	19-Nov-94	1,900,000				WISE	

COLOR CODE	MINE/PROJECT & LOCATION	DAM TYPE	DAM FILL MATERIAL	DAM HEIGHT (meters)	STORAGE VOLUME (cu. meters)	ICOLD TYPE	INCIDENT DATE	RELEASE VOLUME (cu. meters)	RUNOUT (km)	DEATHS	Source Color Code	SOURCES	NOTES
	Payne Creek Mine, Polk County, Florida, USA					1A	2-Oct-94	6,800,000				WISE	
	Merriespruit, near Virginia, South Africa, Harmony 2, 3	US paddock	T	31	7,040,000	1B-OT	22-Feb-94	600,000	4	17	202	ICOLD, WISE, Rico	
	Olympic Dam, Roxby Downs, South Australia					3	14-Feb-94	5,000,000				WISE	Designed groundwater leakage from unlined tailings impoundment into groundwater
	Minera Sera Grande: Crixas, Goias, Brazil	DS then US	CST	41	2.25Mt	2A-SI	Feb-94	None			214	ICOLD	
	Fort Meade, Florida, Cargill phosphate (#3 of 3)					1A	2-Jan-94	76,000				WISE	
	Longjiaoshan, Daye Iron Ore mine, Hubei					1A	1994			31		Wei	
	Marcopper, Marinduque Island, Mogpog Philippines(12/6) (#1 of 2)					1B	6-Dec-93			2		Piplinks	Siltation dam failure. Mogpog River and Mogpog town flooded.
	TD 7, Chingola, Zambia	US	T&E	5		1A-OT	Aug-93	100 t			200	ICOLD	
	Itogon-Suyoc, Baguio gold district, Luzon, Philippines					1A-OT	26-Jun-93				199	ICOLD, Piplinks	
	Marsa, Peru					1A-OT	Jan-93			6		WISE	
	Kojkovac, Montenegro	WR	E		3,500,000	2B-ER	Nov-92	none			198	ICOLD	
	Saaiplaas, South Africa, 2		CST			1A-IS	19-Mar-92				Table 1	ICOLD	3 separate events within 4 days
	Maritsa Istok 1, Bulgaria		Ash	15	52,000,000	1A-ER	1-Mar-92	500,000			218	ICOLD, WISE	
	Tubu, Benguet, No.2 Tailings Pond, Padcal, Luzon, Philippines				80,000,000	1A-FN	2-Jan-92	80,000,000			197	Piplinks	
	Iron Dyke, Sullivan Mine, Kimberley, BC, Canada	US		21		1A-SI	23-Aug-91	75,000			196	ICOLD	
	Soda Lake, California, USA	US	E	3		2A-EQ	17-Oct-89				111	ICOLD	
	Silver King, Idaho, USA	DS	E	9	37,000	2A-OT	5-Aug-89	Small			108	ICOLD	
	Big Four, Florida, USA	CL	E			2A-FN	1989				14	ICOLD	
	Cyprus Thompson Creek, Idaho, USA	CL	CST	146	27,000,000	2A-SE	1989				34	ICOLD	

COLOR CODE	MINE/PROJECT & LOCATION	DAM TYPE	DAM FILL MATERIAL	DAM HEIGHT (meters)	STORAGE VOLUME (cu. meters)	ICOLD TYPE	INCIDENT DATE	RELEASE VOLUME (cu. meters)	RUNOUT (km)	DEATHS	Source Color Code	SOURCES	NOTES
	Southern Clay, Tennessee, USA	WR	E	5		1A-SE	1989	300			112	ICOLD	
	Stancil , Maryland, USA	US	E	9	74,000	1A-SI	1989	38,000	0.1		116	ICOLD, Rico	
	Unidentified, Hernando, County, Florida, USA #2	US	E	12	3,300,000	1A-OT	Sep-88	4,600			163	ICOLD	
	Jinduicheng, Shaanxi Province., China	US		40		1A-OT	30-Apr-88	700,000		~20	195	ICOLD, WISE	
	Consolidated Coal No.1, Tennessee, USA,	DS	MW	85	1,000,000	2A-ST	19-Jan-88	250,000			121	ICOLD, WISE	
	Rain Starter Dam, Elko, Nevada, USA	WR	ER	27	1,500,000	3-	1988				98	ICOLD	
	Unidentified, Hernando, County, Florida, USA	DS	E	12		2A-FN	1988				164	ICOLD	
	Surigao Del Norte Placer, Philippines (#1 of 3)					1A	9-Jul-87					Piplinks	
	Montcoal No.7, Raleigh County, West Virginia, USA					1A	8-Apr-87	87,000	80			WISE	tailings flow 80 km downstream
	Bekovsky, Western Siberia	US	Argillite, aleurolite	53	52,000,000	1A	25-Mar-87	None			212	ICOLD	
	Xishimen, China	US	T	31		1A-SI	21-Mar-87	2,230			194	ICOLD	
	Montana Tunnels, MT, USA	DS	MW	33	250,000	3-	1987				87	ICOLD	
	Marianna Mine #58, PA,	US	E	37	300,000	2A-SI	19-Nov-86				77	ICOLD	
	Mankayan, Luzon, Philippines, No.3 Tailings Pond		E			1A-ST	17-Oct-86				193	ICOLD, Piplinks	Siltation of the Abra River which affected 9 municipalities
	Lepanto, Mankayan, Benguet, Philippines					1A	17-Oct-86					Piplinks	Siltation of the Abra River which affected 9 municipalities
	Pico de Sao Luis, Gerais, Brazil		T	20		1A-ER	2-Oct-86				192	ICOLD	
	Rossarden, Tasmania	WR	E	7.5	200,000	1B-OT	16-May-86				190	ICOLD	
	Story's Creek, Tasmania	Valley side		17	30,000	1B-OT	16-May-86	Minimal			191	ICOLD	
	Itabirito, Minas Gerais, Brazil	Gravity	Masonry	30		1A-ST	May-86	100,000	12	7	189	ICOLD, WISE, Rico	
	Mineral King, BC, Canada	CL	CST	6	Small	1B-OT	20-Mar-86				188	ICOLD	
	Huangmeishan, China					1A	1986			19		WISE	

COLOR CODE	MINE/PROJECT & LOCATION	DAM TYPE	DAM FILL MATERIAL	DAM HEIGHT (meters)	STORAGE VOLUME (cu. meters)	ICOLD TYPE	INCIDENT DATE	RELEASE VOLUME (cu. meters)	RUNOUT (km)	DEATHS	Source Color Code	SOURCES	NOTES
	Spring Creek Plant, Borger, Texas, USA			5	30,000	1A-OT	1986				114	ICOLD	
	Bonsal, North Carolina, USA	WR	E	6	38,000	1A-OT	17-Aug-86	11,000			17	ICOLD	
	Stava, North Italy, 2, 3	US	CST	29.5	300,000	1A-SI	19-Jul-85	200,000	8	269	117	ICOLD, WISE, Rico	
	La Belle, Pennsylvania, USA	DS	MW	79	1,230,000	2A-FN	17-Jul-85				68	ICOLD	
	Cerro Negro No. (4 of 5)	US	CST	40	2,000,000	1A-EQ	3-Mar-85	500,000	8		30	ICOLD WISE, Rico	
	Veta de Agua	US	T	24	700,000	1A-EQ	3-Mar-85	280,000	5		178	ICOLD, WISE, Rico	
	El Cobre No. 4	DS	CST	50		2A-EQ	3-Mar-85				44	ICOLD	
	Niujiaolong, Shizhuyuan Non-ferrous Metals Co., Hunan					1A	Jan-85	731,000	4.2	49		Wei	
	Marga, Chile					1B-OT	1985				76	ICOLD	
	Ollinghouse, Nevada, USA	WR	E	5	120,000	1A-SE	1985	25,000	1.5		91	ICOLD, Rico	
	Texasgulf 4B Pond, Beaufort, Co., North Carolina, USA	WR	T	8	12,300,000	2A-SI	Apr-84				122	ICOLD	
	Mirolubovka, Southern Ukraine	US	E&T	32	80,000,000	2A-SI	15-Jan-84	-			210	ICOLD	
	Battle Mt. Gold, Nevada,	DS	E	8	1,540,000	2A-SI	1984				11	ICOLD	
	Virginia Vermiculite, Louisa County, Virginia, USA	WR	E	9		1A-ST	1984				179	ICOLD	
	Clayton Mine, Idaho, USA	CL	T	24	215,000	2A-ST	2-Jun-83				32	ICOLD	
	Golden Sunlight, MT, USA	CL	CST			3-	5-Jan-83				51	ICOLD	
	Grey Eagle, California, USA	DS	E			3-	1983				53	ICOLD	
	Vallenar 1 and 2					1B-OT	1983				175	ICOLD	
	Sipalay, Philippines, No.3 Tailings Pond	WR	MW		37,000,000	1A-FN	8-Nov-82	28,000,000			187	ICOLD, WISE, Piplinks	Dam failure, due to slippage of foundations on clayey soils. Widespread inundation of agricultural land up to 1.5 m high
	Royster, Florida, USA	US	T	21		1A-FN	1982				102	ICOLD	
	Ages, Harlan County, Kentucky, USA					1A	18-Dec-81	96,000	163	1		WISE	
	Dixie Mine, Colorado, USA					1B-U	Apr-81				39	ICOLD	

COLOR CODE	MINE/PROJECT & LOCATION	DAM TYPE	DAM FILL MATERIAL	DAM HEIGHT (meters)	STORAGE VOLUME (cu. meters)	ICOLD TYPE	INCIDENT DATE	RELEASE VOLUME (cu. meters)	RUNOUT (km)	DEATHS	Source Color Code	SOURCES	NOTES
	Balka Chuficheva, Russia	US	CST	25	27,000,000	1A-SI	20-Jan-81	3,500,000	1.3		211	ICOLD, WISE	
	Texasgulf No. 1 Pond, Beaufort Co., North Carolina, USA	WR	E		24,700,000	2A-SI	1981				123	ICOLD	
	Veta de Aqua A					1A-EQ	1981				176	ICOLD	
	Veta de Agua B					1A-EQ	1981				177	ICOLD	
	Tyrone, New Mexico, Phelps-Dodge	US	CST	66	2,500,000	1A-SI	13-Oct-80	2,000,000	8		94	ICOLD, WISE, Rico	
	Sweeney Tailings Dam, Longmont, Colorado, USA			7		1A-SE	May-80				119	ICOLD	
	Kyanite Mining, Virginia, USA			11	430,000	2A-OT	1980				67	ICOLD	
	Churchrock, New Mexico, United Nuclear	WR	E	11	370,000	1A-FN	16-Jul-79	370,000	110		173	ICOLD, Wikipedia, Rico	
	Union Carbide, Uravan, Colorado, USA	US	T	43		2A-SI	Mar-79				172	ICOLD	
	Incident No. 1, Elliot, Ontario, Canada	WR	E	9		3-	1979				35	ICOLD	
	Suncor E-W Dike, Alberta, Canada	WR	MW	30		2A-SI	1979				118	ICOLD	
	Arcturus, Zimbabwe	US	T	25	1.7-2.0 Mt	1A-OT	31-Jan-78	39,000	0.3	1	185	ICOLD, WISE, Rico	
	Mochikoshi No. 1, Japan (1 of 2)	US	T	28	480,000	1A-EQ	14-Jan-78	80,000	8	1	84	ICOLD, WISE, Rico	Dam failure due to earthquake
	Norosawa, Japan	DS		24	225,000	2B-EQ	14-Jan-78				90	ICOLD	
	Hirayama, Japan	DS		9	87,000	2B-EQ	1978				56	ICOLD	
	Mochikoshi No. 2, Japan (2 of 2)	US	T	19		1A-EQ	1978	3,000	0.15		85	ICOLD, Rico	dam failure due to aftershock
	Syncrude, Alberta, Canada	CL	T			2A-FN	1978				120	ICOLD	
	Madison, Missouri, USA	WR	E	11		1A-OT	28-Feb-77				74	ICOLD	
	Homestake, N. Mexico, USA	US	T	21		1A-ST	Feb-77	30,000			59	ICOLD	
	Pit No. 2, Western	US	T	9		1A-SI	1977				96	ICOLD	

COLOR CODE	MINE/PROJECT & LOCATION	DAM TYPE	DAM FILL MATERIAL	DAM HEIGHT (meters)	STORAGE VOLUME (cu. meters)	ICOLD TYPE	INCIDENT DATE	RELEASE VOLUME (cu. meters)	RUNOUT (km)	DEATHS	Source Color Code	SOURCES	NOTES
	Unidentified, Hernando, County, Florida, USA	CL	E	6		2A-FN	1977				162	ICOLD	
	Western Nuclear, Jeffrey City, Wyoming, USA #2					1A-SI	1977	40			180	ICOLD	
	Kerr-McGee, Churchrock, New Mexico, USA	WR	E	9		1A-FN	Apr-76				64	ICOLD	
	Zlevoto No. 4, Yugoslavia	US	T	25	1,000,000	1A-SI	Mar-76	300,000			184	ICOLD, WISE	
	Dashihe. China	US		37		2A-EQ	1976				36	ICOLD	
	Unidentified, Idaho, USA	DS	E	34		2A-SI	1976				149	ICOLD	
	Cadet No. 2, Montana,	CL	E	21		2A-SI	Sep-75				18	ICOLD	
	Madjarevo, Bulgaria	US	T	40	3,000,000	1A-ST	1-Apr-75	250,000			219	ICOLD	
	Carr Fork, Utah, USA			10		1A-ST	Feb-75				22	ICOLD	
	Dresser No. 4, Montana,	CL	E	15		1A-FN	1975				40	ICOLD	
	Keystone Mine, Crested Butte, Colorado, USA					1B-U	1975				65	ICOLD	
	Mike Horse, Montana, USA	US	T	18	750,000	1B-OT	1975	150,000			79	ICOLD	
	PCS Rocanville, Saskatchewan, Canada	US	T	12		3-	1975				92	ICOLD	
	Unidentified, Green River, Wyoming, USA	WR	E	18		3-	1975				161	ICOLD	
	Heath Steele main dam, Brunswick, Canada	WR	R,E	30		2A-FN	1975				186	ICOLD	
	Bafokeng, South Africa	US	T	20	13,000,000	1A-SE	11-Nov-74	3,000,000	45	12	7	ICOLD, WISE, Rico	
	Golden Gilpin Mine, Colorado, USA			12		1B-U	Nov-74				50	ICOLD	
	Deneen Mica Yancey County, North Carolina, USA	US	CST	18	300,000	1A-SI	Jun-74	38,000	0.03		37	ICOLD	
	Silver King, Idaho, USA	DS	E	9	37,000	1A-OT	16-Jan-74	6,000			109	ICOLD	
	Galena Mine, Idaho, USA #2	US	MW	9		1A-OT	15-Jan-74	3,800	0.61		49	ICOLD, Rico	
	Berrien, France	US	R	9		1A-SE	1974				10	ICOLD	
	GCOS, Alberta, Canada	US	T	61		2A-SI	1974				47	ICOLD	

COLOR CODE	MINE/PROJECT & LOCATION	DAM TYPE	DAM FILL MATERIAL	DAM HEIGHT (meters)	STORAGE VOLUME (cu. meters)	ICOLD TYPE	INCIDENT DATE	RELEASE VOLUME (cu. meters)	RUNOUT (km)	DEATHS	Source Color Code	SOURCES	NOTES
	Unidentified, Mississippi, USA #2	US	T	20		2A-FN	1974				153	ICOLD	
	Unidentified, Canaca, Mexico	US	T	46		1A-OT	1974				159	ICOLD	
	Ray Mine, Arizona, USA inc #2	US	T	52		2A-SI	5-Feb-73				101	ICOLD	
	(unidentified), Southwestern USA	US	E	43	500,000	1A-SI	1973	170,000	25		169	ICOLD, WISE, Rico	noted as "Southwestern US" in WISE
	Earth Resources, N M,	US	T	21		1A-OT	1973				41	ICOLD	
	Ray Mine, Arizona, USA	US	T	52		1A-SI	2-Dec-72				100	ICOLD	
	Buffalo Creek, West Virginia, USA					1A	26-Feb-72	500,000	64.4	125	Table 1	ICOLD, WISE, Rico	Tailings traveled 27 km downstream, 125 people lost their lives, 500 homes were destroyed. Property and highway damage exceeded $65 million
	Galena Mine, Idaho, USA	US	E	14		2A-ER	1972				48	ICOLD	
	Cities Service, Fort Meade, Florida, phosphate					1A	3-Dec-71	9,000,000	120		31	WISE, Rico	
	Pinchi Lake, BC, Canada	WR	E	13		2A-ER	1971				95	ICOLD	
	Western Nuclear, Jeffrey City, Wyoming, USA					1A-ST	1971				181	ICOLD	
	Mufulira, Zambia			50	1,000,000	1A-MS	Sep-70	68,000		89	88	ICOLD, WISE	Saturated slime tailings deposited in a TSF #3 over subsidence feature flowed into an underground mine killing 89 miners.
	Maggie Pye, United Kingdom, clay	US	T	18		1A-SI	1970	15,000			75	ICOLD, WISE	
	Park, United Kingdom	WR	T	3		1A-OT	1970				93	ICOLD	
	Portworthy, United Kingdom	DS	R	15		1A-ST	1970				97	ICOLD	
	Unidentified, Mississippi, USA	US	T	15		1A-OT	1970				152	ICOLD	
	Williamsport Washer, Maury County, Tennessee, USA			21		1A-U	1970				182	ICOLD	
	Bilbao, Spain					1A-SI	1969	115,000	0.035	?	15	ICOLD, WISE	
	Monsanto Dike 15, TN,	DS	E	43	1,230,000	2A-SE	1969				86	ICOLD	
	Stoney Middleton, UK					1A-SI	8-Feb-68				217	ICOLD	
	Hokkaido, Japan	US	T	12	300,000	1A-EQ	1968	90,000	0.15		57	ICOLD, WISE, Rico	

COLOR CODE	MINE/PROJECT & LOCATION	DAM TYPE	DAM FILL MATERIAL	DAM HEIGHT (meters)	STORAGE VOLUME (cu. meters)	ICOLD TYPE	INCIDENT DATE	RELEASE VOLUME (cu. meters)	RUNOUT (km)	DEATHS	Source Color Code	SOURCES	NOTES
	Agrico Chemical, Florida, USA					1A-U	1968				1	ICOLD	
	IMC K-2, Saskatchewan, Canada	US	T	30		3-	1968				60	ICOLD	
	Climax, Colorado, USA					1A-U	2-Jul-67	12,000			33	ICOLD	
	Mobil Chemical, Fort Meade, Florida, phosphate					1A	1-Mar-67	2,000,000			83	ICOLD, WISE	250,000 m3 of phosphatic clay slimes, 1.8 million m3 of water. Spill reaches Peace River, fish kill reported
	Unidentified, United Kingdom	DS		20		1A-SI	1967				144	ICOLD	
	Unidentified, United Kingdom #3	DS	MW	14		2A-SI	1967				145	ICOLD	
	Unidentified, United Kingdom #2	DS	E	30		2A-SE	1967				146	ICOLD	
	Alberfan, Wales						21-Oct-66	112,000		144		Wikipedia	Coal tip (waste rock pile) failure
	Mir mine, Sgorigrad, Bulgaria	US	T			1A-U	1-May-66	450,000	8	488	81	ICOLD, WISE	Tailings wave traveled 8 km to the city of Vratza and destroyed half of Sgorigrad village 1 km downstream, killing 488 people.
	Williamthorpe, UK		MW			1A-OT	24-Mar-66				183	ICOLD	
	Unidentified, Texas, USA	US	T	16		1A-SE	1966	130,000			154	ICOLD, WISE	
	Gypsum Tailings Dam (Texas, USA)	UP		11	7,000,000	1A-SE	1966	85,000	0.3			WISE, Rico	Summary of Research on Analyses of Flow Failures of Mine Tailings Impoundments, J. K. Jeyapalan, J. M. Duncan, and H. B. Seed
	Derbyshire, United Kingdom	DS		8		1B-FN	1966	30,000			38	ICOLD	
	Williamthorpe, UK #2					1A-FN	1966				216	ICOLD	
	Tymawr, United Kingdom Inc#2			12		1A-OT	29-Mar-65				125	ICOLD, WISE	
	Bellavista. Chile	US	T	20	450,000	1A-EQ	28-Mar-65	70,000	0.8		12	ICOLD	The tailings failures of March 28, 1965, were from La Ligua, Chile, earthquake. This accounts for a significant part of the large number of earthquakes in the period of 1960-1970. About half of the failed dams were abandoned, and half were located at operating mines. (see Villavicencio et al, 2014)
	Cerro Blanco de Polpaico, Chile	WR	R	9		2A-EQ	28-Mar-65				26	ICOLD	
	El Cerrado, Chile	US	T	25		2B-EQ	28-Mar-65				42	ICOLD	
	El Cobre New Dam	DS	CST	19	350,000	1A-EQ	28-Mar-65	350,000	12		43	ICOLD, WISE	
	El Cobre Old Dam	US	T	35	4,250,000	1A-EQ	28-Mar-65	1,900,000	12	>200	45	ICOLD, WISE, Rico	

COLOR CODE	MINE/PROJECT & LOCATION	DAM TYPE	DAM FILL MATERIAL	DAM HEIGHT (meters)	STORAGE VOLUME (cu. meters)	ICOLD TYPE	INCIDENT DATE	RELEASE VOLUME (cu. meters)	RUNOUT (km)	DEATHS	Source Color Code	SOURCES	NOTES
	Hierro Viejo, Chile	US	T	5		1A-EQ	28-Mar-65	800			55	ICOLD	
	La Patagua New Dam,	US	T	15		1A-EQ	28-Mar-65	35,000	5		69	ICOLD, Rico	
	Los Maquis No. 1	US	T	15		2B-EQ	28-Mar-65				70	ICOLD	
	Los Maquis No. 3	US		15	43,000	1A-EQ	28-Mar-65	21,000	5		71	ICOLD, Rico	
	Ramayana No. 1, Chile	US	T	5		1A-EQ	28-Mar-65	150			99	ICOLD	
	Sauce No. 1, Chile	US	T	6		2A-EQ	28-Mar-65				104	ICOLD	
	Sauce No. 2, Chile	US	T	5		2B-EQ	28-Mar-65				105	ICOLD	
	Sauce No. 3, Chile	US		5		2B-EQ	28-Mar-65				106	ICOLD	
	Sauce No. 4, Chile	US	T	5		2B-EQ	28-Mar-65				107	ICOLD	
	Cerro Negro No. (1 of 5)	US	T	46		2B-EQ	28-Mar-65				27	ICOLD	Cracking due to EQ
	Cerro Negro No. (2 of 5)	US	T	46		2B-EQ	28-Mar-65				28	ICOLD	Cracking due to EQ
	Cerro Negro No. (3 of 5)	US	T	20	500,000	1A-EQ	28-Mar-65	85,000	5		29	ICOLD WISE, Rico	Dam failed due to EQ
	El Cobre Small Dam	US	T	26	985,000	2B-EQ	28-Mar-65				46	ICOLD	
	American Cyanamid, Florida #2					1A-U	1965				4	ICOLD	
	N'yukka Creek, USSR	WR	E	12		2A-FN	1965				89	ICOLD	
	Unidentified, Idaho, USA	DS	E	18		2A-SI	1965				150	ICOLD	
	Alcoa, Texas, USA			19	4,500,000	1A-U	Oct-64				2	ICOLD	
	Utah construction, Riverton, Wyoming, USA					2A-OT	16-Jun-63				174	ICOLD	
	Mines Development, Edgemont, South Dakota, USA					1A-U	11-Jun-62	100			80	ICOLD	
	Huogudu, Yunnan Tin Group Co., Yunnan	US				1A	1962	3,300,000	4.5	171		Wei	
	American Cyanamid, Florida					1A-U	1962				3	ICOLD	

COLOR CODE	MINE/PROJECT & LOCATION	DAM TYPE	DAM FILL MATERIAL	DAM HEIGHT (meters)	STORAGE VOLUME (cu. meters)	ICOLD TYPE	INCIDENT DATE	RELEASE VOLUME (cu. meters)	RUNOUT (km)	DEATHS	Source Color Code	SOURCES	NOTES
	Unidentified, Peru					1A-EQ	1962				135	ICOLD	
	Union Carbide, Maybell, Colorado, USA					1A-U	6-Dec-61	280			171	ICOLD	
	Tymawr, United Kingdom					1A-U	Dec-61				124	ICOLD	
	Lower Indian Creek, MO,	US	E			2A-SI	1960				72	ICOLD	
	Union Carbide, Green River, Utah, USA					1A-OT	19-Aug-59	8,400			170	ICOLD	
	Grootvlei, South Africa	US	T			1A-SI	1956				54	ICOLD	
	Unidentified, Peace River, Florida, USA 3/52	WR	E	8		1A-SI	Mar-52				168	ICOLD	
	Unidentified, Alfaria River, Florida, USA	WR	E	8		1A-SI	Feb-52				156	ICOLD	
	Unidentified, Peace River, Florida, USA 9/51	WR	MW	6		1A-SE	Sep-51				165	ICOLD	
	Unidentified, Peace River, Florida 7/51	WR	MW	30		1A-SE	Jul-51				166	ICOLD	
	Unidentified, Peace River, Florida, USA2/51	DS	E			1A-SE	Feb-51				167	ICOLD	
	Kimberley, BC, Canada, iron	US	T			1A-SI	1948	1,100,000			66	ICOLD	
	Castle Dome, Arizona, USA	US	T			1A-SE	29-Sep-47	150,000	0.1		25	ICOLD	
	Hollinger, Canada	US	T	15		1A-FN	1944				58	ICOLD	
	Captains Flat Dump 3, Australia		T			1A-U	1942	40,000			20	ICOLD	
	Kennecott, Utah, USA	US	T			1A-FN	1942				63	ICOLD	
	Kennecott, Garfield, Utah, USA	US	T			1A-SI	1941				62	ICOLD	
	St. Joe Lead, Flat Missouri, USA	US	T	15		1A-OT	1940				115	ICOLD	
	Captains Flat Dump 6A, Australia	US	T			1A-SI	1939				21	ICOLD	
	Simmer and Jack, South Africa	US	T			1A-SI	1937				110	ICOLD	
	Barahona, Chile	US	CST	61	20,000,000	1A-EQ	Oct-28	2,800,000			9	ICOLD	
	Unidentified, South Africa					1A-U	1917				136	ICOLD	

APPENDIX 2

Technical Documentation on Canonical Correlation Analysis

TECHNICAL DOCUMENTATION ON CANONICAL CORRELATION ANALYSIS

CANONICAL CORRELATION ANALYSIS (CCA)

Canonical correlation considers the relationship between two data sets one normally considered a “criteria” data set the other and “explanatory” data set. For our CCA analysis the criterion data set (Y1) were the Very Serious Failure and Serious Failures. The explanatory data set (Y2) were the three mining metric variables shown to have the highest correlation with these failure categories copper ore production (Cu prod), copper grade (Cu grade), and copper cost to produce (Cu cost).

Table A2.1 - Input Data Set

Decade	Very Serious Failures	Serious Failures	Other Failures	Other Accident	Non-Dam Failures	All Failures	Cu prod (K tonnes)	Cu grade (%)	Cu cost $/tonne	Cu price $/tonne
1940 – 49	0	0	5	0	0	6	2,545	1.52	$35	$3,633
1950 – 59	1	0	7	0	0	7	3,680	1.21	$48	$5,076
1960 – 69	3	4	25	17	2	51	5,004	1.10	$55	$5,112
1970 – 79	4	8	23	15	3	53	7,445	1.01	$38	$5,895
1980 – 89	5	9	22	14	4	54	10,575	0.95	$20	$3,871
1990 – 99	9	9	10	3	1	32	16,437	0.93	$15	$3,292
2000 – 09	7	8	5	1	0	21	23,658	0.85	$20	$4,256
	======	======	======	=======	=======	======	======	=====	======	=======
Total/Ave	29	38	97	50	10	224	69,344	1.54	$33	$4,448

Abbreviations:

Cu Price = Copper price ($/tonne)
Cu Prod Cost = Cost to produce copper concentrate from copper ore, including waste disposal ($/tonne)
Cu Grade = grade of copper in the ore (%)
Cu Prod = copper ore production (thousand metric tonnes)
Other Failures = tailings dam failures and incidents other than Serious or Very Serious Failures
Serious Failures = Serious tailings dam failures
Very Serious Failures = Very Serious tailings dam failures

Sources: USGS (Metal Statistics) 2014, Schodde 2010, ICOLD 2001, WISE 2015 & additional

DEVELOPMENT AND VETTING OF INPUT DATA

These final selections were based on a rigorous and thorough exploration of the structure of the data within each set and of the inter-relationships among data elements. After settling on the above data set it was vetted against two criteria for proper use and meaningful interpretation of CCA: Multivariate Normality and Multicollinearity.

We had pre-determined CCA to be the best multivariate analysis technique for our consideration of how the “Mining Metric” affects TSF failure frequency and severity globally. We were not looking at this relationship on a time series basis but on a criteria and explanatory basis for which CCA was specifically developed. CCA is used mostly for looking at whether and how intentional or known environmental conditions or interventions affect a given set of observed conditions. (E.g. and more typically, whether the elements of a diet and exercise program, as a program, have more positive effect on measures of health and which elements are most strongly related to the desired or expected outcome.) At least one major economic study (Malacarne 2014) published in the Mathematica

Journal also employed CCA. That study explored whether and to what extent behavior of the major stock exchanges of developed nations influenced the behavior of the exchanges of developing nations.

CCA is perfectly suited to our study because although price is the fixed element against which all mines must perform, the other elements of the Mining Metric are subject to miner control and or have great variability one mine to another within the expressed averages, including how much production to undertake at a given head grade and mine specific cost of production, and how much cash flow is available for nonrevenue generating parts of the operation like waste and waste water management.

DEVELOPMENT OF INPUT VARIABLES

The main defining criteria for severity classifications are apparent on a sort by Release Volume (column L) and Run Out (column M) (See Appendix I), or even a visual inspection. The category Very Serious Failures has had clarity in all analysis from the outset in its relationship to the key Mining Metric variables, and much stronger alone than in combinations we experimented with. Similarly, combinations of coding for other incidents didn't have the clarity we finally found in these final 5 major failure groups. Among these 5 groups (classifications) as shown in the correlation matrix in Table 3.1 only the two high severity codes had significant correlations with Mining Metric variables.

Similarly with the Mining Metric variables we found that the original raw data had greater clarity than any combinations we formulated. For example, on noting the lower correlation of price with failures variables, we created a variable called "price cycle" that coded each decade on the basis of length of trend up or down. Since cumulative production is a surrogate for the exponential growth in global accumulated tailings volume we initially focused on that but found that cumulative production had consistently lower correlations with any coding of failure categories, and so settled on using production as reported by the USGS metal statistics. We tried to improve correlations with various other formulations. But in all cases the actual raw measurements of cost, grade and production were found to have the highest correlations with high severity failures events.

We also had explored an "all metals" basis in lieu of using Cu only and found that no combination of all metals had the same strength of correlation as Cu production alone. (Possibly because Cu production so closely tracks Global GDP). USGS Metal Statistics (2014) includes price, but there were no other comparable sources for average head grade or average production costs. Only Cu afforded the possibility of looking at the interrelationships over the entire century 1910-2010.

For almost any analysis there were too many empty cells for a complete Y1 and Y2 set prior to 1940. Therefore we ended up with a workable data set of only two Y1 variables and three Y2 variables for only 7 decades out of the 10 in the century. At the outset, therefore, we knew that our workable data set was much smaller than what is normally considered the minimum for CCA, and that that would limit the statistical significance of conclusions, but not preclude a meaningful glimpse into the relational behavior of the two data sets.

APPROPRIATE AGGREGATION LEVEL OF FAILURE DATA

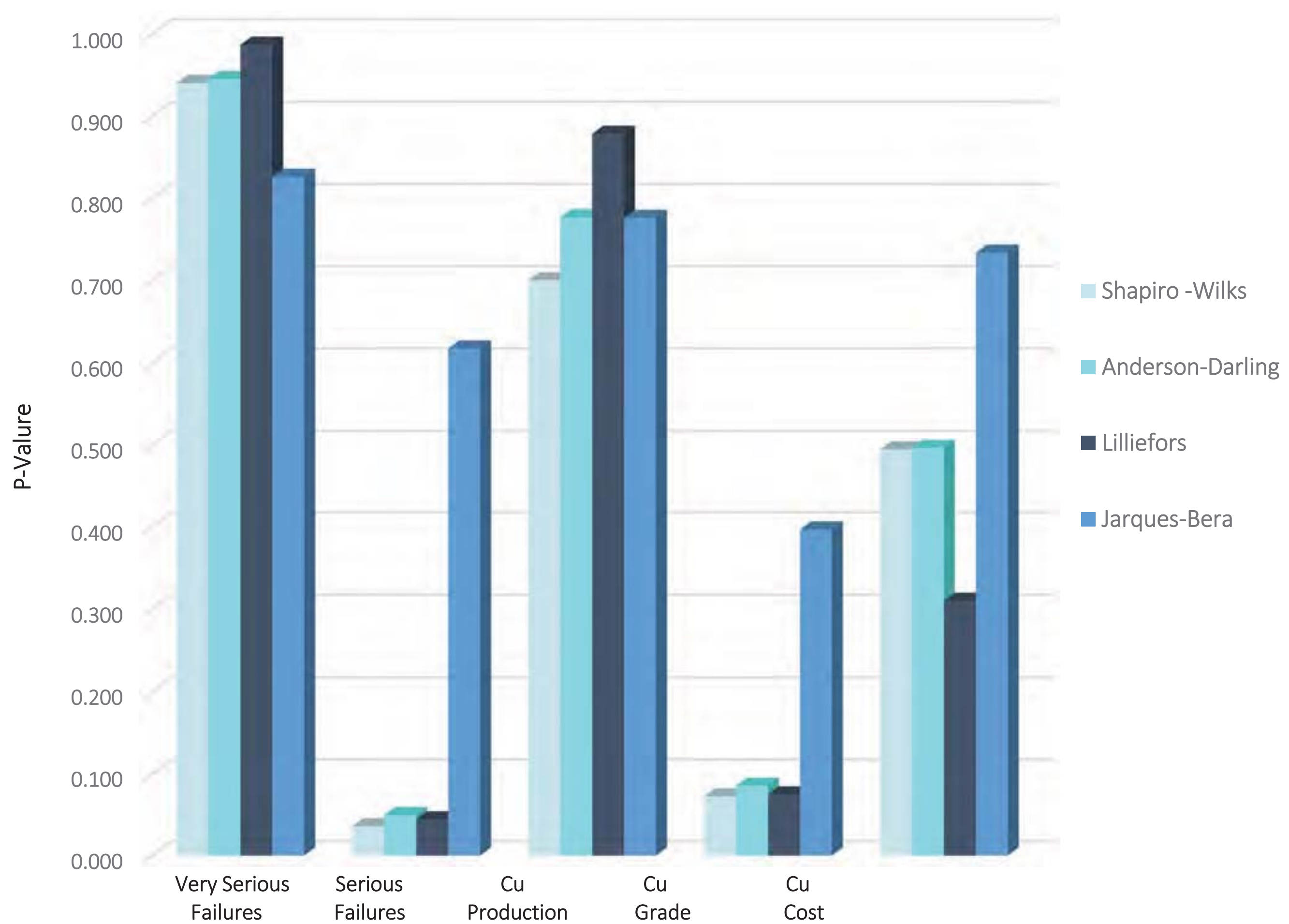
To determine the most appropriate level of aggregation for the failure data sets we looked at aggregations by 1, 2, 5 and 10 years building from the earliest year, 1910. The decade 1910-1920 is the earliest recorded ICOLD TSF incident. We found that the clarity of inter relationships was not apparent at aggregations below 5 years and was most clear at aggregations by decade.

Ideally, there should be 20 observations for each variable which would have required aggregations of 3 years or less. This is also true of the U.S. Census or any other phenomenon that looks at small incremental changes or incidents over a long period of time and the interrelationship with other inter-census changes. These changes would not be apparent or meaningful at smaller levels of aggregation as the many elements of population change (age, ethnicity, household size) have constant small changes day-by-day, month-by-month, which don't reveal the magnitude of net effect or net change until a meaningful level of aggregation is established. Ten years happens to be the apparent optimum level of aggregation for analysis of frequency and severity of TSF failures and reportable incidents.

VETTING OF INPUT DATA SET ON REQUIREMENTS FOR CCA

The proper use of CCA for descriptive analysis requires no assumptions of distribution. To test the significance of the relationships between canonical variates, however, the data should meet the requirements of multivariate normality (MVN). We were not able to conduct a full multivariate normality as it was not an option in XLSTAT©. The normality of each variable within the data set is not a proof of MVN, but all elements of a data set that does meet the requirements of MVN must meet univariate tests of normality. We therefore used the results of univariate tests on each of the 5 input variables as an approximation of MVN as did Malacarne (2014). XLSTAT© automatically gives output for 5 different normality tests and is presented in Figure A2.1, below: P values at 95% confidence intervals are presented for each test on each variable. (The higher the P value the more likely the sample/observation set is drawn from a population with a normal distribution.) The alpha level was 0.05 (95% confidence limits). The closer to 0.05 alpha value the P value is the less certainty that the data set is from a population with a normal distribution. Each test involves different assumptions and approaches to testing for a normal distribution.

Figure A2.1 Tests of Normality
Comparisons on Major Measures By Variable



These results are being presented as a point of interest to get some insight to the data set. All of these measures are known to be robust with very small data sets and normally 20 is the smallest data set they should be performed on. Interesting and not unexpected to note that all 5 variables satisfied only the Jarques-Bera that is most often the case with econometric data sets. Jarques-Bera, alone requires no known mean or standard deviation and is based on skewness and kurtosis. It is interesting to note that "Serious Failures" and "Cu Grade" satisfied the criteria for normality only on the Jarques-Bera. In the case of "Serious Failures" that could be due to its "curvature". (See Figure 3.2) In the case of Cu Grade it may be due to the small difference min to max. Despite the results, this is not conclusive of MVN but strongly suggests that and supports that our use of CCA for exploration is reasonable.

Multicollinearity must not exist for meaningful use and interpretation of CCA Each data set was also tested via principal Component Analysis for Multicollinearity and the eigenvalues for each were very high, 1.880 for the failures data set accounting for 94.013 % of variability and 2.546 for the mining metric data set accounting for 84.8% of variability. Again there is a tendency to robustness in small data sets with very strong linearity. We are concluding only that the data set seems to satisfy the requirement for no Multicollinearity.

All of these results support that CCA is a suitable analytic tool for this Y1, Y2 data set and that the results can be meaningfully interpreted, albeit with acknowledged limitations on affirming statistical significance.

Table A2.2 – Failures Data Eigenvalues

	F1	F2
Eigenvalue	1.880	0.120
Variability (%)	94.013	5.987
Cumulative %	94.013	100.000

Table A2.3 – Mining Metric Data Set Eigenvalues

	F1	F2	F3
Eigenvalue	2.546	0.377	0.077
Variability (%)	84.859	12.570	2.571
Cumulative %	84.859	97.429	100.000

Our aim was not statistical significance, but a better understanding of the nature and structure of the relationship between the high severity failures and the mining metric variable affecting all mines and all miners. CCA offered that and is particularly well suited to exploration of relationships within complex systems and complex multi causal effects.

TSF failures resist any efforts to definitively map what specific combinations of events will result in failure, but we can meaningfully explore the contribution of various elements.

Our data set of "causes" most often associated with failure is itself raggedly incomplete and not systematically recorded for every failure. CCA allows analysis of the relationships between any two sets of system known to be much more complex than just the effects studied through CCA. It allows an open exploration of inter relationships and their intensity without in any way discounting other factors that may contribute as much or more to both likelihood of failure and severity of failure.

OUTPUTS OF CCA

Canonical Correlation Analysis (CCA) is most usually and almost universally defined as "the problem of finding two sets of basis vectors, one for data set Y1 and the other for data set Y2, such that the correlations between the projections of the variables onto these basis vectors are mutually maximized."

CCA seeks a pair of linear transformations, one for each of the sets of variables such that when the set of variables are transformed the corresponding co-ordinates are maximally correlated. The linear transformations are synthetic variables. One "synthetic variable" or canonical variate is create for each data set.

CCA is in some respects similar in principal to dimensional analysis in engineering employed to statistically explain or explore the complex relationships producing observed measurements. CCA similarly “discovers” the relationships that may not otherwise be apparent in univariate correlation analysis or which may be understated or not detected at all in univariate analysis (because of interrelationships within and between the two sets)

UNIVARIATE CORRELATION MATRIX

The univariate correlation matrix is a standard CCA output and presents the relationships in the entire data set, and is used to assess both the degree of independence and the degree of individual variable to variable relationships across all variables in the selected arrays Y1 (the severity of failure array) and Y2 (the Mining Metric array). These values are the same as those shown in Table 3.1 for the full original data set. This data set was pre-selected for the CCA for the strength of the correlations between the two failure severity classes (Y1) and the three selected mining metric variables (Y2).

Table A2.4 – CCA Output Correlation Matrix

		Y1		Y2		
		Very Serious Failures	Serious Failures	Cu prod	Cu cost	Cu grade
Y1	Very Serious Failures	1	0.880	0.860	-0.788	-0.794
	Serious Failures	0.880	1	0.720	-0.682	-0.884
Y2	Cu prod	0.860	0.720	1	-0.782	-0.756
	Cu cost	-0.788	-0.682	-0.782	1	0.497
	Cu grade	-0.794	-0.550	-0.756	0.497	1

EIGENVALUES

The principal output of a canonical correlation analysis are the canonical functions (variates) which seek to maximize explained variability between the two arrays (Y1 and Y2). Each function produced is an equation (similar to the equations created in regression analysis) but instead of explaining the relationships in terms of causality, it seeks to define the dimension (strength) of the relationship between (or in larger data sets among) the arrays. Essentially it asks are these arrays independent of one another, or does there appear to be an influence of the two arrays on one another. As many canonical functions are produced as there are variable sets

The first exploration of these canonical functions is the eigenvalue which measures how much variability is explained by each of the canonical functions. The closer the eigenvalue is to zero the less likely the two arrays form a diagonal matrix, i.e. have a linear correlation to one another which might therefore be suitable for linear modeling (regression analysis).

Table A2.5 – Eigenvalues

	Failures Data Eigenvalues	
	Canonical Function 1	Canonical Function 2
Eigenvalue	0.903	0.528
Variability (%)	63.1	36.9
Cumulative %	63.1	100.000

In this case the Eigenvalue for the first canonical function, F1, 0.903 strongly indicates a diagonal matrix. F1 explains explained 93.9% of the total variability between the two arrays indicating a very strong diagonal matrix. The second function, F2, calculated to be maximally independent of the first, in our data set also contributes to explaining 36.9% of the relationship between the two arrays.

We would expect any two data sets with similar within set patterns to produce very high eigenvalues but this FI result is higher than any produced from randomly generated arrays with similar slope and range for each variable. So this does add to our understanding of the strength of the linear relationship between Y1 and Y2.

Wilks Lambda

Wilks' Lambda is a test of the null hypothesis that the data sets are independent of one another as measured via the canonical coefficients. The lower the Wilk's Lambda, the less likely that the data sets Y1 and Y2 are independent. The following results means it is unlikely that the two data sets are independent of one another.

Table A2.6 – Wilks' Lambda Test

	Lambda	F	DF1	DF2	Pr > F
F1	0.046	2.451	6	4	0.202
F2	0.937				

F-Value

The value of the F approximation (a probability distribution) for testing the significance of the Wilks' Lambda corresponding to this row and those below it. If F is an approximation, as here, it is generated as appropriate to the test. Similar F values were XLSTAT generated for the actual (4.6) and the control (4.58) data sets. The first F-value tests the significance of the 1st and 2nd canonical correlations.

DF1

The numerator degrees of freedom of the above F-ratio.

DF2

The denominator degrees of freedom of the above F-ratio.

PR>F (Probability Level)

This is the probability value for the above F statistic. A value near zero indicates a significant canonical correlation. A cutoff value of 0.05 or 0.01 is often used to determine significance at the 95%h of 99% level.

This result is below a 95% confidence level (0.05) but is still strong (92%) I.e., if we accepted the null hypothesis that the two data sets are independent of one another there is a 92% chance we'd be wrong.

Again any data sets with similar variable ranges and slopes would also produce similarly strong results, but several trials with made up data sets did not yield results as strong as the actual data sets. For example, the data set in Table A2.5 below produced lower eigenvalues and higher Wilks Lambdas than the actual data (although the Wilks result in the control set is significant at a higher level than the actual data set).

The data set in Table A2.5 has very similar slope and pattern to the actual failures and mining metric data sets. The synthetic data are plotted in Figure A2.2.

The very high R-Squared as for the actual data set. The eigenvalue not as high and the Wilk's Lambda not as low.

The real data shows a strength of relationship that is not present in synthetic data sets with similar dimensionality and slope for each of the 5 variables.

Table A2.7 – Synthetic Data Set

Decade	Synthetic Very Serious Failures	Synthetic Serious Failures	Synthetic Cu Grade	Synthetic Production Cost	Synthetic Cu Production
1	46.8	49.95	70.14	66.0	50.00
2	39.0	42.18	65.13	52.8	51.55
3	31.2	57.72	46.76	24.2	53.15
4	46.8	43.29	33.40	17.6	54.80
5	54.6	53.28	31.73	13.2	56.49
6	54.6	62.16	30.06	8.8	58.25
7	62.4	57.72	20.04	6.6	60.05
8	54.6	65.49	18.37	6.6	61.91
9	70.2	61.05	15.03	4.4	63.83
10	62.4	69.93	5.01	4.4	65.81
11	70.2	68.82	3.34	6.6	67.85
12	62.4	69.93	1.67	11.0	69.95

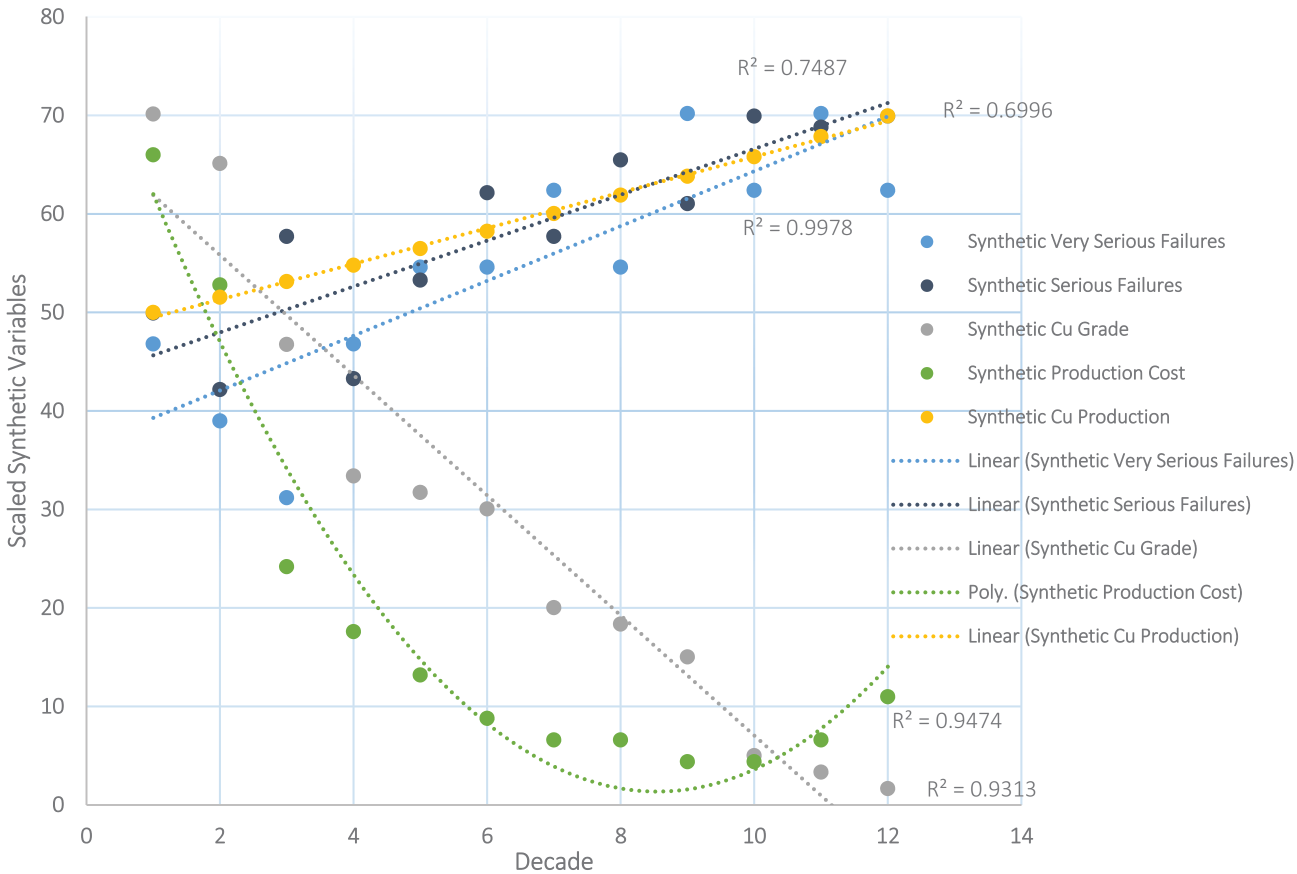
Table A2.8 – Synthetic Data Set Wilks' Lambda Test

	Lambda	F	DF1	DF2	Pr > F
F1	0.114	4.583	6	14	0.009
F2	0.932				

Table A2.9 – Synthetic Data Set Eigenvalues

	Canonical Function 1	Canonical Function 2
Eigenvalue	1.880	0.120
Variability (%)	94.013	5.987
Cumulative %	94.013	100.000

Figure A2.2 – Graph of Synthetic Values



Canonical Correlations

The canonical correlations (also called variates) are the two synthetic variables resulting from the projections of each data set onto a base vector maximizing the mutual variability between the two data sets. The result for each function, F1 and F2 describes the amount of variability accounted for. The higher the value the greater the amount of variability explained by the functions. Function F1 explained 95% of the variability.

Table A2.10 – Canonical Correlation Values

	F1	F2
Canonical Correlation	0.950	0.727
Eigenvalue	.903	0.528
Wilks' Lambda	0.046	0.472

CORRELATION BETWEEN F1 CANONICAL VARIATE AND DATA SET VARIABLES

The aim of CCA is to discover whether dimensions of relationship exist between y1 and y2 variables that were not apparent in the graphs, charts and univariate analysis vis-a-vis one to one correlations. The correlations are shown below between both correlations (F1 and F2), and each variable in each data set, Y1 and Y2. They reveal a stronger influence of grade and cost to produce and reaffirmed the primary dominant relationship with production volume on both categories of failure severity. It also brought out stronger relationships in general between Serious Failures and the Mining Metric variables than were revealed in univariate and graphic analysis. The relationship between Serious Failures and the mining metric may be via cost of production.

Very Serious Failures had a much stronger and opposite correlation with F2 than did Serious Failures, - 0.388 v. - 0.096. The main component in F2 is ore production (-0.588) with cost also strong, 0.450.

In F1 which is much more strongly correlated with Serious Failures, grade is the principal element, 0.929. (While the second variate, F2 does not have the Wilks and Eigen Values of F1, it does it does illustrate that serious failures is a distinctive and separate failure severity group despite its many commonalities with very serious failures. As the possibility of larger data sets grow going forward (i.e. more information from 2000 onward) it may be possible to explore those differences more fully.

The first canonical variate, F1, had very strong correlations with all variables in each of the two data sets (Y1, failures and Y2, mining metric elements. Both of the failure variables and production volume had very strong negative correlations with F1: -0.922, -0.995. Cu Ore Production (-0.802). Cu Cost (0.755) and Cu Grade (0.929) were also highly correlated with F1. F1 is a therefore a nearly complete expression of the very strong relationship between the two data sets with each of the mining metric elements.

Table A2.9 - Input and Canonical Variable Correlations

Correlations between input variables and canonical variables (Y1):

	F1	F2
Very Serious Failures	-0.922	-0.388
Serious Failures	-0.995	0.096

Correlations between input variables and canonical variables (Y2):

	F1	F2
Cu Production	-0.802	-0.558
Cu Cost	0.755	0.072
Cu Grade	0.929	0.368

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http://faculty.arts.ubc.ca/dwhistler/325ClassNotes/chapNorTest.pdf

APPENDIX 3

Documented TSF Very Serious Natural Resource Losses

1990 - 2010

Documented TSF Very Serious Natural Resource Losses 1990 – 2010

TSF Failure	Year	Original Currency (Millions)	Failure Year M US$	2014 M US$	Ore	Release (M m^3)	Run Out (km)	Deaths	Source
Kingston Fossil Plant, Harriman, Tennessee, USA	2008	US 1,200	$1,200	$1,300		5.4	4.1		http://www.sourcewatch.org/index.php/TVA_Kingston_Fossil_Plant_coal_ash_spill
Taoshi, Linfen City, Xiangfen, Shanxi Province, China	2008	US 1,300	$1,300	$1,429	Fe	0.19	2.5	277	Wei, Yin, Wang Ling,Wan(2012), http://wmr.sagepub.com/content/31/1/106.full.pdf+html
Baia Mare, Romania	2000	US 179	$179	$246	Au	0.1	5.2		(1) http://www.wise-uranium.org/mdafbm.html (2) http://viso.jrc.ec.europa.eu/pecomines_ext/docs/bmtf_report.pdf
Los Frailes, Spain	1998	EU 275	$301	$437	Zn/Cu/ Pb	4.6	5		http://www.wise-uranium.org/mdaflf.html
Marinduque Island, Philippines	1996	P 180 + US 114	$123	$185	Cu	1.6	27		(1) http://www.slideshare.net/no2mininginpalawan/major-tailings-dam-disasters-in-the-philippines-alyansa-tigil-mina-atm-april-2011-7819384Philippines (2) http://newsinfo.inquirer.net/479345/marinduque-folk-lose-case-vs-mine-firm (3) http://opinion.inquirer.net/63421/marinduque-is-pushed-to-the-wall (4) http://www.slideshare.net/jillentot/environmental-damages-and-health-hazards-caused-by-marcopper?related=1 (5) Bennagen, 1998
Omai, Guyana	1995	US 100	$100	$156	Au	4.2	80		(1) http://www.thefreelibrary.com/Cambior+Inc.+Announcement.-a055509330 (2) http://www.monitor.net/monitor/9-18-95/eyewitness.html (3) http://ejatlas.org/conflict/omai-gold-mine-tailings-dam-guyana (4) http://www.multinationalmonitor.org/hyper/issues/1995/11/mm1195_04.html
Merriespruit, South Africa	1994	R 100	$29	$46	Au	0.6	2	17	http://floodlist.com/africa/merriespruit-tailings-dam
	Average US$2014: $543			====== $3,799					

NOTES:

A. HISTORICAL CURRENCY CONVERTERS

(1) http://unix4.outcoursing.com/currency-converter/us-dollar-usd_zar-south-african-rand.htm/1994

(2) http://www.x-rates.com/historical/

2005-2015 selected currencies

(3) http://fxtop.com/en/currency-converter-past.php?A=275&C1=EUR&C2=USD&DD=01&MM=01&YYYY=1998&B=1&P=&I=1&btnOK=Go%21

Converts from any one currency to another for any given date 1953-2015

(4) http://www.usinflationcalculator.com/

Advances value of $US from any year from 1913 to any year up to 2015

B. DOCUMENTED TSF VERY SERIOUS NATURAL RESOURCE LOSSES

(1) **TAOSHI, LINFEN CITY, XIANGFEN COUNTY, SHANXI PROVINCE**

US2008 $1,300 million = US2014 $1,429 million

This failure released approximately "1.9 × 105 m^3 tailings. The tailings flowed as far as 2.5 km downstream and covered about 35 hectares of land. … The tailings destroyed many houses, caused 277 deaths, 33 injuries, and caused about US$ 1.3 × 10^7 in direct losses. The failure also resulted in very serious social impacts."

Source:
(a) http://wmr.sagepub.com/content/31/1/106.full.pdf+html

(2) **BAIA MARE**

US2000 $179 million = US2014 $246 million

Operated by AURUL, a joint-venture between Esmeralda Exploration of Australia and REMIN the Romanian state owned mining company.

"On Dec. 16, 2000, Tom Garvey, the head of a European Union task force investigating the spill said there is no doubt the mine was at fault and is responsible for the environmental disaster." No doubt whatever it was a direct result of a hundred tonnes plus of cyanide going into the Pau, the Somas and the Tisza River and killing everything in its wake," he said.

The investigation concluded that the accident was caused by the inappropriately designed tailings dams, the inadequate monitoring of the construction and operation of those dams and by severe - though not exceptional - weather conditions. (Australian Broadcasting Corporation Dec. 16, 2000) (1)

Excerpts from the Baia Mare International Task Force Investigation

"As a result, it is the conclusion of the BMTF that the accidents were caused:

- Firstly, by the use of an inappropriate design of the TMF;
- Secondly, by the acceptance of that design by the permitting authorities; and
- Thirdly, by inadequate monitoring and dam construction, operation and maintenance"(2)

"Furthermore there was a problem in the case of Baia Mare with the stability of the embankment walls themselves. This arose because the Baia Mare facility used a recognized technique of embankment or dam wall construction (called 'construction by operation') which called for the gradual deposition of tailings of sufficiently coarse grade on the starter walls to ensure stable and continuous growth of the height of the embankment walls.

However, the mix of tailings used did not have the ratio of coarse to fine grades stipulated in the design and, in addition, the hydrocyclones used to distribute the tailings within the pond could not operate in the very low temperatures experienced before the accident. As a result the embankment wall construction was interrupted at a critical time, leading to a reduction in the 'freeboard', and consequently to wall breaching and overflow." (2)

"In effect, these were two accidents waiting to happen, waiting for the necessary trigger of adverse weather conditions which was bound to come sooner or later." (2)

On July 11, 2000, the Hungarian Government lodged a $179 million compensation claim against Esmeralda Exploration. (1)

Sources:

(a) http://www.wise-uranium.org/mdafbm.html
(b) http://viso.jrc.ec.europa.eu/pecomines_ext/docs/bmtf_report.pdf

(3) **LOS FRAILLES**

EU1998 €275 million = US1998 $301.4 million = US2014 $437 million

Operated by Boliden Ltd. Sweden via subsidiary Boliden-Apirsa

On November 20, 2001, the Andalusian Government and the Spanish Environmental Ministry announced to sue for damages. Both Administrations have spent more than Pesetas 40,000 million (Euro 240 million / US$ 210 million) for the clean-up of the spill. (El País Nov. 21, 2001)

On December 14, 2001, Boliden Apirsa signed agreements with the Regional Government of Andalucía and with the workers council and unions regarding environmental restoration plans and severance payments. The mining company had presented a plan of environmental restoration and abandonment of the mine valued in 8,269 million pesetas (EUR 50 million / US$ 45 million). The workers council, however, estimated that at least an additional 5,000 million pesetas (EUR 30 million / US$ 27 million) were required. I.e. future work estimated by regional government of Andalusia at $72 million (beyond what was sent as of 11/21/2001)

In the agreement obtained, the Regional Government had to accept the payment with assets of the company for lack of sufficient funds available. But it reserved the right to claim from Apirsa's Swedish parent company Boliden Ltd any additional funds that might be required in the future. (El País Dec. 15, 2001)

The environmental group Ecologistas en Acción has decided to draw the case on the penal responsibility for the tailings dam failure before the Constitutional Court. (El País Feb. 1, 2002)

On April 23, 2002, the advisor of Environment, Fuensanta Coves, indicated that the legal services of the Regional Government are completing the statements of civil claims against Boliden-Apirsa, to demand a part of the funds used to repair the damages of the accident. The Andalusian Administration has invested more than 152 million Euros (around 25,000 million Pesetas) in the recovery, and it anticipates to spend another 10 million Euros in 2002. El País April 24, 2002) (i.e. total costs as of 2002 put at $162 EU)

On July 2, 2002, the Environmental Council of the Andalusian Government approved the initiation of civil actions against the mining company to try to recover part of the 152 million Euros (25,000 million pesetas) spent to decontaminate the affected zone. (El País July 3, 2002)

On July 31, 2002, the Environment Council of the Andalusian Government concluded the removal of the 10,000 cubic meters of muds that still were stored in the river basin of the Guadiamar. The Environment Council furthermore announced that it will come to the reforestation of the affected zone in October 2002. (El País August 1, 2002)

On August 2, 2002, the Council of Ministers imposed a penalty of 45 million Euros on Boliden, the highest ever by environmental damages in Spanish history. Nevertheless, the fine covers only about one sixth of the cleanup cost of 276 million Euros spent by the administrations so far. (El País / El Mundo, August 3, 2002)

Boliden announced it is not willing to pay a single cent. (ABCe August 5, 2002)

The Andalusian Government plans to impose another penalty of 86 million Euros on Boliden to recover the cost it has spent on the cleanup. (El País August 6, 2002)

Boliden claims damages from the Spanish construction company Dragados: Boliden's Spanish subsidiary Boliden Apirsa has filed a notice of litigation against Dragados y Construcciones S.A., a member of the construction company Dragados S.A., listed in Spain, in connection with the failure of the tailings dam at the Los Frailes mine, Spain, in 1998. Boliden´s claim against Dragados amounts to a minimum of **1 billion SEK (107 million Euro).** The formal claim will be presented to a Spanish court in October. (Boliden Sep 26, 2002)

On Nov. 16, 2002, the regional government of Andalusia filed a civil suit to recover from Boliden 89.8 million euros ($89.9 million) in damages and cleanup costs. (Reuters Nov. 22, 2002)

On Jan. 2, 2003, the Primera Instancia número 11 court of Seville rejected the civil demand of the regional government of Andalusia against Boliden. (ABCe Jan. 4, 2003)

The regional government of Andalusia now has decided to demand from Boliden recovery of 89.9 million euros in damages by the administrative route. (ABCe Nov. 5, 2003)

"As previously announced, Boliden's Spanish subsidiary Boliden Apirsa filed a notice of litigation against the Spanish company Dragados y Construcciones S.A. Now Boliden Apirsa has filed the final claim in a court in Madrid. Boliden's claim against Dragados amounts to around EUR 115 million." (Boliden Jan. 23, 2004)

Source:

(a) http://www.wise-uranium.org/mdaflf.html

(4) **OMAI**

US1995 $100 million = US2014 $156 million

Operator: Cambior, subsidiary Golden Star Resources in partnership with INVESCOR of Denver, via subsidiary Omai Mines Ltd in which Guyana Government had 4% interest.

Class Action Lawsuit for $2B dismissed against Cambior & claimants ordered by Guyana court to pay all defense costs of all named mining interests and their insurers. (3) The dismissal and general outcome viz a viz environmental damages is widely considered a failure of environmental justice. There has been no systematic accounting of actual damages by the Guyana Government or any NGO only the imposition of a $100 million fine.

"Several months before the disaster, the company told the government that because it had underestimated the amount of waste it would produce, it would need to build a second tailings dam and partly because of the cost would be unable to pay any royalties and taxes to the government until the year 2002, just three years before the mine's is expected to close. The news had reportedly caused dismay in government circles. As Omai is the largest open pit gold mine in South America, the government expected it to contribute substantially to its revenues." (2)

"The disaster only added fuel to an already difficult relationship. Instead of being a source of revenues, the mine is now a cause of more environmental expenditures for the government, whose foreign debt sometimes consumes as much as 70 percent of its tax revenues.

"There have been warnings of a disaster in the making for months. In March, the operators of the mine warned that disposal of the waste water was a problem, and prophetically suggested the mine might need to close in August if no other way was found to deal with the waste. A small spill occurred in May and in June the government announced an investigation into whether company plans to discharge effluent into the river were environmentally sound.

"Roger Moody, the Mining Advisor to the Amerindian People's Association of Guyana (APA) and the author of several works assessing the socio-economic impact of mining projects, was invited to Guyana last December by the APA, who expressed concern about earlier reported pollution incidents at Omai.

"He was unable to get permission to visit the site. He told American Reporter News Bureau yesterday that "the mine was hastily built, ill planned and an example of greed masquerading as the hope of a poor country." The mine is a subsidiary of Invesco, Inc., a Denver, Colorado-based mutual fund giant. Among that company's outside directors is the CEO of Atlanta 1996 Olympic Games. The Canadian engineering company Knight Piesold hired by Omai Gold Mine to build the tailings dam say they were very embarrassed by being associated with the failure.

"The company has built hundreds of tailings dams and this is the first time something has happened like this," a company spokesman said. However, the firm believes that Omai further developed the tailings dam after Knight Piesold left the project, raising the walls from the 25 metres state Knight Piesold had designed to a height of 45 metres.

"The initial cyanide spill in May was reported as being due to a power failure which had prevented sluice gates from being closed. This suggests that the gates were already open at the time of the failure, perhaps for a deliberate controlled discharge of effluent.

"Such a deliberate release is entirely plausible. Omai Mines had intended from the very first to release overflows from the polluted tailings dam into the river in its original Environmental Impact Statement to the previous Guyanese government. The current government apparently inherited a tacit agreement to this controlled release, along with a five percent equity share in the mine.

"A major force in bringing the mine to reality was Canadian mining investor Robert Friedland, who at the time was reeling from a gold mine's tailings dam disaster at Summitville, Indiana, the most expensive such failure in the U.S. in recent times.

"The Environmental Protection Agency (EPA) has estimated that the final cost of clearing up the cyanide and heavy metal pollution at the Summitville mine will be about $120 million. Friedland is still wanted for questioning by the EPA.

"After the Summitville disaster, Friedland invested in Omai Gold Mines Ltd. through Golden Star Resources, the subsidiary of Canadian-based Cambior, Inc. and Invesco, which operates a $9 billion mutual fund specialized in high-risk securities from "emerging nations." Golden Star Resources is now a 35 participant in the mine. Friedland is now believed to have sold his holding in the Omai mine and to have moved on to establishing one of the world's largest new gold mines on Lihir Island in Papua New Guinea." (2)

"The Québec Superior Court dismissed the case in August 1998, on the grounds that the courts in Guyana were in a better position to hear the case. A lawsuit against Cambior was filed in Guyana, but it was dismissed by the High Court of the Supreme Court of Judicature of Guyana in 2002. A new suit was filed against Cambior in 2003 in Guyana again seeking damages for the effects of the 1995 spill. In October 2006, the High Court of the Supreme Court of Judicature of Guyana ordered the dismissal of the 2003 action and ordered the plaintiffs to pay the defendants' legal costs. (3)

"In August 1998, within the three-year limitation period, a similar Representative Action was filed in Guyana. OMAI has now been served with the Action claiming to represent some 23,000 individuals in Guyana and seeking US $100 million as compensation for damages. The Action remains open to challenge in numerous respects, and Cambior and OMAI have instructed their attorneys to contest it vigorously" (1)

Sources:

(a) http://www.thefreelibrary.com/Cambior+Inc.+Announcement.-a055509330
(b) http://www.monitor.net/monitor/9-18-95/eyewitness.html
(c) http://ejatlas.org/conflict/omai-gold-mine-tailings-dam-guyana
(d) http://www.multinationalmonitor.org/hyper/issues/1995/11/mm1195_04.html
(e) https://ujdigispace.uj.ac.za/handle/10210/7295

(5) **MARINDUQUE**

Natural Damage Rehabilitation:

Tailings rehabilitation: Dredging of Boac River (Bennagen, 1998, Table 13)

US1996 $114 million = US2014 $172 million (www.usinflationcalculator.com)

Socioeconomic Loss:

Present Value of Current and Future Foregone Income for 10 years = P1996 $180 million
(At a discount rate of 15%, see Bennagen, 1998, Table 7)

P1996 $180 million = US1996 $8.77 million (www.x-rates.com/historical)

US1996 $8.77 million = US2014 $ 13.23 million (www.usinflationcalculator.com)

TOTAL: US1996 $122.8 million = US2014 $185 million

Operator Placer Dome Subsidiary Marcopper Mining

"This may be the amount used in some or all of the claims filed against Marcopper by fisher folk & other private citizens (which was not sustained). These damages are clearly not about clean up and only partly about loss of the rivers other functions in the ecosystem. We have therefore treated them as an amount separate from the $100 million government suit against Placer.

Background & Summary Notes

"The banks of the Boac River still hold tall mounds of tailings that were left to continuously pump acid and heavy metals into the river after another catastrophic dam failure filled that river with mine waste in 1996. These contaminated rivers no longer support the livelihood and economic activities of nearby villages, as they once did. Placer Dome, which had managed two copper mines in Marinduque, fled the Philippines in 2001, leaving the mess behind.

In spite of a long legal struggle with competent American lawyers, on Sept. 17 Marinduque provincial administrator Eleuterio Raza told the Inquirer that Barrick had offered the province around $20 million, take it or leave it." (4)

The cleanup of mine waste in contaminated sites around the world indicates that rehabilitation on a scale that is required in Marinduque can easily run into hundreds of millions of dollars. (4)

"Numerous independent scientific studies of the ravages of mining on Marinduque, including by the United States Geological Survey, confirm the ongoing toxic impacts of uncontained mine waste and unrehabilitated rivers and coastal areas. Furthermore, numerous dams and structures have not been maintained since the mine ceased operations in 1996. Placer Dome's own consultants, Canada's Klohn Crippen, warned in a 2001 report, leaked just before Placer Dome fled the Philippines, of "danger to life and property" related to inadequate mine structures holding back waste." (4)

The incident resulted in the release of 1.6 million cubic meters of tailings along a 27km span of the river system and coastal areas near the river mouth of the island province. The impact on the river eco system was extensive. The devastating effects of the pollution on the river and costal ecosystems was of such a magnitude that a UN Assessment Mission declared the accident an environmental disaster. Boac River was left virtually dead. The onrush of tailings downstream displaced the river water, which in turn flooded low lying areas destroying crop farms and vegetable gardens along the banks and clogging the irrigation waterways to rice fields."

Oxfam, an international development and humanitarian aid agency with projects in the Philippines was approached by Marinduque community members for help. Oxfam Australia's Mining Ombudsman took their case and released a report. The report calls on Placer Dome to complete an environmental clean-up, adequately compensate affected communities, and take steps to prevent future disasters. The report updates similar findings made by the United States Geological Survey in July 2004. As of 2005 Placer Dome (which ran the mine at the time of the disaster) was the sixth largest gold mining company in the world and was listed on the Toronto Stock Exchange, but was acquired by Barrick Gold in 2006. At the time of the incident Marinduque was identified as among the 44 poorest of the 80 provinces in the Philippines

On October 4, 2005, the provincial government of Marinduque sued Marcopper's parent company, Placer Dome, for $100 million in damages.

Sources:

(a) http://www.slideshare.net/no2mininginpalawan/major-tailings-dam-disasters-in-the-philippines-alyansa-tigil-mina-atm-april-2011-7819384Philippines
(b) http://newsinfo.inquirer.net/479345/marinduque-folk-lose-case-vs-mine-firm
(c) http://opinion.inquirer.net/63421/marinduque-is-pushed-to-the-wall
(d) http://www.slideshare.net/jillentot/environmental-damages-and-health-hazards-caused-by-marcopper?related=1
(e) Bennagen, 1998. Estimation of Environmental Damages from Mining Pollution: The Marinduque Island Mining Accident, Ma. Eugenia Bennagen, Economy and Environment Program for Southeast ASIA, November, 1998

(6) **MERRIESPRUIT**

R1995 R100 million = US1995 $29 million = US2014 $46 million

Operator Harmony Gold

Despite the well documented and oft cited magnitude of loss, an entire village and many lives, and despite a judicial inquest there was no authoritative estimate of the economic value of that damage. The R100 cited above gave no source and no details and clearly is a significant under accounting of damage from a run out of this volume and length.

"Little attention was given to the environment. The identified need in this study was therefore to investigate the consequences of the disaster on the environment, a need which derives from the uniqueness of this particular disaster and its consequences. The Department of Minerals and Energy require the submission of an Environmental Management Program Report (EMPR) on all prospecting and mining operations. It is clear that, in the compilation of such an EMPR, Harmony Gold Mine neglected to establish a Management Plan to regulate the physical impact of the disaster on the environment, mainly because no attention was given to disasters in the Aide-Memoir."

Damages were estimated at R100 million (1)

The year before the disaster, a leak was reported, so all deposition was cancelled in to that particular compartment. Extra water was filtered into another compartment. Before the dam failed, the conditions were considered unsafe and unfit. The freeboard (which contained the extra water) did not have the ability to hold half a metre of extra water. But still, nothing was done. (1)

Management failures at Merriespruit:

- The inquest judge laid the blame for the disaster at the doors of the contractor, the mine, and certain of the contractor's and mine's employees. Failings of these parties that were illuminated at the inquest were as follows:
- There was no review process for the operation of the storage that involved an independent reviewer. The mine's and contractor's familiarity with the chronic problems of the storage resulted in complacency about their seriousness.
- The only involvement of a trained geotechnical engineer in the problems of the storage was that of an employee of the contractor, who became involved occasionally, only by request, and whose roles and responsibilities were ill defined.
- There were regular meetings between the mine and the contractor. However, decisions were poorly recorded, which led to confusion about responsibilities and agreed actions.
- The contractor's office at the mine did not keep the head office adequately informed of happenings at the storage. The head office was ignorant of problems and potential problems at the site and could thus not take corrective action.
- The contractor's local office was aware that water was being stored in the storage by the mine, but it took no action and did not inform either head office or seek the advice of its geotechnical engineer.
- Although the contractor had operated the storage since its inception, he had never been requested to upgrade the facilities of the storage and so bring it in line with acceptable practice, as spelled out in the industry guideline. (Chamber of Mines of South Africa 1979, 1983). Thus, the storage continued to be operated without a return-water pond. This necessitated storing water in the storage.
- Remedial measures taken to restore the stability of the northern wall were ad hoc and not the result of an adequate geotechnical investigation and design. (3)

Source:

(a) https://ujdigispace.uj.ac.za/handle/10210/7295
(b) http://floodlist.com/africa/merriespruit-tailings-dam
(c) https://books.google.com/books?id=OdFp3wKyxJoC&pg=PA453&dq=merriespruit+slimes+dam+1994&source=gbs_toc_r&cad=4#v=onepage&q=merriespruit%20slimes%20dam%201994&f=false

(7) **TENNESSEE FOSSIL PLANT**

US2008 $1,200 million = US2014 $1.3 billion

Owner Operator Tennessee Valley Authority

On December 22, 2008, a retention pond wall collapsed at Tennessee Valley Authority's (TVA) Kingston plant in Harriman, Tennessee, releasing a combination of water and fly ash that flooded 12 homes, spilled into nearby Watts Bar Lake, contaminated the Emory River, and caused a train wreck. Officials said 4 to 6 feet of material escaped from the pond to cover an estimated 400 acres of adjacent land. A train bringing coal to the plant became stuck when it was unable to stop before reaching the flooded tracks.

Originally TVA estimated that 1.7 million cubic yards of waste had burst through the storage facility. Company officials said the pond had contained a total of about 2.6 million cubic yards of sludge. However, the company revised its estimates on December 26, when it released an aerial survey showing that 5.4 million cubic yards (1.09 billion gallons) of fly ash was released from the storage facility.

The TVA spill was 100 times larger than the Exxon Valdez spill in Alaska, which released 10.9 million gallons of crude oil. Cleanup was expected to take weeks and cost tens of millions of dollars.

According to reports filed with the EPA by the Tennessee Valley Authority, the 2008 TVA Kingston Fossil Plant coal ash spill resulted in a discharge of 140,000 pounds of arsenic into the Emory River -- more than twice the reported amount of arsenic discharged into U.S. waterways from all U.S. coal plants in 2007. (1)

In April 2009, TVA Chairman Bill Sansom said the company is facing "upward pressure" on its rates, stemming from several challenges, including the Kingston coal ash spill. TVA has already spent $68 million on cleanup, and it estimates the final cost could surpass $800 million, not including fines and lawsuits. The Associated Press reported on April 11 that TVA had already spent over $20 million purchasing 71 properties tainted by the coal-ash spill and is negotiating to buy more.

Although falling fuel prices have enabled TVA to cut much of a 20 percent rate increase that took effect in October 2008, the company is considering another increase in October 2009 to mitigate these expenses. TVA will set its fiscal 2010 budget and rate changes in August.

In September 2011, it was reported that TVA estimated the total cost of the cleanup will be $1.2 billion. The utility is self-funding, so ratepayers in the seven-state region are paying the tab with higher electric bills. (1)

On August 23, 2012, U.S. District Judge Thomas Varlan ruled that “TVA is liable for the ultimate failure of North Dike which flowed, in part, from TVA’s negligent nondiscretionary conduct.” The litigation involves more than 60 cases and more than 800 plaintiffs, and will allow their claims of negligence, trespass, and private nuisance to move to Phase II proceedings, meaning each plaintiff must prove the elements of his or her respective negligence, trespass, and/or private nuisance claims by a preponderance of the evidence.

In a Sep. 27, 2010 report, TVA's inspector general Richard Moore said poor coal ash control practices and the Tennessee Valley Authority management culture led to the huge December 2008 spill. The report on the inspector general's website describes the giant spill of coal sludge laden with selenium, mercury, and arsenic as "one of the largest environmental disasters in U.S. history." TVA said the description of the event as one of the largest disasters is "not supportable." Moore refused to change. (1)

(ATLANTA – May 18, 2010) The U.S. Environmental Protection Agency (EPA) Region 4 today has approved the Tennessee Valley Authority’s (TVA) selected cleanup plan for the next phase of coal ash removal at the TVA Kingston site in Roane County, Tenn. The cleanup plan, one of three alternatives proposed to the public earlier this year, requires TVA to permanently store on site all of the ash being removed from the Swan Pond Embayment, which includes land and bodies of water adjacent to the TVA coal ash disposal area. The embayment area will then be restored to conditions that protect human health and the environment. (2)

Sources:

(a) http://www.sourcewatch.org/index.php/TVA_Kingston_Fossil_Plant_coal_ash_spill
(b) http://yosemite.epa.gov/opa/admpress.nsf/2ac652c59703a4738525735900400c2c/106c22e4bc722561852577270062c9de!OpenDocument
(c) http://www.epakingstontva.com/default.aspx

In the Dark Shadow of the Supercycle

Tailings Failure Risk & Public Liability Reach All Time Highs

August 17, 2017

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1.0 SUMMARY

In the shadow of the "supercycle",[1] and propelled by its dysfunctional economic dynamics, risk and public liability from mine tailings storage facility (TSF) failures reached all-time highs. The annual failure rate for significant TSF events escalated from a 50-year average of 0.56 "Very Serious"[2] TSF failures per year (33/50) to 1.0 (10/10) for the period of price run up 2000-2010 as described by the HHWI (Rossen 2015), a 78% increase. In this period, copper prices steadily climbed reaching an all-time post war[33] high of $9411 per tonne ($2015) in 2011 as compared to the prior 50 year average price of $5133. These facts challenge the widely held notion that failures are mainly shaped by falling prices. They point to more fundamental and still not fully examined or understood root causes of TSF failures. What is apparent in the forensic record is that the increasing severity and frequency of high consequence failures reflects in part an aging infrastructure at depleted mines which are no longer economically viable even at record high prices. There are also indications that many were never viable and were just abetted into existence through venture capital on loosely regulated exchanges or advanced on faulty feasibility studies.

The 78% increase in failure frequency and severity which occurred during the price run up of the present "supercycle" challenges the widely held notion that failures are a response to tightening economic conditions (falling prices) and points to more fundamental and still not fully examined or understood root causes of tailings storage facility failures.

What seems apparent and even widely understood within the industry, though not yet widely acknowledged, is that we have reached the outer limit of the mining industry's long standing metric that ever lower grades of ores can be mined through "economies of scale". A close examination of the data

[1] Supercycle refers to a multi-year period of sustained price increases in commodities and raw materials.
[2] We define "Very Serious" TSF failures as those involving a release of 1 million cubic meters or more
[3] The all-time annual average high since 1896 was 1916 at $13,572 ($2015) followed by 1917 at $11,876

seems to show that the "mining metric"[4] stopped working sometime in the mid 90's. It is from this turning point in the efficacy of the mining metric that conditions which evolve to catastrophic failure began forming, incubating and progressing. During the supercycle as prices climbed to all-time post war highs, grades, even at major producing large mines, dropped to all-time lows. Meanwhile, post supercycle, mines still premised on this economies of scale model continue to be put forward with unverifiable claims of economic viability, or with no demonstrated reasonable expectation of full compliance with external environmental and other community protective standards.

Post Mt. Polley, and now post Fundão, none of the much publicized government and industry studies and reforms address any of the key root causes of high severity high consequence tailings failures or commit to any changes in law and policy that will be effective in preventing the man-made losses.

The data increasingly dictate that we need a completely new approach to navigate this new era of mining. We need action to identify and correct already accrued public liability the end of this era has left behind in standing operating TSFs. This can only be accomplished through the application of comprehensive law and policy addressed to these two imperatives. The fragmented legal frameworks for mining in most prevalent use today globally have conclusively demonstrated their failure to adequately protect the public interest. Post Mt. Polley, and now post Fundão, none of the much publicized government and industry studies and reforms address the key root causes of high severity high consequence tailings failures or commit to any changes in law and policy that will be effective in preventing man made losses in existing, not yet closed marginal mines.

This paper endeavors to make that case forensically with global failures data 1916-2015 (Chambers-Bowker) in the context of assessments by leading mining analysts including Deloitte (2013), McKinsey (2015), Aguirregabiria & Luengo (2016), Ernst & Young (2015), and Price Waterhouse (PWC 2016).

Remains of Bento Rodriguez after failure of Samarco's Fundao tailings dam.

[4] The "mining metric" is higher mine production necessitated by lower grades of ore, a century of declining prices offset by declining costs per ton. The metric is to continuously develop the resource through economies of scale, larger and deeper footprints, more efficient operations, bigger and better bulk mining technology.

2.0 THE CO-ENTANGLEMENT OF SUPERCYCLE ECONOMICS & TSF FAILURES

It is irrefutable that the frequency and consequence of Very Serious Failures and of Serious[5] Failures is continuing to increase at alarming rates, that the trend emerged and grew post 1990 and that it is in large part a consequence of conscious decisions made at the mine-level to make up for fundamental mine and miner specific economic disadvantages viz. global economics. Short cuts on waste management, especially of tailings management, were and are a fast, easy, under the radar way to try to meet the high production volumes and low cash costs investors insist on (Bowker & Chambers 2015, Bowker & Chambers 2016). The dysfunctional, reactive economics of the supercycle are expertly analyzed and well characterized by Deloitte in their 2014 market trend analysis. "*In their relentless pursuit of growth in response to pressure from investors and analysts, companies developed massive project pipelines. Some also developed marginal mines, hoping commodity prices would buoy poor project economics. In their headlong pursuit of volume, many mining companies abandoned their focus on business fundamentals. They compromised capital allocation decision making in the belief that strong commodity prices would compensate for weak business practices. Rather than maintaining a long-term view of the market, many acted opportunistically.*" (Deloitte 2013).

It is irrefutable that the frequency and consequence of "Very Serious" and of "Serious" failures is continuing to increase at alarming rates, that the trend emerged and grew post 1990 and that it is in large part a consequence of conscious decisions made at the mine level to make up for fundamental economic disadvantages viz. global economics.

Price Waterhouse Coopers, looking at the performance of the top 40 over the supercycle, note that much of the massive commitment of capital to expansion and production at any cost ended up as impairment write offs: "*… from 2010-2015, the top 40 have impaired the equivalent of a staggering 32% of the capex incurred*". They note that $36 billion, or 68 % of the total impairments, were taken by Glencore, Freeport Vale and Anglo American and that "*2015 saw the first widescale mothballing of marginal projects*". The top 40 took a collective net loss of $27 billion and investors punished them for "squandering the benefits of boom" and for "poor capital management and investment decisions". (PWC 2016).

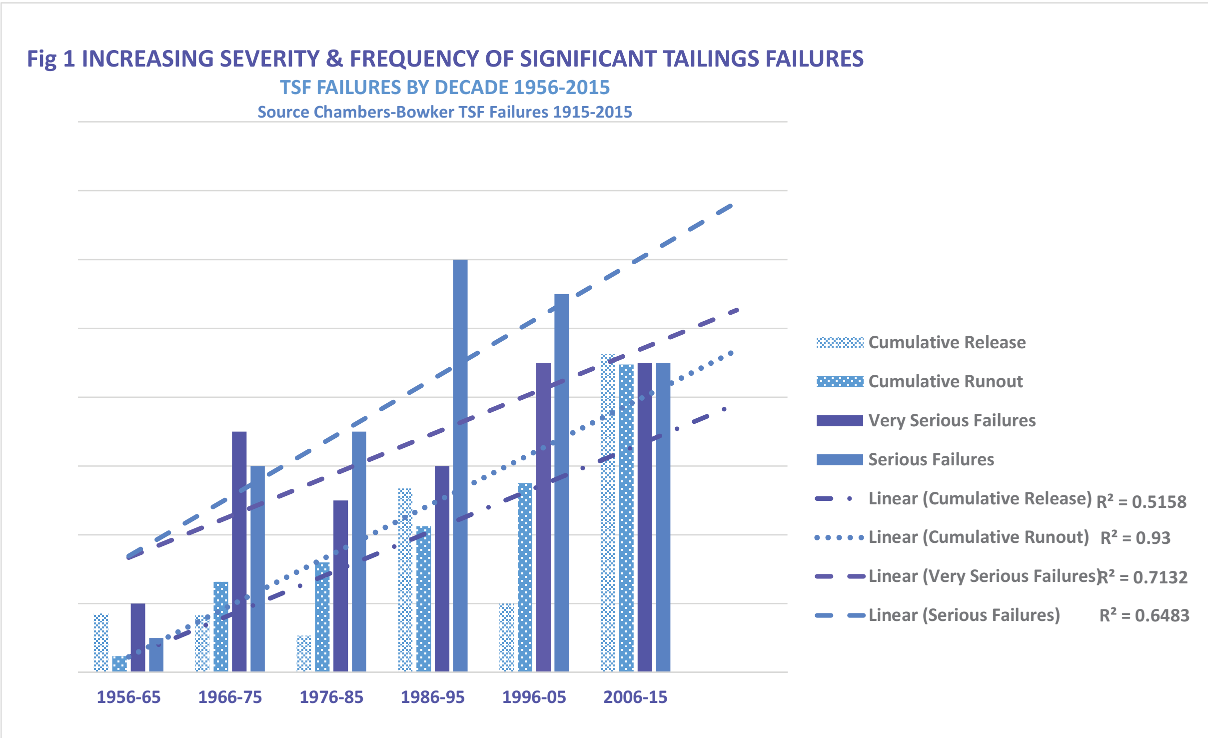
It is in this dysfunctional "maximum production at any cost" dynamic of the supercycle that we see a dramatic upturn in the frequency and severity of failures, and in which there is with very little doubt a higher global portfolio risk of accrued and unexamined public liability. As presented in Section 3, below, changes in waste rock to metals ratios for gold suggest the possibility of a more than 100% increase in the level of potential unexamined risk (SRSrocco 2016).

Immediately following we describe the degree of consequence and severity that the supercycle has directly caused and present the evidence that before the supercycle even began, the mining metric of economies of scale had begun to come unraveled and the consequence of that manifest in the early appearance of the high severity failure trend.

[5] We define "Serious" TSF failures as those with a release of greater than 100,000 m^3, but less than 1 million m^3.

3.0 DIMENSIONS OF THE SUPERCYCLE'S IMPACT ON TAILINGS FAILURE TRENDS

As graphically illustrated in Figure 1, below. The absolute number of major failures, and the severity of all failures as indicated by cumulative release, and cumulative runout per decade has steadily escalated reaching all new highs. (The present decade (2006-2015) captures the steepest part of the price run up of the Supercycle and just the beginning of the steep and sudden downward leg.) It is important to note



that the escalation of severity as measured in release volumes and run out distance for all recorded events is nearly parallel with the slope of the trend lines of both Serious and Very Serious failures. That is, the overall magnitude of all significant events is increasing and affecting ever larger surrounding areas.

TABLE 1 ANTICIPATED INCREASES IN FREQUENCY & SEVERITY 2016-2025				
Time Period	**Very Serious >1M m3**	**Serious >100k m3**	**Cumulative Release M m3**	**Cumulative Run Out km**
2006-2015 (actual)	9	9	895	92
2016-2025 (predicted)	15	15	937	110
Projected % change	+67%	+67%	+5%	+8%

Estimating major failures by proven actuarial methods (Bowker & Chambers 2015) and projecting cumulative runout and release by trend line, the overall severity profile for the coming decade, 2016-2025 (Table 1), will be 67% higher for both major failure categories and severity will reach all-time highs with more modest projected increases of 5% and 8% respectively.

Although not statistically significant by normal standards of minimum observation size, the fit to a linear trend line and the strong r-square values for both Serious and Very Serious failures and for the two severity elements shown in Figure 1 above completes the compelling and persuasive forensic evidence of increasing frequency and severity of TSF failures.

The data set on all 290 events in the failures data base is shown in Table 2 below with predictions for 2010-2020 and for 2016-2025 on a per million tonnes of Cu ore production basis. The 2010-2020 projection has increased from 11 to 13 based on the additional 5 years of failures and substantially more complete information on pre 2010 failures. Predictions for 2016-2025 are 15 for both high severity categories, an annual rate 67% higher than the 2006-2015 decade.

TABLE 2 TSF RELATED FAILURES & EVENTS BY SEVERITY 1906-2015

Decade	Dam Failures "Significant Events"			"Other Events"		Total
	Very Serious	Serious	Other	Non-Fail	Non-Dam	
1906-15	0	1	0	0	0	1
1916-25	0	0	1	0	0	1
1926-35	1	0	0	0	0	1
1936-45	1	0	7	0	0	8
1946-55	1	1	5	0	0	7
1956-65	3	1	30	0	1	35
1966-75	7	6	37	0	4	54
1976-85	5	7	36	2	2	52
1986-95	6	13	34	3	0	56
1996-05	9	11	17	0	0	37
2006-15	9	9	16	3	1	38
	=======	=======	=======	=======	=======	=======
Occurred	42	49	183	8	8	290
pred 2010-20	13	13	n/av	n/av	n/av	n/av
pred 2016-25	15	15	n/av	n/av	n/av	n/av

4.0 ROOT CAUSES OF FAILURE BEYOND PROXIMATE CAUSE

Virtually all Very Serious Failures in recorded history were preventable, either by better design or by better operational management. It is widely recognized now that "proximate cause"[6] of failure is not a matter of force majeure, unforeseeable and uncontrollable events, "black swans"[7], or ordinary human error, but a result of conscious decisions at odds with Best Practice, Best Knowledge and Best Available Technologies. Of course, the proximate cause of all TSF dam failures is geophysical and structural in nature, but the *root cause* is a failure to design, build and manage TSFs to known Best Practice, Best

[6] Proximate cause in insurance is defined as an event sufficiently related to a legally recognizable injury or loss to be held to be the cause of that injury. Here we mean looking at the precipitating final physical cause of a major failure

[7] A "black swan" is a high severity loss that results from the cumulative effect of a large number of small, unforeseeable, unpredictable events or conditions.

Knowledge, and Best Available Technology. Though few put it in these plain terms, the Mt Polley Expert Panel was very clear.

In Brazil and in British Columbia, professional practice and regulatory guidance allowed unrestrained reliance on "observational method"[8] as circmstances beyond TSF design requirements and assumptions arose.

It is widely recognized now that proximate cause of failure is not a matter of force majeure, unforeseeable and uncontrollable events, "black swans", or ordinary human error. The almost universal conclusion of all dam failure analysis focused on proximate cause is a failure to apply and adhere to known best practice and best knowledge.

The Fundão dam had serious construction flaws in the base drain and filters, concrete decant galleries were structurally deficient, operational deviations allowed structurally weak slimes to be deposited in areas where they were prohibited by the operating plan, and the dam crest was moved and constructed over these slimes causing the dam failure (Fundão Review Panel 2016).

At Mt Polley, the miner deviated from the construction design, and the review committee found the dam would not have failed if the original design had been followed, despite the undiscovered lacustrine layer (Mt Polley Expert Panel 2015).

All of the earthquake triggered failures in Chile in the 1960s were found to be associated with the prevalent use of upstream construction for TSFs in an area known to be prone to frequent, high severity earthquakes (Villavicencio et al. 2014).

With the exception of recent updates to law and policy in Australia (West Australia ,South Australia, and New South Wales), we are not aware of any other legal framework for mining that enforces a primary Best Practice/Best Available Technologies performance standard life of mine. Regulatory agencies defer to industry; do not formally adopt existing guidelines like MEND or ANCOLD (2012); and, largely depend on their own or consulting engineers without independent review to make key decisions affecting risk and viability. As the Mt Polley Expert Panel noted, the standard applied in this prevailing framework often puts economic exigencies and production schedules ahead of the public interest.

This focus on proximate cause in the autopsy of catastrophic events on the one hand and the determined avoidance of BAT or Best knowledge in law and policy sets up a dynamic where it's easy to look to short cuts on all aspects of mine waste management practice without raising any concerns on the part of regulators or investors.

It is widely acknowledged even by the industry and major industry trade groups that Best Knowledge and Best Practice and Best Available Technology will not be universally

[8] The observational method is a field response to changed conditions or conditions different from expected without study or systematic analysis. The Mt Polley Report notes "The Observational Method … relies on recognition of the potential failure modes, an acceptable design to deal with them, and practical contingency plans to execute in the event observations lead to conditions that require mitigation. The lack of recognition of the critical undrained failure mode that prevailed reduced the Observational Method to mere trial and error."

applied without a legal mandate. Neither MAC (2015) nor ICMM (2016) have adopted such standards as mandates for their members. The British Columbia Ministry of Energy and Mines (BC MEM) response to the Mt Polley Expert Panel recommendations avoided several of the main recommendations of the Mt Polley Expert Panel to the point where BC MEM requirements will not adequately protect tailings dams from future failures (Chambers 2015).

The focus only on proximate cause in the autopsy of catastrophic events on the one hand, and the determined avoidance of Best Available Technology, Best Knowledge and Best Practice in law and policy on the other, sets up a system wherein it's easy to look to short cuts on all aspects of waste management practice without raising any concerns on the part of regulators or investors. (To BC MOM's credit, they did flag the exact location of failure two years before and did press for a full buttress which was resisted and contested.) (Mt Polley Expert Panel 2015).

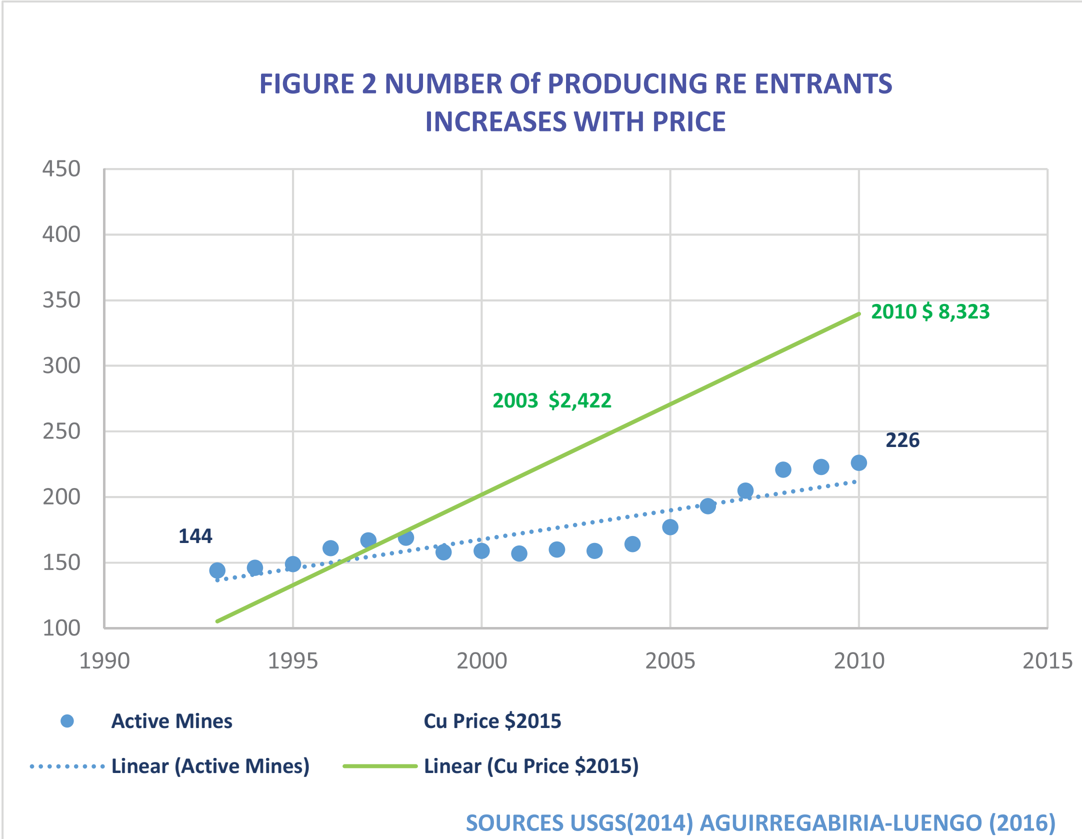
More importantly, the focus on proximate cause fails to address or understand the more fundamental root causes that result in these deviations where law does not require and enforce adherence to the application of best practices in all phases of TSF design, construction, operation, and closure, or to require expert independent review of key decisions affecting public risk and economic viability.

4.1 The Root Cause of Inherent Economic Weakness at the Mine-level

A recent study of actual annual mine records of 330 mines comprising 85% of world copper production sheds some light on the economics that may apply for all metals, and may hold keys to a deeper understanding of the relevant economic red flags of possibly incubating failure conditions (Aguirregabiria & Luengo 2015). The study reports that on average only 52% of mines were active at any time in their study period, 1993-2010 (173/330) and that 32% produced no mined output at all during the supercycle (maximum active was 226). This suggests the possibility that from 30% to 52% of all "still open" copper mines globally may not be economically feasible and cannot be expected to generate revenue sufficient to cover production costs. In many instances perhaps mines should never have been developed in the first place. Certainly no one would dispute that there are many mines that have never been profitable and that have frequently been in and out of production due to price sensitivity.

.. only 52% of "operating" {copper} mines were active 1993-2010 (173/330) ; 32% produced no mined output at all during the Supercycle. This suggests the possibility that from 30% to 45% of all "still open" copper mines globally may not be economically feasible In many instances perhaps many of these mines should never have been developed in the first place.

As Figure 2, based on the Aguirregabiria & Luengo (2015) report shows, in the run up of the supercycle the active participation among the 330 mines swelled from 144(44%) to 226(68%) (viz. an average of 173 active at any one time). It is in this increased re-entry, and often expansion of economically fragile mines (also see PWC 2016) that the trend to ever increasing severity and frequency of catastrophic TSF failures has manifested.



It is generally recognized within the industry that a widespread "cleansing" is both needed and well underway post supercycle metals price peak in 2011. Regulators meanwhile continue to avoid enforcement and duck corrections at these marginal mines hoping for a return of prices that will allow problems to be addressed out of mine revenues that are not likely to ever come again. They fear that enforcement actions may trigger bankruptcy, as occurred at the short-lived re opening of the Yellow Giant Mine. Government is not treating the widespread shedding of these marginal mines as possibly separating the deeper pockets who tried to force production out of these mines from the liabilities that may have accrued.

It is generally recognized within the industry that a widespread "cleansing" is both needed and well underway post supercycle metals price peak in 2011. Regulators meanwhile continue to avoid enforcement and duck corrections at these marginal mines hoping for a return of prices that will allow problems to be addressed out of mine revenues that are not likely to ever come again

As no known regulatory authority keeps a publicly accessible inventory of TSFs or TSF "significant events," it isn't possible to identify which of these re-entries and expansions in the supercycle are those in the failures data base. However, Mt Polley was definitely one of these, reopening in 2005 after a 4-year standby following a brief and unprofitable prior period of operation. Forensically, if not statistically, it is strongly suggested in the case records and histories that weak economics is common among failed mines. Further, it has been possible to persuasively map the connection between significant failure events and global mine economics (Bowker & Chambers 2015, Bowker & Chambers 2016), and these relationships in turn point to characteristics at the mine-level that are likely indicators of failure risk underlying what ultimately becomes the proximate cause of failure, the last event in the chain manifesting as physical failure.

4.2 The Directly Measureable Relationship between Failure Trends & Global Mine Economics

The global economic history of metallic mining is best and most frequently described with four key variables: (1) volume of metals produced from mines, (2) realized price for that volume, (3) costs to produce, and (4) grade of ore to the mill. Over the past 100 years, the key dynamic of metallic metal mining globally for all metals has been declining grades and declining prices punctuated by a few short term supercycles. As grades fell across all metals for discoveries, reserves and head grades, economic feasibility and the possibility of profit has turned mainly on the economics of ore production made possible through open pit mining. The cost to move a tonne of ore from the ground to the mill is completely independent of grade and of the ultimate price that will result.

This brings two additional key variables into play as the background economics that result in high failure frequency and severity: (1) ore production volume, and (2) the mining cost per tonne of ore.

Mine economist Richard Schodde (Schodde 2010) correctly mapped the major historic role the unit cost of ore production has played in holding the line against falling grades, and against the long term decline in prices. He calculated that while overall mine costs (C1), 1900-2010 had declined by 50% in real dollars, that when distributed over ore volume, the per-tonne of ore production cost had declined 87%. This is what made the mining metric workable and profitable for some but not all. Schodde argued that the decline in ore production costs would continue to "grow the resource" even as grades continued to fall (discovery, reserves and as milled). What the World Bank detected was the dramatically widening gap between ore production volumes and mined metals output (World Bank 2006).

This gap could also be described as declining yields on the economics side and exponential growth in wastes on the environmental side. In only 8 years from 2005 to 2013 the decline in yields for gold was 29%, from 1.68 g/t in 2005 to 1.20 g/t in 2013. On a waste rock to metals basis that translates to a 117% increase from 52 tonnes/oz to 113 tonnes/oz (SRSrocco 2014). It is to this gap of ever declining yields, and its relationship to the emerging trends of catastrophic failures that prior research (Bowker & Chambers 2015, Bowker & Chambers 2016) and this paper are addressed.

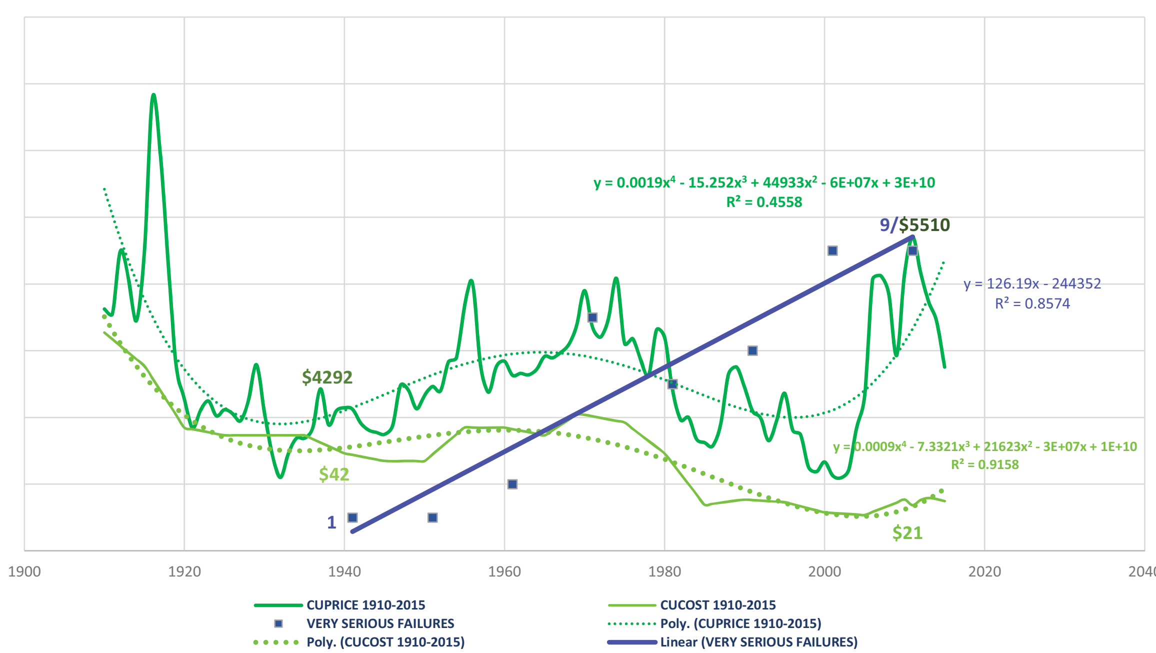
***In only 8 years from 2005 to 2013 the decline in yields for gold was 29%, which on a waste rock to metals basis translates to a 117% increase from 52 tonnes/oz to 113 tonnes/oz**. (SRSrocco 2014)*

The previously established correlations between failure severity and these five key mining economics parameters (Cu Ore, Cu Grade, Cu Metal, Cu Cost, Cu Price) is reaffirmed in failures and mine economics data as of December 31, 2015, as shown in Table 3, below.

Table 3 CHANGES IN CORRELATIONS 1940-2009 As Known July 2015 v As Known July 2016					
DATE/SEVERITY	VERY SERIOUS	SERIOUS	CUORE	CUGRADE	CUCOST
Ver Ser Jul'15	1	0.880	0.860	-0.794	-0.788
Ver Ser Jul'16	1	0.903	0.953	-0.825	-0.754
Serious Jul'15	0.880	1	0.720	-0.884	-0.682
Serious Jul'16	0.903	1	0.824	-0.843	-0.801
Sources: Bowker Mining Economics 2016 Chambers-Bowker TSF Failures 1915-2015					

What emerges with more complete data on pre-2010 failures than we had in July 2015 and the additional six years of data (2010-2015) is an interesting, new view of the relative strength of correlations in the two high severity failure categories. Ore production is reaffirmed as the most dominant but with much higher correlations with both severity categories, 0.953 for very serious and 0.824 for Serious. Grade clearly emerges as much more dominant for Very Serious Failures and copper production cost (CUCOST) emerges as much less important for Very Serious Failures and much more important for Serious Failures. Overall there is more clarity on Serious Failures, and it is now apparent they are shaped by the same forces as Very Serious Failures.

FIG 3 FAILURE TREND INCREASES DESPITE LOWER ORE PRODUCTION COSTS & EXPONENTIAL PRICE RISES

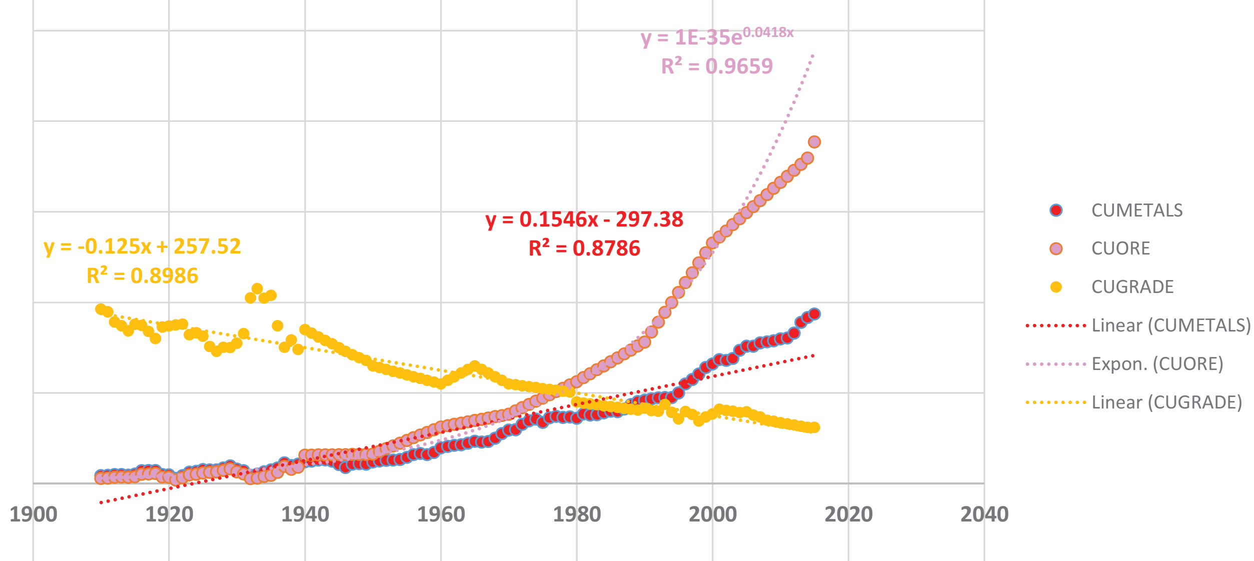


SOURCES: BOWKER CHAMBERS MINE ECONOMICS DATA BASE 2016, CHAMBERS BOWKER TSF FAILURES 1915-2015

As is clear in Figure 3, below the rising trend of Very Serious Failures emerges despite the long term offsetting effects of lower ore production unit costs that accompany the plunge in as-milled grades.

The World Bank noted this shift in the relationship between finished metals production and ore production as of 2000 (World Bank 2006). As we previously mapped, that spread continued to widen through 2009 (Bowker & Chambers 2015). In the 6 years since 2009 the spread is even more pronounced primarily as a result of an even steeper and faster decline in available ore grades that the industry neither foresaw nor prepared for. This increasing spread between metals production from mines and ore production needed to attain that level of production very clearly begins around 1990, almost a full decade before the start of the supercycle. (See Fig 4 below) A closer look at what was happening to grades (Fig 5) as prices rose over the supercycle reveals the key impetus for failure.

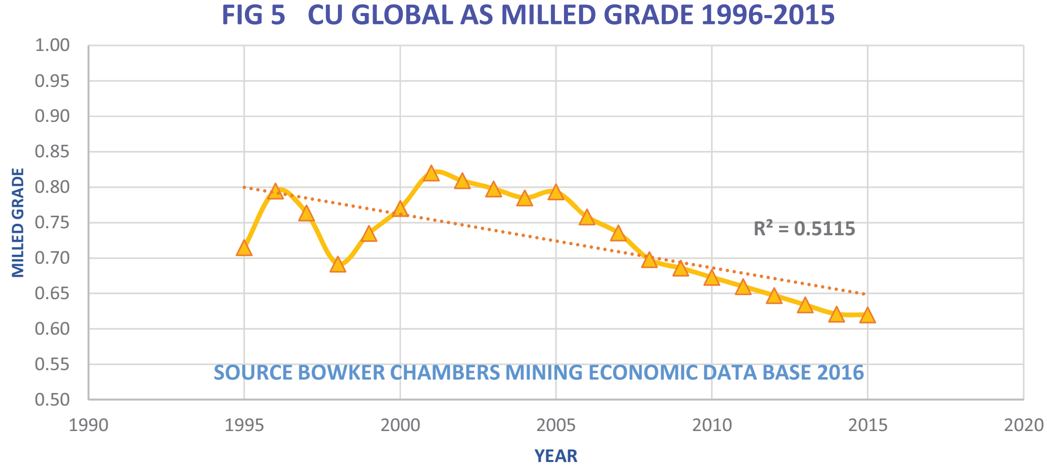
FIG 4 THE SHIFT IN THE MINING METRIC FROM THROUGHPUT TO GRADE



SOURCE BOWKER CHAMBERS MINING ECONOMICS DATABASE 2016

Over the entire period of the supercycle, as shown in Figure 5, "as milled" grades have dropped significantly, affecting not only smaller economically marginal mines but the behemoth Chilean and Top-40 producers as well.

FIG 5   CU GLOBAL AS MILLED GRADE 1996-2015



SOURCE BOWKER CHAMBERS MINING ECONOMIC DATA BASE 2016

As devised by ICOLD (ICOLD/UNEP 2001) and carried on by WISE (WISE 2016), the tailings dam failures data base captures no data on geological, geochemical or econometric descriptors of the mines with failed TSFs. The data on physical characteristics of the TSF facility (height, capacity, type of construction) and severity (run out release deaths) is sporadically reported, even for catastrophic failures. We have nevertheless been able to piece together some mine-level econometric markers on some of the mines with Very Serious Failures post 1990. The data on 7 of 18 mines with Very Serious Failures post 1996 strongly indicate that the econometric markers of these mines are significantly below global averages.

Data on 7 of 18 mines with very serious failures post 1996 strongly indicate that the econometric markers are significantly below global averages. At those which are primarily copper mines average reserve grade at failure was 0.37 as compared with a global average head grade at producing mines of 0.76. Of mines with very serious failures 1992-2010 the realized grade taking account of all metals was 1.10 as compared to 2.25 for all copper mines.

Average reserve grade as of failure for the 6 mines which are primarily copper producers was 0.37 as compared with a global average head grade at producing copper mines of 0.76. Of 7 mines with Very Serious Failures 1992-2010 the Cu equivalent grade (i.e. taking account of other metals produced or translating all metals into Cu equivalent) was 1.10 as compared to a “realized grade” of 2.25 reported by Aguirregabiria & Luengo (2015) for their 330 producing copper mines. These are imperfect and non-exact comparisons but they are also strongly persuasive that mines which produce Very Serious TSF Failures are poor performers viz. average global econometrics. This in turn suggests a significant public interest in giving independent authoritatively verified economic feasibility a specific and prominent place in mine and mine expansion approval, and in life-of-mine and life-of-facility regulatory oversight.

As a project moves to the development stage, the higher the grade, the more robust the projected economics of a project. And for a mine in production, the higher the grade, the more technical sins and price fluctuations it can survive.” (Dashkov 2013). Continuing in this analysis Dashkov goes on to declare that volume and throughput (the Scholz foundation for profitability of low grade mines) is “no longer king,” and that grade is “now king” in determining which mines will be successful and which will fail.

These adverse grade deviations at the mine-level translate to, and are determinant of, higher costs to produce, as well as of larger waste volumes per unit of metal produced.

The fundamentals of how this plays at the mine-level is simply and succinctly expressed by Andrey Dashkov, Senior Analyst, Casey Research**:** “As a project moves to the development stage, the higher the grade, the more robust the projected economics of a project. For a mine in production, the higher the grade, the more technical sins and price fluctuations it can survive.” (Dashkov 2013). Continuing in this analysis Dashkov goes on to declare that volume and throughput (the Scholz foundation for profitability of low grade mines) is “no longer king,” and that grade is “now king” in determining which mines will be successful and which will fail. This was essentially validated by Bowker & Chambers (2015) as the context and main driver in the emerging prevalence of catastrophic failures.

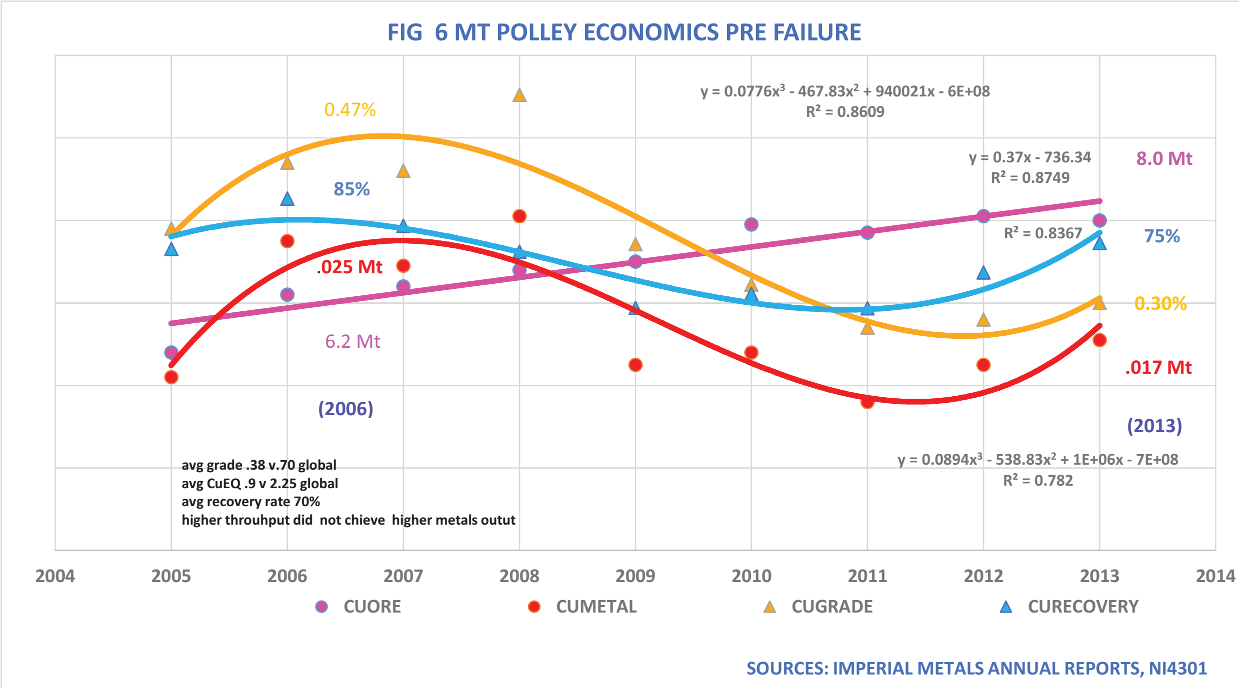
Dashkov's analysis is that a grade advantage is a critical determinant of ability to survive serious technical flubs and dramatic unpredictable price fluctuations. As a norm for all metals, this means that smaller, lower grade mines will suffer more and have more physical manifestations of their economic stress than larger, higher grade mines. Very simply, smaller, lower grade mines operated by junior and midsize miners have less cushion. They have to ride too close to the edge of financial viability viz. global metals markets and major producers to try to stay in production. They also have less access to high quality capital markets, paying more and operating under more onerous terms of credit than the top producers. This is a factor that George Ireland has frequently cited as creating financial instability and uncertainty, when the due dates of credit don't match up with cash flow needs, expected revenue generation, and production capacities of the mine. This mismatch can actually lead to failure or involuntary investor takeover elevating uncertainty and instability (Sylvester 2012).

Mine level data is strongly persuasive that mines which produce "Very Serious" TSF failures are "poor performers" viz. average global econometrics. This in turn affirms a significant "public interest" in giving "economic feasibility" a specific and prominent place in mine and mine expansion approval and in life of mine, life of facility regulatory oversight.

In gold, as a respected analyst Mark Fellows explains, a 10% fall in global average ore grade gives rise to a $50/oz. rise in average global production costs (Fellows 2010). At the mine-level, a difference between a gold mine with 1.72g/t and 2.2 g/t translates to a likely cost difference of $100/oz. in total production costs. These are the actual differences at the Gold Ridge mine, Guadalcanal, in 2009. This mine never achieved profitability, not because of political unrest, but because of the low quality of the deposit compared to the quality of ores shaping world markets. Gold Ridge, with approximately 20 million cubic meters tailings storage capacity with a long history of many owners, frequent interruptions, and continually falling recovery rates (another emerging consequence of mining very low grade ores), under ownership of landowners with limited technical competence, has hovered on the brink of complete failure by overtopping for two years (Ausenco 2007).

4.3 Further Exploration of the Dimensionality of Relationship between Failures and Global Mining Economics

If the legal frameworks for mining mandated the maintenance of public information on the tailings facilities and their larger context of mine and miner on the mines they have approved (or are reviewing), it would be possible to directly compare mine-level with global economic profiles and develop proven "failure risk" markers that might help intercept the incubation of failure conditions early enough for correction before the failure occurs. This information doesn't exist in any permitting regime we have seen. We know from the mine-level narrative of catastrophic failures that poor vetting, shoestring economics and production schedules ahead of safety were very much the key backstory at Mt Polley, which never attained "economic feasibility". From the outset it was plagued by low grades and low recovery rates. A careful reading of all annual reports and of the NI 43-101 prepared by an in-house



geologist indicates that the reopening in 2005 was based on sparse 4-year old data that was not independently verified or re-examined. Life of mine Average Cu Grade was 0.38 vs 0.70 global; higher throughput did not achieve higher metals output as recovery grades constantly were below expected. Imperial processed 29% more ore in 2013 as compared with 2006, its year of peak grade but produced 3.2% less metal. As is obvious in Figure 6 falling grades parallel metals output**.** Life of mine to failure, the Very Serious failure rate for Mt Polley is 0.011 per million tonnes of ore to the mill vs 0.0004 globally, that is 27 times higher than the global failure performance.

The amount of debt Samarco had amassed for the 2010 expansion put great weight on going forward. They did not stop to fix the Fundao dam, or to create more long term capacity onsite (Amira et al. 2010). Piecing this economic back story together for all failures into a data base has so far been impossible. But it is still possible to probe more deeply the dimensionality of the connection between failures and global economics over time at the aggregate level via canonical correlation analysis (CCA). CCA is a way of exploring whether two data sets, in our case the failures data set and the global economics data set, are independent. It can also help identify the dimensions of cross influences or common unidentified external influences (e.g. technical incompetence, brain drain, improper application of technology, geographic shifts in production advantage, excessive debt lost productivity).

Prior research on failures 1940-2009 (Bowker & Chambers 2015) utilizing CCA strongly indicated that TSF failures and copper economics data sets are interdependent, and this is reaffirmed with data through 2015 (see Data base for technical documentation). More than 95% of the total variance is explained through the two canonical variables for both the pre-2010 and pre-2015 data sets. In both, extremely high eigenvalues (0.950 and 0.854), cumulatively explain 100% of the variation. These results strongly indicate the presence of a clear and powerful correlation between failures data and economics data that is linear in nature. The results also further suggest that there are no “missing variables” (no external latent variables commonly affecting both data sets). The Wilks Lambda variables for the entire CCA model for both pre 2010 (0.011) and post 2010 (0.007) data sets are extraordinarily low, supporting the assertion that the two data sets, failures and econometrics, are not independent.

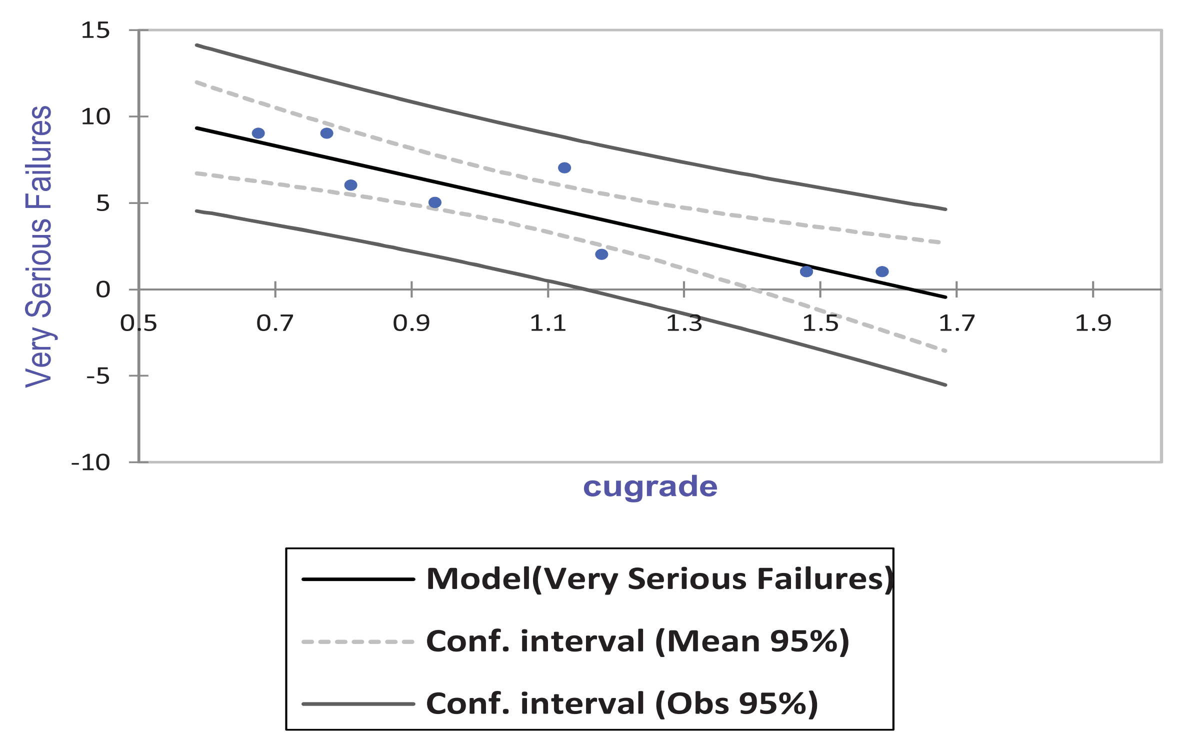
TABLE 4 VERY SERIOUS FAILURES Correlations W Canonical Variables		
	1940-2009	1936-2015
CUPROD	-0.8285	-0.9136
CUGRADE	0.6064	0.8827
CUCOST	0.3982	0.5373

What is most notable though over only 6 years (2010-2015) is the change in the composition of the canonical variables again pointing to the strong influence of grade, as show in Table 4 below.

In the canonical variable most closely associated with Very Serious Failures, the correlations with the three mining economics variables is stronger for all 3 post 2010 v pre 2010. The most dramatic change is with grade from 0.6064 pre 2010, to 0.8827 post 2010.

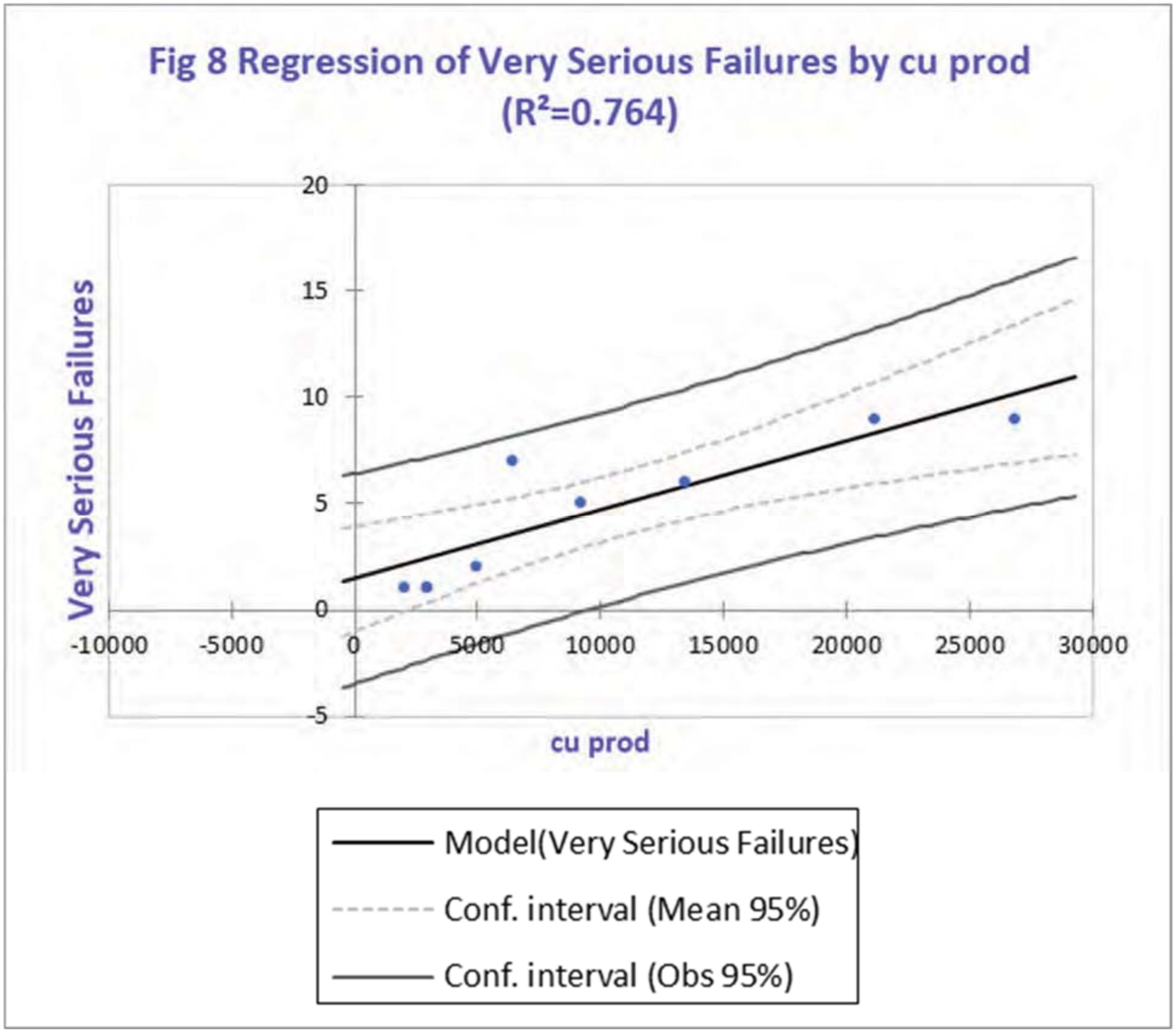
The eigenvalues imply a very strong simple linear relationship between Very Serious Failures and both grade and ore production volume.

Fig 7 Regression of Very Serious Failures by CUGRADE (R^2=0.793)



We undertook examination of these relationships through linear regression, again not to establish statistical significance but just to describe the relationships.

The regression of Very Serious Failures by grade explained 79% of the total variance as shown in Figure 7. Again, this confirms the very strong influence of global average mill grade on catastrophic failures.



The regression of Very Serious Failures by ore production volume (copper production - CUPROD), essentially tailings waste volume, explained 76% of total variance as shown in Figure 8.

5.0 CONCLUSIONS

Overlaying the supercycle autopsies of some of the world's top mining analysts onto what we previously documented in Bowker & Chambers (2015) explains the extent and nature of dysfunctions in global mine planning, development and operation that shaped what we previously had mapped and inferred from our data.

In their independent examination of the supercycle, there is a clear consensus among the world's top mining analysts that we have crossed the threshold into a new and as yet unclear era of mining. If it is understood at all, the industry its regulators and even its key investment analysts have not publicly recognized that present discovery and as milled grades have reached levels that are beyond presently known technology that had previously worked to create economic viability for low grade large scale mines. No regulatory agency known to us has recognized the need to reexamine the large scale low grade mining projects like KSM, Pebble, and Polymet that were originated in the frenzy of the supercycle on assumptions that were never proven in the first instance, and which are very clearly no longer true. No regulatory agency known to us has recognized that the supercycle was a time of pushing marginal mines and their existing infrastructure beyond design capacity and that, as at Mt. Polley and Samarco, those are practices in which failure incubates and matures.

Overlaying the supercycle autopsies of some of the world's top mining analysts onto what we previously documented in Bowker & Chambers 2015 explains the extent and nature of dysfunctions in global mine planning, development and operation that shaped what we previously had mapped and inferred from our data.

Neither the industry itself nor its regulators are taking realistic account of the implications of the fact that somewhere between 1/3 and 1/2 of all technically "operating" mines are no longer economically viable or never were viable. Such a high incidence of stranded assets does not indicate wellness for the industry as a whole. Regulators passively stand by while the wholesale dumping of these mines assuming that production will resume, that jobs will be retained, and that new revenue will finance identification and correction corrections of any potential flaws in infrastructure aggressively pushed into production levels beyond planned capacity. These are not assumptions supported by available data or expert economic analysis.

There is a clear consensus among the world's top mining analysts that we have crossed the threshold into a new and as yet unclear era of mining. If it is understood at all, the industry its regulators and even its key investment analysts have not publicly recognized that present discovery and as milled grades have reached levels that are beyond presently known technology that had previously worked to create economic viability for low grade large scale mines.

There isn't enough data to say what % of these no longer viable mines have TSF's large enough to cause catastrophic failure, but we have confidence in our prediction methods which accurately predicted the 9 very serious failures 2006-2015. We have confidence that the fall out of the supercycle dysfunctions will manifest in higher than previously expected Serious and Very Serious Failures. The data and our proven method of prediction tell us that the expected number of high severity failures is greater (13) than previously estimated for the decade 2010-2020, and that we can expect a record high of at least 15 in each high consequence category for 2016-2025.

The eminent eloquent and wise Dr. Dirk Van Zyl, who served on the Mt Polley expert panel, characterized Bowker & Chambers 2015 as "unnecessarily alarmist". That characterization is proven wrong by what we now can all more clearly see as a significantly elevated and not fully examined global portfolio risk of failure. History itself proves that characterization wrong. We had pieced together a patchwork quilt of costs and legal judgments on post 1990 Very Serious failures predicting $6 billion in 11 Very Serious failures 2010-2020. Samarco alone has damages that exceed that hobbled together estimate by at least 3 fold from a TSF with only a capacity of only 60 M m3. We now reasonably anticipate 13 not 11 Very Serious failures and an additional 13 Serious Failures based on actual ore production volumes and compilation and reconciliation of independent expert predictions post 2015.

Portfolio public liability risk is not going to simply self-correct to less elevated levels.

Nether MAC nor ICMM nor any mining jurisdiction we are aware of has undertaken any reforms that will be effective in lowering public liability portfolio risk.

In Risk Management we live by that old adage "an ounce of prevention is worth a pound of cure". Waiting for revenue that will never come to fix broken and no longer serviceable infrastructure is not in the public interest. It offers neither prevention nor hope of cure for whatever already formed catastrophic losses are maturing to final event.

Continuing to advance and tout mega scale low grade projects conceived in the supercycle and on the basis of its cowboy economics offers no reform, no future with better outcomes.

Regulators have clearly chosen protection and support for the mining industry over reducing public risk and public liability. That and past long standing issues of enormous gravity have brought a loud public backlash in anti-mining anger in the form of extreme and reactive legislation with outright complete prohibitions on all metallic mining , bans on open pit mining, bans of varying degrees on all upstream construction and in the case of Maine, a state with only two mines ever, both failures with a high unresolved unfunded public consequence, a tantamount complete ban in the form of a requirement to prefund in cash an independently assessed worst case scenario.

Regulators have clearly chosen protection and support for the mining industry over reducing public risk and public liability. That and past issues of long standing gravity have brought a loud public back lash in anti-mining anger and disgust in the form of extreme and reactive legislation. If regulators and the industry do not address themselves more actively to pubic risk and public liability than they have done to date 3 years after Mt Polly and two years after Samarco, it is reasonable to expect that elevated public outrage will spawn more of these public opinion driven reactionary extreme anti mining proposals.
While all that unfolds as it will, our data say the public liability risk continues to elevate and the consequence of failure continues to grow.

If regulators and the industry do not address themselves more actively to pubic risk and public liability than they have done to date 3 years after Mt Polly and two years after Samarco , it is reasonable to expect that elevated public outrage will spawn more of these public opinion driven reactionary extreme anti mining proposals.

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Perspectives in Ecology and Conservation 15 (2017) 145–151

ABECO ASSOCIAÇÃO BRASILEIRA DE CIÊNCIA ECOLÓGICA E CONSERVAÇÃO

Perspectives in ecology and conservation

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Essays and Perspectives

Fundão tailings dam failures: the environment tragedy of the largest technological disaster of Brazilian mining in global context

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ARTICLE INFO

Article history:
Received 9 December 2016
Accepted 3 June 2017
Available online 22 July 2017

Keywords:
Iron ore
Post-disaster management
Ecosystem collapse
Large-Scale Mining Industry
Doce River

ABSTRACT

After the collapse of the Fundão dam, 43 million m^3 of iron ore tailings continue to cause environmental damage, polluting 668 km of watercourses from the Doce River to the Atlantic Ocean. The objectives of this study are to characterize the Fundão Tailings Dam and structural failures; improve the understanding of the scale of the disaster; and assess the largest technological disaster in the global context of tailings dam failures. The collapse of Fundão was the biggest environmental disaster of the world mining industry, both in terms of the volume of tailings dumped and the magnitude of the damage. More than year after the tragedy, Samarco has still not carried out adequate removal, monitoring or disposal of the tailings, contrary to the premise of the total removal of tailings from affected rivers proposed by the country's regulatory agencies and the worldwide literature on post-disaster management. Contrary to expectations, there was a setback in environmental legal planning, such as law relaxation, decrease of resources for regulatory agencies and the absence of effective measures for environmental recovery. It is urgent to review how large-scale extraction of minerals is carried out, the technical and environmental standards involved, and the oversight and monitoring of the associated structures.

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Introduction

One year after the collapse of the Fundão tailings dam, more than 43 million m^3 (Samarco, 2016a) of iron ore tailings are still causing environmental damage, polluting 668 km of watercourses from the Doce River Basin to the Atlantic Ocean. The volume of pollutants and the extent of ecosystems affected have assumed unprecedented proportions, involving the Brazilian Atlantic Forest – one of the world's biodiversity hotspots (Mittermeier et al., 2005), estuarine, coastal and marine environments. Furthermore, it affected other priority areas for the conservation of biodiversity and cultural heritage (MMA, 2007), such as the iron geosystems of the Quadrilátero Ferrífero (Carmo and Kamino, 2015).

In less than 48 h after the collapse of the Fundão dam, a task force was created by the Attorney General's Office of the State of Minas Gerais, whose main objective was to ascertain the facts of the environmental tragedy and the repercussions on the 17 districts and 36 municipalities directly affected by the mud wave. The Minas Gerais State Public Prosecutor's Office (MPMG) worked intensively and produced, through related teams, hundreds of technical documents and expert reports. Among these studies, the 120 technical documents of the Prístino Institute (inspection reports, technical reports, studies and maps), developed in partnership with the MPMG Geoprocessing Nucleus, identified the main environmental damage, which supported most of the results of the present work (CAOMA, 2016).

The mining company Samarco Mineração S/A (a joint venture between Vale S/A and BHP Billiton), responsible for the Fundão dam, produced technical documents stating the main emergency measures adopted and the initial planning for environmental recovery. According to the company, the emergency recovery work

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http://dx.doi.org/10.1016/j.pecon.2017.06.002
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prioritized the remaining structures of the dams of the mining complex and, after that, tailings containment dams were constructed within the limits of its properties (Samarco, 2016a,b). Up to the present moment, priority has been given to the recovery of the physical environment. However, a federal agency responsible for inspection and monitoring for the purposes of environmental quality recovery, evaluated the inadequacy and inconsistency of the data presented. It stated that the actions performed by the company are still insufficient to guarantee the reduction of the damage caused by the tailings, resulting in 13 notices of infraction and an environmental fine (IBAMA, 2016a).

The main objectives of the study, based on this scenario, are: detail the structural features and possible structural failures that led to the collapse; improve the understanding of the scale of the disaster from the detailed measurement of the damage caused to ecosystems, protected areas, real estate and cultural heritage; compare and highlight this disaster in the context of global tailings dam failures; detail cases of the post-disaster actions and reflect on whether lessons had been learned about Brazilian tailings failure.

Fundão Tailings Dam: structural features and environmental damage

Structural features

The Fundão dam was one of the megastructures of the Germano mining complex, located in the municipality of Mariana, Minas Gerais, southeastern Brazil. The mining complex had an installed capacity of 23 million tons/year of iron ore concentrate. In addition to Fundão, the complex contained two more dams: Santarém and Germano, the latter being the highest dam in Brazil, with a height of 175 m and a projected volume of up to 160 million m^3 of tailings (Samarco, 2013). Open pit mines, piles of sterile material deposits, industrial plants and pipelines are also part of the Germano Complex.

The Fundão dam began operating in 2008 and was designed to contain a total of 79.6 million m^3 of fine tailings (mud) and 32 million m^3 of sandy tailings during its 25-year lifespan (SUPRAM, 2008). In November 2015, Fundão contained 56.4 million m^3 of iron ore tailings deposited in merely seven years of operation, a result of the never-before attained records of Brazilian production in the years 2013 to 2015 (IBRAM, 2015). In order to accommodate this volume, it was necessary to construct dikes, using the sandy reject itself as a construction material from the upstream embankment method (Ávila, 2012).

Unforgiving structures

Among the most common methods of tailings disposal, the one with the greatest economic advantage is the upstream embankment. However, it poses a significant challenge to the geotechnical engineer, due to the fact that water is the primary instability agent. Indeed, dams using the upstream embankment method are considered "unforgiving structures" and represent up to 66% of the worldwide reported mine tailings dams failures (Rico et al., 2008; Ávila, 2012; Kossoff et al., 2014).

According to Prieto (2014), the main disadvantages and restrictions of this technology are: foundation of later lifts is on unstable tailing slime, unused in earthquake zones; high level of monitoring using instrumentation required during operation; and recommendation that the rate of raised tailings dams be, preferably, no more than 5 m/year.

Since the beginning of the operation, in 2008, the Fundão dam had presented several anomalies related to drainage construction defects, upwellings, mud and water management errors and saturation of sandy material. In some situations, emergency measures were implemented (Samarco, 2016b), one of them known as retreat of the dam axis was begun in 2013. According to Samarco (2016b), the retreat represented: ". . . a temporary solution, it was decided to realign the dam on the left shoulder by moving it behind the section of the gallery to be filled with concrete, in order to allow the continuation of the landfill embankment. (. . .) The retreat would move the crest closer to the water of the reservoir and the mud contained within, but it was anticipated that the dam would quickly return to its original alignment once the buffering operations were done."

However, the retreat was maintained until the collapse of the dam. According to Samarco (2016b): "As the dam embankment continued, surface upwellings began to appear at the retreat of the left shoulder at various elevations and on various occasions during 2013. The saturated mass with sandy tailings was growing, and in August 2014, the drainage carpet controlling this saturation reached its maximum capacity. Meanwhile, the mud under the landfill was responding to the increase in the load that was being deposited by the embankment. The way in which it responded, and the consequent effect on the sands, was what finally made the sands liquefy."

Technical reports on the Fundão disaster (Samarco, 2016b) concluded that the collapse was due to liquefaction of the material, a phenomenon that occurs when solid materials (sandy tailings) lose their mechanical resistance and present fluid characteristics. Basically, the disaster occurred because of some key factors such as: structural damage to the starter dike, resulting in increased saturation; the attempt to solve structural problems with a concrete gallery that caused the axis of the dam to retract (Fig. 1), later being raised on mud; and the unforeseen deposition of sludge in critical regions. In upstream embankment dams, it is essential for the stability of the structure that the deposition of unsaturated sandy tailing create a beach, at least 200 m wide, immediately upstream of the dam crest.

Environmental and cultural damage

The total collapse of the Fundão tailings dam took place on 05 November 2015, between 3:00 pm and 4:00 pm. About 43 million m^3 of tailings (80% of the total contained volume) were unleashed, generating mud waves 10 m high, killing 19 people and causing irreversible environmental damage to hundreds of watercourses in the basin of the Doce River and associated ecosystems (Samarco, 2016b).

Most of the tailings (>90%) remained along the 120 km stretch between the Fundão dam and the hydroelectric power plant reservoir Risoleta Neves (UHE-RN), located in the municipalities of Rio Doce and Santa Cruz do Escalvado (Samarco, 2016a). The tailings remained in the Doce River channel, downstream of the UHE-RN along 548 km, reaching the Atlantic Ocean. Forty downstream municipalities were affect and hundreds of thousands of people (included indigenous) were left without access to clean water (Neves et al., 2016; IBAMA, 2015b). Therefore, in this study, the environmental damage was grouped into two sections: one upstream and the other downstream of the UHE-RN.

High resolution orthorectified satellite images (spatial resolution of 50 cm) were used to identify the environmental damage caused by the mass displacement along the 120 km upstream stretch of the UHE-RN. Two moments were compared: (1) a mosaic of images obtained prior to the dam burst (World View-2, World View-3 and GeoEye); (2) images obtained after the dam burst (World View-2, Pléiades and World View-3). The images were imported into the Geographic Information System and the elements were converted into vectors overlapping the area stained by the tailings. Information for the 548 km downstream stretch

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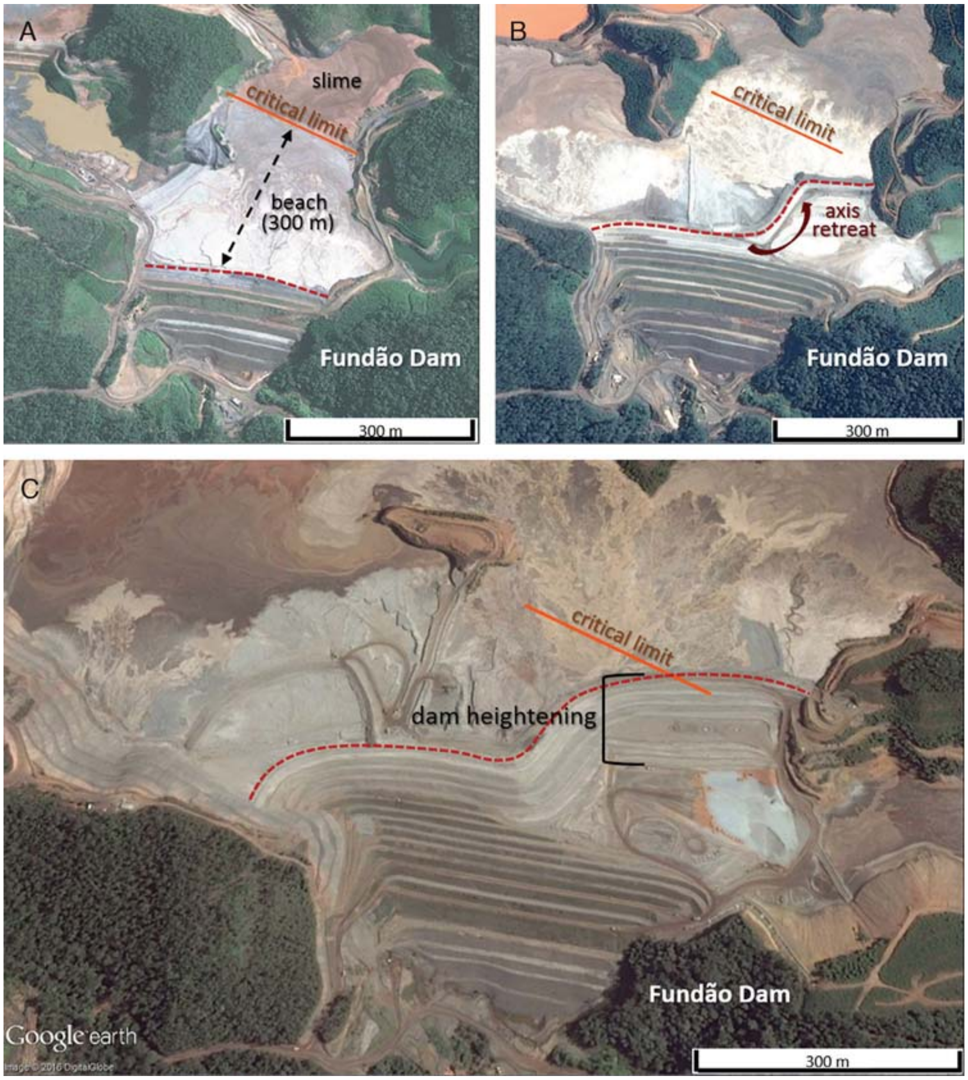

Fig. 1. Fundão dam. (A) Formation of sandy tailings beach 300 m wide and upstream mud deposition, highlighting the critical limit of contact between sandy tailings and sludge (orange line), image of 2011. (B) Retreated axis for emergency works in a concrete gallery brings the crest closer to the critical limit of contact between sandy tailings and sludge (red dashed arrow), image of 2013. (C) Embankment of the dam displaced axis in the critical limit of contact between sandy tailings and sludge (orange line), 2015 image. Adapted from *Google Earth Pro*.

of the UHE-RN, was obtained from technical reports produced by regulatory and inspection agencies and available literature.

Over a span of only 12 h, along the 120 km stretch between the Fundão dam and the UHE-RN, the mass displacement created a patch of 2020 ha and the tailings accumulated in channels, in floodplains and in the UHE-RN reservoir. This UHE-RN has not yet resumed electric power production due to the huge volume of tailings deposited in the reservoir, around 10 million m^3.

The tailings directly hit 135 identified semideciduous seasonal forest fragments, in a 298 ha of vegetation suppression, located on the banks of Gualaxo do Norte and Carmo Rivers and its tributaries. The tailings also directly hit 863.7 ha of Permanent Preservation Areas associated to watercourses, which were in protected areas, as defined by the federal forest code. Santarém Stream (11.9 km impacted), Gualaxo do Norte River (68.4 km) and Carmo River (24.7 km) were the main rivers and streams completely silted by the tailings. In addition, 294 small creeks were affected by the tailings (Fig. 2 and Fig. S1). Little attention has been given to the pollution potential of the tons of chemical compounds (flocculants and coagulants), specifically sodium hydroxide, which spilled out along with the tailings.

Out of the 806 buildings directly hit by the tailings, at least 218 were completely destroyed. These were residences, public buildings, commercial real estate, centennial churches and ancient farms distributed among 10 districts of five municipalities: Mariana, Barra Longa, Ponte Nova, Santa Cruz do Escalvado and Rio Doce. Bento Rodrigues, just 6 km from the Fundão dam, was the most damaged district with 84% of the affected buildings totally destroyed, followed by Paracatu de Baixo (40% of the buildings hit were destroyed). A total of 21.1 km of rural roads, 12 bridges/passages and the small hydroelectric plant of Bicas were also damaged.

Areas of cultural heritage also suffered greatly. Damages include, at least two archeological sites, six places of historical and cultural interest, more than 2000 sacred pieces/material heritage, five caves, a 2.2 km stretch of the Estrada Real and preserved areas of the landscape complex in the junction of the Carmo and Piranga Rivers and the urban complex of Bento Rodrigues. One of the main cultural heritage assets irreversibly affected was the São Bento chapel, an 18th-century building surrounded by stone walls (Fig. 3 and Fig. S2).

Parts of three tourist routes (*Estrada Real, Estrada Parque Caminhos da Mineração and Caminho de São José*) were also severely

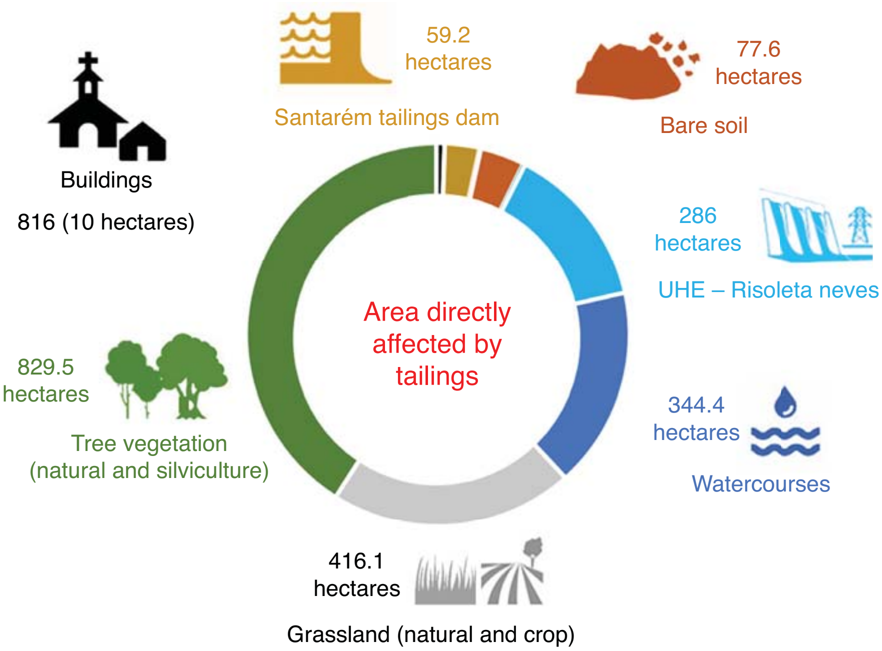

Fig. 2. Elements of natural and cultural heritage damaged by the Fundão tailings dam.

Fig. 3. Buildings affected by the Fundão tailings dam: (A) District of Bento Rodrigues, Mariana and (B) Urban area of the municipality of Barra Longa.

impacted causing losses to the local economy. The mud affected, irreversibly, areas of important archeological and speleological potential had not yet been studied, made it impossible to evaluate the exact impact on the loss of scientific knowledge.

The flood plains favored a larger accumulation of tailings (Fig. S3), on average more than 50 cm high, and in some places estimated at more than 3 m thick (Samarco, 2016a; IBAMA, 2016b). The mass displacement was so intense that it excavated the soil and altered original river beds. The tailings stain damaged four protected areas: APA Barra Longa, APE Ouro Preto/Mariana and the Biosphere Reserves of the Espinhaço Mountains and Atlantic Forest (UNESCO, 2011). There was also damage to Priority Areas for Biodiversity Conservation (MMA, 2007; Drummond et al., 2005) – named Quadrilátero Ferrífero and Florestas da Borda Leste do Quadrilátero – and in key areas for the conservation of six rare Brazilian plants (*sensu* Giulietti et al., 2009), named SE204 – Ouro Preto.

Considering that the disaster occurred in one of the most important regions for biodiversity conservation, it is estimated that the loss was significant (Fernandes et al., 2016). Tons of fish from 21 different species died in large numbers (IBAMA, 2015a). Isolated reports have identified the death of large mammals, such as the South American tapir (*Tapirus terrestris* L.), as well as turtles, birds, amphibians and invertebrates. However, no study has been published by the scientific community to account for long-term effects in main ecological components and populations of endemic species of flora and fauna.

Along the Doce River, downstream of the UHE-RN, about 5.5 million m^3 of tailings were deposited in the first days after the disaster (Samarco, 2016a; IBAMA, 2016a). Very fine tailing particles caused severe changes to the physico-chemical characteristics of the Doce River and estuarine region, increasing the turbidity levels in Minas Gerais up to 6000 times (600,000 NTU) higher than the upper limit established by law for this parameter (SEMAD, 2015). However, the impact monitoring conducted by Samarco presented very low volume of data and several physical and chemical parameters were not reported. This situation was a consequence of using inadequate methods for monitoring impacts (IBAMA, 2016c,d,e).

Three different types of sediment layers from the tailings (IBAMA, 2015a, 2016c,d,e) were detected at the mouth of the Doce River: a thick sediment deposited along the mouth; a plume deposited on the bottom; and another thinner widespread plume on the surface (floating plume). Some early projections predicted the plume would have little impact and the pollutant material would dissipate in a few months (Puff, 2015). A year after the dispersion started, 170 km of beaches were contaminated by mud, 110 km to the north of the Doce River mouth and 60 km to the south. The plumes have already spread over more than 770 km^2, having a huge effect on protected coastal zones such as the Comboios Biological Reserve, an important place for spawning of sea turtles, Santa Cruz Wildlife Refuge and APA Costa das Algas. In the

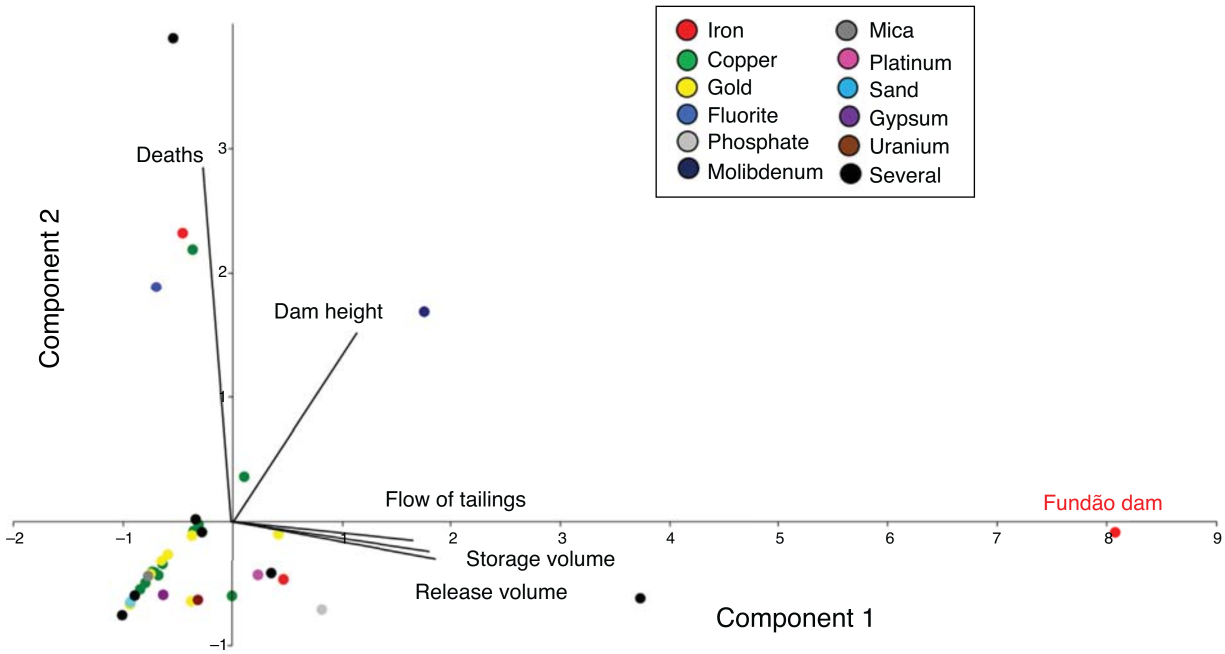

Fig. 4. Distribution pattern of 36 cases of dams disasters (colors identify the types of tailings, according to the processed ore) based on structural characteristics, distance and volume of tailing releases, and deaths by Principal components analysis. See Table S1 for details of 36 cases.

long-term, the pollution plumes could reach regions near the city of Rio de Janeiro (IBAMA, 2015a, 2016c,e; Marta-Almeida et al., 2016).

The tragedy of the largest technological disaster of Brazilian mining in the worldwide context

Based on a survey of 308 cases of mining dam collapses in the world (1915–2016, see Table S1), the Fundão dam disaster can be regarded as the largest technological disaster, considering the volume of tailings released and the geographical extension of environmental damage. The volume of tailings released by collapse of Fundão (43 million m^3) is the largest ever registered. It is followed by the one in the Philippines/1992 (32.2 million m^3); Canada/2014 (23.6 million m^3 of gold and copper residues); and Philippines/2012 (13 million m^3 of copper residues). The extent of the damage caused by Fundão is the largest ever recorded with pollutants spread along 668 km of watercourses. It is followed by the one in Mexico/2014 (420 km of contamination by copper residues), Bolivia/1996 (300 km contamination by lead-zinc residues) and Canada/1990 (168 km contamination by uranium residues).

When compared with the cases registering the highest number of deaths, the case of Fundão (19 deaths) is the ninth most serious cases of the last century. The three disasters that caused the greatest number of deaths were: Bulgaria, 1966, lead-zinc tailings (488 deaths); Chile, 1965, copper tailings (300) and China, 2008, iron tailings (277).

We used principal components analysis (PCA) to understand the distribution pattern of the dams based on structural characteristics and the main damage caused by such collapses (Table S1, Fig. 4). This study used information from about 36 cases of collapses (12% of global cases), which presented data related to the parameters: dam height, storage volume, released volume, tailings flow distance and deaths. The first two variables are technical characteristics related to the national dam safety policies (Brasil, 2010) for risk potential assessment, and the other variables represent the extent of effective damage (Azam and Li, 2010; Kossoff et al., 2014).

The first two axes of the PCA explained 53.8% and 21.9% of the variation, respectively. The variables that best explained the distribution of data in component 1 were released volume (0.96), storage volume (0.86) and tailings flow distance (0.83). The analysis indicates that most collapses are related to copper mining (11 dams), gold (7 dams) and iron mining (3 dams). Similar features keep most collapses grouped in the scatter plot (Fig. 4).

These cases exhibited distinct characteristics that make it the world's largest mining environmental disaster. However, the collapse of the Fundão dam was the most devastating and could be classified as the largest technological disaster in the context of global dam failures.

Post-disaster management

After carrying out measures for minimizing post-disaster risks, a stage involving emergency engineering work, tailings/sediments removal is considered essential and the most frequent action adopted in events of disasters with mass displacement, including collapses of mining tailings dams (UNESCO, 2010; Kossoff et al., 2014; Bowker and Chambers, 2015).

Two examples of cases of the post-disaster management to remove tailings released by the collapse of the dam occurred in Spain and in Hungary. The collapse of the dam in Andalusia, Spain, in 1998 (UNEP, 2001) released more than 2 million m^3 of zinc tailings containing sulphide-related trace elements (As, Cu, Pb, Zn and Ca) and more than 4 million m^3 of acidic water, which were deposited over 45 km of channels and floodplains on the Guadiamar and Agrio Rivers. A plan for cleaning was presented to authorities three days after the disaster with the adoption of a cleaning protocol of the affected areas (Ginige, 2014). In 12 months, about 7 million m^3 of tailings and contaminated materials that had accumulated in the river channels, floodplains and on infrastructure works were removed. The material removed was then deposited in the exhausted mining pit of the company responsible for the dam (WWF, 2002). Another case occurred in 2010 in Ajka, Hungary. Damage control action was taken after the release of over 600,000 m^3 of tailings in the environment over 14 km. The red mud was collected along the affected areas and disposed of inside a dam, which had already been reconstructed, seven days after the collapse

(Kátai-Urbán, 2010). In the medium and long term, an extensive cleaning of debris and materials dragged by the flow of tailings was conducted (Jávor and Hargitai, 2011).

Compared with cases presented, one year after the Fundão tragedy, Samarco has conducted only 0.17 million m^3 clean-up actions in urban area of Barra Longa (Samarco, 2016c), and not yet removed, monitored or properly disposed of tailings deposited in rivers, streams and flood areas. This goes against the premise of total removal of tailings in rivers affected supported by federal government. According to this premise, the company Samarco should evaluate each area regarding the possibility of total removal and proper disposal of tailings, employing alternative treatment techniques. Tailing management should only be considered as a second alternative when technical infeasibility to remove it is proven (IBAMA, 2016b).

As a wide-ranging strategy of landscape recovery (UNEP, 2001; Hudson-Edwards et al., 2003), the tailings removal should be a priority action for the regions affected by the Fundão collapse. The tailing is a source of fine inhalable particulate material, composed of minerals such as hematite, martite, magnetite and goethite. Studies show that prolonged inhalation of particulate material originated from iron mining is associated with the increase in cases of respiratory and cardiovascular diseases (Braga et al., 2007; Gomes et al., 2011). Leaching tests and toxicological bioassays performed in the region of Bento Rodrigues suggest that the tailings and contaminated soils represent a potential risk of cytotoxicity and cellular DNA damage, due to the indication of a high potential of mobilization of elements such as iron, aluminum, manganese and arsenic from the tailings into the water (Segura et al., 2016). Veronez et al. (2016) also indicate genotoxic and biochemical effects induced by iron ore, from the experimental studies which indicated the Fe and Mn accumulation can induce oxidative stress during the metamorphosis of *Lithobates catesbeianus* (L.) tadpoles.

Lessons learned?

In 2013 the Public Prosecutor's Office prepared a statement, based on a technical report from the Prístino Institute (Greenpeace, 2015), expressing concern about the risks of revalidating the Operational License of the Fundão Dam. In the statement, the Public Prosecutor's Office of the State of Minas Gerais requested that the environmental licensing body demand that Samarco carry out the following actions: perform periodic geotechnical and structural monitoring of the dikes and dam, with a maximum interval of one year between samplings; present a contingency plan in case of risks or accidents, especially in relation to the community of Bento Rodrigues, a district of the municipality of Mariana, MG; and perform rupture analysis (DAM – BREAK) of the dam, expected to be delivered to SUPRAM (Regional Superintendence of Environmental Regulation).

After the tragic environmental disaster caused by the collapse of the Fundão dam, several articles addressed a setback in the environmental legal regulations. Law relaxation, the decrease of resources for regulatory agencies and the absence of effective environmental recovery measures were often mentioned (Fearnside, 2016; Fernandes et al., 2016; Garcia et al., 2016; Wanderley et al., 2016).

In addition to the legal regulation, the case of the collapse of the Fundão dam made clear that the structures built using the upstream embankment method, widely used in Minas Gerais and Brazil, bring several environmental and social risks, which are no longer acceptable as management techniques to deal with mining waste and residues.

The Brazilian legal system includes principles that demand that entrepreneurs as well as environmental licensing bodies adopt the Best Available Technologies (BAT) to protect the constitutional rights to an ecologically balanced environment for the present and future generations, under art. 225 of the CR/1988 c/c art. 2nd and 4th of Law 6938/1981 (Loubet, 2015).

In Brazil, the dimension of the risk generated by the method in question inspired the NBR 13028, of the Brazilian Association of Technical Standards (ABNT), which deals with the "development and presentation of dam projects for the disposal of tailings, sediment containment and water reservation". This standard states the conditions required for the development and presentation of a project of tailings disposal, in dams and in mining, to comply with conditions of safety, hygiene, functionality, economy, abandonment and minimization of impacts to the environment, within the legal standards. In its 1993 version, item 4.2, clearly states: "the construction of dams using the upstream embankment method is not recommended". For reasons unknown, in the 2006 version of the same standard, this recommendation no longer appears. The Public Ministry, to guarantee that the construction of dams using the upstream embankment method be avoided, filed a legal action against the State of Minas Gerais to prevent the public administration from granting or renewing environmental licenses for this type of dam structure.

In July 2016, a bill was presented by popular initiative to the Legislative Assembly of the State of Minas Gerais establishing safety standards for mining tailings dams. The draft dealt with the improvement of dam risk management. In October 2016 the bill was already being considered in the State Legislature under PL number 3695/2016.

World disasters caused by mining tailings are closely related to the increase in demand for mineral commodities by global markets, leading to a high rate of disasters occurring in a period of 24–36 months after a soar in overall prices (Davies and Martin, 2009), which was exactly the case of Fundão. Bowker and Chambers (2015) highlighted the recent increase in the rate of severe and very serious disasters caused by tailings dam failures and argued that this trend is a consequence of modern technologies that allow the exploitation of reserves with even smaller ore concentrations. This situation results in a huge increase in the storage capacity of mining tailings dams (Wanderley et al., 2016). Bowker and Chambers (2015) estimate society's billion-dollar costs related to disasters caused by tailings dams and highlight the urgent need for changes in regulatory systems to fit this global trend.

Apart from its sheer magnitude, the collapse of Fundão is the seventh such case that has occurred in Minas Gerais alone since 1986 (Felippe et al., 2016). There is an undeniable need to review environmental standards, for more rigorous control of the hundreds of mining tailings dams in Brazil. In addition, it is fundamental to review how large-scale mineral extraction is carried out, as well as to encourage the use of alternative technologies for the disposal of tailings such as disposing of them in abandoned caves or dewatered stockpiling (dry stacking) for tailings disposal (Gomes et al., 2016). These measures may contribute to minimize the conversion of new, natural areas into megastructures to contain tailings, and to avoid the potential risk of environmental damage to creeks and rivers and associated ecosystems.

Appendix A. Supplementary data

Supplementary data associated with this article can be found, in the online version, at doi:10.1016/j.pecon.2017.06.002.

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Re: Comments on the Final Environmental Impact Statement (FEIS) NorthMet Mining Project and Land Exchange

From: Center for Science in Public Participation

The Center for Science in Public Participation provides technical advice to public interest groups, non-governmental organizations, regulatory agencies, mining companies, and indigenous communities on the environmental impacts of mining. David Chambers has 40 years of experience in mineral exploration and development – 15 years of technical and management experience in the mineral exploration industry, and for the past 25 years he has served as an advisor on the environmental effects of mining projects both nationally and internationally. Stu Levit has 25 years of experience in reclamation and law. He has worked as a Land Reclamation Specialist for the Montana Department of State Lands, Abandoned Mine Reclamation Bureau, and on the legal aspects of water quality issues for the Coeur d'Alene Tribe in Idaho and the Confederated Salish and Kootenai Tribes in Montana.

GENERAL COMMENTS

Although there have been modifications to the mine proposal from the 2013 SDEIS, there are still a number of issues that have not been adequately addressed. These include:

- The design for the tailings dams still continues the use of outdated and inherently unsafe upstream-type dam construction, even when it is feasible (but more expensive) to employ safer centerline-type dam construction technology;
- The Hydrometallurgical Residue Facility is still being sited on a risky foundation which jeopardizes the long-term integrity of the double-lined containment system;
- The FEIS does not present even a basic analysis of the Financial Assurance required for both mine closure and post-closure water treatment, even though the company is willing to make this information available for public review;

- Even though the rail cars that have been redesigned could still spread as much as 32,000 pounds of ore per year along the railroad corridor, the FEIS considers that there is no potential for impact to either air or water resources;
- The FEIS still makes overly optimistic assumptions about the ability to collect seepage water from the tailings impoundment and Category 1 Waste Rock Storage Area; and,
- The FEIS continues to refuse to consider underground mining as a potential alternative for purely economic reasons, even though it can be demonstrated that underground mining could be economic viable.

SECTION COMMENTS

3.2.2.4 Financial Assurance

The Financial Assurance for the project is not given any detailed analysis in FEIS, even though this information is available from the mining company. The rationale for this decision is given as:

> *The level of engineering design and planning required to calculate detailed financial assurance amounts is not currently available, ...*

This is clearly not the case. Any responsible mining company will need to know what the potential financial liabilities of post-closure costs will be in order to provide a proper, and legally-required for the company's investors, financial estimate of the profitability of the proposed project.

In fact, PolyMet has stated:

> *Although NEPA and MEPA regulations do not require a discussion of financial assurance, PolyMet has provided an initial estimate of expected financial assurance needs that could be included in environmental review process (Reference (11)). (PolyMet 2015g, p. 35)*

The problem with presenting at least a preliminary financial assurance calculation appears to be with the agencies, not with PolyMet.

The level of detail available from the company is likely more than sufficient to provide a detailed estimate of the financial assurance. This estimate would provide the public with two very important pieces of information:

(1) The magnitude of the financial surety amount; and,

(2) Most importantly, the methodology which will be used to calculate the final financial surety.

It is an important part of the EIS process to know approximately how much money will need to be provided as a financial surety for closure and post-closure requirements, and that appropriate procedures and inclusive items are being used to estimate this amount.

The financial surety clearly has the potential to affect the human financial/social environment, and is an estimate for which the information is clearly available at the EIS stage.

It is noted in section 1.4.5 Financial Assurance, that:

> *Minnesota Rules, part 6132.1200 requires that before a Permit to Mine can be granted, financial assurance instruments covering the estimated cost of reclamation should the mine be required to close for any reason at any time must be submitted and approved by the MDNR. (FEIS, p. 1-18)*

Even though a financial analysis is required "before a Permit to Mine can be granted" this critical element is not disclosed in the FEIS. In addition to the claim that this level of detail available is not sufficient – which is clearly not true – this is also justified by asserting:

> *There are no applicable federal financial assurance requirements that would be incorporated into the Permit to Mine,... (FEIS, p. ES-55)*

The financial assurance for post-closure is a very important financial element of this project because long-term water treatment will be required. If the assumptions used in calculating the financial assurance for long-term water treatment are not inclusive of all potential costs, or if the assumptions for long-term investment return and/or inflation are not conservative enough, then the public will either be saddled with paying for the long-term water treatment with public funds, or the public will suffer the environmental impacts of not treating this contamination. These are potential significant impacts on the public.

A Federal agency must prepare an EIS if it is proposing a major federal action significantly affecting the quality of the human environment (the Army Corps of Engineers and the Forest Service are co-preparers of this EIS).

In 40 CFR Chapter V - Council on Environmental Quality, Part 1502 - Environmental Impact Statement

> *§ 1502.1 Purpose.*
>
> *The primary purpose of an environmental impact statement is to serve as an action-forcing device to insure that the policies and goals defined in the Act are infused into the ongoing programs and actions of the Federal Government. It shall provide full and fair discussion of significant environmental impacts and* ***shall inform decisionmakers and the public of the reasonable alternatives which would avoid or minimize adverse impacts or enhance the quality of the human environment.*** *Agencies shall focus on significant environmental issues and alternatives and shall reduce paperwork and the accumulation of extraneous background data. Statements shall be concise, clear, and to the point, and shall be supported by evidence that the agency has made the necessary environmental analyses. An environmental impact statement is more than a disclosure document. It shall be used by Federal officials in conjunction with other relevant material to plan actions and make decisions. (****emphasis added****)*

In 40 CFR Chapter V - Council on Environmental Quality, Part 1508, Terminology and Index, it states:

> *Section 1508.14 Human environment*
>
> *"Human environment" shall be interpreted comprehensively to include the natural and physical environment and the relationship of people with that environment. (See the definition of "effects" (Sec. 1508.8).) This means that economic or social effects are not intended by themselves to require preparation of an environmental impact statement.* ***When an environmental impact statement is prepared and economic or social and natural or physical environmental effects are interrelated, then the environmental impact statement will discuss all of these effects on the human environment. (emphasis added)***

A financial surety that is not adequately calculated could have an impact on the human environment as described in the CEQ regulations, and therefore should have been analyzed in the FEIS.

The "preliminary cost estimate for closure" for NorthMet is as large as $200 million during operation (FEIS, p. 3-142). In addition, it appears that this cost estimate is for closure costs only, and does not include post-closure water treatment. It is typical that post-closure water treatment costs are as large as the direct closure costs, and if this is the case then the amount for the current financial surety is being considerably underestimated.

3.2.2.1.8 Engineered Water Controls

Category 1 Stockpile Water Containment System and Cover

The engineered cutoff wall or "hydraulic barrier" is critical to meeting water quality requirements for the proposed project. Key factors for the cutoff wall will be:

(1) How well the cutoff wall can be grouted into the fractured bedrock to avoid contaminants moving under the wall in more permeable sediments;

(2) How effective the collection system on the upstream side of the cutoff wall is at removing pressure on this barrier; and,

(3) The permeability contrast between the cutoff wall and the adjacent sediments.

Cutoff Wall Contact with Bedrock

The cutoff wall is to be installed by:

> • *Cutoff Wall – the cutoff wall will be constructed using trenchless in-situ construction techniques whereby a mechanical mixer is inserted into the ground along the cutoff wall alignment. As the mixer 'walks' down the cutoff wall alignment, it mixes the soil along the cutoff wall location with bentonite. The soil-bentonite mixing occurs in-situ and an open trench is not utilized. DeWind One-Pass Trenching and Hayward Baker are examples of companies that provide such services. At locations where boulders are encountered that interfere with trenchless construction, the boulders will be removed using conventional excavation methods. Small diameter cobbles and boulders are expelled from the excavation as part of the trenchless construction process. (PolyMet 2015l, Attachment G, FTB Containment System Slope Stability Impacts p. 5)*

It might be difficult for an in-situ mechanical mixer to determine when it is really at the bedrock interface since no geologic logging or permeability measurements are being made. If the mechanical mixer does not reach fractured bedrock, for any reason, an zone of relatively higher permeability for contaminants to escape could be created.

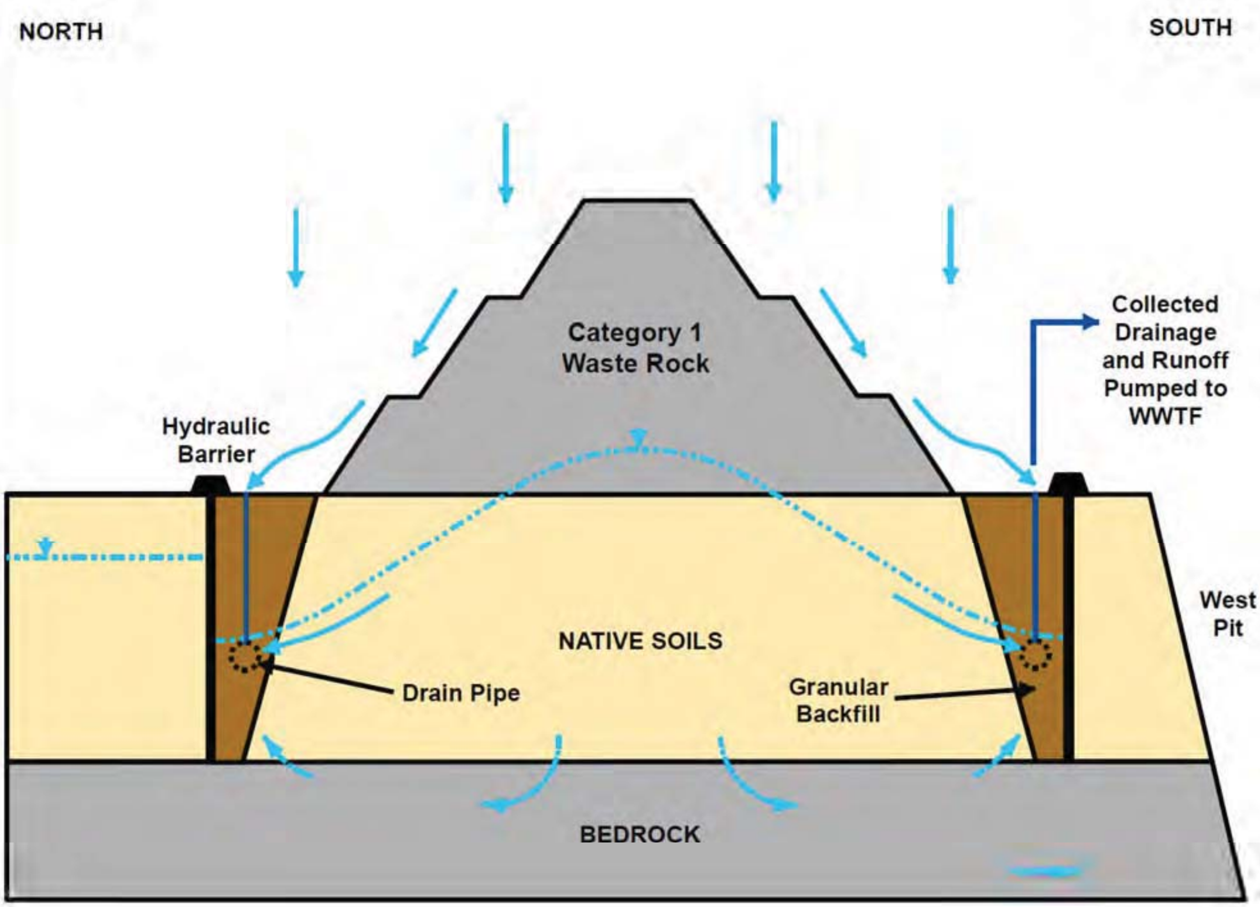
Collection System

Cartoon diagrams of the seepage collections systems at the Category 1 Waste Rock Facility (Figure 3.2-11) and the Tailings Impoundment (Figure 3.2-28) are presented on the following pages. If the seepage collection trenches are not extended to bedrock, as is depicted in Figure 3.2-28, or if the "drain pipes" are not placed at or near the bottom of the trenches, there will be more pressure attempting to push fluid through the cutoff wall.

In the Geotechnical Data Package, Volume 1 – Flotation Tailings Basin (PolyMet 2015l), the seepage collection trenches for the tailings basin it is noted that:

> *The seepage collection trench and drain pipe depth has not yet been finalized, but we assume an average depth of 8 feet to prevent system freezing and maintain operations through-out winter (exact depth will be determined during final design and construction). (PolyMet 2015l, p. 4)*

Even though the cutoff wall is designed to be low-permeability, more hydraulic pressure on this barrier will mean more seepage through or under the barrier.



Water Flow

Not To Scale

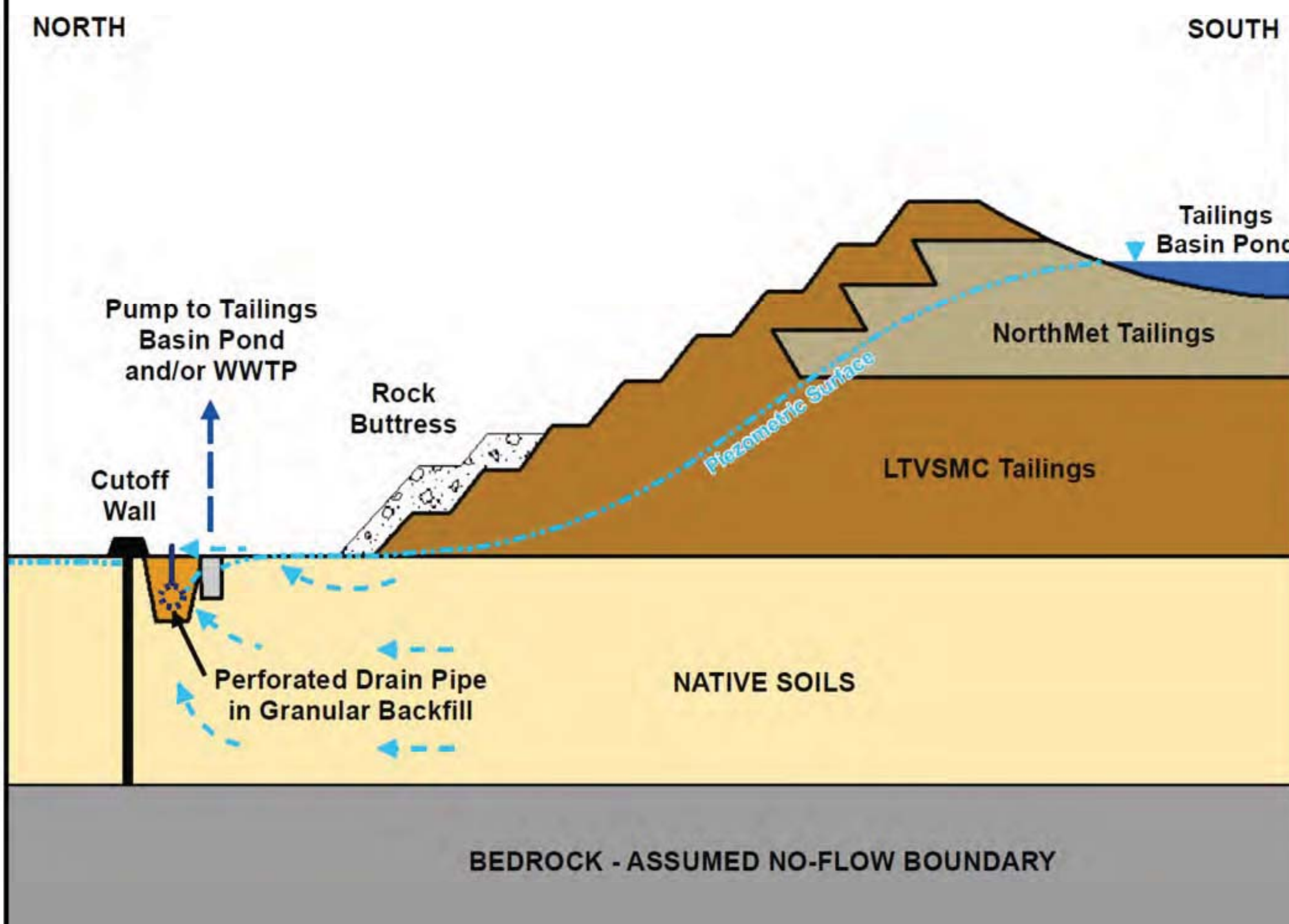
Figure 3.2-11

Conceptual Representation of the Category 1 Stockpile Containment System - Years 1-13

NorthMet Mining Project and Land Exchange FEIS

Minnesota

November 2015



Not to Scale

MNDNR US Army Corps of Engineers St. Paul District

Figure 3.2-28
Conceptual Cross Section - Tailings Basin Groundwater Containment System
NorthMet Mining Project and Land Exchange FEIS
Minnesota

November 2015

Permeability Contrast

> *The cutoff wall would have a hydraulic conductivity specification of no more than 1x10^{-5} centimeters per second (cm/sec). (FEIS, p. 3-47)*

CSP2's comments on DSEIS recommended 1x10^{-6} cm/sec. It's not clear why 10^{-5} is adequate, in that it only provides approximately one order of magnitude difference with some groundwater-carrying sediment units. For example, at the Tailings Basin (PolyMet 2015l, p. 13):

- *Depth to bedrock ranges from 2 to 47 feet with an average depth of approximately 20 feet. Bedrock was competent, with a near surface fracture zone.*
- *Groundwater levels were at or just below the ground surface.*
- *Hydraulic conductivity of the glacial till ranged from 1.5x10^{-3} ft/s (4.6x10^{-2} cm/s) to 1.7x10^{-6} ft/s (5.2x10^{-5} cm/s) with a geometric mean of 5.1x10^{-5} ft/s (1.5x10^{-3} cm/s).*
- *Hydraulic conductivity of the upper portion of the bedrock ranged from effectively zero (the borehole produced no water) to 2.4x10^{-5} ft/s (7.3x10^{-4} cm/s), with a geometric mean (excluding the zero inflow locations) of 1.9 x 10^{-6} ft/s (5.8 x 10^{-5} cm/s)*

Also see Table 4.2.2-5 Bedrock and Surficial Aquifer Hydraulic Conductivity Estimates at the Mine Site.

Reclamation and Long-3.2.2.3.12 term Closure Management

Tailings Basin Reclamation

The plan for closure of the Tailings Basin include:

> *The pond would remain in the reclaimed Tailings Basin with a wetland around its perimeter. In general, the pond's maximum lateral extent would be maintained to be no closer than 625 ft from the interior edge of the Cell 1E/2E dams. ... The pond and wetland would continue to lose water via seepage, but at a reduced rate compared to operations, as a result of the bentonite amendment of the tailings surface. (FEIS, p. 3-133)*

While the design of the bentonite cap will assist in limiting oxidation of the tailings, it violates one of the critical recommendations of the Mt Polley Expert Panel (2015) – that there be no wet closures" for tailings ponds. The Panel said:

> *For new tailings facilities. BAT (Best Available Technology) should be actively encouraged for new tailings facilities at existing and proposed mines. Safety attributes should be evaluated separately from economic considerations, and cost should not be the determining factor.*

and;

> *The goal of BAT for tailings management is to assure physical stability of the tailings deposit. This is achieved by preventing release of impoundment contents, independent of the integrity of any containment structures. In accomplishing this objective, BAT has three components that derive from first principles of soil mechanics:*
>
> *1. Eliminate surface water from the impoundment.*
>
> *2. Promote unsaturated conditions in the tailings with drainage provisions.*
>
> *3. Achieve dilatant conditions throughout the tailings deposit by compaction.*

and;

Where applicable, alternatives to water covers should be aggressively pursued.

The design for the Tailings Basin at NorthMet meets none of these criteria. Cells 1E & 2E contain a relatively small amount of tailings. It would be feasible to design a new tailings dam, using the Category I waste rock as construction material, based on centerline design that could do away with the dangerous upstream construction method. The Tailings Basin already incorporates a cutoff wall so no additional seepage collection would need to be planned.

A centerline dam design could also incorporate enhanced tailings drainage, which would promote the unsaturated conditions recommended by the Mt Polley Expert Panel. While this design might increase the amount of seepage to be treated, the Panel also noted:

The Panel recognizes that creating dry tailings may increase the amount of water requiring treatment or storage.

In the Panels opinion, the additional water treatment is more than compensated by the long-term stability achieved by maintaining the tailings in an unsaturated state.

Upstream Dam Engineering

As mentioned above, upstream-type dam construction is the most problematic type of tailings dam construction, and is associated with a majority of tailings dam accidents. Upstream dam construction has been banned as a dam construction practice in Chile as the result of the large number of seismic and other dam failures associated with this type of structure.

Three noted experts on tailings dams published "The 10 Rules" for upstream tailings dams in 2002 Martin et al 2002). They note that:

It is also important to note that these rules are not options and are not interchangeable with alternative concepts of soil mechanics. These rules exist based upon the fundamentals of soil behavior, the experience of numerous tailings dam failures and the experience of well-managed facilities that perform as intended. Of the 10 rules, a "score" of 9/10 will not necessarily have a better outcome than 2/10, as any omission creates immediate candidacy for an upstream tailings dam to join the list of facilities that have failed due to ignoring some or all of the rules.

Rule number 2 is:

2. A sufficiently wide beach-above-water (BAW), relative to the ultimate height of the dam, must be maintained at all times, to achieve segregation of the coarser tailings sizes and to form a relatively strong, wide, drained (unsaturated), and/or dilatant (non-contractant during shear) outer shell. ***The dam slope must not be underlain by tailings slimes (beach-below water - BBW), unless the designer has satisfied Rule 4 below. (emphasis added)*** *The shell must be of sufficient width to retain the "bursting pressures" (Casagrande and MacIvor, 1970) of the upstream contractant beach sands or slimes if they liquefy.*

They further emphasize:

The rules for the design of an upstream constructed tailings dam are ***not optional guidelines*** *for individual designers to randomly select components they can "fit" into their conception of a safe facility. (**emphasis added**)*

Even for the proposed upstream-type construction, the proposed dam is violating one of the 10 Rules for upstream dam construction.

3.2.3.4.1 Underground Mining Alternative

In all of the EIS versions the alternative of underground mining has been screened and rejected. Underground mining offers a number of environmental benefits, and would conceivably offer a lesser number of jobs for a similar amount of time as the proposed alternative.

As noted by GLIFWC in its comments in FEIS Appendix B:

> *The document states that for an alternative to be evaluated it must meet 5 screening criteria:*
>
> 1. *Be technically feasible*
> 2. *Be available*
> 3. *Offer significant environmental benefits over the proposed project*
> 4. *Meet the purpose and need*
> 5. *Be economically feasible*

As GLIFWC points out, all of these criteria can clearly be met, with the exception of the last. The only rationale that is used to eliminate the alternative is economic feasibility.

CSP2 has performed its own analysis of the possible underground mining, assuming an underground room & pillar mining cost of $44/ton for backfill, and $39/ton without backfill.[1] We disagree with the assumption in AGP 2011 that the recovery rate for room & pillar mining would be limited to 55%. The method can be more flexible than that assumed by AGP. Mining methods tend have site-specific modifications that are made to make the method most flexible and productive at that site. For our purposes we have assumed that the mining method chosen will allow a recovery rate similar to that for open-hole stoping (90%).

The assumptions/rationale for these cost assumptions are detailed in the table below:

[1] CSP2 has used mining cost estimates from the FEIS as starting points for its analysis. It does not necessarily agree with the assumptions and methods used for the FEIS calculation of mining costs.

CSP2 NorthMet Underground Mining Cost Estimates

Mining Rate (tpd)	Mining Cost ($/ton)	Milling Cost ($/ton)	General (Admin) & Contingency ($/ton)	Total Cost ($/ton)	Pre Production Capital Costs (millions $)	Source
7500	28	12.5	3.5	**44.0**	**225.0**	**Room & pillar with backfill:** mining cost from InfoMine 2009 / Foth 2012; p. 9, milling cost from InfoMine 2009 / Foth 2012, p. 11; general & contingency from Foth 2012, p. 8; Pre-production capital costs from InfoMine 2009 / Foth 2012, p. 11
7500	23	12.5	3.5	**39.0**	**205.0**	**Room & pillar without backfill:** mining cost from InfoMine 2009 / Foth 2012; p. 10, milling cost from InfoMine 2009 / Foth 2012, p. 11; general & contingency from Foth 2012, p. 8; Pre-production capital costs from Copperwoodl, MI, 2009 / Foth 2012, p. 11

References:

AGP 2011	Zurowski, Gordon, AGP Mining Consultants, Memo to Jim Tieberg re High Level Underground Costs, November 11, 2011
Appendix B	Appendix B Underground Mining Alternative Assessment for the NorthMet Mining Project and Land Exchange Environmental Impact Statement, September 27, 2013
Foth 2012	Bornhorst, Theodore J, LLC, Economic Assessment of Conceptual Underground Mining Option for the NorthMet Project, Subconsultant to Foth Infrastructure & Environment, LLC, October 2012, Underground Mining Alternative Assessment for the NorthMet Mining Project and Land Exchange EIS, Appendix B, Attachment 1
Foth 2013	Bornhorst, Theodore J, LLC, Response to USEPA Questions Regarding: Economic Assessment of Underground Mining Report Dated October 2012, Underground Mining Alternative Assessment for the NorthMet Mining Project and Land Exchange EIS, Appendix B, Attachment 2
InfoMine 2009	InfoMine USA, Inc., 2009, Mining Cost Service, Section CM, Cost Models, cited in Foth 2012
PEG 2009	Zurowski, Gordon, PEG Mining Consultants, Memo to Jim Tieberg re High Level Underground Costs, July 30, 2009

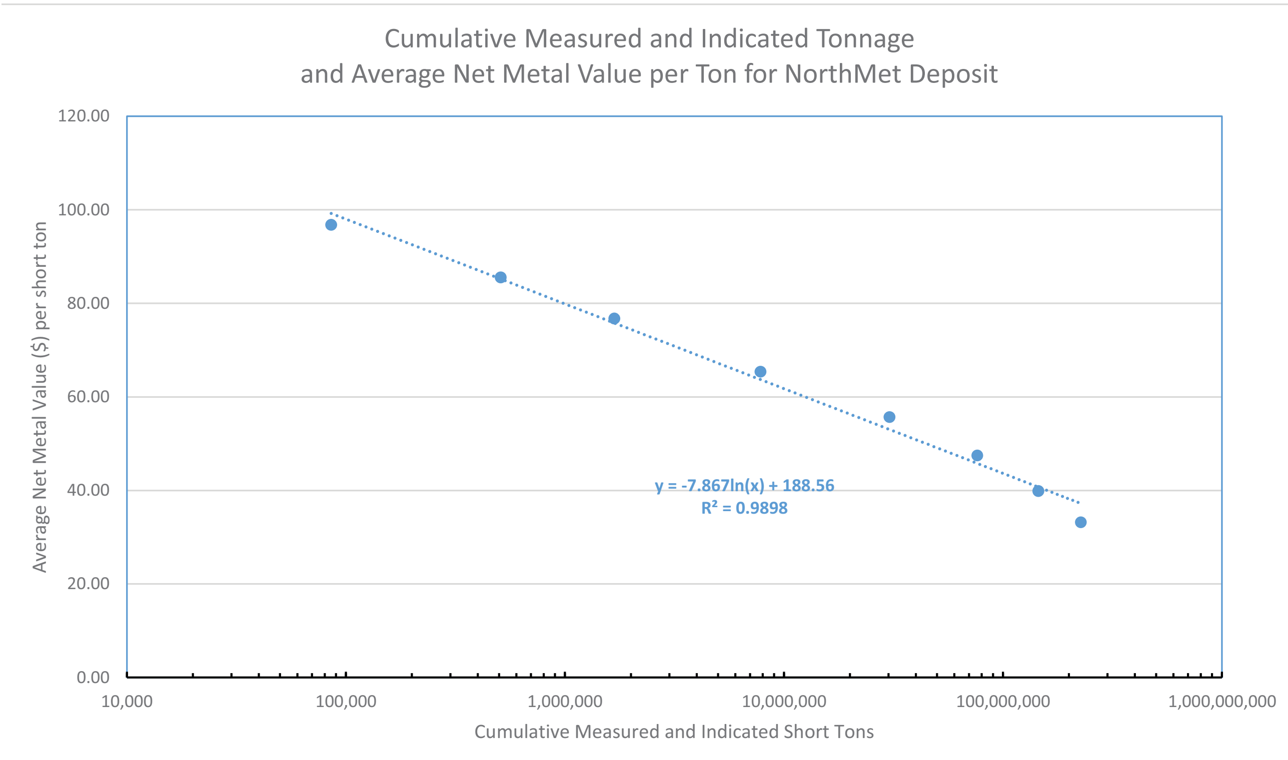
Similarly, the total operating and total pre-production costs are estimated based on the assumptions/rationale as detailed in the table below:

CSP2 Total Operating and Total Pre-Production Capital Cost Estimates Applied to Economic Assessment of Conceptual Underground Mining at NorthMet

from: Foth 2012 / Appendix B

Extracted Tonnage (million short tons)	Underground Daily Rate of Production (tons/day)	Productive Life of Mine (years)	Total Operating Costs ($/ton)	Total Pre-Production Capital Costs ($)	
18	7,500	6 to 7	44	225,000,000	**With Backfill:** Extracted tonnage from AGP 2011 p. 3; underground rate - see Foth 2012, p.9; total operating costs from InfoMine 2009 / Foth 2012, see Foth Cost Estimates; Pre-production capital costs from InfoMine 2009 / Foth 2012, p. 11
18	7,500	6 to 7	39	205,000,000	**Without Backfill:** Extracted tonnage from AGP 2011 p. 3; underground rate - see Foth 2012, p.9; total operating costs from InfoMine 2009 / Foth 2012, see Foth Cost Estimates; Pre-production capital costs from InfoMine 2009 / Foth 2012, p. 11

As was done in Appendix B of the FEIS, for comparison purposes both costs and revenue were based on 2012 data. The average net metal value for the amount to be mined, 18 million tons, was calculated based on the average net metal value per short ton developed in Appendix B, Foth 2012. Since a value for 18 million tons was not explicitly stated in Foth 2012, the data was extrapolated (in the graph below) to yield an average net metal value per short ton of $57.13 (2012 US dollars).



Cumulative Measured and Indicated Tonnage and Average Net Metal Value per Ton for NorthMet Deposit

from: Foth 2012 / Appendix B

Cumulative Measured and Indicated (short tons)[1]	**Average Net Metal Value** ($ per short ton)	**CSP2 Estimates of Average Net Metal Value (in blue) based on the data graphed above** ($ per short ton)	
227,017,162	33.18	37.19	
145,066,201	39.86	40.72	
76,373,821	47.46	45.76	
30,369,759	55.66	53.02	
18,000,000		57.13	Economic tonnage estimate from AGP 2011, p. 3
15,000,000		58.57	
7,817,279	65.37	63.70	
1,682,328	76.72	75.78	
509,229	85.54	85.18	
85,614	96.77	99.21	

Notes: [1] Cumulative measured and indicated tonnage and associated grade provided by AGP

Economic Analysis of Underground Mining of the NorthMet Deposit

from: Foth 2012 / Appendix B

Extracted Tonnage at 95 % rate of extraction and 5 % dilution (tons)	Total extracted net metal value (dollars)	Total revenue (average net metal value minus 5 % royalty) (dollars)	Total Operating Cost (dollars)	Operating Profit (Revenue minus operating cost) (dollars)	Pre-production capital cost (dollars)	Operating Profit minus pre-production capital costs (dollars)	Daily production (tons)	Life of mine for economic analysis (years)	
2,000,000	129,847,971.83	123,355,573.24	160,000,000.00	-36,644,426.76	125,000,000.00	-161,644,426.76	1,000	6	
5,000,000	318,769,570.88	302,831,092.34	370,000,000.00	-67,168,907.66	125,000,000.00	-192,168,907.66	2,000	7	
10,000,000	604,406,603.41	574,186,273.24	682,000,000.00	-107,813,726.76	150,000,000.00	-257,813,726.76	3,000	9	
15,000,000	875,343,935.13	831,576,738.38	934,500,000.00	-102,923,261.62	175,000,000.00	-277,923,261.62	4,000	10	
18,000,000	1,028,426,829	977,005,487	792,000,000	185,005,487	225,000,000	-39,994,513	7,500	6 to 7	room & pillar with backfill
18,000,000	1,028,426,829	977,005,487	702,000,000	275,005,487	205,000,000	70,005,487	7,500	6 to 7	room & pillar without backfill
20,000,000	1,134,125,150.76	1,077,418,893.23	1,130,000,000.00	-52,581,106.77	200,000,000.00	-252,581,106.77	5,000	11	
25,000,000	1,376,867,161.05	1,308,023,803.00	1,412,500,000.00	-104,476,197.00	200,000,000.00	-304,476,197.00	5,000	14	
30,000,000	1,633,916,992.93	1,552,221,143.28	1,470,000,000.00	82,221,143.28	250,000,000.00	-167,778,856.72	7,500	11	
35,000,000	1,857,679,184.93	1,764,795,225.68	1,715,000,000.00	49,795,225.68	250,000,000.00	-200,204,774.32	7,500	13	
50,000,000	2,511,252,374.91	2,385,689,756.16	2,450,000,000.00	-64,310,243.84	250,000,000.00	-314,310,243.84	10,000	14	
75,000,000	3,496,138,949.08	3,321,332,001.63	3,637,500,000.00	-316,167,998.37	300,000,000.00	-616,167,998.37	10,000	21	
100,000,000	4,360,816,362.32	4,142,775,544.20	4,700,000,000.00	-557,224,455.80	400,000,000.00	-957,224,455.80	15,000	18	

Notes:

- In situ average net metal value per ton from Table 2 determined for specific tonnage by log 10 linear extrapolation minus treatment charge.
- Applicable day rate of production and associated total operating costs and pre-production capital costs from Table 3. Economic analysis life of mine based on day rate of production rounded to even year; once life of mine is fixed daily rate of production allowed to vary to accommodate rounding in simple cash flow analysis.
- Rate of extraction and dilution discussed in text. Total extracted net metal value includes deduction for treatment charge as given in Table 1.

The economic analysis of underground mining in the table above indicates that, in 2012, a case for positive net operating profit for underground mining without backfill can be made. Tweaking of assumptions for underground mining with backfill, especially of the operating costs, might also make underground mining with backfill potentially profitable. This is reasonable because all of assumptions made for these analyses are very rough, and if nothing else these calculations suggest that a more detailed analysis of underground mining is warranted.

It should not be necessary to show that underground mining in profitable in every circumstance, only that it is possible under some reasonable set of conditions, like the economic climate of 2012, which was not ideal, to operate at a profit.

It is also relevant to note that in this section it is stated:

> *This alternative would involve mining the NorthMet Deposit as defined by the proposed open pit boundary. (FEIS, p. 3-159)*

If the phrase "as defined by the proposed open pit boundary" refers to the horizontal and vertical extent of the proposed pit, then a significant part of the ore body is being left out of the potential ore calculations.

Theme ALT 10

Rejected Alternative: Paste Tailings Placed on a Lined and Covered Facility

It is noted in the FEIS:

> *Industry standard for dry stacking includes the use of a basin liner. Construction of a basin liner on the existing LTVSMC tailings basin has been evaluated and determined not to be feasible." (FEIS, p. A-315)*

A liner could be added on top of the existing tailings, and the existing tailings dams (~75 feet high now) would need to be reinforced with a buttress.

The reason that it is "not feasible" to put a liner on top of the existing tailings is that the tailings are not stable enough to allow it - which says something about the stability of the proposed impoundment.

Cells 1E and 2E have enough capacity to hold the dry tailings, but it is likely that the existing tailings, which are approximately 60 feet thick, could not be compacted enough to provide the necessary stability upon which to build a dry stack.

Interesting enough, the Hydrometallurgical Residue Facility is proposed to be built on approximately the same thickness of existing tailings.

Theme WR 018

Capture Efficiencies Described for the Tailings Basin

It is noted in the FEIS:

> *These new models consider the presence of an upper more-permeable bedrock zone directly below the slurry wall, with hydraulic properties based on 2014 packer tests conducted in five boreholes along the proposed capture system alignment. Sensitivity analyses have included variable bedrock hydraulic conductivity and different upper bedrock zone thicknesses up to 100 feet. The model results predict that the overall groundwater capture efficiencies of the proposed Tailings Basin surface and groundwater seepage containment system would be substantially greater than 90 percent. This analysis supports the conclusion that the assumption of 90 percent or greater groundwater capture efficiency is justified. (FEIS, p. A-546)*

And from Theme PD 08:

> *The north, west, and east seepage containment systems would capture 100 percent of surface seepage under expected conditions, and 90 percent, 90 percent, and 100 percent, respectively, of groundwater seepage. The Tailings Basin South Seepage Management System would capture 100 percent of surface water (Barr 2015e, as cited in the FEIS). (FEIS, p. 439)*

Given the proposed method of installing the slurry wall at the tailings impoundment, there is a significant possibility that keying the slurry wall into bedrock, even fractured bedrock, will not be 100% attainable. The modeling described in the statements above assume this is possible.

It is not apparent that the modeling took this possibility into account. If the slurry wall is not keyed 100% into bedrock significant leakage could result, especially with the way the seepage collection trenches are designed. We are concerned that the assumed capture efficiencies are too high.

5.2.2.3.5 Proposed and Recommended Mitigation Measures

Monitoring, Adaptive Management, and Mitigation

Adaptive Management has many definitions, but for the purposes of mining it might be defined as: "a structured, iterative process of robust decision making in the face of uncertainty, with an aim to reducing uncertainty over time via system monitoring (Wikipedia, Nov 2015)."

PolyMet proposes to use Adaptive Management for a number of critical aspects of the proposed mining project. Adaptive management is proposed for Water Management (probably the most critical program), the Stockpile Cover System, the Mine and Plant Site Waste Water Treatment Systems, Tailings Basin Pond Bottom Cover System, Wetland Monitoring, the geotechnical stability of the Waste rock stockpiles, Tailings Basin, and Hydrometallurgical Residue Facility, the Mine and Plant Site Air Quality Management Plans, and Long-Term Post-Closure Monitoring and Maintenance.

The US Department of Interior describes adaptive management as:

> *... management as the interplay of decision and assessment components, in an iterative process of learning by doing and adapting based on what's learned. Adaptive management involves key activities such as stakeholder engagement, resource monitoring, and modeling, none of which is sufficient by itself to make a decision process adaptive. The integration of these components is what defines an adaptive approach to natural resource management. (Williams and Brown 2012, p. 11)*

Simply monitoring activities and occasionally changing them when problems are discovered does not constitute adaptive management. The USGS notes that:

> *"Many people in the field of natural resource conservation now claim, sometimes wrongly, that adaptive management is the approach they use to manage resources (Failing et al. 2004). The current popularity of adaptive management is somewhat at odds with its rather modest record of documented success, a record based at least in part on an inadequate framing of many management problems, poorly designed monitoring, and incomplete implementation of the adaptive process itself. (Williams and Brown 2012, p. 6)*

This review seeks to distinguish adaptive management as a technical, substantive process from a generic definition of adaptation that encompasses lesser attempts to simply apply modifications to failed plans.

While adaptive management may take many forms, to be genuine it should ensure a substantive mix of critical elements for planned and unplanned impacts that are applied over time. In the permit context, it should have clear prospective, prescriptive commitments to application and measurement. These should generally consider and include, but may not be limited to, defining and employing:

1. Stakeholder involvement
2. Management objectives
3. Predictive models
4. Monitoring protocols
5. Decision making protocols
6. Follow-up monitoring
7. Assessment
8. Learning and feedback
9. Institutional learning for both the regulators and mines
10. Timing and costs associated with both the adaptive management and its implementation (success and failure) including post reclamation and post-bond release.

(See generally Williams and Brown 2012)

PolyMet proposed use of Adaptive Management is problematic because most of its applications do not include important features of adaptive management. Most of the proposed Adaptive Management Plans are in fact more akin to normal project management where project activities and plans are modified as necessary and appropriate based on changed conditions, failed activities, leaks, improvements in available technologies, etc. For example, PolyMet's EIS Rock and Overburden Management Plan, Section 6.0, Reporting and Adaptive Management, states that:

> *Adaptive management is a system of management practices based on clearly defined outcomes and monitoring requirements to determine if management actions are meeting the desired outcomes; and, if not, implementing changes that will best ensure that outcomes are met or re-evaluated. Adaptive management recognizes the uncertainty associated with estimates based on exploration drilling for a 20-year Mine Plan. Adaptive management measures will be developed through the Environmental Review process, permitting, and during operations, reclamation, and long-term closure to define when changes are needed. (PolyMet 2014h, p. 38)*

This seems to be a fairly self-evident definition of adaptive management – promising little more than modification of plans to implement changes in response to failure to meet previously committed outcomes. Adaptive management surely recognizes uncertainty (such as that uncertainty associated with estimates based on exploration drilling) but it requires a substantive process for dealing with that uncertainty and responding to the outcomes from that uncertainty on the ground. Much of the PolyMet plan does not commit to these steps or outcomes in terms of adaptive management – rather it commits to develop adaptive steps in the future. To be properly considered by regulators and the public, these steps should be included at the permitting stage – not as the mine, and its failures, unfold.

Adaptive management can be a functional iterative tool for many situations but it could also be misused/misapplied as a means to delay planning or other steps necessary for a complete mine and reclamation plan - both of which are necessary prior to agency approval of a mining permit. The USFS (USDA 2005) has noted:

> *"adaptive management does not postpone action until 'enough' is known, but acknowledges that time and resources are too short to defer some action" (Lee 1999, p. 5)*

Specific examples of these types of deficiencies include, but are not limited, to the following sections.

Rock and Overburden Management Plan

As discussed above, the Rock and Overburden Management Plan establishes a questionable definition of adaptive management. It seems to primarily promise to modify its Plan to accommodate variances from its pre-mine estimates. (See PolyMet 2014h, p. *38).* This is insufficient on many levels – the most important being that the Plan identifies only one mitigation measure that could be taken – when actual adaptive management should require considering (with regulators and stakeholders) a variety of reasonable alternatives, considering their effectiveness, and considering their costs (both for analysis purposes and for bonding). These were not done.

Adaptive Water Management Plan

The Adaptive Water Management Plan contains numerous components of what this review considered essential for adaptive management. An example is contingency planning and review. On the other hand, the Adaptive Water Management Plan states:

> *To achieve the specific purpose of treatment for each of the Project phases, the operating configuration and the operating requirements of individual process units within the WWTF or the capacity of the WWTF may need to be modified. Thus, the WWTF is considered an adaptive engineering control. The WWTF treatment processes can be adapted, as necessary, to meet the actual conditions encountered during the Project and estimated by water quality monitoring and continued model updating. (PolyMet 2014d, pp. 32-33)*

The use of the word "adaptive" in response to the need for modification does not make the plan adaptive management (or adaptive engineering). It means that the plan may need modified in response to deficiencies and the failure to meet plan objectives or regulatory guidelines/requirements. As described above, genuine adaptive management requires more than simple adaptation.

The Waste Rock Stockpile Cover system (Section 3) describes select elements that are important to adaptive management. These include specific events/conditions that could trigger change, possible changes that could achieve necessary changes, etc. The discussion is deficient in other important elements such as modeling and stakeholder participation.

Similarly Section 4's Plant Site Adaptive Water Management discussion includes modeling comparison and modification – which are important adaptive management elements. This section goes further than some, with notable details for the Waste Water Treatment Plant. But it does not go far enough to include elements to differentiate it from traditional, standard mine permitting where actual and modeled conditions are compared so as to guide future modeling and plan adjustment.

When discussing the potential for softening pretreatment water to improve the membrane life and process effectiveness the Adaptive Plan states that:

> *.... Generally, ripple effects from this adaptive management strategy will be small compared to current impacts and could be effectively mitigated. (PolyMet 2014d, p. 84)*

It is insufficient to simply conclude/promise that adaptive management strategy impacts will be less than current impacts. The EIS should evaluate the components of the adaptive management plan – which will ensure that it is sufficiently robust and afford regulators and the public an opportunity to evaluate it prior to permitting. Moreover, it is insufficient to simply promise that impacts can be mitigated – they must be shown to be mitigatable and/or at a minimum the mine must make clear commitment (backed by a financial surety) that impacts will actually be mitigated or resolved.

In the Section 5 discussion of Floatation Tailings Basin Pond adaptive management, the plan describes actual test projects, reporting and modifying the model, modifying the design, circumstances triggering modification, and options for modified performance. (PolyMet 2014d, pp. 94-95). These are important features for adaptive management and do not appear in most of the FEIS's "adaptive management" sections.

The Adaptive Management plan for Non-Mechanical Treatment Systems describes:

> *The Non-mechanical treatment systems are adaptive engineering controls because they will be designed and operated based on site-specific conditions using the knowledge that is gained during the operating and reclamation phases of the Project. The specific adaptive management approach for each non-mechanical system is outlined in the development plans (Sections 6.2.3, 6.3.3, and 6.4.3). (PolyMet 2014d, p. 104)*

Like the previous section, this section is more akin to normal mine operations where systems are adjusted to improve performance. However this section's focus on site-specific conditions and using knowledge to guide operations and modifications uses language more common to adaptive management - creating a potential for more robust application. The subsequent sections do not deliver actual adaptive management.

Importantly, the Plan includes provisions to help ensure that the Financial Assurance includes the costs of developing non-mechanical treatment for the Category 1 Stockpile, West Pit Overflow, and Floatation Tailings Basin. It is especially critical that adaptive management plans include adequate financial assurance development and modification to ensure that at all times the mine is operating there is sufficient bond for the state to at any time take over the site and fully operate, maintain, and close the mine. Such provision for bonding must be considered in all management planning.

Water Management Plan - Mine & Water Management Plan - Plant

Even reviewed together with the Adaptive Water Management Plan, as with numerous chapters, what are described as adaptive management plans for the Mine's (PolyMet 2015r) and Plant's (PolyMet 2015i) water management could be more accurately described as contingencies or responses to deficiencies/failures. They are not unreasonable and include important elements to respond to plan failures - but they do not contain critical elements necessary for actual, meaningful, substantive adaptive management. This includes their mitigation references to the Adaptive Water Management Plan (PolyMet 2015d).

By describing plans as adaptive management plans, the EIS raises the bar on what should be included. It is not sufficient to just monitoring activities and commit to possibly implementing from a list of contingencies when a problem is discovered. This is not adaptive management - it is the mine operator responding to a problem without clear commitment to meaningful adaptive process or outcome.

Air Quality

NorthMet Project Air Quality Management Plan - Mine, Version 4 states:

> ***6.3 Adaptive Management***
>
> *The Mine Site FEC (Fugitive Emission Control) Plan includes some adaptive management provisions to address the potential need for adjustments or modifications to the plan. Data from the meteorological monitoring system will be integrated with the data from the Mine Management System (water/chemical application, road usage, observed dust notifications) along with daily fugitive dust observation forms. These data will be reviewed, at a minimum, on a semi-annual basis to aid in analyzing trends and to determine if FECs are effective. The Mine Site FEC Plan will be modified as needed based on these reviews or other improvements that have been identified.*
>
> ***6.4 Available Mitigations***
>
> *Additional mitigations are available if necessary to achieve compliance, including:*
>
> - *revision to Mine Site FEC Plan*
> - *planting of trees or other vegetation along unpaved roads or around other potential dust generating activities to aid dust settling before reaching the ambient air boundary. (PolyMet 2014m, pp. 12-13).*

The concept that "some adaptive management provisions" could constitute adaptive management implies that adopting selected provisions somehow satisfies having a complete plan. These steps do not constitute adaptive management so much as they indicate a willingness to fix problems that evolve. To truly be adaptive management the above, limited provisions need to be part of a full adaptive management scheme. As written it is unclear that the mine will do what it promises or what is necessary for effective implementation.

NorthMet Project Air Quality Management Plan - Plant, Version 7, Issue Date: December 5, 2014 states that:

> *7.0 Reporting and Adaptive Management*
>
> *One time and periodic reporting will be required by the air emission permit. Specific reporting requirements are dictated by applicable federal and state air quality rules, Attachment A, and any other requirements anticipated in the permit.*
>
> *The subsections below provide a reasonable initial proposal for reporting requirements, based on knowledge of applicable regulations and professional experience and judgment. The final operating and maintenance requirements will be agreed upon between PolyMet and MPCA, during the permitting process and include public comment where applicable. (PolyMet 2014n, p. 31).*

The Plant plan does not contain or commit to meaningful adaptive management. It appears to be a fairly typical (non-adaptive management) mine regulatory proposal.

Flotation Tailings Management Plan

As discussed above, review and modification of plans does not alone constitute adaptive management. The Floatation Tailings Management Plan (PolyMet 2015n) describes a list of possible mitigation measures but otherwise fails to include other essential adaptive management components. An essential umbrella over these steps is a clear analysis that the measures proposed can accomplish what they purport to accomplish, an assessment of their likely effectiveness, and the costs associated with their implementation, failure, modification, and related bonding. Without these elements the EIS's proposed adaptive management is more a plan to plan than an "observational method of adaptive management." (PolyMet 2015n, pp, 5, 34)

5.2.4.2.2 Transportation and Utility Corridor

Indirect Effects

The FEIS analyzes the effect of spillage & blowoff as dust effects to vegetation (Section 5.2.4.2.1) and potential air quality impacts (minimal – Section 5.2.74.2). However, the primary concern for ore scattered along the transportation corridor is on water quality. At the Flambeau mine a small stream located next to the now-closed ore loading terminal is still contaminated with copper from non-point sources. This contamination is most probably related to dust blowoff from the ore trains, since copper is elevated in the entire sub-drainage (Chambers and Zamzow 2009).

According to the FEIS, the ore trains would travel on a:

> *… railroad, which would generally be used to transport ore from the Mine Site to the Plant Site using* ***three to four trains****, each consisting of* **sixteen to twenty 100-ton, side-dumping ore cars** *… (Section 3.2.2.2.4 Use During Operations, p. 3-85,* ***emphasis added****)*

The amount of ore spillage was originally estimated to be 6.14 tons per year for unrefurbished cars (PolyMet 2015q). However, after reworking the door hinges, PolyMet estimates the amount of spillage can be reduced 95% (PolyMet 2014a).

> *The quantity of ore that could potentially spill through the door and hinge gaps of a single refurbished ore car is estimated to be 0.20 tons per year. (FEIS, p. 5-164)*

The maximum number of cars (see the ***emphasis added*** above) is:

> Four trains x 20 cars per train = 80 cars

then:

> 0.2 tons/yr/car x 80 cars= 400 pounds/yr/car
>
> 400 pounds/yr/car x 80 cars (FEIS p. 3-85) = 32,000 pounds/yr

32,000 pounds/yr of ore falling from the modified rail ore cars onto the railroad corridor is still a potentially significant amount.

It was also noted in Section 8 – Major Differences of Opinion:

> *GLIFWC does not believe that monitoring of the creeks along the rail line will be effective in preventing or minimizing impacts because once detected in monitoring, the impact will have already occurred. GLIFWC states that cleanup of ore dust in an aquatic environment is a long and difficult process. (FEIS, p. 8-24)*

and;

> *The rail line between the mine and the processing plant is approximately 8 miles long, 1 mile of which is over wetlands, and crosses over at least 3 creeks. ... Because transport will deposit some level of ore and ore dust along the rail line, methods for control of contaminated runoff from along the rail line must be developed and implemented in the mine plan. (FEIS, Appendix C Tribal Agency Position Supporting Materials)*

Especially given that it is estimated that as much as 32,000 pounds/yr of ore can fall on railroad corridor, we must agree.

Baseline soil sampling along the rail route should be established, and regular soil sampling should be conducted to detect soil contamination before it leads to non-point source pollution of streams.

5.2.14.2.2 Tailings Basin

It was noted earlier in the FEIS that:

> *The inclusion of relatively large zones of finer-grained tailings within this outer shell reduces the drainage ability of the shell, increasing the phreatic surface, and reduces the localized shear strength due to the generally weaker behavior of the finer-grained tailings. There were instances during the operation of the LTVSMC Tailings Basin where significant amounts of fine tailings and slimes settled near the perimeter dams. (4.2.14.2.2 Development of the Existing LTVSMC Tailings Basin, p. 4-427)*

It is proposed to use Cement Deep Soil Mix (CDSM), and other measures like drains, to stabilize the tailings in order to allow upstream dam construction to continue. Also noted in the FEIS is that the geometry and physical changes to the embankments (such as CDSM) were incorporated into the design so that all computed slope stability Factors of Safety met or exceeded the Factors of Safety required by the NorthMet Geotechnical Modeling Work Plan (PolyMet 2015l, Attachment A).

It is also noted in the Geotechnical Data Package for the Tailing Basin (PolyMet 2015l) that:

> *The appropriate approach hinges on the extent, composition and continuity of stringers within the deposit as subsequently described. Several types of evidence support the conclusion that* ***heterogeneity within the deposits is localized****, so widespread and continuous stringers of the weakest material (slimes) are unlikely and isotropic parameters are appropriate. (PolyMet 2015l, p. 63,* ***emphasis added****)*

This is a necessary assumption for time-efficient modeling. It is also a critical assumption in terms of keeping the costs of the modeling task in a reasonable range. However, if this assumption is wrong, even in relatively local regions of the dams, then the modeling is wrong.

To illustrate the complexity of the tailings near and under the dam structures, Figure 5.2.14-6 for cross section F of the tailings basin, the most critical cross section in terms of potential instability.

> *Cross Section F, which intersects the northern dam of Cell 2E, as shown in Figure 5.2.14-4, was selected to represent the critical cross section for stability analysis purposes as it is the maximum section based on height as measured from the downstream toe to the proposed final crest, some layers of the weaker fine tailings and slimes extend close to the dam, and the original starter dam is underlain by peat. (FEIS, p. 5-657)*

The dam in the area of Cross Section F (see Figure 5.2.14.6 on the following page) will also require underdrains in order to provide long-term stability (PolyMet 2015l, p. 73).

This building complexity only raises the chances for misinterpretation and oversimplification in the modeling. Simpler models are better.

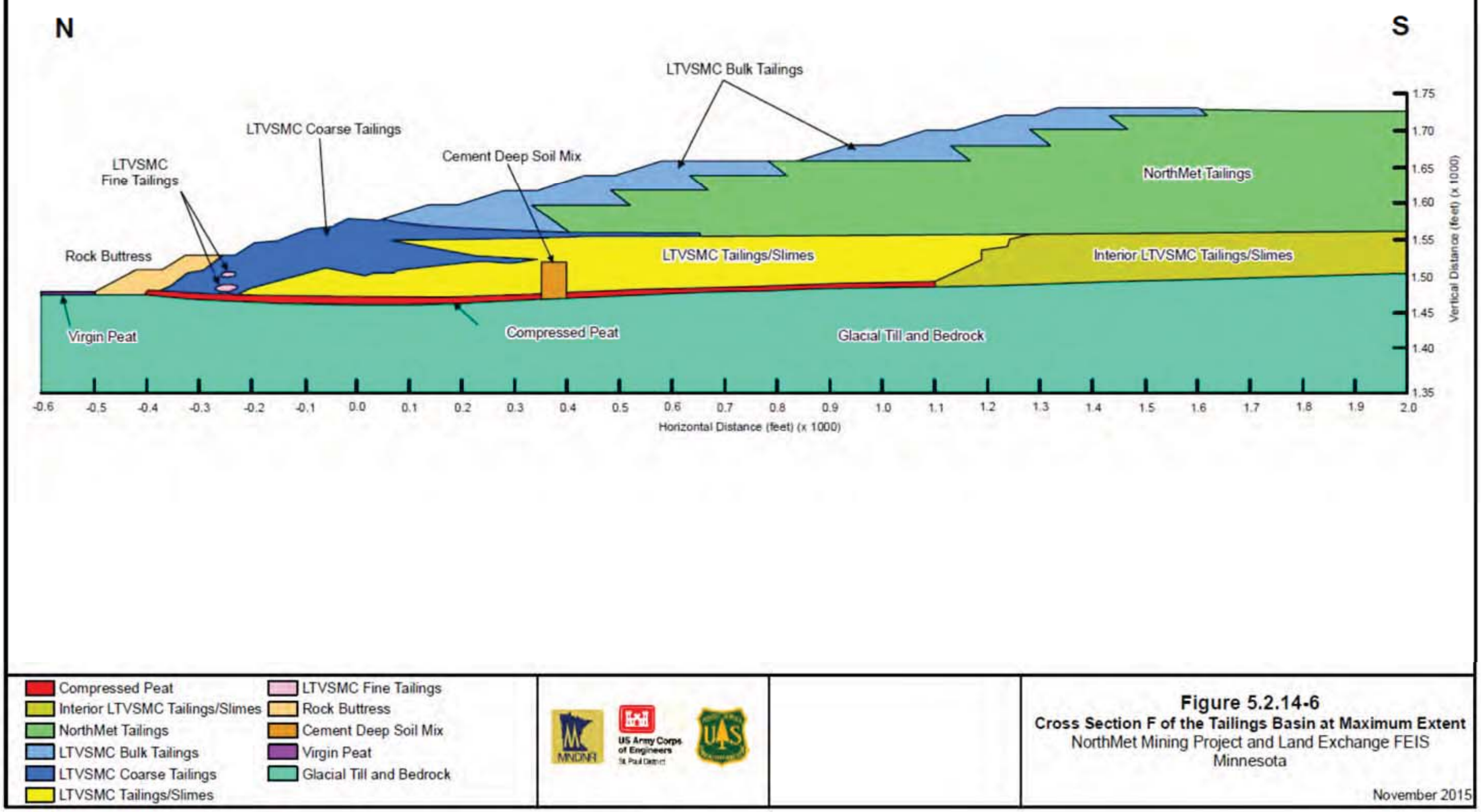


Figure 5.2.14-6
Cross Section F of the Tailings Basin at Maximum Extent
NorthMet Mining Project and Land Exchange FEIS
Minnesota

November 2015

The Cement Deep Soil Mix columns are designed to provide more weight bearing capacity in the tailings to attempt to justify further upstream-type dam construction. This is illustrated in (PolyMet 2015l) Figure 5-2. Note also the complexity of the geology illustrated in this figure.

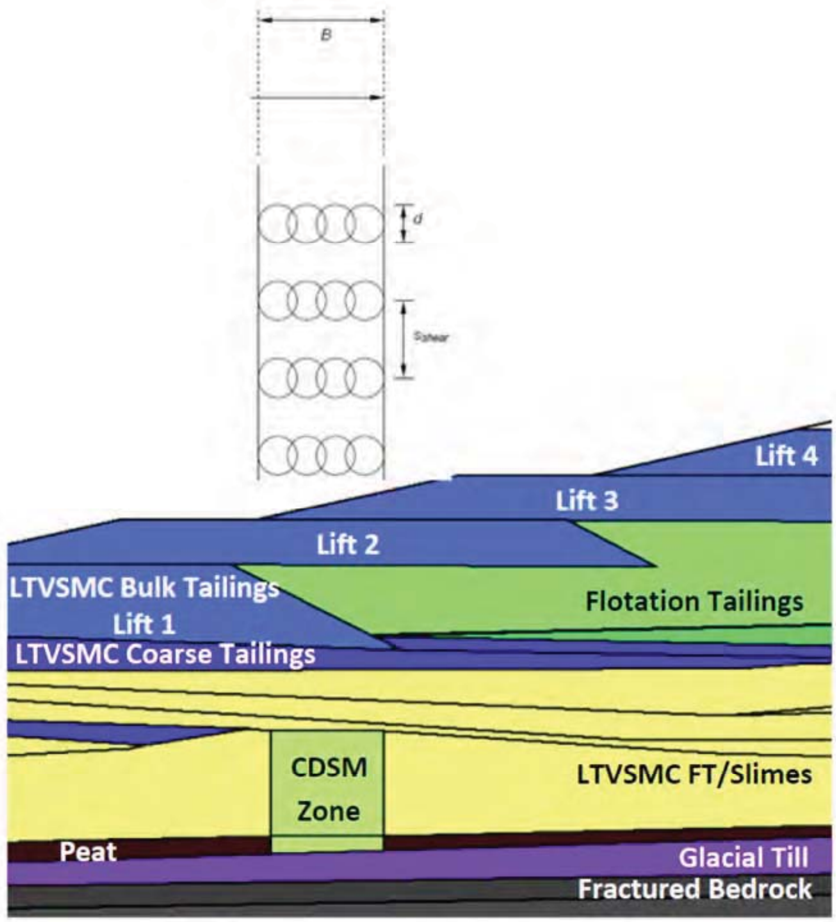


Figure 5-2 **Schematic Plan View (projected above Lift 3 slope) and Cross-Section of Section F CDSM Zone**

Of note is that all of the cross section views of the CDSM the top of the CDSM Zone terminates before the top of the LTVSMC tailings, and bottoms in glacial till, not bedrock.

Since this is done consistently we must assume it is intentional, yet there is no explanation in the FEIS or PolyMet 2015l as to why these columns are not extended to the top of the LTVSMC tailings.

5.2.14.2.3 Hydrometallurgical Residue Facility

One of the critical design parameters for the Hydrometallurgical Residue Facility is maintaining the integrity of the liner system. PolyMet plans on building the Hydrometallurgical Residue Facility on top of multiple layers of tailings and peat (see Figure 5.2.14-9).

In order to stabilize the underlying material PolyMet is proposing to:

- *Install wick drains (if required); and*
- *Place, monitor, and remove a preload fill in the existing LTVSMC Emergency Basin to preconsolidate existing material, thereby reducing future anticipated settlements to mitigate the potential future strains. (FEIS, p. 5-662)*

> *A preload would be placed on the existing LTVSMC Emergency Basin to consolidate the foundation materials before construction of the Hydrometallurgical Residue Facility. Wick drains may be used to help accelerate the consolidation time by increasing the effective hydraulic conductivity of the tailings due to decrease in flowpath length. Some portion of this load would be removed before construction, and the remaining material would be graded to provide sufficient drainage slope and provide a suitable foundation material for the facility. The material would rebound a small amount after the preload is removed.* ***The aggregate settlement at a representative location within the Emergency Basin, considering the maximum anticipated tailings thickness in the foundation, is computed to be 3.9 ft. The material at this location is modeled to consolidate an additional 1.4 ft by the end of operations of the Hydrometallurgical Residue Facility****.* ***(emphasis added)***
>
> ...
>
> *Strain in the Hydrometallurgical Residue Facility liner system would result from differential settlement in the facility foundation between points along the liner. (FEIS, p. 5-667)*

Adequate factors of safety should be guaranteed by installing engineered facilities verified by quality control, when possible – not by modeling.

A less technologically demanding, and safer, method of insuring the stability of the foundation of the Hydrometallurgical Residue Facility is to remove the problematic material down to bedrock. The material removed could be placed in the tailings basin, and not only would the subgrade be stable, but more room for hydrometallurgical residue would be gained.

In the case of the Hydrometallurgical Residue Facility placing the liner on the granite bedrock is possible.

Problems with the bottom liners likely could not be fixed without removing all of the waste. Safety, not cost, should drive liner foundation design considerations. The short-term attempt to save money by attempting to consolidate the tailings and peat underlying the proposed Hydrometallurgical Residue Facility might backfire in the long run.

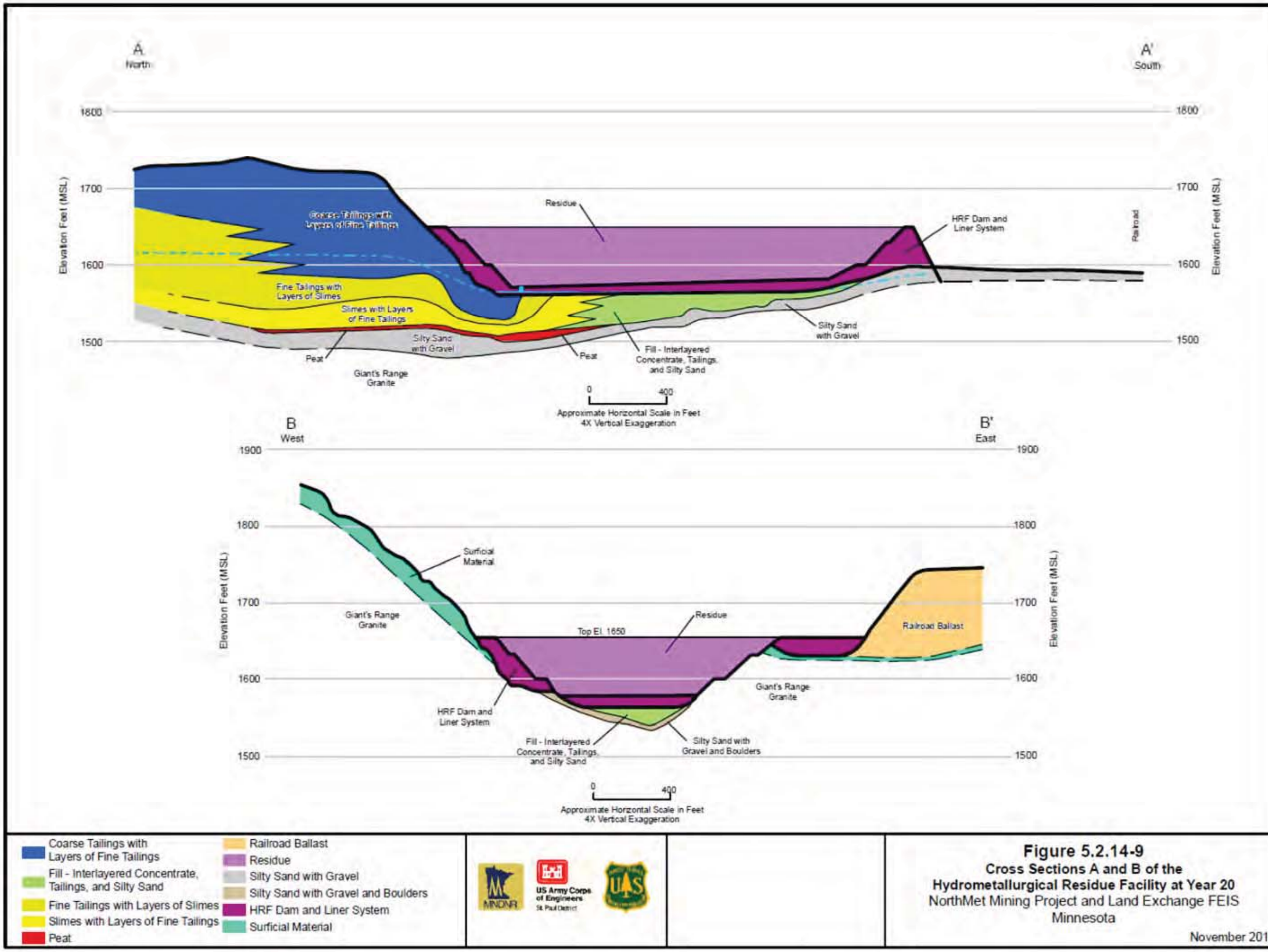


Figure 5.2.14-9
Cross Sections A and B of the Hydrometallurgical Residue Facility at Year 20
NorthMet Mining Project and Land Exchange FEIS
Minnesota

November 2015

Hydrometallurgical Residue Facility Design and Construction

Both the Hydrometallurgical Residue Facility (PolyMet 2014c, p. 11) and the Tailings Basin (PolyMet 2015l, p. 66) use a design earthquake event with a peak ground acceleration of 0.024g (2,475 year return period). The choice of a 2,475-year return design earthquake is not adequate for a structure that must hold in perpetuity.

The Maximum Credible Earthquake, not the 2,475-year return event – which is significantly less than an Maximum Credible Earthquake – should be used for the design event for all permanent structures, both dams and waste rock. The Maximum Credible Earthquake is recommended to be a 10,000-year return period earthquake (ICOLD 2001).

That using the 2,475-year return significantly underestimates the effect of an earthquake on a tailings dam or water rock pile. This can be seen by looking at the horizontal accelerations (g) in Table 6-2 Summary of Probabilistic Seismic Hazard Analysis Results (PolyMet 2015l). The 975-year return period earthquake has a maximum acceleration of 0.025g. The 2,475-year return period earthquake has a maximum acceleration of 0.055g, over twice that of the 975-year event. The maximum horizontal acceleration from a 10,000-year event would be significantly larger than that for a 2,475-year event.

Table 6-2 Summary of PSHA Results

			Seismic Source		
		Unit of Measure	Nearfield Earthquake	Farfield Earthquake	Combined Event (Simultaneous Nearfield and Farfield Earthquakes)
2,475-Year Return Period	Spectral Acceleration	g	0.055	0.016	0.061
	Peak Period	Seconds	0.1	2.0	0.1
975-Year Return Period	Spectral Acceleration	g	0.025	0.006	0.030
	Peak Period	Seconds	0.2	1.0	0.1
475-Year Return Period	Spectral Acceleration	g	0.013	0.000	0.017
	Peak Period	seconds	0.2	N/A	0.2
Hazard Probability		N/A	0.00052	8.13×10^{-5}	0.00062
Mean Distance		Miles	100	763	200
Mean Magnitude		M	5.62	7.73	5.92

Even if the legal requirement is only for a 2,475-year return design earthquake, from an engineering and safety standpoint PolyMet and its consultants should not accept the minimum required. They should do what safety and conservative management requires.

Thank you for the opportunity to comment on this permit.

Sincerely:

David M. Chambers, Ph.D., P. Geop

Stuart M. Levit, M.S., J.D.

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Protections 2016 Ft. Collins, Colorado, USA, 7-9 September 2016
2nd International Seminar on
Dam Protection Against Overtopping
ISBN: 978-1-1889143-27-9 DOI:

Root Causes of Tailings Dam Overtopping: The Economics of Risk & Consequence

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ABSTRACT

This paper examines overtopping failures of embankments at tailings impoundments from 1915-2015 and compares the severity of consequence for overtopping failures to that of other causes of failure. We find that the distribution by severity of consequence for overtopping at active mines is not significantly different from any of the other established "causes of failure." Further we find that the distribution by severity within and across all active TSF recorded failure causes (N=125) is also reflected in the mean distribution of severity for all of our recorded TSF failures (N=267) suggesting that a common root cause, rather than the individual causes of failure, may determine the severity of failure. We look here at the demonstrated link between severity of consequence of failure and the economic dynamics of the "Mining Metric" over 100 years (Bowker and Chambers 2015) as it applies to overtopping. We offer what is available from authoritative sources on the economics backstory of known overtopping failures and crises. We conclude that the deviations from best available technology and best applicable practices at the mine level are conscious choices driven by economics and that without a reframing of the professional, regulatory, and legal frameworks for mining these choices will continue to be made even where proven technology and new promising technology are available and better suited to a given mining asset. Solutions that will prevent mine failure require not only the work of evolving consensus on best available technology/best applicable practices, but also the recognition of root causes which build to catastrophic failure. A complete solution cannot be attained without accountability to best knowledge, best practice, best effective technology in mining law and regulation, as permit standards, as standards for oversight for life-of-mine and of life-of-tailings storage facilities.

Keywords: *overtopping failures, embankments, tailings impoundments, severity of consequence of failure*

1. INTRODUCTION

Overtopping (OT) is one of 8 codes developed by ICOLD for their survey reported in Bulletin 121 (UNEP/DTIE 2001). The original framing of data elements developed by ICOLD continued with WISE (2014) who are the official global record keepers of significant unplanned incidents at above ground tailings impoundments (Tailings Storage Facilities – TSFs). Previous research (Bowker and Chambers 2015) utilized the only direct measures of severity in WISE (2014), run out and release volume, supplemented as necessary by other authoritative narratives and compilations containing direct information on consequence, to develop severity of failure classes. In our data base 267 failures of tailings storage facilities could be divided into three major classes: Very Serious Failures; Serious Failures; and, Other Failures. As is shown in Figure 1, outside of foundation failures (FN) and erosion (ER) - both high severity low frequency causes of loss - no one cause of failure is any more correlated with high severity losses than any other. All others are very close to the mean of the 125 TSF failures at active mines which had complete cause-of-failure codes (i.e. all codes are similar to one another in severity profile). The 125 events which are failures at active mines only, mirror the distribution of severity in the 267 database events of 1915-2015.

This suggests that there is a common root cause that shapes severity of consequence. The data suggests that all customarily used causes of failure, including overtopping, are only a final event in a cluster of other factors that determine severity of failure. In addition to the essential continuing collaborative push to identify and develop best practice and best technology to prevent catastrophic public loss, the law and regulation of mining must look to these root causes beyond available or potentially workable technology.

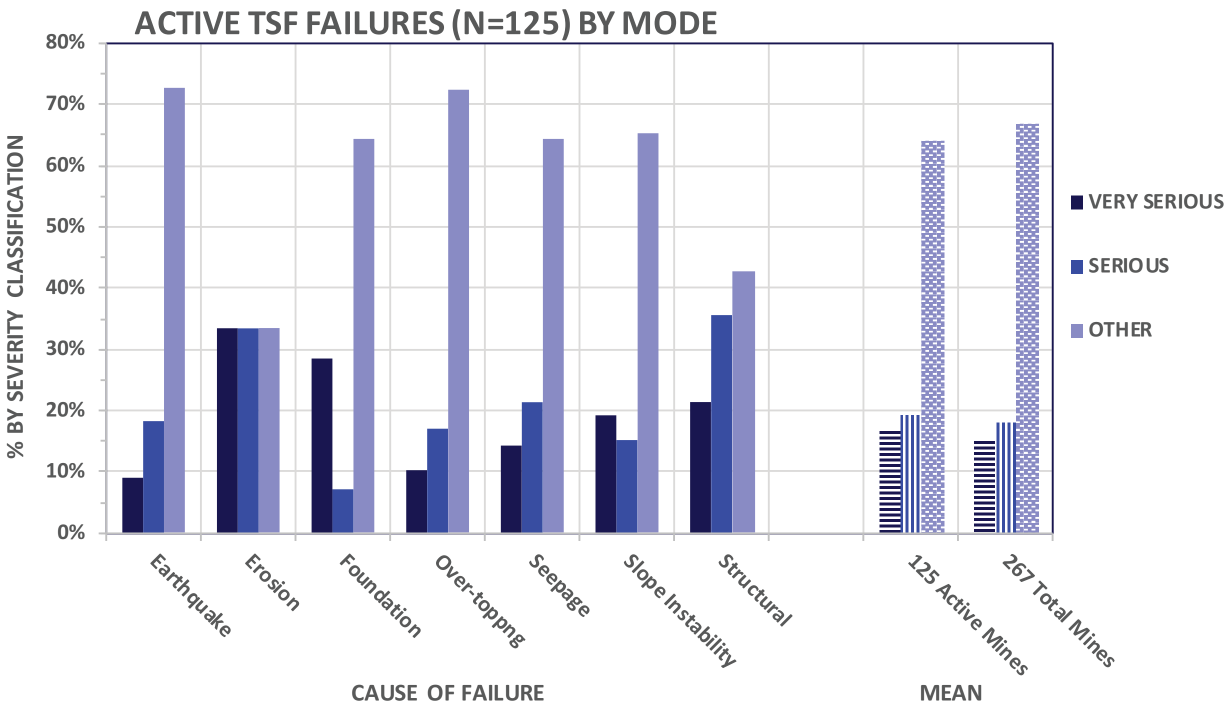

Figure 1. Failure Modes by Severity Classification for Active TSF Failures

We know from experience, from historical narrative, and from the research of Rico et.al. (2007) that key risk indicators reside in the characteristics of the tailings storage facility itself: type of construction; construction materials; height; length; total storage volume; and, adequacy of drainage. However, there is insufficient data on many of these elements in the official compilation for analysis of correlation with severity of consequence in the event of failure, and it is difficult to fill in the blanks from publicly available reports and records. Obviously, even though the larger the facility, the larger the possible failure, it is an oversimplification to think about reducing risk by not allowing large dams. The key issue in effective loss prevention strategy is recognizing and incorporating technologies and practices that will reduce the long-term rate of dam failures to a level that is acceptable to society.

It is widely recognized as well that regulatory and legal gaps are a common root cause shaping severity of failure. The June 9, 2016, police investigation in Brazil supporting the indictment of a Vale senior engineer (in addition to the 6 already indicted, including Samarco's former President) put it very well, correctly observing that behind the failure was a conscious choice to allocate all resources to higher throughput volumes with no corresponding investment in additional waste management technology (Kiernan 2016a). Vale's own independent evaluation of the last planned expansion in 2010 endorsed proceeding with no identified possibility of managing tailings waste (Amira et. al. 2010). Nothing in Brazilian law or regulatory requirements, or Minas Gerais permitting requirements, reviewed or objected to that.

In British Columbia, at Mt. Polley, Imperial Metals was grandfathered from new law and regulation on hazard rating and stricter inspection. As is common practice in most regulatory regimes, existing permitted facilities are seldom held accountable to new and higher standards. Even though Imperial Metals deviated from best available technology (BAT) and best applicable practices (BAP) as recommended by the original designer of the dam, the official finding was that Imperial broke no laws and was in violation of no regulations (McCrae 2015). The Mt Polley Expert Panel found that if the original design had been followed, the failure would not have occurred (Mt Polley Expert Panel 2015).

The BC Ministry of Energy and Mines, like most regulatory agencies, allows the dual use of the facility for storage of water. In May, 2014, prior to the dam failure, there had been an overtopping of the dam, but quick action by mine personnel prevented failure of the dam (Mt Polley Expert Panel 2015). Overtopping at Mt Polley could also have led to a dam failure, with similar damage as eventually occurred in the foundation failure in August in 2014.

Throughout Canada and all over the world new mines and new dams are approved within regulatory and legal structures that do not hold miners to best available technology and best applicable practices. Until this changes, it is clear that the industry will not consistently choose best available technology and best applicable practices unless required to do so. Geoffrey Blight emphasizes this in his authoritative and informed re-visitation of several notorious failures, among his very last works and summing a life time of excellence and insight on design and management of tailings storage facilities (Blight 2010).

While we cannot examine these other root causes systematically through any official or recognized recorded history, we can and did examine the global historical relationship between the primary economic parameters of the Mining Metric (the main strategy for mining continually falling grades of ore over a century of falling prices across all metals) against changes in the level of consequence in mine failures over the 100-year period ending 12/31/2009 (Bowker and Chambers 2015). That work demonstrated the strong correlation between the Mining Metric and the emergence of a pattern of higher severity of failure over a sustained 100-year trend of falling prices.

2. THE EMERGENCE OF INCREASING SEVERITY AND ITS RELATIONSHIP WITH THE MINING METRIC OVER 100 YEARS

2.1. The Emerging Trend of Increased Severity

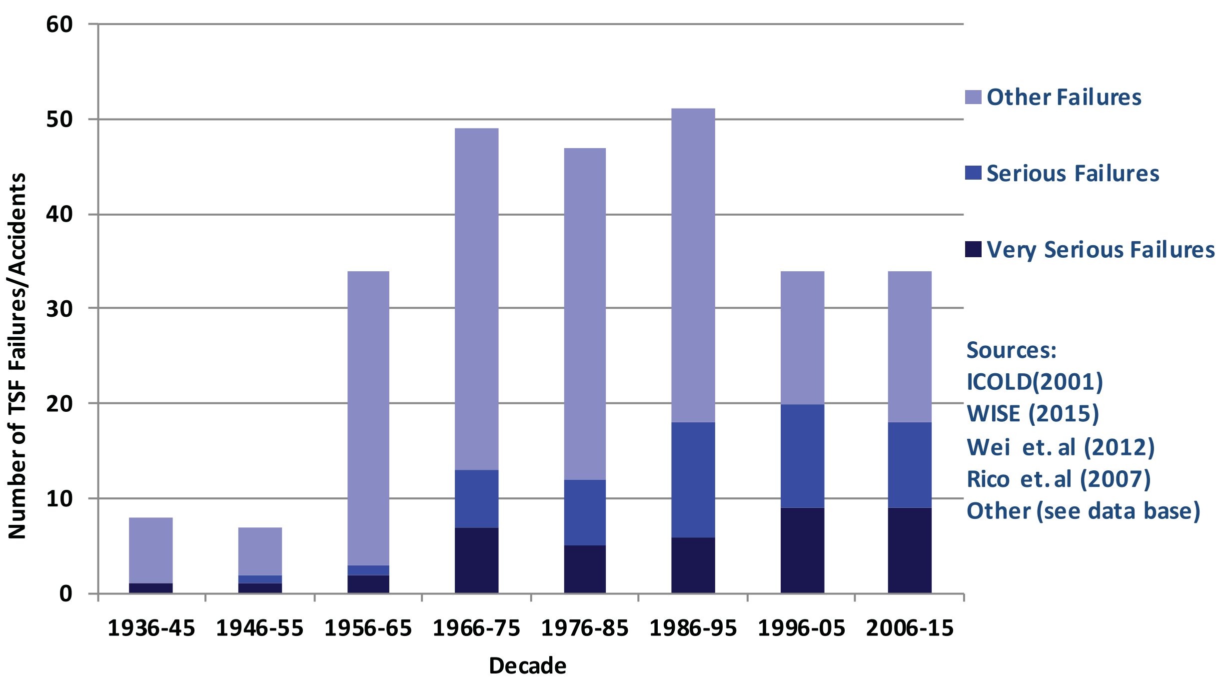

Figure 2. Increasing Severity of TSF Failures Globally by Decade 1936-2015

The overall distribution by severity over time is shown in Figure 2 for the period 1936 through 2015 by decade. The increasing severity of consequence in the recent three decades is apparent. Over the 80 years 1936-2015 the expected rate of very serious failures is 5 per decade (40/8). In the last three decades the rate has been 8.0 (24/3), a 60% elevation above the 100-year average. These data as of 12/31/2015 trend to an expected count of 10 for the decade 2010-2020, as predicted in Bowker Chambers 2015. Important to note that since completing Bowker Chambers (2015) we discovered 7 additional Very Serious Failures as of 12/31/2009, indicating the degree of underreporting in the ICOLD and WISE compilations even for the most significant TSF failures. It was reported by Kiernan (2016b) that ICOLD spokesman Emmanuel Grenier said "ICOLD doesn't include the structures in its 58,000-entry World Register of Dams due to internal concern that their high failure rates would tarnish the reputation of all dams." Meanwhile slurry depositions at upstream earthen dams as at Samarco, continue to be planned, built and used. It is our hope that this paper will help bring large tailings dams into a common practical understanding and advocacy for the engineering and stewardship it takes for very large dams to avoid accrual of public liability at a level beyond funding or recovery.

2.2. Severity Distribution by Cause of Failure Code

Table 1, based on all recorded/known failures through 12/31/2015, shows Overtopping Failures at active mines in the context of all other cause of failure codes developed by ICOLD. This table is for the 125 incidents with listed causes of failure at operating mines recorded out of 267 Very Serious, Serious, and Other Failures recorded in our data base. The data in this table is also presented graphically in Figure 1. With the exception of Erosion (ER) and Foundation Failures (FN) which are low-frequency high-severity, the distributions by severity across the other 5 codes are very similar. Also the pattern by cause of TSF failure code at active mines (N= 125, excluding inactive failures) is the same as the severity distribution for the 267 records in Table 1 (N=267, inclusive of inactive mines), again pointing to a common root cause of severity of consequence beyond the coded cause of failure or the operating status of a mine at failure.

Table 1. Failure Modes by Severity Classification for Active TSF Failures

TSF FAILURE MODE BY SEVERITY CLASS (N=125)

ICOLD Classification	Active-EQ	Active-ER	Active-FN	Active-OT	Active-SE	Active-SI	Active-ST	Total Active (N=125)	Total All (N=267)
Cause → Failure ↓ Severity ↓	Earthquake	Erosion	Foundation	Over-toppng	Seepage	Slope Instability	Structural	TSF Failures by Severity	TSF Failures by Severity
Very Serious	2 (9%)	2 (33%)	4 (29%)	3 (10%)	2 (14%)	5 (19%)	3 (21%)	21 (17%)	41 (15%)
Serious	4 (18%)	2 (33%)	1 (7%)	5 (17%)	3 (21%)	4 (15%)	5 (36%)	24 (19%)	48 (18%)
Other	16 (73%)	2 (33%)	9 (64%)	21 (72%)	9 (64%)	17 (65%)	6 (43%)	80 (64%)	178 (67%)
TOTAL	22 (100%)	6 (100%)	14 (100%)	29 (100%)	14 (100%)	26 (100%)	14 (100%)	125 (100%)	267 (100%)

2.3. The Economics of Best "Waste Care" Practices in Ever Decreasing Global Grades

The economics of the Mining Metric 1910 to 2010, the basis of Bowker Chambers 2015, are shown in Figure 3. The term Mining Metric refers to a hypothesis often attributed to Edgar A. Scholz that production volume and scale of extraction would enable profitable mining of the globally depleted quantity of quality reserves even against a trend of falling prices. At the aggregate global level, that worked until about 1990. As is clear from Figure 3, which uses copper to show the same pattern present in all metals, falling prices were offset by falling ore production costs which absorbed the extra costs of increased ore production needed to attain the same level of finished metal output as grades fell steadily. Mining economist Richard Schodde's analysis showed that ore production costs declined 4-fold while milling costs declined 2-fold over the century (Schodde 2010). The decline in ore production costs was the key to threading the needle of profitability in an overall trend of falling prices, and falling grades which necessitated ever higher volumes of ore production and mill throughput to generate a given desired output of final product.

Larger higher grade mines had an easier run up until 1990 compared with the many smaller, inexperienced, less well financed miners who came into production after 1950. Those mines and miners were not as able to attain a satisfactory margin because of lower grades and generally higher production costs due to their more limited ability to achieve Scholz's economies of scale. They showed their vulnerability through more frequent periods of standby. None of the mines Scholz himself developed according to his hypothesis ever actually attained profitability. Taseko and its sole asset in development, Gibraltar being the latest example. Gibraltar was one of the original Scholz mines.

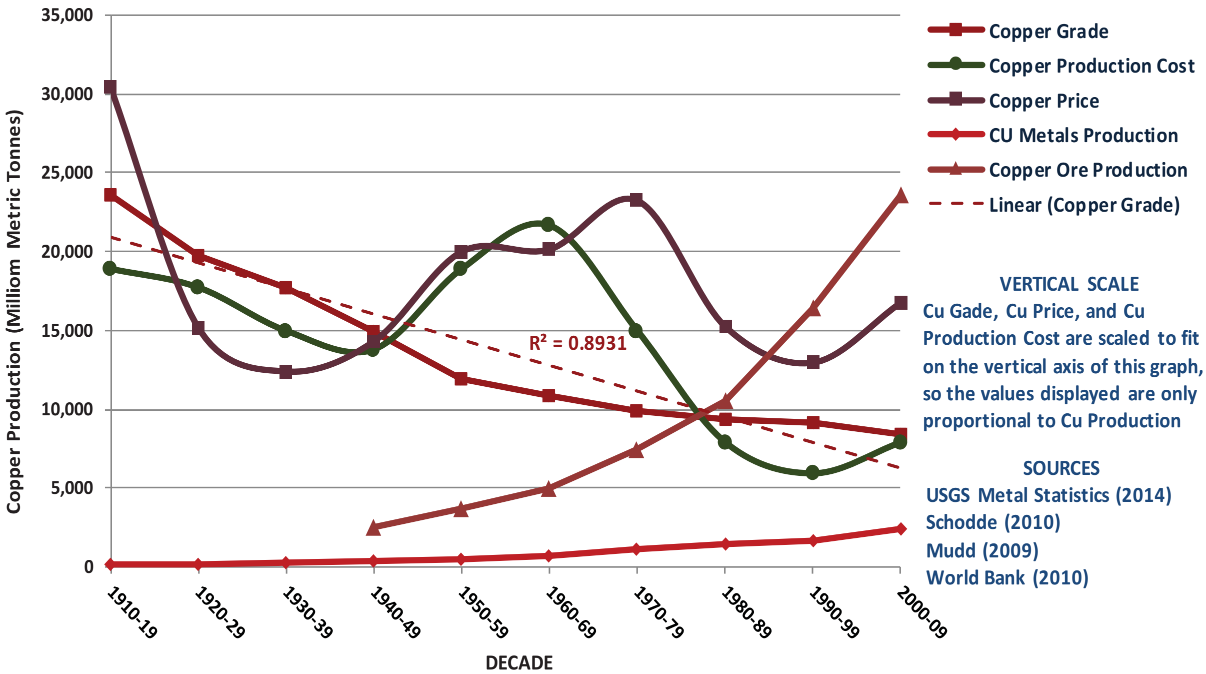

Figure 3. Mining Metric 1910-2010 Declining Prices Offset by Lower Production Costs

In 1990 the sustaining relationship between falling prices and concurrently falling ore production costs changed, both heading upward but with continued declines in grade and a very significant upward swing in ore production volumes needed to offset those lower grades. Some overproduction also occurred in this "super cycle" so the huge upward swing in ore production is not 100% attributable to continuing decline in grade. It is in this period of change in the relationship between as-milled grade and ore production volume that the trend to increasing severity of failures emerged. It is important to note that while there was an upswing in copper price, prices in real dollars ($2009) during the super cycle never reached their previous cycle-high in 1970. This limited recovery in price viz. historic prices, in combination with upward swing in ore production costs, set up the economic squeeze manifesting as an emerging trend of higher severity failures. As at Samarco, the largest tailings storage facility failure in recorded history, the manifestation of the economic squeeze is no longer limited to small mines and junior and mid-sized miners.

Obviously the high volume/lower cost ore production possible through open pit mining and the need that created for bigger and higher tailings facilities dramatically increased the risk profile of the global tailings inventory as noted by Dr. A. Mac G. Robertson in his key note address at the 2011 tailings conference (Robertson 2011). However, what we can see from examining reports on existing mines is that increased tailings capacity is being created at older tailings storage facilities with smaller footprints, not by the design and development of new TSFs specifically engineered to handle the higher volumes, longer lives, and higher throughputs. Major throughput expansions rarely include a systematic reevaluation of existing TSF capacity, or a reevaluation of tailings management needs inherent in the planned expansion. This is the Samarco failure story. There was no capacity and no plan or space to create capacity. They knew as a result of much lower grades the level of fines in the tailings generated for the two years prior to failure precluded use of dry stack and/or paste, as a way to get more and safer life out of the existing modern era facility which went on line in 2009 (Amira et. al. 2010). Vale vigorously denounced Brazil's plan to ban large upstream dams like the Fundao acknowledging that mining ore in Minas Gerais was not possible economically with such a ban (Eisenhammer and Nogueira 2016). Brazil backed down.

2.4. The Link Between Severity of Failure, Grade of Ore and Production Costs

The methods used to establish the relationship between the Mining Metric and the emergence of a trend of increasing severity of failures is shown in Bowker and Chambers (2015) and its technical appendices.

Our final input data set for this analysis is shown in Table 2, and is also available in machine readable form at the CSP2 website along with more comprehensive technical documentation.

Table 2. Input Data Set

Decade	Very Serious Failures	Serious Failures	Other Failures	Other Accidents	Non-Dam Failures	All Failures	Cu Prod (K tonnes)	Cu Grade (%)	Cu Prod Cost $/tonne	Cu Price $/tonne
1940 – 49	0	0	5	0	0	6	2545	1.52	$35	$3,633
1950 – 59	1	0	7	0	0	7	3680	1.21	$48	$5,076
1960 – 69	3	4	25	17	2	51	5004	1.1	$55	$5,112
1970 – 79	4	8	23	15	3	53	7445	1.01	$38	$5,895
1980 – 89	5	9	22	14	4	54	10575	0.95	$20	$3,871
1990 – 99	9	9	10	3	1	32	16437	0.93	$15	$3,292
2000 – 09	7	8	5	1	0	21	23658	0.85	$20	$4,256
	======	======	======	======	======	======	======	====	======	======
Total/Ave	29	38	97	50	10	224	69,344	1.54	$33	$4,448

Sources: USGS Metal Statistics (2014), Schodde (2012), ICOLD (2001), WISE & additional

Abbreviations:

Very Serious Failures = Very Serious tailings dam failures

Serious Failures = Serious tailings dam failures

Other Failures = tailings dam failures and incidents other than Serious or Very Serious Failures

Cu Prod = copper ore production (thousand metric tonnes)

Cu Grade = grade of copper in the ore (%)

Cu Prod Cost = Cost to produce copper concentrate from copper ore, including waste disposal ($/tonne)

Cu Price = Copper price ($/tonne)

The key relationships between severity and Mining Metric elements were identified in an analysis of univariate correlations as shown in Table 3. In analysis leading up to 1990, costs of production canceled out price (as Schodde had correctly concluded in his analysis) and had stronger relationships with severity.

Table 3. Univariate Correlation Between Mining Metric Variables and Severity Variables

Variables	Cu Ore Production	Cu Grade	Cu Prod Cost
Very Serious Failures	0.860	-0.794	-0.788
Serious Failures	0.720	-0.884	-0.682

Abbreviations:

Very Serious Failures = Very Serious tailings dam failures

Serious Failures = Serious tailings dam failures

Cu Prod Cost = Cost to produce copper concentrate from copper ore, including waste disposal

Cu Grade = grade of copper in the ore

Cu Prod = copper ore production

Sources: USGS Metal Statistics (2014), Schodde (2010), ICOLD (2001), WISE (2015) & additional

While all three mining metric variables: ore grade; ore production volume; and, unit costs for ore production, correlated highly with both Very Serious and Serious Failures, in univariate analysis it is interesting and important to note the different patterns of correlation between the two severity classes. Very Serious Failures have a much stronger relationship with ore production volume and costs of production as compared with Serious Failures which have a stronger relation with as-milled grade. These are exceptional correlations with the economic descriptors of the Mining Metric.

Exploring these relationships further to see how much of the variation in severity was accounted for by each of the key Mining Metric variables, we used canonical correlation analysis (Bowker and Chambers 2015). Canonical Correlation Analysis (CCA) is almost universally defined as "the problem of finding two sets of basis vectors, one for data set Y1 (the two high severity classes) and the other for data set Y2, (the Mining Metric variables) such that the correlations between the projections of the variables onto these basis vectors are mutually maximized."

CCA seeks a pair of linear transformations, one for each of the sets of variables such that when the set of variables are transformed the corresponding co-ordinates are maximally correlated. The linear transformations are synthetic variables. One "synthetic variable" or canonical variate is created for each data set (F1, F2 for our two sets), as shown in Table 4.

Table 4. Correlations between Input Variables and Canonical Variables

Y1 Data Set of Severity by Decade				Y2 Data Set of Mining Metric Values by Decade		
	F1	F2			F1	F2
Very Serious Failures	-0.922	-0.388		Ore Production	-0.802	-0.558
Serious Failures	-0.995	0.096		Copper Grade	0.929	-0.072
				Copper Cost	0.755	0.45

The principal output of a canonical correlation analysis are the canonical functions (variates) which seek to maximize explained variability between the two arrays (Y1 and Y2). Each function produced is an equation (similar to the equations created in regression analysis) but instead of explaining the relationships in terms of causality, it seeks to define the dimension (strength) of the relationship between (or in larger data sets among) the arrays. Essentially it asks are these arrays independent of one another, or does there appear to be an influence of the two arrays on one another. As many canonical functions are produced as there are variable sets (in our case 2).

The proper use of canonical correlation analysis for descriptive analysis requires no assumptions of distribution. To test the significance of the relationships between canonical variates, however, the data should meet the requirements of multivariate normality. We were not able to conduct a full multivariate normality as it was not an option in XLSTAT©. The normality of each variable within the data set is not a proof of multivariate normality, but all elements of a data set that does meet the requirements of multivariate normality must meet univariate tests of normality. All variables met Jarques-Bera normality test, a test most frequently applied in economics.

The first exploration of these canonical functions is the eigenvalue which measures how much variability is explained by each of the canonical functions. The closer the eigenvalue is to zero the less likely the two arrays form a diagonal matrix, i.e. have a linear correlation to one another which might therefore be suitable for linear modeling (regression analysis).

As can be seen in Table 5, the canonical correlation itself, 0.950 accounted for 95% of the variability between the two data sets. In this case the Eigenvalue for the first canonical function, F1 = 0.903, strongly indicates a diagonal matrix.

Table 5. Canonical Correlation Values

	F1	F2
Canonical Correlation	0.950	0.727
Eigenvalue	0.903	0.528
Wilks' Lambda	0.046	0.472

Wilks' Lambda is a test of the null hypothesis that the data sets are independent of one another as measured via the canonical coefficients. The lower the Wilk's Lambda, the less likely that the data sets Y1 and Y2 are independent. The results, a Lambda of 0.046, significant at a 92% confidence level means it is unlikely that the two data sets are independent of one another. That is, if accepted the null hypothesis that the two data sets are independent of one another there is a 92% chance we'd be wrong. A further explanation of canonical correlation analysis and its interpretation may be found in Appendix II of Bowker and Chambers (2015). As evident in Table 5, which includes the correlations for the second factor as well (F2), it added little to the overall understanding of the mutual dependence between severity and Mining Metric.

It is reasonable to infer that the influences of declining grades, production cost, and ore volume produced (virtually same as tailings volume) on severity of consequence exist also at the level of "cause of failure" for overtopping and all other coded causes of failure. The Mining Metric has a strong linear relation with each of the two high severity classes for all such recorded significant failures and incidents. Therefore, the Mining Metric must also have this same linear relationship with severity at the level of cause of failure code. How this works at the mine level is illustrated in the discussion below, which places some well-known overtopping events in the context of the global economics of the Mining Metric.

2.5. Economic Root Causes of Well Known Overtopping Failures: Grade as a Key Root Determinant

The fundamentals of how this plays at the mine level is simply and succinctly expressed by Andrey Dashkov, Senior Analyst, Casey Research**:** "As a project moves to the development stage, the higher the grade, the more robust the projected economics of a project. And for a mine in production, the higher the grade, the more technical sins and price fluctuations it can survive." (Dashkov 2013). Continuing in this analysis Dashkov goes on to declare that volume and throughput (the Scholz foundation for profitability of low grade mines) is "no longer king," and that grade is "now king" in determining which mines will be successful and which will fail. This was essentially validated by Bowker and Chambers (2015) as the context and main driver of the emerging prevalence of catastrophic failure. This applies equally to failure by any of the 8 causes of failure, including overtopping.

Dashkov's analysis that a grade advantage is a critical determinant of ability to survive serious technical flubs and dramatic unpredictable price fluctuations, a norm for all metals, means that smaller, lower grade mines will suffer more and have more physical manifestations of their economic stress than larger, higher grade mines. Very simply, smaller, lower grade mines operated by junior and midsize miners have no cushion. They have to ride too close to the edge of financial viability viz. global metals markets and major producers to stay in production. They also have less access to high quality capital markets, paying more and operating under more onerous terms of credit than the top producers at higher grade mines, a factor that George Ireland has frequently cited as creating financial instability and uncertainty when the due dates of credit don't match up with cash flows needs, expected revenue generation, and production capacities of the mine. This mismatch can actually lead to failure or involuntary investor takeover elevating uncertainty and instability (Sylvester 2012).

In gold, as a respected analyst Mark Fellows explains, a 10% fall in global average ore grade gives rise to a \$50/oz rise in average global production costs (Fellows 2010). At the mine level, a difference between a gold mine with 1.72g/t and 2.2 g/t translates to a likely cost difference of \$100/oz in total production costs. These are the actual differences at the Gold Ridge mine, Guadalcanal, in 2009. This mine never achieved profitability, not because of political unrest, but because the low quality of the deposit compared to the quality of ores shaping world markets. Gold Ridge, with a 20 million cubic meter capacity tailings storage facility with a long history of many owners, frequent interruptions, and continually falling recovery rates (another emerging consequence of mining very low grade ores), under ownership of landowners with no technical competence, has hovered on the brink of complete failure by overtopping for two years. Blight and Fourie (2004), George Ireland (Sylvester 2012) and Irwin Wislesky (Moore 2016), among others, all cite technical competence, technical mistakes, and caliber of mine operators as an unexamined and significant back drop to mine failures.

Merriespruit, South Africa, is one of the most famous and examined cases of overtopping failure (1994) in history. The mine's long demise and its many manifestations of economic stress over the entire course of Harmony Gold's ownership illustrates another aspect of how an economic squeeze shaped by global markets and major global producers affects the viability of a given mine. The dam that failed began construction in 1978 (Fourie 2011), at the end of a period of South Africa's global dominance in gold. "As a proportion of world production (excluding that of the U.S.S.R.), South Africa's production peaked in 1971 at 79.1 per cent. [it fell} consistently, to a 1985 level of 55.8 per cent, under the combined influence of declining total South African production and increasing output from elsewhere, particularly North America, Australia, and Brazil. As a percentage of new world supply, which includes imports from the Communist sector, South Africa's contribution decreased from 78.7 per cent in 1970 to 47.3 per cent in 1985" (Janisch 1986). South African gold mines still had exceptional grade compared to emerging producers, but Merriespruit and most others were all costly underground mines whilst the emerging major markets were all open pit mines. The economic advantage of much higher grade wasn't enough to overcome the disadvantage of much higher costs to extract and process it compared to other emerging markets and top producers, placing further economic stress on Merriespruit and all South African mines.

3. CONCLUSIONS

As we see most stunningly at Samarco's largest failure ever in recorded history, without clear standards in law and regulation viz. best available technology and best applicable practices, and adequate competent independent life of facility oversight, efforts to attain profitability will continue to lead to choices at the mine level that can eventually lead to catastrophic loss. As we see at Mt Polley, that eventual catastrophe could emerge as any one of several causes of loss. Mt Polley could have been an overtopping failure of the same magnitude as the foundation failure. Samarco's Fundao dam, it is important to note, was only put in service in 2009, had an Independent Tailings Review Board, and highly regarded expert advisers. In British Columbia, Imperial Metals was found not to be in violation of any law and regulation, even though its economically driven deviations from best available technology and best applicable practices culminated as one of the 10 largest failures ever in recorded history. Similar economic forces and economically driven decision making that maximize throughput have shaped the wrong choices that have led to very serious failures throughout recorded history.

Preventing overtopping is an essential and key part of global overall loss prevention. Identifying best technology and best practice for achieving overtopping prevention is essential to preventing accruals of public liability, protecting investors, local communities, and non-replaceable natural resources. Most importantly a better understanding of causes of loss establishes a basis for intervention in time to prevent tailings dam failures. If we can recognize the early warning signs that could evolve to catastrophic loss, we will be able to address actions and changes to prevent loss. We can expect future losses to routinely exceed the severity of Mt. Polley. With so many very large upstream tailings dams in use and continuing to be authorized, we can also expect others to fail at the scale of Samarco, a very small impoundment compared to others standing and in operation.

Law and the use of truly independent expert panels speaking for the public interest cannot operate effectively without this continual collaborative work. In this paper we are suggesting that best available technology and best applicable practices must be partnered with reforms in law and regulation, and an awareness of the role of economics to build solutions that will save investors, communities, and natural resources from the unfundable damages of catastrophic failure at super-sized tailings storage facilities. If permits continue to be given to mines that are not economically competitive in the present and emerging global markets, even with apparent or agreed compliance with best available technology and best applicable practices, there will be economic pressure on mine operators to bend the rules. Only truly independent life of mine review and oversight, extending the technical knowledge, and developing the understanding and capacity of regulators can prevent further dam failures.

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Fundão Tailings Dam Review Panel

Report on the Immediate Causes of the Failure of the Fundão Dam

Panel:
Norbert R. Morgenstern (Chair)
Steven G. Vick
Cássio B. Viotti
Bryan D. Watts

August 25, 2016

EXECUTIVE SUMMARY

The Fundão Tailings Dam failed on November 5, 2015 in a liquefaction flowslide that initiated at the dam's left abutment. This Investigation was performed to determine its cause.

In structuring its investigation process, the Panel systematically identified and evaluated multiple causation hypotheses. It further imposed hypothesis testing by means of the following three questions that the candidate failure mechanism should be able to explain:

1. Why did a flowslide occur?
2. Why did the flowslide occur where it did?
3. Why did the flowslide occur when it did?

Forensic methods adopted by the Panel integrated multiple lines of evidence: observations from eyewitness accounts; data and imagery in geographic information system (GIS) format; field evidence from subsurface exploration by the Panel and others; advanced laboratory testing; and sophisticated computer modeling. Responding to the above three questions for hypothesis testing demanded a high level of quantification and exhaustive detail in each of these aspects of the Investigation's evidence-based approach.

To understand the failure first requires understanding the materials the dam contained and their properties. There were two types of tailings, both produced in slurry form and delivered in separate pipelines to the Fundão impoundment. *Sand tailings,* or simply *sands,* are a mixture of sand-sized and finer silt particles. The sands are relatively free-draining, but when loose and saturated are susceptible to *liquefaction*, a process whereby the material loses nearly all of its strength and flows as a fluid. The *slimes*, on the other hand, are much finer and clay-like in nature—soft and compressible with low permeability. How these two materials interacted is key to understanding the failure.

Another central aspect is how their deposition was influenced by a series of unplanned occurrences during the dam's construction and operation. Together, these incidents established the conditions that allowed the failure to take place. These included: (1) damage to the original Starter Dam that resulted in increased saturation; (2) deposition of slimes in areas where this was not intended; and (3) structural problems with a concrete conduit that caused the dam to be raised over the slimes.

It was originally planned to deposit sands behind a compacted earthfill Starter Dam, then raise it by the upstream method to increase progressively its capacity. These sands, in turn, would retain slimes deposited behind them such that the two materials would not intermingle. To preserve the free-draining characteristics of the sands, a 200 m beach width was required to prevent water-borne slimes from being deposited near the dam crest where they would impede drainage. A high-capacity drainage system at the base of the Starter Dam would allow water to drain from the sands, reducing saturation.

The first incident occurred in 2009 shortly after the Starter Dam was completed. Due to construction defects in the base drain, the dam was so badly damaged that the original concept could no longer be implemented. Instead, a revised design substituted a new drainage blanket at a higher elevation.

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Together with the revised design there was a fundamental change in the design concept whereby more widespread saturation was allowed and accepted. This increase in the extent of saturation introduced the potential for sand liquefaction.

The second incident associated with slimes and water management occurred over an extended period of time in 2011 and 2012 while the new design was being constructed. During operation, the 200 m beach width criterion was often not met, with water encroaching to as little as 60 m from the crest. This allowed slimes to settle out in areas where they were not intended to exist.

Another incident occurred in late 2012 when a large concrete conduit beneath the dam's left abutment, the Secondary Gallery, was found to be structurally deficient and unable to support further loading. This meant that the dam could not be raised over it until it had been abandoned and filled with concrete. In order to maintain operations in the interim, the alignment of the dam at the left abutment was set back from its former position. This placed the embankment directly over the previously-deposited slimes. With this, all of the necessary conditions for liquefaction triggering were in place.

As dam raising continued, surface seepage began to appear on the left abutment setback at various elevations and times during 2013. The saturated mass of tailings sands was growing, and by August, 2014 the replacement blanket drain intended to control this saturation reached its maximum capacity. Meanwhile, the slimes beneath the embankment were responding to the increasing load being placed on them by the rising embankment. The manner in which they did so, and the consequent effect on the sands, is what ultimately caused the sands to liquefy.

As the softer slimes were loaded, they compressed. At the same time, they also deformed laterally, squeezing out like toothpaste from a tube in a process known as *lateral extrusion*. The sands immediately above, forced to conform to this movement, experienced a reduction in the horizontal stress that confined them. This allowed the sands to, in effect, be pulled apart and in the process become looser.

To replicate this process in the laboratory, the Panel applied these stress changes to the Fundão sand. The saturated specimen completely and abruptly collapsed, losing nearly all its strength—a laboratory demonstration of liquefaction. The Panel then undertook a program of numerical modeling to determine whether stress changes similar to those imposed in the laboratory would have also occurred in the field. Using computer simulation of how the slimes deformed during embankment construction, and tracking the corresponding response of the sands, comparable stress conditions that caused the sands to liquefy in the laboratory were reproduced computationally. Simply put, what is known to have occurred during the failure was replicated in the laboratory, and what occurred in the laboratory is shown to have occurred at the left abutment of the dam.

A related aspect of the failure was the series of three small seismic shocks that occurred about 90 minutes earlier. By then the left abutment of the dam had reached a precarious state of stability. Computer modeling showed that the earthquake forces produced an additional increment of horizontal movement in the slimes that correspondingly affected the overlying sands. Although the movements are quite small and the associated uncertainties large, this additional movement is likely to have accelerated the failure process that was already well advanced.

Hence the failure of the Fundão Tailings Dam by liquefaction flowsliding was the consequence of a chain of events and conditions. A change in design brought about an increase in saturation which introduced the potential for liquefaction. As a result of various developments, soft slimes encroached into unintended areas on the left abutment of the dam and the embankment alignment was set back from its originally-planned location. As a result of this setback, slimes existed beneath the embankment and were subjected to the loading its raising imposed. This initiated a mechanism of extrusion of the slimes and pulling apart of the sands as the embankment height increased. With only a small additional increment of loading produced by the earthquakes, the triggering of liquefaction was accelerated and the flowslide initiated.

Immediately following this Executive Summary is an inventory of structures and their locations to help the reader become oriented to the various features associated with the site.

INVENTORY OF STRUCTURES

Term	Figure Reference
Alegria Mine	1
Auxiliary Foundation (Base) Drain	2
Conveyor	1
Dike 1	1
Dike 1A (a.k.a. Old Dike 1A)	2
Dike 2	1
El. 826 m Blanket Drain	2
Fabrica Nova Waste Pile	1
Fundão Dam	1
Germano Buttress	1
Germano Main Dam	1
Grota da Vale	1
Kananets®	2
Left Abutment (LA)	2
Main Gallery	2
Overflow Channel	2
Plateau	2
Principal Foundation (Base) Drain	2
Reinforcement (Equilibrium) Berm	2
Right Abutment (RA)	2
Santarem Dam	1
Secondary Gallery	2

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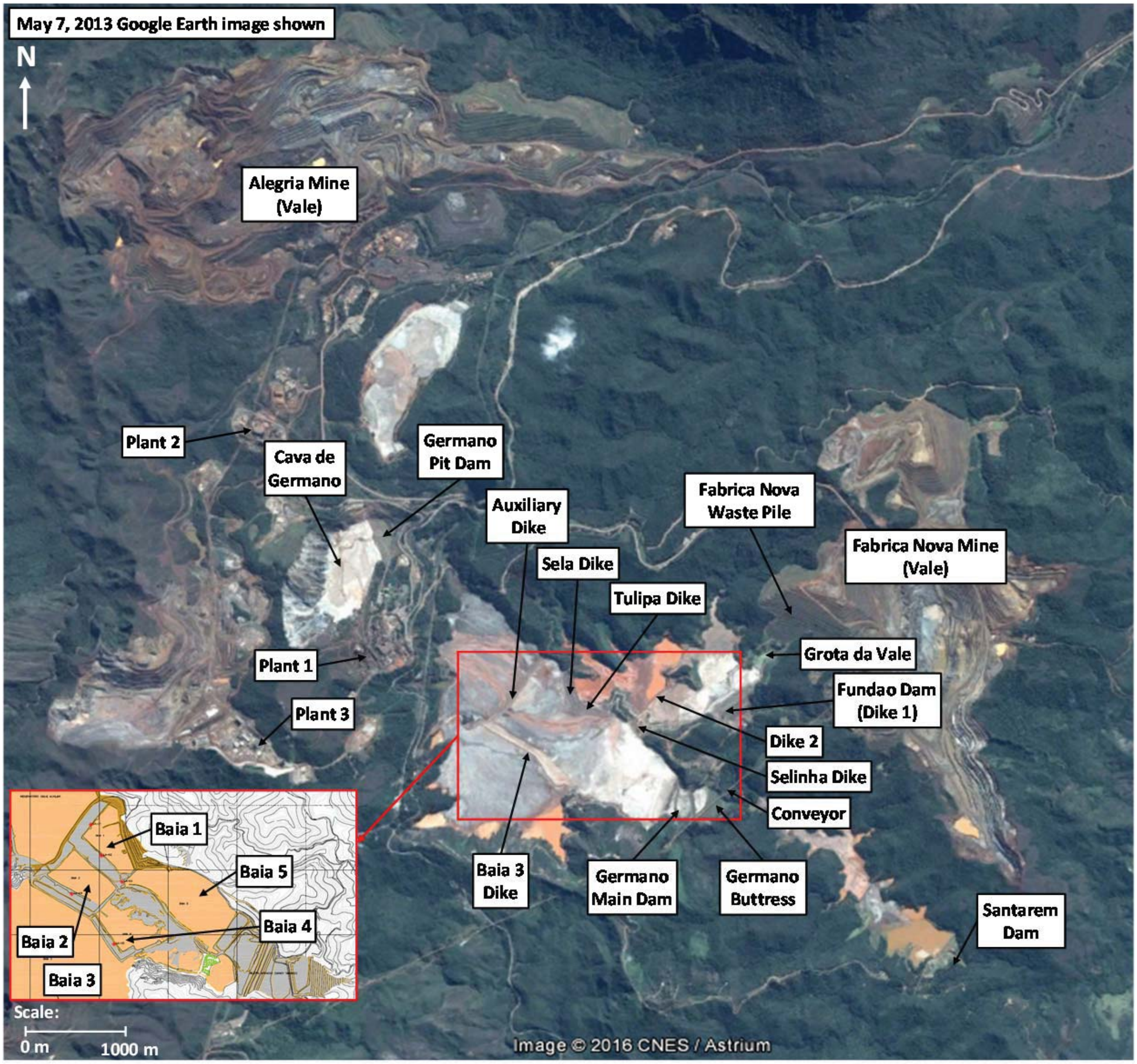

Figure 1 Inventory of structures – Samarco Site

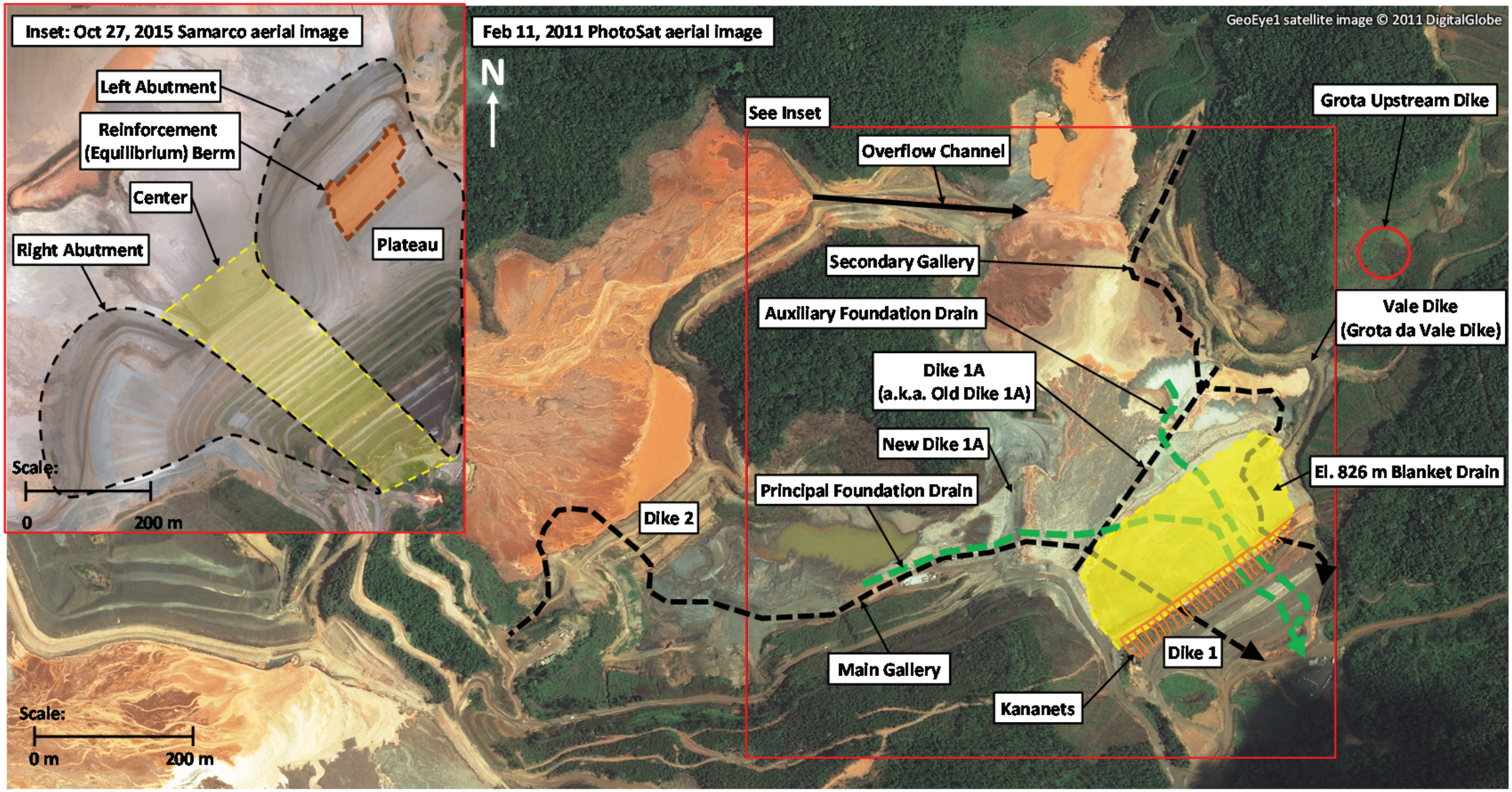

Figure 2 Inventory of structures – Fundão Dam

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1 INTRODUCTION

1.1 The Failure

On the afternoon of November 5, 2015, the Fundão Tailings Dam in Minas Gerais collapsed. Its crest had reached El. 900 m, making the dam 110 m high. Several dozen people were working on or near the dam at the time. Some were hauling and spreading tailings for raising the dam, others were constructing gravel blanket drains in anticipation of the next stage of construction, and still others were engaged in the daily activities required to operate and maintain the tailings system.

Sometime after about 2:00PM[1] many in the Germano plant complex felt a tremor lasting several seconds. Although windows rattled and objects fell from tables, there did not appear to be any serious damage. Work resumed.

At 3:45PM shouts came over radio that the dam was collapsing. A cloud of dust had formed over the left abutment[2], and those closest to the area designated the “setback” could see cracks forming at the recently-constructed drainage blanket. The slope above them was beginning to undulate “like a wave” as if it were “melting,” bringing the dam crest down after it. The tailings that had been solid ground just minutes before transformed into a roiling river, overtopping but not breaching the downstream Santarem Dam, then entering the town of Bento Rodriguez shortly thereafter enroute to its ultimate destination in the sea.

Eyewitness descriptions and videos definitively establish several things. The first is that the Fundão failure initiated at the dam’s left abutment, not at the right side or its downstream toe. The second is that the failure occurred due to flow liquefaction of the tailings, a process whereby water pressures in the interstitial voids between the tailings particles increased to such an extent that the mass of material lost strength and flowed like a liquid. And third is that this transformation from solid to liquid was complete and abrupt, leaving a fluid of apparent viscosity and hydraulic behavior little different from water in just seconds.

The question remains as to what triggered liquefaction and what factors promoted its occurrence. That is the focus of this report.

1.2 The Investigation

This Investigation of the Fundão Tailings Dam failure was commissioned by BHP Billiton Brasil Ltda., Vale S.A. and Samarco Mineração S.A. The firm of Cleary Gottlieb Steen & Hamilton LLP (CGSH) was engaged to conduct the Investigation with the assistance of a panel of experts. The Fundão Tailings Dam Review Panel (Panel) includes four members, all specialist geotechnical engineers in water and tailings dams: Norbert R. Morgenstern (Chair), Steven G. Vick, Cássio B. Viotti, and Bryan D. Watts.

[1] All times in this report refer to local Brazilian time.

[2] The conventions *left* and *right* indicate direction, location, or orientation as seen by an observer looking downstream. The left and right *abutments* are where the constructed dam meets the respective valley sides.

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The Panel's Terms of Reference defined the scope of its activities. Specifically, the Panel was instructed to provide its independent and unbiased professional judgment and expertise in determining the immediate cause(s) of the incident.

In accomplishing this purpose, the Panel could examine any or all of the following:

- geotechnical designs of the Fundão Tailings Dam and structures associated with the dam, including both intact and breached embankments, and including both the original design and all lifts of the embankment structure;
- interpretation of results of geotechnical investigations and associated laboratory testing of the Fundão Tailings Dam;
- patterns, trends, and relationships in instrumentation behavior of the Fundão Tailings Dam;
- interpretation of instrumentation and performance data in relation to the Fundão Dam's behavior;
- materials, methods, procedures, and quality assurance/quality control practices for the construction and modification of the Fundão Dam;
- water balance and water quality as they relate to the incident;
- seismic activity in the region on the day of the incident;
- operational procedures and planning for tailings deposition and water management at the Fundão Dam;
- inspection and surveillance procedures and implementation, including reports issued by the Independent Tailings Review Board (ITRB) and other outside auditors;
- the Engineer of Record's field reviews;
- issues identified by the National Department of Mineral Production (DNPM) and the Brazilian federal and state environmental agencies in the course of their oversight;
- the design and structure of other similar tailings dams in the vicinity; and
- other matters the Panel deems appropriate to be examined.

Seismologists Gail Atkinson and Ivan Wong provided the Panel with input in their field of expertise. The firm of Klohn Crippen Berger provided analytical, field, and laboratory support, and the firm of TÜV SÜD provided local assistance in Brazil.

The Panel was provided with available information and witnesses necessary to achieving its purpose. The Panel was asked not to assign fault or responsibility to any person or party, or to evaluate environmental or other downstream effects or damages. None of the Panel members had performed previous work for Samarco or was currently engaged in any other assignment for BHP Billiton Brasil or Vale during the conduct of the Investigation.

During the course of the Investigation, the Panel conducted the following activities:

- site inspection and meetings;
- meetings with eyewitnesses and technical personnel;
- compilation and review of project documents;
- assembly of GIS data and imagery;
- reconstruction of tailings stratigraphy;
- compilation and assembly of pre-failure subsurface and laboratory data;
- subsurface investigations at the site and laboratory testing;
- compilation and interpretation of instrumentation data;
- analytical studies:
  - seepage modeling;
  - consolidation modeling;
  - stability analysis;
  - deformation analysis; and
  - dynamic response analysis.
- geologic assessment;
- fault tree analysis; and
- preparation of this report.

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2 HISTORY

This section provides a compilation of historical facts and circumstances considered by the Panel to be most relevant to understanding the failure, with particular emphasis on the left abutment where the failure is known to have initiated. The complete history is much more extensive, and no attempt is made here to review it in its entirety.

2.1 The Concept (2004-2007)

Beneficiation of iron ore at Samarco's Germano Complex results in two distinct kinds of tailings produced and transported in slurry form as separate streams. *Sands*, or sand tailings, are actually composed of both sand and silt-sized particles in roughly equal proportion. During deposition, they form a gently-sloping beach through which transport water drains fairly rapidly. *Slimes,* on the other hand, are fine-grained and clayey in nature. The clay-sized particles remain suspended and eventually settle in standing water to produce a softer material of lower permeability.

At Germano, a way was devised to use these two types of tailings and their different characteristics to best advantage. The sands were deposited to form a buttress or "stack" that retained the slimes discharged separately behind it. The sands, in turn, were retained by an earthfill or rockfill *starter dam* at the downstream toe of the stack, as illustrated on Figure 2-1 for Samarco's Germano Buttress structure. Over time, the Germano Starter Dam was raised according to the *upstream method* or *upstream construction*. With this procedure, the dam crest moves progressively upstream over previously-deposited tailings as the dam is raised.

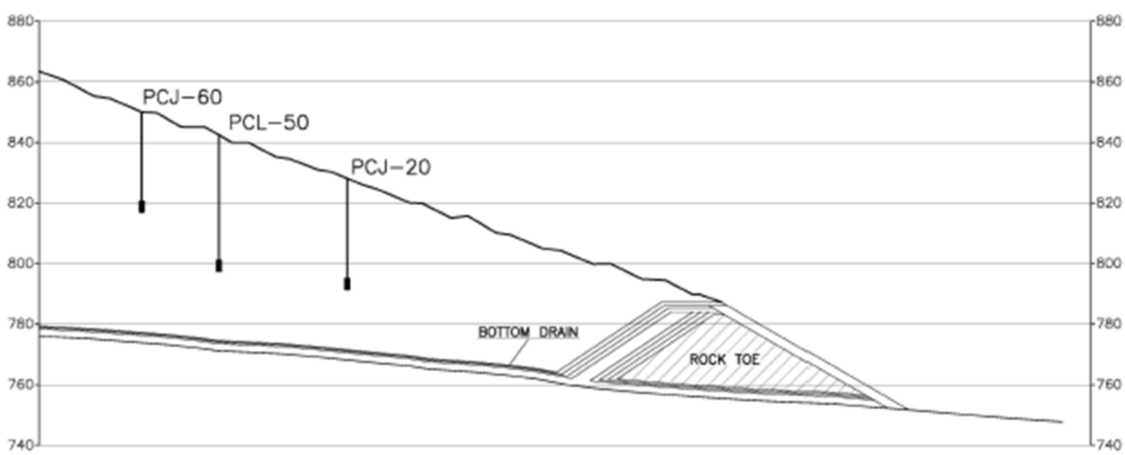

Figure 2-1 Germano Buttress (Pimenta de Ávila 2011)

Adequate drainage of the sands was the key to this concept. Figure 2-1 shows that drainage was promoted by highly-pervious bottom drains underlying the sand and extending beneath the Starter Dam to prevent water from accumulating and saturating the deposit. The absence of any significant water pressure was to be confirmed with the piezometers shown in the figure. Provided that no slimes were present to impede downward drainage and that the sands remained unsaturated, resistance to liquefaction—a well-known vulnerability of upstream construction—could be assured.

By 2005, the existing tailings facilities at Samarco's Germano operation were nearing capacity, and a new third pellet plant would increase production of both sand and slimes. The adjacent Fundão Valley was chosen as a new tailings site. In the layout that emerged, the sands and slimes would initially be physically separated, with sands deposited behind Dike 1 and slimes behind Dike 2, as represented on Figure 2-2. Retention of the slimes required that the sands deposited between the two dikes always remain at a higher elevation throughout the raising process. This was a matter of reservoir geometry, and the dikes in Figure 2-2 had been strategically positioned for sands and slimes in 70% and 30% proportion of the total received from all plants.

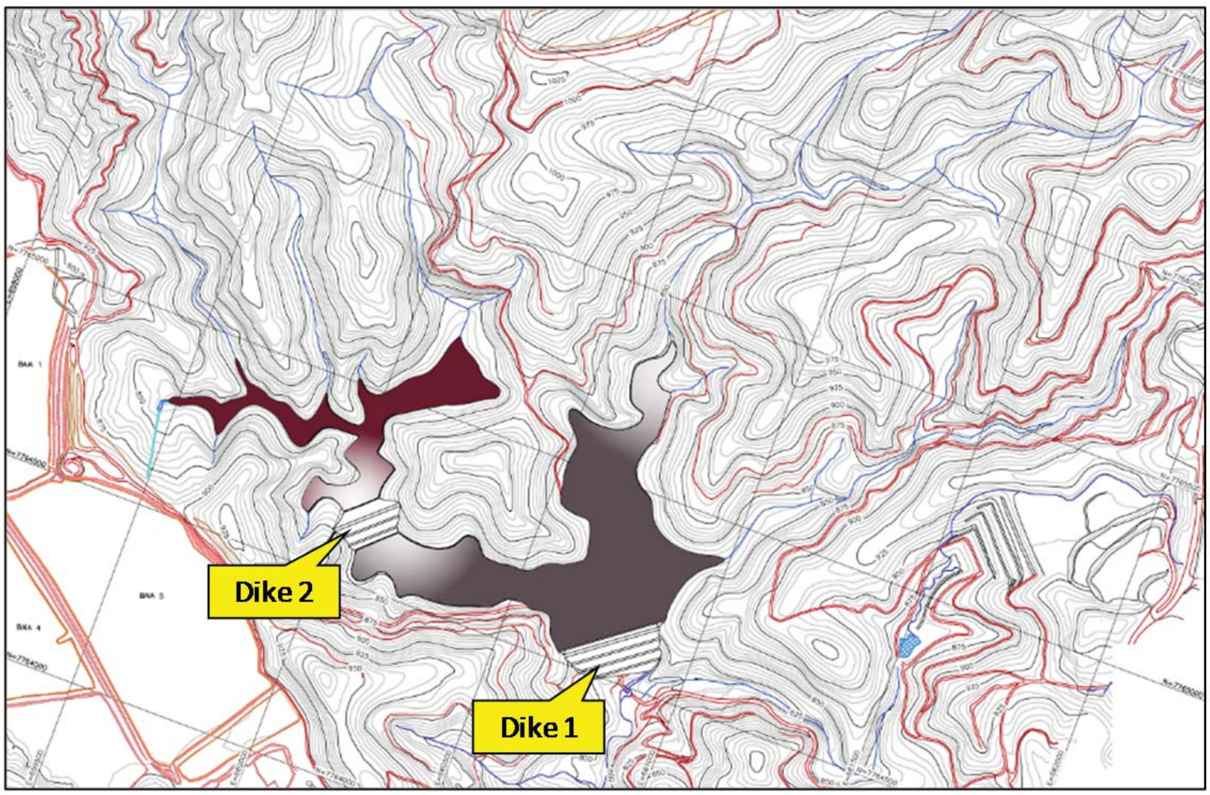

Figure 2-2 Fundão Dikes 1 and 2

Two alternative methods were considered for raising Dike 1 after filling the space between the two dikes with sand. One was *centerline* raising depicted on Figure 2-3 using compacted sand tailings in the downstream slope. This alternative was not selected, with the drained stack concept shown on Figure 2-4 adopted instead. The Dike 1 Starter Dam would be a conventional earthfill structure constructed of compacted saprolite soils to crest El. 830 m, with subsequent upstream raising with sand tailings to El. 920 m.

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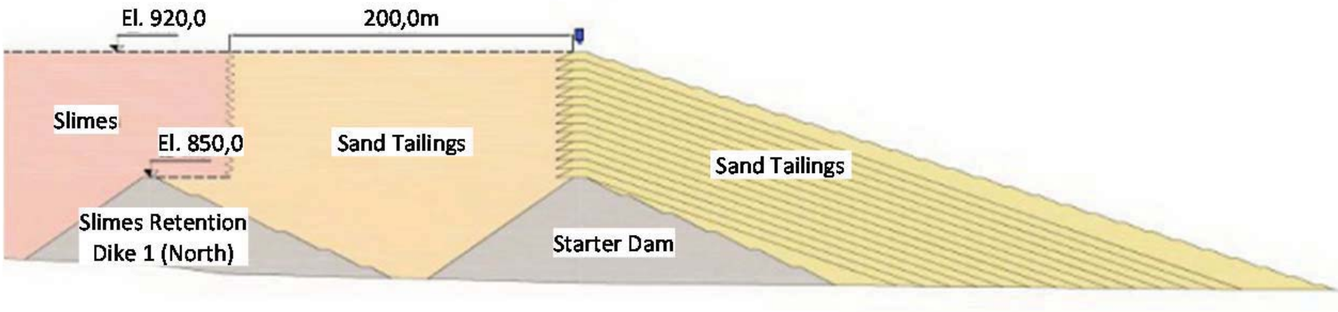

Figure 2-3 Centerline raising of Dike 1 considered but not implemented

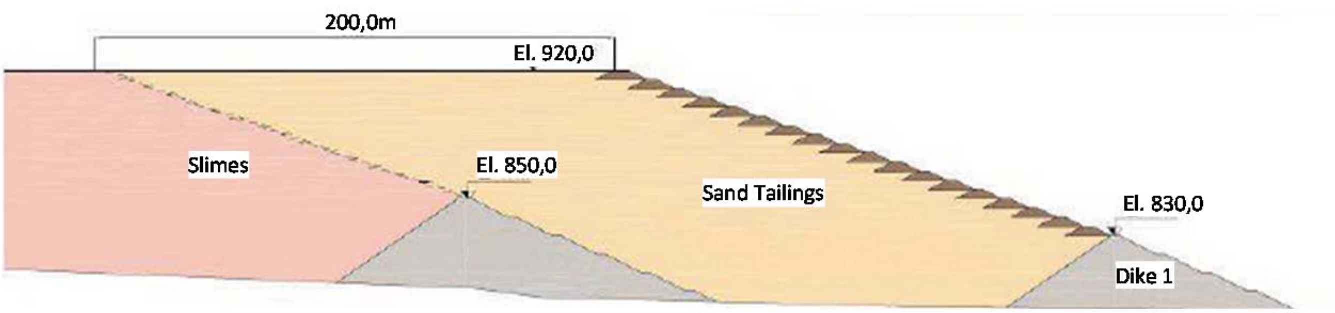

Figure 2-4 Upstream raising of Dike 1 by the "drained stack" concept

Thus, the Germano Buttress structure became the prototype for Fundão. Like its predecessor, the Dike 1 Starter Dam for Fundão would be underlain by a high-capacity base drain of gravel and rock. This would connect to another drain on the Starter Dam's upstream face, along with other complimentary drainage features—all to minimize saturation in the sand deposit behind it.

A remaining design consideration was how to evacuate surface water inflows from ordinary precipitation, floods, and discharged tailings slurry. This would be accomplished by two concrete galleries, 2 m diameter decant conduits of reinforced concrete extending beneath the tailings deposit and Dike 1 itself. The Main Gallery would be beneath the right abutment and the Secondary Gallery beneath the left as indicated on Figure 2-5.

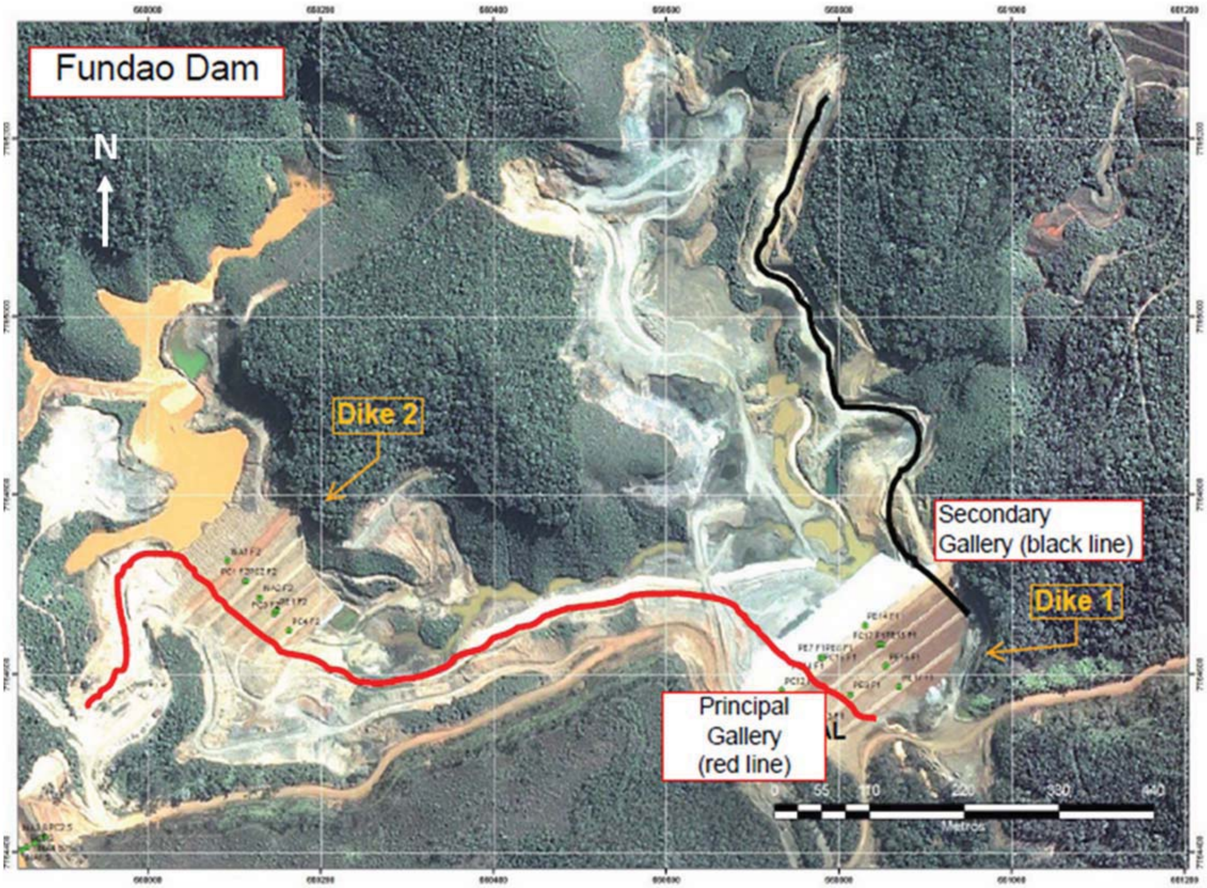

Figure 2-5 Main (Principal) and Secondary Galleries

In the Panel's estimation, this design concept for Fundão offered several advantages. With the dam located in a narrow valley constriction, the site was efficient, requiring a modest amount of dam fill for the storage volume achieved. Once above the valley floor, the reservoir expanded to provide large capacity relative to the area it occupied. But the concept also had certain vulnerabilities. The design was not adaptable to variation in the proportion of sands and slimes received. And most importantly, it depended on achieving adequate drainage of the sands.

2.2 The Piping Incident (2009–2010)

Construction of the Dike 1 Starter Dam, with its requisite drains and galleries, was completed in October, 2008. Shortly after full-scale discharge of sand tailings began on April 13, 2009, large seepage flows carrying fines appeared on the downstream slope above the main underdrain as shown on Figure 2-6, conditions symptomatic of the process of *piping* or *internal erosion*.

Figure 2-6 Internal erosion effects on downstream slope of Dike 1

An Emergency Action Plan in place for the dam at that time was immediately implemented. The reservoir was lowered, a berm was constructed over the affected portion of the dam slope, and provisions were made for holding the reservoir's remaining contents in the downstream Santarem Dam should failure occur. Engineering investigations later revealed serious construction flaws in the base drain and its filters, including a portion of the drain's outlet that had never been completed. This allowed water pressure within it to build until causing the slope to erode and slump.

As these investigations continued, the impending rainy season made it too late to fully restore the drainage features to their original condition, making it impossible to repair the damage. Instead, all of the drains were sealed. With this, the most important element of the original design concept became inoperative.

Additionally, the balance between sands and slimes crucial to the dam raising plan was changed. Filling of Dike 2 had begun earlier than anticipated, making its slimes level higher, not lower, than the projected sands in Dike 1. At the same time, reduction in pellet production reduced the amount of sand available while delivery of slimes continued. This required construction of yet a third dike between Dikes 1 and 2, designated Dike 1A, to provide additional slimes capacity. It was November 2010 before all of the measures made necessary by the piping incident were finally completed.

It remained to devise a new design concept to replace the old one.

2.3 The Recovery (2011–2012)

A revised design for raising Dike 1 to El. 920 m was first described in the 2011 Operations Manual, then updated in the 2012 version when the dam had reached crest El. 845 m. The central feature was the addition of a blanket drain on the surface of the tailings to replace the inoperative base drain below them. As shown on Figure 2-7, the new blanket drain was at El. 826 m just below the Starter Dam crest. Figure 2-8 depicts how the blanket drain would become embedded within the tailings during raising of the dam, intercepting seepage that could otherwise emerge on the slope and reduce its stability. In order to augment capacity for discharging the collected seepage flows, the blanket drain also contained slotted pipes called "Kananets®".

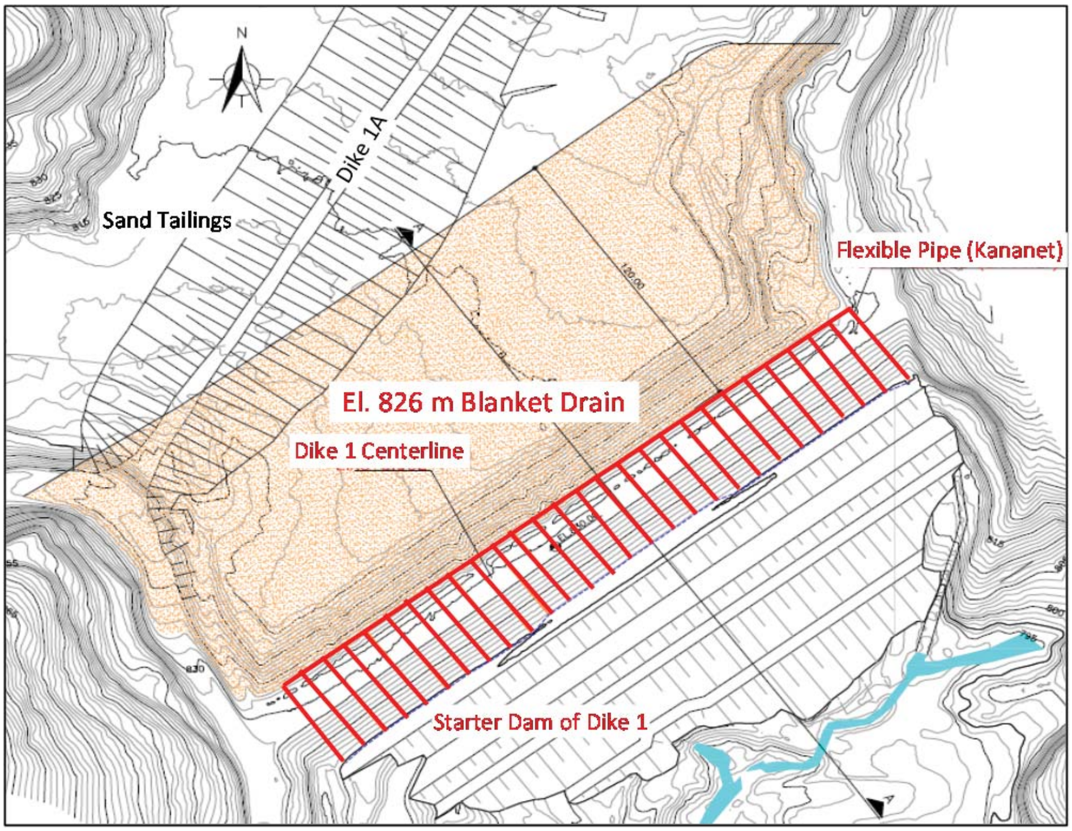

Figure 2-7 Blanket drain (plan view) on tailings surface at El. 826 m

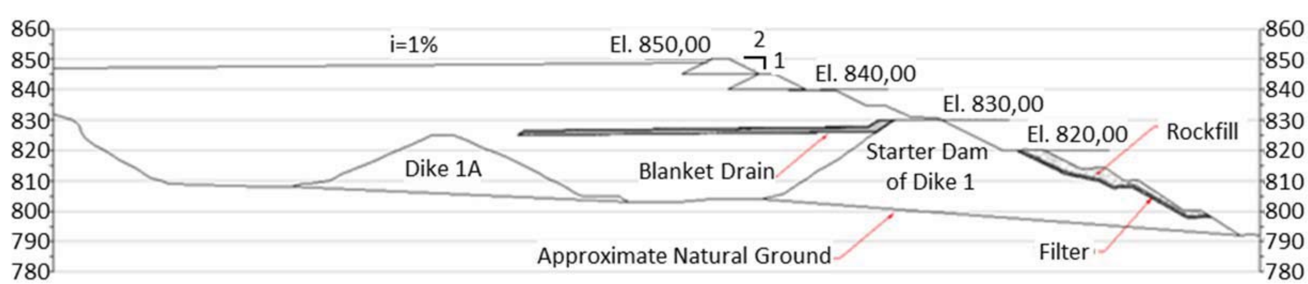

Figure 2-8 El. 826 m blanket drain (section) showing extent behind Dike 1

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Comparing Figure 2-8 to Figure 2-1, it can be seen that the new blanket drain represented an attempt to replicate the drained-stack concept by providing drainage for the overlying tailings. But the sands below this drain would remain saturated, as would much of the tailings behind it. Once the base drain became inoperative, the control of saturation embodied in the original design concept could not be restored.

A requirement common to both the original and revised designs was that the sands be free-draining. To ensure that low-permeability slimes would not be deposited where they could impede this drainage, water containing the slimes had to be restricted from the area of sand deposition. To do so, a 200 m minimum beach width had been specified in the original 2007 Operations Manual, a provision retained in the 2011 and 2012 versions.

But as operation proceeded, this beach-width criterion was not consistently achieved. As explained in greater detail in Section 5.1.3, a new Overflow Channel was conveying water and slimes from Dike 2 to the rear of the Dike 1 reservoir, making beach management more difficult. No longer were the sands and slimes physically separated; the interface between them could only be controlled by adjusting the amount of sand spigotted from the dam crest in relation to the amount of slimes-laden water being introduced. As plotted on Figure 2-9 and documented in Appendix B, during much of 2011 and 2012, beach widths violated the 200 m minimum more often than not, at times encroaching to as little as 60 m from the crest.

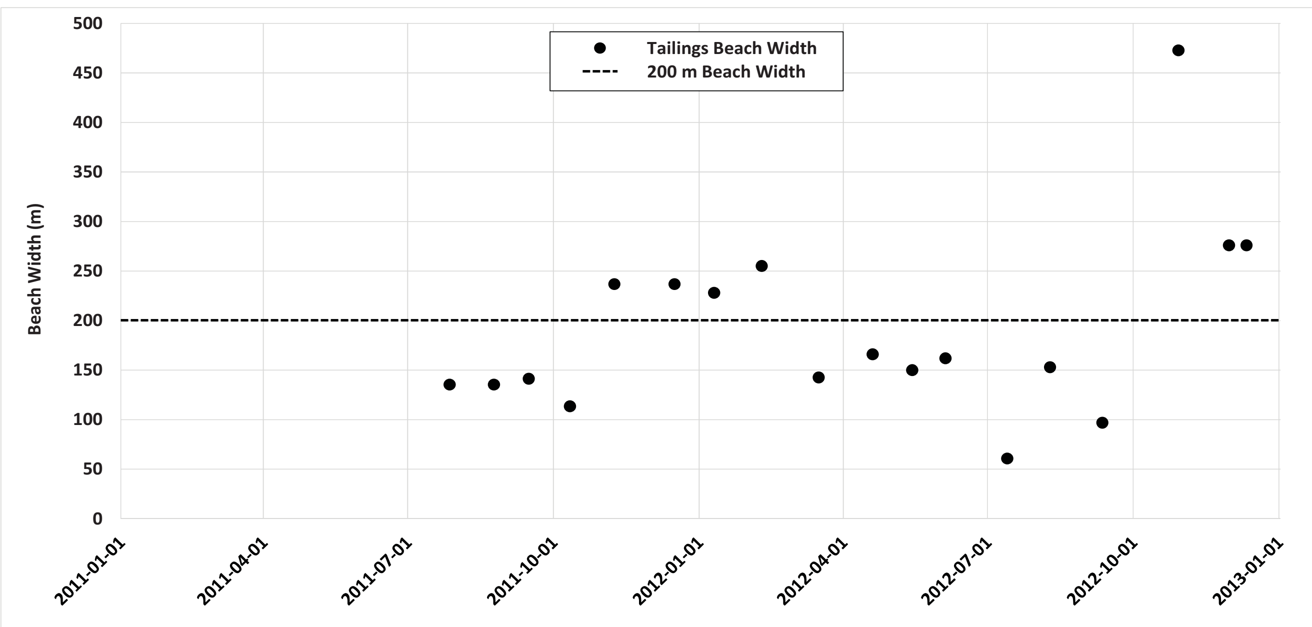

Figure 2-9 Monthly beach width measurements by Samarco, 2011-2012

2.4 The Setback (2012–2014)

Even as recovery from the 2009 Starter Dam piping incident remained underway, new conditions were developing that would directly affect the left abutment. The galleries shown on Figure 2-5 that evacuated water from the Fundão impoundment were found to be structurally deficient. This first

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became evident for the Main Gallery at the right abutment when in July, 2010 a vortex appeared in the reservoir above it, showing that tailings and water were entering. Inspections revealed cracking and structural damage from foundation settlement and construction defects. Were either of the galleries to collapse, uncontrolled release of tailings from the reservoir or failure of the dam would be possible. So in January, 2011 a program of jet grouting was initiated to repair the Main Gallery and return it to service.

Similar conditions were discovered for the Secondary Gallery, and jet grouting was undertaken there as well. But by July, 2012, it was apparent that jet grouting had not cured these problems. After a sinkhole appeared in the tailings overlying the Secondary Gallery in November, 2012, repair efforts were abandoned. Instead, plans were made to plug both galleries by filling them with concrete from their outlets to a point beneath the projected crest of the 920 raise in order to prevent their collapse. Moreover, it was discovered from structural analyses that the Secondary Gallery could not support tailings higher than El. 845 m, some 10 m lower than the tailings already were at that time.

Because the height of tailings at the left abutment already exceeded the load capacity of the Secondary Gallery, the dam could not be raised any further over this area until the plugging operation was completed. As a temporary solution, it was decided to realign the dam at the left abutment by moving it back behind the portion of the gallery to be filled with concrete so that embankment raising could continue. This realignment shown on Figure 2-10 became the “setback”.

The setback would move the crest closer to the reservoir water and the slimes it contained, but it was anticipated that the dam would be quickly returned to its original alignment as soon as the plugging operations were done. At the same time, as will be explained more fully in Section 5, moving the crest back from its original alignment would also place it closer to, if not over, areas where beach encroachment and slimes deposition had already occurred.

Filling of the Secondary Gallery was completed on August 22, 2013. Meanwhile, dam raising had continued, with seeps that began to appear at the left abutment as early as June 26, 2012, at El. 845 m. In February, 2013, three-dimensional seepage modeling of the 920 raise showed that additional drains would be needed at the abutments if seepage breakout were to be prevented. This analysis was borne out when seepage, saturation, and cracking began appearing at several locations at the left abutment during 2013. The first such incident occurred in March at El. 855 m, followed by another seep in June at El. 855 m. Both were treated by constructing a drain. A third seep on November 15 appeared at El. 860 m and was accompanied by slumping of the slope shown on Figure 2-11. Another drain was provided to address this condition. On December 26, seepage occurred at El. 860 m and there was cracking on the left abutment crest at El. 875 m.

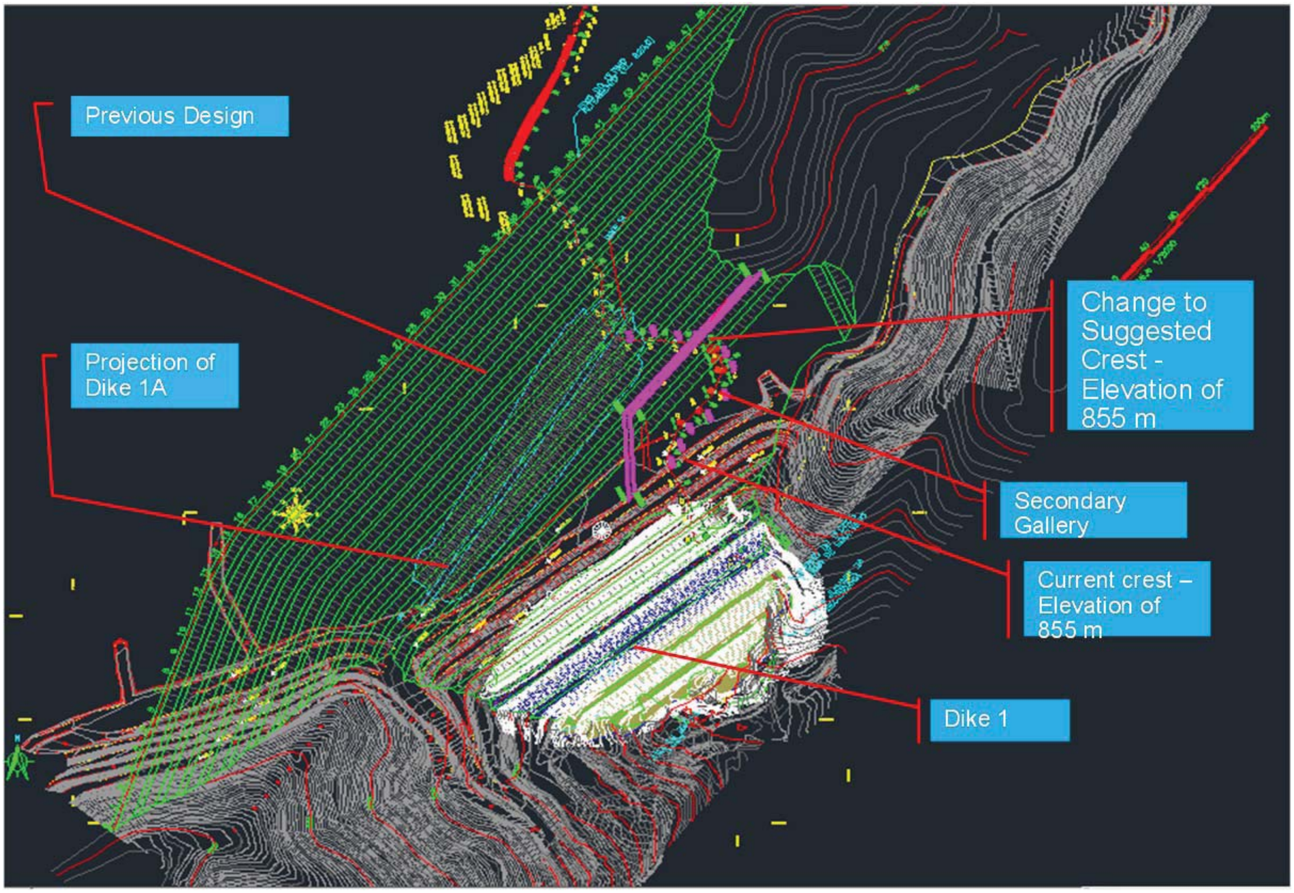

Figure 2-10 Left abutment setback proposed in June, 2012

Figure 2-11 November, 2013 seepage, cracking, and slumping at left abutment El. 860 m

Following these 2013 episodes of seepage and cracking, it had become apparent by January, 2014 that the El. 826 m blanket drain was no longer sufficient and that additional drains would be needed at the left abutment. This coincided with plans for an entirely new project for future raising of the dam by an additional 20 m from its then-planned maximum elevation of 920 m. Not only would this new El. 940 m raise add needed drainage features to the left abutment; it would eventually integrate them with an independent drainage system entering from the adjacent Grota da Vale and Fabrica Nova waste pile. As shown on Figure 2-12, the result would be what the Panel considers to be a complex and elaborate drainage system.

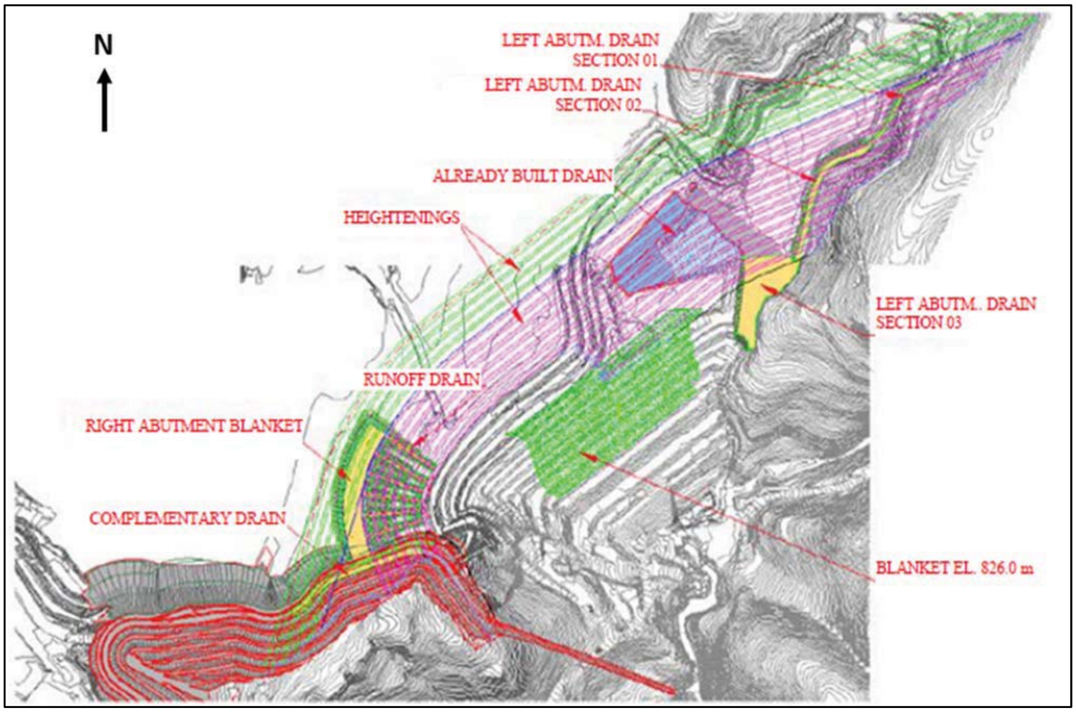

Figure 2-12 Proposed drainage scheme for 940 raise

The more immediate effect was that construction of additional drains in the left abutment area would require the setback to be maintained until they were completed. This entailed further delay in restoring the original alignment. As a result, the setback had risen at an average rate of 18 m/yr during 2013 and 3.0 m in September, a monthly record. In the 18 months since the setback decision had been made, the dam had grown by more than 20 m, and by January 2014 the Fundão Dam looked like Figure 2-13.

Figure 2-13 Fundão Dam in January, 2014 showing left abutment setback and adjacent Grota da Vale

2.5 The Slope Incident (August 2014)

Just after sunrise on August 27, 2014 a series of cracks much more extensive than anything that had occurred the previous year were discovered that extended behind the dam crest, emerged at the toe, and encompassed most of the slope as shown on Figure 2-14. Accompanying the cracking was shallow saturation at the toe, as shown on Figure 2-15.

Figure 2-14 August 27, 2014 cracking at left abutment setback

Figure 2-15 Cracks on dam crest and saturation at toe of slope, August 27, 2014

Stabilizing the slope became paramount, and construction was quickly mobilized to do so. Within two weeks, the reinforcement or "equilibrium" berm shown on Figure 2-16 was completed.

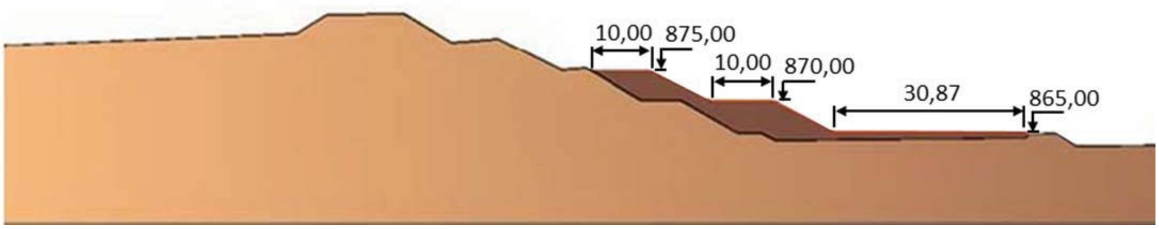

Figure 2-16 Reinforcement berm for left abutment setback, August, 2014

Construction of the left abutment drain was still ongoing, and it was not until a year later, August, 2015, that the drain was completed and fill placement over the area it covered could resume.

October, 2015 was a period of intense activity on the left abutment. The dam crest was being raised to El. 900 m, preparations were being made for cyclone sand placement on the El. 875 m bench, while at the same time the reinforcing berm was being extended by raising the El. 875 m and El. 895 m benches. The net result was that the monthly increase in crest height of 2.9 m—an annualized rate of rise of 35 m/yr—rivaled the record of 3.0 m set in 2013.

2.6 The Earthquakes (November 5, 2015)

Explosions are detonated every day at mines throughout the region, so the small-magnitude seismic events they produce are not unusual. At the same time, while larger earthquakes are rare in Brazil, small earthquakes in Minas Gerais are relatively common. Either way, the tremor on the afternoon of November 5, 2015 was not unprecedented.

According to felt reports at the plant about 2 km from Fundão, shaking was strong enough to cause a computer to fall from a tabletop, but not so strong as to produce structural damage other than minor cracking.

Detailed analysis of instrumentally-recorded events and mine records show that on November 5, 2015, two blasts occurred at a nearby mine within seconds of each other just after 1PM. This was almost three hours before the failure. Later at around 2:15PM a series of three small-magnitude earthquakes occurred over a period of four minutes on the afternoon of November 5, 2015. They preceded the failure by some 90 minutes with the time sequence shown in Table 2-1 below and occurred almost directly beneath the Fundão deposit.

Table 2-1 Pre-failure earthquakes and mine blasts on November 5, 2015 (E.g., Atkinson 2016)

Local time	Moment Magnitude M_w	Distance from Fundão	Identification
1:01:49PM	2.1	2.6 km	mine blast
1:06:06PM	2.3	2.6 km	mine blast
2:12:15PM	2.2	< 2 km	earthquake (foreshock)
2:13:51PM	2.6	< 2 km	earthquake (main shock)
2:16:03PM	1.8	< 2 km	earthquake (aftershock)
3:45PM			*Dam failure*

The implications of the earthquakes will be discussed in Section 6.

2.7 The Collapse (November 5, 2015)

On the afternoon of November 5, 2015, most activity was on or near the right abutment where drains were being constructed, placing several workers in a position to see along the length of the dam crest. On the left abutment, fill was being placed on the El. 875 m bench of the setback in preparation for start-up of cyclone sand placement. Figure 2-17 shows the locations of eyewitnesses engaged in these and other activities at the time of failure.

Figure 2-17 Eyewitness locations on the afternoon of November 5, 2015

The first thing noticed by many workers on the dam, including those at locations 4 and 6, was a cloud of dust drifting up from the left side heralding the failure. A worker at 4 watched as waves developed in the central portion of the reservoir, accompanied by cracks forming on the left side and blocks of sand moving up and down on the left abutment setback. Another worker at 5 saw a crack open up along the crest of the left abutment setback then propagate in both directions, beginning closer to the left abutment, reaching it, then progressing to the right. And at location 9 at the toe of the dam, witnesses experienced an avalanche of mud-like tailings cascading down from the left abutment, but no movement of the starter dike itself.

These observations establish that failure originated at the left abutment setback and that the Starter Dam did not participate in the failure mechanism. However, these workers on the dam crest had been unable to see precisely how and where the failure began, and by the time they made these observations the first stages of failure were already well advanced.

Other observers at the left abutment had a closer view of the developing failure sequence. Workers at locations 1 and 2 were the first to see the failure initiate near the left abutment drain where they were standing, placing the time at 3:45PM. Here, a sudden jet of dirty water "exploded" out of the drain. The first movement and cracking was also reported at the exposed drain and along the adjacent edge of the plateau, placing the exit of the rupture surface at or around El. 857 m. A worker at 1, who was standing on the plateau, felt it begin to move beneath him and crack around him, detaching from the setback slope and moving downstream.

Next to move was the lower slope of the setback. Eyewitnesses at 2, 3, and 5 describe slope movement having propagated "from the bottom up" on the lower benches, not from the crest down, placing the seat of movement at lower elevations. A worker at 3 observed a small bulldozer on the El. 875 m bench moving or being pushed outward, placing the head of the incipient failure at or above this elevation. At first, the lower slope progressed slowly forward "like a snake." Remaining intact and moving as a unit, it then bulged, becoming grossly distorted as movement accelerated, coming down "like a wave," or as if it were "melting". Subsequently, a witness at 3 characterized the violent turbulence of the fluidized mass as "going in somersaults" downstream.

Taken together, these eyewitness observations can be synthesized into the sequence of events at failure initiation portrayed on Figure 2-18.

Figure 2-18 Failure initiation sequence

By the time the events on Figure 2-18 had occurred, the growing failure would have become apparent to the observers on the crest at locations 4, 5, and 6 as it progressed back behind the crest and into the reservoir. Only then did the central and right sides of the dam begin to disintegrate.

A conveyor crossing the Fundão stream channel about 1300 m downstream from the offset crest stopped functioning at 3:49PM, four minutes after failure is reported to have begun at 3:45PM. From this, it is ascertained that the flowslide was moving at about 11 m/s by the time it reached the conveyor. It is calculated that 32 million m^3 of tailings was lost, representing 61% of the impoundment contents—an unusually high proportion in relation to tailings dam failure statistics. In a matter of hours, the Fundão Dam was gone, and what once had been Figure 2-19(a) became Figure 2-19(b).

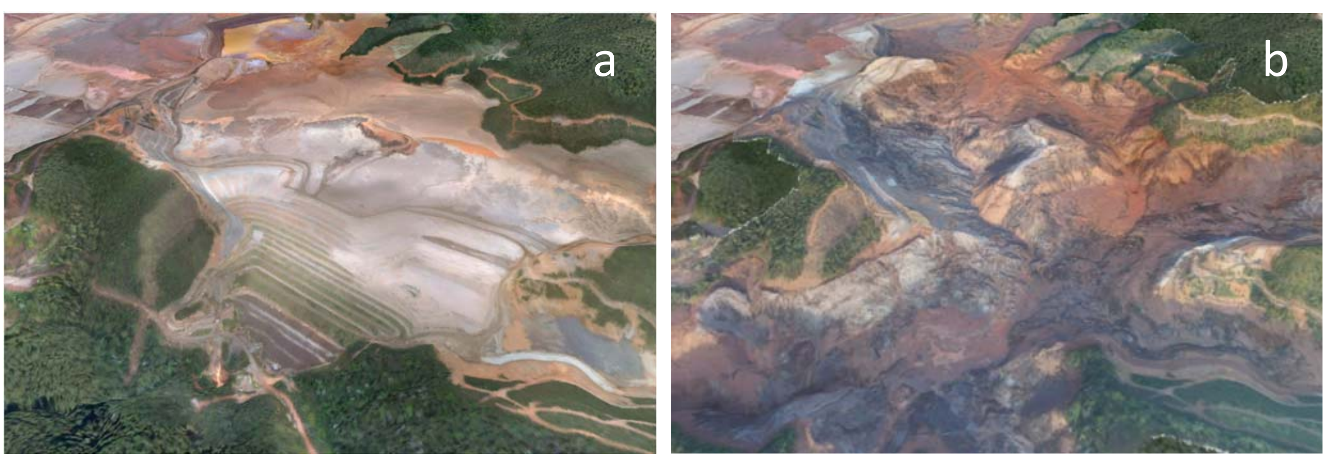

Figure 2-19 Fundão damsite and reservoir (a) before, (b) after failure

3 WHAT DID THE PANEL DO?

3.1 Diagnostic Strategy

The methodologies adopted and activities conducted during the Panel's Investigation were important to its outcome. The instruction to the Panel in its Terms of Reference was to determine the immediate cause or causes of the breach of the Fundão Tailings Dam on November 5, 2015. This is fundamentally a diagnostic exercise as reflected in the overall framework adopted by the Panel. The Panel's diagnostic strategy consisted of three parts:

1. *Hypothesis formulation.* Candidate failure modes were identified based on known causes of tailings dam failures as they pertain to specific conditions of the Fundão Dam.
2. *Hypothesis screening.* The candidate failure modes were screened using a process of elimination to arrive at one or more that were most consistent with the evidence.
3. *Hypothesis testing.* The surviving failure modes were tested for their ability to predict conditions that occurred at times and locations other than those on November 5, 2015 at the left abutment.

With regard to the third item of hypothesis testing, the Panel developed criteria that its causation conclusion should meet. These took the form of three questions:

1. **Why did a flowslide occur?** That the failure occurred by flowsliding is self-evident but not by itself informative. Any explanation of the failure must go beyond this to determining the events, conditions, and mechanisms that allowed flowsliding to occur.
2. **Why did the flowslide occur where it did?** In principle, there were many places on the Fundão Dam where failure might have occurred. The failure hypothesis must explain what was different about the left abutment that caused the failure to occur there and not at some other location.
3. **Why did the flowslide occur when it did?** Failure occurred when the embankment at the left abutment reached El. 898 m following a series of small earthquakes. The failure explanation must establish why failure did not occur at some previous time at lower elevation and the relationship, if any, between the failure and the earthquakes. The hypothesis must also explain why flowsliding did not occur in association with the cracking incident of August, 2014.

As the tests of the Panel's hypothesis, these three questions constitute the central topics of the remainder of this report and the framework around which it is built.

3.2 Investigation Methodology

The Panel also followed a systematic structure in its investigative efforts. The elements of the Investigation and the tasks that comprised them are described below, with reference to the related appendices.

- **Reconstruction of the dam and its properties.** Most if not all of the key physical evidence was destroyed when the dam washed downstream with the failure. A virtual representation of the dam and its internal composition therefore had to be reconstructed through a lengthy and painstaking process consisting of:
  - *Compilation of digital topographic data and imagery in GIS format*. This allowed the progression of dam raising and tailings deposition to be tracked over time. The methodology adopted is described in Appendix A.
  - *Reconstruction of design, construction, and operational history*. This was done through assembly and interpretation of documents, photographs, and aerial imagery, as described in Appendix B.
  - *Subsurface exploration and laboratory testing*. This incorporated both pre-failure data and independent Panel field investigations at surrogate locations. It allowed estimation of pre-failure engineering properties of dam materials, as contained in Appendices C and D.
- **Compilation of instrumentation data.** The dam contained a large number of instruments that measured internal water pressures, flows, and movements. Together, this data provides a record of the dam's engineering behavior, allowing trends and changes to be tracked throughout its life. Instrumentation data is contained in Appendix E.
- **Synthesis of eyewitness interviews.** The Fundão failure was witnessed by a large number of people at different locations on and near the dam. Their accounts are of content and value unusual for dam failure investigations of this kind and provide insight into the processes that were taking place during the hours and minutes leading up to the failure.
- **Analytical studies.** With the reconstructed dam, instrumentation data, and eyewitness accounts in place, the Panel was able to simulate the operation of potential failure mechanisms and related processes through a variety of numerical modeling techniques:
  - *Consolidation modeling.* This was to evaluate the effects of loading rate on pore pressure development and is described in Appendix F.
  - *Seepage modeling.* This provided information on internal flow and pressure conditions at times and locations where measured instrumentation data was not available. Seepage modeling is described in Appendix G.
  - *Stability analysis.* This provided the calculated degree of embankment stability under various conditions at various times and is found in Appendix H.
  - *Deformation analysis.* Closely linked to stability, deformation modeling provides further insight into failure-related processes and mechanisms as contained in Appendix I. The deformation analysis is central to identifying the causative liquefaction trigger mechanism, and the concluding section of this report is devoted to the development of this topic.
  - *Dynamic response analysis.* This numerically simulates earthquake shaking and is found in Appendix J.
- **Seismological studies.** Conducted independently from the Panel's Investigation, seismological studies provided key input that is contained in a separate report.

3.3 Potential Failure Modes and Triggers

The Panel considers that the evolutionary character of its design and operation makes the Fundão Dam extraordinarily complex. This is reflected in the large number of potential failure modes, which in turn makes a structured process for their evaluation mandatory. Appendix K details how the approach to hypothesis formulation and screening of Section 3.1 was implemented. First, the following potential failure modes were considered:

1. overtopping;
2. internal erosion;
3. Starter Dam foundation or embankment sliding; and
4. liquefaction.

All but liquefaction were ruled out as being inconsistent with physical evidence and/or eyewitness accounts.

Amplifying on liquefaction as the cause of flowsliding, the second stage was to evaluate liquefaction triggering mechanisms, again adopting the same hypothesis formulation and screening process. Here, the Panel used fault trees to structure the assessment in real time, modifying them as the Investigation unfolded. Applied as a heuristic aid rather than a reliability application, formal fault tree symbology was not necessary or adopted. The Panel's final fault tree for liquefaction triggering is shown on Figure 3-1.

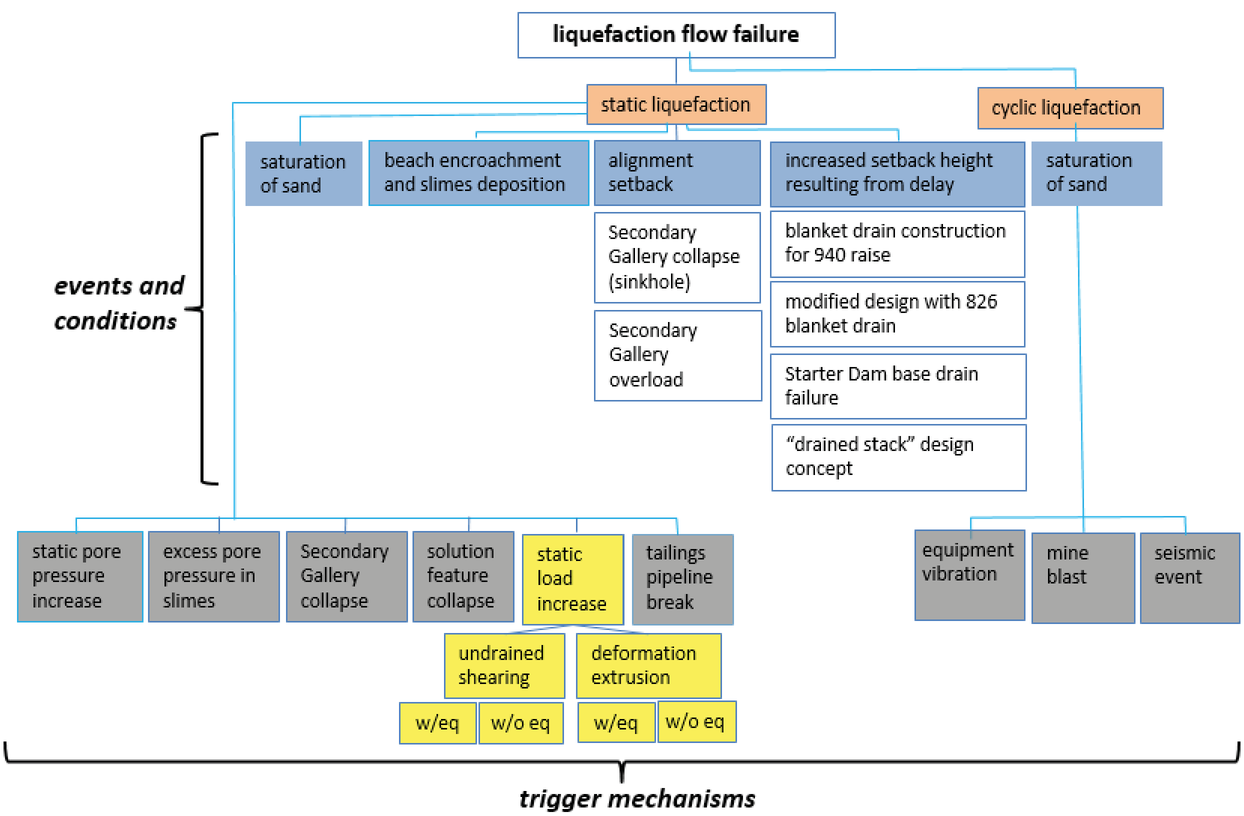

Figure 3-1 Fault tree for liquefaction triggering

The top event on Figure 3-1 is liquefaction flow failure. The next tier of events represents the two fundamental liquefaction processes: static and cyclic, either of which might have been operative. In this representation, cyclic-induced liquefaction flow failure is distinct from cyclic pore pressure contribution to static liquefaction.

The bottom tier of candidate initiating events represent liquefaction trigger mechanisms, and those shaded in grey were ruled out for reasons developed in Appendix K. These are:

- cyclic liquefaction:
  - equipment vibration;
  - mine blasting; and
  - seismic-induced.
- static liquefaction:
  - static pore pressure increase;
  - excess pore pressure in slimes;
  - Secondary Gallery collapse;
  - solution feature collapse; and
  - tailings pipeline break.

The surviving liquefaction trigger mechanism is static load increase, shown in yellow on Figure 3-1 with its two subsidiary processes: undrained shearing and deformation-related extrusion. Both of these might be operative either with or without cyclic pore pressure contribution from the November 5, 2015 earthquake series.

Also important on Figure 3-1 are the antecedent events and conditions shaded in blue that allowed or promoted static liquefaction at the left abutment. These are: (1) saturation of the sand; (2) water encroachment that allowed slimes deposition on the tailings beach; (3) the alignment setback; and (4) the increased height of the setback resulting from continued raising of the dam.

These four factors are central elements of the following sections of this report.

4 WHY DID A FLOWSLIDE OCCUR?

Determining why a flowslide occurred necessarily involves considering the conditions required for liquefaction, the first of which is saturation. In this regard, Section 2 presented the original “drained stack” design concept, that in the view of the Panel was not in principle amenable to liquefaction, and explained design changes that brought about an extent of saturation not anticipated in that concept. The new design allowed saturated conditions within the tailings, as evidenced by the extensive system of piezometers intended to measure it and limiting criteria established to evaluate it.

Another requirement for liquefaction concerns the properties of the materials involved, in this case sand tailings. This section shows how their void ratio—a measure of their propensity to expand or contract during shearing—influenced their susceptibility to liquefaction during the kind of rapid failure that occurred. Along with this is a related requirement for liquefaction: a reduction in strength during rapid shearing that produces flow behavior.

The Panel found no credible pre-failure assessment of liquefaction for the Fundão Dam in any of the documents it reviewed. Nor did it find any boring or cone penetration test (CPT) penetrating the full depth of the tailings that would have made such an assessment possible. For these reasons, the Panel has relied on its own analyses to determine why a flowslide occurred, the first test it has imposed on its explanation of the failure.

4.1 Strength Behavior

When load is applied to soil particles as a shear stress, *shearing* is said to occur. If these particles are in a tightly-packed arrangement—such as dense sands or stiff clays—the soil particles must first move apart to order to move past each other during shearing. This produces an increase in volume of the soil mass, and such soils are said to be *dilatant*. Generally speaking, dilatant soils are strong, which is why mechanical compaction is commonly used to achieve this condition.

By contrast, when shearing a loose particle arrangement—for example, loose sands or soft clays—the opposite occurs. The particles move together and the soil mass compresses. Soils displaying this tendency for volume decrease are called *contractive*. Hydraulically-placed and uncompacted materials such as tailings are often contractive.

When the soil mass is saturated, the spaces between the particles, or *voids*, are filled with water. If the soil is contractive and shearing occurs, the water may inhibit the particles from moving together so that the water itself carries part of the load. This produces pressure in the water, or *pore pressure.* But since water has no strength, the strength of the saturated soil mass can be reduced. Whether or not this occurs depends on whether or not the water escapes from the voids. And this, in turn, depends on yet another necessary condition for flowsliding—the rate of shearing.

Shearing a contractive, saturated soil slowly enough for pore pressure to dissipate as fast as it is generated produces a *drained* condition. Pore pressure does not develop and the soil retains its strength. On the other hand, if shearing occurs too rapidly for pore pressure to dissipate, *undrained* shearing is said to occur. In the case of Fundão, the failure developed within minutes and clearly occurred under undrained conditions. But in addition, the undrained strength of contractive sands decreases markedly under the large strains imposed during flowsliding. It is this characteristic that gives flowslides their speed and mobility.

4.2 Tailings Volume Change, Undrained Strength, and Liquefaction

Different loading conditions can induce static liquefaction. Figure 4-1 provides stress paths for test data on Fundão sand tailings, where p´ is mean effective confining stress and q is shear stress. Stress paths for two tests are shown, both consolidated to the same stress at the start of shearing.

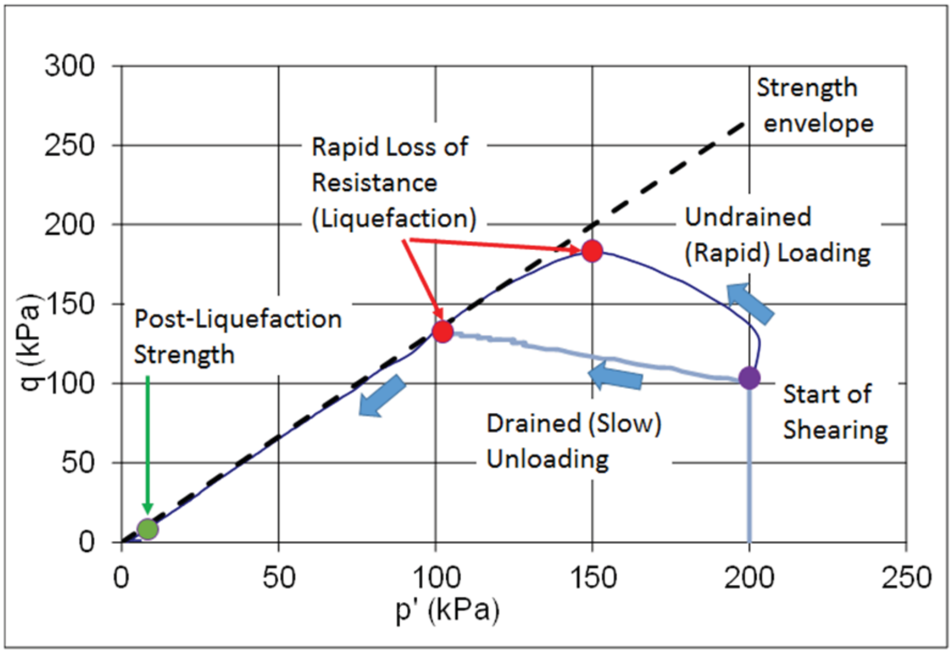

Figure 4-1 Stress paths for undrained loading and drained unloading of sand, Fundão test data

In the first test, conventional undrained loading is applied to simulate rapid shearing. When the stress path reaches the strength envelope it reaches a condition of liquefaction. As shearing resistance reduces due to changes in pore pressure, it progresses downward along the envelope and strength rapidly diminishes until arriving at a very low *post-liquefaction* strength.

The second test represents a different stress path central to understanding the Fundão failure. Instead of being loaded, the sample is laterally unloaded to simulate horizontal spreading. In addition, the unloading process is conducted slowly under drained conditions. As seen on Figure 4-1, the behavior on reaching the strength envelope is the same as before: liquefaction occurs, strength rapidly decreases, and the same post-liquefaction value results. In both tests, the loss of strength accompanying liquefaction is dramatic and nearly instantaneous, so much so that this behavior is sometimes referred to as *collapse*. The parallels between this kind of behavior in the laboratory and that which occurred during the Fundão failure are evident.

Thus, if the necessary conditions are present, liquefaction can occur under either slowly-imposed or rapidly-imposed changes in stress that can be produced by either loading or unloading. The essential point is that it is the rate of shearing, not necessarily the rate of loading, that controls liquefaction of contractive materials, and the change in shear resistance derives from the intrinsic properties of the soil.

The most important such property is the tendency for volume change during shearing. This depends on two factors: first, how loose or dense the soil is, as characterized by its *void ratio*; and second, the level of stress it experiences. Figure 4-2 plots void ratio e versus effective stress p´. At any given effective stress, there exists some void ratio at which there is no tendency for either increase or decrease in volume during shearing. The *critical state line* (CSL) is the locus of these points and delineates the boundary between dilatant (volume increase) and contractive (volume decrease) conditions.

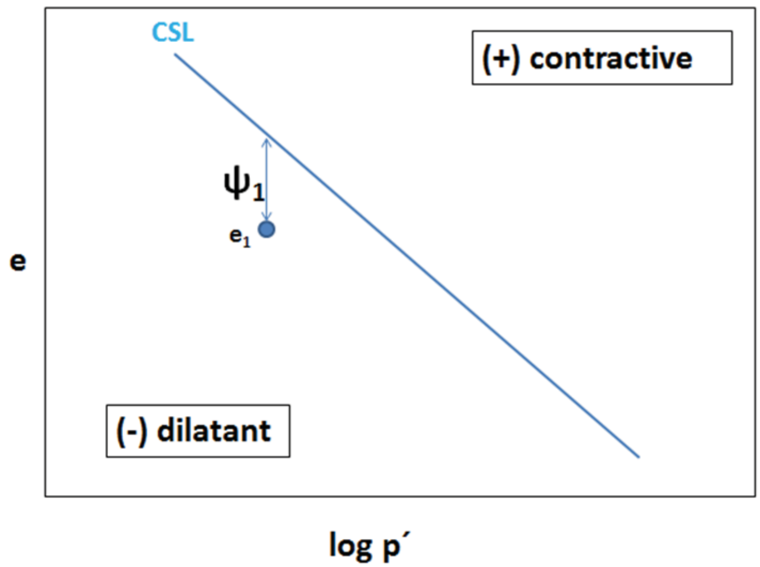

Figure 4-2 Definition of state parameter

The degree of contractiveness or dilatancy can be characterized by the *state parameter* ψ, shown on Figure 4-2 for some existing void ratio e_1. State parameter is defined as the difference in void ratio between e_1 and the void ratio on the CSL at the same mean effective stress. The magnitude of ψ, or the vertical distance of e_1 from the CSL, expresses the degree of contractiveness or dilatancy at that void ratio, with a negative sign convention for dilatancy and positive for contractiveness.

The relationships shown on Figure 4-2 are for constant stress. Figure 4-3 shows what happens when stress increases, for example when loading from embankment raising is imposed.

At the initial void ratio e_1 the tailings are dilatant with negative ψ_1, meaning that they act like a dense sand from an undrained strength standpoint. Under imposed loading and effective stress increase, compression occurs and e_1 reduces to e_2 in Figure 4-3. Now e_2 lies on the other side of the CSL and state parameter ψ_2 has positive sign. Thus, a material that initially had the dilatant behavior of a dense sand takes on the contractive characteristics of a loose sand as a result of the increased stress it now experiences.

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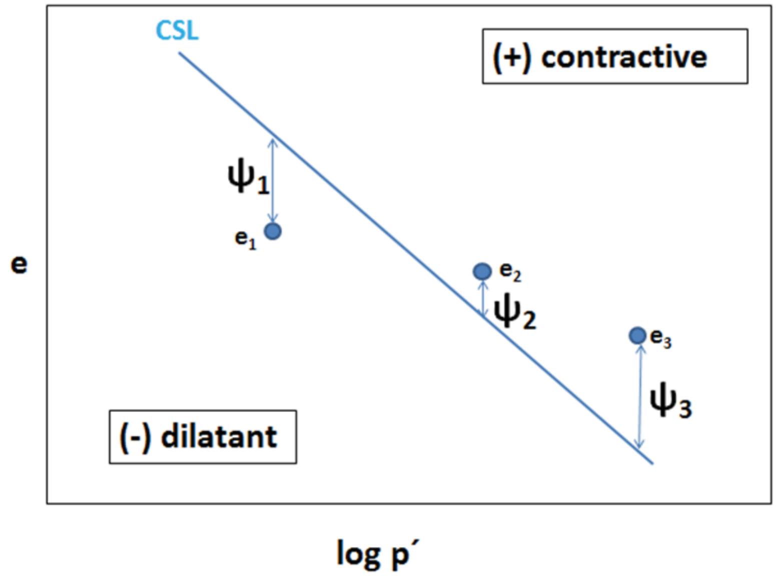

Figure 4-3 Change in state parameter for increasing stress

As loading continues and effective stress increases still more, e_2 reduces to e_3 as the result of further compression, and the magnitude of ψ_2 increases to ψ_3. Thus, not only can continued loading transform a dilatant material into a contractive one, it can also increase its degree of contractiveness.

These principles are applied to the Fundão tailings sand on Figure 4-4 that provides a statistical summary from Appendix C of CPT data on sand tailings obtained by Samarco prior to the failure in early 2015. The five CPTs were located along two transects on the Fundão tailings beach, one behind the left abutment setback and another in the central portion. While in theory contractive materials are those having $\psi > 0$, in practice $\psi > -0.05$ is often adopted as the boundary (Shuttle and Cunning 2007).

Shaded areas on Figure 4-4 indicate relative proportions of contractive material. Upper and lower histograms are for the left abutment and central transects, respectively, at locations given in Appendix C.

On this basis, approximately 70% to 80% of the sand tailings within 75 m of the dam crest are indicated to have been contractive, and 95% or more at greater distance up to 180 m. This demonstrates that the majority of hydraulically-discharged Fundão sand tailings satisfied the contractiveness requirement for liquefaction flowsliding. This is confirmed by CPT-based liquefaction criteria developed by Robertson (2010) on Figure 4-5 that supplements state parameter with liquefaction field case histories. These include flow liquefaction of the Nerlerk offshore berm for the encircled point labeled 19, 20, and 21, a case that figures prominently in the Fundão assessment as subsequently explained in Section 6.

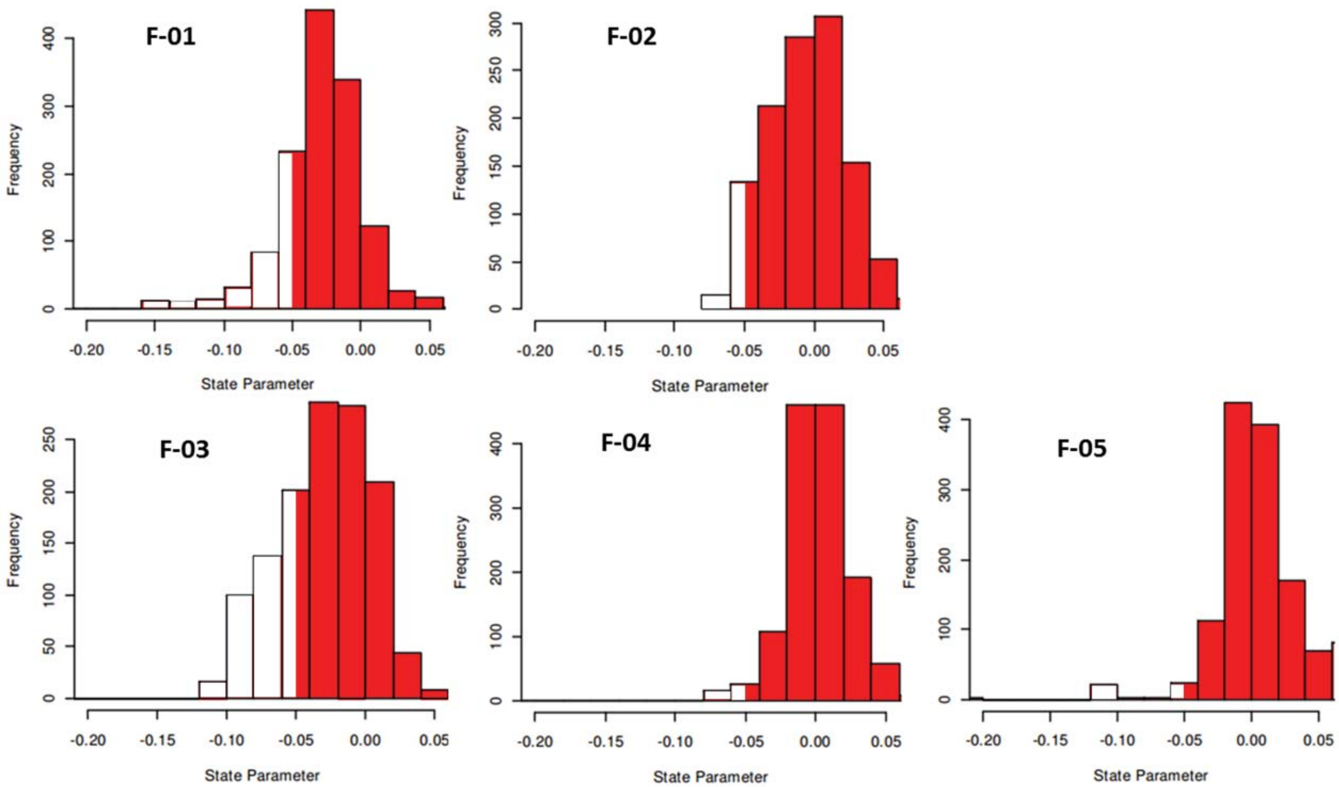

Figure 4-4 Histograms of state parameter for Fundão sand tailings

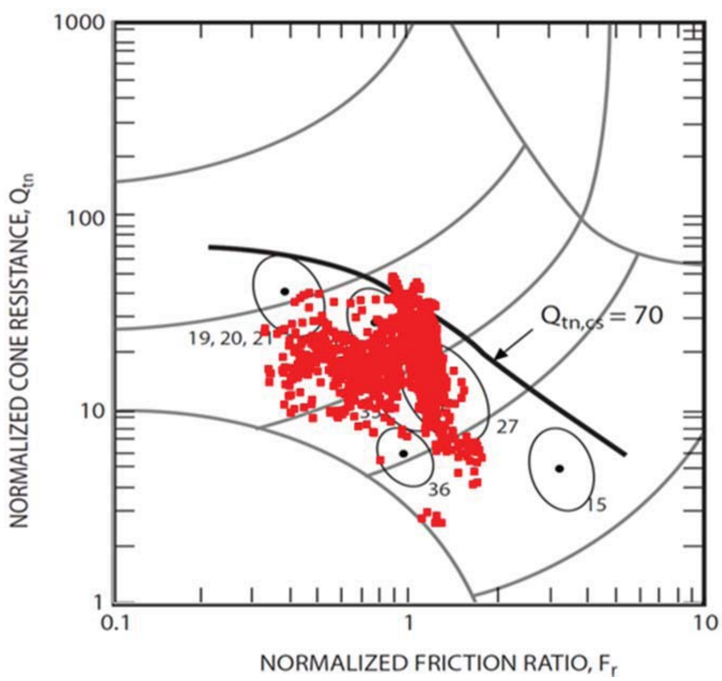

Figure 4-5 Robertson (2010) liquefaction criterion for Fundão CPT F-02 data

The 2015 Fundão CPT data also demonstrate the propensity for reduction in undrained strength of the sand tailings subject to the large deformations that accompany flowsliding. This can be shown by comparing undrained yield (peak) strength to critical (also known as residual or post-liquefaction) undrained strength. Figure 4-6 applies the CPT correlations of Sadrekarimi (2014) for undrained yield strength in simple shear and critical undrained strength.

Figure 4-6 shows that mean undrained strength ratio dropped from 0.21 before the flowslide to 0.07 during it, demonstrating that the Fundão sand tailings were susceptible to significant loss of strength.

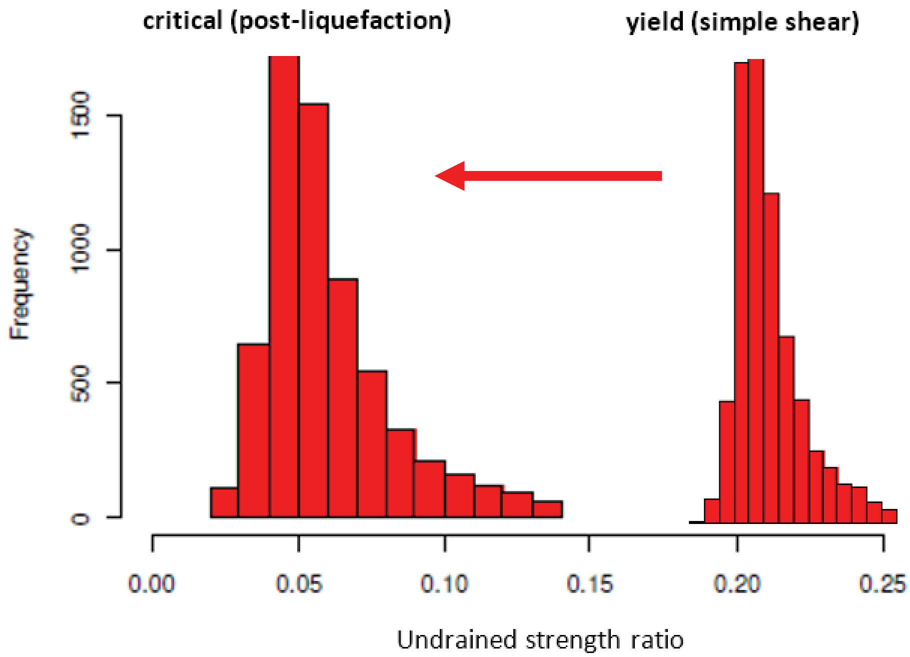

Figure 4-6 Yield (pre-flowslide) and critical (post-flowslide) undrained strengths for aggregated 2015 Fundão CPT data

4.3 Saturation

Saturation is another necessary condition for liquefaction flowsliding. It is useful to chart how saturation conditions of the Fundão Dam changed over time in response to events during the dam's evolution. To begin with, and as explained previously in Section 2.1, the "drained stack" concept of the Germano Buttress provided the model for the original Fundão design. With its high-capacity base drain extending beneath the dam and the sand tailings behind it, the aim was to reduce saturation and the accompanying effects on stability.

The original concept became inoperative after the damage sustained to the Starter Dam in the 2009 piping incident. The revised design that emerged relied instead on a blanket drain at El. 826 m near the top of the sand that had nearly filled the Starter Dam by that time. The El. 826 m blanket drain, which included the Kananet® pipes, was called upon to carry nearly all of the seepage as the dam grew higher and the impoundment larger with time. It became increasingly unable to do so, resulting in expanding volumes of saturated tailings.

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The progression of conditions that promoted saturation is best illustrated by the following series of figures that integrates this information. Figure 4-7 shows the blanket drain in July, 2011, shortly after its completion the previous November. At that time the drain spanned the entire width of the Starter Dam. In this early configuration—which most closely resembled the original drained-stack concept—the impoundment size was limited and the drain was beneath the discharged tailings where it could intercept downward drainage to maximum effect.

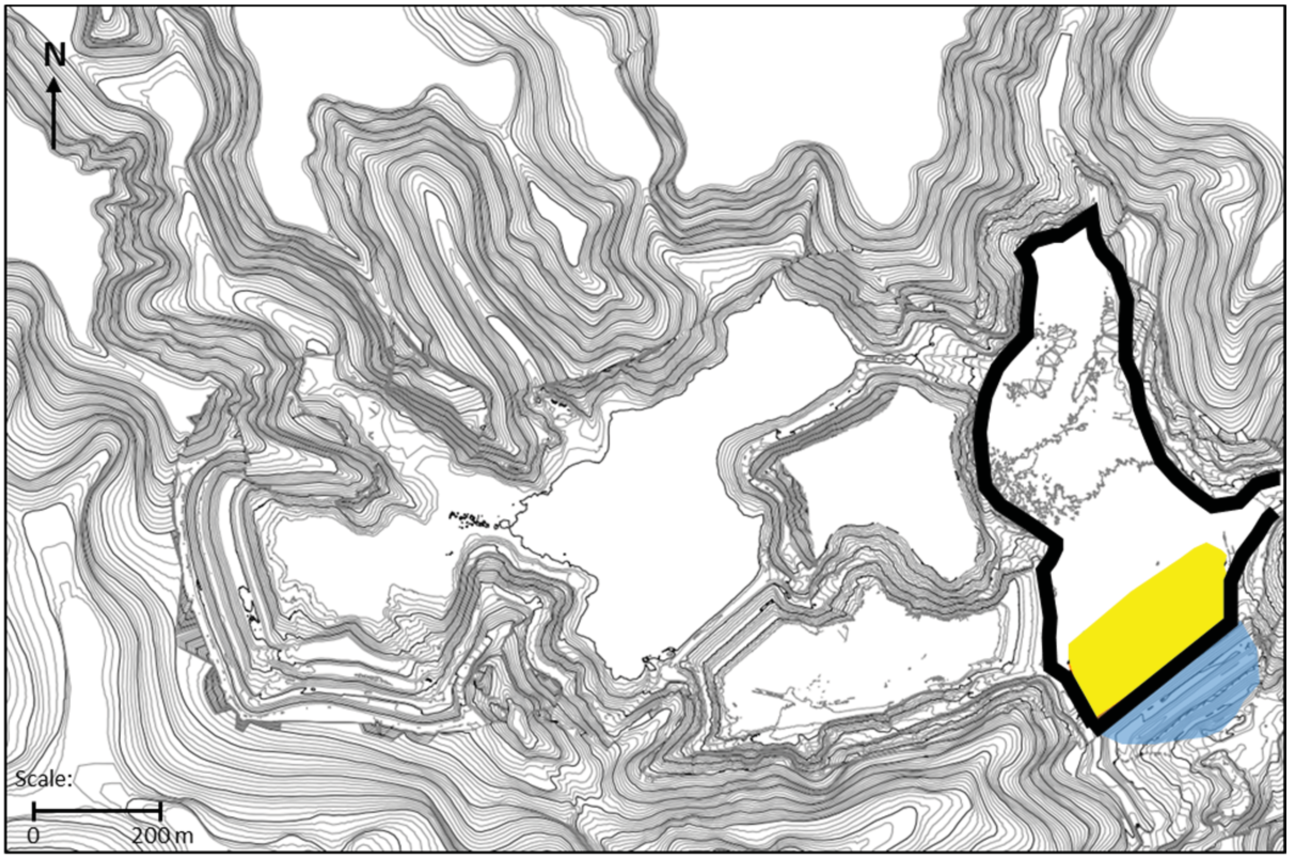

Figure 4-7 July, 2011 configuration showing El. 826 m blanket drain (yellow), Starter Dam embankment (blue) and impoundment outline

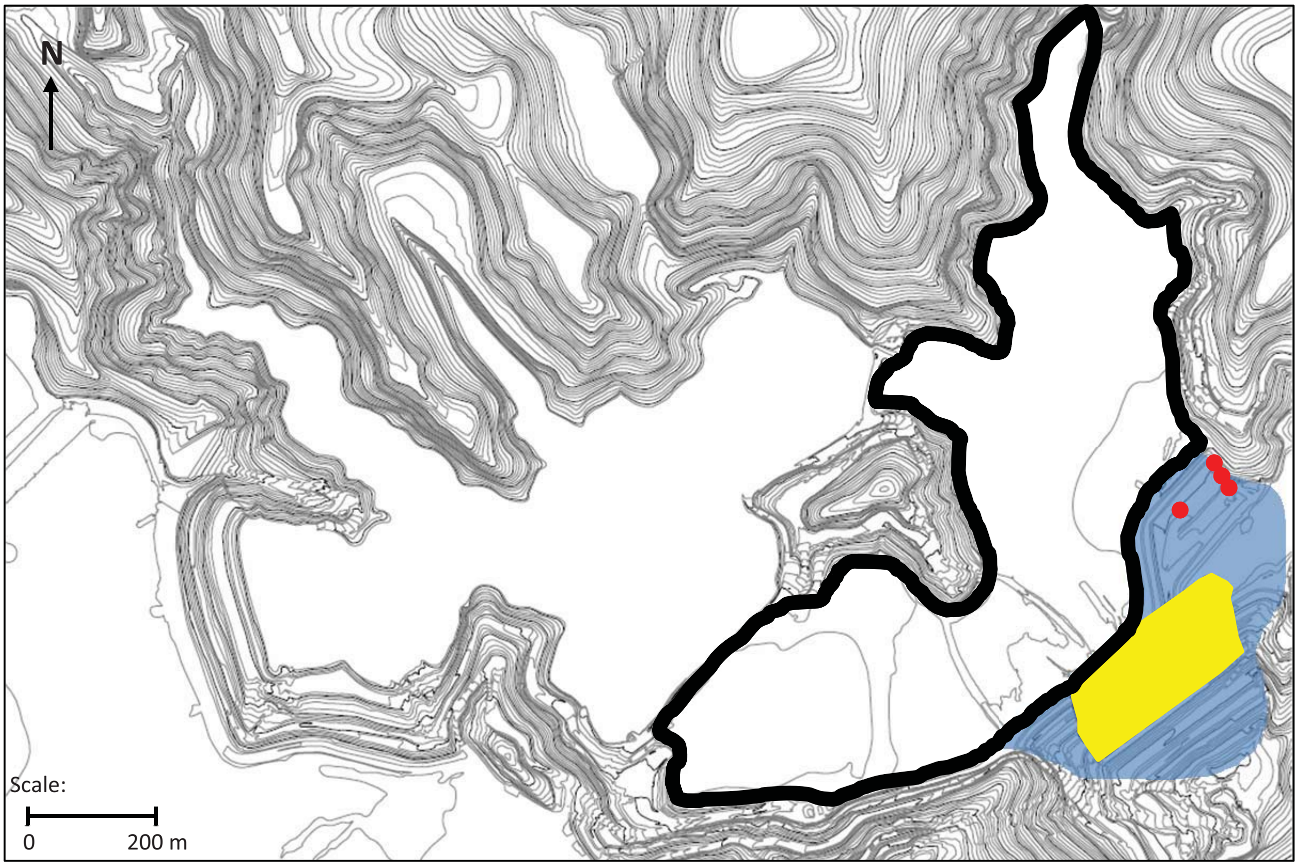

Figure 4-8 August, 2013 configuration showing El. 826 m blanket drain, raised dam, impoundment outline, and left abutment seeps (red dots)

Figure 4-8 shows that by August, 2013 both the embankment and impoundment had widened considerably as the dam grew higher, expanding beyond the limits of the drain on both sides. This had the effect of funneling seepage flow into the much narrower drain, and in the process raising the saturation level in the tailings. At the same time, the impoundment was moving upstream and becoming more distant from the drain as upstream dam raising progressed, also increasing the volume of saturated tailings.

With the left abutment setback by then in place, seeps appeared at El. 855 m in March and June, 2013 as the rising saturation reached the tailings surface, and again in November and December at El. 860 m. This shows that the saturation level at the surface of the left abutment rose some 5 m in elevation during the course of 2013. Localized drains constructed to treat these seeps had mostly near-surface effects, preventing further seepage breakout on the embankment face but not significantly reducing saturation in the tailings mass behind them.

Figure 4-9 shows that by August, 2014 the impoundment had nearly doubled in size, principally on the right side. With this enlargement came a seep on the right abutment at El. 855 m in July, followed by another in January the next year. As upstream raising continued, the impoundment became even further removed from the drain, expanding the volume of saturated tailings behind it still further.

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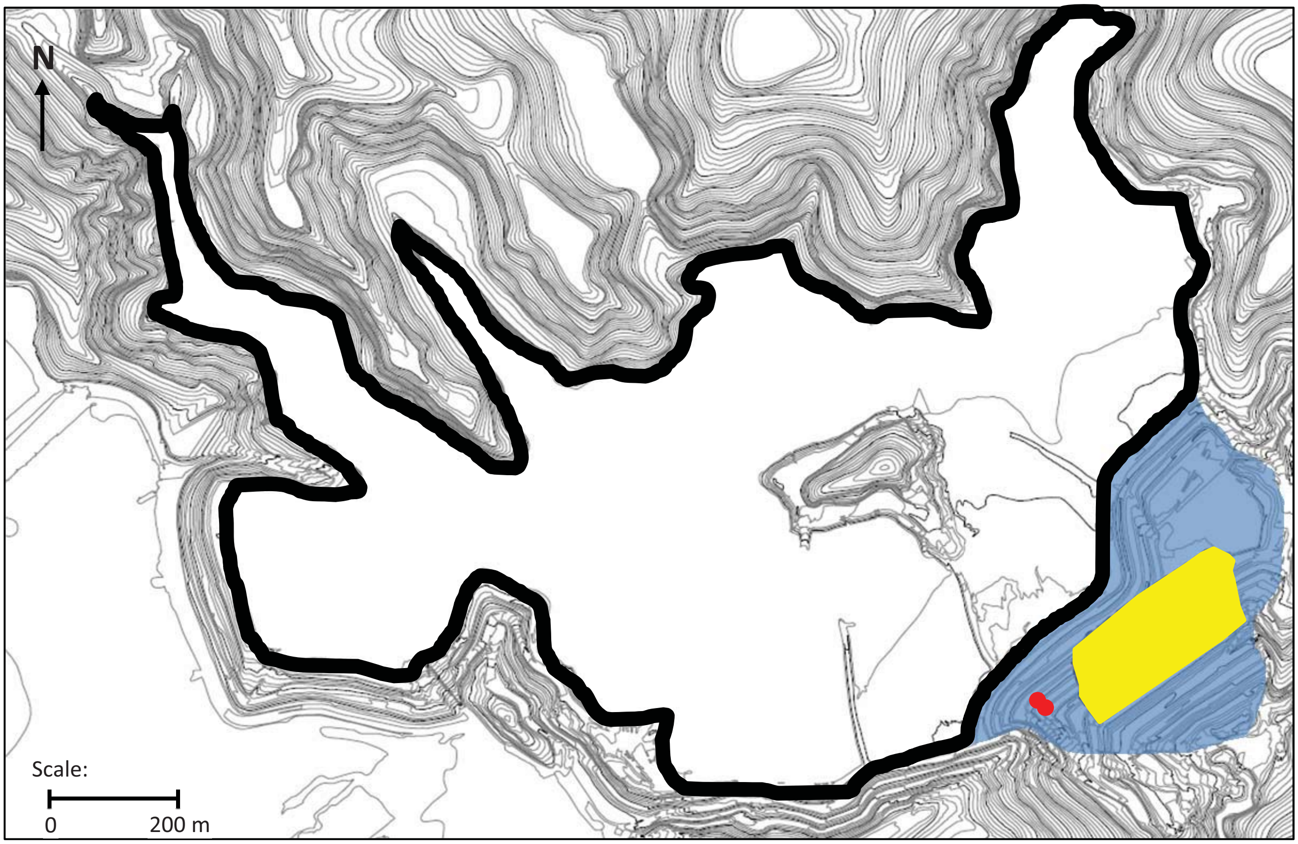

Figure 4-9 August, 2014 configuration showing El. 826 m blanket drain, raised dam, impoundment outline, and right abutment seeps (red dots)

Besides these incremental effects, a more fundamental change occurred on or about August, 2014 when three things happened simultaneously. As shown on Figure 4-10, flow from the El. 826 m blanket drain stopped increasing, then dropped briefly, partially recovered, and remained essentially unchanged thereafter. Also, flows from the Starter Dam base drain (a remnant of the original base drain salvaged after the 2009 piping incident) stopped diminishing and began increasing. In addition, artesian flow appeared at the toe of the Starter Dam.

An explanation consistent with these events is that the El. 826 m blanket drain with its Kananet® outlet pipes reached its maximum capacity. With the drain unable to divert additional seepage, saturation on the right abutment increased, breaking out in July at El. 855 m. At the same time, the diminished effectiveness of the blanket drain caused base drain flow to reverse its previous trend and begin increasing, while related increase in flow into the foundation caused artesian conditions to appear at the Starter Dam toe. And, as discussed subsequently in detail, all of these things were accompanied by the shallow saturation and unprecedented cracking of the left abutment that also occurred in August, 2014.

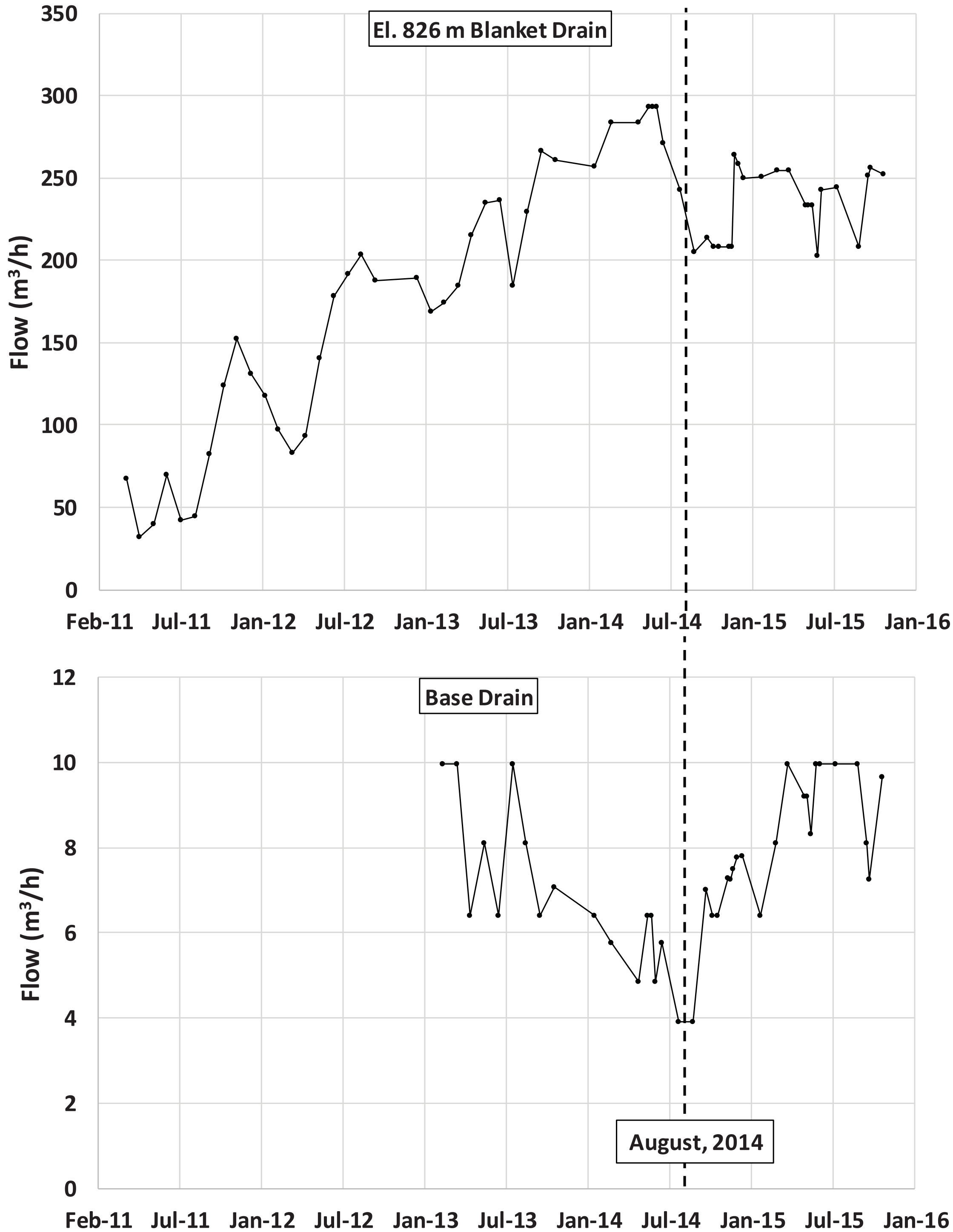

Figure 4-10 Measured flows from El. 826 m blanket drain and Starter Dam base drain

Figure 4-11 shows the drain configuration in November, 2015 with the ever-expanding impoundment. In preparation for the new 940 raise, the new blanket drain at El. 860 m had been completed at the left abutment setback and a companion drain at the right abutment was under construction. Had failure not intervened and had the dam alignment been restored, both of these new drains would have underlain the tailings much as the El. 826 m blanket drain once did. But as it was, neither had any effect on the tailings saturation that had already developed.

Figure 4-11 November, 2015 configuration showing El. 826 m and El. 860 m blanket drains, raised dam, and impoundment outline

Figure 4-12 summarizes the time-sequence of these impoundment drainage provisions. As upstream raising continued and the impoundment expanded, the El. 826 m blanket drain became increasingly more distant from the tailings it was intended to drain and eventually could no longer keep pace with rising saturation levels. With this, the saturated conditions necessary for liquefaction flowsliding were satisfied.

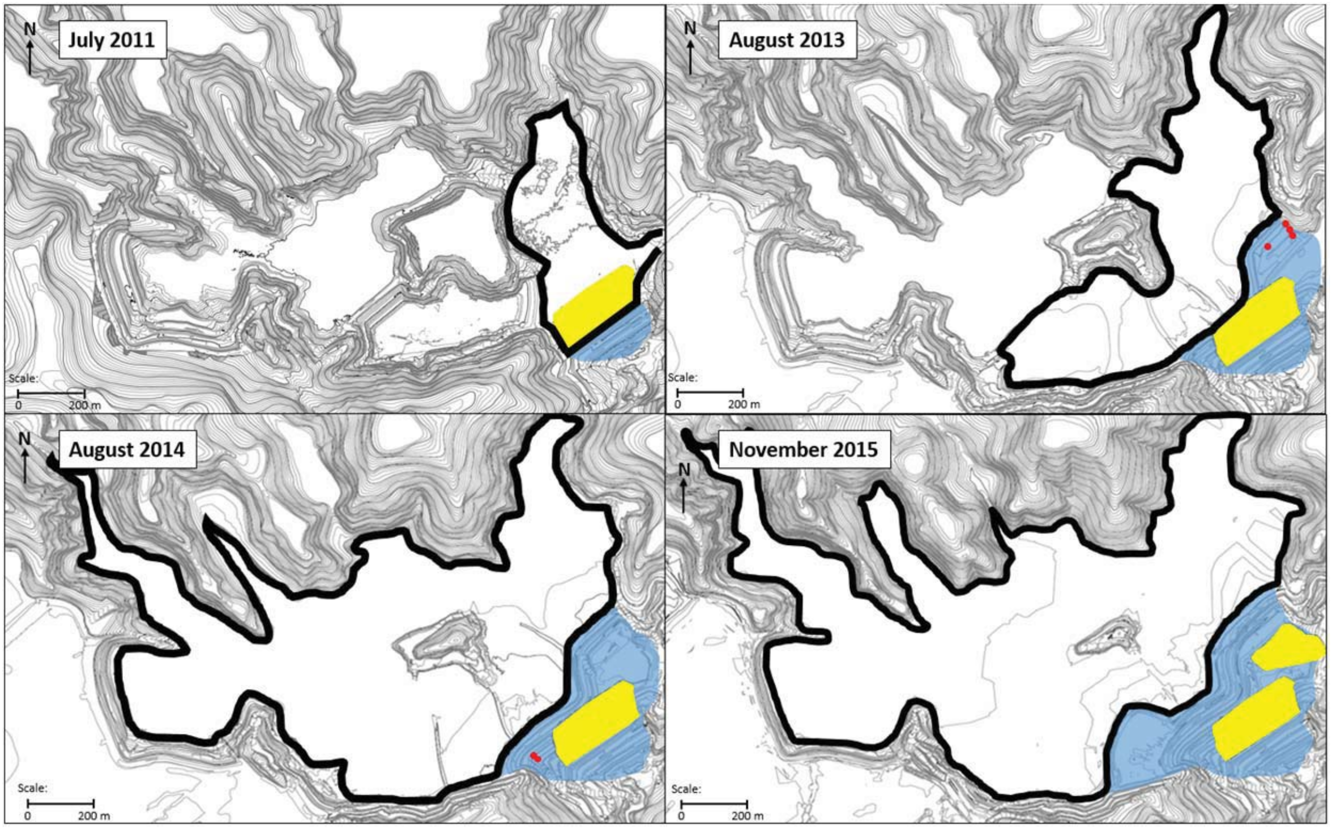

Figure 4-12 Progression of impoundment and drainage provisions with time

The beginning of this section posed the question: Why did a flowslide occur? In response, it has been shown that all of the necessary conditions were present. The sand tailings were contractive, they were saturated, and they were susceptible to severe loss of strength during the rapid failure that developed.

But an important factor has yet to be addressed. And that concerns the slimes.

5 WHY DID THE FLOWSLIDE OCCUR WHERE IT OCCURRED?

The eyewitness accounts summarized in Section 2.7 show that the flowslide of November 5, 2015 initiated on the left abutment where the dam had been set back from its former alignment. Section 4 has established why the flowslide occurred. It remains to explain why the flowslide initiated at the left abutment and not at some other location.

To do so requires identifying features or properties unique to the left abutment. In this respect, the defining feature of the left abutment setback was the presence of slimes beneath the embankment slope. The following discussions explain the characteristics of the slimes, where they were deposited, and how the setback influenced their effect on the embankment. Comparing these factors at the left and right abutments shows why the failure initiated at the former and not the latter.

5.1 The Slimes

5.1.1 Slimes Characteristics

Two types of tailings, *sands* and *slimes* were produced in the plants and conveyed in separate slurry pipelines to the Fundão and Germano impoundments. The sands are cohesionless and the slimes cohesive in character. As indicated on Figure 5-1, the two materials are readily distinguished by their color, the sands being gray and the slimes variously described as red, brown, or chocolate color.

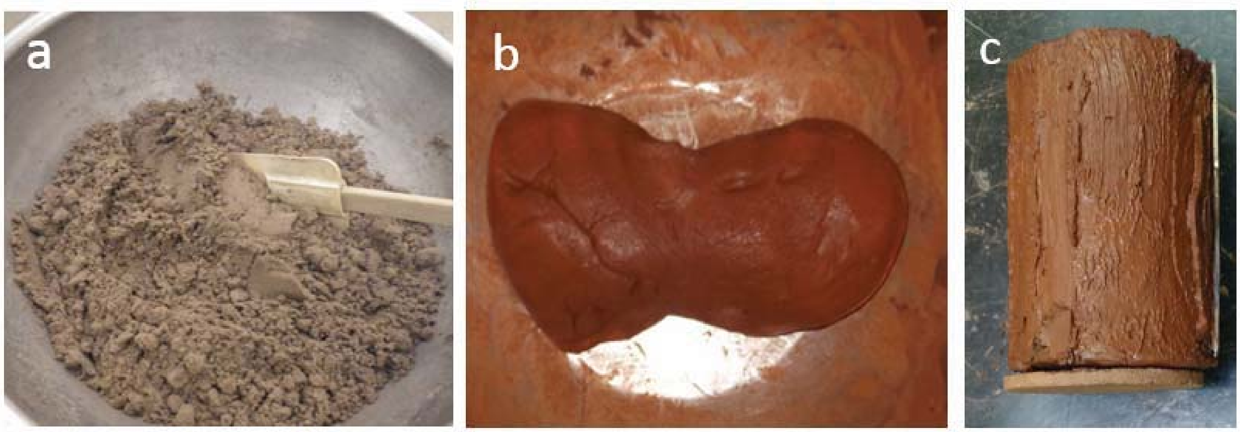

Figure 5-1 Sand and slimes tailings. (a) sand; (b) remolded slimes; (c) intact slimes specimen

The gradations of the two materials are compared on Figure 5-2, which shows that the sands contain approximately 40% silt, while the slimes are comprised entirely of silt and clay-sized particles.

The slimes contain only a small proportion of conventional clay minerals illite and kaolinite, the majority being hematite and goethite with some quartz. X-ray diffraction analysis of mineral composition is shown on Table 5-1.

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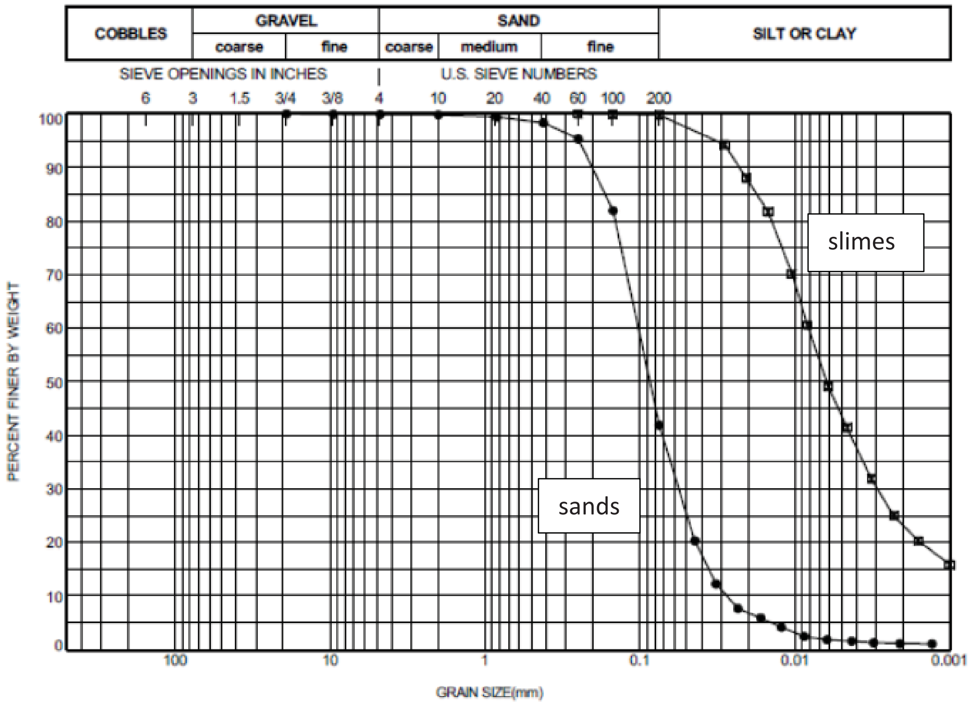

Figure 5-2 Sands and slimes gradation

Table 5-1 Slimes mineralogy

Mineral	Ideal Formula	#1 Slimes
Chalcopyrite ?	$CuFeS_2$	< 0.1
Goethite	α-$Fe^{3+}O(OH)$	30.9
Hematite	α-Fe_2O_3	42.9
Illite-Muscovite	$KAl_2AlSi_3O_{10}(OH)_2$	1.4
Kaolinite	$Al_2Si_2O_5(OH)_4$	4.4
Plagioclase	$NaAlSi_3O_8 – CaAl_2Si_2O_8$	1.1
Quartz	SiO2	19.2
Total		100.0

The concentration of iron-derived minerals in the slimes gives them a high specific gravity of nearly 4.0 that distinguishes them from the lighter sands. Where laboratory testing was available, the Panel was able to use specific gravity as a marker to distinguish the relative proportion of sands and slimes in tested samples. Despite the near-absence of clay minerals, the slimes classify as low-plasticity clay from Atterberg limits, with corresponding low permeability. Index properties of the slimes are given on Table 5-2, along with those for sands for comparison.

Table 5-2 Index properties

Property	Sands	Slimes
percent minus 0.074 mm	40-45	98-100
percent minus 0.002 mm	<2	20-25
specific gravity	2.8-2.9	3.9-4.0
plasticity index	non plastic	7-11
permeability	$3x10^{-4}$ cm/s	< 10^{-6} cm/s

While the two materials in unadulterated form are easily distinguishable based on these measured properties, they are often mixed in various proportions in the field. Without laboratory testing, slimes can be difficult to identify from ordinary soil classification techniques, making their signature color their distinguishing characteristic in the field.

From the standpoint of the behavior of the Fundão Dam, the most important engineering property of the slimes that distinguishes them from sands is deformability. The slimes are softer and more compressible, as indicated by the compression curves on Figure 5-3. It will be shown in the following section that deformability of the slimes was a central factor in triggering liquefaction in the sands.

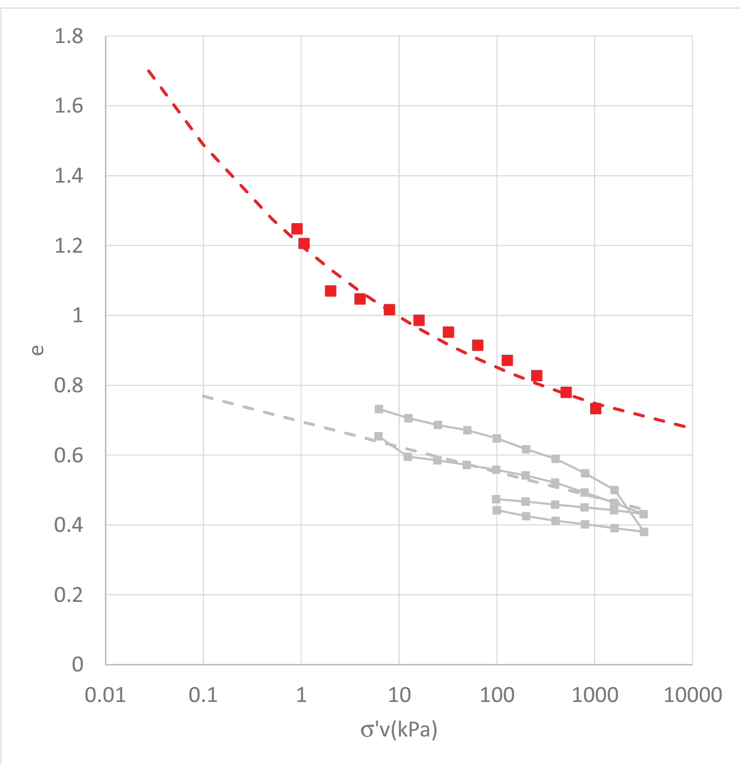

Figure 5-3 e log p curves for sand (grey) and slimes (red) from laboratory and field data; dashed lines used in modeling

5.1.2 Slimes Deposition and Identification

The tailings deposition process governs how sands and slimes are distributed areally and with depth. Figure 5-4 depicts this process in an idealized way when sands are discharged onto an above-water beach and from there into ponded water containing suspended slimes. Spigotting deposits exclusively sand tailings on the beach, while predominantly slimes sediment from the ponded water at greater distance. Between these two areas is an intermediate zone where intermixing of sands and slimes occurs at times when sands are being discharged. When sand discharge is temporarily suspended or relocated elsewhere, slimes layers are deposited that become embedded in and interlayered with the intermixed materials.

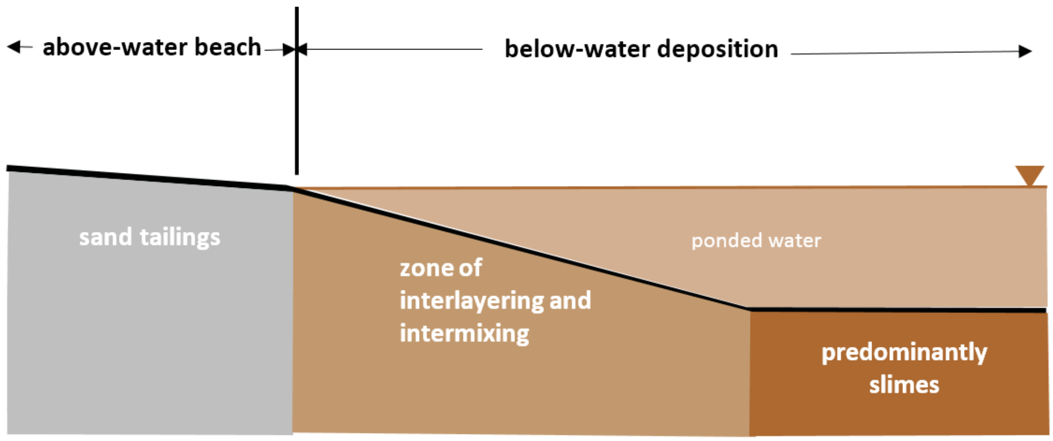

Figure 5-4 Idealized process of sands and slimes deposition

Figure 5-4 represents conditions at a particular moment in time, but the actual process is dynamic and constantly changing. The location of the interface between the beach and ponded water depends on both the depth of water—which varies according to precipitation inflows and water release—and the amount of tailings reaching that location from the sand discharge pipeline—which is regularly relocated. Thus, the dimensions of the three zones are always shifting. For the same reasons, they change with depth as the deposit accretes.

Figure 5-4 constitutes the conceptual basis for reconstructing tailings stratigraphy during the Investigation. The slimes-laden ponded water can be readily identified by its red color. Imagery from a variety of sources and related records provide snapshots of ponded water location and configuration that, when assembled and tracked over time, produce a three-dimensional representation of sands and slimes. The procedures used to create this tailings deposition model are described in Appendix A, and key findings are presented in Appendix B.

The Panel's development of the tailings deposition history involved review and distillation of hundreds of documents and records, most importantly: (1) publically-available satellite photographs; (2) drone photographs and post-2012 topography; (3) monthly Samarco instrumentation reports; (4) weekly construction reports; (5) consultant reports; and (6) interviews with Samarco engineering staff. The consultant reports were the primary source of drill holes, cone penetration tests and laboratory data on the tailings sands and slimes. The engineering data from these reports are summarized in Appendices C and D.

Topographic information was assembled in Civil 3D using 40 different sets of basin-wide topographic surfaces at successive dam heights from 2009 to November, 2015. The quality of that topographic information increased considerably after Samarco initiated its drone program in early 2013. With the drone aerial photographs and topography, it was possible to input stripped ground topographic surfaces into Civil 3D to model the as-constructed base of Dike 1. This was especially useful because the abutments were stripped as the dam was raised. Some 15 different stripped surfaces were stitched into Civil 3D.

All information was assembled in a geographic information system (GIS) which enabled data to be queried and displayed in multiple views in real time. The Civil 3D model and the GIS system became the common source of the topography, stratigraphy and groundwater data used in the suite of analyses for this work.

5.1.3 Slimes Mapping

It can be recalled from Section 2.1 that the original design concept for Fundão was predicated on free-draining conditions in the sand tailings comprising the embankment. Achieving this required that drainage not be impeded by deposition of lower-permeability slimes. This was to have been assured by physically separating and separately discharging of the sand and slimes tailings behind Dikes 1 and 2, respectively.

Section 2.2 described how this concept was abandoned after the Dike 1 Starter Dam was seriously damaged by piping and internal erosion. In addition, subsequent structural problems with the Main and Secondary Galleries made it necessary to re-route water and slimes from the Dike 2 impoundment into Dike 1. These problems happened during 2011 and 2012 when tailings that would later underlie the left abutment setback were being deposited. The aerial image in Figure 5-5 shows that an Overflow Channel was constructed from the Dike 2 slimes reservoir to the Dike 1 reservoir from January, 2011 to July, 2012. This Overflow Channel introduced slimes into the Dike 1 reservoir.

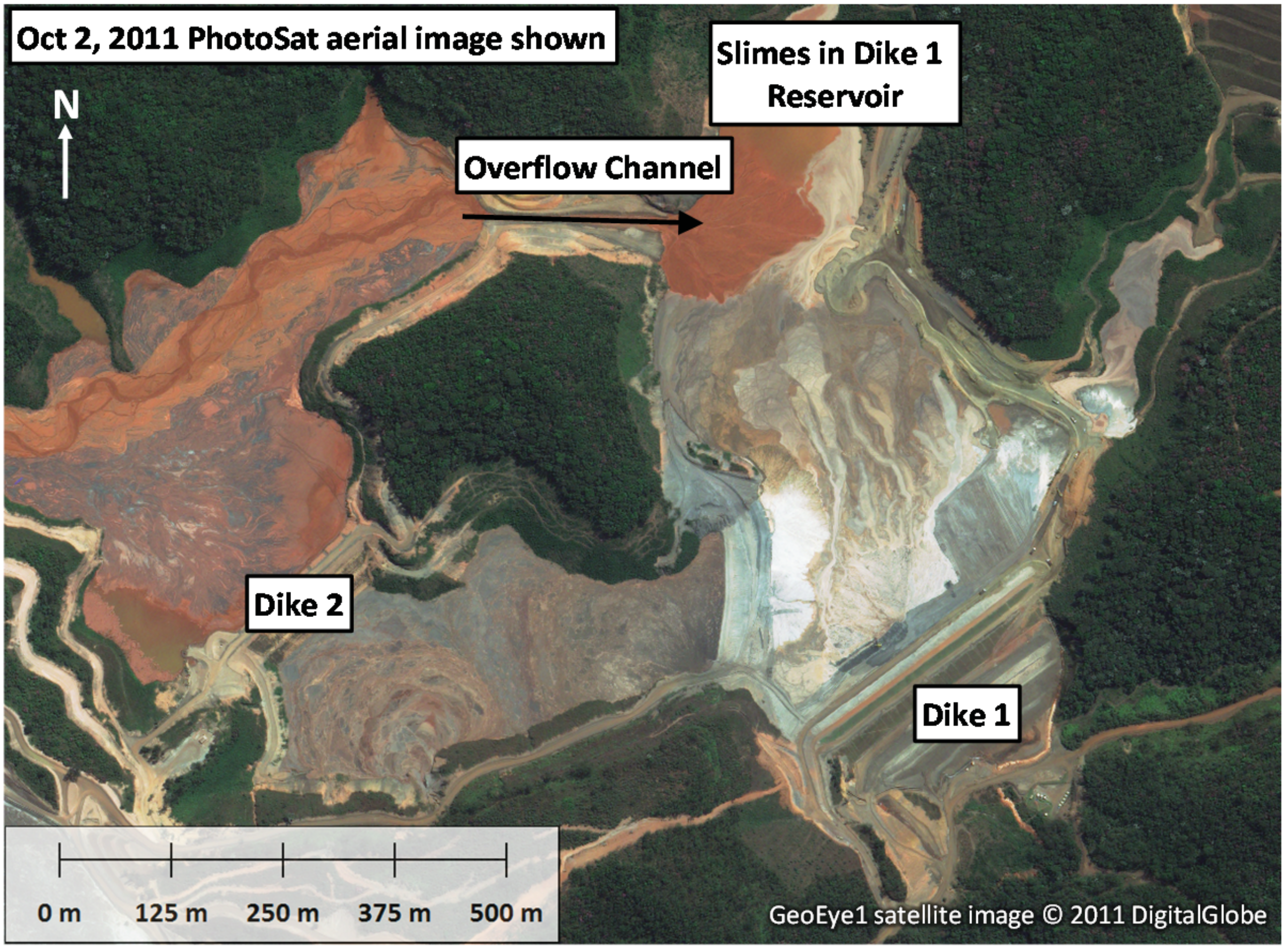

Figure 5-5 Slimes Overflow Channel from Dike 2 reservoir to Dike 1 reservoir

The Overflow Channel was closed in August, 2012 but not before slimes were deposited between El. 824 m and El. 850 m. Operation of the Overflow Channel and resulting slimes deposition is illustrated by imagery at selected dates during late 2011 and early 2012 on Figure 5-6.

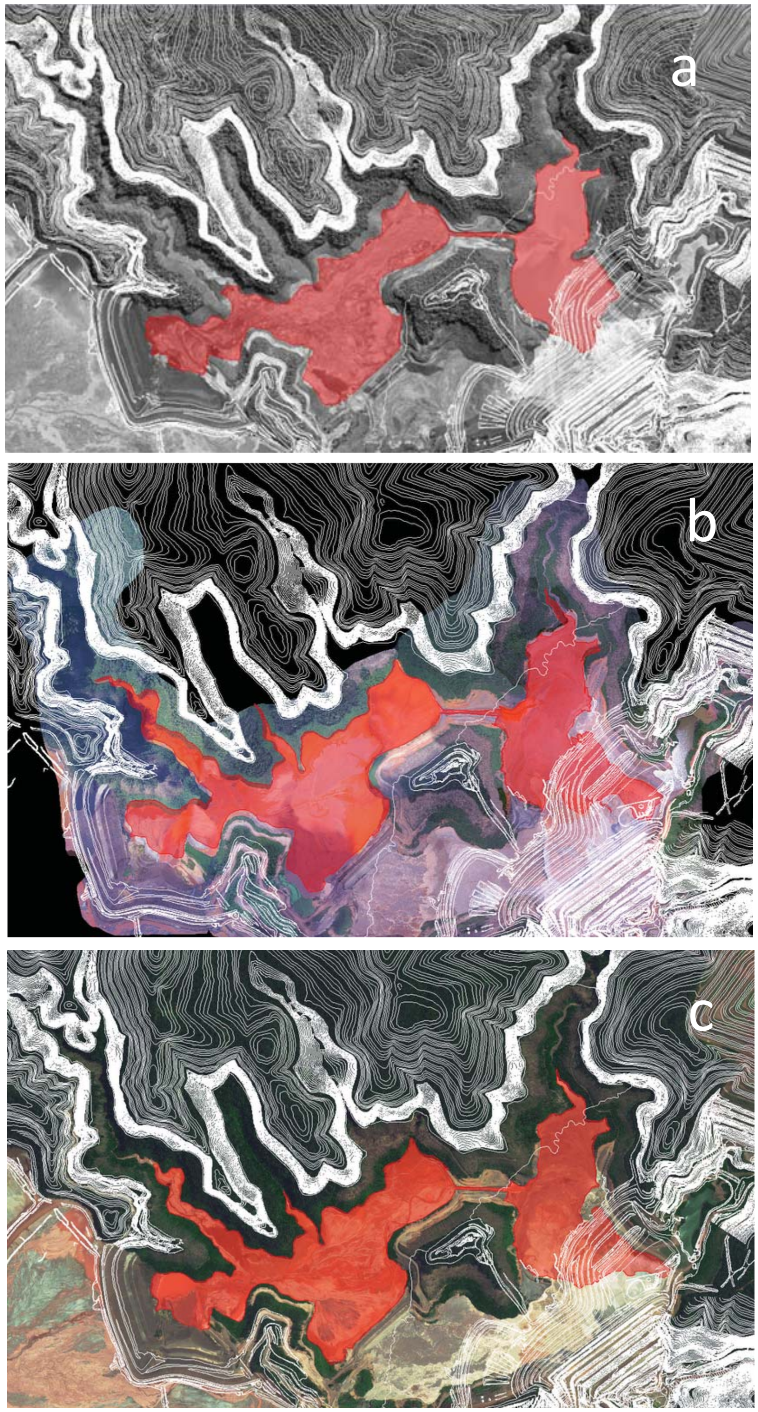

Figure 5-6 Slimes deposition (a) September 20, 2011; (b) January 21, 2012; (c) March 3, 2012. Slimes highlighted in red; final embankment contours in white

5.1.4 Drill Hole Information

The locations of slimes inferred from mapping can be compared to logs of drill holes through the left abutment tailings. Figure 5-7 shows borings and CPTs that penetrated a target interval of El. 830 m to El. 850 m. These holes were drilled at different times and surface elevations, but for reference they are superimposed on imagery from January, 2012. Also on Figure 5-7 for comparison are the outlines of slimes mapped at El. 841 m on that date that also appear on Figure 5-6b.

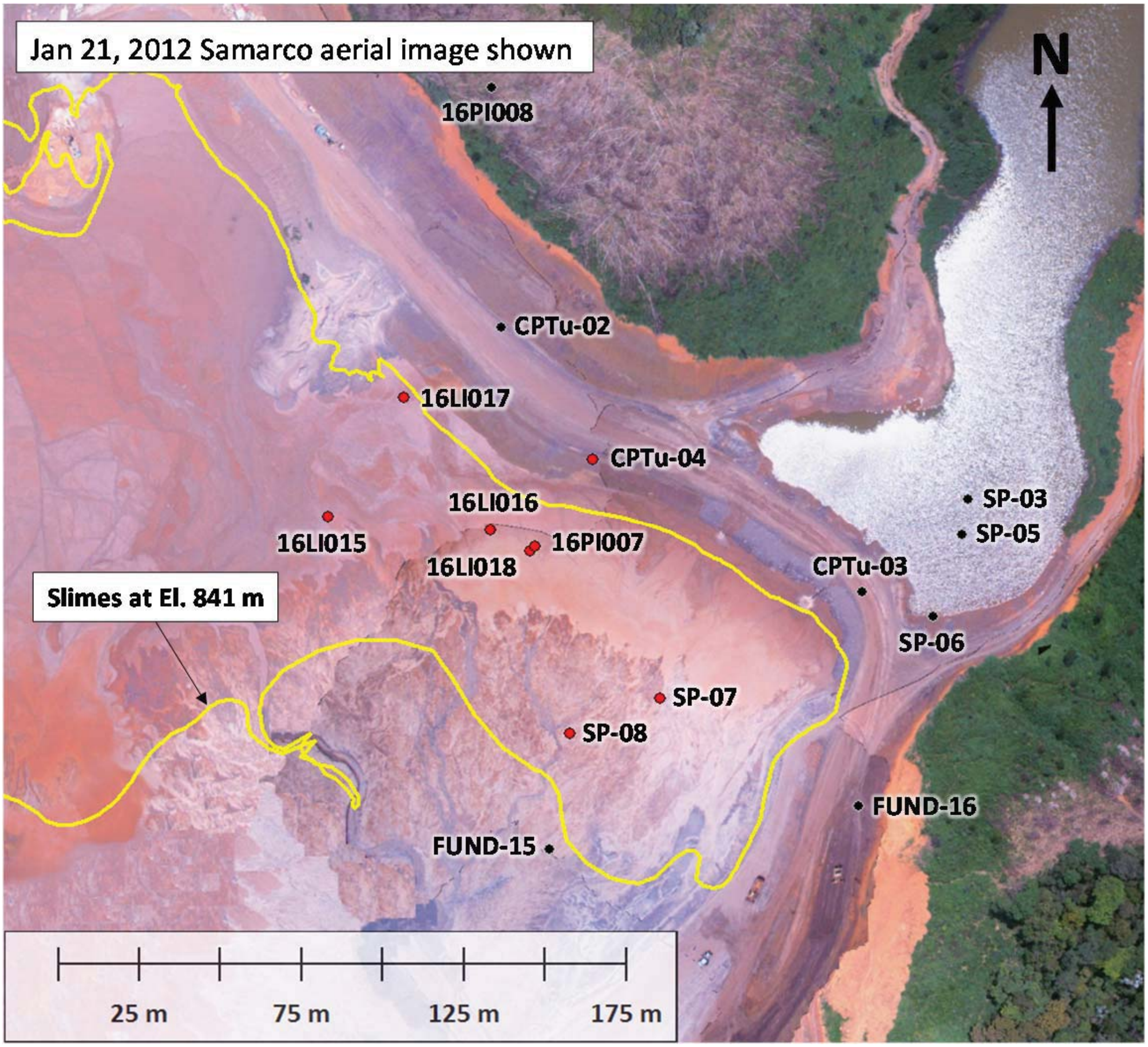

Figure 5-7 Left Abutment Drill Holes. Red circles indicate slimes within target interval of El. 830 m to El. 850 m.

No laboratory testing was conducted on recovered samples from any of the borings, so sample descriptions on the logs rely on visual classification alone with associated uncertainties as discussed in Section 5.1.1. For purposes of this assessment, slimes were taken as material logged as red or brown in color, as opposed to grey for sands, often noting the presence of clay in varying amounts. Slimes were taken in CPTs as materials having apparent fines content of 100% that are not associated with road fill or other introduced materials. On this basis, Figure 5-7 shows holes indicating the presence of slimes within the target elevation interval in red and holes with no such indications in black. It can be seen that the drill hole information corresponds to the area of mapped slimes. A number of other holes not shown on Figure 5-7 were drilled over the Secondary Gallery for purposes of investigating its foundation conditions. With a surface elevation at or near El. 835 m, they mostly penetrated tailings below the target interval.

Within those holes where slimes were identified, their distribution is more difficult to determine because the tailings were not continuously sampled. However, SP-07 is notable in having distinguished two discrete clay layers corresponding to slimes: a 2 m thick layer at El. 836.36 m and a deeper 2 m layer at El. 828.36 m. Also, CPTu-04 penetrated slimes layers up to several centimeters thick. It is reasonable then from the drill holes to categorize discrete layers of slimes as ranging from a few centimeters to a few meters in thickness, with the remainder of the slimes material intermixed with sand in varying proportions. This characterization is consistent with the zone of interlayering and intermixing portrayed on Figure 5-4.

5.1.5 Slimes Mass Balance

The distribution of slimes estimated from drill hole information can be supplemented on a broader level using a mass balance approach. To this end, slimes production records were compared with the potential slimes volumes between El. 840 m and El. 850 m. A mass balance was derived by assuming a dry unit weight of the slimes taken from measurements of recently-deposited slimes sampled in Germano as part of this Investigation. The mass balance assumptions and calculations are given in Appendix B.

The mass balance provides a measure of the volumetric proportion of constituent slimes and slimes layers within a specified zone. Figure 5-8 portrays the results in cross section. Here, the region designated *predominantly slimes* is nearly 100% slimes, *interbedded slimes* is estimated to contain 20% or more slimes overall, and the zone of *isolated slimes* less than this amount. Although distinct boundaries are shown between these regions, actual conditions are transitional in character.

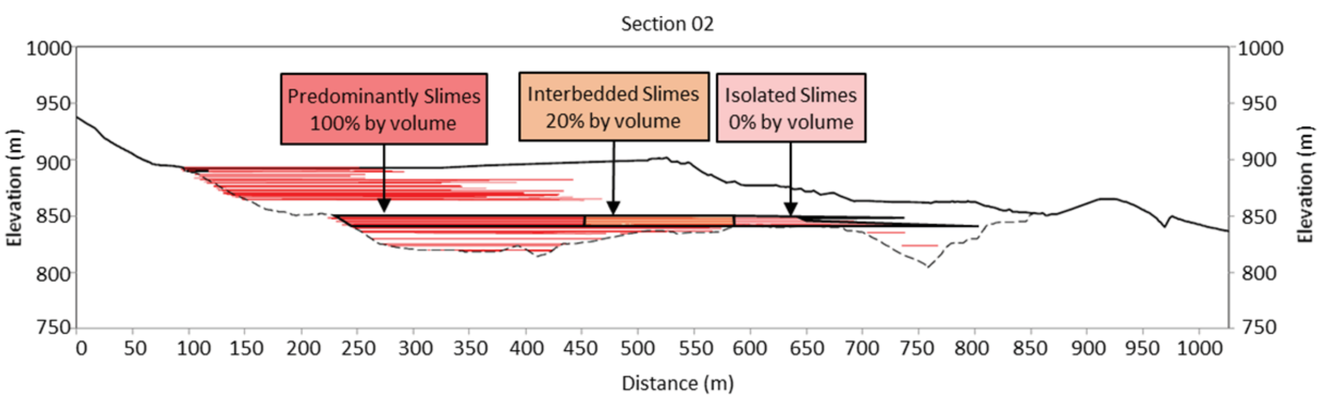

Figure 5-8 Distribution of slimes at left abutment

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The preceding discussions have shown that slimes were present at the left abutment, in particular between El. 830 m and El. 850 m, and that their concentration increased with distance behind the dam. How these conditions influenced the embankment requires accounting for the setback.

5.2 The Left Abutment Setback

5.2.1 Events and Circumstances

Circumstances surrounding the modification of the dam alignment that resulted in the setback have been reviewed in Section 2.4. Due to structural problems and construction defects, the dam could not continue to be raised over the Secondary Gallery until repairs had been made. But when these repairs proved unsuccessful, the Secondary Gallery had to be abandoned and filled with concrete. To accommodate tailings storage requirements during these periods, beginning in October, 2012 the dam alignment at the left abutment was shifted back from its former location as shown on Figure 5-9. This created what is called here the *setback*, with the vacant area in front of it the *plateau* or *platform*.

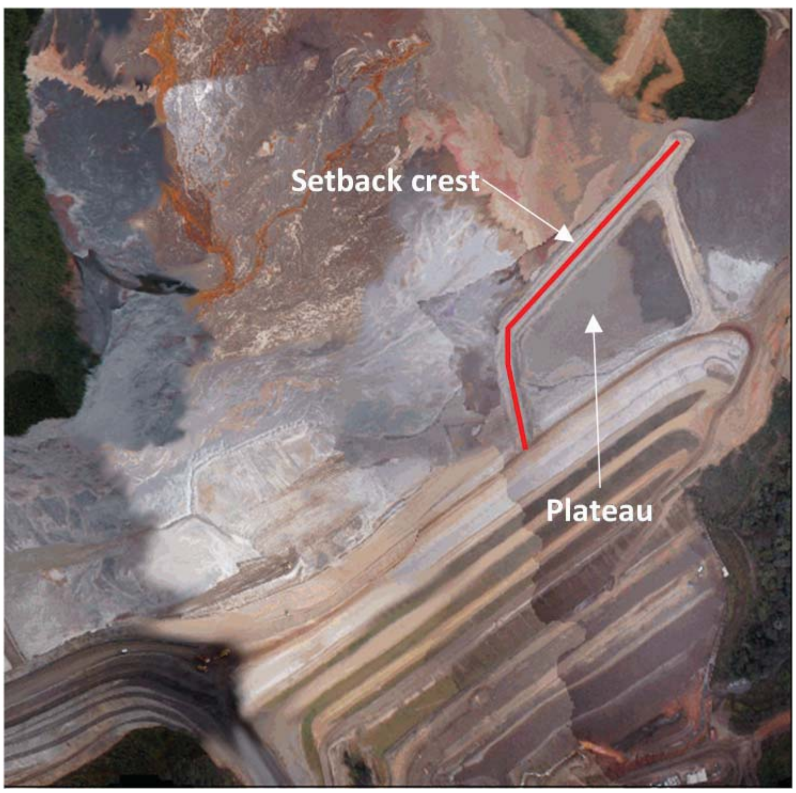

Figure 5-9 Aerial photograph of the setback alignment in October, 2012

The setback was initiated when the plateau was at approximately El. 855 m. By the end of 2013, the crest had risen to El. 877 m, or about 22 m high.

Starting in August, 2013, the first compacted fill was placed to rebuild the setback portion of the embankment and return it to its former alignment while dam raising continued. This occurred until August, 2014 when the slope showed serious signs of distress as reviewed in Section 2.5. The setback was immediately buttressed with a tailings sand berm. By then, the crest had reached El. 885 m, or 30 m high.

Infilling of the setback was further delayed by requirements for the proposed raise of the Fundão Dam to a crest elevation of 940 m. Design analyses concluded that more drainage would be needed to reduce the phreatic surface on both abutments, including a large blanket drain at the left abutment to be constructed in four stages. The first stage was a blanket drain at El. 860 m on the setback plateau. Construction began in November, 2014 and did not conclude until August, 2015 when setback infilling was resumed.

The setback had significant effects. Moving the embankment back toward the impoundment caused it to be raised over the slimes deposited in 2011 and 2012. In addition to influencing foundation conditions, these slimes also changed the seepage regime, elevating the phreatic surface on the left abutment.

5.2.2 Slimes Configuration

The combined effect of the slimes deposition described in Section 5.1 and the setback is best shown by a series of illustrations representing various points in time. The extent of newly-added tailings since the previous time step is indicated on the sections and in plan view on the insets to Figure 5-10. Compacted tailings that were mechanically placed during embankment raise construction are distinguished from the hydraulically-discharged sands by lines separating the two on the cross sections.

Figure 5-10 shows that the alignment setback caused all or most of the embankment to be constructed over slimes. In addition, as the embankment became higher and its crest moved upstream, more of the embankment slope became underlain by the higher proportion of slimes in the interbedded region.

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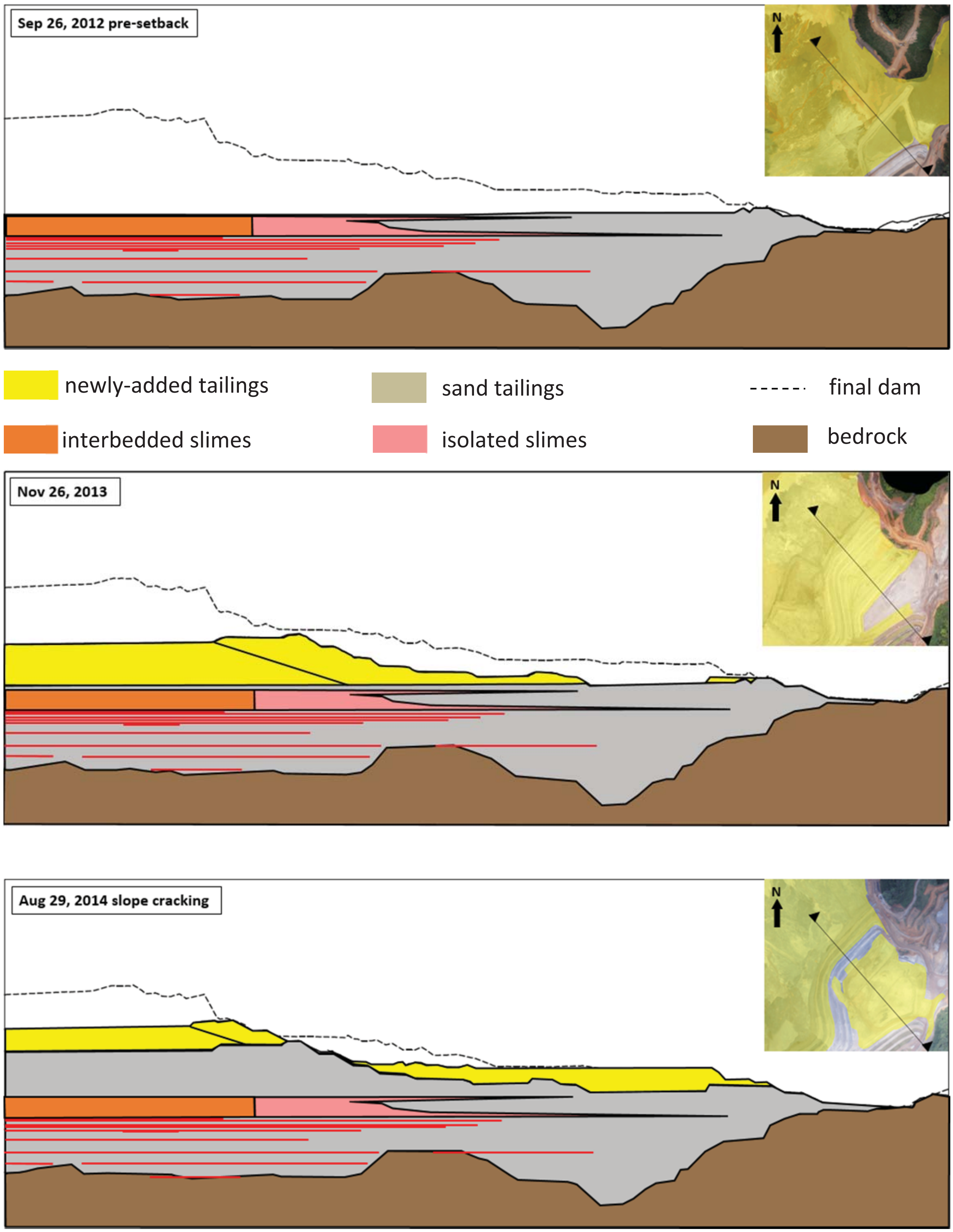
Sep 26, 2012 pre-setback
N
newly-added tailings
sand tailings
final dam
interbedded slimes
isolated slimes
bedrock
Nov 26, 2013
N
Aug 29, 2014 slope cracking
N

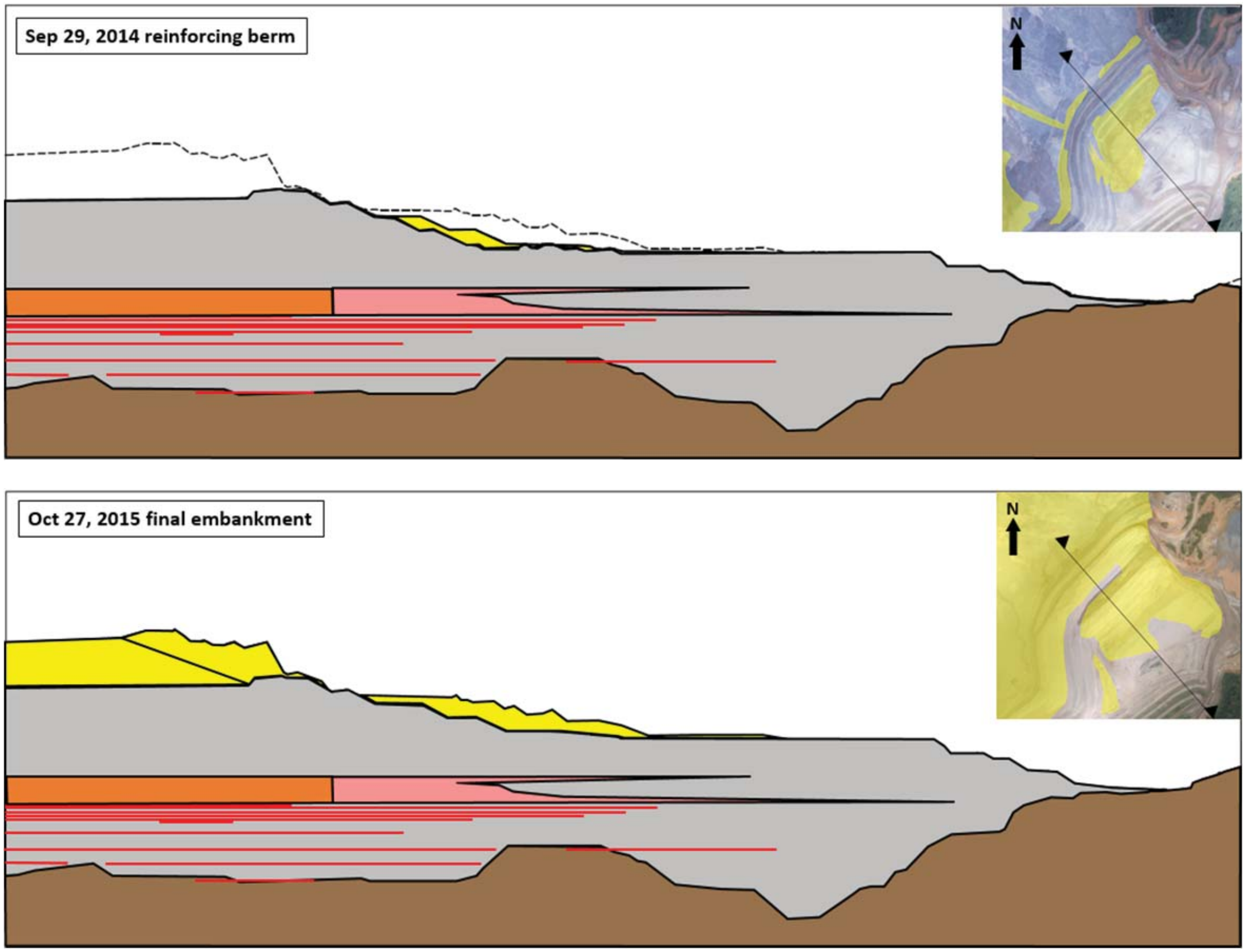

Figure 5-10 Sequential raising of setback embankment over slimes

The areal extent of the slimes depicted on Figure 5-10 deposited in 2011 and 2012 beneath the left abutment setback is shown on Figure 5-11, where again it can be seen that by the time the embankment reached its final height, slimes would be present beneath the entire setback slope and much of the plateau area.

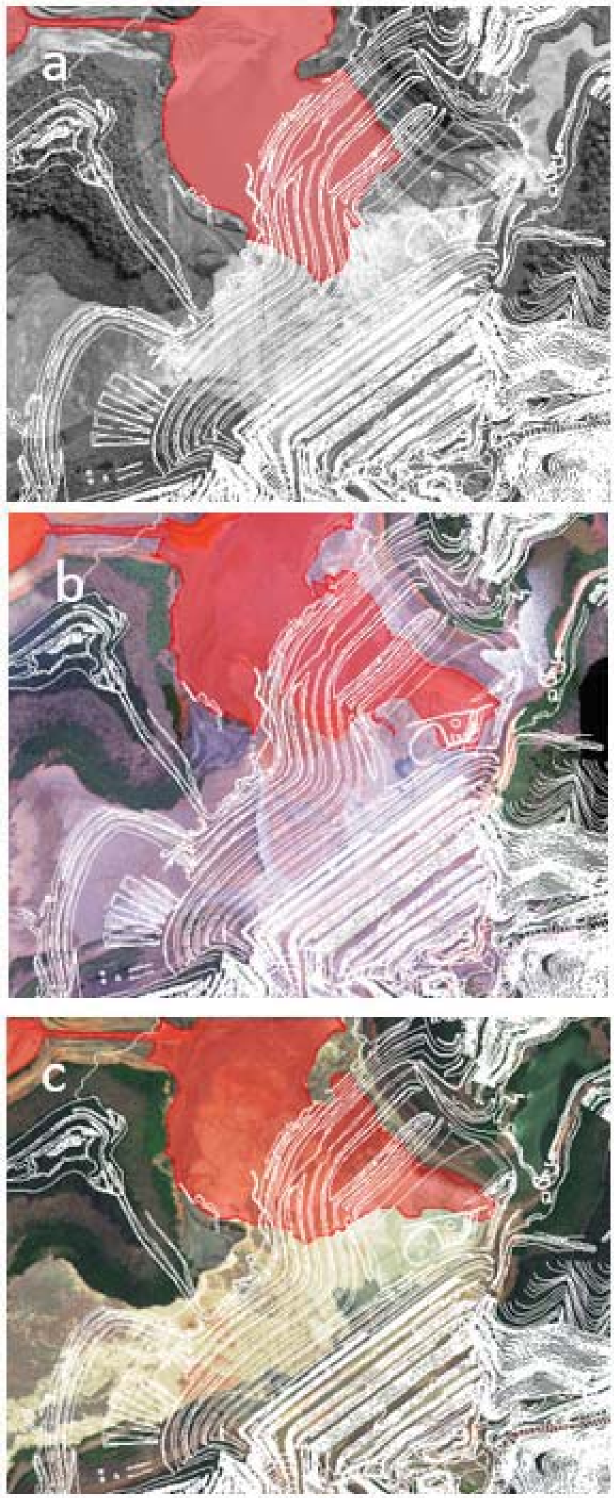

Figure 5-11 Slimes beneath final embankment: (a) September 20, 2011; (b) January 21, 2012; (c) March 3, 2012. Slimes highlighted in red; final embankment contours in white

5.2.3 Rate of Rise

As indicated on Figure 5-12, the rate of rise of the dam crest at the left abutment varied during the life of the setback. The average 1.3 m/mo during 2015, or an annualized rate of 15.7 m/yr, was intermediate between rates experienced during 2013 and 2014. Also during 2015 in the months immediately prior to failure, raising accelerated from as low as 0.4 m to 2.9 m. The small negative rate of rise in September, 2014 was produced by regrading related to construction of the reinforcing berm and is not consequential to the overall trend.

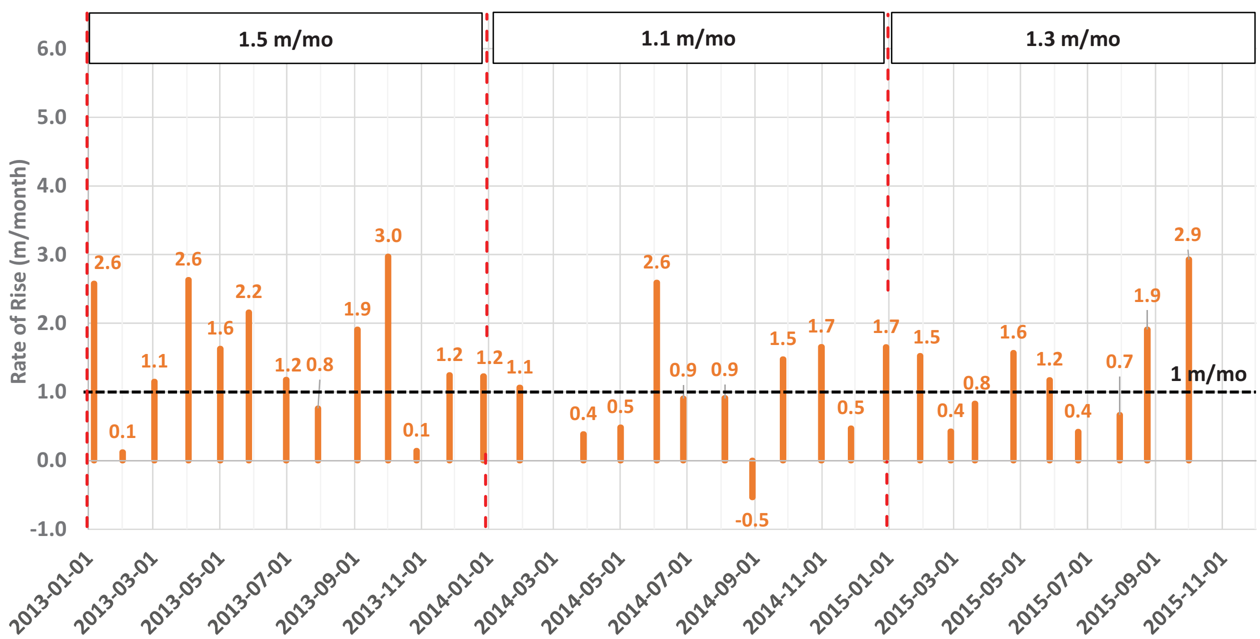

Figure 5-12 Rate of dam crest rise at left abutment setback

5.3 Comparison of Left and Right Abutments

Thus far, it has been established that slimes existed beneath the embankment slope at the left abutment as a consequence of their earlier deposition together with the setback of the dam alignment. To explain why failure initiated here and not elsewhere, it is useful to compare conditions at the left abutment to those at the right, where failure resulted from and was preceded by flowsliding on the left. The question then becomes why failure initiated at the left abutment and not the right. This requires comparing conditions at the two locations.

5.3.1 Right Abutment Conditions

Figure 5-13 shows Section AA at the right abutment and its internal composition based on mapping as described in Section 5.1.3. The distinguishing feature of the right abutment compared to the left is the nearly complete absence of slimes beneath the embankment slope. The only region of slimes lies below El. 825 m where it is confined both upstream and downstream by natural ground. This is in sharp contrast to the left abutment on Figure 5-8 where slimes can be seen to extend beneath virtually the entire length of the slope.

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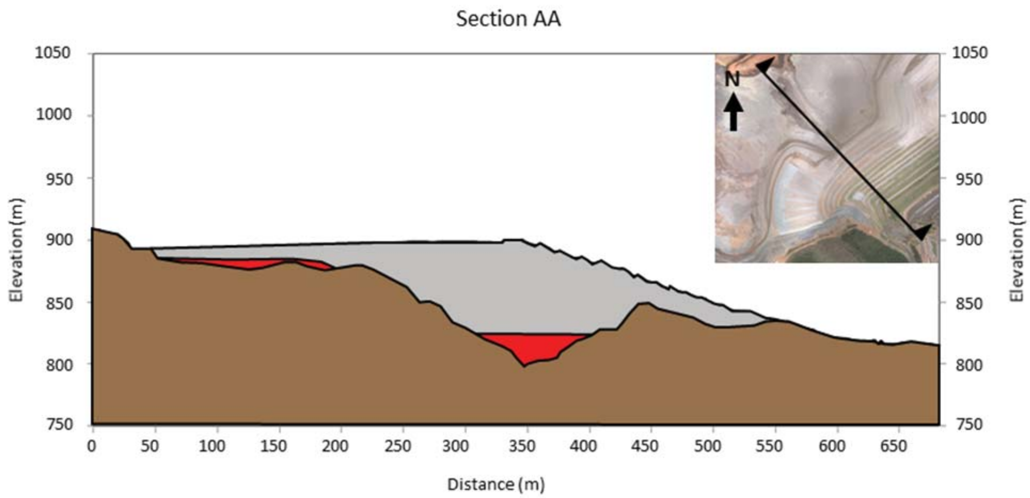

Figure 5-13 Slimes at right abutment Section AA

The left and right abutments can be compared more directly by overlaying the respective sections. Figure 5-14 provides an overlay of Section 01 at the left abutment and right abutment Section AA, coincident at the respective dam crests. Geometrically, the greater overall steepness and extended length of the 3.0H:1.0V right abutment slope stands out.

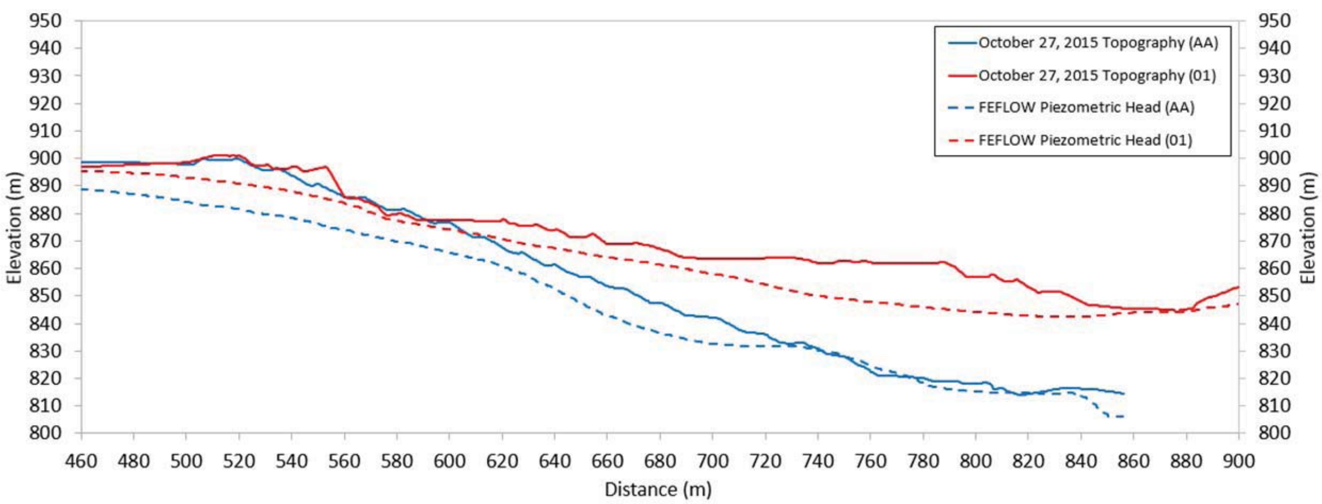

Figure 5-14 Geometry and piezometric comparison of left and right abutments

Also depicted on Figure 5-14 is the difference between the piezometric surface at the two locations, with generally higher conditions on the left that reflect the influence of slimes. This is illustrated in more detail on Figure 5-15.

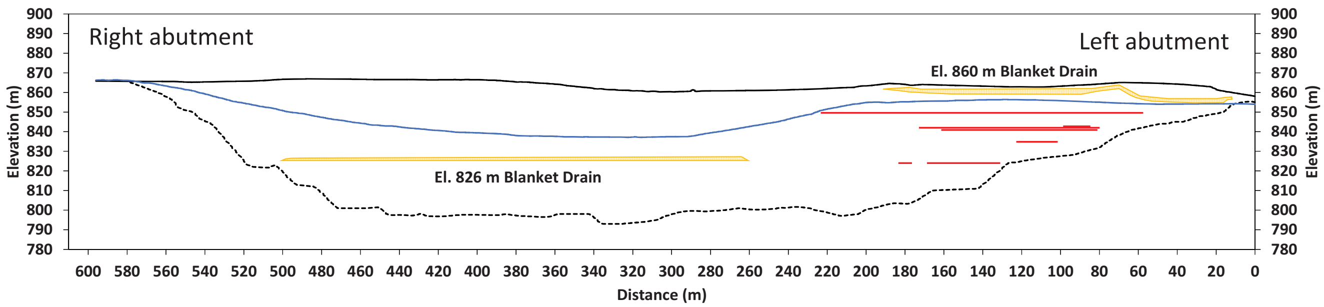

Figure 5-15 Longitudinal section from FEFLOW, view looking upstream. Phreatic surface shown in blue, El. 826 m blanket drain in yellow, slimes in red.

Figure 5-15 is based on results of a 3D steady state and transient seepage analysis of the tailings impoundment performed as part of the Investigation and described in Appendix G. The estimated phreatic surface from this modeling is shown on the longitudinal section. On the left abutment, inflows from Grota da Vale together with the slimes maintain the phreatic surface at a higher elevation. In contrast to the right abutment, the slimes on the left extend outward toward the El. 826 m blanket drain and limit the lateral extent of its influence.

The rate of rise of the embankment crest at the right abutment is shown on Figure 5-16, which can be compared to conditions at the left from Figure 5-12. Average annual rates are generally similar except during 2015 when the right abutment was raised at a higher rate of 1.6 m/mo compared to 1.3 m/mo on the left. The right abutment experienced an unusually high rate of rise of 5.4 m/mo in August, 2015 on a newly-initiated realignment of the crest that was set back from the former location to allow for drain construction depicted on Figure 2-12.

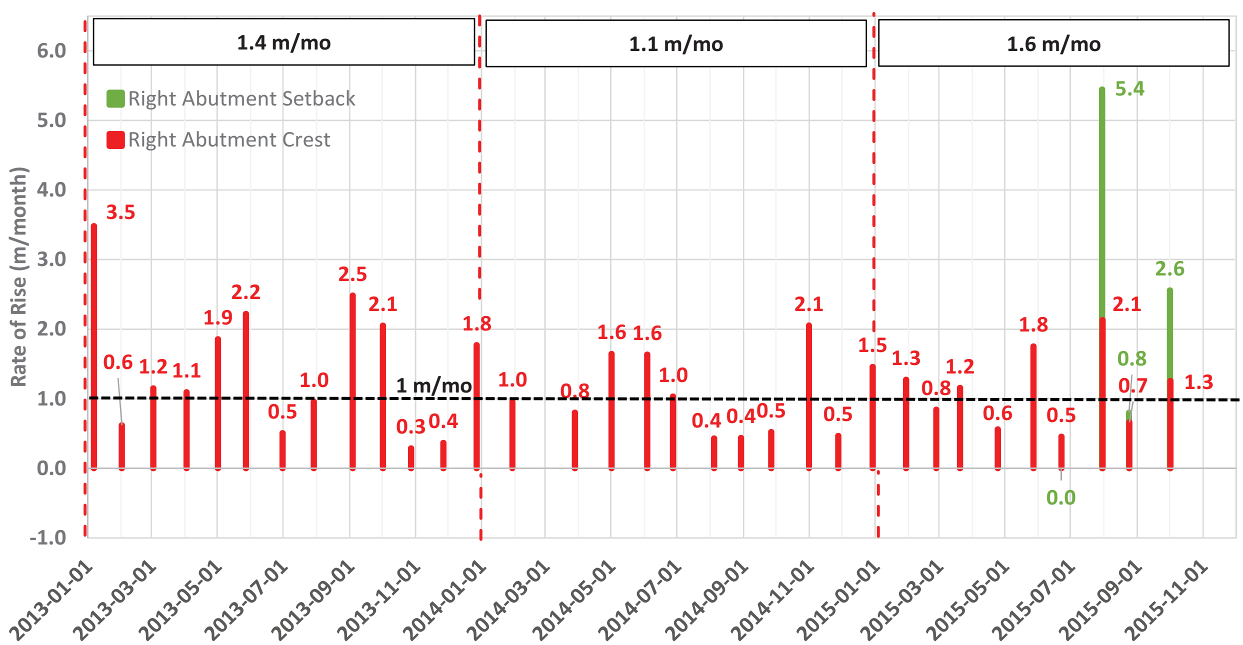

Figure 5-16 Rate of rise at right abutment

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Conditions conducive to failure at the right abutment therefore include greater slope steepness and higher rate of rise than the left abutment, although these are mitigated somewhat by the lower piezometric conditions. The net effect can be evaluated by means of stability analysis.

5.3.2 Right Abutment Stability

Section 4.1 reviewed the aspects of soil behavior that give rise to flowsliding in saturated, contractive materials. Shearing that occurs slowly enough to allow pore pressures to dissipate is said to occur under *drained* conditions, while *undrained* conditions pertain to rapid shearing associated with flowsliding.

Stability analyses for these two conditions adopt corresponding strength parameters. *Effective-stress analysis*, or ESA, uses friction angle and cohesion to represent drained shearing, while *undrained strength analysis*, or USA, for undrained shearing uses undrained strength typically expressed as a ratio to the effective vertical overburden stress.

Stability analyses for the right abutment Section AA on November 5, 2015 are shown on Figure 5-17 for both ESA and USA, where a calculated factor of safety (FS) less than 1.0 indicates failure. The ESA adopts a friction angle of 35 degrees and 5 kPa for compacted tailings fill and 33 degrees and zero kPa for hydraulically-discharged tailings, while the USA uses an undrained strength ratio in compression of 0.25 for tailings below the piezometric surface.

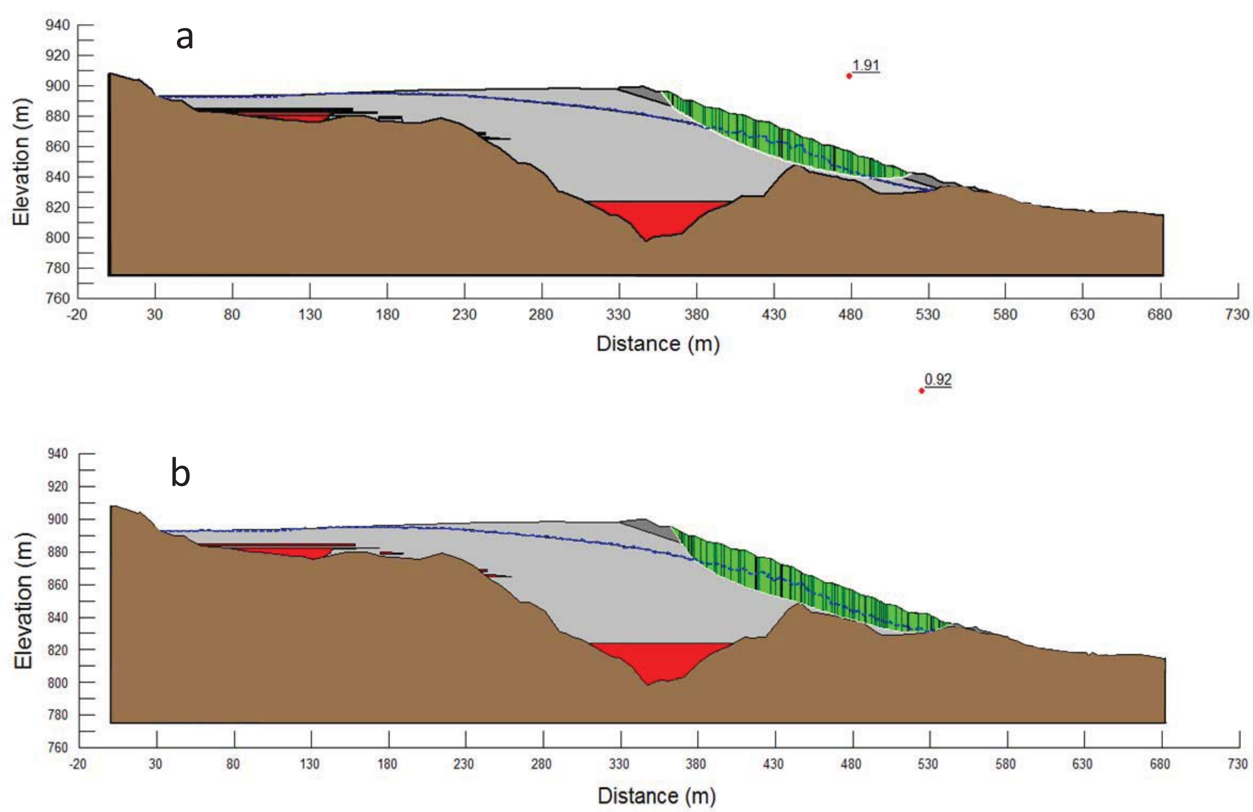

Figure 5-17 Stability analyses at right abutment Section AA; (a) effective stress (ESA); (b) undrained strength (USA)

Figure 5-17 shows FS = 1.91 for ESA conditions and FS = 0.92 for USA. Hence, with a USA factor of safety less than 1.0, rapid failure and associated flowsliding should have initiated at the right abutment if undrained conditions had been operative. The fact that failure did not initiate there means that undrained strength was not fully mobilized and that drained conditions represented by the ESA prevailed at the right abutment. It is equally apparent that the rate of embankment rise at the right abutment was not itself sufficient to mobilize undrained strength in the sand tailings.

5.4 Flowslide Occurrence at the Left Abutment

The conditions discussed in this section make it possible to answer the question of why failure initiated at the left abutment rather than at the right. Two main factors were operative. First, compared to the right abutment, the left abutment had higher and more adverse piezometric conditions. But most importantly, the embankment slope was underlain by slimes at the left abutment causing undrained strength to be mobilized, conditions that did not exist at the right abutment. Undrained shearing and subsequent reduction in undrained strength—the phenomenon of static liquefaction—resulted in the left abutment flowslide. The triggering mechanism for static liquefaction, and the role of the slimes in producing this mechanism, are explained in the following section.

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6 WHY DID THE FLOWSLIDE OCCUR WHEN IT OCCURRED?

6.1 Triggering Mechanisms

Section 4 of this report outlined the conditions required for liquefaction flowsliding. These are: the presence of loose, contractant tailings; the existence of saturated conditions; and rapid failure producing undrained conditions with accompanying reduction in undrained strength. The Fundão flowslide occurred because all of these necessary conditions were present.

Section 5 considered why the flowslide occurred at the left abutment. It was shown that softer, more compressible slimes were deposited in areas intended to be exclusively sand beach, and that the setback of the dam alignment resulted in these materials being present beneath the embankment slope as it was further raised upstream. By contrast to the left abutment, undrained strength was not mobilized at the right abutment, which was not underlain by slimes, and neither were high rates of construction sufficient to induce liquefaction there. The reason that the flowslide occurred at the left abutment is that the presence of slimes-enriched tailings inhibited drainage, enhanced saturation, and promoted undrained shearing.

This section considers what caused liquefaction flowsliding to occur as it did on November 5, 2015. As part of its assessment, the Panel noted that the failure did not occur earlier in the left abutment construction sequence when conditions such as those in the August, 2014 cracking incident were manifested, but that it did occur shortly after the earthquake sequence earlier that day. The timing of the failure event, and the operative conditions at this and other times, goes to the question of liquefaction triggering. Clearly, the softer slimes at the left abutment—and in particular how they responded to increased stress during dam raising—must play a prominent role in any theory of causation.

Drawing on their own experiences and those of others, Martin and McRoberts (1999) have emphasized the need for a physical trigger to initiate rapid shearing, and they catalog numerous potential triggers:

1. Oversteepening at the toe due to erosion, localized initially-drained slumps and construction activities such as excavation.
2. Loading due to rapid rate of impoundment raising, steepening at the crest, and construction activities at the crest.
3. Changes in pore pressure due to increased pond levels, accelerated rates of construction, movements, and other processes.
4. Overtopping due to severe storms, failure of diversion facilities, seismic deformation resulting in loss of freeboard.
5. Vibrational loading due to earthquakes, construction traffic, blasting.

These and other physical trigger mechanisms unique to the Fundão Dam have been considered in Appendix K. It is evident from the above that when contractant tailings are present in the structural portion of a tailings dam, the evaluation of liquefaction triggering is a formidable task.

Of particular interest was static liquefaction initiated by a rise in phreatic surface alone. In the early deliberations of the Panel, this ranked highly as a potential cause. As discussed in Appendix K, the levelling-off of piezometric pressures in the months prior to the failure provides evidence against it. Cyclic liquefaction also received attention, but had it been the sole triggering mechanism, the right abutment would have failed before the left, as discussed in Appendix K.

As summarized on Figure 3-1, the surviving candidate liquefaction triggering mechanisms are static load increases generated directly by either undrained shearing of the slimes or by deformations at the base of the sand leading to collapse. Either trigger mechanism might be augmented by earthquake effects if shown to be consequential. While their end result is similar, their stress paths differ. In the case of undrained shearing, the question arises whether the load due to embankment construction, coupled with the deformation of the underlying slimes-enriched material, can directly induce pore pressures in the loose sand sequence that would lead to undrained failure.

The shearing mechanism is based on undrained shear in the underlying slimes material together with mobilized frictional resistance in the overlying sands. Liquefaction in the overlying sands would be induced by uncontrolled deformation of a sliding mechanism with a factor of safety of unity. An example of a tailings dam that exhibited this failure mechanism is the Los Frailes dam in Spain. In a later portion of this section and in Appendix I, the relationship between undrained strength in the underlying slimes and factor of safety will be presented, as will the sliding developed in the slimes associated with the factors of safety. It will be shown that sliding in the slimes would induce failure associated with a deformation mechanism prior to the initiation of shear failure. In addition, the shear failure mechanism as presently analyzed does not take into account additional three dimensional resistance, which will be substantial. Moreover, shear failure mechanisms are often accompanied by the development of a down-drop block, or graben, at the initiation of the movement, and eyewitness reports do not provide any evidence of such a feature. For both the analytical results to be discussed and the additional items mentioned above, the Panel favors the deformation mechanism as the basis for initiating the failure when it occurred.

The alternative deformation-related trigger mechanism is termed here *lateral extrusion*, with reference to horizontal spreading of the softer slimes due to loading that induces a corollary elongation effect in the overlying sands. The mechanism of lateral extrusion is somewhat more indirect. It asks whether stress changes in the sand above the slimes-enriched layer, as it is undergoing deformation, result in a stress path that leads to collapse and static liquefaction.

Although not included on the list of Martin and McRoberts (1999), lateral extrusion as a static liquefaction trigger mechanism is not new or without precedent. It has been identified by Jefferies and Been (2016) in their discussion of the static liquefaction of the Nerlerk berm, where they state that:

> "The dangerous nature of declining mean-stress paths in terms of liquefaction behavior, caused by basal extrusion, was not understood in 1983."

Much of the subsequent content of this section is devoted to the lateral (basal) extrusion mechanism, incorporating developments much more recent than those referred to above.

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The case of static liquefaction of sands associated with the 1938 failure of the Fort Peck Dam has some similarity to the Nerlerk case in that it has been interpreted to have been caused by shear failure of a weak shale foundation. The difference resides in the basal straining mechanism but the net result in creating stress changes in the saturated sand above is similar.

An important case of static liquefaction occurred at the Germano Complex itself in 2005. As shown in Figure 6-1, a low dike being raised over interlayered and intermixed slimes in the Baia 4 area experienced a sudden, high-mobility failure that moved rapidly over a distance of 80 m. The Baia 4 failure was attributed to liquefaction.

Figure 6-1 2005 Baia 4 static liquefaction failure

The Baia 4 failure provided the basis for determining parameters for slimes-rich layers used in modeling of the Fundão failure. Specifically, parameters relating to the peak undrained shear strength of these layers and the reduction in strength at large deformations were derived by back calculation from pre-failure and post-failure conditions. In these respects, the Baia 4 failure provided an important link between the theoretical studies conducted by the Panel and actual experience at the Germano Complex itself (see Appendix C for details).

Numerical simulations were also grounded in other field experience from the Germano Complex. As explained in Appendix I, field loading trials in 2008 and 2013 provided deformation response and consolidation properties for the slimes-rich layers.

6.2 Loading Conditions

As discussed in Section 4, the ability for rapid loading to result in rapid failure, and hence liquefaction of loose saturated sands, is well understood. However, this is not the case with slower loading.

Sasitharan et al. (1993) demonstrated that a loose granular deposit can collapse as a result of slow loading, as well as during rapid loading, mobilizing a resistance that is much less than the ultimate frictional resistance.

Skopek et al. (1994) demonstrated the mechanics of collapse by following the loading paths utilized above with dry sand and found a sudden volume decrease at essentially the same stress condition, consistent with the data noted above. These two sets of experiments demonstrate the value of the collapse testing to find a yield surface separating collapsing from non-collapsing states in loose, contractive soils like tailings.

Testing is also of value in understanding the role of cyclic loading from earthquakes leading to liquefaction. This is illustrated on Figure 6-2.

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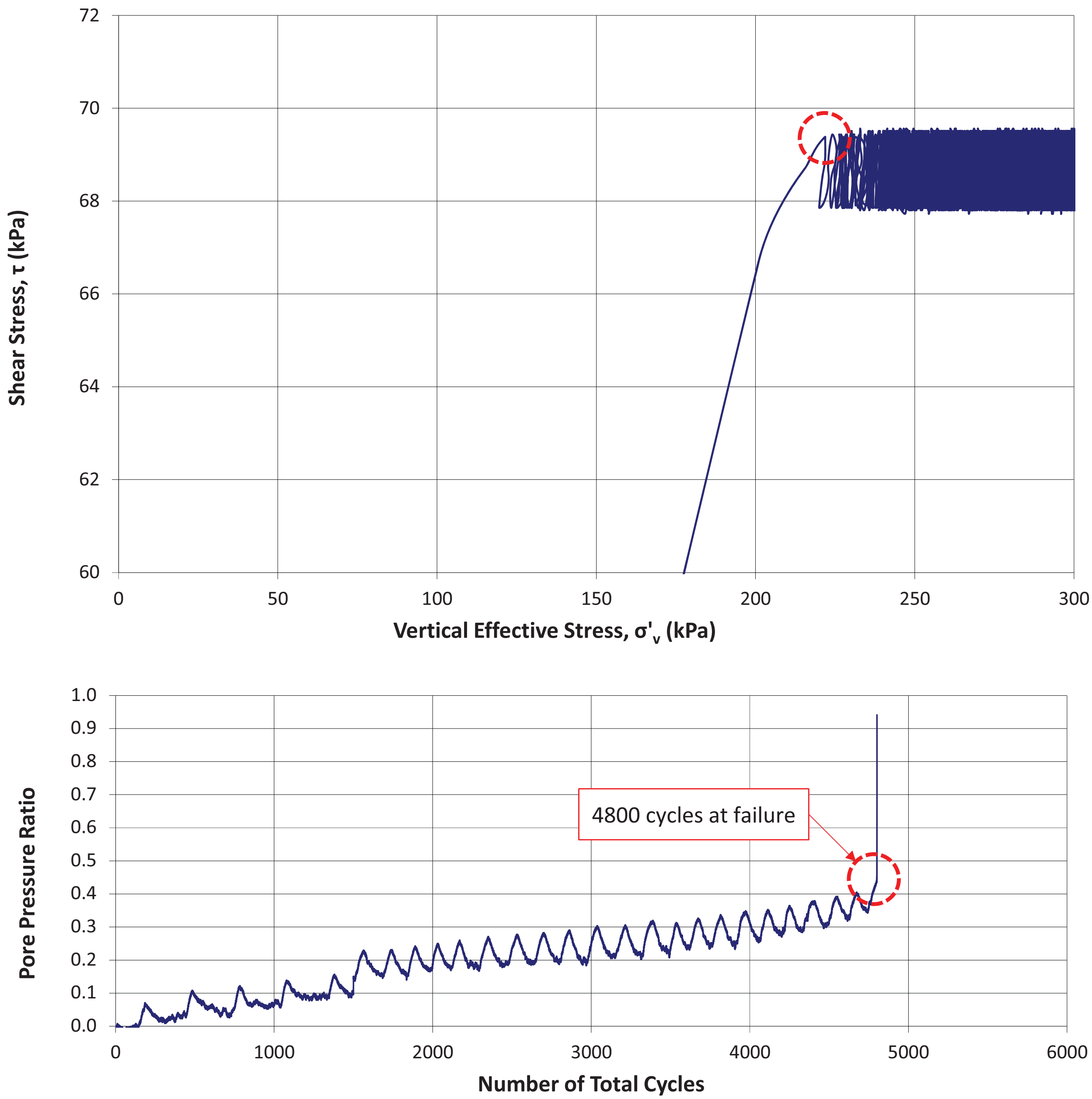

Figure 6-2 Stress path during cyclic loading

During cyclic loading shear stresses vary with time, and this can induce an increase in pore pressure resulting in a reduction of p’. As shown, the stress path migrates to the yield surface and, upon intersection, liquefaction under the applied static stresses results. The sensitivity to cyclic loading depends upon the magnitude of the cyclic shear stress, the duration of dynamic loading, the existing static stresses, and the state of the tailing sands. This topic is discussed in more detail in Section 6.5, where cyclic tests representing the specific earthquake loading for the Fundão Dam will be presented.

In addition to the stresses applied, the loading due to embankment raising must also be considered. This was based on actual survey data, as illustrated in Appendix B. As previously shown on Figure 5-12, the rate of rise at the left abutment reached a value of 2.9 m/month. There is no evidence that this rate had a material effect on the sands, but it could certainly induce an increment of undrained loading on the underlying slimes.

6.3 Ground Conditions

The assessment of ground conditions that influence the formulation of the trigger mechanisms relies on piezometric data prior to the failure and CPT profiles to determine the contractive/dilative behavior of the deposit in the vicinity of the left abutment.

With respect to the piezometric data, Appendix E contains the plots of piezometers in the vicinity of the left abutment up to the time of the failure. Seepage simulation, summarized in Appendix G, extrapolates this data and its trends. The significance of this data in assessing potential trigger mechanisms is also discussed in Appendix K of this report.

With respect to the CPT profiles, the Panel has utilized the Robertson (2010) procedure previously shown on Figure 4-5 for evaluating the contractant/dilatant behavior of the deposits, primarily because it incorporates liquefaction failures of tailings dams and other deposits for immediate comparison. All CPT interpretations are available in Appendix C.

There have been three campaigns of CPT testing in the vicinity of the left abutment where failure was initiated. They also have a bearing on the slumps that developed in 2014; see Section 2 of the Report:

1. April, 2014;
2. September, 2014 – March, 2015; and
3. June, 2015.

The first campaign was limited in scope and reliability but does provide some information in the vicinity of the large-scale cracking that developed in August, 2014. This CPT data does not indicate contractive behavior, but rather dilatant or close to the contractant/dilatant boundary. This is consistent with the absence of any significant mobility of the affected material and the observation that none of the small slumps that preceded it in 2013 propagated by undrained retrogression. There is some indication of densified layering within the profiles, suggesting that the mass of tailings adjacent to the slope may have benefitted from densification associated with construction traffic on the beach.

The second campaign from September, 2014 to March, 2015 has the most data in the region of interest. CPTs F-01 to F-05, which explore from El. 854 m to El. 889 m, all reveal contractant characteristics of beach material, consistent with the existence of potential collapse behavior. Relevant data has previously been summarized on Figure 4-4.

The third campaign, conducted in June, 2015, provided less insight into conditions at the left abutment because it covered a substantial area, even outside of Fundão. Sounding FUND-06 encountered dilatant sand over an isolated loose layer between El. 862 m and El. 864 m. Sounding FUND-07 encountered dilatant sand from El. 895 m to El. 886 m, followed by soft phyllite to a depth of El. 865 m.

6.4 The Lateral Extrusion Mechanism

6.4.1 Detailed Description

The lateral extrusion mechanism is predicated on the presence of saturated, loose sands overlying soft slimes, with confinement of the slimes that varies according to the constructed profile.

As the structure increases in height, the slimes are loaded vertically but tend to extrude or spread laterally, rather like squeezing toothpaste from a tube. In doing so, the overlying sands tend to move with the slimes but lack ductility. As a result, stress changes arise that tend to reduce lateral confinement of the sands. This induces collapse of saturated sand or development of cracks in unsaturated material.

This mechanism, without liquefaction, is well known to designers of embankments on soft clays. Under these conditions the lateral deformation of the foundation often results in vertical tensile cracking of the overlying embankment fill. At failure, the shear strength of the fill cannot be relied upon because of the absence of shear resistance along the open cracks that it sustains.

6.4.2 Extrusion and Collapse of Saturated Loose Sand

The Panel has experimentally demonstrated the lateral extrusion mechanism leading to collapse by conducting drained triaxial compression tests that adopt a specially-designed stress path. Designated *extrusion collapse* tests, they simulate the reduction in horizontal stress in the sand due to slimes extrusion while keeping the vertical stresses constant. Details of the test procedures and data obtained are provided in Appendix D, which includes the results from the tests that were performed. The results of two tests are shown on Figure 6-3, one on a contractive specimen with an initial state parameter $\Psi = +0.01$ and another with $\Psi = +0.04$. These values are on the contractive side of CPT data previously shown on Figure 4-4 and as such tend to bracket the characteristic state in the field. In both cases, as the shear stress along this loading path approaches the ultimate friction line, collapse occurs in an abrupt and sudden manner after only small deformations in the sand. This testing provides both a qualitative demonstration and a quantitative reference for collapse of Fundão sand associated with the extrusion mechanism.

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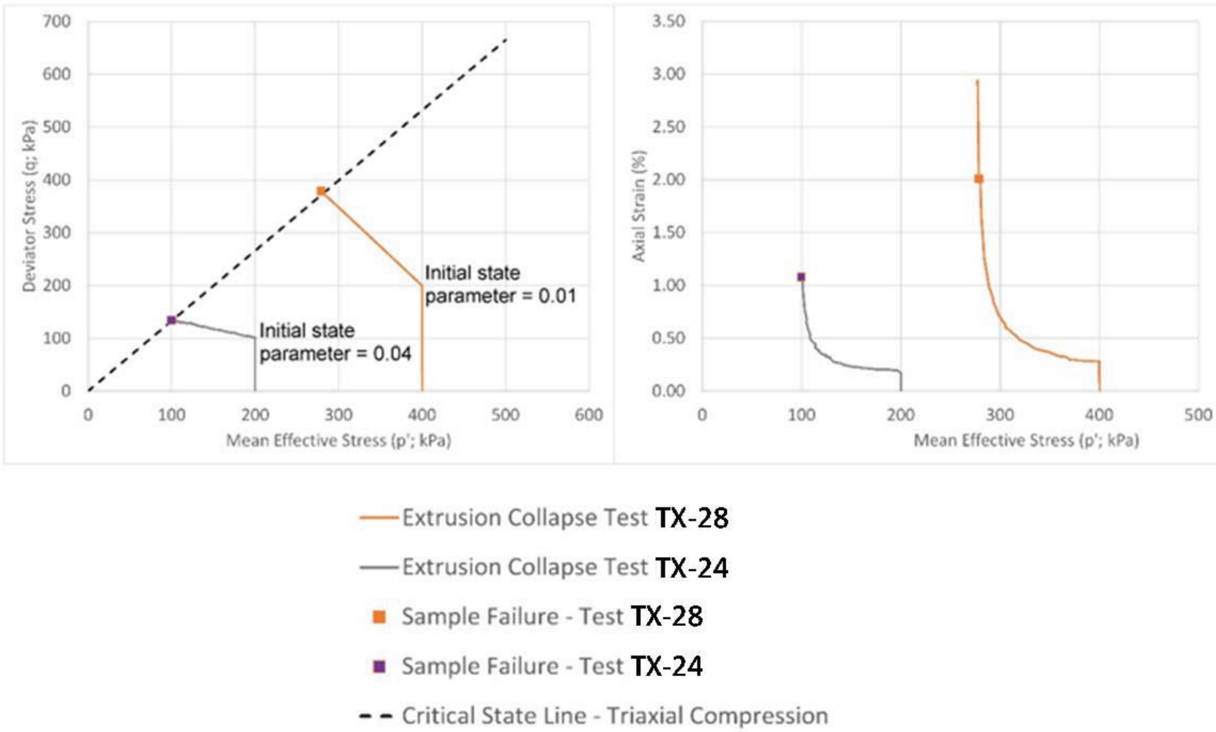

Figure 6-3 Extrusion collapse tests on Fundão sand

6.4.3 Numerical Simulation - Formulation

In order to analyze the lateral extrusion mechanism and resulting collapse, the Panel has undertaken numerical simulation of the construction of the Fundão Dam left abutment section. This analysis follows the history of construction, the evolution of piezometric pressures, the deformation within the slimes and sands, and the spatial variation of the state parameter. An important output from the analysis will be the demonstration of stress paths to failure comparable to those utilized in Section 6.4.2 for quantifying collapse behavior.

The cross-section adopted for the analysis is based on Section 01, at the left abutment provided in Appendix B. Materials within the cross-section have been grouped into the following material types:

1. Bedrock: All materials below the "stripped ground" survey were assigned to this material type;
2. Uncompacted tailings sand not intermixed or interbedded with slimes;
3. Slimes/sand deposits in varying proportions, designated as:
   - predominantly slimes;
   - mixed sand and slimes;
   - interbedded slimes; or
   - isolated slimes; and
4. Compacted sand.

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The embankment configuration was modeled at four-month time intervals throughout the majority of the construction history, starting at the end of 2011. For the final six months (June to November, 2015), this time interval was reduced to monthly in order to gain additional resolution of model response close to the time of failure. Details of the modeling process and its formulation are presented in Appendix I.

The geotechnical properties for each of the materials listed above constitute a fundamental input to the modeling. Formulations of increasing complexity were adopted in an iterative manner to provide a check on model performance. This gave confidence in the results from analyses based on the most complex formulation for loose sand behavior, the critical state model NorSand presented by Jefferies and Been (2016). Parameter sensitivity analyses were completed for the critical state model to assess variations of the influence of the strength and continuity of the slimes layer.

Elastic properties for the sand were based on shear wave velocity measurement in Appendix C converted to an approximate large strain modulus. The elastic properties for the slimes were based on one-dimensional consolidation test data calibrated to a 2008 field loading trial by Samarco described in Appendix F.

The shear strength for beached sand was set at a frictional angle of $\phi' = 33°$, based on tests conducted by the Panel in Appendix D. The compacted tailings sand was modelled with a friction angle of $\phi' = 35°$ and 5 kPa cohesion, in accordance with the values used by others during designs.

The slimes were given a peak shear strength of $\phi_p = 12.4°$, equivalent to an undrained strength ratio of 0.22. This reduced linearly to one-third of the initial value at a plastic strain of 20%, reflecting a modest sensitivity. Support for this formulation is provided in Appendix C from back-calculation of the Baia 4 failure described in Section 6.1.

Critical state parameters assigned to the uncompacted tailings sand were derived from triaxial compression laboratory tests provided in Appendix D. One parameter needed for the critical state formulation was derived from modeling single-element response (equivalent to a laboratory test) as discussed in detail in Appendix I. In addition, it was necessary to declare an initial state parameter to seed the analysis. Following recommendations of Jefferies and Been (2016) and utilizing CPT data from the 2015 field campaign, this seed value was set at $\Psi = -0.02$.

It is also necessary to characterize the various sand-slimes mixtures listed above. Here, no direct experimental information is available, hence judgment is needed to establish both elastic and strength properties. Both elastic and strength properties of the slimes described above were blended with those of the sand in accordance with estimated proportions of those materials within the cross-sections. "Predominantly Slimes" were treated as pure slimes and "Isolated Slimes" were considered as pure sands. "Mixed Sand and Slimes" were a 50:50 mixture and "Interbedded Slimes" were taken to be 80% sand. The resulting properties are summarized in Appendix I.

The formulation of the behavior of the sand/slimes mixtures and their relative proportions is the greatest source of uncertainty in the analysis. As a result, sensitivity analyses have been conducted to explore how variations in assumed sand/slimes behavior influence the model results.

One final element in the formulation of the analysis is the treatment of the pore-water pressures. The pore-water pressures were assigned by setting the phreatic surface based on the integration of piezometric response provided by the hydrogeologic model summarized in Appendix G. As such, no stress-induced pore pressures are considered.

As shown in Appendix F, the slimes appear to fully consolidate on average over the loading history from 2011 to failure. However, in the model it is assumed that increments of loading generate an undrained response. The pore pressures developed are assumed to dissipate prior to the next load increment and do not accumulate over time.

6.4.4 Numerical Simulation - Results

An important check on any complex numerical model is to replicate the experimental information that constitutes the building blocks of the comprehensive constitutive relationship needed to undertake more complex analysis. Figure 6-4 and Figure 6-5 show the results of a simulation of a drained triaxial compression test and an undrained stress-controlled triaxial compression test. The latter follows a stress path simulating the effect of the extrusion mechanism in the slimes on the overlying sand developed in Section 6.4.2 above. The correspondence between numerical simulation and experiment is encouraging. The model strength result is about 3% less than the experimental value and it will be used as a reference to assess proximity of the simulation to collapse.

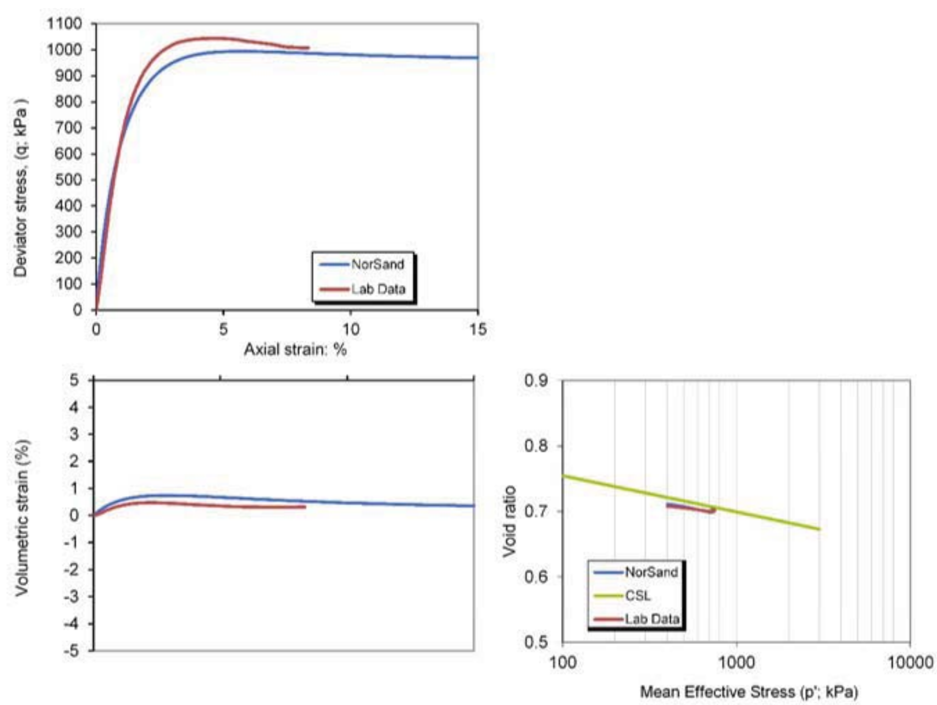

Figure 6-4 Simulated drained triaxial compression test (Test ID TX-12)

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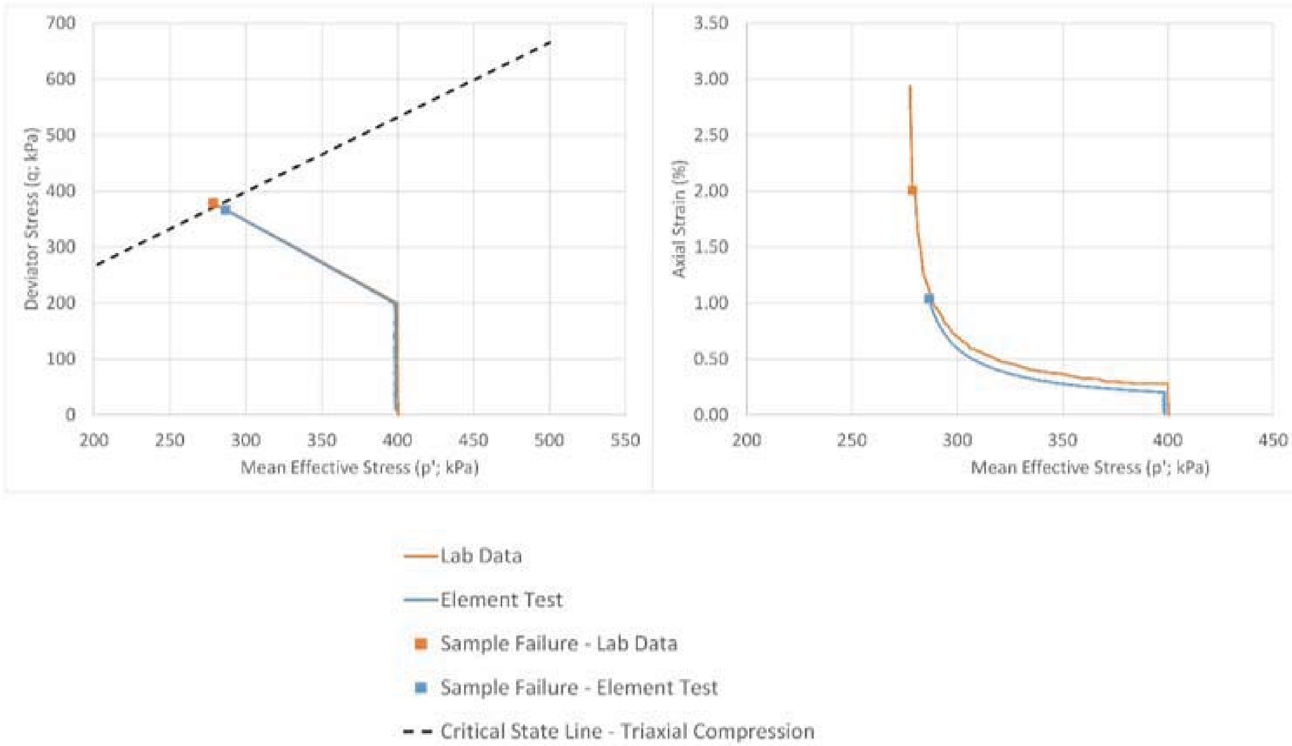

Figure 6-5 Simulated extrusion collapse test (Test ID TX-28)

Appendix I presents the results and general conclusions from a variety of simulations intended to explore the sensitivity to assumptions with respect to the distribution of slimes-enriched deposits and to their assumed geotechnical properties. In the view of the Panel, the case that best represents the evolution of collapse in the saturated loose sands overlying the slimes rich deposits is presented on Figure 6-6.

The Mobilized Instability Ratio (MIR) is a criterion for the triggering of collapse. It is defined as the ratio of the deviator stress and mean effective stress to the ratio at the onset of collapse. The color zonation represents the MIR related to the collapse strength determined from laboratory tests. The maximum value computed is 80%. Numerical convergence limitations inhibit the modeling from progressing further. However, the information available from the simulation provides compelling support for the hypothesis that collapse was triggered by lateral extrusion of the slimes-rich deposits.

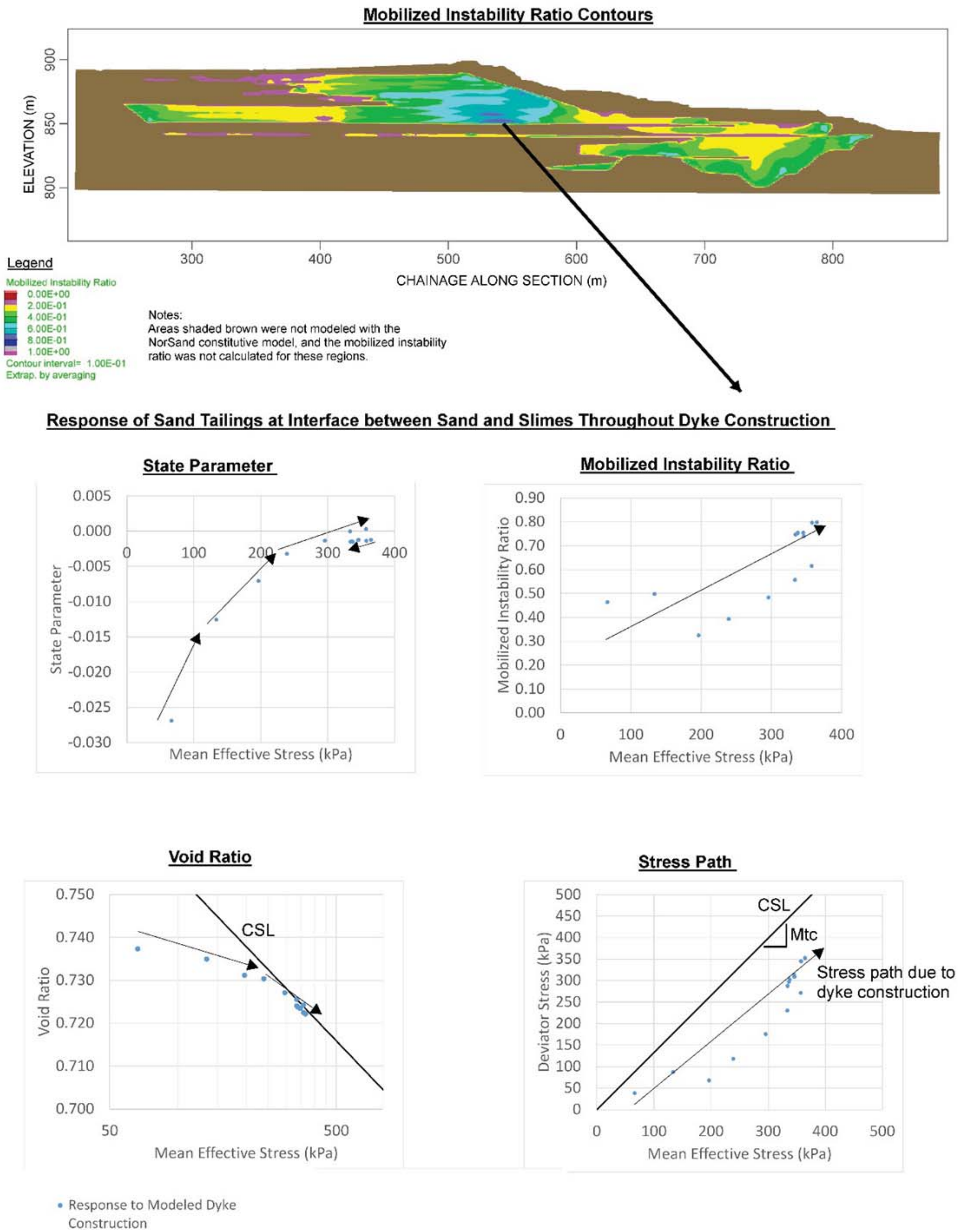

Figure 6-6 Mobilized Instability Ratio

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Figure 6-6 also plots the stress path calculated throughout construction of the Fundão Dam. Operation of the lateral extrusion mechanism is cumulative during construction as reflected by the results plotted. The stress path has been calculated at the base of the sand which is the location where collapse would be initiated. The calculation indicates that 80% of the available collapse resistance has been mobilized with the strength as prescribed in the analysis and determined by laboratory tests. Numerical instability, from a computational perspective, precluded advancing the calculations further.

Figure 6-7 provides a comparison of laboratory data from Figure 6-5 with the simulated field stress path on Figure 6-6. It shows that the field stress path displays similarity to the controlled laboratory stress path and is migrating towards the ultimate strength line. As noted above, numerical convergence limitations preclude completing the analysis.

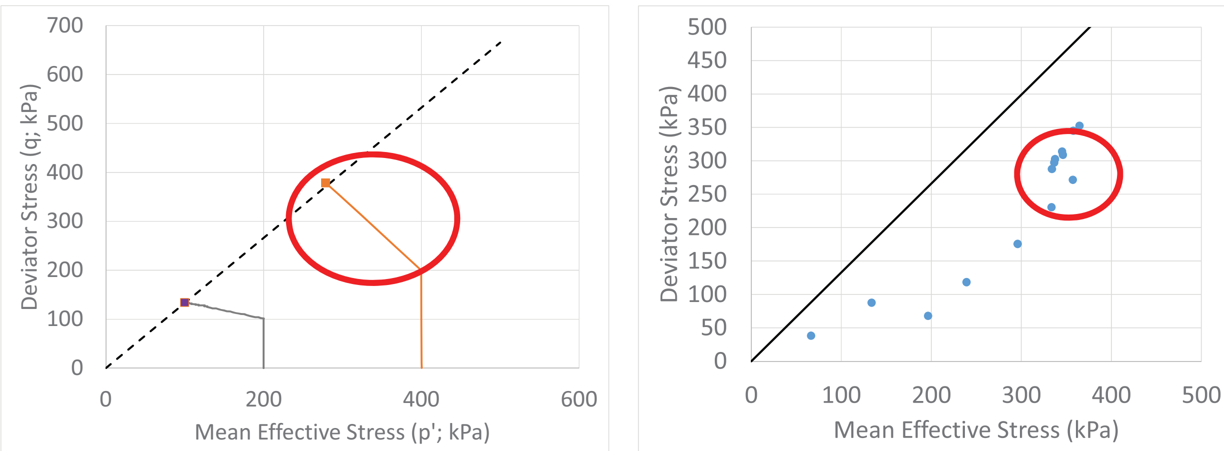

Figure 6-7 Comparison of laboratory and simulated field stress path

Figure 6-8 plots horizontal deformations along the slimes/sand interface at various stages of construction of the Fundão Dam. It illustrates that the largest lateral movements occur beneath the slope and downstream of the crest. This implies compressive straining in the downstream direction and extension straining in the upstream direction. Extension strains result in a reduction of horizontal confinement consistent with the lateral extrusion hypothesis.

It is also of interest to note that the maximum horizontal displacements beneath the lower part of the slope coincide with eyewitness reports of slope movement having initiated on the lower benches, as described in Section 2.7.

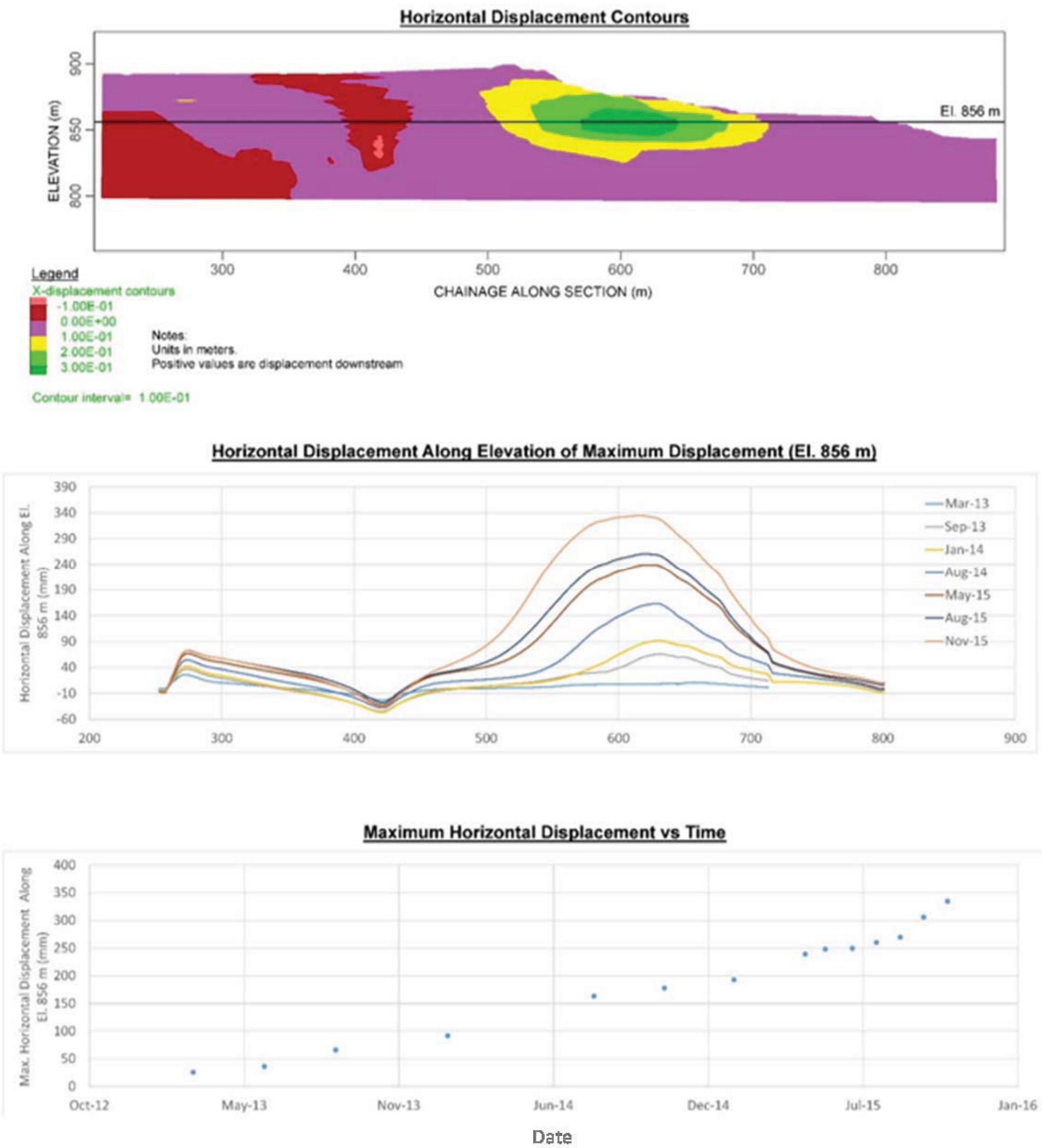

Figure 6-8 Horizontal displacements at sand/slimes interface

6.5 Displacements to Trigger Liquefaction by Lateral Extrusion

In order to determine the sliding deformation that would overcome the limitations of numerical convergence issues and meet a MIR of unity, the numerical analysis has departed from following the loading history and now imposes a specified slip to calculate the MIR response. As shown in Appendix I, a sliding displacement of 600 mm is required for an MIR of unity. By extrapolation, from

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past values relating MIR and mobilized shear strength it is found that the sliding displacement of 600 mm would be calculated if the undrained strength ratio were equal to 0.14. This value is consistent with the sensitivity of the slimes.

6.6 Comparison Between Shearing Mechanism and Lateral Extrusion

In order to use this critical sliding displacement to evaluate the relative likelihood of the lateral extrusion mechanism triggering liquefaction versus other mechanisms, it is necessary to compare the 600 mm value with slip associated with a shearing mechanism that could develop due to the mobilization of low strengths in the slimes-rich layers. The shearing mechanism is a sequence involving undrained yielding of the slimes-rich layer leading to a general shear failure throughout the dam slope, which in turn results in an acceleration of displacements that triggers liquefaction. In order to evaluate which of these mechanisms was the more probable liquefaction trigger, the Mohr-Coulomb model discussed in Section 6.4.3 was used to estimate the magnitude of deformations that would develop at the onset of general shear failure due to yielding in the slimes-rich layer. Details are presented in Appendix I.

The pattern of displacements resulting in November, 2015 if a factor of safety of unity was approached is shown on Figure 6-9. The pattern of displacements is similar to that shown previously for the NorSand model analyses.

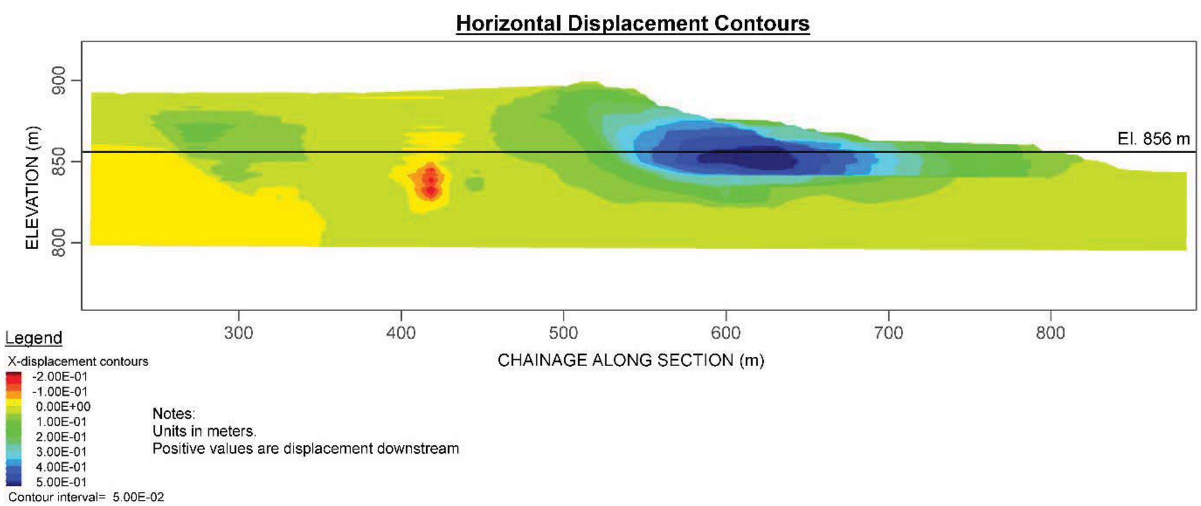

Figure 6-9 Horizontal displacements resulting from Mohr-Coulomb analysis approaching a factor of safety of unity

The deformation model used for failure analysis is equivalent to limit equilibrium analysis and hence provides a linkage between sliding displacement at failure and factor of safety. As discussed earlier in this section, a factor of safety of unity represents a trigger for the onset of liquefaction. The deformations associated with the onset of liquefaction with the shearing mechanism are much greater than those associated with the lateral extrusion mechanism. Therefore, liquefaction would be initiated by lateral extrusion prior to the development of shear failure.

The Panel regards the results from the numerical simulation as providing compelling support for the lateral extrusion mechanism accounting for the occurrence of the flowslide on November 5, 2015.

6.7 The Role of Earthquakes

6.7.1 Earthquake Loads

The Panel has relied on the Atkinson Report (2016) for evaluating the seismic history at the damsite and for recommending ground motions to be considered in response analyses (Atkinson 2016). The seismology report summarizes the regional seismicity and the instrumental records that were obtained from the earthquakes that occurred just prior to the collapse of the dam. It concludes that the site experienced natural earthquakes, as summarized in Table 2-1, with a Moment Magnitude, M_w, of up to 2.5 and epicenters close to the dam. As reviewed in Appendix K, earthquake loading from such small shocks would not usually be considered consequential to structures with robust design and operation. However, as discussed in detail above, the dam was in a very fragile state at the time of the earthquakes and the question arises whether the earthquakes hastened its collapse. Hence, a more detailed evaluation was warranted.

Understandably, there is considerable uncertainty in the determination of ground motions, and the Panel requested that the seismologists provide a range of ground motions and associated estimates of likelihood. These records form the basis of the dynamic response analysis needed to calculate the magnitude of stresses in the dam induced by the earthquake and the duration of earthquake loading. Both the median and 84th percentile (mean plus one standard deviation) ground motions were used for the dynamic response analyses.

6.7.2 Dynamic Response Analysis

Details of the dynamic response analyses are presented in Appendix J, and the soil properties used in these analyses are summarized in Appendices C and D.

Prior to calculating the dynamic response of the dam to the prescribed earthquake loading, the dynamic response was calculated at the site where the earthquake was experienced and where the subjective intensity characterization was first assembled. The intent of the calibration was to confirm that, within the bounds of the uncertainty associated with these analyses, the calculated ground motions were reasonable. Calculations were conducted by means of an industry standard method called SHAKE. The seismological advisors concurred that the calculated response was acceptable.

The recommended median and 84th percentile ground motions were then used to calculate the cyclic stresses and number of significant cycles to be considered in assessing the dynamic response of the dam. These ground motions are used to calculate both potential pore pressure development in saturated sand above the slimes as well as potential displacements in the slimes-rich deposits.

Cyclic Loading and Pore Pressure Response

It was the intent to apply the cyclic loading discussed above to a test specimen of sand on the brink of collapse, having been brought to that state by reducing horizontal stresses following the path associated with the lateral extrusion mechanism. However, it was not practical to apply the small

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stresses calculated, and significantly larger stresses were applied during testing. Figure 6-2 illustrates the type of response that would indicate that the imposed earthquakes could have a significant effect on failure of the dam. In specific tests undertaken on the fragile test specimen, many more cycles (>1000) were applied than the 4-5 indicated by the calculations. Details of the testing are summarized in Appendix D. Collapse occurred only after more than 1200 cycles at stresses significantly larger than indicated by the analysis to have been produced by the earthquakes and no specific excess pore pressures were generated.

The Panel concludes that no cyclic induced pore pressures resulted from the assumed earthquake loading.

Cyclic Loading and Sliding in Slimes

Another potential result of the imposed earthquake loading is to induce deformations in the slimes-rich deposits as a result of the cyclic stresses discussed above. These deformations are calculated by adopting the earthquake motions computed at the top of the slimes and imposing them directly into analysis to calculate the seismic induced slip using a classical method entitled Newmark-type analysis and using a well-accepted computer program called SLAMMER. Details of these calculations are presented in Appendix J.

Reflecting the uncertainty in the prescribed ground motions, six time histories were selected from those recommended by Atkinson (2016). Those selected reflected the upper-bound of the range evaluated in the seismic study. The average calculated displacement was 5 mm. This can be compared with the rate of displacement calculated by the deformation analyses prior to failure. Estimates of rate of displacement from both NorSand and Mohr-Coulomb analyses indicate rates of approximately 1 mm per day. Hence, the displacements calculated from the SLAMMER analyses are of limited significance when compared with the rates of displacements associated with static loading alone. Nevertheless, given the proximity of the dam to collapse due to prior construction loading, this likely accelerated the failure process that was already well-advanced

6.8 Timing of the Failure

The introductory portions of this section posed the following three questions, which are answered below.

Why did flowsliding not occur on August, 2014 cracking incident?

The August, 2014 cracking incident did not display the mobility associated with flowsliding indicating that liquefaction did not occur. Figure 4-3 previously explained in principle how increased loading can cause a formerly dilatant material to become contractant. This effect is displayed in the NorSand model of conditions prevailing on or about August of 2014. As shown on Figure 6-10 on August, 2014 the sand is on or close to the CSL, which is a boundary between contractant and dilatant behavior. At this point the sand became loose enough to exhibit volume change and associated cracking but was not sufficiently loose to exhibit liquefaction. Also, Section 4.3 explained that a fundamental change in seepage patterns happened on or about the same time. Together these two changes, one in sand behavior and the other in saturation, produced the observed effects.

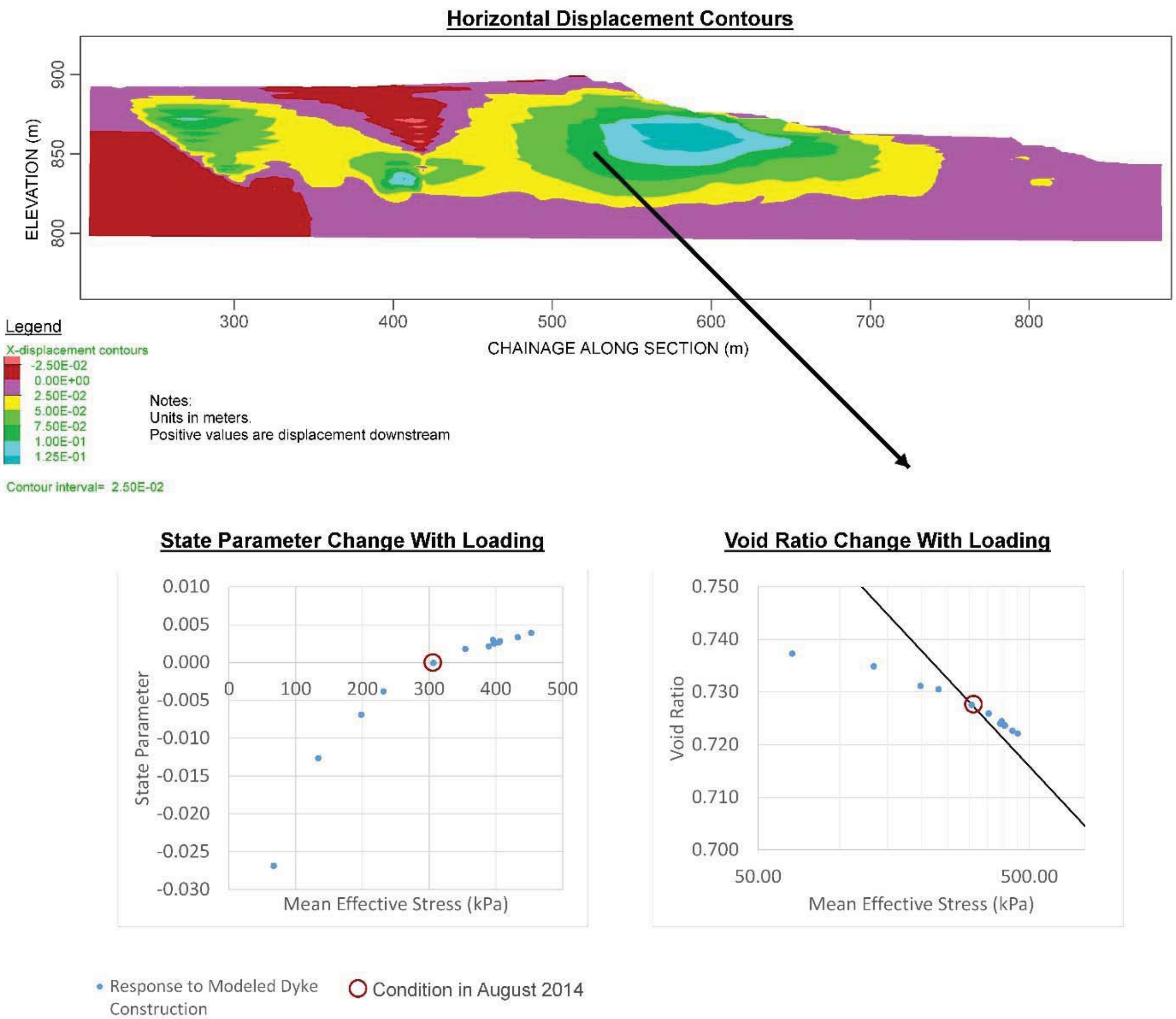

Figure 6-10 Example NorSand model output

Why did flowsliding occur under the conditions that prevailed on November 5, 2015?

The Panel concludes that the flowslide that occurred on November 5, 2015 was instigated by a lateral extrusion mechanism seated in the slimes-rich deposit at depth in the embankment that resulted in a reduction of lateral confinement of the overlying contractant and saturated sand. The extrusion mechanism created sufficient sliding displacement to generate a MIR of unity which is the criterion for triggering collapse.

Why did flowsliding occur following the earthquakes?

The earthquakes were small and would normally not be regarded as consequential for an ordinary dam. The Fundão Dam was subjected to lateral extrusion in the slimes-rich deposits beneath the left abutment and stress readjustment associated with this mechanism was leading it to collapse and liquefaction.

The prescribed earthquake motions have two potential effects on the dam. One is cyclic stresses that induce pore pressures in the sand and the other is cyclic stresses that induce deformations in the slimes-rich deposits. Experiments conducted on samples of sand representative of the stresses prior to collapse did not develop any pore pressure response due to the applied earthquake motions. However, the same earthquake motions applied to the behavior of the slimes-rich deposits indicated sliding displacements in the range of several millimeters. These displacements are of limited significance when compared with the displacements associated with static loading alone. Given the proximity of the dam to collapse due to prior construction loading, these earthquake induced displacements likely accelerated the failure process that was already well-advanced.

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7 CONCLUSIONS

The mandate to the Panel was to conduct an investigation into the cause(s) of the breach of the Fundão Tailings Dam on November 5, 2015. To fulfill this mandate, the Panel was expected to provide its independent and unbiased professional judgement and expertise in determining the immediate cause(s) of the Incident and that this report would identify these immediate cause(s).

The Panel has responded to its mandate by framing three questions with accompanying answers. These questions and a summary of the responses, presented below, identify the immediate causes.

Question 1: Why Did a Flowslide Occur?

The original design concept for the Fundão Dam employed an unsaturated sand zone to support the weak slimes zone. Unsaturated sand is not amenable to liquefaction and hence the original design was robust in this regard. However, difficulties were encountered in executing the design and a modified design was put forward and adopted. As part of this modification, a change in the design concept was also adopted and saturated conditions were permitted to develop in the sand.

The flowslide required three conditions to develop: (1) saturation of the sand; (2) loose uncompacted sand; and (3) a trigger mechanism. Depositing sand tailings by hydraulic means resulted in loose conditions. The growth in the saturated conditions is well-documented. Hence, all the conditions prevailed for liquefaction to develop resulting in a flowslide, provided it was triggered. Triggering is discussed in the response to Question 3.

Question 2: Why Did the Flowslide Occur Where It Occurred?

Eyewitness accounts revealed that the flowslide initiated on the left abutment, where the dam had been set back from its former alignment. Studies of the depositional history associated with the growth of the Fundão Dam revealed that slimes encroached into the area preserved for sand deposition alone. The design incorporated a 200 m zone separating the two deposits but historical information reveals that slimes had encroached into the area on a number of occasions. The presence of slimes introduces a barrier to downward drainage and a zone of potential weakness that might affect stability. Deposition in the area of the right abutment was almost slimes free.

The setback was implemented to accommodate repairs to a deficient conduit at the base of the impoundment as well as the construction of additional horizontal blanket drains to facilitate subsequent dike-raising. This change in geometry resulted in substantial embankment loading over slimes-rich deposits. This distinguishes the left abutment area from the right and accounts for the location of flowslide initiation.

Question 3: Why Did the Flowslide Occur When It Occurred?

The initiation of a flowslide requires not only the presence of saturated contractant tailings but also a trigger mechanism to initiate the process that mobilizes undrained shearing and hence flowsliding. Following an evaluation of potential trigger mechanisms, the Panel concluded that lateral extrusion initiated the failure.

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The lateral extrusion mechanism develops as the dam increases in height, loading the slimes-rich zone vertically which tends to extrude or spread laterally, rather like squeezing toothpaste from a tube. This results in stress changes in the overlying sands which reduce their confinement, leading to collapse.

This mechanism for collapse was modelled by tests in the laboratory and by computational modeling that predicted to an acceptable degree that collapse should have occurred about the time that the dam was raised to the height that was attained on November 5, 2015.

The role of the earthquakes that occurred just prior to collapse was also investigated quantitatively. Calculations with recommended design motions reveal that about 5 mm of displacement may have been induced in the slimes. Given the proximity of the dam to collapse due to prior construction loading, this likely accelerated the failure process that was already well-advanced.

ACKNOWLEDGEMENTS

We would like to express our appreciation to Klohn Crippen Berger, which provided indispensable engineering and organizational assistance. We especially wish to recognize the efforts of Cindy Wang, without whose creative skill and determination in synthesizing reams of information and data our efforts could never have proceeded far. In addition, we wish to recognize the contribution of Joe Quinn not only for his leadership and analytical modeling but also his role in the field and laboratory studies. The contribution of Chris Dickinson and his colleagues resulted in an important synthesis of all seepage related issues.

Most of all, the Panel wishes to acknowledge the sacrifices of the victims, their loved ones, and those left homeless. We can only hope that our work may in some small measure help to prevent such occurrences from ever happening again.

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GEOPHYSICAL RESEARCH LETTERS, VOL. 40, 1402–1408, doi:10.1002/grl.50334, 2013

Probable maximum precipitation and climate change

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Received 13 February 2013; revised 4 March 2013; accepted 7 March 2013; published 12 April 2013.

[1] Probable maximum precipitation (PMP) is the greatest accumulation of precipitation for a given duration meteorologically possible for an area. Climate change effects on PMP are analyzed, in particular, maximization of moisture and persistent upward motion, using both climate model simulations and conceptual models of relevant meteorological systems. Climate model simulations indicate a substantial future increase in mean and maximum water vapor concentrations. For the RCP8.5 scenario, the changes in maximum values for the continental United States are approximately 20%–30% by 2071–2100. The magnitudes of the maximum water vapor changes follow temperature changes with an approximate Clausius-Clapeyron relationship. Model-simulated changes in maximum vertical and horizontal winds are too small to offset water vapor changes. Thus, our conclusion is that the most scientifically sound projection is that PMP values will increase in the future due to higher levels of atmospheric moisture content and consequent higher levels of moisture transport into storms. **Citation:** Kunkel, K. E., T. R. Karl, D. R. Easterling, K. Redmond, J. Young, X. Yin, and P. Hennon (2013), Probable maximum precipitation and climate change, *Geophys. Res. Lett.*, *40*, 1402–1408, doi:10.1002/grl.50334.

1. Introduction

[2] Climate change can be described in terms of the temporal evolution of the full probability density function (pdf) of variables that characterize the state of the atmosphere and the climate system. An important set of these variables have been designated as essential climate variables [*GCOS*, 2009]. Changes in the tails of the pdfs of some of these variables receive particular attention for climate change impacts and risk assessment.

[3] Increases in heavy precipitation events have been documented in many regions of the globe [*IPCC*, 2012] with substantial variations in the spatial distribution of statistically significant trends [*Bonin et al.*, 2011; *Kunkel et al.*, 2013]. Similarly, most areas of the U.S. are projected to see increases through the 21st century [*IPCC*, 2012], including areas that did not have statistically significant trends in the 20th century. Given these observed and projected changes, precipitation-sensitive information and applications would benefit from incorporation of best estimates of future changes, based on observed trends, model projections, or a combination of these.

[4] One informational product used for planning, probable maximum precipitation (PMP), is defined as the greatest accumulation of precipitation for a given duration meteorologically possible for a design watershed or a given storm area at a particular location at a particular time of year [*WMO*, 2009]. A better term for this concept might be potential maximum precipitation (PMP) to avoid any confusion that such an amount is probable, but instead, is potentially possible. A principal application for PMP values is the design of infrastructure for water retention (dams) or routing, where failure would be catastrophic.

[5] PMP values translate into a return period very much longer than the longest return periods traditionally used in applied climatology products, such as the 100 year return period amount (the amount that in a stationary climate has a 1% chance of occurring any given year or on average once in every 100 years). For example, the 24 h, 100 year return period amount for Urbana, IL, is about 175 mm, but the 24 h PMP amount ranges from about 225 mm for a 51,800 km^2 area to over 900 mm for a 26 km^2 area [*Schreiner and Riedel*, 1978].

[6] The long lifetimes of dams and similar structures ensure that they will experience the impacts of future climate change. We contend that in any future assessment of dam safety risk or other infrastructure where failure can lead to catastrophic consequences, ignoring climate change–induced new probabilities of extreme events, is likely to lead to a false sense of security. In this paper, we discuss the factors influencing PMP estimates for a range of time and space scales and whether any statements can be made about future changes in these factors.

2. Estimation of PMP

[7] PMP values are, in principle, most dependent upon atmospheric moisture, transport of moisture into storms, persistent upward motion, and strong winds where orographic uplift is important [*WMO*, 2009; *Trenberth et al.*, 2003]. The general approach, using data and physical judgment, is to estimate the precipitation that would occur if all the relevant factors in a particular place and situation achieved their optimum values simultaneously and remained in place for the specified duration over the basin area. We review the factors below.

a. *Convergence and vertical motion*

[8] In past analyses, estimation of the maximum value of horizontal low-level wind convergence and upward motion

All supporting information may be found in the online version of this article.

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0094-8276/13/10.1002/grl.50334

was not considered to have a robust theoretical basis. Instead, the lengthy and numerous data records of precipitation have been used to identify historical extreme observed storm precipitation (P_{storm}). P_{storm} values serve as indirect measures of maximum low-level moisture convergence and persistent upward motion.

[9] *Schreiner and Riedel* [1978] developed the most recent estimates of PMP values for much of the U.S. east of the Rocky Mountains. They based their analysis on a set of 55 extreme storms occurring at scattered locations during the period 1878–1972 whose precipitation totals constitute the P_{storm} set. These storms are assumed to approximate a maximum for precipitation and vertical motion lasting for a given duration. This set of the most extreme precipitation events was gleaned from the pooled data of the entire observing network representing hundreds of thousands of station-years of data. As such, the implied return period of events of this magnitude for any individual location is much longer than the 95 years over which these events occurred. All of the storms used in this eastern U.S. analysis were warm season events. The western U.S. is different, as most of the most extreme events occur during fall or winter [e.g., *Corrigan et al.*, 1999].

b. *Atmospheric water vapor*

[10] A second component entering into the empirical estimation of PMP is the maximum atmospheric total column water vapor (precipitable water, PW) that is possible for a given location and season. The maximum possible value, PW_{max}, is estimated as the observed maximum historical precipitable water [*Schreiner and Riedel*, 1978].

[11] A U.S. climatology of extreme PW values from 50 years of radiosonde observations (www.crh.noaa.gov/unr/?n=pw), representing a first-order approximation and a distribution of 50-year recurrence interval value, indicates maximum PW values of nearly 75 mm are found in summer for stations near the Gulf Coast. Since the radiosonde network is relatively sparse in space (400 km mean spacing) and time (12 h interval between observations) and the period of record is short in duration (only 50 years), these extreme PW values probably underestimate PW_{max}, even in a stationary climate. Although transient values within the cores of storms may be higher than the atmospheric conditions sampled by the radiosonde network [*Holloway and Neelin*, 2010], such transient values are most likely not representative of the inflow regions of storms.

[12] The observed record of precipitation, extending back to the late 19th century, is considerably longer than the record of radiosonde observations. Thus, the PW estimate for a climatology of maximum observed "storm events," PW_{storm}, has previously been based upon the climatology of maximum surface dewpoint temperatures persisting for a minimum duration of 12 h. Since the atmosphere during torrential rains of several hours duration (such as in tropical cyclones) typically approaches a pseudoadiabatic temperature state, this temperature-humidity profile has been assumed as a limiting extreme for PW_{max} [*Schreiner and Riedel*, 1978]. We believe this is an appropriate equilibrium assumption for linking PW_{max} to PMP. A criticism of this assumption was presented by *Chen and Bradley* [2006], whose analysis of extreme events in the central U.S. indicated a 7% overestimate of PW_{max} using the above criteria. Their conclusion could be a result of surface humidity, upper-level dryness, or storm dynamics peculiar to that geographical region or undersampling by the short-term data record. But any current overestimates would apply equally to future changes and our interest in this study is in changes relative to current values.

[13] Given the durations (6–72 h) for which PMP estimates are made, sustained high moisture flow into storms is necessary. One meteorological type is "open" precipitating systems fed by persistent large-scale winds and oceanic moisture sources far upwind. For example, on the west coast of the U.S., atmospheric rivers of deep tropical moisture from the Pacific Ocean [*Dettinger*, 2011] can extend inland and create intense precipitation over the upwind slopes of the mountain ranges. For the eastern half of the U.S., the major source of moisture is the Atlantic Ocean/Caribbean Sea/Gulf of Mexico.

[14] Some extreme rains over days are associated with "closed" precipitating systems which depend on more localized sources of water. The best example is a stationary tropical cyclone over a coastline next to a warm ocean; such events have produced historic PMP events, reflecting a strong positive feedback system involving surface wind, evaporative moisture supply, and precipitation and condensational heating to sustain the energy of the winds [*Price*, 1981]. However, these isolated storms also experience negative feedbacks limiting the supply of moisture and the lifetime of the system, including subsidence of dry air around the storm periphery, and wind-induced evaporative cooling of the ocean beneath, reducing the oceanic source moisture [*Schade*, 2000].

[15] For shorter time scales and smaller basins, some mesoscale convective systems may be relevant to PMP. These systems are normally propagating and are partially "open" as they traverse humid air masses. Their lifetime is often limited by the negative feedback of subsidence of cooled, dry air to the surface. Such systems may be sustained in a region of weak (or negative) upper-level inertial stability, which encourages the divergent branch of the convective circulation [*Coniglio et al.*, 2010].

c. *Physical synthesis*: *Linking PMP and atmospheric water vapor*

[16] Traditionally, the calculation of PMP assumes a statistical equilibrium relationship between PMP and PW_{max} linking the "storm event" data (i.e., P_{storm}) with estimates of extreme values:

$$PMP_{est}/PW_{max} = P_{storm}/PW_{storm} = N_{cycles} \quad (1)$$

indicating that PMP_{est} increases proportionately to PW_{max} as inferred from dewpoint temperature, for a given storm climatology. N_{cycles} is the number of water replacement cycles for the column during the duration of precipitation, and the underlying assumption is that N_{cycles} is the same for the PMP calculation as for the storm data.Expression (1) can alternatively be expressed as the ratio of time scales

$$N_{cycles} = (\tau_{dur})/(\tau_{repl}) \quad (2)$$

where (τ_{repl}) is the replacement time scale for the water in the column and (τ_{dur}) is duration time over which the total precipitation is accumulated. Since air rises over the depth of the rain system H in time (τ_{repl}), its average vertical velocity is $W = H/(\tau_{repl})$. Hence, we can further rewrite any of the above as

$$W/H = P_{rate}/PW \quad (3)$$

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where $P_{rate} = P/(\tau_{dur})$ is the average precipitation rate (intensity) over the duration (τ_{dur}). Thus, these relationships reflect the underlying assumption in PMP estimation that the average vertical motion for the equilibrium assumption is assumed to be the same for the PMP case as for the storm data. Thus, W, as calculated here, is an "efficiency" parameter representing the estimate of maximum persistent upward vertical motion (W_{max}) consistent with the column water budget in an extreme precipitation event. Over topography with slope S, one expects W to be proportional to the product of upstream wind and S.

[17] For the aforementioned example of the point 24 h PMP at Urbana, IL, the maximum PW at this site is roughly 64 mm. This leads to a value of N_{cycles} of about 15, an approximate replacement time scale of slightly less than 2 h, and an average vertical velocity of about 1.5 m s^{-1}. The values of PMP for basins decrease with increasing basin size because P_{storm} values decrease as the size of the area over which precipitation is averaged increases. For the location of Urbana, the 24 h PMP value for an area of 51,800 km^2 (the largest area estimated by *Schreiner and Riedel* [1978] is about 230 mm, about ¼ the point value, and W is similarly reduced to about 0.4 m s^{-1}.

3. Possible Effects of Climate Change on Extreme Precipitation

[18] The radiative energy imbalance caused by increases in greenhouse gas concentrations is highly likely to continue the increases in ocean heat storage and a rise in sea surface temperatures (SSTs) that have already been observed [*Trenberth et al.*, 2007]. The warming ocean will in turn lead to a rise in evaporation and atmospheric water vapor content, following the Clausius-Clapeyron relationship for saturation water vapor pressure. A probable consequence is the intensification of the hydrologic cycle and PMP over land and ocean. The effect of this intensification on changes in PW_{max} values over land was investigated by analyzing future (2041–2070 and 2071–2100) and control (1971–2000) simulations from the Coupled-Model Intercomparison Project phase 5 (CMIP5) archive. Seven GCM simulations were examined (listed in supplementary online material). The model data were first regridded to a common grid of 2° latitude by 2.5° longitude, comparable to the largest basin sizes for PMP applications. For each grid point, the maximum value over the entire 30 year period of the 12 h persisting PW (PW_{max}) was identified. Finally, a multimodel mean map was produced. The analysis was performed for two representative concentration pathways (RCP), the RCP4.5 and the RCP8.5.

[19] Figure 1 shows the global pattern of maximum PW (top) and its projected percentage changes for 100 years in the future (middle). The analysis reveals projected increases across all grid cells, indicating general global moistening of the atmosphere. The overall global patterns of contemporary PW_{max} (top) and the absolute magnitudes of the future differences (supplementary online material) are very similar: moisture increases are a maximum in regions where they are currently large. These changes in PW content represent changes in the pattern of latent energy content and are focused in the tropical belt of latitudes, particularly the oceanic ITCZs, western Pacific warm pool, and adjacent Asian monsoon regions.

[20] The patterns of fractional percentage changes (middle) are quite different from that of absolute changes, indicating somewhat larger changes toward the poles. Over large parts of the Northern Hemisphere, the percentage increases are in the range of 20%–30% by 2071–2100. At high latitudes and over some land areas, particularly Eurasia, the increases are more than 30% by the end of the 21st century. For North America and surrounding ocean areas, there are increases of 20%–30% by 2071–2100 with the greatest increases over the western U.S. (where the actual PW_{max} values remain relatively low).

[21] The results for 2041–2070 and for the RCP4.5 simulations (supplementary online material) indicate increases for 2041–2070 of roughly half of the 2071–2100 results and for RCP4.5 about half of the results of the RCP8.5 simulations, in approximate correspondence to the difference in greenhouse radiative forcing. The fractional changes in mean water vapor concentrations (not shown) are larger, but only by a small amount, than the changes in the maximum values shown in Figure 1. The maximum values of PW typically occur in July or August in most of the contiguous U.S., except along the west coast, where a fall (either September or October) maximum is simulated (results not shown).

[22] The increases in PW_{max} are a robust result in the model simulations and have a strong theoretical basis, the Clausius-Clapeyron equation, linking the increases to increasing temperature. The PW_{max} increases are large and, if incorporated into PMP estimates, would have major implications for design of dams and other long-lived and critical runoff control structures. An important question then is whether any other meteorological factors may change in ways that offset, or add to, the expected changes in PMP attributed to an increase in PW. The key issue is whether the vertical motion "efficiency" variable W changes in the future. From equation (3), logarithmic differentiation equates the difference in fractional changes of PW and PMP to that of W. Over resolved sloping topography, the fractional change of W would be proportional to that of the upslope wind component.

[23] Although previous assessments of PMP assumed that there is no theoretical basis for determining a maximum value of vertical motion, there are conceptual simplifications for the space and time scales of PMP. Spatially, the scales of PW_{max} and PMP are rather large away from sharp topography. The relatively long durations of PMP applications (several hours to days) are also long compared to the time scale of transient convective elements. It follows that an idealized subsynoptic scale model of intense, persistent rain events can consist of a steady state, two-dimensional flow of saturated (moist adiabatic) atmosphere columns converging toward the precipitation zone.

[24] Following the discussion in section 2b, these examples illustrate the type of situations that may result in PMP events:

- Radial inflow of high PW air into a slow-moving tropical cyclone and ascent in the inner wall rainband (e.g., the U. S. 24 h rainfall record of 1092 mm at Alvin, TX, during Hurricane Claudette in July 1979 [*Hebert*, 1980]; this value is close to the 24 h PMP value of approximately 1200 mm).
- Flow of moist air toward an extratropical cyclone front that is stationary as a result of synoptic-scale flow (e.g. Illinois state record 24 h rainfall of 430 mm at Aurora on 18 July 1996) [*Changnon and Kunkel*, 1999].

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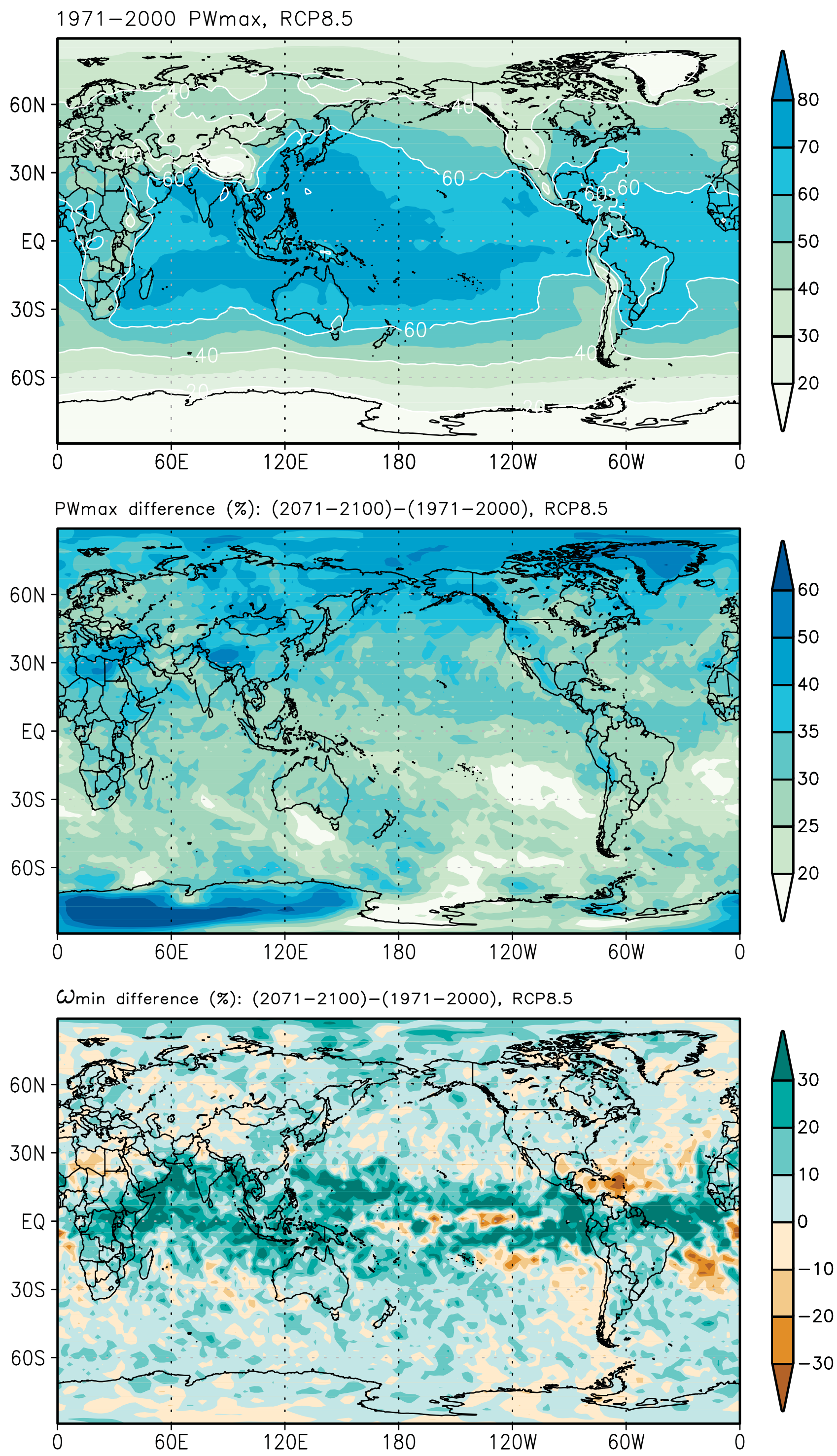

Figure 1. Fractional changes (%) of maximum precipitable water (PW_{max}) and upward motion (ω_{min}) projected by seven CMIP5 climate models. These are multimodel mean differences (future minus present) in the 30 year maximum values under the RCP8.5 scenario, for 2071–2100 relative to the 1971–2000 reference value for (middle) 12 h precipitable water and (bottom) 6 h upward motion. (top) The 30 year maximum precipitable water for 1971–2000 (mm), averaged over the same seven climate models.

- Sustained low-level jet sustaining a mesoscale convective system (e.g., the Nashville, TN flood of 1–2 May 2010 with 48 h rainfall exceeding 400 mm) [*Moore et al.*, 2012].
- Upslope advection of moist air masses by synoptic-scale winds encountering mountainous topography (e.g., the 6–7 November 2006 event in Washington and Oregon, where 3 day rainfall exceeded 700 mm) [*Neiman et al.*, 2008]. The persistence of topographically forced PMP events is then due to the synoptic-scale wind system.

[25] How will climate warming affect these types of meteorological phenomena? *Knutson et al.* [2010] assessed the state of knowledge regarding future projections of tropical cyclones. They indicated rainfall rates were likely to

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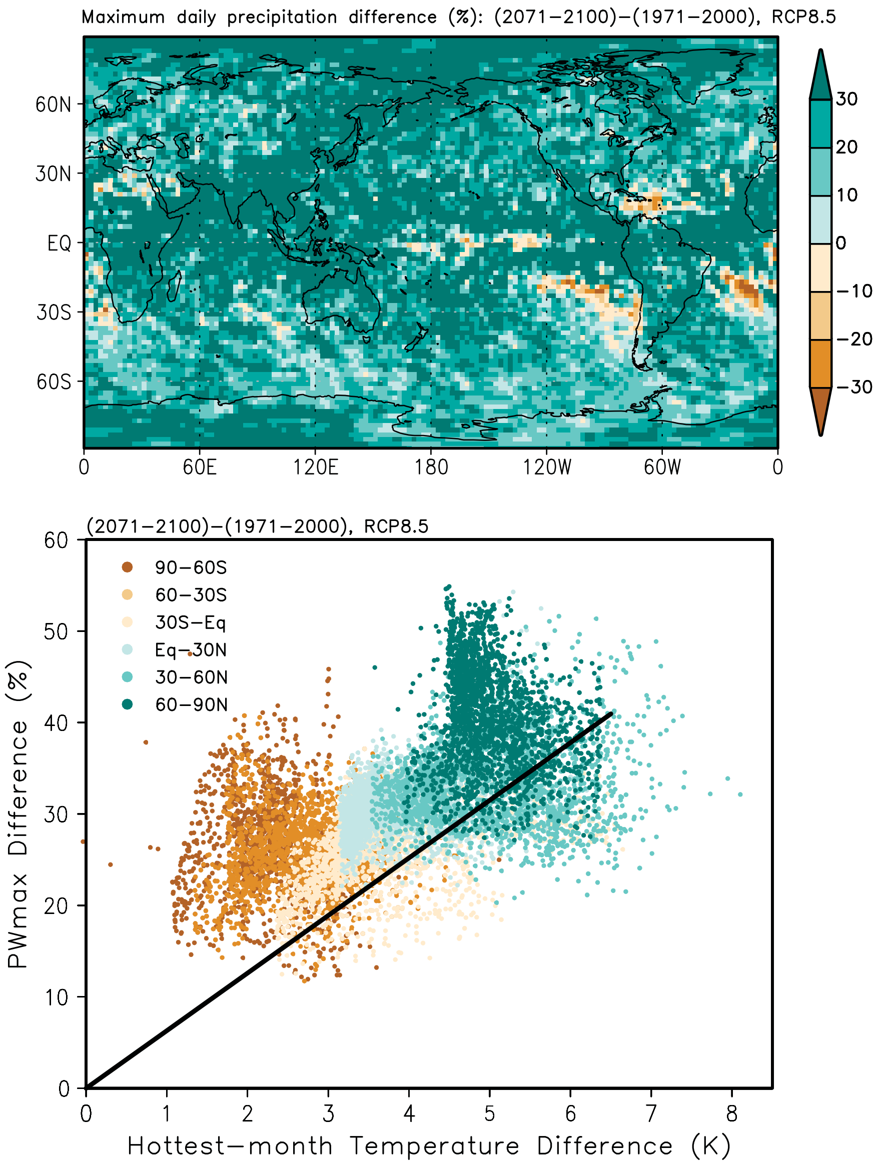

Figure 2. Fractional changes (%) of precipitation and PW_{max} projected by seven CMIP5 climate models. (top) These are multimodel mean differences (future minus present) in the 30 year period maximum daily precipitation for 2071–2100 under the RCP8.5 scenario, relative to the 1971–2000 reference value. (bottom) A scatterplot of grid point differences (future minus present) of the 30 year maximum precipitable water versus 30 year average temperature of the climatologically warmest month at 850 hPa for 2071–2100 with respect to 1971–2000 for the RCP8.5 scenario. The straight line represents a slope of 6.3% K^{-1}, the approximate value of the derivative of the saturation vapor pressure with respect to temperature at 288 K.

increase due to the general increase in water vapor concentrations based on theoretical considerations and high-resolution climate models. They estimated increases of +20% near the tropical cyclone center by the late 21st century under the A1B emissions scenario. For stationary fronts, studies of extratropical cyclones in the CMIP5 models find mixed changes [*Colle et al*., 2013] in the eastern U.S. with roughly equal areas of increases and decreases; thus, there does not appear to be a compelling reason to expect large changes in the maximum vertical motion produced by extratropical cyclones. Regarding atmospheric rivers, *Dettinger* [2011] finds that extreme atmospheric river episodes in the western U.S. actually increase in a multimodel ensemble from CMIP3.

[26] To further explore these characteristics in the CMIP5 simulations, fractional changes (%) in three relevant modeled variables were analyzed: the 30 year maximum values of (a) 6 h upward motion (ω_{min} where $\omega = dP/dt$ and P = pressure), (b) 6 h horizontal wind speed, and (c) daily precipitation. The results for ω_{min} for RCP8.5 for 2071–2100 are shown in Figure 1 (bottom). Note that these values are relevant to the largest scales for which PMP estimates are provided, as the model resolution is not sufficient to resolve small-scale upward motion and intense precipitation. The differences between 2071–2100 and 1971–2000 over the contiguous U.S. are mostly positive, and both the positive and negative magnitudes are mostly less than 10%, considerably smaller than the water vapor increases. Thus, the model

simulations do not show changes in maximum upward motion that could negate the increases in water vapor. Globally, the largest changes in upward motion are increases of greater than 20% at tropical latitudes while the largest areas of decreases of more than -10% are mostly in subtropical latitudes. The changes at mid and high latitudes are mixed in sign and mostly less than 10% in magnitude.

[27] Topographically forced vertical motion will be an important, perhaps even dominant, factor in extreme precipitation storms in certain areas such as the West Coast and along the Appalachian Mountains. This uplift will be directly related to the horizontal wind speed integrated from the upwind land/ocean surface to the crest. The CMIP5 models results (supplementary online material) do show areas of decreases in maximum horizontal wind speed over the western U.S. where topographic uplift is important. However, the magnitudes of the decreases are less than 6% almost everywhere and again much smaller than the water vapor increases.

[28] Model simulations are known to produce more intense precipitation under anthropogenic forcing [e.g., *Trenberth*, 2011]. Here we examine the most extreme precipitation values. Changes in the 30 year period maximum daily precipitation (Figure 2, top) are consistent with the above results. Increases are generally in the range of 10%–30% over the CONUS and mostly above 20% or similar to the changes in water vapor concentration. The highest daily precipitation accumulation during a 30 year period is extreme, but far less extreme than PMP values. Nevertheless, these results suggest water vapor changes are the dominant control on the magnitude of extreme precipitation, at least at the scale of the resolution of these model values, which is similar to the largest basin scale for which PMP values have been estimated (i.e., 51,800 km^2). Globally, large increases in maximum daily precipitation are simulated nearly everywhere. The few areas of spatially coherent decreases (Caribbean, eastern south Pacific, and south Atlantic) are mostly in subtropical areas where there are both decreases in maximum upward motion and smaller (than surrounding areas) increases in maximum precipitable water.

[29] The changes in the 30 year maximum PW as a function of changes in the temperature of the climatologically warmest month at 850 hPa generally follow the Clausius-Clapeyron relationship (Figure 2, bottom). The individual grid point values at low and midlatitudes cluster around a slope of 6.3% K^{-1} line, which is the approximate value of the derivative of saturation vapor pressure with respect to temperature at 288 K. The changes at high latitudes are generally somewhat greater than the nominal 6.3% K^{-1} value. This result also suggests a strong tie to temperature change and the overall robustness of the model PW projections. Note that the simulated temperature changes are generally smaller in the Southern Hemisphere than in the Northern Hemisphere, reflecting the moderating effects of the larger ocean area.

4. Summary

[30] Climate model simulations indicate a substantial increase in water vapor concentrations during the 21st century will occur. Since the imbalance in the radiative energy budget arising from an increase in greenhouse gases will almost surely be manifested in an increase in ocean heat content, there is high confidence in this model outcome. This increase in ocean heat content in turn will lead to an increase in atmospheric water vapor concentrations. The model simulations indicate that the changes in maximum water vapor concentrations, which are a principal input to PMP estimation techniques, will change by an amount comparable to mean water vapor changes, and ultimately to an accelerated water cycle with heavier extreme rains. The magnitude of the maximum water vapor changes follows approximately a quasi-exponential Clausius-Clapeyron relationship with temperature.

[31] Conceptual considerations suggest there are no compelling arguments for either increases or decreases of comparable magnitude in other factors used as inputs to PMP, specifically maximum vertical motion and horizontal wind speed. Indeed, model-simulated changes in the maximum values of these variables are too small to offset the water vapor changes. Model simulated-increases in extreme precipitation confirm the dominant role of water vapor in controlling such extremes. We conclude that the most scientifically sound projection is that PMP values will increase in the future and raise the risk of damaging floods. These conclusions apply not only to the U.S. but also globally to almost all other areas.

[32] **Acknowledgments.** This work was partially supported by NOAA through the Cooperative Institute for Climate and Satellites–North Carolina under Cooperative Agreement NA09NES4400006 and by the NOAA Climate Program Office, Climate Observations and Monitoring Program. We acknowledge the World Climate Research Programme's Working Group on Coupled Modeling, which is responsible for CMIP, and we thank the climate modeling groups (listed in Table 1 of the supplementary material) for producing and making available their model output. For CMIP the U.S. Department of Energy's Program for Climate Model Diagnosis and Intercomparison provides coordinating support and led development of software infrastructure in partnership with the Global Organization for Earth System Science Portals. We thank Mike Coniglio for advice on mesoscale convective systems.

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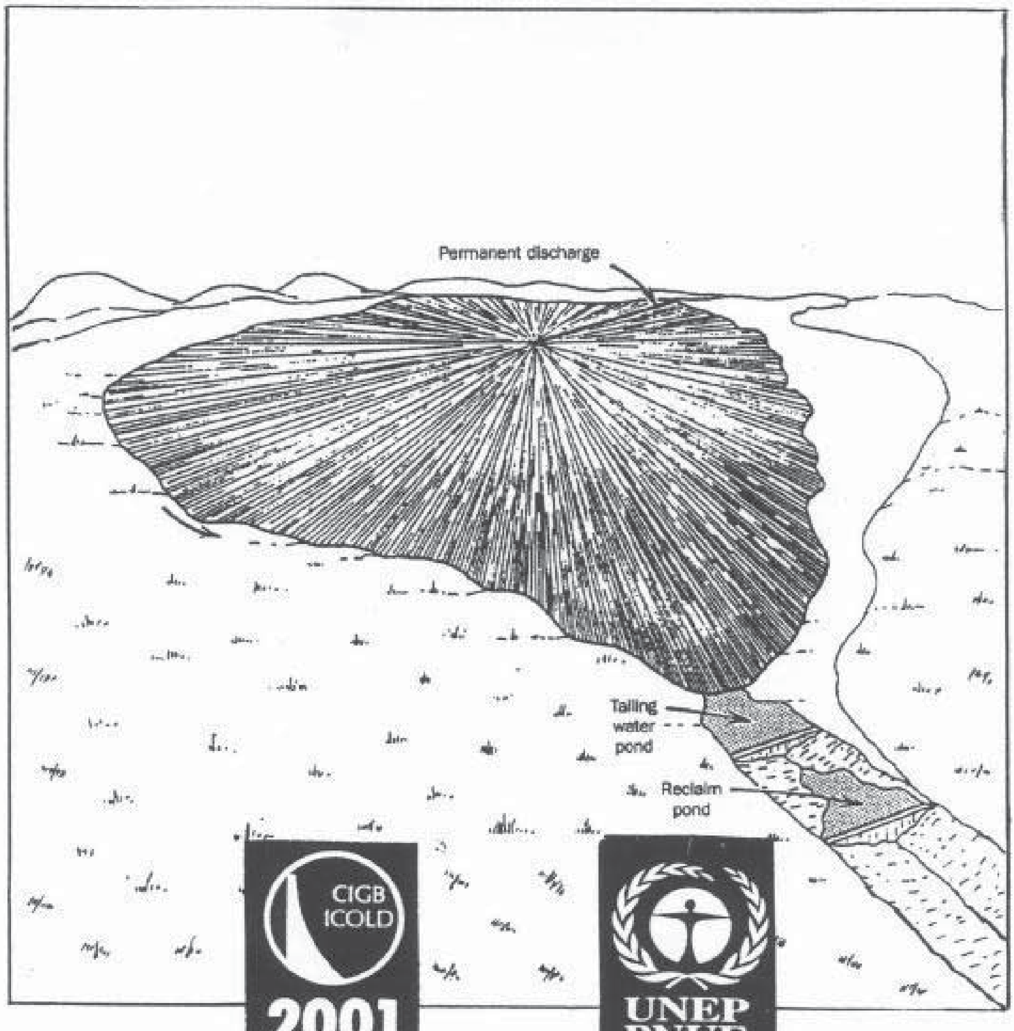
TAILINGS DAMS
RISK OF DANGEROUS OCCURRENCES

Lessons learnt from practical experiences

Bulletin 121



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Tailings storage using the Thickened Central Discharge Method
(from Robinsky, 1979)

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TAILINGS DAMS
RISK OF DANGEROUS OCCURRENCES

Lessons learnt from practical experiences

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SUMMARY

Tailings dams are built to retain impoundments of tailings, and when possible, material extracted from the tailings themselves is used in their construction. They have many features in common with embankment dams built to retain water reservoirs, and in many cases are built as water retaining dams, particularly where there is a requirement for the storage of water over the tailings, or the stored tailings have to be protected by a covering of water to prevent aerial pollution.

While the methods used for the design and construction of embankment dams can be applied to tailings dams, there are major differences between the two types. Embankment dams are prestigious structures used to profitably store water, whereas tailings dams are required for the storage of unwanted waste, desirably at minimum cost. Embankment dams are usually built to full height during one period of construction, having been designed and their construction supervised by competent engineers (controlled by law in many countries). Modern tailings dams are often designed by competent consulting engineers, but because they are built slowly in stages over many years, and conditions may also change with time, supervision of their construction may become faulty.

Guidelines for the design, construction and closure of safe tailings dams have been given by many publications, including ICOLD Bulletins Nos. 45 (1982), 74 (1989), 97 (1994), 98 (1995), 101 (1995), 103 (1996), 104 (1996), 106 (1996), ANCOLD (1999). If the recommendations given in these guidelines were to be closely followed, the risk of a failure or dangerous occurrence with a tailings dam and impoundment would be greatly reduced. Unfortunately the number of major incidents continues at an average of more than one a year. During the last 6 years the rate has been two per year.

With the intention of trying to determine the causes of these incidents, 221 case records have been collected. They are given both in brief detail and discussed in general terms. The main causes of these reported cases of failure and incidents were found to be lack of control of the water balance, lack of control of construction and a general lack of understanding of the features that control safe operations. There were one or two cases of unpredictable events and other cases caused by unexpected climatic conditions, including earthquakes, although it can be argued that with today's knowledge, allowance should have been made for these events.

Water retaining dams in most countries are controlled by legislation, and in some countries the legislation applying to embankment dams retaining water are equally applied to tailings dams. There appears to be a requirement for a more extensive application of legislation to the non-revenue raising activity of storing waste tailings, in order to reduce the occurrences of tailings dam failures and unsatisfactory behaviour. Up-to-date information can be obtained from a "Chronology of major tailings dam failures" compiled by WISE Uranium Project that can be found on the website http://www.antenna.nl/wise-database/uranium/mdaf.html

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RÉSUMÉ

Les barrages de stériles sont construits pour le stockage de stériles et, dans la mesure du possible, des matériaux extraits des stériles eux-mêmes sont utilisés pour la construction. Ces ouvrages ont beaucoup d'aspects comparables à ceux des barrages en remblai classiques créant des retenues d'eau, et dans bien des cas ils sont construits de la même façon, particulièrement lorsqu'il est exigé de retenir l'eau au-dessus des stériles ou lorsque les stériles mis en dépôt nécessitent une protection par une nappe d'eau afin d'éviter une pollution par voie aérienne.

Bien que les méthodes de conception et de construction des barrages en remblai classiques puissent s'appliquer aux barrages de stériles, ces deux types d'ouvrage présentent d'importantes différences. Les barrages en remblai sont des ouvrages prestigieux stockant l'eau dans un but utilitaire, alors que les barrages de stériles sont nécessaires pour la mise en dépôt des rejets et ce au moindre coût. Les barrages en remblai sont habituellement construits en une seule étape, leur conception et leur construction étant supervisées par des ingénieurs compétents (avec application de prescriptions légales dans beaucoup de pays). Les barrages de stériles modernes sont souvent conçus par des bureaux d'études compétents mais, du fait de leur construction lente par étapes s'étendant sur de nombreuses années, et du changement possible de conditions dans le temps, le contrôle de leur construction peut devenir défectueux.

Des recommandations relatives à la conception, à la construction et à la fermeture des barrages de stériles ont été présentées dans plusieurs publications ; on peut citer notamment les Bulletins de la CIGB n° 45 (1982), 74 (1989), 97 (1994), 98 (1995), 101 (1995), 103 (1996), 104 (1996), 106 (1996), et ANCOLD (1999). Si les recommandations données dans ces documents étaient suivies de près, les risques de rupture ou d'incident grave concernant les barrages et dépôts de stériles seraient beaucoup réduits. Malheureusement, le nombre d'incidents majeurs continue à dépasser en moyenne un taux de un par an. Au cours des six dernières années, ce taux a été de deux par an.

En vue de déterminer les causes de ces incidents, 221 cas de ruptures et d'incidents ont été répertoriés. Ils sont décrits brièvement et analysés en termes généraux. On a déduit que les principales causes des ruptures et incidents répertoriés ont été : un manque de maîtrise du bilan hydraulique, un mauvais contrôle de la construction et un défaut général de compréhension des principes dont dépend la sécurité des opérations. On a recensé un ou deux cas d'événements imprévisibles et d'autres cas dus à des conditions climatiques exceptionnelles et à des séismes, malgré la grande attention portée à de tels événements grâce aux connaissances actuelles.

Dans la plupart des pays, les barrages stockant de l'eau sont soumis à une législation, et dans certains pays la législation relative aux barrages en remblai s'applique également aux barrages de stériles. En vue de réduire les risques de rupture ou de mauvais comportement des barrages de stériles, il apparaît nécessaire

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The ICOLD Tailings Dams Committee has concluded that effective reduction of the cost of risk and failure can only be achieved by a commitment from Owners to the adequate and enforced application of available engineering technology to the design, construction and closure of tailings dams and impoundments over the entire period of their operating life.

d'étendre l'application de la législation aux activités non rentables de stockage des stériles. Des informations mises à jour peuvent être obtenues en consultant le document « Chronology of major tailings dam failures » établi par WISE Uranium Project et disponible sur le site web http://www.antenna.nl/wise-database/uranium/mdaf.html

Le Comité des Barrages et Dépôts de Stériles de la CIGB a conclu qu'une réduction effective des coûts de risque et de rupture ne pouvait être obtenue que par un engagement des maîtres d'ouvrage et une stricte application des technologies disponibles à la conception, à la construction et à la fermeture des barrages et dépôts de stériles pendant toute leur durée d'exploitation.

FOREWORD

The disposal of wastes in our overcrowded world has become a serious problem. Even domestic wastes in the developed countries presents a difficult disposal problem. Due to the nature of mining and mineral processing, the volumes of mining wastes are significantly larger than those of both domestic and industrial wastes. The chemical characteristics of the waste (particularly mobility of metal constituents) are often of concern. The volumes of mine waste greatly exceed the total volumes of materials handled by civil engineering throughout the world. The crushed rock passed through the processing plant to extract the desired product is discharged from the tail end of the plant as the waste tailings, and in many parts of the world forms the greatest volume of mine waste, although at open-pit mining operations the volume of waste rock may exceed the volume of tailings. The fine particulate tailings are commonly stored in impoundments retained by tailings dams. The material is placed hydraulically and so is loose and nearly saturated. Any major movement of the retaining boundaries of the impoundment can induce shearing strains that disturb the structure of the tailings mass, inducing a rapid rise of pore water pressures and liquefaction of a section of the impoundment, causing even greater pressures to be applied to the retaining boundaries. Failure of the retaining dam can release liquefied tailings that can travel for great distances, and because of its greater weight, destroying everything in its path. Water will flow through and around buildings, but liquefied tailings can destroy the structures. The tendency is for tailing dams to become ever higher and impoundments ever larger.

Similarities between tailings dams and embankment dams designed to retain water, have enabled many of the design techniques used with embankment dams to be applied to produce safe tailings dams, but despite great improvements, there has been a reported failure of a tailings dam almost every year for the past three decades. The damage caused by these failures in the form of human casualties, destruction of property, disruption of communications, pollution of the environment and economic loss to the mining industry is enormous. The purpose of this Bulletin is to discuss some of these failures and see what lessons can be learnt from them, to identify improvements that would reduce the occurrence of these failures.

The Bulletin was prepared by the British Sub-Committee on Tailings Dams, using with permission and agreement, the USCOLD collection of 185 case records published in 1994, the 26 cases found by Mining Research Services for UNEP, published in 1996, and 12 examples known by members of the ICOLD Committee. During final compilation of the case records, some duplications were found, so that the total number became 221. All members of the ICOLD Committee contributed to the final draft and liberal use has been made of their publications, e.g. Askari et al (1994), Penman et al (2000), Strachan (1999) and Williamson (1999). Special mention must be made of Mr Kulesza who categorised the 221 cases given in the Appendix. The Bulletin has been reviewed both by the ICOLD Committee on Tailings Dams and Waste Lagoons and by UNEP. Valuable comments have also been received from the National Committees of ICOLD and we are grateful to everyone involved.

A.D.M.Penman
Chairman, ICOLD Committee on Tailings Dams
and Waste Lagoons

PREFACE

D'Appolonia (1976) said, ‘Any attempt at construction of a tailings embankment that does not take into account the design-construct process is in my opinion doomed to great distress.' He pointed out that construction goes on for very many years, during which time many conditions may change, so it is essential to maintain a flexible approach and amend the design as required.

Rev Michael West (1998) said, “It is my own view that many mine accidents, and perhaps especially the most serious arise from over-familiarity with a potential hazard. Some of the worst accidents which have been associated with mud rushes and tailings movements are terrible examples - Mufulira, Stava, Harmony”.

Pierre Londe, President of ICOLD in a lecture given at AIT in 1980, about lessons from earth dam failures, said that man learned little from success but a lot from his mistakes: learning from our errors is vital for improving our knowledge and promoting safer design. He gave eight recommendations applicable to tailings dams. These were:

(1) Look out for over-consolidated clay formations, and use residual strength parameters;
(2) Look out for loose saturated sandy formations and study their liquefaction potential;
(3) Analyse the floods in terms of probability of occurrence and corresponding probability of damage downstream;
(4) Use the most recent hydrological methods;
(5) Provide ample and well graded filters and drains for preventing piping;
(6) Test for dispersion potential of fine clayey soils;
(7) Incorporate instrumentation as a vital part in the design of a dam for monitoring its safety;
(8) Provide a thorough and careful surveillance of dams and appurtenant structures on a continuing basis.

Guidelines for the safe design and construction of tailings dams and waste lagoons have been published in the following ICOLD Bulletins:

a)	No.45.	Manual on Tailings Dams and Dumps.	1982
b)	No.44a.	Bibliography.	1989
c)	No.74.	Tailings Dam Safety.	1989
d)	No.97.	Tailings Dams – Design of Drainage.	1994
e)	No.98.	Tailings Dams and Seismicity.	1995
f)	No.101.	Tailings Dams. Transport, Placement and Decantation.	1995
g)	No.103.	Tailings Dams and the Environment.	1996
h)	No.104.	Monitoring of Tailings Dams.	1996
i)	No.106.	A Guide to Tailings Dams and Impoundments.	1996

Knowledge about the factors that control the behaviour of tailings dams has improved greatly during the past 20 years, also the consequences and public perception of tailings dam failures has increased considerably, causing managers and owners to become more aware of the risks involved in the construction of impoundments. Many factors influence the behaviour of tailings impoundments; accidents and other incidents are often the result of inadequate site investigation, design, construction, operation, or monitoring of the impoundment, or a combination of these.

Every site and dam is unique so direct application from one to another is seldom possible. However, there are a number of common principles and the lessons learned from incidents at one dam can be applied in general terms to other situations. This Bulletin is intended to give general advice that can be of help to all those responsible for impoundments and tailings dams.

1. INTRODUCTION

1.1 OBJECTIVES

The aim of this Bulletin is to highlight and learn from some of the difficulties commonly encountered to help owners, managers, contractors and other personnel responsible for the day to day construction of tailings dams, to avoid similar difficulties. It is intended to be of help to all those connected in any way with tailings dams and waste lagoons: government officials concerned with regulations, planning officials concerned about the requirements needed for planning permission, and those concerned about the continuing stability of existing tailings dams.

Examples are given of accidents and failures, together with some examples of effective remedial measures.

1.2 BACKGROUND

In 1964 ICOLD approved a proposal for a 'Study of known failures and incidents arising from rock foundations for dams'. This title was modified during 1965-6 to 'Failures and accidents to large dams'. The ICOLD Committee on Failures and Accidents to Large Dams expanded its report to include events occurring during construction and major repairs. At the 1973 Executive Meeting of USCOLD, the Committee was authorised to proceed with a report covering incidents to USA dams during the period 1960 to 1972. *The report specifically excluded mine tailings and refuse dams.* Questionnaires were sent to a large number of dam owners in 1966 and 1973, resulting in information on 349 significant incidents. These were catalogued under 8 headings:

- F1 Major failure of a dam during operation, resulting in complete abandonment.
- F2 Major failure of a dam during operation that could be repaired.
- A1 Accident to dam that had been operating for some time, that was rectified before failure occurred.
- A2 Accident during initial filling.
- A3 Accident before filling began.
- AR Accident in a reservoir during operation, that did not cause trouble to the dam itself.
- DDC Damage to a partly constructed dam.
- MR Major repair required due to deterioration, or upgrade to comply with more modern standards.

The effect of improving understanding of the behaviour of dams that has resulted in improved methods of design and construction, is shown by the recorded failures during various periods.

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During the 50 years 1850 to 1900, 13% occurred
During the decade 1900 to 1910 there were 7% failures
During the decade 1910 to 1920 there were 4.8 % (*)
During the decade 1920 to 1930 there were 2%
From 1930 to 1940 and 1940 to 1950 there were less than 1%
During the decade 1950 to 1960 there were about 0.2%.

Fuller details about this study of incidents to dams, excluding tailings and refuse dams, can be found in ICOLD (1974), as well as in ASCE (1975) and (1988).

The ICOLD Committee on Tailings Dams and Waste Lagoons has attempted a similar approach for tailings dams, but has encountered a reluctance amongst the owners of tailings dams to expose incidents or failures unless they came into the public domain through the media or published papers. In North America, the Tailings Dams Committee of USCOLD collected data about tailings dam incidents from published literature, responses to questionnaires and anecdotal information. They collected 185 cases and published their findings in USCOLD (1994). The Committee agreed that this publication could form the basis of this present report, which is published jointly between ICOLD and UNEP (United Nations Environmental Programme). This latter organisation sponsored a literature search by the Mining Journal Research Services, going back only to 1980. The Research Services contacted 52 organisations in 18 countries. Of this group, 23 agreed to participate and were sent questionnaires from which 20 replies were received. A list of 26 incidents was compiled, in addition to the cases collected by USCOLD. The ICOLD Committee also collected published cases, many passed on to the Research Services, making a grand total of 221 cases.

(*) In the main text and the appendix, the Anglo-Saxon usage (full stop and comma) has been adopted for the quantities.

2. PRE-REQUISITES FOR SAFE TAILINGS DAMS

Satellite imagery has led us to the realisation that tailings impoundments are probably the largest man-made structures on earth. Their safety, for the protection of life, the environment and property, is an essential need in today's mining operations. These factors, and the relatively poor safety record revealed by the numbers of failures in tailings dams, have led to an increasing awareness of the need for enhanced safety provisions in the design and operation of tailings dams. The mining industry has a less than perfect record when tailings dam failures are reviewed. Examples of notable failures that have been costly to life, the environment and to asset value, are given by Table 1.

Table 1. Examples of tailings dam failures

October 2000	Martin Country Coal Corporation, Kentucky, USA. 0.95 million m^3 coal waste slurry released into local streams. Fish kill in River Tug and drinking water intakes had to be closed.
Sept 2000.	Aitik mine: Sweden: 1.8 million m^3 water released.
March 2000	Borsa: Romania: 22,000 t tailings contaminated by heavy metals released.
Jan 2000	Baia Mare: Romania: 100,000 m^3 cyanide contaminated water with some tailings released.
April 1999	Placer, Surigao del Norte: Philippines: 700,000 t cyanide contaminated tailings released; 17 homes buried.
Dec 1998	Haelva: Spain: 50,000 m^3 acidic and toxic water released.
April 1998	Aznalcóllar: Spain: 4-5 million m^3 toxic water and slurry released.
Oct 1997	Pinto Valley: USA: 230,000 m^3 tailings and mine rock.
Aug 1996	El Porco: Bolivia: 400,000 t involved.
March 1996	Marcopper: Philippines: 1.5 million tonnes tailings released.
Sept 1995	Placer: Philippines: 50,000 m^3 released, 12 killed.
Aug 1995	Omai: Guyana: 4.2 million m^3 cyanide slurry released.
Feb 1994	Merriespruit: South Africa: 17 lives lost, 500,000 m^3 slurry flowed 2 km.

July 1985	Stava: Italy: 269 lives lost, tailings flowed up to 8 km.
Jan 1978	Arcturus: Zimbabwe: 1978: 1 life lost, 20,000 m^3 flowed 300 m.
Nov 1974	Bafokeng: South Africa: 12 deaths, 3 million m^3 slurry flowed 45km.
Feb 1972	Buffalo Creek: USA: 125 lives lost, 500 homes destroyed.
Sept 1970	Mufilira:Zambia: 89 deaths, 68,000 m^3 into mine workings.

Other examples:

Saaiplaas:	South Africa	1992
Jinshan:	China	1986
Pica Sao Luiz:	Brazil	1985
El Corbre:	Chile	1965

Attention at the design stage to the critical issues that can affect the long term safety of a tailings dam, will pay dividends throughout the life of the dam. The primary features affecting the design of a tailings disposal facility include:

a) The rate of tailings delivery and potential future changes.
b) The properties of the tailings and potential future variations.
c) The influence of additives on the properties, e.g. thickener flocculants.
d) The properties of the disposal area site foundations.
e) Possible influence of seismic loading.
f) Rainfall and evaporation rates.
g) Requirement for water cover and its depth.

From the point of view of the dam itself, factors affecting stability include:

1) Detailed foundation conditions.
2) Ultimate height and angle of the outer slope.
3) The rate of deposition and the detailed properties of the tailings.
4) Provision of adequate drainage. *See Bulletin No. 97 (1994).*
5) Seismic influences. *See Bulletin No. 98 (1995).*
6) Control of hydrology to avoid overtopping or dangerous rises of the phreatic surface within the dam body. *See Bulletin No. 101 (1995).*

To begin at the beginning, the choice of site is all important. In general tailings are transported as a slurry in a pipeline or, sometimes, in an open flume where flow can be caused by gravity alone, so that the disposal site can economically be some distance from the processing plant, giving a fairly wide choice of site position. The selected site must be of adequate size so that the tailings can be deposited throughout the life of the impoundment at safe rates of rise or embankment staging and with a final volume to satisfy the predicted volume of mineral extraction.

The filter under-drainage system is a critical facility that has often been overlooked in the past, resulting in dangerously high phreatic surfaces within the body of the tailings dam. As is well known, the outer slopes of a tailings dam are very sensitive to the level of the phreatic surface. Capillary rise above the measured position of the phreatic surface can make the tailings in this zone to be close to full saturation. This condition can produce unexpectedly large rises of the phreatic surface from remarkably small amounts of rainfall.

From the point of view of the tailings delivery and deposition system, safe design will incorporate the appropriate selection of a system that will ensure that breakages, wear and tear and maintenance be kept to a minimum, and that the supernatant pool is always contained safely, with adequate statutory freeboard. This is of particular importance in the early deposition stages of a new facility, where careful design is required. With paddock type construction, tailings must always be deposited first into the outer paddocks to build up the surrounding dam to safely contain the impoundment at all times. The outer banks can, if necessary, be raised more rapidly by use of hydro-cyclones, or by influencing the deposited beach slope.

Effective quality control and monitoring of the construction process for compliance with the design and works specification will ensure long-term effectiveness of these components, i.e. the starter dam; the filter drains; the decant facility; installed instrumentation *(see Bulletin No. 104, 1996)* and the tailings delivery system.

Controlled management of the deposition process and the operating functions of a tailings dam will significantly enhance the safety of a tailings disposal facility. Tailings dams usually have a significant deposition life, commonly more than twenty years, so safety management and checking for compliance with the design, or modifying the design to accommodate changed circumstances is an essential component of the operation. The safe day to day operation must be managed by correct planning procedures, that involves measurement of the volume and properties of the tailings slurry being delivered to the impoundment, and monitoring the construction activities in detail.

The process of implementing decisions associated with the assessment, toleration and reduction of risks can be termed safety management. Owners and operators have specific responsibilities for their dams and the need to formulate safety management procedures. Technical and managerial approaches should be utilised to improve safety and reduce risk. Continuing day to day safety of the dam-impoundment system will depend on some form of observational method involving surveillance and monitoring, using suitable instrumentation to reveal internal conditions.

An increase in safety is provided at an increase in cost and a balance has to be found between dam safety and economy. Cost effective risk reduction involves defining the acceptable level of risk, reducing the risk of the dam breaching to an acceptable value and implementing emergency management procedures to endeavour to ensure that there is no loss of life should the dam breach. The approaches to risk reduction for the dam-impoundment system can include structural improvements to the dam and ancillary works, improved surveillance, monitoring and maintenance. The approaches to risk reduction for the downstream valley system include the preparation of inundation maps, estimation of the time of arrival of flood wave at different locations and the duration of inundation and the implementation and maintenance of emergency warning procedures and systems. Unlike water, liquefied tailings do not drain away and the deposits left on roadways can seriously hamper emergency services. The weight of tailings is such that it can cause great damage, much greater than that of an equivalent flood of water, demolishing buildings rather than just flowing through them. The difficulty in knowing when to give warning makes the operation of emergency procedures very difficult.

Planning and management activities should include:

a) Staff training.
b) Planning of deposition cycles and positions.
c) Planning for dam geometry control.
d) Planning for maintenance activities.
e) Planning of measurement and monitoring activities.
f) Planning for responses to emergencies and for contingencies.

Recorded measurements made during dam construction will include:

a) The volumes and properties of delivered tailings slurry.
b) Levels of the dam crest and of the water pool, giving freeboard values.
c) Position of the phreatic surface.
d) In situ properties of the deposited tailings.
e) Seepage discharge flows from the filter drains.
f) Record of any uncontrolled toe seepage or other signs of distress.

Monitoring will include:

a) Regular visual inspections of the dam and its facilities from ground level and possibly also from the air.
b) Inspection of all measurement records with checks for compliance.
c) Recording and reporting all non-compliances and arranging remedial action.
d) Close liaison with designers.
e) Close liaison with the controlling authorities.

When judging the condition and safety of a tailings dam on the basis of instrumentation results and observations, consideration must be given to the possibility of unusual loading conditions, particularly from extreme meteorological events, which must be met by the reserves inherent in the structural system.

In conclusion it can be said that the provision for safe tailings dams can be made by careful attention to the critical components of the dam at the design stage, by effective quality control of the pre-deposition works construction and to professional management and monitoring of the deposition and operating process, all by experienced engineers and operators.

3. OVERVIEW OF DAM AND TAILINGS DAM INCIDENTS

The International Commission on Large Dams (ICOLD) and the National Committees of its 81 member countries, provides a forum for technical interaction amongst dam designers and constructors, and has recognised the importance of learning from failures and accidents with dams. ICOLD has numerous technical committees that publish Bulletins giving guidance to various aspects of dam design, construction and monitoring. One committee studied cases of failures and incidents amongst the dams of the world, and published 'Lessons from Dam Incidents' (ICOLD 1974). ICOLD also holds a congress every three years, and the Transactions of these Congresses contain extremely valuable information: they can be described as milestones along the path of progress in our subject. The U.S. Committee on Large Dams (USCOLD) conducted an incident survey of dams in the United States, updated by their Committee on Dam Safety with the results published in 1988 (USCOLD 1988). Over 500 incidents were tabulated, consisting of dam failures, accidents and major repairs to dam facilities. Embankment dams comprise approximately 73% of the dams in operation, and about 75% of the recorded incidents were related to this type of dam. Approximately 24% of the incidents were failures, and about 42% of the incidents were major repairs, with the remaining 34% described as accidents. This incident data is summarised graphically in Fig.1, where the type of incident has been compared with its cause. Conclusions that can be drawn from Fig.1 (*) are that the majority of dam failures are associated with overtopping, and the majority of incidents are associated with spillways and other facility structures.

Designers and operators of tailings dams also gain important information from both the satisfactory and unsatisfactory behaviour of both existing embankments and tailings dams. The focus of assessing dam performance is not to pass judgement on an industry or attach blame to the designer or constructor involved with a failure or an accident, but to gain knowledge from the records of performance and apply that knowledge to the next design or project. An ICOLD technical committee on tailings dams was formed following the Mexican Congress of 1976, and it has published several manuals and guides on tailings dams based on experience with tailings dam performance. A list of these publications is given in the Preface.

In addition to their incident survey of dams in the United States, USCOLD also made a tailings dam incident survey, updated in 1994. It contained 185 tailings dam incidents consisting of failures, accidents, seepage and examples of behaviour that did not meet design criteria, that had occurred during the period 1917 to 1989. This collection has been supplemented by a recent tailings dam incident survey made by the United Nations Environmental Programme (UNEP), and has added 26 cases to the 185 incidents.

The historical record of the incident survey data is shown by Fig.2 in terms of the number of incidents per five-year period, and differentiates between the USCOLD and

(*) All the Fig. are given at the end of the main text (Chapter 10)

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the UNEP/ICOLD collected data. It should be noted that all the figures showing tailings dam data are plotted with the number of incidents as the dependent variable. Fig.2 shows an increase of incidents starting in the mid-1960s. This increase is most likely due to the larger number of tailings dams constructed after 1960 combined with the more thorough documentation of tailings dam operation. The USCOLD and UNEP surveys have provided a total of 211 incidents for comparison. Clearly all incidents have not been reported, and this collected number form a subset of the actual number of tailings dam incident that have occurred from the early 1900s to 1996. The following discussion of results is presented in terms of the number of incidents, rather than downstream effects or cost of repair or remediation. A comparison of the number of incidents with tailings dam height is given by Fig.3, where it will be seen that about 57% of the incidents occurred in dams less than 20 m high. A comparison of actual dam failures with dam height is given by Fig.4 which shows that approximately 63% of the failures occurred in dams lower than 20 m.

When an impoundment has become completely filled, or when tailings production ceases, the tailings dam and its retained impoundment is described as inactive. This does not make them immune to accidents or failures and, as shown by Fig.7, relatively few incidents were associated with inactive dams. A cause of failure of inactive dams is increase of pool water level, bringing the pool closer to the dam crest thereby bringing the phreatic surface to the downstream slope, leading to eventual slips and overtopping. A method to avoid the overtopping of closed impoundments consists of building a subsidiary dyke some distance back from the crest of the tailings dam to permit the level of the pond water to rise during an emergency without risking the main dam. The section of such a dam in Germany is shown by Fig.5.

As can well be imagined, the type of construction of the tailings dam must play a part in its behaviour. One of the earliest and the most common types of construction is by the upstream method: tailings dams constructed with tailings by the upstream method have been documented in South Africa in the early 1900s *(types of construction have been described in Bulletin 106)*: traditional embankment dams when used to retain impoundments, are referred to as water retaining type dams. Because of the risk of failure of dams built by the upstream method, particularly when subject to earthquake shaking, the downstream method was developed; and there is a centreline method that is a compromise between the former two methods. As Fig.6 shows clearly, there are many more incidents with dams built by the upstream method than with other types, but also there are many more of these type of dam than of other types amongst the examples that we are considering.

An indication of the causes of incidents, for active and inactive tailings dams are given by Fig.7, where it will be seen that the leading cause of incidents for active dams are slope instability, overtopping and earthquake. For inactive dams, the leading causes of incidents are overtopping and earthquake. Fig.8 similarly shows incident cause for active dams, separating failures form accidents. Finally, Fig.9 shows the total incidents with their cause separated by tailings dam type. This figure also indicates that the leading causes for incidents are slope instability, earthquake and overtopping: particularly so for dams constructed by the upstream method. The incidents must be reviewed in terms of the number of particular dam types in operation. The upstream method is the oldest and most commonly used method of tailings dam construction. This method, as pointed out by Mittal and Morgenstern (1977), was used at sites prior to the use of foundation investigation and slope stability analyses. On the other hand, dams built by the newer centreline method show relatively few incidents, but it must be recognised that significantly fewer dams of this type exist as compared with the number built by the upstream method.

From the review of water-storage dam incidents, key design factors include regional conditions (climatic conditions, design storm event for spillway sizing, and the design earthquake) and site specific conditions (dam and reservoir foundations and slope conditions, and on-site construction materials). Water storage dams are designed to retain and release water over a range of operating level conditions, as well as pass flows from large storms. Key periods of observation include dam construction, initial reservoir filling, and the response time to initial filling. Maintenance and repair become more important as exposed structures, such as spillways and gates age and wear.

With tailings dams, key design factors include the same regional and site-specific conditions, but also include the mill production schedule and tailings characteristics. Tailings dams and impoundments are designed to retain material that is initially discharged into the impoundment as a slurry, and operated to provide separation of the tailings solids from the transporting water. Upon successful reclamation, the impoundment is essentially a storage facility for solid materials, with as little impounded or entrained liquid as possible, although it is very difficult to reduce the considerable volume of water retained within the pore space between the small particles constituting the tailings. Additional drainage can be provided either by including drainage layers into the impoundment during construction or by installing vertical and/or horizontal sand drains at a later stage. Research work on the effect of water chemistry on the density of settling tailings and the effects of colloids in the water has been described by Vaughan (1999). The application of vibrations of audio frequency during sedimentation was found to increase the density of the deposit.

The impoundment is typically designed to contain water from a range of operating water-level conditions, as well as contain impounded water from large storm events. The containment capacity for free water must be maintained despite the ever increasing level of the retained tailings solids. To reduce capital costs, tailings dams and associated impoundment areas are usually constructed in stages or phases. Protection from storms is maintained by diversion of upstream drainage and providing adequate capacity for precipitation directly on the impoundment area.

In comparison with water storage dams, the key period of observation for a tailings dam is the entire operating life of the impoundment to its completion and during its reclamation. This differs completely from water-storage dam operation, because tailings dam construction and impoundment filling are taking place throughout the mill operation period. Settlement, consolidation and drainage of the tailings is occurring throughout the operation, as well as for a period afterwards. Maintenance and repairs take place during operation, and become less important after reclamation as operating features are closed or removed.

The design and operation of a tailings dam has been described as an example of the observational approach in geotechnical engineering, as discussed by Peck (1969), where the design is based on the best information available at the time, with planned contingencies and subsequent modifications based on careful observation and monitoring of the initial construction and operation. This is the case if the initial design was based on the key design factors listed above, followed by careful observation and monitoring. The tailings dam incidents discussed here reflect deficiencies in this observational approach.

The incident survey results show that there is no overriding cause or mechanism for all tailings dam incidents, but it is intended that readers may find examples close to cases under their control, and benefit from the experiences recorded here. As shown by Fig.9, incidents are due to a number of factors including overtopping, static and seismic instability, seepage and internal erosion, external erosion, structural and foundation conditions. In addition, no type of tailings dam or operational condition is particularly immune from incidents. Although the majority of tailings dam incidents are associated with active tailings impoundments, a range of types of incidents have been recorded at inactive impoundments.

4. COMMON REASONS FOR FAULTY BEHAVIOUR

Water is used in the grinding and processing of mineral ore for the extraction of the required metals, so that at the tail end of the processing, the discarded tailings is in the form of a slurry of water and mineral particles that will flow in a channel or pipeline. Tailings that come from chemical and other plants is often stored in a similar way, but the material is not always suitable for dam construction.

A great deal of water is included in mineral tailings that come to an impoundment. The material in its wet form is unsuitable for dam construction. Water can be removed by drainage under the downward pull of gravity, by evaporation to the atmosphere and by consolidation caused by gravitational action on the mineral particles, which settle inside the mass of the impoundment, leaving free water that can be decanted or pumped from the surface. The coarsest particles used for the construction of the dam have the highest permeability of the tailings material so that the water in the pores can escape when there are adequate drainage outlets.

With a dam being built by the upstream method, the coarsest particles settle out nearest to the crest, with reduction of particles sizing with advancement down the gently sloping beach until only the finest particles are left to be carried into the pond where they settle. While the surface of the water in the pond is visible, with distance moving downstream towards the dam crest, the level of the water within the tailings, referred to as the phreatic surface, falls as it approached the downstream slope where it can drain out or evaporate. In some early dams, built without drains, the positions for the phreatic surfaces under various conditions, are shown by Fig.10. Variations of placement, particle sizes etc., cause the body of an impoundment to be extremely anisotropic with regard to permeability. This results in equi-potential lines that are far from vertical, which can cause an incorrect measurement of the position of the phreatic surface by open standpipes that have long intake filters. Also layers of materials of low permeability can produce perched phreatic surfaces, as indicated by Fig 11(e). In fact it is essential for the stability of the dam that the phreatic surface is kept well back from the downstream slope. When it is allowed to move downstream by allowing the pond to move too close to the dam crest, local small rotational slips will occur where the phreatic surface meets the downstream slope.

Downstream slopes may suffer slight gullying caused by rainfall, and first signs of a close phreatic surface may be given by water issuing into one of these gullies. If there is no remedial action, small slips may develop, deepening the gully and moving the surface of that part of the downstream slope even further back into the phreatic surface. This leads to larger slips that continue to eat away the base of the slope, until the whole downstream face becomes unstable. In the extreme the slips may reach up as far as the dam crest, removing sufficient of it to permit of overtopping and failure of the dam. Thus it can be seen that complete control of the water regime is of the utmost importance to maintain the stability of the dam and impoundment.

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Dams built by the downstream method use the coarsest fraction from the tailings, usually separated by the use of hydro-cyclones. These simple machines, illustrated by Fig.11, have no moving parts. The tailings supply enters the cylindrical shaped body tangentially, producing rapid rotation, throwing the coarse fraction to the periphery where it moves down to discharge from the bottom nozzle, which is made of adjustable diameter so that the sizes of the discharged particles can be controlled. The smaller the rate of discharge permitted, the larger the sizes. The finest fractions from the centre of the swirling mass pass through a central pipe to discharge from the top of the hydro-cyclone, usually connected by delivery pipes to be discharged into the impoundment.

Not all tailings contain a sufficient volume of coarse (sand size) particles for the construction of a downstream type dam and changes have to be made in the construction method to incorporate imported fill and/or change to the upstream method from time to time. Provision must be made at all times during construction to ensure that good drainage is being built into the dam to ensure that there is no danger of the phreatic surface from advancing to the downstream slope. In general, dams built by the downstream or centreline method are much safer than those built by the upstream method, particularly when subject to earthquake shaking.

In the exceptional circumstances of the dam being constructed of material that, when in place, proved to be less permeable than the tailings it was retaining, the phreatic surface in the dam can rise to dangerous levels, causing instability. An unidentified case has been reported of the failure of a downstream slope due to this cause. Clearly extensive drainage should have been provided in this dam fill.

It must be pointed out that a permeability value is not a property of a particular material. The value varies with the density of the fill, its degree of saturation and particularly with the values of existing effective stresses.

Appropriate remedial measures carried out at the right time can prevent expensive and often fatal occurrences. The responsibility for the safety of tailings dams must lie with the owner or operator. The owner or operator of the tailings dam and retained impoundment has to ensure that a competent person is engaged to have overall charge of construction and that he is given the power to make modifications to the methods of construction being used and in the extreme to stop further materials from being delivered to the dam and impoundment. This may require the enterprise to operate an emergency (or just a second) disposal facility into which the tailings flow could be diverted, although in North America, emergency impoundments may not be considered feasible due to the cost of obtaining permits, provision of a liner and the freeboard requirements demanded for an emergency impoundment. It is often said that there should be an operations manual for each tailings dam, written by the designer and modified by him during the years of operation to ensure that it always provides practical advice. In North America an operations manual is a required part of the permitting process, and it is modified into standard operating procedures or simplified guidelines for operational personnel. This approach can be dangerous if the personnel operating construction do not fully understand the implications of some of the recommendations. The manual may also have become a lengthy document and staff may not have time to re-study the manual and from it work out the correct response to an emergency. There is no substitute for a competent engineer to be fully responsible for the safe construction of the dam.

Regular inspection is an important part of checking that the construction procedure is correct and that no hidden mistakes are being built into the structure. Combined with regular inspection is the correct and regular monitoring of all instrumentation and the taking of measurements etc. Aerial photography and satellite imagery can be of great help to obtain the overall view of the way the impoundment is developing. But, apparently efficient inspection may prove deceptive. There is a case where failure occurred, with an inspector driving on and around the dam, inspecting it on a semi-continuous basis. Unless the inspector has a good knowledge of the behaviour of tailings dams and has at his disposal instrumentation to reveal to him the conditions within the body of the dam and its foundations, such inspection may be not only useless but also highly dangerous by giving to management a false sense of security. An example of the weakness and danger of such inspection was given by the example of Incident No.206; further details are given in Section 6.

5. RISK MANAGEMENT

The failure of a tailings dam and the uncontrolled release of the impounded waste may have serious consequences for the public safety, the environment and the Owner or Operator. Some of the types of consequences can include the following:

- Economic Consequences: Included under this heading are the costs of repair or reconstruction of the dam and impoundment and the effects on the operator of the facility of a temporary lack of storage for waste.
- Public Safety: Public attention has increasingly focused on matters relating to safety and a hazard which may affect a large number of people in a single catastrophe is less acceptable than every day hazards which may in aggregate cause far more deaths but in each incident affect only one or two individuals. An imposed and involuntary exposure due to living close to some hazard is much less acceptable than a voluntary exposure to a high risk activity.
- Environmental Damage: The release of a substantial quantity of waste material which then flows over a large area of surrounding ground may cause massive environmental damage, particularly if the waste is toxic. There are also risks associated with incremental events over a longer term such as dust dispersion, groundwater contamination, landslide or ground instability.
- The Risk Management process involves carrying out a Risk Assessment to assess the potential failure modes and consequences, a Risk Management Plan to reduce the risks through design or operations, and a Contingency Plan to develop an optimal response to failures.

5.1 RISK ASSESSMENT

Risk analysis can be qualitative or quantitative. The term quantitative risk assessment (QRA) refers to the technique of assessing the frequency of an unwanted event and its measurable consequences in terms such as number of fatalities or cost of damage (Dise & Vick, 2000). QRA techniques are advocated by many regulatory bodies to assess the safety of modern complex plants and their protective systems (U.S. Department of the Interior, Bureau of Reclamation, 1999) It can be questioned whether QRA, as used in the process industries, involving large numbers of fault trees and event trees is appropriate for the assessment of the safety of tailings dams and waste impoundments. Qualitative Risk Assessment techniques are commonly based upon the Failure Modes Effects Analysis (FMEA) which was developed as a result of the Bhopal and Challenger disasters. McLeod and Plewes (1999) further developed the methodology to quantify the environmental and socio-economic consequences of failure for tailing facilities.

The risk assessment process has many variables and approaches (Vick, 1999). The ICOLD subcommittee on Dam Safety is currently preparing a Bulletin on Risk Assessment and, although it is addressed towards water storage dams, the same principles apply to tailing facilities.

Risk analysis should consider the main components of the facility which include:

Dam and Foundations:

- Has the dam been designed by competent engineers, with due regard for foundation conditions, internal drainage, slope stability, seismic loading, and contaminant containment?

- Where tailings or cycloned sand are used for construction, has the structure been assessed with the same rigour as an earth/rockfill dam?

- Is the dam instrumented and/or monitored, so as to reveal any abnormal behaviour?

Water Management Systems:

- Are the decant systems secure and have all pipes through the dam or foundation been adequately sealed and/or protected against piping failure?

- Is there sufficient flood storage capacity?

- Are spillways and/or diversions adequate for the design floods?

- Are there any hazards associated with the tailing delivery lines and water reclaim lines?

Closure:

- Has the structure been designed to accommodate potential changes in operating conditions over the closure period, e.g. erosion, floods, sediment inflows or natural landslides, etc.?

- Are the closure works suitable to reduce the potential for contaminant transport?

5.2 RISK MANAGEMENT

There is increasing recognition of the role that risk management has in dam safety assessments (Bowles et al, 1997). The risk assessment process will identify a number of risks associated with the tailings facility. The objective of the Risk Management Plan is to apply compensating factors to reduce the level of risk. The main areas of compensating factors include the following:

- Design: These may be civil works to increase the safety (e.g. berms), additional technical and environmental studies to increase the level of confidence in the assessment.

- Security: This could include both passive and active security systems to safeguard the public and the operating facilities.

- Monitoring and inspection systems: This allows early response to changes and identifies conditions which may be changing over the life of the facility. This includes the requirements of quality assurance and quality control(QA/QC) throughout the operations.

- Maintenance programs: These include such items as maintenance of diversion and water management structures, collection or treatment facilities, access roads, etc.

- Management: This includes supervision requirements, training of staff, reporting and Corporate/Public assurance.

Trade-off studies are required to optimise the cost/benefit of various compensating factors with the proportionate reduction in risk. Owners and operators have specific responsibilities for their dams and the need to formulate risk management procedures.

5.3 RISK CONTINGENCY PLAN

All facilities carry some degree of risk, even after risk management plans have been implemented. A risk contingency plan is therefore required to address those risks which cannot be eliminated. Contingency plans are required to address the required action and to mitigate the consequences if the event occurs. Contingency plans are needed to address issues of responsibility, notification, emergency response, technical monitoring and technical response, and other issues.

As with any structure theoretically capable of catastrophic failure, tailings dams require a contingency plan to be developed to deal with a possible accident. As tailings accidents may involve either physical and chemical consequences for persons and the environment, both of these aspects need to be considered. In addition to the requirements for support equipment it is necessary to have a clear communications and coordination plans in order to manage the response.

Emergency response plans necessarily require that the potentially affected community understands what it must do in case of accident. Public anxiety after a spill from tailings is greatly reduced if understanding of the real consequences has been built before an incident. Such understanding is impossible to achieve after an incident because learning ability is diminished by high anxiety levels, and a low level of trust at that time.

Emergency response plans are required components for new tailings impoundments in the United States of America and some other countries. The

UNEP's APELL (Awareness and Preparedness for Emergencies at the Local Level) Program has been designed to help companies, local government and the emergency services put a coordinated plan together to improve public preparedness in case of industrial accidents, including those that may arise at mine sites. The APELL program involves all the affected partners and actors, it is not a unilateral plan on the part of only the company or the emergency services.

An informed public will initially question the acceptability of any externally imposed risk. It is common for the company to undertake a thorough risk management plan and it can be a very valuable tool to incorporate the local stakeholders into the process (McLeod and Plewes, 1999). This allows the potentially affected community as well as the "response" organisations to properly understand the issues and the best emergency preparedness arrangements.

The APELL program has been applied in many industrial situation around the world. The APELL handbook and supporting material is available in over 20 languages, and there is a pool of United Nations and national experts to assist companies and communities in developing emergency preparedness plans. The procedure is directly applicable to tailings facilities.

6. LESSONS LEARNT FROM THIS STUDY

6.1 GENERAL ASSESSMENT OF THE LESSONS LEARNED

A study of the case records contained in this Bulletin may give an insight into the behaviour of a particular tailings dam of concern to the reader. It is expected that all readers will find items of interest that can be applied to dams and impoundments for which they are responsible. A general assessment of the lessons learnt from this collection of case records are given by the several points that follow:

- Although our understanding of the behaviour of embankment dams has improved to the extent that they can be designed to behave correctly, and many of the design features can be applied to the design of tailings dams, tailings dams continue to fail. During the decade 1979 to 1989 there were 13 failures. The decade before, 1969 to 1979 had at least one failure every year, and the most recent decade, 1989 to 1999 suffered 21 reported failures. It must be emphasised, however, that failures can occur without reaching the public domain. Only the more serious cases that attract media attention are the ones we hear about.

  These numbers of failures may well simply reflect the ever increasing numbers of tailings dams being constructed together with the increasing numbers of closed tailings dams. Consideration of the total number of operational and closed impoundments, the percentage of failures would be seen as decreasing with advancing time. This, unfortunately, is of little comfort to the owner of one of the dams that fails, although it would give an indication that the methods used by the profession for the design and construction of tailings dams are steadily improving.

- It can be argued that failures are due to inadequate management. The art and science of geotechnical engineering and geology, plus the detailed research studies of the behaviour of embankment dams, has given designers sufficient information to enable of the design of safe tailings dams. Major differences between embankment dams built to retain water reservoirs, and tailings dams, are that the embankment dams are designed by specialist consultants, who supervise their construction, certify their correct completion and supervise the first filling of the reservoirs. Subsequently in Britain and in many other countries, dams by law, are continuously supervised to check on their continuing satisfactory behaviour. This is done with the aid of instrumentation, installed in the dams during their construction, as well as visual observations and checks on valves, spillways and other auxiliary works. Partly because of the slow rate of construction this approach is not used with tailings dams, and many of their failures are caused by lack of attention to detail. Often there appears to be no responsible person in full charge of the tailings dams, and it is unusual for them to be well instrumented.

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In many very advanced countries, such as those of North America, tailings dams are designed, constructed and operated under similar regulations and reviews as water storage dams. A difference between the two types relate to physical loading from: (1) the staged manner in which tailings dams are typically constructed, and (2) the progressively increasing loading of the tailings on the impoundment foundation with time.

- Lack of control of the hydrological regime is one of the most common causes of failure. Of the cases reported here, the majority of failures were due to overtopping, slope instability, seepage and erosion; all caused by a lack of control of the water balance within the impoundments. Correctly placed piezometers and open tube standpipes can show the levels of the phreatic surface and give warning of dangerously high conditions. There should always be provision for diverting water and tailings discharge away from an impoundment in difficulties. Alternative discharge, possibly into another impoundment, should always be available. Removal of water from the pond should be an uninterrupted continuous process and blockage or damage to pump barge or any form of decant should not be allowed to occur. Damage to vertical decants can be caused by consolidation of the tailings and negative skin friction inducing high vertical loading. Ice, attaching itself to a decant tower can impose damaging bending and twisting moments caused by water level changes and wind forces. The ever increasing loading on culverts passing under a dam and impoundment, particularly when height has been increased to give above design capacity, crushing damage may prevent adequate discharge, particularly during exceptional conditions. Unsatisfactory amendments to the designed outlet arrangements, as occurred with Incident No.117, should not be allowed. Second decant facilities and/or standby pump barges should be available for emergencies. Any such standby facilities must be regularly tested to ensure that they will work when required. Culverts should be monitored and inspected regularly to detect first signs of distress.

  All impoundments and their retaining dams need to be able to accommodate extreme hydrologic events, up to the Probable Maximum Flood. Water retaining dams are normally provided with spillway facilities designed to pass the PMF (older dams are being modified to accord with modern law). With many tailings dams, however, the tailings fluids are not permitted to be discharged, so upstream flood waters must be fully diverted so as not to enter the impoundment and storage capacity (adequate freeboard) and careful management of tailings pond water must be sufficient to accept all flood waters falling directly on to the impoundment or entering via incorrectly diverted streams.

- Unsatisfactory foundation conditions can not always be detected by the site investigation made for the design stage, and some deep conditions have been missed, as illustrated by examples Incident No. 207 and 209. Careful measurements of movements, the use of deep inclinometers and knowledge of the pore pressure conditions in the foundation soils might be expected to show up unsatisfactory conditions, but with brittle soils there may be very little forewarning.

- Many older dams were not provided with adequate drainage and often, particularly when mine ownership has changed hands, records of design assumptions are not available. In these circumstances it is advisable for the condition of the dam to be determined by a full inspection and site investigation; when absent, instrumentation can be installed, so that the behaviour of the dam can be observed during its continuing use and raising. Sometimes additional drainage can be obtained by the use of horizontal boring to install tube-well type filtered pipe drains.

- Remedial measures, as may be derived from the above examples of unsatisfactory behaviour, include toe weighting with rockfill placed over correctly designed filter layers, over the downstream slope to improve stability. When there is time, and imminent slope failure is not expected, stability can be improved by drainage. Horizontal drains can be installed from the downstream slope, and the foundation can be partially drained with vertical drains installed through the lower part of the downstream slope.

- Many of the cases represent a lack of care. In retrospect the actions that were the cause of failure were due to a lack of appreciation of the mechanisms that trigger failure. Considering the cost to the mine owners of these failures, it might be expected that a much greater care would be taken of the tailings dams. Having someone in charge of the dams at all time, supported by good instrumentation and regular inspection and review would be a minimum requirement. The cost of insurance against dam failure and its consequences must be extremely high. The action of the insurance companies in demanding correct control of tailings dams, with the incentive of lower premiums for those mine owners who establish a satisfactory approach to their tailings dams may be a way towards a reduction in the numbers of failures that continue to occur.

  It should be noted that in North America and elsewhere, a tailings impoundment cannot start operation without prior approval from regulatory agencies who typically require both an internal technical review as well as a public review of the design, operation and reclamation plans for the impoundment. A reclamation bond placed with the regulatory agency by the mining company sufficient to cover the cost of site reclamation is also required. Large projects outside North America seeking external financing, require approval of the technical aspects of the design, as well as approval of construction, operation and reclamation.

6.2 SPECIFIC CONSIDERATIONS, WITH EXAMPLES OF INCIDENTS

6.2.1 Site selection and investigation

The position of the mine is fixed by the position of the ore body, but the position for the impoundment may be controlled by several factors including environmental considerations, local regulations, consideration of the local hydrological and seismic conditions, as well as the geographical and geotechnical conditions.

Published papers and a book about site selection methods include Robinson et al (1980), who provide a qualitative and semi-quantitative method; Robinson and Moss (1981) describe the use of these methods for several mill sites, while Keeney (1980) deals with site selection for power plants and other civil works. A successful example of the site selection process for mine facilities has been documented by Crouch and Poulter (1983). Comprehensive coverage of site selection and site investigation has been given by Clayton et al (1982).

Consideration of the position for a new impoundment must take account of the risk of damage to life, property and the environment should failure occur both during operation and after closure. It could be argued that the site for the tailings dams at Stava, Incident No 117, had been badly chosen in view of the vulnerability of the downstream town and hotels. The Merriespruit tailings dam, Incident No.202, had been sited very close to an existing township on ground sloping down into the town, apparently with no regard to the risk it imposed.

Tailings can usually be transported over considerable distances relatively cheaply, so that the choice of site may not be as limited as might at first appear to be the case. This can result in greater freedom to select a site which is relatively free of constrains and where the consequences of failure can be reduced considerably. There may also be benefits to the mining operation as a whole and not just to that element related to tailings disposal.

Example. As an example, rock containing copper is mined high in the Andes in Chile, and many mines crushed, ground and processed this ore near the mine. Lorries brought the copper concentrate down the narrow mountain tracks, distances exceeding 160 km to smelters or for export. In winter they could be held up for long periods. The narrow, steep sided valleys near the mines were used for the impoundments, but fairly high dams were needed for the storage of appreciable volumes of tailings, and there was a growing risk of downstream damage if the dams should fail. Much better sites for large impoundments could be found on the flatter land at the foot of the Andes where the rainfall was less and the rate of evaporation greater than in the mountains. In some cases low hills could be joined by tailings dams to enclose large areas.

The ore was still crushed and ground at the mine, ready for processing, but was then carried by pipeline or concrete flumes, distances exceeding 80 km, down to the flatter land, where the processing plant was constructed adjacent to the site for the impoundment. In this way the distance that the concentrate had to be carried was greatly reduced, could be continued though the winter, and tailings storage was greatly simplified, with provision for much greater storage than had been possible adjacent to the mines. The better climate at the lower levels was also an advantage for the construction of the tailings dams.

Example. At the McLaughlin mine in North America, the processing plant and tailings impoundment were sited approximately 7 km from the mine. This was done so that the tailings impoundment was in a location with favourable topography and founded on clayey subsoils for tailings solution containment. The ore was crushed and ground at the mine and conveyed as a slurry by pipeline to the processing plant and tailings impoundment. The site selection process for the project is described by Crouch and Poulter (1983).

Before the proposed site for a tailings dam can be confirmed, it is essential that the site is thoroughly investigated to check on predicted seismic activity and, if the impoundment is to be situated within a river valley, to determine the Probable Maximum Flood.

<u>Example</u>. The Sarcheshmeh tailings impoundment situated in a valley 20 km north of the Sarcheshmeh Copper Plant of Iran, was mainly constructed for the purpose of storing the tailings from the copper plant, as well as providing a source of water for the plant. Inaccuracy in the estimated total annual discharge of the river made during the feasibility studies has resulted in a change of use for the impoundment. Instead of the retaining dam being considered as a tailings dam, it has had to become a water reservoir, and this change has posed various problems. The actual flow during the first 7 years of operation has been more than 2 ½ time the estimate and this has necessitated release of water downstream, despite some environmental limitations. This, combined with the continuing output of tailings from the plant, has necessitated raising the heights of both the main and saddle dams by 15 m. Details have been given by Askari (1991) and Askari et al (1994).

Information about the geotechnical properties of the foundation are an essential pre-requisite for the design of the dam.

<u>Example</u>. Ok Ma, Papua New Guinea. A major landslide occurred on the left bank of the river Ma during early stages of the construction of an embankment dam intended for tailings retention. The site was in one of the most favourable valleys hydrologically in the area, even though it was about 18 km from the processing plant. According to Fookes and Dale (1992) the site investigation had been made by well known site investigation specialist companies over a period of six months. 23 boreholes were made, together with 3 test pits and a geophysical survey using 20 seismic refraction lines. During the period there were 10 professional geologists and geotechnical staff on the site.

The early stages of construction involved excavation of the colluvial and taluvial materials from the base of the valley to obtain an effective cut-off. Due to concern about the residual shear strength of the Pnyang Mudstones, further investigation was to proceed in parallel with the excavation and 44 more boreholes were drilled. During this time a highly professional review team made visits to the site on behalf of the Government. Their reports, although expressing concern about problems associated with the excavation, gave no indication of any concern about the possibility of the major landslide that occurred.

The main site investigation began in March 1982 but prior to completion of an access road in September 1983, access to the site had been by long treks through the jungle or by helicopter. Excavation at the left abutment had reached a depth of 14 m when a landslide developed on 16th December 1983, moving 6 m towards the river and involving an area of 11 ha. Shortly afterwards, during the night of 6th-7th January 1984 a much larger landslide occurred covering an area of 122 ha, resulting in the abandonment of the site.

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This example is given to demonstrate the very great difficulty in making accurate predictions about the behaviour of a site. Fortunately the majority of sites chosen for the storage of waste tailings and the construction of tailings dams do not involve such complications.

Sinkholes, old mine shafts and weaknesses above active underground mine workings are very difficult to detect during site investigations. Karstic foundation bedrock can offer the opportunity for the surface overburden to collapse when its bridging action over a cavity is broken by increases of stress or softening due to increased water that could be introduced through the presence of a tailings dam and impoundment.

Example. Londe (1976) described a case of rotational failure of a part of the toe of a tailings dam caused by collapse into an old and disused mine shaft that had not been effectively sealed when abandoned.

Example. Incident No. 73. The failure of the Iwiny tailings dam in Poland, described by Wolski (1996), was thought to have been caused by a sink hole forming underneath the dam due to loss of ground into a highly fractured and faulted red sandstone bedrock. The dam had been built across a fault with a fracture zone 20 m thick, and it was here that failure occurred. The underground mine had advanced to within 200 m of the dam axis, pumping from it had lowered the water table and rock blasting in the mine caused small earth tremors. The failure, during the night of 13th December 1967, released about 4.6 x 10^6 m^3 of tailings, more than 20% of the stored volume, causing 18 deaths.

Example. Incident No.88. Mufilira Mine, Zambia. The depression formed in the ground surface by the extraction of materials from the underground mine was being used as an apparently convenient storage basin for tailings. Under the weight of the tailings and the increased water soaking down from the impoundment, on 25th September 1970 the ground gave way, releasing a large volume of liquefied tailings into the mine workings. 68,000 m^3 of tailings funnelled down into the mine workings 600 m below in 15 minutes, killing 89 miners. The remaining mass was stabilised by de-watering with tube wells to enable mining to continue.

6.2.2 Starter dam

Starter dams are commonly constructed from locally obtained fill, that may be soil and/or rock, including coarse waste and overburden from the mining operation if that is close enough to the tailings dam site. Design may require the dam to function as a drain or to form an impervious barrier and the fill chosen should reflect these requirements. If acting as a drain, pore water from the impounded tailings passing into the starter dam should be collected in a seepage collection pond outside the toe of the embankment and pumped back into the tailings pool or directly back to the processing plant. When the pore water from the tailings is toxic, containing such chemicals as cyanide, regulations may require the whole impoundment to be lined with clay or a synthetic liner, when a drainage blanket would be placed between the liner and the impounded tailings to promote drainage from the tailings. Seepage from the impoundment directly into the ground would only be allowed if the pore fluid did not affect the quality of the groundwater. Permitted seepage from the tailings would either have an initial water quality similar to that of the groundwater or have mobile constituents removed from solution by attenuation in the soils between the tailings and the groundwater. In practice such an ideal approach can not always be applied.

Example. A tailings impoundment was needed to accept tailings from a rock washing plant. A site was chosen where there was a 20 m high mound of refuse consisting mainly of stripped overburden, that could act as a starter dam. The land sloped gently towards this mound that was parallel to a double track railway line. A site investigation showed that the mound was not homogeneous and it was difficult to define representative soil parameters. In view of this, conservative design parameters were derived from a back analysis of the existing situation. In considering the stability of the whole scheme when tailings would be placed behind this starter dam, and then raised by the upstream method, the position of the phreatic surface, which was almost impossible to predict, was crucial. A stability analysis was used to determine a maximum height position for a developing phreatic surface, and to measure this a large number of standpipe piezometers were installed.

The impoundment was put into operation and filled with tailings to the crest level of the mound, when the first of the supplementary dykes was built above and upstream of the crest and the level of the tailings continued to rise. Standpipe readings were taken weekly and the phreatic surface remained comfortably low. But suddenly, between two of the weekly readings, the level came up by several metres. This caused the downstream slope of the mound to begin moving downstream towards the railway line. Because of the risk of a sudden slide, the railway was closed and some people living nearby were evacuated. Large cracks appeared in the downstream slope as the toe continued to move towards the railway, as indicated by Fig.12. The movement was arrested by removing the upper part of the supplementary dyke, loading the toe with 10 000 tonnes of rockfill and drawing water from the moving mound and from the tailings immediately upstream of it. The subsequent study revealed a layer of rock fragments within the mound. Water from the tailings had reached the level of the upstream edge of this layer, permitting the rapid entry of water into the body of the mound.

Problems can arise when the initial starter dam is not constructed to a sufficient height. The capacity that will be impounded by a starter dam of a given height must be balanced against the predicted rate of tailings delivery to the new impoundment to ensure that the expected rate of rise of dam height by the method of construction that will be used, will enable the storage capacity of the impoundment to increase at or above the expected rate of tailings delivery. It should be noted that because the rate of delivery of suitable tailings for dam construction from a new mine cannot be relied upon, the initial starter dam height should be made greater than might be expected.

6.2.3 Unsatisfactory foundations

This is allied to site investigation because the weaknesses revealed by the dam failures should have been detected during the site investigation. This does not necessarily imply that the site investigation was inadequate, but the dam designer may not have made the correct assumptions from the site investigation reports. Karl Terzaghi's biography [Goodman 1999] contains numerous examples of how he evaluated foundation problems from a detailed engineering and geomorphological standpoint in ways that previous engineers had overlooked.

Example. Incident No. 187. Prior to the construction of this dam the surface layers of clayey soil had not been stripped. The result was that the base of the dam slid forwards, causing failure.

Example. Incident No. 183. The 8 m high tailings dam had been built on gently sloping ground on a clay stratum about 6 m thick overlying a shale/mudstone bedrock. Its impoundment had been filled completely 8 years before it failed, releasing a flow of liquified tailings that travelled 100 m, covering a main road to a depth of 3 m. The failure was attributed to artesian water pressure in the bedrock developed by the seepage of water from other impoundments further up the slope, combined with tensile strains induced in the clay stratum by old underground mine workings. A line of relief wells was installed to control the groundwater pressures in the area.

Example. Incident No. 68. A clay layer in the foundation of this dam sheared when the dam reached a height of 79 m. This caused a 240 m long section of the dam to slump. It was stabilised by the use of rock drains and toe weighting.

Example. Incident No. 207. This dam was founded on about 50 m depth of material overlying lava flows. There appears to have been a layer of weaker material lying over the lava flows, and when the dam had reached a height of 25-30 m, the dam slid forwards on this deep layer. The dam was stabilised by draining the layer into excavated tunnels from the existing underground mine workings, and constructing a rock buttress at the dam toe that would act as toe weighting.

Example. Incident No. 209. The Aznalcóllar tailings dam at the Los Frailes mine near Seville in Spain, begun in 1978, failed in April 1998 when it had reached a height of 27 m. A length of the dam of about 600 m swung forwards like a door, forming a breach about 45 m wide. The dam section remaining intact, releasing an estimated 5.5×10^6 m^3 of acidic tailings that flooded over an area of approximately 2.6×10^3 hectares of farmland. The dam was founded on about 4 m thickness of alluvium, overlying marl which may have contained pre-formed slip surfaces, developed during earlier geological conditions. The impoundment, about 2 km long and 1.2 km wide, was along one side of the flat valley of the river Agrio. The tailings in the impoundment was particularly heavy, with a bulk density of 28 kN/m^3; almost three times the weight of water.

The failure occurred where a dividing dam met the main dam. In 1995 some grouting had been carried out from the main dam crest to reduce leakage. 42 relief wells were placed towards the end of 1997 along the downstream toe, through the alluvium and about 1-1 ½ m into the marl, and for several months they collected 1000 m^3/hr that was pumped back.

There were no eye witnesses because failure developed during the night, but there was some evidence to suggest that the main dam failed north of the dividing dam, prior to the bodily movement of the main dam adjacent to and south of the dividing dam, on a shear plane deep in the marl, as indicated by Fig.13. In addition to the physical conditions at the breach, the river flow measurements showed two peaks, one at 03.25 hours with a second at 08.30, indicating that failure occurred in two stages. Subsequently piezometric heads in the marl were found to be above dam crest, as indicated by Fig.13. This may have been due to the high density of the impounded tailings producing construction pore pressures in the heavily overconsolidated marl that had migrated horizontally towards the river and come underneath the dam. Deep boreholes have disclosed an aquifer 80 m below the dam, showing a head of 80m, that would have restricted drainage from the marl. According to Eriksson and Adamek (2000) the cause of the failure was a fault in the marl 14 m below ground surface. Bodily movement of the marl, carrying with it the intact dam towards the river, would be assisted by the reduction of effective stresses on the horizontal feature caused by the high pore pressure. Several eminent geotechnical engineers have expressed the view that such behaviour would have been extremely difficult to predict.

Example. Fort Peck dam. This 76 m high hydraulic fill dam on a sand foundation overlying weathered shale bedrock containing seams of bentonite, was in effect a tailings lagoon between two shoulders of coarser hydraulic fill. It failed two weeks after it had reached full height, on 22nd September 1938, producing a flow slide. 8 million m^3 of fill and foundation sands moved out a maximum distance of 430 m in 3 minutes on a level surface and 80 men were lost. As with the Aznalcóllar tailings dam at the Los Frailes mine, Incident No 209, a massive section of the dam swung out upstream like a great earthen gate hinged on the east abutment. The failure was attributed to the fact that the shearing resistance of the weathered shale and bentonite seams was insufficient to withstand the shearing forces imposed by the spreading action of the dam. This case is given as an example because it has been extensively studied and has become a classic. It has been described by Casagrande (1965, 1971).

6.2.4 Lack of stability of the downstream slope

The dangers of allowing the phreatic surface to move so far downstream as to intersect the downstream slope of a dam are very well known. Some of the earliest work during the 1930s of the United States Bureau of Reclamation in instrumenting their dams was for the purpose of finding the position of the phreatic surface. They installed standpipe piezometers in homogeneous dams as soon as construction was complete so that the rise of phreatic surface during reservoir impounding could be monitored. In some dams they were surprised to find that water rose in the standpipes before water had been put in the reservoir. This was due to construction pore pressures, created by the weight of new fill which compressed the lower fill and built up a pressure in the water trapped in the pores of the soil. The rate of construction increased the weight faster than the pore water could escape from the soil. At the end of construction, the continuing escape of pore water from the fill lowered the pore pressure, but the rise of water in the reservoir could increase it again.

The same thing happens in a tailings dam, except that usually the rate of construction raises the vertical height sufficiently slowly so that construction pore pressures can dissipate, and appreciable pressures seldom develop within the dam itself. Pore pressures are raised however by the rising impoundment, which usually rises at the same rate as dam construction; ie the dam is raised only to keep a freeboard distance above the level of the rising impoundment. A normal situation, with a dam being built by the upstream method, is indicated by Fig.14. By maintaining a wide beach, the pool of visible water is kept well away from the longitudinal axis and crest of the dam and there is more opportunity for the coarse fractions from the tailings to settle out on to what is becoming the body of the dam; only the finest fraction being carried into the settling pool. The rate of settlement depends on size of particle, its specific gravity, and its activity: an assessment can be obtained from Stoke's Law. The rate of clearance, i.e. the time for the smallest particles to sink below the surface to leave some clear water, can be so small that a large area of pond is needed to cater for the volume of tailings being discharged into the impoundment, and a compromise has to be reached between the area of the surface of the pond, and its closeness to the dam crest.

Trouble can arise when pond level rises, saturating the beach and bringing the edge of the open water closer to the dam axis. When rain erosion has cut gullies in the downstream slope of the dam, the phreatic surface, pushed downstream by the advancing pond, can reach the deepest of these gullies, causing water to issue into the base of the gully. This is an extremely dangerous situation because the issuing water loosens and carries away material from the dam slope, thereby steepening it locally until a small rotational slip occurs, bringing more material down into the gully to be washed away by the flow of water which increases as the effective slope of the dam is moved ever further back below the position for the phreatic surface. If this behaviour continues for too long, unobserved, larger and larger rotational slips occur, endangering the stability of the whole dam. Kealy and Busch (1979) analysed the effect of a high phreatic surface using circular slip surfaces as a simple illustration of the effect described above. They showed that the factor of safety against the occurrence of a rotational slip fell from 2.6 to only 1.1 when the phreatic surface reached the downstream slope of their dam.

The phreatic surface can be moved back from the slope to improve stability by the installation of horizontal bored drains. The California Division of Highways has been using such horizontal drains since 1939, according to Smith and Stafford (1957). They drilled holes near the base of the slope with a slight upward inclination so that water could flow out by gravity. Holes of 80 to 150 mm diameter were drilled then fitted with perforated metal pipes. In some cases the hole collapsed before the pipes could be inserted. Currently slotted PVC pipes are installed with the aid of a hollow stem continuous flight auger, which acts as a casing while the slotted drain pipes are inserted. Another method uses expendable fishtail bits on 75 mm hollow drill rods, using flush water. After the hole is drilled, the drain pipe is inserted and the fishtail bit dropped off so that the drill rods can be removed. The optimum length and spacing for the drains to lower the phreatic surface below the failure surface has been determined by Kenny et al (1976). As a remedial measure, additional drainage has been installed in the form of pumped vertical wells (Incident No.25), but the installation of horizontally bored under drainage, although clearly attractive, has not yet been applied extensively to tailings dams.

<u>Example.</u> Incident Nos. 124 and 125. The British Clean Waters Acts required coal mines to collect the waste fines previously discharged from their processing plants into rivers. At the Ty Mawr colliery in South Wales lagoons were formed in the existing dumps of coarse discard on the valley side, without any true design considerations. The tailings was pumped up to these lagoons in the expectation that surplus water would seep into the coarse discard. The downslope bank of the first of these lagoons

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ruptured in December 1961, and the released flow damaged an overhead ropeway that carried the coarse discard to the mountainside tips. A second lagoon, built in the same way, failed in March 1965, releasing a greater volume of tailings that reached the colliery yard, damaging cars and almost going down the shaft. These incidents are illustrated by Fig.15.

Example. Incident No. 184. A 15 m high tailings dam at the Zletovo lead mine in Yugoslavia failed in March 1976 due to a high phreatic surface reaching its steep downstream slope of 1 on 1.5. Prior to failure there were signs of leakage issuing from the slope quite high up. The flow of released tailings contaminated the nearby river and the water supply from this river to the town of Stip had to be discontinued for more than 24 hours.

Example. Incident No.7. Bafokeng, 1974. 3 million m^3 of liquefied tailings escaped and travelled 45 km. The flow destroyed mine buildings, some went down the shaft carrying with it the cage and killing 12 miners. The case has been described by Jennings (1979).

Example. Incident No.25. Castle Dome, USA, 1950. Sand dyke failed due to excessive seepage and high phreatic conditions. Pumped vertical wells were used to drain the sand and reduce the height of the phreatic surface.

Example. Incident No. 202. Merriespruit, Virginia, South Africa. This 31 m high tailings dam had been constructed just up slope of the township of Merriespruit. Dampness on the downstream slope and some small slips had caused the impoundment to be closed. It continued to be used for the occasional discharge of waste water containing some tailings. This continued for some time, the extra tailings slowly pushing the pond further towards the dam and reducing the freeboard. The movement of the pond was recorded by satellite imagery (by chance a satellite passed over the site on a regular basis) and it was seen that the decant became isolated so that no further discharge could come from the pond. A rainstorm when 30 to 55 mm fell in ½ hour was the last straw and the dam failed during the evening of 22nd February 1994. A description of this failure was given by Blight (1997b).

Example. Incident No. 213. Fernandinho tailings dam, Brazil. This 40 m high dam failed very suddenly. The failure was blamed on the very steep downstream slope of 1 on 1.11 combined with high pore pressures within the dam. This case is mentioned under “6.2.7. Flow slides” below.

Example. Incident No. 214. Minera Serra Grande tailings dam, Brazil. The impoundment was formed in a natural valley 3 km from the processing plant, retained by the dam constructed from cycloned tailings. Construction began in 1989, and the downstream slope of the tailings dam suffered a major slip on 26th February 1994. The slip did not sever the dam crest so there was no release of tailings, but the mining operation was suspended until the safety and integrity of the dam was re-established. Emergency repairs took 3 weeks, resulting in a revenue loss equivalent to 8500 ozs of gold.

The failure was attributed to a rise of the phreatic surface caused by badly constructed and ineffective filter drains under the starter dam, exacerbated by heavy rainfall during late 1993 and early 1994. The 27m high starter dam had been made impervious through being constructed from unsuitable fill which, in an attempt to construct a strong dam, had been well compacted. Mistakenly following the practice used for water retaining dams, a grout curtain had been constructed under the starter dam, preventing drainage of pore water. Contributory factors were that the design and operating manuals had not been developed so that there was no short or long term strategy. Operating staff were inexperienced in the adopted method of dam construction and thus were unable to interpret the warning signs that were being given by the piezometer readings. Initial construction was supposed to be by downstream cycloning from the crest of the starter dam. Unfortunately the cyclones did not arrive until 8 months after operations began and meanwhile tailings had been discharged freely into the impoundment and due to operator inexperience, freeboard was rapidly lost, causing the starter dam to be raised by 3 m in July 1991. The method of deposition was changed to the upstream method in September 1991 thereby restoring freeboard. During 1993 the outward appearance of the tailings dam looked reasonably healthy, aside from the common problem of too much water in the pool. There was minimum seepage from the downstream slope; also minimum seepage from the filter drains which should have been discharging freely. The rising phreatic surface rose above the crest level of the impervious starter dam and then exited from the slope above it, resulting in the large slip.

Example: Incident No.54. Grootvlei gold mine, South Africa, 1956. Downstream slope failure occurred after a prolonged period of rain, when the pond spread over the tailings beach and encroached on the dam crest. The dam breached releasing a flowslide of tailings that carried away about a third of the impoundment contents.

Example: Incident No.62. Kennecott copper mine, USA, 1941. Rainfall was thought to have increased the saturation of the downstream slope, causing minor rotational slips which initiated breaching and released tailings from the impoundment.

Example: Incident No.100. Ray copper mine, USA, 1972, 52 m high dam. Slope instability over a 150 m length of the dam was thought to have been associated with a perched water table above a layer of slimes of low permeability deposited on the dam 20 years earlier. A wetted zone had been observed on the downstream slope for some time before a rotational slip occurred, leading to a breach. Released tailings covered a section of an adjacent railway.

Example: Incident No.185. Arcturus tailings impoundment, Zimbabwe. This rectangular impoundment , 310 x 150 m serving a gold mine and in operation, was retained by dams that had been built by the upstream method from the coarse fraction of the tailings obtained by the usual beach separation. It had been built over a very slight ridge that ran under the long direction on the east side so that the land sloped at about 1 on 10 towards the west side, causing base drainage to flow to the dam forming that side. There appeared to be no under-drainage layer. The downstream slopes were 1 on 1.3 to 1 on 1.1 (vert on hor). There were two decant towers each 448 mm diameter, on the central axis of the impoundment and about 100 m apart, with the surface of the tailings impoundment sloping uniformly towards them with a fall of about 1.5 m from the beach. Six years prior to the failure a large arcuate scar 60 m wide, 1.5 m across and 10 m deep, formed in the slope of the west dam at about mid length, suggesting the beginning of a rotational slip. It was stabilised with 1040 tons of rock waste. A week before failure, cavities developed in the west dam and were filled with waste rock. There was evidence that the surface of the impoundment was beginning to slope towards the west side and to keep pond water away from the dam crest, four days before failure, a low bund was built parallel to and 17 m east of the west dam crest. Delivery pipes were moved to enable spigotting from the bund. During these last four days, following a period of heavy rain, 183 mm of rain fell and was trapped between the dam crest and the new bund. Sink holes developed in this 17 m wide strip. At 19.30 on 31st January 1978 with a reported loud bang, the west dam failed where the scar had formed six years earlier, releasing a flow slide that formed a breach 55 m wide in the dam.

Boreholes put down through the 25 m height of the impoundment just upstream of the dam, revealed tailings 16 to 20 m deep with about 70% silt and 8% clay, at a voids ratio of 0.96 to 1.05. Attempts had been made to improve drainage by driving steel pipes into the dam during the two or three years before the failure. It is possible that as a remedial measure, provision of horizontally bored drains through the toe of the dam and into the bottom of the impounded tailings might have lowered the phreatic surface sufficiently to have prevented the formation of the large rotational slip.

Further details of this event have been given by Shakesby and Whitlow (1991). In general, preventative measures consist of proper tailings water management. Remedial measures include reducing the water level in the impoundment, placing free draining, coarse material as toe weighting over the lower part of the downstream slope, and improving drainage. This has been done in some cases by installing horizontal drains into the toe of the slope.

Advice on drainage is given by ICOLD Bulletin No 97. Original design may have included drainage and drains of various kinds may have been built into the dam body during construction. Choking or other malfunction of these drains can seriously affect the position of the phreatic surface, and could endanger stability. Blockage can be caused by inadequate filters around the drains, allowing fine tailings to get into the drain. Drains that exit to the atmosphere and carry relatively small flows can allow chemical oxidation of the draining fluids, producing solid precipitates that can slowly block the drains with a well cemented hard material that cannot be cleaned away by surging. To avoid oxidation, a drain can be kept full of water by placing a small weir at its outlet, raising the water level around the drain by a small amount; insufficient to affect the draining function of the drain.

Dagenais (1976) warned that lack of drainage is the main cause of failures and requires great attention. ICOLD Bulletin 97 concludes, 'Careful and correct drain construction during the early stages of dam construction constitutes a cheap insurance against future expensive remedial work'.

Consideration of the Minera Serra Grande tailings dam failure concluded that a tailings dam must be recognised as a sensitive structure involving high risk to life, the environment, property and profitability of the company (the failure resulted in a revenue loss equivalent to 8500 ozs of gold). Consequently, tailings disposal must be subjected to the appropriate levels of design, management and supervision in relation to the risks. Specifically, the design should be inclusive of the conceptual and feasibility detail, construction and operating procedure and closure aspects. No phase can be done in isolation of the others and continuity and responsibility must be maintained through the entire process. The operating procedures manual is the vital link between design and operation and must be generated together with the necessary pre-operational programs for operating personnel.

Example. Incident No.34. Cyprus Thompson Creek, USA, 1989. An original drain included a 15 cm diameter PVC pipe wrapped in filter cloth. Fine tailings were seen to be being discharged from the drain, and a sinkhole of 2.5 m diameter and 1.2 m depth developed in the downstream slope. It was assumed that the filter cloth protection had failed, allowing piping into the drain.

Example. Incident No.154. Gypsum tailings impoundment, USA, 1966. Dam breached due to choking of the under-drains.

Preventative measures include careful drain and filter material design, sizing of drains and filters with adequate factors of safety, and proper water management during operations.

Lack of stability can also be caused by a variety of conditions in addition to those of the phreatic surface reaching the downstream slope, as described above; which is perhaps the most common cause of instability that can lead to dam failure. Excessive height at a given slope angle, exacerbated by gullying and high rainfall, can cause local slope failures which if repaired in time, need not lead to dam failure in the sense of the formation of a breach. Toe erosion causes local steepening and can result in local rotational slips. Freezing of the downstream slope can prevent evaporation from the slope and so increase pore pressures within the body of the dam. Prolonged freezing can also cause ice lenses to form, drawing pore water towards the surface of the slope. During thaw the released water can initiate slope instability.

The following examples relate to toe erosion.

Example. Lead and zinc mine at Mojkovac, Montenegro, 1992. Tailings retained by a 20 m high embankment dam built alongside the Tara river, a tributary of the Drina which in turn, joins the Danube. The dam of fill consisting of a clay with sand and gravel, had been covered on the upstream side by plastic sheeting to prevent water from the tailings from polluting the river. Floods raised river level 3 m, bringing it to the dam toe where sufficient erosion occurred to cause a rotational slip that reduced the 3 m wide crest to only 1 m. The plastic sheeting prevented water from the impoundment from entering the fill and the dam did not breach. Remedial work consisted of diverting the river to a course away from the dam toe, and rebuilding the downstream shoulder to a flatter slope.

Example. Incident No.95. Pinchi Lake, British Columbia, 1971. A 13 m high dam of homogeneous section built of compacted glacial till. Water decanted from the impoundment flowed in an unlined channel parallel with the downstream toe. Erosion of the channel produced downcutting of 4 m, causing cracking and deformation of the downstream slope of the dam. Movements were seated within the lacustrine foundation sediments at the depth of the eroded channel. Movements were stabilised by construction of a toe weighting berm and relocation of the channel.

6.2.5 Superimposed loads

Loads can be added to dams retaining tailings by increasing the height of the dam, retaining the same slope angle, and by adding materials to the surface of the tailings impoundment.

Example. Incident No.72. Lower Indian Creek lead mine, USA, 1960. Earthfill dam built in 1953 to a height of 14 m and subsequently raised several times by the addition of earthfill. In 1960 slumping occurred in the 1 on 2 downstream slope. The dam was saved by the addition of rockfill toe weighting, placed with an outer slope of 1 on 3. The dam remained in service and was raised between 1971 to 1976 with cycloned sand fill, ultimately reaching a height of 25.3 m.

Example. Incident No.75. Maggie Pye china clay, UK, 1970. Dam 18 m high suffered slope failure immediately after completion of a dyke to raise the dam, following a period of heavy rain. High pore pressures, together with the added weight of the new fill, were thought to be the causes for the failure. 15 000 m^3 of tailings were released.

Example. Incident No. 206. Failure of Manila Mining Corporation's Tailing Pond No 5 - Philippines. Failed into sea, extending 200 m seawards, at 09.30 on 2nd Sept 1995, about 50,000 cubic metres of material was released. 17 people were working on the tip at the time, and a farmer with his wife were walking along the shore. Of these 12 were killed, including MMC's Environmental Inspector, Nelson Cayomo, whose daily inspection reports of the impoundments indicated no signs of failure. The farmer was killed but his wife was saved. Lots of heavy plant lost too. Dam was built 1985/6 by the shoreline of Placer Bay. On 21st Dec 1986, Typhoon Ameng washed away a portion of the dam at the seafront. Another collapse occurred on 9th July 1987, both incidents releasing effluent with high levels of cyanide resulting in fishkill. The impoundment had been filled to capacity by July 1995, when dam crest was 17 m above sea level. The crest was about 10 m wide and was used as a two way road for heavy plant. The closed impoundment began being used as a dump for mine waste rock, and at collapse contained more than a million cubic metres of mine waste, earth, boulders, rock and leach pad debris plus seven 10 wheel trucks, 32 dump trucks, 3 dozers, 1 loader and a land cruiser (carrying the inspector). Failure was thought to be due to high rainfall raising the phreatic level, but the toe of the dam was over reclaimed land, and the breached portion coincides with the former shoreline.

6.2.6 Problems with decants

Water ponded in a tailings impoundment is removed by evaporation, pumping from a floating barge, or decanting into a tower that exits the impoundment through a culvert or pipe beneath the tailings dam.

One of the most common causes of unintended dangerous rises of pond water levels is inadequate behaviour of decants. This may be produced by debris blockage, crushing and/or fracture of the outlet passing under the dam, or by unanticipated flood.

Damage to decant towers caused by ice are discussed in Section 6.2.9 below.

Example. Incident No.23. Casapalca, Peru. Several tailings dams built by upstream method, up to 107m high used a complex array of pipe type decant structures and inadequately sized stream bypass channels. Five separate dam failures resulted from failures in these systems.

Example. Incident No.16. Blackbird, USA. Cobalt mine. Metal culvert under the dam corroded and partially collapsed. Suspended tailings discharged into downstream drainage. No embankment breach.

Example. Incident No.49. Galena silver mine, USA, 1974. Three tailings impoundments had been built in sequence within a narrow valley. During a rain on snow event that caused a 100 year flood, a blockage diverted a large portion of the flood into the uppermost impoundment. Its decant could not accept this large flow, causing the upper dam to fail by overtopping, leading to a cascade failure of the others. Released tailings covered about 5 acres of land, including part of a highway and main line railway.

Example. Incident No.119. Sweeney, USA, 1980. The dam was breached due to piping around the decant outlet conduit.

Example. Incident No.117. Stava, Italy, 1985. Two tailings dams were built for a fluorite mine in the mountains of northern Italy. They were built one above the other on sloping ground formed by fluvio glacial sediments, using starter dams with construction at later stages by the upstream method. The tailings contained sufficient coarse, angular sand to enable the downstream slope to be built at an inclination of 1 on 1.2. Special drainage for natural runoff at the site was not provided. Concrete pipes were laid on the ground to be under the lower impoundment, encased in concrete to form a square section. Upward facing openings were made in each length of pipe to act as decants for tailings water, and as the height of the impounded tailings rose, one by one these opening were closed by concrete plugs. When the first dam reached a height of 26 m, a second dam was begun upstream of the first impoundment, with the starter dam founded on natural ground at the limit of the existing impoundment. Similar encased concrete pipes were laid under the footprint for the starter dam, and continued up the slope to act as decants for the second impoundment. But when the second impoundment had reached a certain level, a blockage occurred in the decant concrete pipe. The blocked length was by-passed with a steel pipe laid on the surface of the tailings and connected to the concrete pipe through a small vertical tower, as indicated by Fig.16. When the second dam reached a height of 29 m it suffered a rotational slip and breached. The released tailings produced the failure of the lower dam. The combined contents of the two impoundments flowed at speeds up to 60 km/hour sweeping away the village of Stava with its several hotels and engulfed part of the small town of Tesero, about 4km downstream. 269 people were killed.

Six months prior to this failure in July 1985, a small slip had occurred in the lower portion of the upper dam in the area where the decant pipe passed underneath, as a consequence of freezing, and water was observed to seep from the area until March. In June a large sink hole developed in the lower impoundment due to failure of the decant pipe and tailings slurry from the sink hole flowed out to the river Stava. Water levels in both impoundments were lowered so that repairs could be carried out. Only four days before the fatal failure, both ponds had been refilled. It is thought that the length of steel pipe sagged under the weight of the settling tailings and pulled out from the vertical short concrete tower, permitting the decanted water to discharge into the body of the lower part of the upper dam. The resulting rise of pore pressure caused the rotational slip and the failure. Further details of this case are given by Berti et al (1988), Chandler (1991) and Chandler & Tosatti (1995).

6.2.7 Flow slides

Failure of a tailings dam itself, while causing an inconvenience, may not have seriously damaging consequences nor cause any loss of life. The serious danger of a breach is the possibility of a subsequent flow slide of liquefied tailings. The stability of embankments against slips is controlled by the available shear strength of the fill and foundations. The shear strength of particulate materials is usually expressed simply and, for the present purpose, quite adequately by the Mohr-Coulomb relationship in terms of effective stress:

$$\tau_f = c' + \sigma' \tan \phi'$$

where τ_f and σ' are the shear strength and normal effective stress respectively on a failure surface in the material, and c' and ϕ' are the effective cohesion and effective angle of shearing resistance of the particulate material. The above equation can also be expressed as:

$$\tau_f = c' + (\sigma - u) \tan \phi'$$

which demonstrates that as the pore pressure u increases, the available shearing resistance decreases, provided the total stress does not change.

The pore pressure is often expressed as a dimensionless ratio (r_u) defined as:

$$r_u = u / \gamma z$$

where γ is the bulk unit weight of the material (soil or tailings) and z is the depth of the considered position below fill or ground surface.

A granular soil in a dense state will generally exhibit a greater maximum or peak effective stress shear strength (ϕ'_p) than the same soil in a loose condition, although this effect is suppressed at large normal stresses. When the dense granular soil is strained beyond the peak strength, there will be a fall in strength to the constant volume or critical state strength (ϕ'_{cv}). This constant volume strength is similar to the maximum strength of the soil in a loose condition where little or no post-maximum fall in strength will occur. For a granular soil, ϕ'_{cv} should be similar to the angle of repose. From a literature review, Bolton (1986) found that for sands ϕ'_{cv} ranged from 30° to 37°.

In a truly undrained condition a saturated granular material with particles in the sand and silt grain sizes i.e. material typical of tailings, behaves as a $\phi = 0$ material.

This was demonstrated by Bishop and Eldin (1950) and Penman (1953). When shearing strains are applied such material can dilate or contract, depending on its density. This means that as shear strains occur the undrained strength of the saturated material can be respectively higher or much lower than its drained shear strength. With contractive material even a small shock can trigger flow liquefaction: the shearing strains imposed on a saturated tailings impoundment caused by deformations of the retaining dam can readily result in a flow slide composed of liquified tailings. This material has the fluidity of water but is very much heavier making its destructive capacity large.

Very loose, normally consolidated saturated material as are tailings in an impoundment, exhibit a peak strength higher than the residual strength at greater strains, when tested in apparatus using strain control. This effect is shown more vividly in stress controlled tests, i.e. tests in which the shearing force is applied by weights: an example given by Castro (1969) is shown by Fig.17. The tests were made on a fine uniform sand with a D_{10} size of 0.1 mm at a relative density of 29% using a confining pressure of 400 kN/m^2. Load was applied in small increments and peak deviator stress was reached in 14 minutes at a strain of 1%, during which time the pore pressure increased to half the confining pressure. But once the peak strength had been reached the strain increased to 19% in 0.17 seconds. The pore pressure climbed to almost the value of the confining pressure and as a consequence the strength fell to almost nothing. This is the mechanism of a flow slide.

Bishop (1972) described tests simulating the conditions in a tailings impoundment when yield of the retaining dam reduces $\acute{o}_3$ while $\acute{o}_1$ remains constant. Although the average effective principal stress decreased, so did the volume and he pointed out that this behaviour gave a warning of the probability of the flow slide phenomena.

Blight (1997a) described the failures of five tailings dams in southern Africa, Bafokeng, Arcturus, Saaiplaas, Merriespruit, and Simmergo. He shows that the occurrence of a mudflow is closely associated with the condition of the ground on to which the escaping tailings move. If the ground surface is dry, it is likely that the tailings will not move far whereas if it is wet, a flowslide is much more likely to ensue.

Examples. These are numerous and result from failure of the retaining tailings dam that may be caused by inadequate shear strength or the additional loading created by earthquake shock. They include Bafokeng [Incident No.7]; Barahona [Incident No.9]; Bilbao [Incident No.15]; the El Cobre dams [Incidents Nos.43 & 45]; Kimberley [Incident No.66]; Mochikoshi [Incidents No 84 & 85]; Stava [Incident No.117]; Merrespruit [Incident No.202]; Iwiny [Incident No.73]; Fernandinho [Incident No.213].

6.2.8 Earthquakes

Dams built by the upstream method are particularly susceptible to damage by earthquake shaking. There is a general suggestion that this method of construction should not be used in areas where there is risk of earthquake. Dams built by the downstream method, in cases where there are sufficient volumes of the coarser fraction in the tailings, or those built from borrowed clayey fill as water retaining dams, are much less prone to damage by earthquake shaking. Seed (1979) said that it was noteworthy that no failures have been reported in dams built of clayey soils even under the strongest earthquake shaking conditions imaginable, and that all cases of slope failure reported have involved sandy soils. Advice on earthquake resistant design of tailings dams is given by ICOLD Bulletin No 98.

Example. Incident No.9. Barahona, Chile, 1928. 61 m high dam built by the upstream method with downstream slopes of 1 on 1. The dam failed during the Talca earthquake of magnitude 8.3, producing a breach 460 m wide. The released tailings flowed down the valley, killing 54 people.

Example. Incident No.12. Bellavista, Chile, 1965. A 20 m high dam built by the upstream method with downstream slope of 1 on 1.4, failed during the La Liqua earthquake of 7.7 magnitude. At the time only 8 m separated the edge of the pond water from the dam crest.

Example. Incident No.57. Hokkaido, Japan, 1968. A 12 m high dam built by the upstream method with a 1 on 3 downstream slope, failed during the Tokachi-Oki earthquake of 7.8 magnitude. 90 000 m^3 tailings flowed from the breach, crossing and blocking a river near the downstream toe.

Example. Incident No.84. Mochikoshi No.1, Japan, 1978. A 28 m high dam built by the upstream method with a downstream slope of 1 on 3, failed during the Izu-Oshima-Kinkai earthquake of 7.0 magnitude. 8 000 m^3 of tailings were released and reached and flowed down a river valley for 7 to 8 km, causing one fatality.

Preventative measures (as mentioned above) include dam construction with cohesive materials, provision for water drainage in the dam, and proper water management.

6.2.9 Ice and faulty water balance

Sufficient freeboard under all circumstances and all along a tailings dam is one of the most important prerequisites for safety. Tailings dams are extremely sensitive to high levels of the phreatic surface that can cause small slips leading to overtopping. Tailings dams built by the upstream method require a dry beach of coarse tailings solids above the pond water level upstream of the dam crest. The beach must never be flooded with water otherwise dangerous seepage conditions can develop. This calls for a sound water balance of the tailings disposal system taking into account all the components of inflow under the varying conditions of operation and the climatic conditions in their seasonal variations. Extreme situations with low frequency of re-occurrence need also to be checked.

In parts of the world subjected to long periods of frost, failures have occurred as an indirect result of freezing.

Example. Incident No.66. Kimberley iron mine, British Columbia, 1948. Slope failure occurred during a period of high snowmelt and spring runoff that raised the phreatic surface while the surface of the slope was frozen. A large tailings flowslide developed and frozen blocks of material were seen in the flowing mass.

Example. Details are given by Casagrande and McIver (1971) of extensive sloughing of downstream slope attributed to freezing and growth of ice lenses, accompanied by the development of piping during the first few days of a spring thaw.

Example. Incident No.221. Baia Mare, Romania. A new impoundment about 1 km wide and 1.5 km long was begun on relatively flat land that rose 7 m uniformly over its length. An outer perimeter bank 2 m high was built from old tailings taken from a disused deposit. Inside this perimeter, a starter dam was built from the same material, in general about 1 m high, but higher on the low side of the impoundment. When tailings came from the processing plant it was put through cyclones on the crest to build by the downstream method to the perimeter bund, when further construction would use the upstream method.

Three old disused impoundments were to be re-worked by cutting into them with water jets, to extract remaining gold and silver. The resultant slurry was pumped to a new processing plant, and the processed tailings, containing high concentrations of cyanide (total approx. 400 mg/l, free about 200 mg/l), pumped a further 6 ½ km tothe new impoundment. Decanted water was pumped back to the old impoundments to provide the water jets, with the aim of forming a closed system with no discharge into the environment.

During the first summer, construction of the dam along the low side of the impoundment and partly up the long sides, progressed well and an ample freeboard was maintained. Evaporation from the disused and new impoundments exceeded precipitation, but this situation changed when winter set in and the volume of circulating water increased. When the temperature fell below freezing, it was no longer possible to operate the cyclones and the whole tailings was discharged into the impoundment, which became covered by ice, in turn covered by the precipitation in the form of snow.

On the last day of January 2000, a change of wind direction brought heavy rain and a sudden increase of temperature to above freezing. Water liberated from the ice and snow, supplemented by the rainfall raised the water level in the impoundment until it overflowed, part way up one of the long sides where dam construction was quite low, cutting a breach 20 to 25 m wide permitting a spill of about 100,000 m^3 of heavily contaminated water.

Decant towers. In regions subject to long periods of frost, unequal ice thrust against a decant tower can generate excessive movements in the tower stem or at the stem to base contact. Lowering the water level below the ice can generate large moments in the tower section and wind plus wave forces across an irregular ice flow can generate torsion forces sufficient to shear off the tower stem. Large releases of water caused by tower failure under these conditions have on occasion transported significant volumes of tailings away from the impoundment. Many of these difficulties can be overcome by coating the tower with rigid closed-cell foam panels fixed around the exterior of the tower to prevent the formation of bond between the ice and the tower.

6.2.10 Impoundments not retained by a dam

a). Tailings discharged into a worked out pit or other depression can be expected to have no greater bulk density than those put into a traditional impoundment: if anything slightly less because there may be no special provision for drainage for the pore water in the tailings. Because of this they are just as susceptible to liquefaction and, should the opportunity arise, will rapidly escape through any opening. Even water, when inadvertently stored then allowed to escape suddenly can cause trouble.

Example. A quarry in Scotland drained through a tunnel into a catchment area above an earthfill dam. During a storm in 1925 the tunnel choked with debris and the quarry filled with water. The increasing pressure of water eventually forced through the debris blockage releasing a large volume of water into the water reservoir. This flood overtopped the dam, causing a breach which released, in addition to the contents of the quarry, the contents of the reservoir, that flowed into the town of Skelmorely killing 5 people. This accident in 1925 contributed to the formation of the British Reservoirs (Safety Provisions) Act of 1930.

Example. Mine tailings from the San Antonio ore deposit have, since 1993, been put into an old open cut mine site (Tapian ore reserve) known as the Tapian Pit. A drainage tunnel leading from this pit was sealed with a concrete plug prior to placing the tailings, so as to make the old pit into a watertight basin. The mine tailings were a fine material, about 75% by weight of less than 63 micron size and 95% less than 200 microns. The tailings were deposited at a consistency of 70/30% solids/liquid, and about 20 million cubic metres had been placed by early 1996.

On 24th March 1996, large quantities of tailings began escaping from the drainage tunnel into the Makulapnit and Boac Rivers. During the following 4 to 5 days, approximately 2.4 million tons of tailings were released. Subsequently the flow of tailings from the tunnel was reduced, but during the following 6 weeks, the total weight lost was approximately 3 million tons. The Makulapnit and Boac Rivers below the failed drainage tunnel were reported to have been severely affected, with the tailings reaching as far as the coastal area adjacent to the mouth of the Boac River. As well as the rivers being degraded as a result of smothering by mine tailings, the area covered by a thick layer of tailings is estimated to extend approximately 3 km along the coast and at least ½ km from the shore line.

A United Nations Expert Assessment Mission who investigated this event, as one of their conclusions, pointed out that the mine owners had an inadequate environmental management structure. No apparent risk assessment of the Tapian tailings pit was carried out and consequently, no effective contingency plan, seepage and/or downstream monitoring programmes were in place at the time when the tunnel plug failed. Both these factors contributed significantly to the Marinduque environmental disaster and the failure of the mine owners to rapidly and effectively stop the flow of the mine tailings into the rivers. Clearly the original design of the concrete plug had failed to adequately consider the very high pressures that would be developed as the level of the tailings in the pit rose. It had not considered the seepage forces that would develop in the ground surrounding the plug, nor the strength and deformation properties of that ground. Had the matter been given consideration prior to placing the plug, a site investigation made from the surface or from within the tunnel itself would have enabled a satisfactory design to have been evolved for the construction of the plug.

b). Thickened Tailings Disposal (TTD) System. The aim of the TTD system is to eliminate the possibility of dam failures and prevent the pollution attributed to conventional impoundments. It also aims to reduce the cost of reclaiming conventional impoundments after closure. These aims are achieved by depositing thickened tailings from a topographical high, or from central ramps or towers, so as to form a self supporting mound or ridge of the stored tailings. This minimises the requirement for confining dams, eliminates the need for a settling pond, and shapes the discharged tailings into a self-draining, easily reclaimable shape, having slopes of 2 to 6 percent.

The removal of much of the process water from the tailings is achieved by passing the tailings through high compression thickeners. $^2/_3$ to $^4/_5$ of the process water is decanted from the thickeners as clear overflow and is recycled back into the process. The thickened tailings do not segregate, so that all the particle sizes stay together forming a homogeneous material with a high capillary suction: when placed layer by layer, it dries to near its shrinkage limit and becomes dilative under earthquake strains, thus preventing liquefaction. It tends to remain saturated almost to the surface, thus preventing the development of acid drainage and so becomes suitable for eventual topsoil and vegetation. Detailed description of the system has been given by Robinsky (1999).

Example. The Falconbridge Ltd 12,000 ton/day copper/zinc Kidd Creek mine in Ontario, Canada, was converted in 1972 to use the TTD system to avoid having to build traditional tailings dams on very soft and sensitive clay foundations. The disposal area of 1420 ha, with an average diameter of 1.5 km is on a topographical high surrounded on three sides by a river. Initially discharge was from a series of spigots from a central ramp, but later a single point discharge was placed at the north end of the area with the intention of moving it progressively to the south to create a ridge so as to allow of progressive reclamation from the north end while deposition continues. The slopes formed are between 2.5 and 3 percent and the only retaining dam required is 10 m high across a small valley. This Kidd Creek operation was the first TTD project and while considered experimental at first, the system has progressively evolved to form a successful method of tailings deposition, now operating in its 24th year.

Example. The open pit nickel sulphide mine at the Mount Keith operation in Western Australia produces 11.5 x 10^6 tonnes of tailings a year. The area is relatively flat and semi-arid, with an average annual pan evaporation of 3800 mm, while the rainfall is only 220 mm. As a cheaper and potentially safer method of tailings storage than the traditional paddock method, mine management chose the thickened tailings disposal system, originally proposed by Professor Robinsky in 1968. Following an intensive field and laboratory investigation, they designed the storage facility to be able to accommodate 250 x 10^6 tonnes of tailings at a rate of up to 15 x 10^6 tonnes/year. The storage area is 1700 ha, with an average diameter of 4.6 km on land with a very slight fall of only 12 to 14 m from west to east. A perimeter bund 14 km long surrounds the site to prevent the spread of any materials that might be carried by rainfall run-off. There is a central riser pipe surrounded by 8 other risers 35 to 45 m high and a fully automated three-train, two stage pumping station able to deliver the thickened tailings to any riser at a rate of 3 x 10^3 m^3 per hour. Underdrainage was installed in the ground surrounding each riser, plus open drainage to collect decanted water. There are back-up systems including spare risers, bypass line, dump valves etc., and the facility is monitored remotely continuously by telemetry. A computer programme gives comparison between predicted and actual mound development using aerial photography. The facility was commissioned in December 1996 and has been operating continuously since. This information was supplied by the mine owners, WMC Resources Limited, Perth.

Example. The Thickened Tailings Disposal (TTD) System has been used at the Peak Gold Mine at Cobar, New South Wales. This underground mine, established in 1992, produces gold together with copper, lead, zinc, but mainly 0.3 x 10^6 tonnes of fine tailings a year. The absence of any significant amount of waste rock from the underground mine made the idea of an impoundment requiring no large dams particularly attractive, and the central discharge avoids the considerable work of having to move the discharge points as required during the construction of the other types of tailings dams. The general arrangement for the storage is shown by Fig.18.

6.3 SAFETY MANAGEMENT

The process of implementing decisions associated with the assessment, toleration and reduction of risks can be termed safety management. Owners and operators have specific responsibilities for their dams and the need to formulate safety management procedures. Technical and managerial approaches should be utilised to improve safety and reduce risk. Continuing day to day safety of the dam-impoundment system will depend on some form of observational method involving surveillance and monitoring, using suitable instrumentation to reveal internal conditions.

An increase in safety is provided at an increase in cost and a balance has to be found between dam safety and economy. Cost effective risk reduction involves defining the acceptable level of risk, reducing the risk of the dam breaching to an acceptable value and implementing emergency management procedures to endeavour to ensure that there is no loss of life should the dam breach. The approaches to risk reduction for the dam-impoundment system can include structural improvements to the dam and ancillary works, improved surveillance, monitoring and maintenance. The approaches to risk reduction for the downstream valley system include the preparation of inundation maps, estimation of the time of arrival of flood wave at different locations and the duration of inundation and the implementation and maintenance of emergency warning procedures and systems. Unlike water, liquefied tailings do not drain away and the deposits left on roadways can seriously hamper emergency services. The weight of tailings is such that it can cause great damage, much greater than that of an equivalent flood of water, demolishing buildings rather than just flowing through them. The difficulty in knowing when to give warning makes the operation of emergency procedures very difficult.

7. CONCLUSIONS AND RECOMMENDATIONS

a) Introduction. Failures of tailings dams continue to occur despite the available improved technology for the design, construction and operation. The consequences of these failures have been heavy economic losses, environmental degradation and, in many cases, human loss.

b) Main reasons for failure. Causes in many cases could be attributed to lack of attention to detail. The slow construction of tailings dams can span many staff changes, and sometimes changes of ownership. Original design heights are often exceeded and the properties of the tailings can change. Lack of water balance can lead to “overtopping”: so called because that is observed, but may be due to rising phreatic levels causing local failures that produce crest settlements.

   Causes include problems of foundations with insufficient investigations, inadequate or failed decants, slope instability, erosion control, structural inadequacies and additional loading of closed impoundments. Most situations have already been solved by engineering technology, indicating that a more systematic application of the specialized knowledge is required.

c) Conclusions. The ICOLD Tailings Dams Committee conclude that the adequate application of available technology of engineering to the design, construction, operation and closure can provide the required cost effective risk reduction.

d) Recommendations. Owners and operators have specific responsibilities to apply safety management procedures to achieve safety improvement and risk reduction. The design, construction, operation and closure of dams and impoundments with risk potential to downstream shall include the following requirements:

   1) Detailed site investigation by experienced geologists and geotechnical engineers to determine possible potential for failure, with in situ and laboratory testing to determine the properties of the foundation materials.

   2) Application of state of the art procedures for design.

   3) Expert construction supervision and inspection.

   4) Laboratory testing for “as built” conditions.

   5) Routine monitoring.

   6) Safety evaluation for observed conditions including “as built” geometry, materials and shearing resistance. Observations and effects of piezometric conditions.

7) Dam break studies.

8) Contingency plans.

9) Periodic safety audits.

Regulatory Authorities should be more concerned about the safety of tailings dams that come under their jurisdiction and should require periodic reviews carried out by appointed specialists. In some countries approval had to be obtained for specific stages of construction, causing the stability, general condition and safety to be automatically checked from time to time.

8. LESSONS LEARNED : IMPLICATIONS FOR POLICY – A UNEP VIEW

Improvements are made by learning the lessons of experience and using them to avoid repeating the mistakes of the past.

This Bulletin is intended to be of help to all those connected in any way with tailings storage facilities including owners, managers, contractors, engineers, personnel responsible for day to day construction, and government officials concerned with regulation. In highlighting accidents, the aim is to learn from them, not to condemn. UNEP commends ICOLD for its thorough work in compiling and analysing this extensive data base of accidents, incidents and remedial action.

The Bulletin clearly shows that many of the tailings dam accidents which have occurred have been preventable. Also that the factors which have contributed to failures have often been common to different sites and over time. It makes the point that the technical knowledge exists to allow tailings dams to be built and operated at low risk, but that accidents occur frequently because of lapses in the consistent application of expertise over the full life of a facility and because of lack of attention to detail.

This is an unsatisfactory situation for an industry under public scrutiny, whose products are an essential part of daily life and which contributes to many economies around the world. Major accidents naturally destroy community and political confidence. By highlighting the continuing frequency with which they are occurring and the severe consequences of many of the cases, this Bulletin provides *prima facie* evidence that commensurate attention is not yet being paid by all concerned to safe tailings management. This is surprising given the high risk which tailings dams demonstrably pose to life, the environment, property and the profitability of companies.

UNEP has seen awareness of the problem escalate dramatically in the last 3 or 4 years within leading companies, industry bodies, governments and community groups. The World Wide Fund for Nature, for instance, commenced a major campaign in 1998 to focus attention on mining waste lagoons in Europe. The International Council on Metals and the Environment (ICME) has partnered with UNEP in mounting two major tailings workshops, in Sweden in 1997 and in Buenos Aires in 1998. In 2000, ICME commissioned Golder and Associates to review the adequacy of existing tailings management guidelines to see if there is a need to consolidate and strengthen codes and guidelines for international use. (The conclusion was that it would be beneficial). In 2000, the Government of Australia and UNEP co-hosted the first intergovernmental Workshop on Environmental Regulation for Accident Prevention in Mining: Tailings and Chemicals Management. UNEP is working with interested governments to establish an ongoing Regulators Forum – with the safety and environmental performance of tailings facilities and approaches to their regulation as a continuing theme.

All of this work will undoubtedly bear fruit over time since it involves a discussion of solutions at the same time as driving home the seriousness of the problem.

Market forces are also escalating the issue, with financiers and insurers now more concerned with the risks and financial consequences of tailings accidents. This is not surprising since in some of the recent cases companies have gone into receivership as a direct result of the costs of an accident, including interrupted production, clean-up costs, reengineering and reconstruction costs and legal proceedings. Criteria for lending will increasingly include requirements for strong safety and environmental management systems and assurance processes, coupled with technical competence.

In addition to the financial imperative, companies also have other drivers, including shareholder and employee concern and reputation in the community at large.

In recent years many companies have taken new approaches to ensure diligence and quality control in the management of tailings. For example, major mining companies have established expert teams, either from within the company or involving external experts, to undertake regular audits of tailings facilities and tailings management systems at their operations. These audits have covered closed facilities as well as operating facilities, since, as this Bulletin also shows, failures occur in both. Safety reports to management, independent of the site operator, have uncovered some emerging problems resulting in remedial action being taken.

At the same time, the mining industry operates with a continual imperative to cut costs due to the relentless reduction in real prices for minerals which has been experienced over the long term, plus the low margins and low return on capital which are the norm. The result has been a shedding of manpower to the point where companies may no longer have sufficient expertise in the range of engineering and operational skills which apply to the management of tailings. Continuity of management and loss of history of operations are related issues.

Greater recognition of the importance of the safe management of tailings and the attention being paid to it is encouraging. Yet both UNEP and ICOLD believe that still more needs to be done, and that arguably the most important area for action may lie at this time with the regulators.

Stuart Cale, a member of the British Sub-Committee of ICOLD and one of the authors of this Bulletin, wrote in an article published in UNEP's *Mining and sustainable development II* special 2000 issue of **industry and environment**,

> "Experience of a wide range of systems for inspection and auditing of tailings management facilities around the world indicates that where policies for regular expert auditing by competent persons have been enforced, failures have been reduced and incidences of untoward discharges have been significantly reduced.
>
> …"The major driving force in reducing the number of tailings dam incidents is, firstly, the adoption of regulations that require regular independent auditing and certification of a facility and, secondly, the recognition of the need for a competent person to undertake the audits, and that the competent person must have experience of tailings management facilities, rather than having general

competence in water dams or civil engineering."

In a Bulletin about Lessons Learned it is appropriate to consider not only the physical lessons but also the policy lessons, whether for companies or for governments. Sound policy can undoubtedly play a role in accident prevention. UNEP is not simply advocating the adoption of more or tougher regulation but, rather, the review of the structure of regulation to assess its effectiveness specifically against the particular goal of ensuring the safe management of tailings dams. In some cases, effective implementation of existing regulation may be the more important factor; in others it may be the further training and skills of regulators to exercise their oversight duties effectively. The result needs to be that governments play their part as one of the agents contributing to the goal of accident prevention.

In putting the spotlight onto the role of regulators, UNEP does not want in any way to downplay the role of the owner and operator. The unequivocal statement made in this Bulletin that "the responsibility for the safety of tailings dams must lie with management" is universally applicable.

The Workshop on Environmental Regulation for Accident Prevention in Mining: Tailings and Chemicals Management, which was co-hosted by the Government of Australia and UNEP in Perth in October 2000, brought together regulators and experts from 20 countries to share latest thinking on effective approaches to regulation of the high hazard aspects of mining. That meeting heard from governments which had experienced major tailings accidents about what they see as important to prevent a recurrence. Despite a diversity of situations in different countries – climates, nature of ore-bodies, size of mines and numbers of mines to be regulated, differing expertise either in the companies or in the government, and so on - common elements emerged in the discussion of what is universally required to make a difference.

The discussion was much more about regulatory tools to ensure diligence and to strengthen quality assurance, than it was about specific standards or technical requirements. There was a recognition that site- and ore specific operational differences must be accommodated and that no two tailings dams are the same. Regulators were generally seeking to strengthen their systems of review, reporting, inspection, oversight and sanction, rather than to prescribe operational parameters for engineers and managers.

The Workshop highlighted a set of things to which regulators need to pay attention in order to reduce the risks and the consequences of accidents. Foremost amongst these were:

> Governments should evaluate the design of tailings systems proposed by operators and inspect those systems. Some specify certain safety factors or weather return periods to be incorporated in design. Independent certification of design was seen as a fundamental requirement, with regulators using expert advice at arm's length from the company. Governments have to address not only sound design but also correct operation, with all aspects of the life of tailings dams being important. Repeated inspection and certification of stability may be warranted and governments should consider requiring independent inspections of the dam against design at adequate intervals, as a condition of

approval.

Some governments already have a role in approving all phases – design, construction, certification, surveillance, closure, emergency planning. The post – closure phase is of particular importance to governments because the stability of closed tailings facilities must be maintained in perpetuity. Also, all aspects of tailings systems need to be covered, since accidents occur in tailings pipelines as well as in dams. Governments need to take a holistic approach to the regulation of tailings dams since a range of expertise and different agencies will be involved. Attention to coordination and integration is therefore important to ensure that no aspects are overlooked.

Also as a permit condition, continuous monitoring should be required, using instrumentation as appropriate, which gives meaningful information on the safety performance of the facility. Governments may require operating changes to reduce risk in the light of monitoring results and operational experience. In the long operating phase of tailings dams, conditions can change substantially and unforeseen circumstances arise which require a flexible response by both operator and regulator.

Regulators and companies need to focus on risk reduction and contingency planning as well as on design, operation and compliance. Regulators need to consider requirements for risk reduction to be cost-effective as well as to allow acceptable levels of risk. Governments should require emergency plans to be developed in high risk situations and include emergency procedures in site inspections. In the case of orphan sites, there needs to be a process of risk assessment and prioritisation leading to preventative remedial action at high risk/high consequence sites.

Governments need to be empowered to act to require corrective action in the case of a problem arising - before it becomes an emergency (if the company has not already done so). Regulators need adequate expertise and access to competent advice in taking these decisions. Requirements to report "near-misses" or critical events can assist governments to be proactive.

Regulators need to be trained and to gain on-ground experience in tailings dam matters in order to be able to interpret signs, reports and data and to take competent decisions. Experience is required to be able to visualise emerging problems and their consequences. Inspection and monitoring protocols should be developed by experienced regulators. Adequate resources need to be available for regulatory purposes and regulators also need to have both the power and the political support to take action or to apply sanctions when necessary.

More generally, it was agreed that regulators should encourage or find ways to reward companies adhering voluntarily to tailings and other codes and guidelines plus formal management systems. Also that regulation should be developed in an open and transparent manner, in consultation with communities, in order to meet concerns and to build support for projects based on an accurate understanding of their risks.

In some countries and in some cases many of these things are already the norm or are on the agenda of regulators. In other jurisdictions they are not. UNEP is of the view that this list is worth promulgating for further discussion and further elaboration as to how some of these things are actually being done or could be done by regulators.

One proposal which was raised but not fully explored concerned the possibility of requiring tailings facility operators to be certified for competency in different aspects of tailings management. UNEP believes this is worthy of further discussion involving the industry, governments and training institutions. The proposed Regulators Forum will provide an opportunity to take some of this thinking to the next level as well as to spread it more widely amongst countries.

UNEP values its association with ICOLD as the leading global authority on tailings dam technical management. We will continue to work with ICOLD and with other partners to ensure that sound engineering knowledge and techniques are applied in the field and that hard-won lessons are not learned only by the specialists. UNEP commends this Bulletin to all owners, operators and regulators.

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10. FIGURES

MCEA Comments Ex. 19

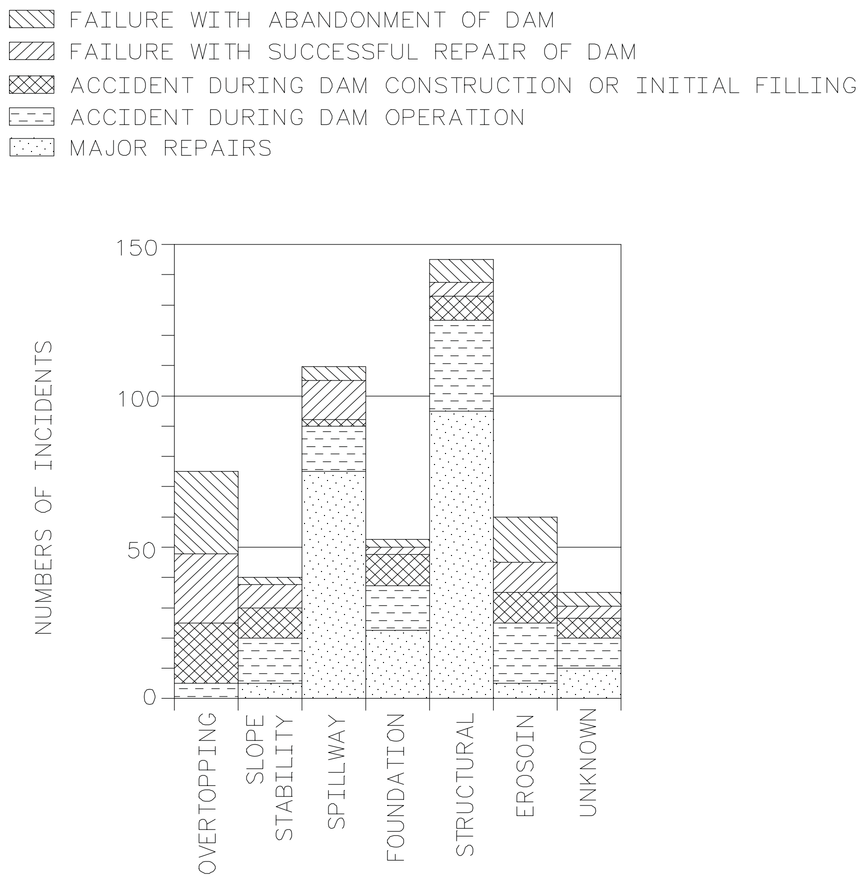


Figure 1. Water storage dam incident comparison.

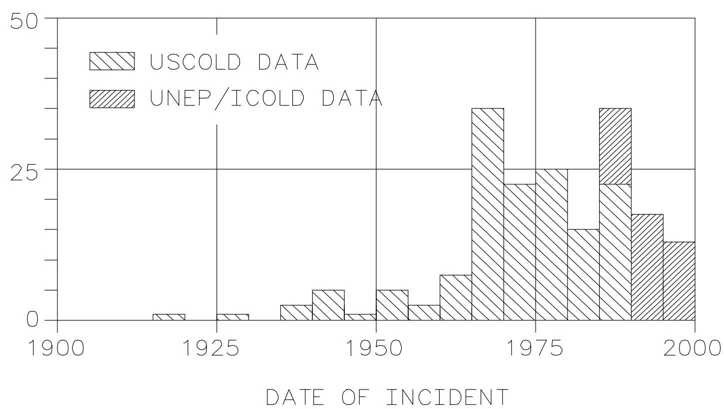


Figure 2. Tailings dam incident history summary: number of incidents per 5 year period.

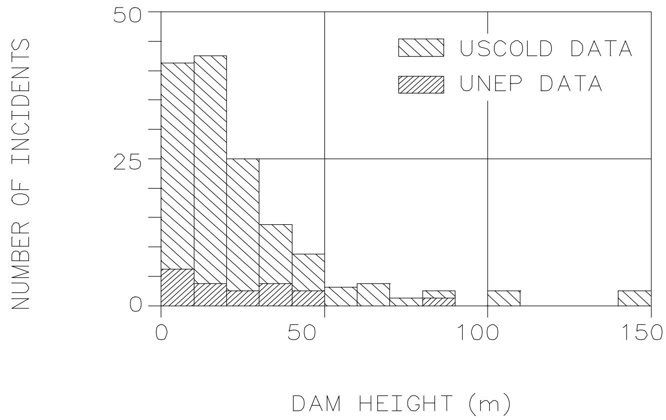


Figure 3. Tailings dam incident and height comparison.

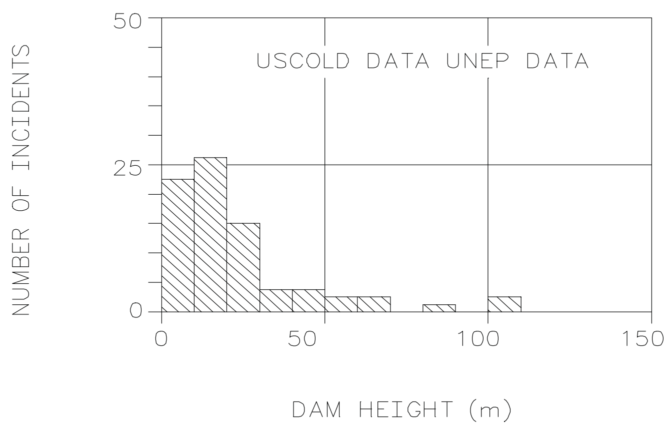
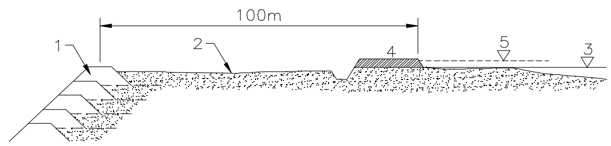


Figure 4. Tailings dam failure and height comparison.



1. DAM (HEIGHT APPROX. 70m)
2. BEACH
3. SUPERNATANT POOL
4. EXTRA DYKE
5. FREEBOARD LEVEL

SECTION (SCHEMATIC)

Figure 5. Extra dyke on a closed tailings impoundment.

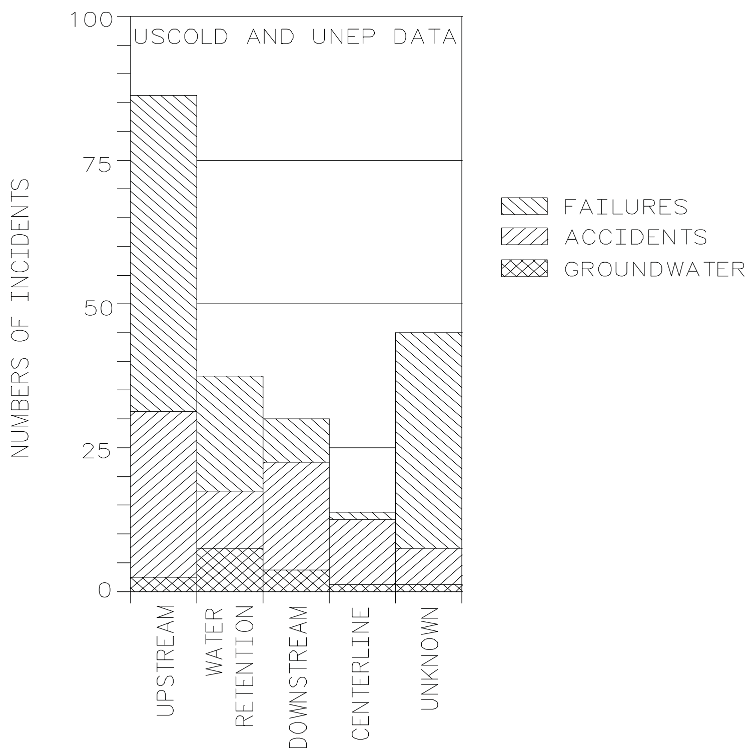


Figure 6. Tailings dam type comparison.

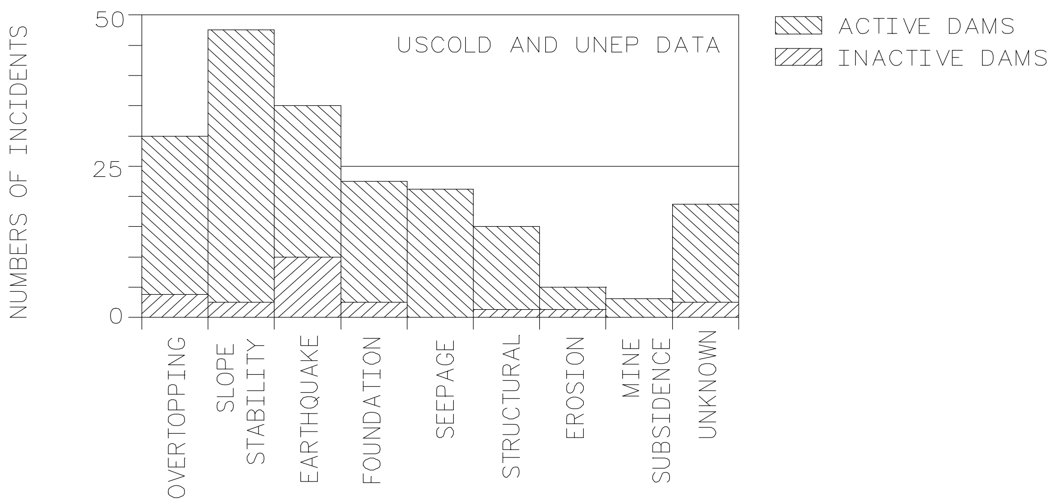


Figure 7. Tailings dam incident cause comparison with dam status.

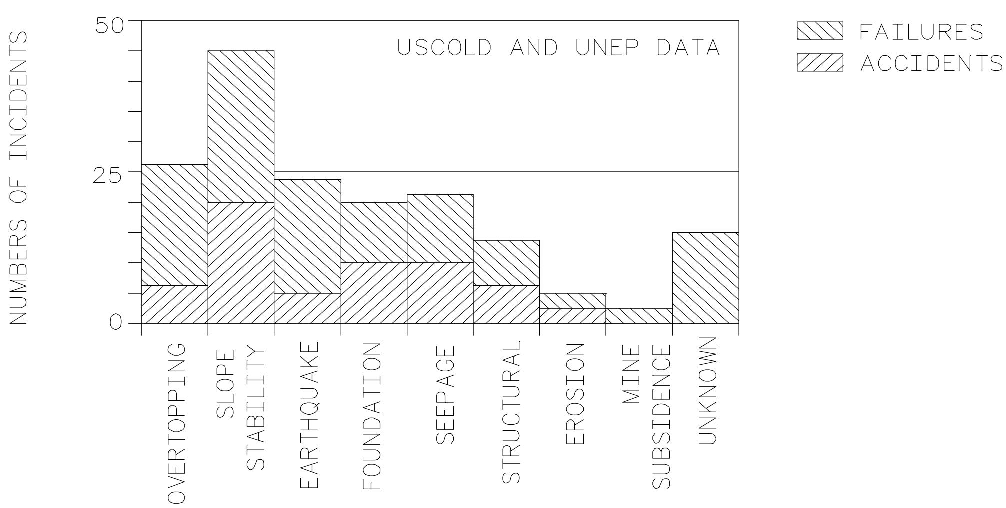


Figure 8. Tailings dam incident cause comparison with incident type for active dams.

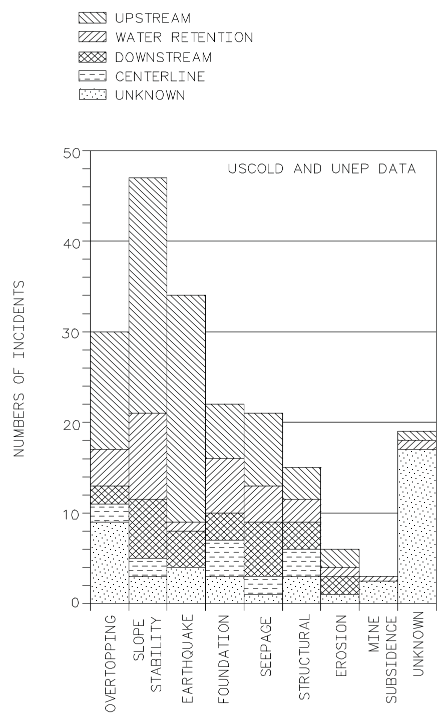


Figure 9. Tailings dam incident cause comparison with dam type.

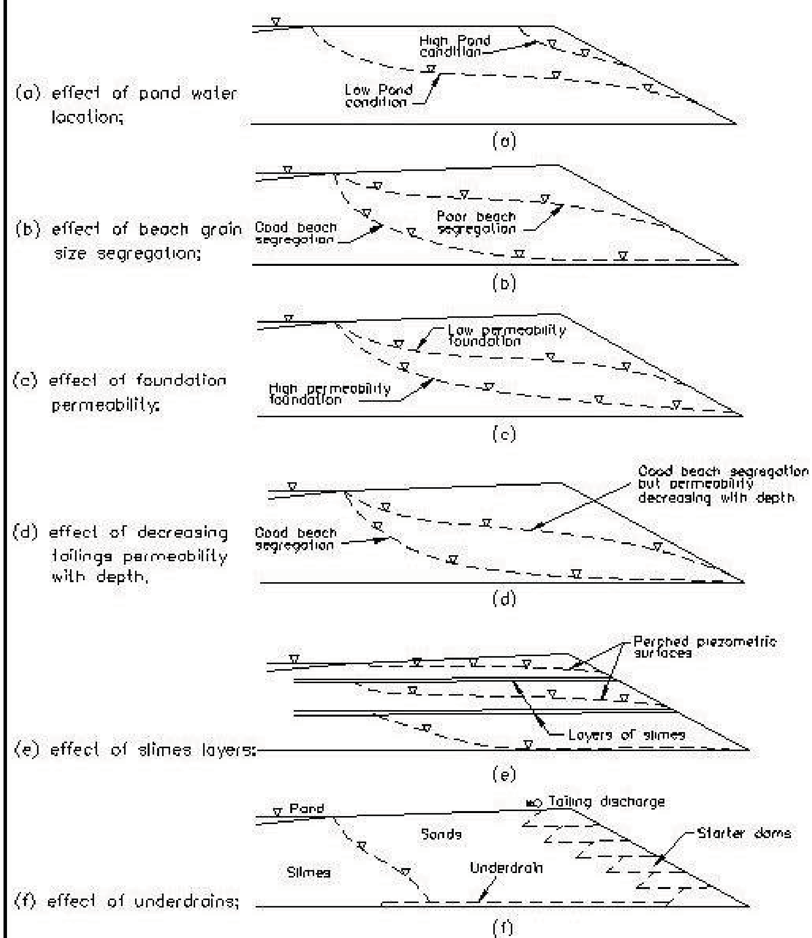


Figure 10. Factors influencing the position of the phreatic surface in dams built by the upstream method.
Shows effect of lack of drainage and layers of low permeability producing perched water surfaces. (After Fell et al 1992)

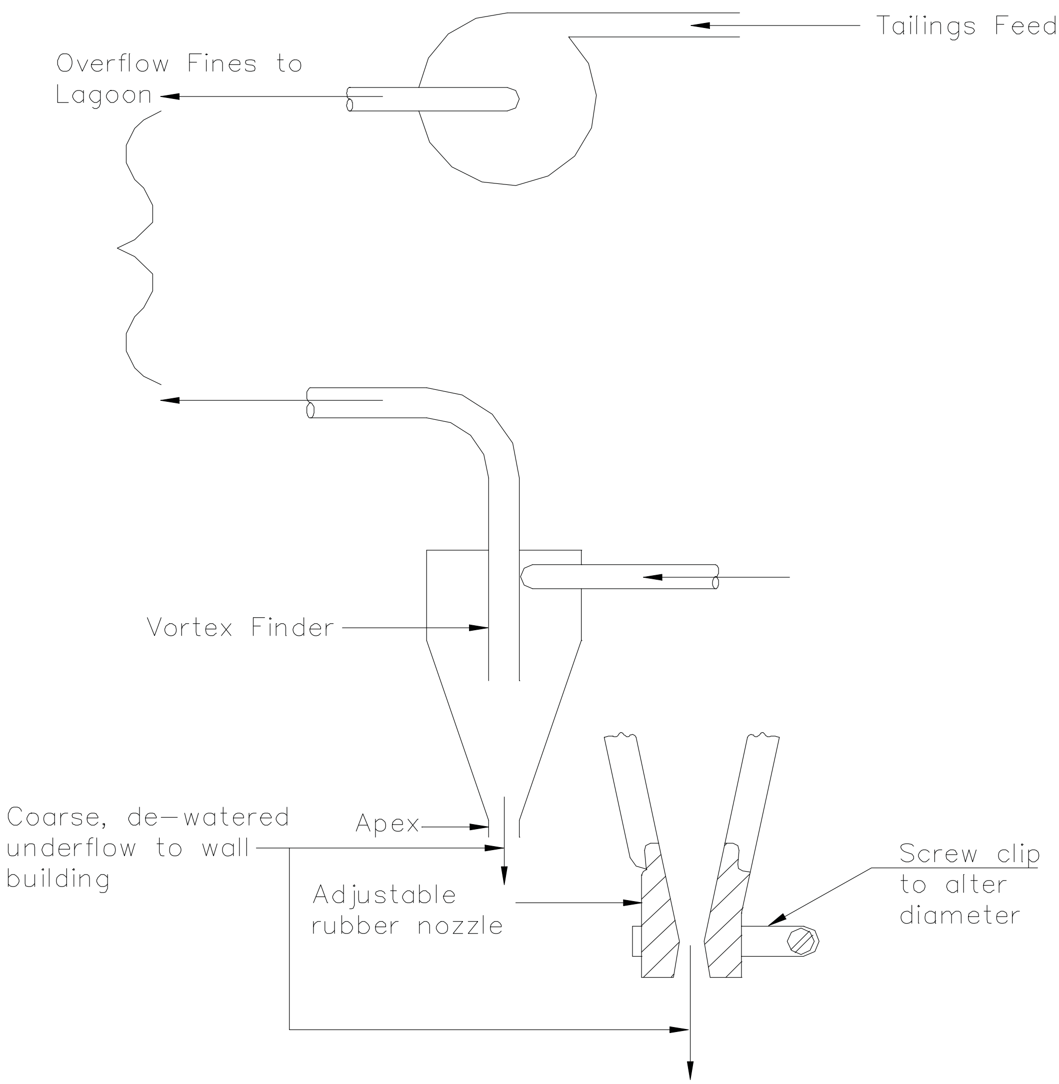
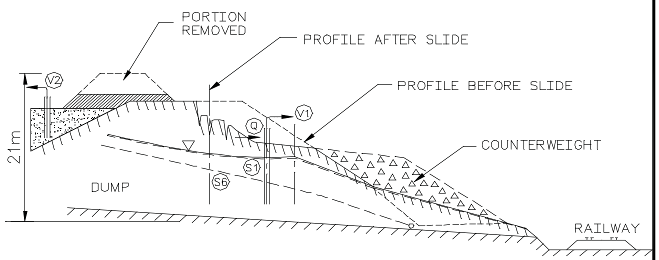


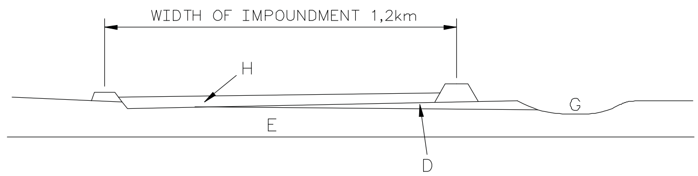
Figure11. Hydro-cyclone.



S1 = PHREATIC LINE 1 DAY BEFORE SLIDE
S6 = PHREATIC LINE 6 DAYS BEFORE SLIDE
Q = WATER ESCAPING FROM HIDDEN COARSE LAYER
V1 = WELLPOINTS FOR REMOVAL OF POREWATER FROM DAM BODY
V2 = WELLPOINTS FOR REMOVAL OF POREWATER FROM SEDIMENTED TAILINGS

0 5 10 15 20m

Figure 12. Section of starter dam formed by an existing dump.



A WASTE ROCKFILL.
B EARTHFILL.
C SLURRY TRENCH CUT–OFF.
D ALLUVIAL SAND AND GRAVEL ABOUT 4m THICK.
E BLUE MARL 60m THICK. $k = 10^{-11 \text{ to } -12}$ m/s.
F SLIP SURFACE. $c' = 0$, $\phi' = 16°$
G RIVER
H TAILINGS. $\gamma = 28$ kN/m^3

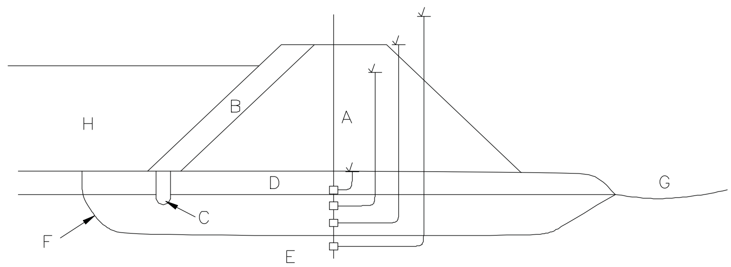


Figure 13. Section of Aznalcôllar Tailings Dam.

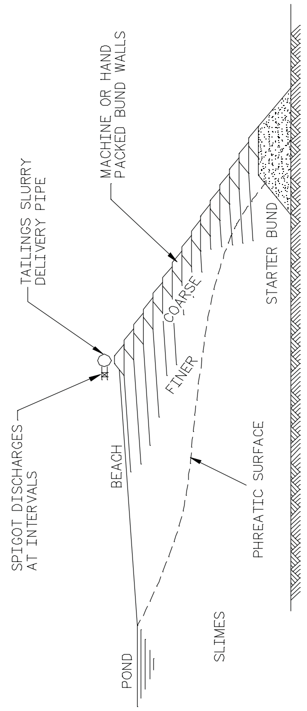


Figure 14. Upstream method of construction with spigots.

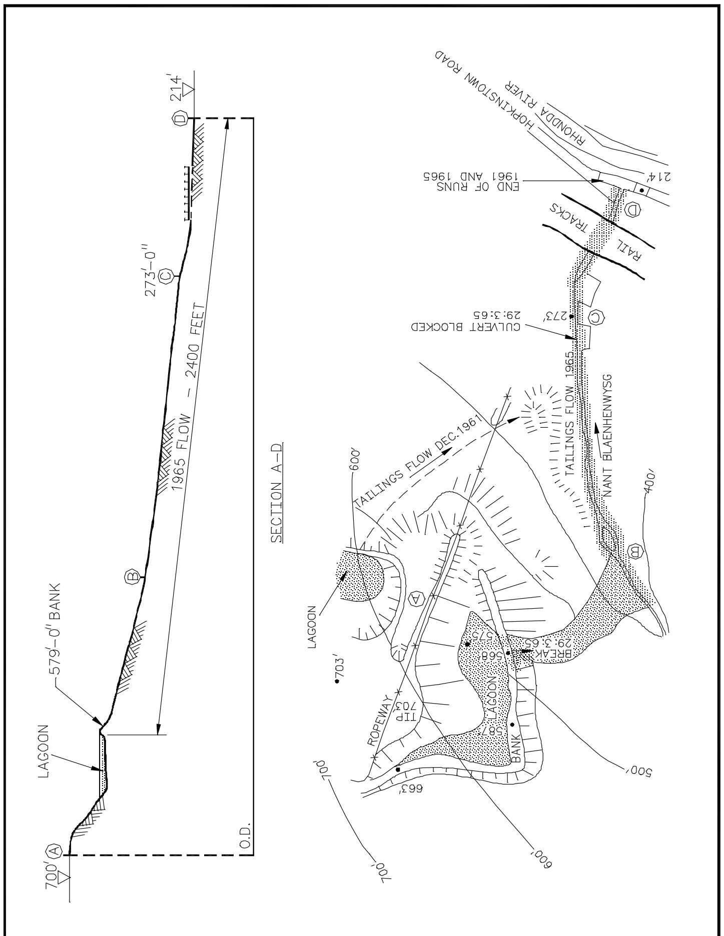


Figure 15. Tailings flows at Ty Mawr colliery, South Wales, in 1961 and 1965.

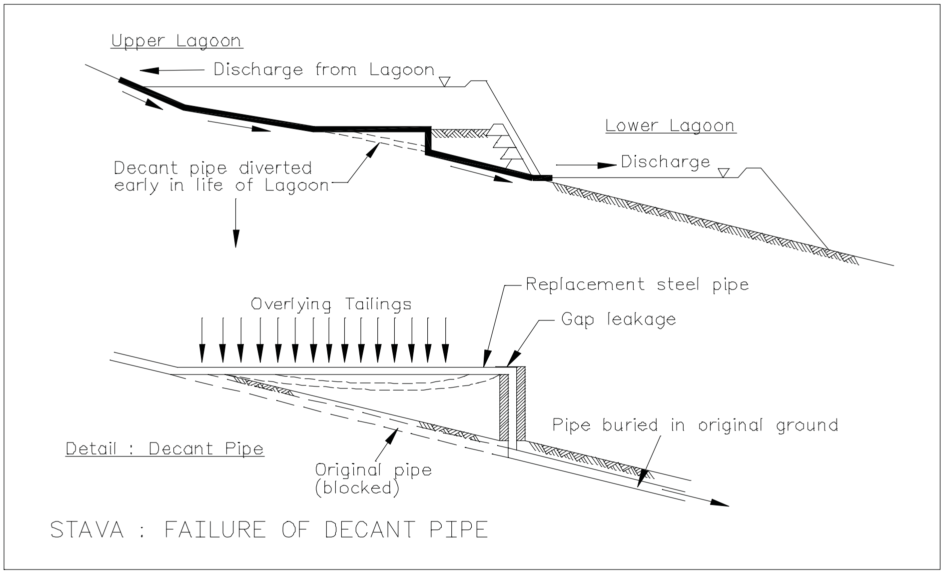


Figure 16. Failure of the decant pipe at Stava.

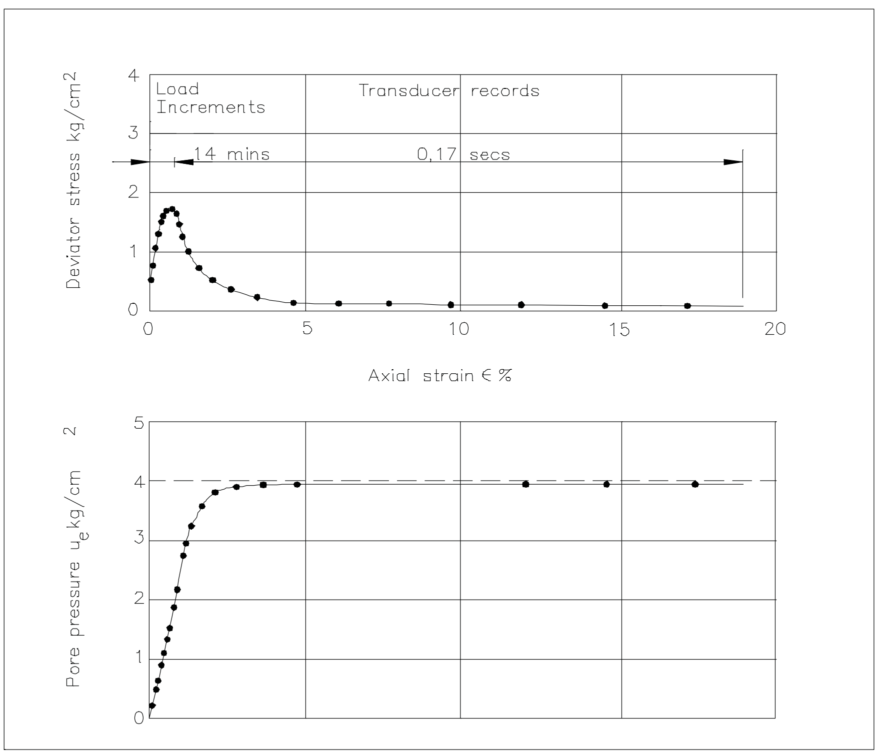


Figure 17. Stress–strain curve for stress controlled consolidated undrained test on saturated loose sand.

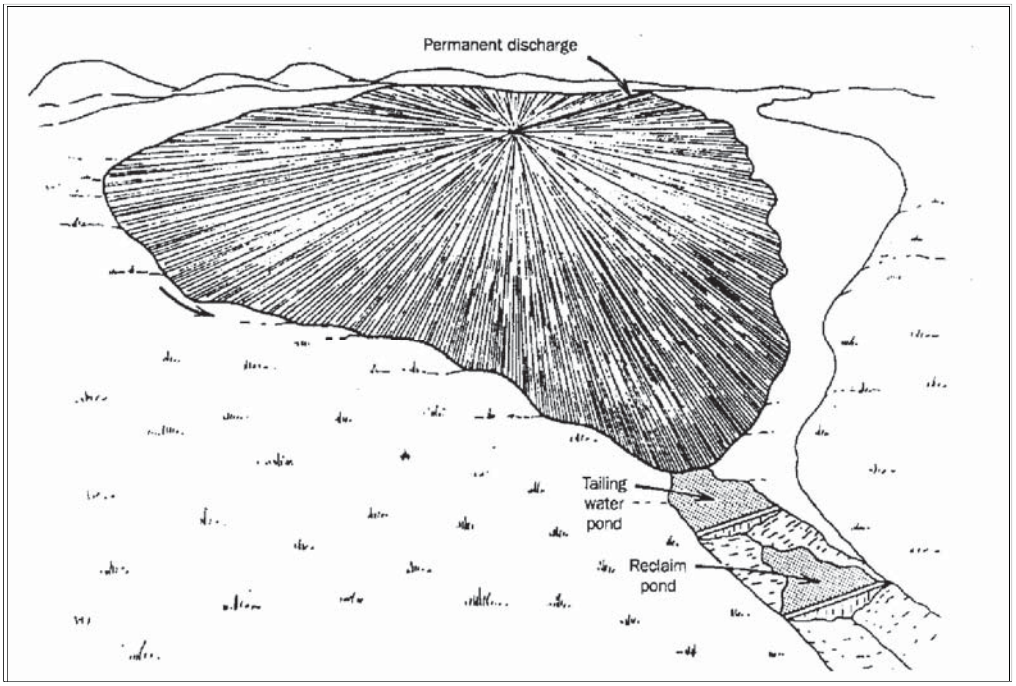


Figure 18. Tailings storage using the Thickened Central Discharge Method. (Reprinted from Robinski 1979)

APPENDIX

TAILINGS DAMS – INCIDENT CASE RECORDS

1. Introduction.
2. Abbreviations.
3. List of tailings dams for which incident cases were collected.
4. Brief descriptions of the 221 cases.

TAILINGS DAMS – INCIDENT CASE RECORDS

1. INTRODUCTION

This Appendix contains individual records for each incident included in the compilation. The abbreviations used are listed in Section 2. The tailings dams for which incident case histories are provided are listed in Section 3, with summary data. Section 4 provides more detailed data and sources for each case, with a brief description of the circumstances of the incident.

The case records were collected by:

Case Nos. 1 to 185:	USCOLD
Case Nos. 186 to 221:	UNEP and ICOLD

2. ABBREVIATIONS

<u>Dam Type</u>

US	upstream
DS	downstream
CL	centerline
WR	water retention
NR	not reported

<u>Dam Fill Material</u>

T	tailings
CST	cycloned sand tailings
MW	mine waste
E	earthfill
R	rockfill

<u>Incident Type</u>

1A	failure, active impoundment
1B	failure, inactive impoundment
2A	accident, active impoundment
2B	accident,inactive impoundment
3	groundwater

<u>Incident Cause</u>

SI	slope instability
SE	seepage
FN	foundation
OT	overtopping
ST	structural
EQ	earthquake
MS	mine subsidence
ER	erosion
U	unknown, or
NR	not reported

3. LIST OF TAILINGS DAMS FOR WHICH INCIDENT CASES WERE COLLECTED

Provided on the pages that follow. Dates are in the format month-day-year.

<u>Note</u>: Summary Page Numbers shown in the List refer to Section 4 “Brief descriptions of the 221 Incident Cases collected.

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ID No.	Mine Name/Location	Summary Page No. (See Note above)	Ore Type	Dam Type	Dam Fill Material	Dam Height (meters)	Storage Volume (Cubic Meters)	Incident Type	Incident Date	Tailings Released (cubic meters)	Tailings Travel (meters)
1	Agrico Chemical,Florida, USA	140	phosphate					1A-U	1968		
2	Alcoa, Texas, USA	142	bauxite			19	4,500,000	1A-U	10-01-1964		
3	American Cyanamid, Florida	141	gypsum					1A-U	1962		
4	American Cyanamid, Florida	141	phosphate					1A-U	1965		
5	Atlas Consolidated, Phillippines	140						1A-MS			
6	Avoca Mines, Ireland	127	copper	WR	T			1A-SI			
7	Bafokeng, South Africa	104	platinum	US	T	20	13,000,000	1A-SE	1974	3,000,000	45,000
8	Bancroft, Ontario, Canada	143	uranium					3-			
9	Barahona, Chile	110	copper	US	CST	61	20,000,000	1A-EQ	10-01-1928	2,800,000	
10	Berrien, France	104	kaolin	US	R	9		1A-SE	1974		
11	Battle Mt. Gold, Nevada, USA	122	gold	DS	E	8	1,540,000	2A-SI	1984		
12	Bellavista. Chile	109	copper	US	T	20	450,000	1A-EQ	03-28-1965	70,000	800
13	Big Four, Florida, USA	119	phosphate	CL	E	18		2A-ST	1986		
14	Big Four, Florida, USA	119	phosphate	CL	E			2A-FN	08-01-1989		
15	Bilbao, Spain	136			R			1A-SI		115,000	
16	Blackbird, Idaho, USA	119	cobalt	CL	MW	15	1,230,000	2B-ST			
17	Bonsal, North Carolina, USA	129	sand & gravel	WR	E	6	38,000	1A-OT	08-17-1985	11,000	800
18	Cadet No. 2, Montana, USA	118	barite	CL	E	21		2A-SI	09-01-1975		
19	Captains Flat Dam 2, Australia	118	copper	CL	E	22		2A-SI			
20	Captains Flat Dump 3, Australia	141	copper		T			1A-U	1942	40,000	

ID No.	Mine Name/Location	Summary Page No. (See Note above)	Ore Type	Dam Type	Dam Fill Material	Dam Height (meters)	Storage Volume (Cubic Meters)	Incident Type	Incident Date	Tailings Released (cubic meters)	Tailings Travel (meters)
21	Captains Flat Dump 6A, Australia	102	copper	US	T			1A-SI	1939		
22	Carr Fork, Utah, USA	138	copper			10		1A-ST	02-01-1975		
23	Casapalca, Peru	107	Lead/zinc	US	T	107		1A-ST			
24	Casapalca	110	Lead/zinc	US	T	107		1A-EQ			
25	Castle Dome, Arizona, USA	104	copper	US	T			1A-SE		150,000	100
26	Cerro Blanco de Polpaico, Chile	133	limestone	WR	R	9		2A-EQ	03-28-1965		
27	Cerro Negro No. 1, Chile	116	copper	US	T	46		2B-EQ	03-28-1965		
28	Cerro Negro No. 2, Chile	116	copper	US	T	46		2B-EQ	03-28-1965		
29	Cerro Negro No. 3, Chile	109	copper	US	T	20	500,000	1A-EQ	03-28-1965	85,000	5,000
30	Cerro Negro No. 4, Chile	108	copper	US	CST	40	2,000,000	1A-EQ	03-03-1985	500,000	8,000
31	Cities Service, Florida, USA	131	phosphate	WR	E	15	12,340,000	1A-U	12-03-1971	9,000,000	120,000
32	Clayton Mine, Idaho, USA	119	silver	CL	T	24	215,000	2A-ST	02-06-1983		
33	Climax, Colorado, USA	140	uranium					1A-U	07-02-1967	12,000	
34	Cyprus Thompson Creek, Idaho, USA	118	molybdenum	CL	CST	146	27,000,000	2A-SE	1989		
35	Incident No. 1, Elliot Lake, Ontario, Canada	134	uranium	WR	E	9		3-	1979		
36	Dashihe. China	115		US		37		2A-EQ	1976		
37	Deneen Mica Yancey County, North Carolina, USA	101	mica	US	CST	18	300,000	1A-SI	06-01-1974	38,000	30
38	Derbyshire, United Kingdom	121	coal	DS		8		1B-FN	1966	30,000	100
39	Dixie Mine, Colorado, USA	142	gold					1B-U	04-01-1981		

ID No.	Mine Name/Location	Summary Page No. (See Note above)	Ore Type	Dam Type	Dam Fill Material	Dam Height (meters)	Storage Volume (Cubic Meters)	Incident Type	Incident Date	Tailings Released (cubic meters)	Tailings Travel (meters)
40	Dresser No. 4, Montana, USA	117	barite	CL	E	15		1A-FN	08-15-1975		
41	Earth Resources, N M, USA	106	copper	US	T	21		1A-OT	1973		
42	El Cerrado, Chile	116	copper	US	T	25		2B-EQ	03-28-1965		
43	El Cobre New Dam, Chile	121	copper	DS	CST	19	350,000	1A-EQ	03-28-1965	350,000	12,000
44	El Cobre No. 4, Chile	125	copper	DS	CST	50		2A-EQ	03-03-1985		
45	El Cobre Old Dam, Chile	109	copper	US	T	35	4,250,000	1A-EQ	1965	1,900,000	12,000
46	El Cobre Small Dam, Chile	116	copper	US	T	26	985,000	2B-EQ	03-28-1965		
47	GCOS, Alberta, Canada	112	oil sands	US	T	61		2A-SI	1974		
48	Galena Mine, Idaho, USA	115	silver	US	E	14		2A-ER	1972		
49	Galena Mine, Idaho, USA	106	silver	US	MW	9		1A-OT	01-15-1974	3,800	610
50	Golden Gilpin Mine, Colorado, USA	143	gold			12		1B-U	11-01-1974		
51	Golden Sunlight, MT, USA	120	gold	CL	CST			3-	05-01-1983		
52	Granisle, BC, Canada	123	copper	DS	MW	24		2A-SE			
53	Grey Eagle, California, USA	126	gold	DS	E			3-	1983		
54	Grootvlei, South Africa	102	gold	US	T			1A-SI	1956		
55	Hierro Viejo, Chile	109	copper	US	T	5		1A-EQ	03-28-1965	800	1,000
56	Hirayama, Japan	126	gold	DS		9	87,000	2B-EQ	1978		
57	Hokkaido, Japan	108		US	T	12	300,000	1A-EQ	1968	90,000	150
58	Hollinger, Canada	104	gold	US	T	15		1A-FN	1944		
59	Homestake, N. Mexico, USA	107	uranium	US	T	21		1A-ST	02-01-1977	30,000	

ID No.	Mine Name/Location	Summary Page No. (See Note above)	Ore Type	Dam Type	Dam Fill Material	Dam Height (meters)	Storage Volume (Cubic Meters)	Incident Type	Incident Date	Tailings Released (cubic meters)	Tailings Travel (meters)
60	IMC K-2, Saskatchewan, Canada	117	potash	US	T	30		3-	1968		
61	Irelyakh, USSR	133		WR	E	10		2A-ST			
62	Kennecott, Garfield, Utah, USA	102	copper	US	T			1A-SI	1941		
63	Kennecott, Utah, USA	105	copper	US	T			1A-FN	1942		
64	Kerr-McGee, Churchrock, New Mexico, USA	129	uranium	WR	E	9		1A-FN	04-01-1976		
65	Keystone Mine, Crested Butte Colorado, USA	142	molybdenum					1B-U	05-01-1975		
66	Kimberley, BC, Canada	102	iron	US	T			1A-SI	1948	1,100,000	
67	Kyanite Mining, Virginia, USA	143	kyanite			11	430,000	2A-OT	1980		
68	La Belle, Pennsylvania, USA	124	coal	DS	MW	79	1,230,000	2A-FN	07-17-1985		
69	La Patagua New Dam, Chile	109	copper	US	T	15		1A-EQ	03-28-1965	35,000	5,000
70	Los Maquis No. 1, Chile	116	copper	US	T	15		2B-EQ	03-28-1965		
71	Los Maquis No. 3, Chile	110	copper	US		15	43,000	1A-EQ	03-28-1965	21,000	5,000
72	Lower Indian Creek, MO, USA	112	lead	US	E			2A-SI	1960		
73	Iwiny, Lower Silesia, Poland	130	copper	WR	E	25		1A-MS			
74	Madison, Missouri, USA	129	lead	WR	E	11		1A-OT	02-28-1977		
75	Maggie Pye, United Kingdom	101	china clay	US	T	18		1A-SI	1970	15,000	35
76	Marga, Chile	142	copper					1B-OT	1985		
77	Marianna Mine #58, PA, USA	111	coal	US	E	37	300,000	2A-SI	11-19-1986		
78	Miami Copper, Arizona, USA	113	copper	US	T			2A-SE			
79	Mike Horse, Montana, USA	111	Lead/zinc	US	T	18	750,000	1B-OT	1975	150,000	

ID No.	Mine Name/Location	Summary Page No. (See Note above)	Ore Type	Dam Type	Dam Fill Material	Dam Height (meters)	Storage Volume (Cubic Meters)	Incident Type	Incident Date	Tailings Released (cubic meters)	Tailings Travel (meters)
80	Mines Development, Edgemont, South Dakota, USA	141	uranium					1A-U	06-11-1962	100	
81	Mir, Bulgaria	110	Lead/zinc	US	T			1A-U	1966		
82	Missouri Lead, Missouri, USA	125	lead	DS	CST	17		2A-ST			
83	Mobil Chemical, Florida, USA	140	phosphate					1A-U	1967	250,000	
84	Mochikoshi No. 1, Japan	108	gold	US	T	28	480,000	1A-EQ	01-14-1978	80,000	7,000
85	Mochikoshi No. 2, Japan	108	gold	US	T	19	480,000	1A-EQ	01-15-1978	3,000	150
86	Monsanto Dike 15, TN, USA	122	phosphate	DS	E	43	1,230,000	2A-SE	1969		
87	Montana Tunnels, MT, USA	126	gold	DS	MW	33	250,000	3-	1987		
88	Mulfilira, Zambia	139	copper			50		1A-MS	1970		
89	N'yukka Creek, USSR	132		WR	E	12		2A-FN	1965		
90	Norosawa, Japan	125	gold	DS		24	225,000	2B-EQ	01-14-1978		
91	Ollinghouse, Nevada, USA	128	gold	WR	E	5	120,000	1A-SE	1985	25,000	1,500
92	PCS Rocanville, Saskatchewan, Canada	117	potash	US	T	12		3-	1975		
93	Park, United Kingdom	130	china clay	WR	T	3		1A-OT	1970		
94	Phelps-Dodge, Tyrone, New Mexico, USA	100	copper	US	CST	66		1A-SI	10-13-1980	2,000,000	8,000
95	Pinchi Lake, BC, Canada	133	mercury	WR	E	13		2A-ER	1971		
96	Pit No. 2, Western Australia	100	rare earth	US	T	9		1A-SI	1977		
97	Portworthy, United Kingdom	121	china clay	DS	R	15		1A-ST	1970		
98	Rain Starter Dam, Elko, Nevada, USA	134	gold	WR	ER	27	1,500,000	3-	1988		

ID No.	Mine Name/Location	Summary Page No. (See Note above)	Ore Type	Dam Type	Dam Fill Material	Dam Height (meters)	Storage Volume (Cubic Meters)	Incident Type	Incident Date	Tailings Released (cubic meters)	Tailings Travel (meters)
99	Ramayana No. 1, Chile	139	copper	US	T	5		1A-EQ	03-28-1965	150	
100	Ray Mine, Arizona, USA	101	copper	US	T	52		1A-SI	12-02-1972		
101	Ray Mine, Arizona, USA	112	copper	US	T	52		2A-SI	02-05-1973		
102	Royster, Florida, USA	104	gypsum	US	T	21		1A-FN	1982		
103	Santander, Spain	137						1A-FN			
104	Sauce No. 1, Chile	115	copper	US	T	6		2A-EQ	03-28-1965		
105	Sauce No. 2, Chile	115	copper	US	T	5		2B-EQ	03-28-1965		
106	Sauce No. 3, Chile	115	copper	US		5		2B-EQ	03-28-1965		
107	Sauce No. 4, Chile	115	copper	US	T	5		2B-EQ	03-28-1965		
108	Silver King, Idaho, USA	124	copper	DS	E	9	37,000	2A-OT	08-05-1989	Small	
109	Silver King, Idaho, USA	121	copper	DS	E	9	37,000	1A-OT	01-16-1974	6,000	
110	Simmer and Jack, South Africa	103	gold	US	T			1A-SI	1937		
111	Soda Lake, California, USA	114	sand & gravel	US	E	3		2A-EQ	10-17-1989		
112	Southern Clay, Tennessee, USA	127	clay	WR	E	5		1A-SE	09-07-1989	300	
113	Southwest US, USA	113		US	T			2A-SI			
114	Spring Creek Plant, Borger, Texas, USA	137	sand & gravel			5	30,000	1A-OT	08-01-1986		
115	St. Joe Lead, Flat River, Missouri, USA	106	lead	US	T	15		1A-OT	1940		
116	Stancil , Maryland, USA	99	sand & gravel	US	E	9	74,000	1A-SI	08-25-1989	38,000	100

ID No.	Mine Name/Location	Summary Page No. (See Note above)	Ore Type	Dam Type	Dam Fill Material	Dam Height (meters)	Storage Volume (Cubic Meters)	Incident Type	Incident Date	Tailings Released (cubic meters)	Tailings Travel (meters)
117	Stava, North Italy	99	fluorite	US	CST	29	300,000	1A-SI	07-19-1985	190,000	4000
118	Suncor E-W Dike, Alberta, Canada	132	oil sands	WR	MW	30		2A-SI	1979		
119	Sweeney Tailings Dam, Longmont, Colorado, USA	136	sand & gravel			7		1A-SE	05-01-1980		
120	Syncrude, Alberta, Canada	119	oil sands	CL	T			2A-FN	1978		
121	TN Consolidated Coal No. 1, Tennessee, USA,	124	coal	DS	MW	85	1,000,000	2A-ST	01-19-1988		
122	Texasgulf 4B Pond, Beaufort Co., North Carolina, USA	131	phosphate	WR	T	8	12,300,000	2A-SI	04-01-1984		
123	Texasgulf No. 1 Pond, Beaufort Co., North Carolina, USA	132	phosphate	WR	E		24,700,000	2A-SI	1981		
124	Tymawr, United Kingdon	137	coal					1A-U	12-1961		800
125	Tymawr, United Kindom	137	coal			12		1A-OT	03-29-1965		700
126	Unidentified	103		US				1A-SI			
127	Unidentified	130		WR	E			1A-ST			
128	Unidentified	114		US	T			2A-ST			
129	Unidentified	103		US	MW			1A-SI			
130	Unidentified	113		US	T			2A-SE			
131	Unidentified	112		US				2A-SI			
132	Unidentified	134	gold	WR	E	8		3-			
133	Unidentified	126	gold	DS	E			3-			
134	Unidentified, Europe	133		WR	R	24		2A-FN			

ID No.	Mine Name/Location	Summary Page No. (See Note above)	Ore Type	Dam Type	Dam Fill Material	Dam Height (meters)	Storage Volume (Cubic Meters)	Incident Type	Incident Date	Tailings Released (cubic meters)	Tailings Travel (meters)
135	Unidentified, Peru	139						1A-EQ	1962		
136	Unidentified, South Africa	141	gold					1A-U	1917		
137	Unidentified, South Africa	114	gold	US	T			2A-FN			
138	Unidentified, South Africa	113	gold	US	T			2A-SE			
139	Unidentified, South Africa	103	gold	US	T			1A-SI			
140	Unidentified, USA	112	copper	US	CST	60		2A-SI			
141	Unidentified, USA	132	magnesia	WR	E	6		2A-FN			
142	Unidentified, United Kingdom	141			R	12		1A-U			
143	Unidentified, United Kingdom	125	coal	DS				2B-SI			
144	Unidentified, United Kingdom	120	coal	DS		20		1A-SI	1967		
145	Unidentified, United Kingdom	122	coal	DS	MW	14		2A-SI	1967		
146	Unidentified, United Kingdom	123	sandstone	DS	E	30		2A-SE	1967		
147	Unidentified, Arizona, USA	113	copper	US	CST	18		2A-SE			
148	Unidentified, Colorado, USA	135		WR	E			3-			
149	Unidentified, Idaho, USA	122	phosphate	DS	E	34		2A-SI	1976		
150	Unidentified, Idaho, USA	122	phosphate	DS	E	18		2A-SI	1965		
151	Unidentified, Indiana, USA	142	coal					1B-OT			
152	Unidentified, Mississippi, USA	106	gypsum	US	T	15		1A-OT	1970		
153	Unidentified, Mississippi, USA	114	gypsum	US	T	20		2A-FN	1974		
154	Unidentified, Texas, USA	104	gypsum	US	T	16		1A-SE	1966	130,000	300
155	Unidentified, Wyoming, USA	134	trona	WR	E	24		3-			

ID No.	Mine Name/Location	Summary Page No. (See Note above)	Ore Type	Dam Type	Dam Fill Material	Dam Height (meters)	Storage Volume (Cubic Meters)	Incident Type	Incident Date	Tailings Released (cubic meters)	Tailings Travel (meters)
156	Unidentified, Alfaria River, Florida, USA	127	phosphate	WR	E	8		1A-SI	02-01-1952		
157	Unidentified, BC, Canada	123		DS	CST			2A-SE			
158	Unidentified, BC, Canada	123		DS	MW	30		2A-SE			
159	Unidentified, Canaca, Mexico	106	copper	US	T	46		1A-OT	1974		
160	Unidentified, Eastern USA	143	coal		MW	150		2A-SI			
161	Unidentified, Green River, Wyoming, USA	134	trona	WR	E	18		3-	1975		
162	Unidentified, Hernando County, Florida, USA	119	limestone	CL	E	6		2A-FN	1977		
163	Unidentified, Hernando County, Florida, USA	105	limestone	US	E	12	3,300,000	1A-OT	09-01-1988	4,600	
164	Unidentified, Hernando County, Florida, USA	124	limestone	DS	E	12		2A-FN	1988		
165	Unidentified, Peace River, Florida, USA	127	phosphate	WR	E	8		1A-SI	03-01-1952		
166	Unidentified, Peace River, Florida, USA	128	phosphate	WR	MW	30		1A-SE	07-01-1951		
167	Unidentified, Peace River, Florida, USA	128	phosphate	WR	MW	6		1A-SE	09-01-1951		
168	Unidentified, Peace River, Florida, USA	120	phosphate	DS	E			1A-SE	02-01-1951		
169	Unidentified, Southern USA	101	copper	US	E	43	500,000	1A-SI	1973	170,000	25,000
170	Union Carbide, Green River, Utah, USA	138	uranium					1A-OT	08-19-1959	8,400	
171	Union Carbide, Maybell, Colorado, USA	141	uranium					1A-U	12-06-1961	280	

ID No.	Mine Name/Location	Summary Page No. (See Note above)	Ore Type	Dam Type	Dam Fill Material	Dam Height (meters)	Storage Volume (Cubic Meters)	Incident Type	Incident Date	Tailings Released (cubic meters)	Tailings Travel (meters)
172	Union Carbide, Uravan, Colorado, USA	111	uranium	US	T	43		2A-SI	03-01-1979		
173	United Nuclear, Churchrock, New Mexico, USA	129	uranium	WR	E	11	370,000	1A-FN	07-01-1979	370.000	110,000
174	Utah construction, Riverton, Wyoming, USA	143	uranium					2A-OT	06-16-1963		
175	Vallenar 1 and 2, Chile	142	copper					1B-OT	1983		
176	Veta de Aqua A, Chile	139	copper					1A-EQ	1981		
177	Veta de Agua B, Chile	139	copper					1A-EQ	1981		
178	Veta de Agua No. 1, Chile	108	copper	US	T	24	700,000	1A-EQ	03-03-1985	280,000	5,000
179	Virginia Vermiculite, Louisa County, Virginia, USA	130	vermiculite	WR	E	9		1A-ST	1984		
180	Western Nuclear, Jeffrey City, Wyoming, USA	135	uranium					1A-SI	1977	40	
181	Western Nuclear, Jeffrey City, Wyoming, USA	139	uranium					1A-ST	03-23-1971		
182	Williamsport Washer, Maury County, Tennessee, USA	140	phosphate			21		1A-U	1970		
183	Williamthorpe, United Kingdom	136	coal					1A-FN	1966		
184	Zlevoto No. 4, Yugoslavia	101	lead/zinc	US	T	25	1,000,000	1A-SI	03-01-1976	300,000	
185	Arcturus, Zimbabwe	105	gold	US	T	25	1.7-2.0 Mt	1A-OT	01-31-1978	39,000 t	300
186	Heath Steele main dam, New Brunswick, Canada	132	copper/zinc	WR	R, E	30		2A-FN	Since 1970s		

ID No.	Mine Name/Location	Summary Page No. (See Note above)	Ore Type	Dam Type	Dam Fill Material	Dam Height (meters)	Storage Volume (Cubic Meters)	Incident Type	Incident Date	Tailings Released (cubic meters)	Tailings Travel (meters)
187	No.3 Tailings Pond, Sipalay, Phillippines	129	copper	WR	MW		37 Mt	1A-FN	11-08-1982	27Mt	
188	Mineral King, BC, Canada	118		CL	CST	6	Small	1B-OT	03-20-1986		
189	Itabirito, Minas Gerais,Brazil	135	iron ore	Gravity	Masonry	30		1A-ST	05-1986	100,000	12,000
190	Rossarden, Tasmania	131		WR	E	7.5	200,000	1B-OT	05-16-1986		
191	Story's Creek, Tasmania	135		Valley side		17	30,000	1B-OT	05-16-1986	Minimal	
192	Pico de Sao Luis, Minas Gerais, Brazil	140			T	20		1A-ER	10-02-1986		
193	No.3 Tailings Pond, Mankayan, Luzon, Phillippines	138	copper		E			1A-ST	10-17-1986		
194	Xishimen, China	99	iron	US	T	31		1A-SI	03-21-1987	2,230	
195	Jinduicheng, Shaanxi Province., China	105	molybdenum	US		40		1A-OT	04-30-1988	700,000	
196	Iron Dyke, Sullivan Mine, Kumberley, BC, Canada	99		US		21		1A-SI	08-23-1991		
197	No.2 Tailings Pond, Padcal, Luzon, Philippines	136	copper				80Mt	A1-FN	01-1992	80Mt	
198	Kojkovac, Montenegro	133	lead/zinc	WR	E		3,500,000	2B-ER	11-1992	none	
199	Itogon-Suyoc, Baguio gold district,Luzon, Phllippines	137	gold					1A-OT	06-26-1993		
200	TD 7, Chingola, Zambia	105	copper	US	T & E	5		1A-OT	08-1993	100t	
202	Merriespruit, nr Virginia, South Africa	111	gold	US paddock	T	31	10Mt	1B-OT	02-22-1994	2.5Mt	2,000
203	Riltec, Mathinna, Tasmania	118		CL	E	7	120,000	2A-SE	01-06-1995	40,000	

ID No.	Mine Name/Location	Summary Page No. (See Note above)	Ore Type	Dam Type	Dam Fill Material	Dam Height (meters)	Storage Volume (Cubic Meters)	Incident Type	Incident Date	Tailings Released (cubic meters)	Tailings Travel (meters)
204	Middle Arm, Launceston, Tasmania	117		CL	E	4	25,000	1A-OT	06-25-1995	5.000	
205	Tailings dam No 1, Omai, Guyana	130	gold	WR	R	44	5,250,000	1A-ER	08-19-1995	4,200,000	
206	Placer Bay, Surigao del Norte, Philippines	131		WR	E	17		1B-SI	09-02-1995	50,000	Out to sea
207	Golden Cross, Waitekauri Valley, New Zealand	136	gold		R	25-30	3Mt	1A-FN	12-1995	None	
208	Marcopper, Marinduque Island, Philippines	138	Copper					1A-ST	03-24-1996	2.4Mt	25,000
209	Los Frailes, nr Seville, Spain	128	Zinc-lead-copper	WR	R	27	15,000,000	1A-FN	04-24-1998	6,800,000	40,000
210	Mirolubovka, Southern Ukraine	111	Iron	US	E and T	32	80,000,000	2A-SI	01-15-1984	none	
211	Balka Chuficheva, Russia	100	Iron	US	CST	25	27,000,000	1A-SI	01-20-1981	3,500,000	1,300
212	Bekovsky, Western Siberia	99	Coal	US	Argillite, aleurolite	53	52,000,000	US	03-25-1987	None	
213	Fernandinho, nr Belo Horizonte, Brazil	103	Iron	US	T	40		1A-SI			
214	Minera Sera Grande: Crixas, Goias, Brazil	121	Gold	DS then US	CST	41	2.25 Mt	2A-SI	02-1994	None	
215	Forquilha, Brazil	138	Iron					1A-OT			
216	Williamthorpe, UK	137	Coal		MW			1A-OT	03-24-1966		
217	Stoney Middleton, UK	136						1A-SI	02-08-1968		
218	Maritsa Istok 1, Bulgaria	110	Ash/cinder		Ash	15	52,000,000	1A-ER	03-01-1992	500,000	

ID No.	Mine Name/Location	Summary Page No. (See Note above)	Ore Type	Dam Type	Dam Fill Material	Dam Height (meters)	Storage Volume (Cubic Meters)	Incident Type	Incident Date	Tailings Released (cubic meters)	Tailings Travel (meters)
219	Madjarevo, Bulgaria	107	Lead/zinc/ gold	US	T	40	3,000,000	1A-ST	04-1975	250,000	
220	Sgurigrad, Bulgaria	102	Lead/zinc/ copper/silver	US	T	45	1,520,000	1A-SI	05-01-1996	220,000	6,000
221	Baia Mare, Romania	107	Gold	DS then US	T	A few m	800,000 (estimate)	1A-ST	01-30-2000	100,000 (estimate)	See Appendix

APPENDIX (continued)

4. BRIEF DESCRIPTION OF THE 221 CASES

- The described cases are those listed in Section 3 of the Appendix.
- Organizations that collected the case histories, and the corresponding Incident Nos. are listed in Section 1 of the Appendix.
- For Abbreviations, see Section 2 of the Appendix
- In the incident summaries that follow, the incidents were sorted in the following order:

Dam Type
and for each Dam Type, Incident Type
and for each Incident Type, Incident Cause
and for each Incident Cause, Date of incident, in reverse chronological order
(most recent listed first)

Where no information is available (NR or U), the cases are listed at the end of each category.

Incident Summaries provided on the pages that follow.

DAM TYPE: US INCIDENT TYPE: 1A
INCIDENT CAUSE: SI

Incident No.: 196
Dam/Mine Name: Iron Dyke
Mine Location: Sullivan mine, Kimberley, British Columbia Ore/Tailings Type:
Dam Height (m): 21;
Dam Type: US Dam Fill Material:
Impoundment Volume (Cu. m):
Incident Information:
Date: 23rd August 1991
Incident Type: 1A Cause: SI
Quantity of Tailings Released (Cu. m):
Tailings Travel Distance (m):
Incident Description:
A length of 300m out of a ring dyke 1,500m long, failed by rotational slip. A foundation embankment of tailings had been built in 1951, and the new ring dyke built in 1975. It was raised every year and heavy construction equipment was running on the dyke. Failure thought to be due to excess pore pressures developed in old foundation embankment due to weight of machines and raised height of dyke. Out of action for a year, cost of remedial works over a million Canadian dollars.
Source: Cominco Ltd., Vancouver, Canada.

Incident No.: 116
Dam/Mine Name: Stancil
Mine Location: Perryville, MD, USA
Ore/Tailings Type: sand & gravel
Dam Height (m): 9
Dam Type: US Dam Fill Material: E
Impoundment Volume (cu. m): 74,000
Incident Information:
Date: 08-25-1989 Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m): 38,000
Tailings Travel Distance (m): 100
Incident Description:
Capping of the tailings was in progress when slope failure breached the embankment over a width of 280 feet. The clayey silt cap, which ranged from 8 to 12 feet thick, is thought to have elevated pore pressures in the clayey tailings impounded by the embankment. A contributing factor may have been saturation of the embankment fill by above- -normal precipitation prior to the failure. The tailings flowslide blocked a creek near the embankment toe, diverting creek discharge, dislodging trees, and destroying tidal vegetation over an area of 1.2 acres beyond the embankment toe.
Source: Maryland Dept. of Nat. Res., Dam Safety Div.

Incident No.: 212
Dam/Mine Name: Bekovsky
Mine Location: Kuznetsk coal basin, Western Siberia, Russia
Ore/Tailings Type: coal/loam, clay
Dam Height (m): 53
Dam Type: US Dam Fill Material: Argillites, aleurolites, loam, clay - hydraulic fill
Impoundment Volume (Cu. m): 52,000,000
Incident Information:
Date: 3-25-1987 at 16:45
Incident Type: lA Cause: SI
Quantity of Tailings Released (cu. m): none
Tailings Travel Distance (m):
Incident Description;
Starter dam 20m high. Raised with 5m high dykes. 7th dyke was being placed over a frozen beach, to raise dam height to 53m. Rotational slip 15m high x 250m long lowered crest 3m and bottom of slip moved 3m downstream. Caused by high rate of filling (260,000 cu m during 2 ½ months). Produced high pore pressures retained under frozen layer that reduced shear strength to very low value. Inspection of 7th dyke in June 1988 showed body completely destroyed by longitudinal cracks, indicating continuing movement. Piezometers were installed and the dam was stabilized with toe weighting. When the dam reached 60m high, no deformations reported.
Source: ICOLD Tailings Committee

Incident No.: 194
Dam/Mine Name: Xishimen
Mine Location: China
Ore/Tailings Type: iron
Dam Height (m): 31
Dam Type: US; Dam Fill Material: T
Impoundment Volume (Cu. m):
Incident Information:
Date: 3-21-1987 at 02:40
Incident Type: lA Cause: SI
Quantity of Tailings Released (cu. m): 2,230
Tailings Travel Distance (m):
Incident Description:
Blocked decant caused pond water to rise too high, causing failure of downstream slope, formation of a breach and escape of tailings
Source: ICOLD Tailings Committee.

Incident No.: 117
Dam/Mine Name: Stava
Mine Location: Northern Italy
Ore/Tailings Type: fluorite
Dam Height (m): 29.5 upper and 26.0 lower Dam Type: US lower, CL upper.
Dam Fill Material: CST

Impoundment Volume (cu. m): 300,000
Incident Information:
Date: 07-19-1985 Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m): 190,000
Tailings Travel Distance (m): 4,000
Incident Description:
Two upstream-type impoundments had been constructed with the upper embankment founded partially on the slimes deposit of the lower. Embankment slopes ranged from 1.2:1 to 1.5:1. Failure of the upper embankment caused the lower embankment to also fail, with the loss of 269 lives in the resulting tailings flowslide. Mechanisms that triggered the failure may have included excess pore pressures in soft foundation tailings due to embankment raising, seepage of ponded water into embankment sands, pressurization of a blocked decant conduit, or excess pore pressures in natural foundation soils in response to rainfall or embankment seepage. For more details, see Section 6.
Source: Berti, et al, 1988; Chandler RJ and Tosatti G (1995*)* The Stava tailings dam failure, Italy, July 1985. Geotechnical Engineering, Pub Instn Civil Engrs, London, vol. 113, no 2, pp 67-79.

Incident No.: 211
Dam/Mine Name: Balka Chuficheva
Mine Location: Lebedinsky (Kursk Magnetic Anomaly), Russia
Ore/Tailings Type: iron/chalk, sand
Dam Height (m): 25
Dam Type: US
Dam Fill Material: Hydraulic fill sand
Impoundment Volume (cu. m): 27,000,000
Incident Information:
Date: 1-20-1981 at 06:30
Incident Type: lA Cause: SI
Quantity of Tailings Released (cu. m): 3,500,000
Tailings Travel Distance (m): 1,300
Incident Description:
The dam retained hydraulically placed chalky and sandy overburden from mine stripping. A breach, which occurred at right end of dam where it joined the valley side, became 55m wide. Resulting ravine formed in the impoundment up to 20m deep, max. width 400m and 1km long. Primary cause: violation of technology in performing hydrodumping works caused pond to be moved down to the dam.
Source: ICOLD Tailings Committee.

Incident No.: 94
Dam/Mine Name: No.3 Tailings Dam, Phelps-Dodge
Mine Location: Tyrone, NM, USA
Ore/Tailings Type: copper
Dam Height (m): 66
Dam Type: US Dam Fill Material: CST
Impoundment Volume (cu. m): 2,500,000
Incident Information:
Date: 10-13-1980, night
Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m): 2,000,000
Tailings Travel Distance (m): 8,000
Incident Description:
The embankment was being raised continuously by constructing perimeter dikes of cycloned sand tailings and discharge of slimes cyclone overflow to the impoundment. During the night, flowsliding occurred through a breached section 215m wide and 35m deep. Tailings flowed down slope and up opposite side, then 8km down the valley. The failure is attributed to a rapid raising rate and insufficient dissipation of pore pressures in the embankment. Alternative explanations advanced include breach due to pipeline rupture as a triggering mechanism for the flowslide.
Source: New Mexico State Engineers Office; Phelps Dodge, Phoenix.

Incident No.: 96
Dam/Mine Name: Pit No. 2
Mine Location: Western Australia, Australia
Ore/Tailings Type: rare earth
Dam Height (m): 9
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1977 Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Tailings produced by mining of ocean beach sands for ilmenite, rutile, and zircon were deposited in a mined-out pit. The pit bottom sloped upward at a 3-4 degree angle, and deposition of tailings by direct spigotting proceeded from the lowest end of the pit and progressed upward in increments behind low tailings dikes. This procedure resulted in deposition of slimes beneath the dikes. An initial localized dike failure in 1976 was attributed to a high phreatic surface in the dike resulting from rainfall and high pond operating levels. A larger failure one year later showed evidence of upthrusting at the toe of the pit, block-type downslope movement of tailings and sand boils within the failed mass. This larger failure had no obvious trigger mechanism, and it was concluded that excess pore pressures in permeable layers

within the tailings or the pit floor initiated liquefaction
Source: Williams, 1979

Incident No.: 184
Dam/Mine Name: Zlevoto No. 4
Mine Location: Yugoslavia
Ore/Tailings Type: lead/zinc
Dam Height (m): 25 Dam Type: US
Dam Fill Material: T
Impoundment Volume (cu. m): 1,000,000
Incident Information:
Date: 03-01-1976 Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m): 300,000
Tailings Travel Distance (m):
Incident Description:
Four tailings impoundments had been constructed in a sidehill configuration by the upstream method using direct tailings spigotting. Embankment slopes ranged from 2:1 to 2.5:1. Failure was attributed to a high phreatic surface and seepage breakout on the embankment face produced by high fines content of the spigotted tailings and insufficient permeability of starter dike materials. The tailings flowslide reached and polluted a nearby river.
Source: Sandic, 1979

Incident No.: 37
Dam/Mine Name: Deneen Mica
Mine Location: Yancey County, NC, USA
Ore/Tailings Type: mica
Dam Height (m): 18
Dam Type: US Dam Fill Material: CST
Impoundment Volume (cu. m): 300,000
Incident Information:
Date: 06-01-1974 Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m): 38,000
Tailings Travel Distance (m): 30
Incident Description:
The dam was constructed of cycloned sands which were hauled by truck and received variable compaction. Slimes were spigotted from the rear of the impoundment, resulting in very soft materials beneath the upstream sand raises. These conditions, combined with the steep 1.5:1 embankment face, resulted in marginal stability. During a heavy rain, the dam overtopped and deep gullies were eroded into the embankment face. This loss of support caused sliding of the downstream slope over its full height and over a width of 200 ft. Slimes were released to an adjacent river. The breached section was reconstructed to prevent further release of tailings, and the impoundment was abandoned due to marginal stability of the remaining portions of the embankment.
Source: Brawner, 1979; North Carolina Dept. of Environ. Health and Nat. Res., Land Quality Section

Incident No.: 169
Dam/Mine Name: Unidentified
Mine Location: Southwestern US, USA
Ore/Tailings Type: copper
Dam Height (m): 43 Dam Type: US
Dam Fill Material: E
Impoundment Volume (cu. m): 500,000
Incident Information:
Date: 1973 Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m): 170,000
Tailings Travel Distance (m): 25,000
Incident Description:
The dam included a 60 foot high zoned earthfill starter dike. Prior to failure, two 15-foot high upstream raises had been added using perimeter dikes of uncompacted clayey soils derived from weathered shales. A third raise of cycloned sand tailings was under construction when the uncompacted shale dikes slumped from increased load and pore pressure. The resulting embankment breach took the form of a narrow gulley down to the level of the starter dike crest, and released about one-third of the impoundment contents in the form of a tailings flowslide. Tailings reached streams and rivers as far as 15 miles away.
Source: Wahler and Schlick, 1976; Lucia, 1981

Incident No.: 100
Dam/Mine Name: Ray Mine
Mine Location: Hayden, AZ, USA
Ore/Tailings Type: copper
Dam Height (m): 52 Dam Type: US
Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 12-02-1972 Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Slope instability along a 500-ft section of the embankment caused failure to occur. Instability is believed to have been related to saturation and perched seepage conditions along a layer of slimes deposited within the embankment 20 years earlier. A wetted zone had been present on the embankment face at the location where failure occurred. Released tailings covered a small section of an adjacent railroad.
Source: Anecdotal

Incident No.: 75
Dam/Mine Name: Maggie Pye
Mine Location: United Kingdom

Ore/Tailings Type: china clay
Dam Height (m): 18
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1970 Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m): 15,000
Tailings Travel Distance (m): 35
Incident Description:
Slope failure occurred immediately after completion of a perimeter dike to raise the embankment and following a period of heavy rainfall. High pore pressures and addition of the perimeter dike fill, possibly also supplemented by vibrations of construction equipment, are thought to have been contributing factors.
Source: Ripley, 1972

Incident No.: 220
Dam/Mine Name: Sgurigrad
Mine Location: Western Bulgaria
Ore/Tailings Type: Lead, zinc, copper, silver
Dam Height (m): 45 Dam Type: US
Dam Fill Material: T
Impoundment Volume (cu. m): 1,520,000
Incident Information:
Date: 05-01-1966
Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m): 220,000
Tailings Travel Distance (m): 6,000
Incident Description:
A rise in pond level during 3 days of heavy rains caused a sudden loss of stability of the dam and liquefaction of the tailings, although the dam was not overtopped. The wave destroyed half of village 1 km downstream, with 107 victims. For more details see Section 6.
Source: ICOLD Tailings Committee

Incident No.: 54
Dam/Mine Name: Grootvlei
Mine Location: South Africa
Ore/Tailings Type: gold
Dam Height (m): Dam Type: US
Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1956 Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
An embankment slope failure occurred after a prolonged period of rain when water covered the tailings beach and encroached upon the embankment crest. About one third of the impoundment contents were lost in the ensuing tailings flowslide.
Source: Donaldson, et al, 1976

Incident No.: 66
Dam/Mine Name: Kimberley
Mine Location: British Columbia, Canada
Ore/Tailings Type: iron
Dam Height (m): Dam Type: US
Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1948 Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m): 1,100,000
Tailings Travel Distance (m):
Incident Description:
The embankment was constructed by direct spigotting of tailings using upstream raising procedures. The foundation is believed to have consisted of low-permeability glacial till. The failure is attributed to freezing of the dam face during a period of high snowmelt and spring runoff that raised the phreatic surface and caused slope instability. A large tailings flowslide was triggered that moved toward, but apparently did not reach, the St. Mary River a few miles away. Frozen blocks of material were observed in the flow failure mass.
Source: Robinson and Toland, 1979

Incident No.: 62
Dam/Mine Name: Kennecott
Mine Location: Garfield, UT, USA
Ore/Tailings Type: copper
Dam Height (m): Dam Type: US
Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1941 Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Breach of the embankment triggered a tailings flowslide. Accounts indicate that rainfall proceeding the failure which may have increased dike saturation, and that "minor shearing" may have initiated the failure.
Source: McIver, 1961; Smith, 1969

Incident No.: 21
Dam/Mine Name: Captains Flat Dump 6A
Mine Location: Australia
Ore/Tailings Type: copper
Dam Height (m):
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1939 Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:

No details of the failure are available, but released tailings were deposited in an adjacent river and caused widespread damage to river flats up to 10 miles downstream.
Source: Ash, 1976

Incident No.: 110
Dam/Mine Name: Simmer and Jack
Mine Location: South Africa
Ore/Tailings Type: gold
Dam Height (m): Dam Type: US
Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1937 Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Embankment breach occurred after a period of rain and in an area weakened by excavation. The tailings flowslide traveled a considerable distance and engulfed a mine train.
Source: Donaldson, et al, 1976

Incident No.: 213
Dam/Mine Name: Fernandinho
Mine Location: nr Ouro Preto highway, 40km from Belo Horizonte, Brazil
Ore/Tailings Type: iron
Dam Height (m): 40
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: lA Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Dam had not been raised since 1984. A central dyke had been built to divide the impoundment. Tailings were placed in one while the other drained and dried. Dried tailings then dug out and placed elsewhere. A truck was on the crest and became stuck in the mud. Two others sent to help and while this going on slip failure began. The crest was 2m above the tailings, but they were placed away from the dam, which had water against it. The rotational movements soon allowed overtopping. A strong noise was heard and the staff of a laboratory 500m d/s ran for their lives up the valley side. The liquefied tailings swept down the zigzag valley like water, but stripping all vegetation. D/s slope of dam was 1 on 1.1 ($\varphi=42^0$)For more details, see Section 6.
Source: ICOLD Tailings Committee

Incident No.: 126
Dam/Mine Name: Unidentified
Mine Location:
Ore/Tailings Type:
Dam Height (m): Dam Type: US
Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A liquefaction flowslide is illustrated that is believed to be related to lateral strains, differential movements, and cracking that occurred at a high-angle corner of the tailings embankment.
Source: Casagrande and McIver, 1971

Incident No.: 129
Dam/Mine Name: Unidentified
Mine Location:
Ore/Tailings Type:
Dam Height (m): Dam Type: US
Dam Fill Material: MW
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The upstream embankment was being raised on an overall slope of 4:1 by constructing perimeter dikes of hauled fill. Instability of the embankment was attributed to vibrations produced by the mine railroad transporting and dumping perimeter dike fill. Tailings flowsliding resulted.
Source: Casagrande and McIver, 1971

Incident No.: 139
Dam/Mine Name: Unidentified
Mine Location: South Africa
Ore/Tailings Type: gold
Dam Height (m): Dam Type: US
Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A failure is illustrated whereby a rotational slide in the embankment face triggered partial liquefaction of the retained slimes. The slide was related to long-term retention of water on the impoundment surface at variance with conventional operating practice in South Africa.
Source: Blight and Steffen, 1979

DAM TYPE: US INCIDENT TYPE: 1A
INCIDENT CAUSE: SE

Incident No.: 7
Dam/Mine Name: Bafokeng
Mine Location: South Africa
Ore/Tailings Type: platinum
Dam Height (m): 20
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m): 13,000,000
Incident Information:
Date: 1974 Incident Type: 1A Cause: SE
Quantity of Tailings Released (cu. m): 3,000,000
Tailings Travel Distance (m): 45,000
Incident Description:
The embankment failed by concentrated seepage and piping through cracks. Embankment breach released a flow failure that reached a river, transporting tailings far downstream. The tailings flowslide inundated a mine shaft, killing 12 miners underground.
Source: Jennings, 1979; Rudd, 1979; Lucia, 1981

Incident No.: 10
Dam/Mine Name: Berrien
Mine Location: France
Ore/Tailings Type: kaolin
Dam Height (m): 9
Dam Type: US Dam Fill Material: R
Impoundment Volume (cu. m):
Incident Information:
Date: 1974 Incident Type: 1A Cause: SE
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The starter dike for an upstream embankment partially breached due to seepage and piping after heavy rains. Damage was repaired and plans were made for raising the dam an additional 20 m by upstream methods.
Source: Londe, et al, 1976

Incident No.: 154
Dam/Mine Name: Unidentified
Mine Location: TX, USA
Ore/Tailings Type: gypsum
Dam Height (m): 16
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1966 Incident Type: 1A Cause: SE
Quantity of Tailings Released (cu. m): 130,000
Tailings Travel Distance (m): 300
Incident Description:
Operation of the impoundment began in 1962 with the construction of a clay starter dike and a sand underdrainage system. The failure is attributed to seepage-related slumping and piping that initiated at the toe and progressed until breach of the embankment and tailings flowsliding occurred. The drainage system is believed to have been ineffective due to insufficient permeability of the sand.
Source: Kleiner, 1976; Lucia, 1981

Incident No.: 25
Dam/Mine Name: Castle Dome
Mine Location: AZ, USA
Ore/Tailings Type: copper
Dam Height (m):
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 1A Cause: SE
Quantity of Tailings Released (cu. m): 150,000
Tailings Travel Distance (m): 100
Incident Description:
The failure of a sand dike occurred due to excessive seepage and high phreatic conditions. The repaired section incorporated many pumped vertical wells to improve internal drainage.
Source: Lenhart, 1950

DAM TYPE: US INCIDENT TYPE: 1A
INCIDENT CAUSE: FN

Incident No.: 102
Dam/Mine Name: Royster
Mine Location: Mulberry, FL, USA
Ore/Tailings Type: gypsum
Dam Height (m): 21
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1982 Incident Type: 1A Cause: FN
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The gypsum embankment was built on soft phosphatic clay slimes, which caused a 900-ft section of the embankment slope to fail. An unknown quantity of low-pH process water was released.
Source: Anecdotal

Incident No.: 58
Dam/Mine Name: Hollinger
Mine Location: Canada
Ore/Tailings Type: gold
Dam Height (m): 15
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1944 Incident Type: 1A Cause: FN
Quantity of Tailings Released (cu. m):

Tailings Travel Distance (m):
Incident Description:
The dam was constructed on 5 to 17 feet of muskeg overlying alluvial sands, clays, and clayey silts. Between 1936 and 1944, 17 separate episodes of foundation sliding occurred, producing subsidence of the embankment crest and lateral spreading. Failures occurred rapidly (within a few minutes) and without warning. Crest subsidence ranged from 4-8 feet when the embankment height was about 15 feet to 20-25 feet, after embankment raising to a height of 50 ft.
Source: Blackshaw, 1951

Incident No.: 63
Dam/Mine Name: Kennecott
Mine Location: Garfield, UT, USA
Ore/Tailings Type: copper
Dam Height (m):
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1942 Incident Type: 1A Cause: FN
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Dam breach was caused by shear failure in weak foundation materials.
Source: McIver, 1961

DAM TYPE: US INCIDENT TYPE: 1A INCIDENT CAUSE: OT

Incident No.: 200
Dam/Mine Name: TD 7
Mine Location: Chingola, Zambia
Ore/Tailings Type: copper
Dam Height (m): 5
Dam Type: US Fill Material: T and E
Impoundment Volume (cu. m):
Incident Information:
Date: 08-1993
Incident Type: 1A Cause: OT
Quantity of Tailings Released (tonnes): 100
Tailings Travel Distance (m):
Incident Description:
Rainstorm caused overilow at time when rate of tailings deposition had increased. Part of dam collapsed. Spillway not adequate for flood.
Source: ZCCM Ltd., Kalulushi, Zambia.

Incident No.: 163
Dam/Mine Name: Unidentified
Mine Location: Hernando County, FL, USA
Ore/Tailings Type: limestone
Dam Height (m): 12
Dam Type: US Dam Fill Material: E
Impoundment Volume (cu. m): 3,300,000
Incident Information:
Date: 09-01-1988
Incident Type: 1A Cause: OT
Quantity of Tailings Released (cu. m): 4,600
Tailings Travel Distance (m):
Incident Description:
The embankment was raised with clay fill over tailings similar in nature to phosphatic clay slimes derived from limestone washing operations. Local shear failures and displacement of soft tailings occurred during construction of upstream raises, and downstream embankment slopes were as steep as 1.3:1. Overtopping of the embankment occurred due to excessive water accumulation during heavy rainfall. Overtopping may have been promoted by settlement of the portion of the embankment constructed on soft tailings, or by shear failures on the steep downstream slope. The narrow breach that resulted released all of the impounded water (about 2 million gallons) but only a limited quantity of tailings, and a major flowslide did not occur. The absence of flowsliding was attributed to abnormally high consolidation and undrained shear strength in the lower portion of the impounded clayey slimes due to underdrainage by a pervious foundation sand layer.
Source: Anecdotal

Incident No.: 195
Dam/Mine Name: Jinduicheng
Mine Location: Shaanxi province, China
Ore/Tailings Type: molybdenum
Dam Height (m): 40
Dam Type: US Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 04-30-1988 at 03:00
Incident Type: lA Cause: OT
Quantity of Tailings Released (cu. m): 700,000
Tailings Travel Distance (m):
Incident Description:
Spillway blockage raised phreatic surface, causing rotational slip in central part of dam: a catastrophic failure; 20 killed
Source: ICOLD Tailings Committee.

Incident No.: 185
Dam/Mine Name: Arcturus
Mine Location: Zimbabwe
Ore/Tailings Type: Gold
Dam Height (m): 25
Dam Type: US paddock Fill Material: T
Impoundment Volume (Mt): 1.7 - 2.0
Incident Information:
Date: 01-31-1978 at 19:30

Incident Type: lA Cause: OT
Quantity of Tailings Released (tons): 30,000
Tailings Travel Distance (m): 300
Incident Description:
Early in the morning, following continuous rain over several days (seasonal total rainfall above average), a breach 55m wide suddenly developed, releasing a flow slide of tailings, blocking and contaminating public waterway. Minor damage to local village. One child killed and another injured.
Source: Chamber of Mines, Harare, Zimbabwe.

Incident No.: 159
Dam/Mine Name: Unidentified
Mine Location: Cananea, Mexico
Ore/Tailings Type: copper
Dam Height (m): 46
Dam Type: US   Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1974 Incident Type: 1A   Cause: OT
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Embankment perimeter dikes were constructed of and upon fine tailings discharged from the rear of the impoundment. Overall embankment slopes were 1.5:1. Overtopping resulted in breach of the embankment, loss of impounded water, and erosional-type gullying of tailings within the impoundment. Flow sliding of the tailings mass, however, did not occur.
Source: Anecdotal

Incident No.: 49
Dam/Mine Name: Galena Mine
Mine Location: Wallace, ID, USA
Ore/Tailings Type: silver
Dam Height (m): 9 Dam Type: US
Dam Fill Material: MW
Impoundment Volume (cu. m):
Incident Information:
Date: 01-15-1974
Incident Type: 1A   Cause: OT
Quantity of Tailings Released (cu. m): 3,800
Tailings Travel Distance (m): 610
Incident Description:
Three tailings impoundments in a sidehill configuration adjoined each other within a narrow valley with a creek at their toe. During a rain-on-snow event, flooding on the creek reached estimated 100-yr. recurrence interval flows. A culvert in the creek upstream from the impoundments became blocked by debris, diverting a large portion of the streamflow into the uppermost impoundment. Lacking sufficient decant spillway capacity for these flows the uppermost embankment breached by overtopping, resulting in cascade failure of all three impoundments. Tailings released in the failure covered about 5 acres, including a short section of highway and railroad track. This incremental damage was insignificant in relation to general flood damages to public and private property.
Source: Montana Div. State Lands

Incident No.: 41
Dam/Mine Name: Earth Resource
Mine Location: Cuba, NM, USA
Ore/Tailings Type: copper
Dam Height (m): 21
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1973 Incident Type: 1A Cause: OT
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Improper operation and inadequate tailings beach deposition allowed ponded water to encroach on the embankment crest and overtopping failure to occur. No flood or extreme precipitation event was associated with this failure.
Source: New Mexico State Engineers Office

Incident No.: 152
Dam/Mine Name: Unidentified
Mine Location: MS, USA
Ore/Tailings Type: gypsum
Dam Height (m): 15
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1970 Incident Type: 1A   Cause: OT
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Overtopping occurred due to accumulation of water in the impoundment from hurricane rainfall. The embankment breached and water was released, but flow failure of the tailings did not develop. The breach was repaired and the embankment placed back into service.
Source: Anecdotal

Incident No.: 115
Dam/Mine Name: St. Joe Lead
Mine Location: Flat River, MO, USA
Ore/Tailings Type: lead
Dam Height (m): 15
Dam Type: US   Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1940 Incident Type: 1A Cause: OT
Quantity of Tailings Released (cu. m):

Tailings Travel Distance (m):
Incident Description:
During embankment raising, a portion of the tailings was discharged from the rear of the impoundment producing a narrow sand tailings beach and accumulation of water near the embankment crest. This water was decanted with a vertical-riser decant system, but inattention to flashboard placement allowed ponded water to rise and overtop the embankment. A narrow breach resulted, with loss of some tailings. The breach was filled with mine waste rock, the impoundment was placed back into service, and the embankment was raised to an ultimate weight of 110 feet without further incident.
Source: Anecdotal

DAM TYPE: US INCIDENT TYPE: 1A
INCIDENT CAUSE: ST

Incident No.: 221
Dam/Mine Name: Aurul S.A. Mine
Mine Location: Baia Mare, Romania
Ore/Tailings Type: Gold
Dam Height (m): A few m; future final height 20 m Dam Type: Initially DS, later US
Dam Fill Material: T (cycloned)
Impoundment Volume (cu. m): Approx. 800,000
Incident Information:
Date: 1-30-2000
Incident Type: 1A Cause: ST
Quantity of contaminated effluent released (cu. m): 100,000 (Estimate)
Tailings Travel Distance (m):
Incident Description:
After extreme weather conditions (ice and snow on the tailings pond, high precipitation: $36L/m^2$), the tailings deposited on the inner embankment (starter dam) became saturated. Stability was affected, causing local displacement, and this subsequently developed into a breach of approximately 23 m in length. The effluent released through the breach filled the area between the starter dam and the outer perimeter dam, both surrounding the impoundment (93 hectares in area), and spilled over the outer embankment. Around 100,000 m^3 of cyanide-rich (50-100 tonnes) effluent contaminated also with some heavy metals was released into the Somes and Tisza rivers and then into the Danube, finally reaching the Black Sea. Significant contamination occurred over a stretch of 150 to 180 m, then became more and more diluted. It caused significant fishkill and destruction of aquatic species in the river system.
Source: UNEP/OCHA Assessment Mission Report, 2000

Incident No.: 59
Dam/Mine Name: Homestake
Mine Location: Milan, NM, USA
Ore/Tailings Type: uranium
Dam Height (m): 21 Dam Type: US
Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 02-01-1977
Incident Type: 1A Cause: ST
Quantity of Tailings Released (cu. m): 30,000
Tailings Travel Distance (m):
Incident Description:
A tailings slurry pipeline on the dam crest ruptured due to a blockage by freezing and pressure buildup. The slurry released eroded a "v"-shaped breach in the embankment, which in turn released tailings and an estimated 2 to 8 million gallons of impounded effluent. All released materials were contained on the mine site.
Source: Teknekron, 1978; New Mexico State Engineers Office.

Incident No.: 219
Dam/Mine Name: Madjarevo
Mine Location: Eastern Rodope Mountain, South-Eastern Bulgaria
Ore/Tailings Type: lead, zinc, gold
Dam Height (m): 40
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m): 3,000,000
Incident Information:
Date: 04-1975 Incident Type: 1A Cause: ST
Quantity of Tailings Released (cu. m): 250,000
Tailings Travel Distance (m):
Incident Description:
Rising of tailings above design level caused overloading of the decant tower and collectors, resulting in structural failure. Tailings flowed through tower and collector into river and backwater of a water retention downstream. For more details see Section 6.
Source: ICOLD Tailings Committee

Incident No.: 23
Dam/Mine Name: Casapalca
Mine Location: Peru
Ore/Tailings Type:
Dam Height (m): 107
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 1A Cause: ST
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A number of tailings dams up to 350 ft in height

were developed over the 50-year mine life in steep, narrow valleys. All utilized a complex array of pipe-type decant structures and inadequately-sized stream bypass channels. Five separate dam failures resulted from failure of these bypass or decant systems.
Source: Brawner, 1979

DAM TYPE: US INCIDENT TYPE: 1A
INCIDENT CAUSE: EQ

Incident No.: 30
Dam/Mine Name: Cerro Negro No. 4
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 40
Dam Type: US Dam Fill Material: CST
Impoundment Volume (cu. m): 2,000,000
Incident Information:
Date: 03-03-1985
Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m): 500,000
Tailings Travel Distance (m): 8,000
Incident Description:
The dam was constructed using a combination of upstream and centerline methods, with downstream slopes of 1.7:1. The dam failed by liquefaction during the M7.8 earthquake of March 3, 1985. Slimes flowed through a narrow breach, reached a creek, and were deposited downstream for a distance of 8 km.
Source: Castro and Troncoso, 1989; Troncoso, 1988

Incident No.: 178
Dam/Mine Name: Veta de Agua No. 1
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 24
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m): 700,000
Incident Information:
Date: 03-03-1985
Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m): 280,000
Tailings Travel Distance (m): 5,000
Incident Description:
The dam was constructed using both upstream and centerline methods with downstream slopes of 1.5:1. During the M7.8 earthquake of March 3, 1985, the dam failed by liquefaction.
Source: Castro and Troncoso, 1989; Troncoso, 1988

Incident No.: 84
Dam/Mine Name: Mochikoshi No. 1
Mine Location: Japan
Ore/Tailings Type: gold
Dam Height (m): 28
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m): 480,000
Incident Information:
Date: 01-14-1978
Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m): 80,000
Tailings Travel Distance (m): 7,000
Incident Description:
The embankment was constructed with a rockfill starter dike and had slopes of about 3:1. Failure occurred by liquefaction during the M7.0 Izu-Oshima-Kinkai earthquake. The flowslide reached and flowed down a river for 7-8 km, causing one fatality.
Source: Marcuson, 1979; Okusa, et.al., 1980

Incident No.: 85
Dam/Mine Name: Mochikoshi No.2
Mine Location: Japan
Ore/Tailings Type: gold
Dam Height (m): 19
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m): 480,000
Incident Information:
Date: 01-15-1978
Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m): 3,000
Tailings Travel Distance (m): 150
Incident Description:
The embankment was constructed with rockfill starter dikes, and had slopes of 2.5:1 to 3:1. Liquefaction failure occurred the day after the January 14, 1978 M 7.0 Izu–Oshima– Kinkai earthquake, and about 5 hours after the two after shocks of M 5.4 and M 5.8.
Source: Marcuson, 1979; Okusa, et. al., 1980

Incident No.: 57
Dam/Mine Name: Hokkaido
Mine Location: Japan
Ore/Tailings Type:
Dam Height (m): 12
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m): 300,0
Incident Information:
Date: 1968 Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m): 90,000
Tailings Travel Distance (m): 150
Incident Description:
The embankment included a low rockfill starter dike at the toe, and was constructed with 3:1 slopes. The embankment failed by liquefaction during the M7.8 Tokachi-Oki earthquake, and the resulting flowslide reached and crossed a river at the downstream toe of the embankment.
Source: Ishihara, et. al., 1990

Incident No.: 12
Dam/Mine Name: Bellavista
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 20
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m): 450,000
Incident Information:
Date: 03-28-1965
Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m): 70,000
Tailings Travel Distance (m): 800
Incident Description:
At the time of the M7-7 1/4 1965 La Liqua earthquake, only 8m separated the edge of the ponded water from the crest of this upstream-type embankment with slopes as steep as 1.4:1.0. According to eyewitness accounts, the face of the embankment slid first, followed by flowsliding of the tailings behind the breach.
Source: Dobry and Alvarez, 1967

Incident No.: 29
Dam/Mine Name: Cerro Negro No. 3
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 20 Dam Type: US
Dam Fill Material: T
Impoundment Volume (cu. m): 500,000
Incident Information:
Date: 03-28-1965
Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m): 85,000
Tailings Travel Distance (m): 5,000
Incident Description:
The upstream type dam experienced strong shaking during the M7-7 1/4 La Ligua earthquake. Eyewitness accounts indicate that surface waves were generated on the liquefied slimes for as long as 1-1/2 minutes after shaking ceased. These waves of liquefied slimes eroded the small perimeter dike on the embankment crest, breaching the embankment and producing a tailings flowslide.
Source: Dobry and Alvarez, 1967

Incident No.: 45
Dam/Mine Name: El Cobre Old Dam
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 35
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m): 4,250,000
Incident Information:
Date: 1965 Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m): 1,900,000
Tailings Travel Distance (m): 12,000
Incident Description:
The embankment failed catastrophically in the M7-7 1/4 La Ligua earthquake of March 28, 1965 by liquefaction. The dam had been constructed according to the upstream method by spigotting from flumes on the crest and was in use as an emergency impoundment at the time of the earthquake. Embankment slopes as steep as 1.2:1.0 and the presence of slimes layers near the face suggest that static stability may have been marginal even before the earthquake. The tailings flowslide destroyed the town of El Cobre, killing more than 200.
Source: Dobry and Alvarez, 1967

Incident No.: 55
Dam/Mine Name: Hierro Viejo
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 5
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 03-28-1965
Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m): 800
Tailings Travel Distance (m): 1,000
Incident Description:
This upstream dam experienced liquefaction flow failure during the M7-7 1/4 La Ligua earthquake. The liquefied tailings traveled a distance of 1 km on the gently sloping valley floor without doing any damage.
Source: Dobry and Alvarez, 1967

Incident No.: 69
Dam/Mine Name: La Patagua New Dam
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 15
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 03-28-1965
Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m): 35,000
Tailings Travel Distance (m): 5,000
Incident Description:
The New Dam was being used to retain mill process water at the time of the M7-7 1/4 1965 La Liqua earthquake, and pond water levels retained by the upstream-type embankment were relatively high. Embankment slopes were a maximum of 1.4:1.0. The dam failed by liquefaction during the earthquake, but no damage was reported.
Source: Dobry and Alvarez, 1967

Incident No.: 71
Dam/Mine Name: Los Maquis No. 3
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 15
Dam Type: US Dam Fill Material:
Impoundment Volume (cu. m): 43,000
Incident Information:
Date: 03-28-1965
Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m): 21,000
Tailings Travel Distance (m): 5,000
Incident Description:
The embankment failed during the M7-7 1/4 La Ligua earthquake by liquefaction. The dam was constructed by the upstream method with slopes as steep as 1.4:1.0. No damage from the resulting flowslide was reported.
Source: Dobry and Alvarez, 1967

Incident No.: 9
Dam/Mine Name: Barahona
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 61
Dam Type: US Dam Fill Material: CST
Impoundment Volume (cu. m): 20,000,000
Incident Information:
Date: 10-01-1928
Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m): 2,800,000
Tailings Travel Distance (m):
Incident Description:
The dam was constructed by cycloning sand tailings to form the outer shell. Embankment slopes were as steep as 1:1, and at the time of failure the last perimeter dike on the embankment crest had been constructed to a height of 55 feet. The dam failed by liquefaction during the M8.3 Talca earthquake of October 1, 1928. A tailings flowslide developed through a breach section approximately 1500 feet wide and flowed down a valley, killing 54 people.
Source: Dobry and Alvarez, 1967; Brawner, 1979; Jigins, 1957

Incident No.: 24
Dam/Mine Name: Casapalca
Mine Location: Peru
Ore/Tailings Type:
Dam Height (m): 107
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A number of tailings dams up to 350 feet in height were developed over the 50-year mine life in steep, narrow valleys. One of these dams failed by seismic liquefaction; no further details are reported
Source: Brawner, 1979

DAM TYPE: US INCIDENT TYPE: 1A
INCIDENT CAUSE: ER

Incident No.: 218
Dam/Mine Name: Ash-Cinder Tailings Dam of Thermal Power Plant “Maritsa Istok 1”, 3rd section
Mine Location: Near Stara Zagora, Central Bulgaria
Ore/Tailings Type: Ash/Cinder
Dam Height (m): 15
Dam Type: Dam Fill Material: Crushed Ash and Cinder
Impoundment Volume (cu. m): 52,000,000
Incident Information:
Date: 03-01-1992
Incident Type: 1A Cause: ER
Quantity of Tailings Released (cu. m): 500,000
Tailings Travel Distance (m):
Incident Description:
Inundation of the beach of the uppermost section of the dam caused erosion failure, with the slurry discharge causing failure of the lower dam sections by piping and overtopping. For more details see Section 6.
Source: Abadjiev and Dimitrov, 1997.

DAM TYPE: US INCIDENT TYPE: 1A
INCIDENT CAUSE: U

Incident No.: 81
Dam/Mine Name: Mir
Mine Location: BulgariaOre/Tailings Type: lead/zinc
Dam Height (m): Dam Type: US
Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1966 Incident Type: 1A Cause: U
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The tailings dam failed with loss of life. No other details are available.
Source: Abadjiev, 1990

DAM TYPE: US INCIDENT TYPE: 1B
INCIDENT CAUSE: OT

Incident No.: 202

Dam/Mine Name: Merriespruit
Mine Location: Merriespruit, nr Virginia, South Africa
Ore/Tailings Type: Gold
Dam Height (m): 31
Dam Type: US paddock Dam Fill Material: T
Impoundment Volume (Mt): 10
Incident Information:
Date: 02-22-1994;
Incident Type: lB Cause: OT
Quantity of Tailings Released (Mt): *2.5*
Tailings Travel Distance (m): 2,000
Incident Description:
Impoundment had been closed following signs of instability of part of ring dam closest to township. Mine continued to use it for storing waste water, containing tailings. This reduced freeboard and isolated decant. Heavy rain caused overtopping during evening. Personnel from mine tried to release water and warn population, some m bed. High phreatic surface caused failure of dam adjacent to houses. 17 killed.
Source: Official inquiry report.

Incident No.: 79
Dam/Mine Name: Mike Horse
Mine Location: MT, USA
Ore/Tailings Type: lead/zinc
Dam Height (m): 18
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m): 750,000
Incident Information:
Date: 1975 Incident Type: 1B Cause: OT
Quantity of Tailings Released (cu. m): 150,000
Tailings Travel Distance (m):
Incident Description:
During extreme runoff from a rain-on-snow event, the slopes of a sidehill diversion ditch became saturated and failed, directing the diverted streamflow into the abandoned impoundment. The decant capacity was insufficient to discharge the inflow, and the embankment was breached by overtopping
Source: Toland, 1977

DAM TYPE: US INCIDENT TYPE: 2A
INCIDENT CAUSE: SI

Incident No.: 77
Dam/Mine Name: Marianna Mine # 58
Mine Location: Washington County, PA, USA
Ore/Tailings Type: coal
Dam Height (m): 37
Dam Type: US Dam Fill Material: E
Impoundment Volume (cu. m): 300,000
Incident Information:
Date: 11-19-1986 Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A slide occurred in the upstream slope as a raise was being constructed of clayey fill over the fine coal refuse (tailings)beach. The cause was undrained shear failure due to rapid loading. The raise was about 14 ft. above the tailings elevation when the slide occurred. Sliding took place over a period of 1-2 minutes. The resulting scarp was about 14 ft. high and 550 ft. long, with as much as 20 ft. of lateral movement. The raise was later successfully constructed to a height of 25 ft. with careful monitoring of piezometers and controlled placement rates.
Source: Pennsylvania Dept. of Environmental Resources, Div. of Dam Safety

Incident No.: 210
Dam/Mine Name: Mirolubovka
Mine Location: Krivoj Rog iron ore basin, Southern Ukraine
Ore/Tailings Type: iron
Dam Height (m): 32
Dam Type: US Dam Fill Material: E, T and R (crushed quartzite)
Impoundment Volume (cu. m): 80,000,000
Incident Information:
Date: 01-15-1984 Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m): none
Tailings Travel Distance (m):
Incident Description:
Starter dam of loam 22m high. Two sand dykes above to raise to 32m. Phreatic surface allowed to rise and ditch cut in crest of starter dam to collect seepage. Starter dam became saturated. Rotational slip developed, said to be caused by incompatibility between real and design values for shear characteristics of foundation soil. Stabilized by toe weighting with rockfill.
Source: ICOLD Tailings Committee.

Incident No.: 172
Dam/Mine Name: Union Carbide
Mine Location: Uravan, CO, USA
Ore/Tailings Type: uranium
Dam Height (m): 43
Dam Type: US Dam Fill Material:
TImpoundment Volume (cu. m):
Incident Information:
Date: 03-01-1979 Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Two slides occurred on the 1.5:1 embankment slope due to snowmelt and internal seepage. Both were shallow, measuring 30-80 ft top width, 150-200 ft base width, and 80-100 ft length. Interim stabilization measures included horizontal

drains and a geofabric-protected drainage blanket. Long-term stabilization that followed consisted of a rockfill berm and underlying drainage zone which flattened the slopes to 2.0:1 to 3.0:1.
Source: Berry and Valarde, 1981; Robinson and Toland,1979; Colo. Div. Water Res.

Incident No.: 47
Dam/Mine Name: GCOS
Mine Location: Alberta, Canada
Ore/Tailings Type: oil sands
Dam Height (m): 61
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1974 Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Several episodes of instability occurred within compacted fill that was being placed over spigotted beach sand tailings during construction of upstream raises. All showed evidence of local liquefaction of the spigotted beach tailings, and took the form of subsidence of the compacted fill accompanied by shearing scarps. These failures were attributed to excess pore pressures that developed in the loose beach sand tailings in response to rapidly-applied loading during fill placement. No lateral translation occurred during failure and overall embankment stability was not jeopardized.
Source: Mittal and Hardy, 1977

Incident No.: 101
Dam/Mine Name: Ray Mine
Mine Location: Hayden, AZ, USA
Ore/Tailings Type: copper
Dam Height (m): 52
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 02-05-1973 Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Instability occurred along a small section of the embankment near the location where embankment failure had previously occurred on Dec. 2, 1972. No tailings were released. The previous failure was related to perched seepage conditions along a slimes layer, and the subsequent accident may have resulted from similar conditions.
Source: Anecdotal

Incident No.: 72
Dam/Mine Name: Lower Indian Creek
Mine Location: Washington County, MO, USA
Ore/Tailings Type: lead
Dam Height (m):
Dam Type: US Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 1960 Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The original earthfill dam was constructed in 1953 to an initial height of 45 feet and raised several times with additional earthfill. In 1959, the spillway washed out in a flood, causing some release of tailings but no breach or damage to the dam embankment. In 1960, the dam was reported to have shown signs of slumping on its 2:1 downstream face and was buttressed with a rockfill toe berm at a 3:1 slope. The dam remained in service was raised from 1971 to 1976 with cycloned sand tailings, and reached an ultimate height of 83 feet.
Source: Missouri Dept. of Nat. Res., Dam and Reservoir Safety Program

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Incident No.: 140
Dam/Mine Name: Unidentified
Mine Location: USA
Ore/Tailings Type: copper
Dam Height (m): 60
Dam Type: US Dam Fill Material: CST
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Slope instability caused a 15m wide section of the embankment slope to drop 8m. This condition was produced when saturated material was bulldozed in an oversteepened condition on the upper portion of the slope. Shortly after, seepage emerged on the unstable section and began to cause retrogressive failure, but dam beach was averted by prompt remedial action.
Source: Wahler and Schlick, 1976

Incident No.: 131
Dam/Mine Name: Unidentified
Mine Location:
Ore/Tailings Type:
Dam Height (m):
Dam Type: US Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):

Incident Description:
Freezing and growth of ice lenses on the embankment face produced extensive sloughing of the embankment slope accompanied by development of piping during the first few days of spring thaw.
Source: Casagrande and McIver, 1971

Incident No.: 113
Dam/Mine Name: Southwest US
Mine Location: USA
Ore/Tailings Type:
Dam Height (m):
Dam Type: US   Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
An embankment slope failure that breached the crest of the dam occurred after an unusually heavy rainfall. Ponded water was well back from the embankment crest at the time of failure, and no slimes or water were released.
Source: Klohn, 1972

DAM TYPE: US INCIDENT TYPE: 2A
INCIDENT CAUSE: SE

Incident No.: 138
Dam/Mine Name: Unidentified
Mine Location: South Africa
Ore/Tailings Type: gold
Dam Height (m):
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A Cause: SE
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Severe seepage and piping eroded a considerable portion of the embankment slope, leaving near-vertical scarps.
Source: Donaldson. Et al. 1976

Incident No.: 78
Dam/Mine Name: Miami Copper
Mine Location: AZ, USA
Ore/Tailings Type: copper
Dam Height (m):
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A Cause: SE
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Extensive damage to the embankment occurred due to seepage-related slumping and ravelling of the face, accompanied by piping and erosional transport of embankment tailings materials.
Source: Hazen, 1924

Incident No.: 147
Dam/Mine Name: Unidentified
Mine Location: AZ, USA
Ore/Tailings Type: copper
Dam Height (m): 18
Dam Type: US   Dam Fill Material: CST
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A   Cause: SE
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The initial starter dike was constructed to a height of 45 feet of relatively impervious sand and gravel on an impermeable foundation. Upstream raising used cycloned sands spigotted from the dam crest, with slimes discharged in the rear of the impoundment. When the embankment reached a height of 60 feet, malfunction of the stationary cyclone system caused uncycloned tailings to be discharged from the rear of the impoundment, with accumulation of ponded water and slimes near the embankment face. This caused seepage to emerge on the embankment face above the starter dike crest and raised the phreatic surface within the embankment to critical levels. Remedial measures included installation of french drains on the embankment face to collect surface seepage, and instituting perimeter discharge of whole tailings from the embankment crest to eliminate accumulation of ponded water in this area.
Source: Robinson and Toland, 1979

Incident No.: 130
Dam/Mine Name: Unidentified
Mine Location:
Ore/Tailings Type:
Dam Height (m):
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A Cause: SE
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A large piping cavity developed in the embankment due to seepage breakout on the embankment face. Piping progressed through the

entire width of the perimeter dike in one day, and severe damage was narrowly averted.
Source: Casagrande and McIver, 1971

DAM TYPE: US INCIDENT TYPE: 2A
INCIDENT CAUSE: FN

Incident No.: 153
Dam/Mine Name: Unidentified
Mine Location: MS, USA
Ore/Tailings Type: gypsum
Dam Height (m): 20
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1974 Incident Type: 2A Cause: FN
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The embankment was constructed and raised with overall slopes of about 3.5:1. When the embankment reached a height of 65 feet, slope instability occurred due to undrained shearing in soft foundation clays that had reached normally-consolidated conditions under the applied embankment loading. Further raising was discontinued, and the impoundment was subsequently abandoned.
Source: Anecdotal

Incident No.: 137
Dam/Mine Name: Unidentified
Mine Location: South Africa
Ore/Tailings Type: gold
Dam Height (m):
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A Cause: FN
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Wedge-type sliding on a thin layer of very soft foundation soil resulted in instability of the embankment, but the crest was not breached and no tailings were released.
Source: Donaldson, et al, 1976

DAM TYPE: US INCIDENT TYPE: 2A
INCIDENT CAUSE: ST

Incident No.: 128
Dam/Mine Name: Unidentified
Mine Location:
Ore/Tailings Type:
Dam Height (m):
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A Cause: ST
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A reinforced concrete decant conduit extended beneath the tailings dam and impoundment. In response to deterioration of the conduit, timber supports were added. Nevertheless, the conduit collapsed under excessive external water pressures, forming a crater on the surface of the impounded slimes. Tailings and timber debris formed a plug inside the collapsed conduit that allowed water pressures inside the plugged section to increase, cracking the conduit and producing concentrated seepage within coarse tailings comprising the embankment. This seepage caused piping of tailings into the rockfill starter dike at the downstream embankment toe. Damage was repaired by removing the debris plug, repairing the conduit, and adding filter zones to the rockfill.
Source: Smith and Connell, 1979

DAM TYPE: US INCIDENT TYPE: 2A
INCIDENT CAUSE: EQ

Incident No.: 111
Dam/Mine Name: Soda Lake
Mine Location: Santa Cruz, CA, USA
Ore/Tailings Type: sand and gravel
Dam Height (m): 3
Dam Type: US Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 10-17-1989
Incident Type: 2A Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A small saddle dike impounding tailings from rock washing operations experienced strong shaking during the Loma Preita earthquake. The dike was located 29 miles from the epicenter and 1400 feet from the main trace of the San Andreas fault. At the time of the earthquake, the impoundment contained little or no ponded water. Extensive sand boils and liquefaction-related features were observed within impounded sediments. Damage to the dam consisted of a large wedge of embankment fill that slid in an upstream direction and extended through the embankment out to the downstream face near the toe. Post-earthquake investigations revealed that the dam incorporated an upstream raise that underwent sliding due to liquefaction of underlying tailings. Adjacent dams confining the

same impoundment that did not incorporate upstream raises experienced no damage.
Source: California Dept. Water Resources., Div. of Safety of Dams.

Incident No.: 36
Dam/Mine Name: Dashihe
Mine Location: China
Ore/Tailings Type:
Dam Height (m): 37
Dam Type: US Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 1976 Incident Type: 2A Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The upstream type embankment was constructed to a height of 37m on 1.6:1 slopes at the time of the1976 Tangshang earthquake. The area experienced a M7.8 main shock, a M7.1 shock 15 days later, and numerous aftershocks of magnitude greater than 5. The dam was located 40 km and 15 km from the first two shocks, respectively. Damage consisted of cracks on the downstream embankment face and tailings beach, accompanied by boils and fissures near the ponded water. The dam did not fail and remained in service.
Source: Morgenstern and Kupper, 1988

Incident No.: 104
Dam/Mine Name: Sauce No. 1
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 6 Dam Type: US
Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 03-28-1965
Incident Type: 2A Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The No. 1 dam had been constructed with 1.7:1 slopes and was in active operation at the time of the M7-7 1/4 La Ligua earthquake. Serious cracking occurred at one corner, but the embankment did not fail.
Source: Dobry and Alvarez, 1967

DAM TYPE: US INCIDENT TYPE: 2A
INCIDENT CAUSE: ER

Incident No.: 48
Dam/Mine Name: Galena Mine
Mine Location: Osburn, ID, USA
Ore/Tailings Type: silver
Dam Height (m): 14 Dam Type: US
Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 1972 Incident Type: 2A Cause: ER
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description: Flooding on the stream adjacent to the toe of four sidehill type impoundments caused erosion damage to the embankments. Damage was repaired and the embankments riprapped.
Source: MHSA

DAM TYPE: US INCIDENT TYPE: 2B
INCIDENT CAUSE: EQ

Incident No. 105
Dam/Mine Name: Sauce No. 2
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 5
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 03-28-1965
Incident Type: 2B Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The No. 2 dam was inactive at the time of the M7-7 1/4 La Liqua earthquake, and suffered minor cracking.
Source: Dobry and Alvarez, 1967

Incident No.: 106
Dam/Mine Name: Sauce No. 3
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 5
Dam Type: US Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 03-28-1965 Incident Type: 2B
Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The No. 3 dam was inactive at the time of the 7-7 1/4 La Ligua earthquake, and suffered minor cracking.
Source: Dobry and Alvarez, 1967

Incident No.: 107
Dam/Mine Name: Sauce No. 4
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 5

Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 03-28-1965
Incident Type: 2B Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The No. 4 dam was inactive at the time of the M7-7 1/4 La Ligua earthquake, and suffered minor cracking.
Source: Dobry and Alvarez, 1967

Incident No.: 70
Dam/Mine Name: Los Maquis No. 1
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 15
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 03-28-1965
Incident Type: 2B Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam experienced strong shaking during the M7-7 1/4 La Ligua earthquake and was adjacent to the active Los Maguis No. 3 dam which failed. The No.1 dam had been out of service for many years, and experienced only slight cracking along the crest and small slides in dry tailings on the sideslopes.
Source: Dobry and Alvarez, 1967

Incident No.: 42
Dam/Mine Name: El Cerrado
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 25
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 03-28-1965 Incident Type: 2B
Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The impoundment, incorporating 3 levels, had been abandoned for 10 years at the time of the M7-7 ¼ La Ligua earthquake. The embankments were constructed with 1.4:1 slopes. The earthquake produced cracks up to 6 feet deep along the entire crest accompanied by several circular slides, especially at corners of the embankment. Crest deformation up to 1 foot also occurred.
Source: Dobry and Alvarez, 1967

Incident No.: 27
Dam/Mine Name: Cerro Negro No. 1
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 46
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 03-28-1965 Incident Type: 2B
Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The Cerro Negro No. 1 dam was inactive at the time of the M7-7 1/4 La Ligua earthquake and adjacent to the No. 3 dam which failed. Its slopes were as steep as 1:1. The No. 1 dam experienced cracking, especially along the crest, and some small slides.
Source: Dobry and Alvarez, 1967

Incident No.: 28
Dam/Mine Name: Cerro Negro No. 2
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 46
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 03-28-1965
Incident Type: 2B Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The Cerro Negro No. 2 dam was inactive at the time of the M7-7 1/4 La Ligua earthquake and adjacent to the No. 3 dam which failed. Its slopes were as steep as 1:1. The No. 2 dam, like the adjoining inactive No. 1 dam, experienced cracking along the crest and small slides.
Source: Dobry and Alvarez, 1967

Incident No.: 46
Dam/Mine Name: El Cobre Small Dam
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 26
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m): 985,000
Incident Information:
Date: 03-28-1965
Incident Type: 2B Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:

The El Cobre Small Dam was adjacent to the New Dam and Old Dam, both of which failed during the M7-7 1/4 La Ligua earthquake. The Small Dam was similar in construction to the Old Dam with steep (1.2:1.0) slopes, but was abandoned at the time of the earthquake, with a desiccated surface crust about 5 m deep. Damage in the form of local slides is reported, but the dam remained essentially intact.
Source: Dobry and Alvarez, 1967

DAM TYPE: US INCIDENT TYPE: 3
INCIDENT CAUSE: NR

Incident No.: 92
Dam/Mine Name: PCS Rocanville
Mine Location: Saskatchewan, Canada
Ore/Tailings Type: potash
Dam Height (m):12
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1975 Incident Type: 3 Cause:
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The impoundment is underlain by a surficial aquifer and a deeper aquifer separated by a till layer. During initial impoundment construction, puncturing of its polyethylene liner was reported due to placement of cover material. As a result, a second liner and cover were placed on top of the first. During operation, leakage of brine into the shallow aquifer was detected. A collector ditch was installed, and improvement in downgradient water quality in the shallow aquifer was reported. Contamination of the lower aquifer was detected several years later. Two downgradient pumpback wells were installed, and water quality improvements occurred.
Source: Tallin and Pufahl, 1983

Incident No.: 60
Dam/Mine Name: IMC K-2
Mine Location: Saskatchewan, Canada
Ore/Tailings Type: potash
Dam Height (m): 30
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1968 Incident Type: 3 Cause:
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The starter dike for the impoundment was constructed of compacted till on a foundation of oxidized till which was jointed and contained sand seams. A cutoff trench of compacted clay underlies the dike to a depth of 4 ft, and a shallow collector ditch was constructed at the toe. The collector ditch proved to be too shallow to completely control seepage. Extension of the ditch down through the oxidized till was planned as a remedial measure.
Source: Kent, et al., 1983

DAM TYPE: CL INCIDENT TYPE: 1A
INCIDENT CAUSE: FN

Incident No.: 40
Dam/Mine Name: Dresser No. 4
Mine Location: Washington County, MO, USA
Ore/Tailings Type: barite
Dam Height (m): 15
Dam Type: CL Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 08-15-1975
Incident Type: 1A Cause: FN
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The apparent cause of failure was embankment sliding along residual and alluvial foundation soils. The tailings flowslide reached a nearby drainage and from there entered a creek.
Source: Missouri Dept. of Nat. Res., Dam and Reservoir Safety Program

DAM TYPE: CL INCIDENT TYPE: 1A
INCIDENT CAUSE: OT

Incident No.: 204
Dam/Mine Name: No 1 tailings dam
Mine Location: Middle Arm, Launceston, Tasmania
Ore/Tailings Type:
Dam Height (m): 4 Dam Type: CL Dam Fill Material: E
Impoundment Volume (cu. m): 25,000
Incident Information:
Date: 06-25-1995
Incident Type: lA Cause: OT
Quantity of Water Released (cu. m): 5,000
Tailings Travel Distance (m):
Incident Description:
Crest formed of tailings, eroded by wave action. Water containing 95mg/litre released into Tamar river. Cause: retained tailings allowed to rise above crest. Cost of remediation estimated A$ 20,000 - 30,000
Source: Inspector of Mines, Tasmania

DAM TYPE: CL INCIDENT TYPE: 1B

INCIDENT CAUSE: OT

Incident No.: 188
Dam/Mine Name: Mineral King
Mine Location: Invermere, British Columbia
Ore/Tailings Type:
Dam Height (m): 6 Dam Type: CL Dam Fill Material: CST
Impoundment Volume (cu. m): small
Incident Information:
Date: 03-20-1986
Incident Type: 1B Cause: OT
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Dam breach caused by high pond overtopping crest. Diversion ditch blocked by ice during onset of spring snowmelt
Source: Energy and Minerals Division, Ministry of Employment and Investment, Victoria V8V 1X4, Canada

DAM TYPE: CL INCIDENT TYPE: 2A
INCIDENT CAUSE: SI

Incident No.: 18
Dam/Mine Name: Cadet No. 2
Mine Location: Washington County, MO, USA
Ore/Tailings Type: barite
Dam Height (m): 21
Dam Type: CL Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 09-01-1975
Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
During initial raising of the starter dike, sand and gravel mill reject with excessive fines content was used as fill in the downstream portion of the raise. This did not provide sufficient drainage, and a slide resulted due to the high phreatic surface. A 10-foot wide berm of gravel and rock fill was placed to a height of about 40 feet to stabilize the area.
Source: Missouri Dept. of Nat. Res., Dam and Reservoir Safety Program

Incident No.: 19
Dam/Mine Name: Captains Flat Dam 2
Mine Location: Australia
Ore/Tailings Type: copper
Dam Height (m): 22
Dam Type: CL Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam was constructed of uncompacted clayey sand and gravel with downstream slopes of 1:1. Considerable seepage at the embankment toe occurred, with damage consisting of multiple cracks and scarps parallel to the crest having a cumulative vertical displacement up to one meter.
Source: Ash, 1976

DAM TYPE: CL INCIDENT TYPE: 2A
INCIDENT CAUSE: SE

Incident No.: 203
Dam/Mine Name: Riltec
Mine Location: Mathinna, Tasmania
Ore/Tailings Type:
Dam Height (m): 7 Dam Type: CL Dam Fill Material: E
Impoundment Volume (Cu. m): 120,000
Incident Information:
Date: 1-6-1995
Incident Type: 2A Cause: SE
Quantity of Water Released (cu. m): 40,000
Tailings Travel Distance (m):
Incident Description:
Leakage of cyanide-contaminated water from base of impoundment into ground water. Dam 1:2 downstream slope, 4m wide crest. Built 3 months before in compacted layers, clay lined. Polluted streams; fish kill. Cessation of operations; owner bankrupt.
Source: Inspector of Mines, Tasmania.

Incident No.: 34
Dam/Mine Name: Cyprus Thompson Creek
Mine Location: Custer County, ID, USA
Ore/Tailings Type: molybdenum
Dam Height (m): 146
Dam Type: CL Dam Fill Material: CST
Impoundment Volume (cu. m): 27,000,000
Incident Information:
Date: 1989 Incident Type: 2A Cause: SE
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
An auxiliary drain at the embankment toe originally installed to drain a spring (900 gpm flow) was noted to be discharging fines. Further inspection revealed a sinkhole 8 ft in dia. and 4ft deep on the downstream slope of the embankment. The original drain included a 6-inch dia. PVC pipe wrapped in filter cloth, and it is thought that some form of failure of the filter cloth may have allowed piping into the drain and sinkhole formation to occur.
Source: Idaho Dept. Water Res., Dam Safety Section

DAM TYPE: CL INCIDENT TYPE: 2A
INCIDENT CAUSE: FN

Incident No.: 14
Dam/Mine Name: Big Four
Mine Location: Polk County, FL, USA
Ore/Tailings Type: phosphate
Dam Height (m):
Dam Type: CL Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 08-01-1989
Incident Type: 2A Cause: FN
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The accident was related to sinkhole-induced subsidence in the karstic limestone foundation of the dam, which retained phosphatic clay slimes. No further details are available.
Source: Anecdotal

Incident No.: 120
Dam/Mine Name: Syncrude
Mine Location: Alberta, Canada
Ore/Tailings Type: oil sands
Dam Height (m):
Dam Type: CL Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1978 Incident Type: 2A Cause: FN
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The embankment is founded on pre-sheared clay shales of low residual strength. Measured foundation movements indicated the potential for foundation instability, and portions of the embankment were re-designed with slopes as flat as 9:1.
Source: Morgenstern, et al, 1988

Incident No.: 162
Dam/Mine Name: Unidentified
Mine Location: Hernando County, FL, USA
Ore/Tailings Type: limestone
Dam Height (m): 6
Dam Type: CL Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 1977 Incident Type: 2A Cause: FN
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description: The impoundment was used to retain tailings from limestone washing operations of similar nature to phosphatic clay slimes. When the embankment reached a height of about 20 feet, concentrated seepage and piping in karstic foundation limestone occurred at the embankment toe. A small ring dike was constructed around the area, and water within it was allowed to rise until pressure head balanced seepage exit pressures. No further piping occurred.
Source: Anecdotal

DAM TYPE: CL INCIDENT TYPE: 2A
INCIDENT CAUSE: ST

Incident No.: 13
Dam/Mine Name: Big Four
Mine Location: Polk County, FL, USA
Ore/Tailings Type: phosphate
Dam Height (m): 18
Dam Type: CL Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 1986 Incident Type: 2A Cause: ST
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A metal pipe outlet conduit penetrated the dam, which impounded phosphatic clay slimes. Corrosion of the pipe caused internal erosion of embankment fill soils into it. Remedial measures included repair of the pipe, backfilling of embankment soils, and regrading.
Source: Anecdotal

Incident No.: 32
Dam/Mine Name: Clayton Mine
Mine Location: Custer County, ID, USA
Ore/Tailings Type: silver
Dam Height (m): 24
Dam Type: CL; Dam Fill Material: T
Impoundment Volume (cu. m): 215,000
Incident Information:
Date: 02-06-1983
Incident Type: 2A Cause: ST
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A tailings pipeline on the dam crest broke during the night, eroding a gully 2-3 ft wide and 5-6 ft deep on the downstream face of the embankment. No impounded tailings were released and the dam was repaired and placed back in service.
Source: Idaho Dept. Water Res., Dam Safety Section.

DAM TYPE: CL INCIDENT TYPE: 2B
INCIDENT CAUSE: ST

Incident No.: 16
Dam/Mine Name: Blackbird

Mine Location: Cobalt, ID, USA
Ore/Tailings Type: cobalt
Dam Height (m): 15
Dam Type: CL Dam Fill Material: MW
Impoundment Volume (cu. m): 1,230,000
Incident Information:
Date: Incident Type: 2B Cause: ST
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The impoundment was constructed in the 1940's and 1950's with a metal culvert to pass perennial streamflows beneath the dam and impoundment. The culvert corroded under the influence of acidic tailings effluent, and at least partially collapsed. No embankment breach resulted, but suspended tailings were discharged to the downstream drainage during periods of high flow through the damaged culvert..
Source: Montana Dept. State Lands

DAM TYPE: CL INCIDENT TYPE: 3
INCIDENT CAUSE: NR

Incident No.: 51
Dam/Mine Name: Golden Sunlight
Mine Location: Whitehall, MT, USA
Ore/Tailings Type: gold
Dam Height (m):
Dam Type: CL Dam Fill Material: CST
Impoundment Volume (cu. m):
Incident Information:
Date: 05-01-1983 Incident Type: 3 Cause:
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The seepage control system constructed for the tailings dam included a primary bentonite-slurry cutoff wall, together with drains beneath the impounded tailings and foundation preparation of clayey soils in the impoundment area. The cutoff wall extended as deep as 60 feet to an impermeable stratum. Tailings discharge began in Feb. 1983, and contamination was detected in downgradient monitor wells in May, 1983. It is estimated that 160,000 gal. of cyanide-bearing effluent leaked past the slurry cutoff between April 1983 and June, 1984, with average concentrations of 1.5 mg/l total and 0.3 mg/l free cyanide. The reason for the leakage is presumed to be an undetected landslide-related discontinuity in the impermeable stratum that was not penetrated by the cutoff. Remedial measures included repair of the cutoff and installation of pumpback wells that returned 400 gpm to the impoundment. It is believed that these measures were effective in containing further seepage. Continued migration of the original contaminant plume was not expected to result in detectable levels of contamination in adjacent surface waters.
Source: Montana Dept. State Lands

DAM TYPE: DS INCIDENT TYPE: 1A
INCIDENT CAUSE: SI

Incident No.: 144
Dam/Mine Name: Unidentified
Mine Location: United Kingdom
Ore/Tailings Type: coal
Dam Height (m): 20
Dam Type: DS Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 1967 Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The failure occurred during regrading operations to stabilize bulging and deformation of the downstream dam slope that had occurred two months previously. Contributing to the failure may have been rise in impoundment fluid levels due to displacement by mine waste being regraded from an adjacent pile. The tailings flow failure covered an area of 4 ha.
Source: Thompson and Rodin, 1972

DAM TYPE: DS INCIDENT TYPE: 1A
INCIDENT CAUSE: SE

Incident No.: 168
Dam/Mine Name: Unidentified
Mine Location: Peace River, FL, USA
Ore/Tailings Type: phosphate
Dam Height (m):
Dam Type: DS Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 02-01-1951
Incident Type: 1A Cause: SE
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam had been raised in height several months prior to the failure using sand fill. At the time of failure, water at least 5 feet deep was in direct contact with the upstream face of the dam, including the interface between the new and old fill. The failure is thought to be related to either the incorporation of logs and brush in the original portion of the structure, or an old decant pipe found at the bottom of the breach. In either case, seepage and piping were the eventual cause of failure. The phosphate clay slimes released produced suspended solids concentrations as high as 8000 ppm in the Peace River.

Source: Anecdotal

DAM TYPE: DS INCIDENT TYPE: 1A
INCIDENT CAUSE: OT

Incident No.: 109
Dam/Mine Name: Silver King
Mine Location: Adams County, ID, USA
Ore/Tailings Type: copper
Dam Height (m): 9
Dam Type: DS Dam Fill Material: E
Impoundment Volume (cu. m): 37,000
Incident Information:
Date: 01-16-1974
Incident Type: 1A Cause: OT
Quantity of Tailings Released (cu. m): 6,000
Tailings Travel Distance (m):
Incident Description:
Rain on heavy snowpack caused the impoundment to fill to capacity, and emergency pumping was insufficient to prevent overtopping with the loss of 2 million gallons of water and about 20% of the impounded tailings. Downstream damage consisted of silting of streambeds. The embankment was subsequently repaired and placed back into service.
Source: Idaho Dept. Water Res., Dam Safety Section

DAM TYPE: DS INCIDENT TYPE: 1A
INCIDENT CAUSE: ST

Incident No.: 97
Dam/Mine Name: Portworthy
Mine Location: United Kingdom
Ore/Tailings Type: china clay
Dam Height (m): 15
Dam Type: DS Dam Fill Material: R
Impoundment Volume (cu. m):
Incident Information:
Date: 1970 Incident Type: 1A Cause: ST
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Dam breach occurred due to structural failure of a decant conduit.
Source: Ripley, 1972

DAM TYPE: DS INCIDENT TYPE: 1A
INCIDENT CAUSE: EQ

Incident No.: 43
Dam/Mine Name: El Cobre New Dam
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 19
Dam Type: DS Dam Fill Material: CST
Impoundment Volume (cu. m): 350,000
Incident Information:
Date: 03-28-1965
Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m): 350,000
Tailings Travel Distance (m): 12,000
Incident Description:
The dam was constructed by cycloning and is inferred to have been raised according to the downstream method with a downstream slope of 3.7:1.0. The impoundment had undergone rapid filling immediately prior to the M7-7 1/4 La Ligua earthquake of March 28, 1965. Eyewitness accounts indicated that the impounded slimes completely liquefied, with waves generated on the surface. Inertial forces combined with increased pressure from the liquefied slimes opened a breach near the abutment, which was rapidly enlarged by the flowslide. The failure, combined with that of the adjacent Old Dam, destroyed the town of El Cobre and killed more than 200 people.
Source: Dobry and Alvarez, 1967

DAM TYPE: DS INCIDENT TYPE: 1B
INCIDENT CAUSE: FN

Incident No.: 38
Dam/Mine Name: Derbyshire
Mine Location: United Kingdom
Ore/Tailings Type: coal
Dam Height (m): 8
Dam Type: DS Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 1966 Incident Type: 1B Cause: FN
Quantity of Tailings Released (cu. m): 30,000
Tailings Travel Distance (m): 100
Incident Description:
The impoundment had been inactive for 8 years at the time of failure. Foundation materials consisted of 20 feet of clay overlying shale/mudstone bedrock. Failure by foundation sliding was attributed to artesian foundation pore pressures produced by seepage from adjacent active impoundments and natural recharge, with subsidence from underground workings as a possible contributing cause.
Source: Thompson and Rodin, 1972

DAM TYPE: DS INCIDENT TYPE: 2A
INCIDENT CAUSE: SI

Incident No.: 214
Dam/Mine Name: Minera Serra Grande Mine
Location: Crixas, Goias, Brazil
Ore/Tailings Type: gold

Dam Height (m): 41
Dam Type: DS over US Fill Material: CST
Impoundment Volume (Mt): 2.25
Incident Information:
Date: 02-1994 Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m): none
Tailings Travel Distance (m):
Incident Description:
Major rotational slip in downstream slope that did not lower the crest. Mine closed for 3 weeks during emergency repairs. Revenue loss equivalent to 8,500 ozs of gold. Starter dam across the valley built of compacted earthfill that was fairly impervious. Filter drains underneath badly constructed and ineffective. Grout curtain cut-off under this earth dam. Abnormal behaviour of the piezometers was no diagnosed. Heavy rainstorms late 1993 and early 1994 brought phreatic surface above starter dam to 'daylight' 20m above downstream toe. For more details, see Section 6.
Source: ICOLD Tailings Committee

Incident No.: 11
Dam/Mine Name: Battle Mt. Gold
Mine Location: Battle Mt., NV, USA
Ore/Tailings Type: gold
Dam Height (m): 8
Dam Type: DS Dam Fill Material: E
Impoundment Volume (cu. m): 1,540,000
Incident Information:
Date: 1984 Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Instability of the downstream slope was caused by poor compaction of fill. The slope was reconstructed and flattened
Source: Nevada Dept. of Conservation and Nat. Res., Div. Water Res.

Incident No.: 149
Dam/Mine Name: Unidentified
Mine Location: ID, USA
Ore/Tailings Type: phosphate
Dam Height (m): 34
Dam Type: DS Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 1976 Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam was constructed with downstream slopes of 1.4:1 and a dam raise had been added the previous winter. During the spring thaw, severe sloughing on the downstream face of the dam occurred, accompanied by extensive downslope creep of heavily saturated fill containing blocks of frozen soil. These conditions were attributed to snow and ice having been incorporated during winter construction of the previous raise.
Source: Anecdotal

Incident No.: 145
Dam/Mine Name: Unidentified
Mine Location: United Kingdom
Ore/Tailings Type: coal
Dam Height (m): 14
Dam Type: DS Dam Fill Material: MW
Impoundment Volume (cu. m):
Incident Information:
Date: 1967 Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A slide occurred in the downstream slope after aperiod of heavy rain and one week following widening of the dam crest by dumping of uncompacted mine waste fill. Both the original and newly-constructed slopes were at the angle of repose, and a high phreatic surface existed within the embankment.
Source: Thompson and Rodin, 1972

Incident No.: 150
Dam/Mine Name: Unidentified
Mine Location: ID, USA
Ore/Tailings Type: phosphate
Dam Height (m): 18
Dam Type: DS Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 1965 Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam was initially constructed and raised using clay and gravel soils with downstream slopes of 1.5:1.0. Slope instability occurred due to lack of internal drainage and the steep embankment slopes. An internal drainage zone was incorporated for subsequent raises of the dam.
Source: Anecdotal

DAM TYPE: DS INCIDENT TYPE: 2A INCIDENT CAUSE: SE

Incident No.: 86
Dam/Mine Name: Monsanto Dike 15
Mine Location: Colombia, TN, USA
Ore/Tailings Type: phosphate
Dam Height (m): 43
Dam Type: DS Dam Fill Material: E

Impoundment Volume (cu. m): 1,230,000
Incident Information:
Date: 1969 Incident Type: 2A Cause: SE
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The tailings dam was constructed as a conventional water-retention type structure with an internal core, clayey gravel shells, and a blanket drain. Operation of the dam was such that ponded water accumulated directly against the upstream face of the dam. Excessive seepage through the dam occurred during the first few years of operation. The reservoir was lowered, an asphalt emulsion was placed on the upstream face, and seepage was substantially reduced. During operation of subsequently constructed downstream dam raises, tailings were spigotted from the embankment crest as a primary seepage-control measure.
Source: Smith, et. al., 1977

Incident No.: 146
Dam/Mine Name: Unidentified
Mine Location: United Kingdom
Ore/Tailings Type: sandstone
Dam Height (m): 30
Dam Type: DS Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 1967 Incident Type: 2A Cause: SE
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Shortly after filling of the first stage of the dam, small slips on the downstream slope, high piezometer pressures, and a break in the decant pipe passing through the dam occurred. These conditions were repaired by placing a filter and buttress on the downstream slope. Following subsequent downstream raising of the dam, seepage occurred at the interface between the new and original fill on the downstream dam slope during impoundment of runoff. This was repaired by placing a synthetic membrane on the exposed upstream face of the dam.
Source: Little and Beavan, 1976

Incident No.: 52
Dam/Mine Name: Granisle
Mine Location: British Columbia, Canada
Ore/Tailings Type: copper
Dam Height (m): 24
Dam Type: DS Dam Fill Material: MW
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A Cause: SE
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The initial stage of the tailings dam was constructed across a bay of a large lake by dumping mine waste, and tailings were discharged into the bay from the dam crest. A sudden piping failure occurred when the tailings beach reached a level one foot above the lake tailwater elevation, carrying a significant quantity of tailings and effluent through the mine waste and into the lake. The condition was repaired by placing a wide zone of cycloned sand over the spigotted tailings beach, thereby pushing ponded water back from the dam. The combined effects of drainage by cycloned sands and reduction of internal seepage gradients prevented further piping, and subsequent dam raises incorporated an upstream filter zone against the placed mine waste.
Source: Klohn, 1979; Klohn, 1980

Incident No.: 157
Dam/Mine Name: Unidentified
Mine Location: British Columbia, Canada
Ore/Tailings Type:
Dam Height (m):
Dam Type: DS Dam Fill Material: CST
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A Cause: SE
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Piping developed at the abutment of a cycloned sand tailings dam when ponded water rose in response to spring runoff and came into direct contact with the sand tailings embankment fill. This condition had been predicted, and an upstream impervious zone had been added to prevent its occurrence. However, careless spigotting of tailings had eroded this zone at the abutment contact. Repairs consisted of dumping impervious fill on the upstream dam face and filling the downstream eroded piping exit area with sand and gravel filter material.
Source: Klohn, 1979

Incident No.: 158
Dam/Mine Name: Unidentified
Mine Location: British Columbia, Canada
Ore/Tailings Type:
Dam Height (m): 30
Dam Type: DS Dam Fill Material: MW
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A Cause: SE
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):

Incident Description:
Piping developed in the sand beach of a tailings dam constructed of mine waste, resulting in the discharge of 10,000,000 gallons of ponded water at a peak rate flow of 48,000 gpm. Considerable property damage occurred. Subsequent investigations revealed that an intended upstream filter zone was either absent or improperly placed by end-dumping, and that water had been allowed to pond too close to the dam face with insufficient beach development. Seepage fluctuations and development of sinkholes on the beach prior to the incident were not properly interpreted as indicating the occurrence of piping because no tailings were observed to be present in downstream seepage discharge. Repairs consisted of spigotting tailings from a new dike constructed on the tailings beach far removed from the main dam. The resulting reduction of internal seepage gradients was sufficient to allow the dam to be completed to full height of about 200 feet.
Source: Klohn, 1979.

DAM TYPE: DS INCIDENT TYPE: 2A
INCIDENT CAUSE: FN

Incident No.: 164
Dam/Mine Name: Unidentified
Mine Location: Hernando County, FL, USA
Ore/Tailings Type: limestone
Dam Height (m): 12
Dam Type: DS Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 1988 Incident Type: 2A Cause: FN
Quantity of Tailings Released (cu. m):Tailings Travel Distance (m):
Incident Description:
Tailings similar in nature to phosphatic clay slimes were impounded behind a dam constructed of clay fill. Downstream raises of the dam were constructed on a foundation that contained slimes from a previous tailings spill. Slope instability occurred during construction of the final raise, with shearing through the weak foundation slimes.
Source: Anecdotal

Incident No.: 68
Dam/Mine Name: La Belle
Mine Location: Fayette County, PA, USA
Ore/Tailings Type: coal
Dam Height (m): 79
Dam Type: DS Dam Fill Material: MW
Impoundment Volume (cu. m): 1,230,000
Incident Information:
Date: 07-17-1985
Incident Type: 2A Cause: FN
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Movements on the downstream slope were first noticed on July 17, 1985, and small movements occurred daily over the following two weeks. During or after a rain on July 31, 1985, further sliding occurred leaving a scarp up to 4 ft high and 800 ft long. Thirteen families were evacuated for a short period of time. Movements were caused by translation-type sliding along a residual foundation clay layer that dipped in a downstream direction. Repairs included rock drains on the downstream face and a buttress of rock and coarse refuse. There had been no further movement through 1989.
Source: Pennsylvania Dept. of Environ. Div. of Dams Safety; LaBelle Processing Company, Uniontown, PA 15401

DAM TYPE: DS INCIDENT TYPE: 2A
INCIDENT CAUSE: OT

Incident No.: 108
Dam/Mine Name: Silver King
Mine Location: Adams County, ID, USA
Ore/Tailings Type: copper
Dam Height (m): 9
Dam Type: DS Dam Fill Material: E
Impoundment Volume (cu. m): 37,000
Incident Information:
Date: 08-05-1989
Incident Type: 2A Cause: OT
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The tailings impoundment was being used for water retention when a mine waste dump founded on a portion of the tailings in the impoundment failed, displacing about 1-2 ac-ft of water and overtopping the dam. The tailings dam was not significantly damaged. Only some silting of downstream stream channels was reported.
Source: Idaho Dept. Water Res., Dam Safety Section; Alta Gold Co., Salt Lake City, UT 84109, USA.

DAM TYPE: DS INCIDENT TYPE: 2A
INCIDENT CAUSE: ST

Incident No.: 121
Dam/Mine Name: TN Consolidated Coal No.1
Mine Location: Marion County, TN, USA
Ore/Tailings Type: coal
Dam Height (m): 85
Dam Type: DS Dam Fill Material: MW
Impoundment Volume (cu. m): 1,000,000

Incident Information:
Date: 01-19-1988
Incident Type: 2A Cause: ST
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam contained an abandoned 3-ft diameter outlet conduit that had been plugged at both ends with concrete, with the conduit interior drained by an 8-inch bleed pipe. The upstream end of the conduit developed a leak which produced inflow greater than the capacity of the 8-inch bleed line to drain it, and water began seeping out on the downstream face of the dam near the toe. The accident drained all 6.5 million gallons of water impounded by the dam and produced severe erosion of the downstream face which exposed the buried conduit. The conduit was completely backfilled with concrete and the dam placed back in service.
Source: Division of Water Pollution Control, Tennessee Department of Health and Environment;

Incident No.: 82
Dam/Mine Name: Missouri Lead
Mine Location: MO, USA
Ore/Tailings Type: lead
Dam Height (m): 17
Dam Type: DS Dam Fill Material: CST
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A Cause: ST
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
In conjunction with raising of the dam using cycloned sand tailings, a foundation drainage system consisting of 6-in dia. Perforated corrugated metal pipe surrounded by filter gravel was installed. When the cycloned sand fill had reached a height of 55 feet above the pipe, a sinkhole 25 ft. in diameter and 20 ft. deep developed on the sandfill surface. The sinkhole was attributed to collapse of the drainage pipe, and investigations showed the pipe to be severely corroded by the slightly acidic pH of the seepage effluent. The pipe ends were plugged, and internal drainage was directed to pervious in-situ foundation soils to the downstream toe of the dam.
Source: Brawner, 1979

DAM TYPE: DS INCIDENT TYPE: 2A INCIDENT CAUSE: EQ

Incident No.: 44
Dam/Mine Name: El Cobre No. 4
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 50
Dam Type: DS Dam Fill Material: CST
Impoundment Volume (cu. m):
Incident Information:
Date: 03-03-1985
Incident Type: 2A Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam was constructed with upstream slopes of 1.9:1, downstream slopes of 4.6:1, and with a blanket drain. Cycloned sands received some compaction during spreading with a bulldozer. During the M7.8 earthquake of March 3, 1985, minor damage occurred in the form of a sloughing of sands in the upper part of the downstream slope and shallow slides in the upper 6 ft of the unsubmerged upstream slope.
Source: Castro and Troncoso, 1989

DAM TYPE: DS INCIDENT TYPE: 2B INCIDENT CAUSE: SI

Dam ID Number: 143
Dam/Mine Name: Unidentified
Mine Location: United Kingdom
Ore/Tailings Type: coal
Dam Height (m): Dam Type: DS Fill Material:
Impoundment Volume (cu.m):
Incident Information:
Date: Incident Type: 2B Cause: SI
Quantity of Tailings Released (cu.m):
Tailings Travel Distance (m):
Incident Description:
Sliding of the upstream slope of the dam occurred during excavation of the impounded tailings to a depth of 40 ft intended to allow re-use of the impoundment. The failure was attributed to rapid-drawdown conditions on the upstream slope following removal of the tailings.
Source: Thompson and Rodin, 1972

DAM TYPE: DS INCIDENT TYPE: 2B INCIDENT CAUSE: EQ

Incident No.: 90
Dam/Mine Name: Norosawa
Mine Location: Japan
Ore/Tailings Type: gold
Dam Height (m): 24
Dam Type: DS Dam Fill Material:
Impoundment Volume (cu. m): 225,000
Incident Information:
Date: 01-14-1978
Incident Type: 2B Cause: EQ
Quantity of Tailings Released (cu. m):

Tailings Travel Distance (m):
Incident Description:
The dam had been abandoned for 13 years at the time of the M7.0 Izu-Oshima-Kinkai earthquake. Boils appeared on the surface of the impounded slimes and the dam was cracked, but no failure occurred.
Source: Okusa, et al, 1980

Incident No.: 56
Dam/Mine Name: Hirayama
Mine Location: Japan
Ore/Tailings Type: gold
Dam Height (m): 9
Dam Type: DS Dam Fill Material:
Impoundment Volume (cu. m): 87,000
Incident Information:
Date: 1978 Incident Type: 2B Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam experienced ground accelerations estimated to be 0.2-0.35 g from the M7.0 Izu-Oshima-Kinkai earthquake. The impoundment had been inactive for about 20 years. The dam experienced cracking and impounded tailings exhibited sand boils, but no failure occurred.
Source: Okusa, et al, 1980

DAM TYPE: DS INCIDENT TYPE: 3
INCIDENT CAUSE: NR

Incident No.: 87
Dam/Mine Name: Montana Tunnels
Mine Location: Jefferson City, MT, USA
Ore/Tailings Type: gold
Dam Height (m): 33
Dam Type: DS Dam Fill Material: MW
Impoundment Volume (cu. m): 250,000
Incident Information:
Date: 1987 Incident Type: 3 Cause:
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The impoundment bottom was lined with a compacted soil-bentonite liner underlain by a sand filter blanket. The liner was eroded at several locations due to (1) concentrated runoff from high-intensity rainstorms prior to tailings deposition, and (2) initial spigotting of tailings and emergency release of reclaim water into the impoundment. Damage to the liner was repaired, but some of the damage may not have been detected, and the integrity of the liner in repaired areas may not have been completely restored. When tailings deposition resumed following liner repairs, routine groundwater monitoring detected elevated levels of process solution immediately downstream from the embankment. Impoundment seepage was estimated as 450 to 650 gpm. A recovery system of pumpback wells was installed downstream from the embankment. This was effective in intercepting the contaminated groundwater and has contained the contaminant plume within the mine site boundaries.
Source: Clark, et al, 1989; Montana Dept. State Lands

Incident No.: 53
Dam/Mine Name: Grey Eagle
Mine Location: Siskiyou County, CA, USA
Ore/Tailings Type: gold
Dam Height (m):
Dam Type: DS Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 1983 Incident Type: 3 Cause:
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam contained an upstream-sloping clay core with granular shells, and was constructed on a jointed rock foundation. Provision was made for installation of a dam seepage collection and pumpback system to return contaminated fluids to the impoundment. Unforeseen problems resulted in higher effluent cyanide concentrations than anticipated, and unprecedented precipitation produced high seepage return flows. Three months after filling began, dam through-seepage and infiltration into the downstream shell reached 400 gpm. Undiluted cyanide concentrations within the internal drainage system reached 20 ppm free and 100 ppm total. This seepage and unattenuated cyanide was not anticipated, and its return to the impoundment had adverse effects on the system water balance. To prevent contamination of surface and groundwater, surface diversion and drainage of downstream-shell infiltration; enlargement of the seepage pumpback system; construction of a treatment plant for dam seepage; and measures to reduce impoundment inflows were proposed.
Source: Hutchinson, et al, 1985; Centurion Gold Ltd., Vancouver

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Incident No.: 133
Dam/Mine Name: Unidentified
Mine Location:
Ore/Tailings Type: gold
Dam Height (m):

Dam Type: DS   Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 3 Cause:
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
An initial zoned earthfill dam was constructed of borrow excavated from within the impoundment. The impoundment was not lined, and complete cutoff of foundation seepage could not be guaranteed at the dam site selected. Seepage mitigation measures incorporated in the design included an extensive underdrain system within the impoundment, a trench drain along the downstream toe of the dam, and extensive piezometer instrumentation. Filling of the impoundment resulted in excessive seepage around the abutments and beneath the trench drain through interconnected zones of sands and gravels within the impoundment area. Remedial measures included construction of a deep trench drain system incorporating pumpback wells downstream from the raised dam toe.
Source: Anecdotal

DAM TYPE: WR   INCIDENT TYPE: 1A
INCIDENT CAUSE: SI

Incident No.: 165
Dam/Mine Name: Unidentified
Mine Location: Peace River, FL, USA
Ore/Tailings Type: phosphate
Dam Height (m): 8
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 03-01-1952 Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam failed by sliding of the downstream and possibly also the upstream slope. Contributing causes may have included (1) retention of clear water above the level of the impounded phosphate clay slimes and directly against the upstream face of the dam, (2) active mining and blasting at adjacent locations downstream from the dam; and (3) inadequate stripping and grubbing of the dam/foundation contact.
Source: Anecdotal

Incident No.: 156
Dam/Mine Name: Unidentified
Mine Location: Alafia River, FL, USA
Ore/Tailings Type: phosphate
Dam Height (m): 8
Dam Type: WR   Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 02-01-1952 Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam incorporated a return-water canal at its downstream toe that impounded water against the downstream face of the dam. Failure occurred when water in the canal was released by a break in its confining dike. The rapid reduction in canal tailwater level probably caused rapid-drawdown instability of the downstream slope of the impoundment dam. The breach was about 100 feet wide, and the resulting release of phosphatic clay slimes impounded behind the dam produced suspended solids concentrations as high as 20,000 ppm in the Alafia River.
Source: Anecdotal

Incident No.: 6
Dam/Mine Name: Avoca Mines
Mine Location: Ireland
Ore/Tailings Type: copper
Dam Height (m):
Dam Type: WR   Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Tailings were confined by dikes of primarily tailings fill placed with a dragline and levelled with a bulldozer. The fill received little or no compaction, and dike slopes ranged from 33 to 38 degrees at or near the angle of repose of the material. Multiple failures of the dikes occurred, depositing impounded tailings into the nearby Avoca River.
Source: Brawner, 1979

DAM TYPE: WR   INCIDENT TYPE: 1A
INCIDENT CAUSE: SE

Incident No.: 112
Dam/Mine Name: Southern Clay
Mine Location: TN, USA
Ore/Tailings Type: clay
Dam Height (m): 5
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 09-07-1989
Incident Type: 1A Cause: SE
Quantity of Tailings Released (cu. m): 300
Tailings Travel Distance (m):
Incident Description:

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A low berm was constructed along the rim of an abandoned clay pit used for retention of process fines and clarification of process water. The berm fill was not compacted and incorporated brush and debris. The failure occurred due to uncontrolled seepage at the foundation contact after a period of rapid impoundment rise. About 80,000 gallons of process water was released.
Source: Division of Water Pollution Control, Tennessee Dept. of Health and Environment

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Incident No.: 91
Dam/Mine Name: Ollinghouse
Mine Location: Wadsworth, NV, USA
Ore/Tailings Type: gold
Dam Height (m): 5
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m): 120,000
Incident Information:
Date: 1985 Incident Type: 1A Cause: SE
Quantity of Tailings Released (cu. m): 25,000
Tailings Travel Distance (m): 1,500
Incident Description:
With no engineering supervision during construction, the dam fill was essentially uncompacted (less than 80% maximum dry density). Collapse of the fill occurred as saturation developed resulting in loss of freeboard, slumping of the slope, and breach of the dam.
Source: Nevada Dept. of Conservation and Nat. Res., Div. Water Res.; Centurion Gold Ltd., Vancouver

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Incident No.: 167
Dam/Mine Name: Unidentified
Mine Location: Peace River, FL, USA
Ore/Tailings Type: phosphate
Dam Height (m): 6
Dam Type: WR Dam Fill Material: MW
Impoundment Volume (cu. m):
Incident Information:
Date: 09-01-1951
Incident Type: 1A Cause: SE
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam was constructed of mine waste, probably sands and clays. At the time of failure, the impoundment contained about 12 feet of phosphatic clay slimes and 1.5 feet of water in direct contact with the upstream embankment face. Failure occurred by seepage and piping on the downstream face of the embankment, with a possible contributing factor being 1.6 inches of rainfall prior to failure. The released slimes produced suspended solids concentrations of 15,000 ppm in a creek immediately adjacent to the impoundment and 800 ppm in the Peace River farther downstream.
Source: Anecdotal

Incident No.: 166
Dam/Mine Name: Unidentified
Mine Location: Peace River, FL, USA
Ore/Tailings Type: phosphate
Dam Height (m): 30
Dam Type: WR Dam Fill Material: MW
Impoundment Volume (cu. m):
Incident Information:
Date: 07-01-1951
Incident Type: 1A Cause: SE
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam had been constructed to a height of 100 feet using draggling-cast mine waste, probably consisting of sands and clays. At the time of failure, phosphate slimes and clear water were impounded to a depth of 25 feet, and failure is believed to have been due to seepage and piping, perhaps exacerbated by considerable rainfall just prior to the failure. The released slimes produced suspended solids concentrations as high as 800 ppm in the Peace River.
Source: Anecdotal

DAM TYPE: WR INCIDENT TYPE: 1A
INCIDENT CAUSE: FN

Incident No.: 209
Dam/Mine Name: Los Frailes
Mine Location: 45 km west of Seville, Spain
Ore/Tailings Type: pyretic zinc-lead-copper
Dam Height (m): 27
Dam Type: WR Dam Fill Material: R
Impoundment Volume (cu. m): 15,000,000
Incident Information:
Date: 01.00 on 04-24-1998
Incident Type: lA Cause: FN
Quantity of Tailings Released (cu. m): 6,800,000
Tailings Travel Distance (m): 40,000
Incident Description:
Dam built of waste rockfill was designed to have an upstream sloping earth core connected to a slurry trench cut-off passing through the alluvial gravels into underlying marl. A 600m long section slid forwards and opened like a gate, the body of the dam remaining intact. Subsequent site investigation indicated that failure occurred along a shear plane about 14m below the base of the dam. Water leakage through an adjacent section may have been a contributory factor.
Source: Boliden News Release 25th to 28th April 1998 and other reports.

Incident No.: 187
Dam/Mine Name: No 3 tailings pond
Mine Location: Sipalay, Negros Occidental, Philippines.
Ore/Tailings Type: copper
Dam Height (m):
Dam Type: WR Dam Fill Material: MW
Impoundment Volume (cu. m): 37 Mt
Incident Information:
Date: 16.00 on 11-08-1982
Incident Type: lA Cause: FN
Quantity of Tailings Released (cu. m): 27Mt
Tailings Travel Distance (m):
Incident Description:
Failure of section of dam due to slippage of foundation on clayey soil; surface materials not removed prior to construction. Inadequate anchoring of starter dam; use of mixed mine waste of very variable particle size.
Source:

Incident No.: 173
Dam/Mine Name: United Nuclear
Mine Location: Churchrock, NM, USA
Ore/Tailings Type: uranium
Dam Height (m): 11
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m): 370,000
Incident Information:
Date: 07-16-1979
Incident Type: 1A Cause: FN
Quantity of Tailings Released (cu. m): 700
Tailings Travel Distance (m):
Incident Description:
Differential foundation settlement, aggravated by a high operating pond level and narrow tailings beach, caused embankment cracking and failure by piping. Post-failure Investigations showed that some foundation soils were subject to in excess of 10% collapse upon saturation. The absence of an adequate sand beach and direct contact of water with the embankment fill allowed piping to occur through cracks in the fill that developed in response to foundation settlement. Roughly 80 million gallons of released effluent traveled to the Rio Puerco, through Gallup, NM, and into Arizona for a distance of 60-70 miles before completely infiltrating into the streambed alluvium.
Source: Nelson and Kane, 1980; Sautter, 1984

Incident No.: 64
Dam/Mine Name: Kerr-McGee
Mine Location: Churchrock, NM, USA
Ore/Tailings Type: uranium
Dam Height (m): 9
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 04-01-1976
Incident Type: 1A Cause: FN
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Differential settlement of foundation soils caused embankment cracking and piping failure. A minor quantity of effluent was released.
Source: New Mexico State Engineers Office

DAM TYPE: WR INCIDENT TYPE: 1A
INCIDENT CAUSE: OT

Incident No.: 17
Dam/Mine Name: Bonsal
Mine Location: Anson County, NC, USA
Ore/Tailings Type: sand & gravel
Dam Height (m): 5
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m): 38,000
Incident Information:
Date: 08-01-1985
Incident Type: 1A Cause: OT
Quantity of Tailings Released (cu. m): 11,000
Tailings Travel Distance (m): 800
Incident Description:
A rainfall of 7 to 9 inches in 12 hours caused an overtopping failure that released clay tailings and 3 million gallons of water into an adjacent stream. The dam is reported to have had an outlet or spillway of unknown type, and the breach occurred at this location. The materials released caused some damage at the sand and gravel plant immediately downstream but were contained in the plant's freshwater pond located downstream from the plant. The dam was repaired and placed back in service.
Source: North Carolina Dept. of Environ. Health and Nat. Res., Land Quality Section

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Incident No.: 74
Dam/Mine Name: Madison
Mine Location: Madison County, MO, USA
Ore/Tailings Type: lead
Dam Height (m): 11
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 02-28-1977
Incident Type: 1A Cause: OT
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam overtopped during an intense 6-inch rainfall due to inadequate spillway capacity.

Tailings flowsliding did not develop, although tailings were eroded by the impounded water flowing through the breach. These tailings were subsequently deposited throughout the city of Fredricktown.
Source: Missouri Dept. of Nat. Res., Dam and Reservoir Safety Program

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Incident No.: 93
Dam/Mine Name: Park
Mine Location: United Kingdom
Ore/Tailings Type: china, clay
Dam Height (m): 3
Dam Type: WR Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1970 Incident Type: 1A Cause: OT
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Overtopping failure occurred due to ice blockage of a decant structure.
Source: Ripley, 1972

DAM TYPE: WR INCIDENT TYPE: 1A INCIDENT CAUSE: ST

Incident No.: 179
Dam/Mine Name: Virginia Vermiculite
Mine Location: Louisa County, VA, USA
Ore/Tailings Type: vermiculite
Dam Height (m): 9
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 1984 Incident Type: 1A Cause: ST
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A pipe spillway through the clay-shale embankment collapsed and caused the dam to fail A downstream impoundment contained the tailings released. The dam was repaired by plugging the old spillway and installing a new spillway.
Source: Virginia Dept. Mines, Minerals and Energy, Div. Mined Land Reclamation

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Incident No.: 127
Dam/Mine Name: Unidentified
Mine Location:
Ore/Tailings Type:
Dam Height (m):
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 1A Cause: ST
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam consisted of a homogeneous section of well-compacted clay shale and sandstone- derived fill. The decant pipe, which penetrated the embankment, ruptured due to minor differential settlement. Embankment failure occurred rapidly.
Source: Casagrande and McIver, 1971

DAM TYPE: WR INCIDENT TYPE: 1A INCIDENT CAUSE: MS

Incident No.: 73
Dam/Mine Name: Iwiny
Mine Location: Lower Silesia, Poland
Ore/Tailings Type: copper
Dam Height (m): 25
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m): 16,000,000
Incident Information:
Date: 09-17-1967
Incident Type: 1A Cause: MS
Quantity of Tailings Released (cu. m): 4,600,000
Tailings Travel Distance (m): about 15,000
Incident Description:
The dam was constructed on alluvium underlain by a 20-m wide fault zone, in a general area of underground mining. 3rd stage of dam was almost completed, no signs of defects, e.g. cracking, seepage, or wet areas were detected. Underground mining was approaching the fault and pumping from the mine had lowered the water table. The dewatering and vibrations from rockbursts are thought to have loosened the fault gouge such that upward stopping created a cavity and ultimately a sinkhole beneath the upstream slope of the dam. A breach occurred near S end of dam: liquefied tailings swept down the valley with a width of 50m to 220m, covering 7 small villages, destroying the railway and killing 18 people. Dam breach and loss of impoundment contents resulted
Source: Wolski, et al, 1976; ICOLD Tailings Committee.

DAM TYPE: WR INCIDENT TYPE: 1A INCIDENT CAUSE: ER

Incident No.: 205
Dam/Mine Name: Tailings dam No 1
Mine Location: Omai gold mine, Guyana
Ore/Tailings Type: gold
Dam Height (m): 44
Dam Type: WR Dam Fill Material: R
Impoundment Volume (cu. m): 5,250,000
Incident Information:

Date: 23.55 on 08-25-1995
Incident Type: lA Cause: ER
Quantity of Tailings Released (cu. m): 4,200,000
Tailings Travel Distance (m):
Incident Description:
Dam of waste rockfill with compacted saprolite core supported by sand filter. Raised above original height progressively to match the impoundment. Rear deposition with water against dam. Piping failure, initially around construction drain pipe, carried core and materials through rockfill. Cyanide contamination caused minor fish kill in Omai river. Pollution of the much larger Essequibo river negligible: Canadian drinking water standards not exceeded.
Source: Reports from Guyana and Vick 1996.

DAM TYPE: WR INCIDENT TYPE: 1A INCIDENT CAUSE: U

Incident No.: 31
Dam/Mine Name: Cities Service
Mine Location: Fort Meade, FL, USA
Ore/Tailings Type: phosphate
Dam Height (m): 15
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m): 12,340,000
Incident Information:
Date: 12-03-1971 Incident Type: 1A Cause: U
Quantity of Tailings Released (cu. m): 9,000,000
Tailings Travel Distance (m): 120,000
Incident Description:
Breach of the dam allowed the phosphatic clay slimes to enter the Peace River, where they were carried in suspension for 120 km. Although the slimes are non-toxic to humans, a vast fish kill occurred when the slimes coated the gills of fish, causing them to suffocate. The cause of the failure is unknown, although the dam was observed to have been intact and with no signs of distress 15 minutes before the failure occurred.
Source: Lucia, 1981; Environmental Science and Technology, 1974

DAM TYPE: WR INCIDENT TYPE: 1B INCIDENT CAUSE: OT

Incident No.: 190
Dam/Mine Name: Rossarden
Mine Location: Rossarden, Tasmania
Ore/Tailings Type:
Dam Height (m): 7.5
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m): 200,000
Incident Information:
Date: 05-16-1986 Incident Type: lB Cause: OT
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Dam built in 1931 on a valley side l90 m above the river, of layered earth and tea-tree matting, in an uncontrolled manner. Water breached the main side of the impoundment area, swept through the impoundment and overtopped the front dam causing failure. The river was polluted.
Source: Inspector of Mines, Tasmania.

DAM TYPE: WR INCIDENT TYPE: 1B INCIDENT CAUSE: SI

Incident No.: 206
Dam/Mine Name: Placer Bay, Tailing Pond No.5
Mine Location: Surigao Del Norte, Philippines
Ore/Tailings Type:
Dam Height (m): 17
Dam Type: WR Dam Fill Material: E
Impoundment Volume (Cu. m):
Incident Information:
Date: 9:20 on 09-02-1995
Incident Type: lB Cause: SI
Quantity of Tailings Released (Cu. m): 50,000
Tailings Travel Distance (m): out to sea
Incident Description:
Inactive impoundment. Waste rock being placed on top of tailings for storage when dam failed into the sea forming 100m breach. Cause partly toe erosion and foundation on reclaimed land. 12 killed including safety inspector, and 14 pieces of heavy equipment that had been working to place waste rock on the impoundment.
Source: Reports from Republic of the Philippines

DAM TYPE: WR INCIDENT TYPE: 2A INCIDENT CAUSE: SI

Incident No.: 122
Dam/Mine Name: Texasgulf 4B Pond
Mine Location: Beaufort County, NC, USA
Ore/Tailings Type: phosphate
Dam Height (m): 8
Dam Type: WR Dam Fill Material: T
Impoundment Volume (cu. m): 12,300,000
Incident Information:
Date: 04-01-1984 Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A 200-ft long shallow slide occurred on the downstream slope of this slimes dam at the point of transition where the slope flattened from 3H:1V to 6H:1V. Clay spoil from adjacent ditch excavation placed on the embankment slope had blocked seepage, which caused the phreatic surface to rise and slope instability to occur. The

slope was repaired by installing filtered drainage trenches.
Source: North Carolina Dept. of Environ. Health and Nat. Res., Land Quality Section; Texasgulf Inc., Raleigh

Incident No.: 123
Dam/Mine Name: Texasgulf No. 1 Pond
Mine Location: Beaufort County, NC, USA
Ore/Tailings Type: phosphate
Dam Height (m):
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m): 24,700,000
Incident Information:
Date: 1981-1983 Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Several slides occurred from 1981 through 1983 along a several thousand foot section of the downstream slope of the phosphate slimes pond dike. The instability was caused by seepage related to clay layers deposited in the dredged dike fill. The slope failures were repaired by installation of filtered drains.
Source: North Carolina Dept. of Environ. Health and Nat. Res., Land Quality Section

Incident No.: 118
Dam/Mine Name: Suncor E-W Dike
Mine Location: Alberta, Canada
Ore/Tailings Type: oil sands
Dam Height (m): 30
Dam Type: WR Dam Fill Material: MW
Impoundment Volume (cu. m):
Incident Information:
Date: 1979 Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Slope instability occurred during construction of the dam. Remedial measures included slope flattening and incorporation of horizontal internal sand zones to enhance pore pressure dissipation.
Source: Morgenstern, et al, 1988

DAM TYPE: WR INCIDENT TYPE: 2A
INCIDENT CAUSE: FN

Incident No.: 186
Dam/Mine Name: Heath Steele main dam
Mine Location: Bathurst, New Brunswick
Ore/Tailings Type: Copper/zinc
Dam Height (m): 30
Dam Type: WR
Dam Fill Material: Rock, glacial till & clay core
Impoundment Volume (cu. m):
Incident Information:
Date: Since 1970s
Incident Type: 2A Cause: FN
Quantity of Tailings Released (cu. m): Seepage of contaminated water
Tailings Travel Distance (m):
Incident Description:
Leakage of water containing copper and zinc. Dam built on fractured bedrock, with no liner or grouting.
Source: DoE, Canada

Incident No.: 89
Dam/Mine Name: N'yukka Creek
Mine Location: USSR
Ore/Tailings Type:
Dam Height (m): 12
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 1965 Incident Type: 2A Cause: FN
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam was constructed as a starter dike for subsequent upstream raising, and operated initially to retain water. During first filling, sinkholes appeared in both abutments, and were initially treated by covering with tailings. When this proved ineffective, a concrete cutoff wall was constructed through the embankment and into the foundation. Sinkhole development was attributed to thawing of foundation permafrost that allowed ice-filled joints in foundation rock to transmit seepage and piping to occur.
Source: Biyanov, 1976

Incident No.: 141
Dam/Mine Name: Unidentified
Mine Location: USA
Ore/Tailings Type: magnesia
Dam Height (m): 6
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A Cause: FN
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Confining dikes were constructed on soft tidal flats foundation materials by casting fill with a dragline and later hauling it with trucks and spreading with bulldozers. The dikes experienced severe deformation and cracking, slumping, and bulging at the toe. Emergency remedial action averted failure.
Source: Wahler and Schlick, 1976

Incident No.: 134

Dam/Mine Name: Unidentified
Mine Location: Europe
Ore/Tailings Type:
Dam Height (m): 24
Dam Type: WR Dam Fill Material: R
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A Cause: FN
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A concrete faced rockfill starter dike was constructed on a karstic limestone foundation. Although a grout blanket was placed over exposed limestone within the impoundment area, a number of sinkholes developed during initial filling that drained the reservoir. One area of sinkhole formation was at the upstream toe of the concrete facing, and investigations revealed interconnected solution cavities in this area as well as extensive caverns in the dam abutment near the crest. Repairs included careful excavation of solution features and plugging with a mixture of mine waste and concrete accompanied by placement of a 10-foot thick mine waste layer over treated areas to bridge and plug potential future sinkholes that might develop.
Source: Robinson and Toland, 1979

DAM TYPE: WR INCIDENT TYPE: 2A
INCIDENT CAUSE: ST

Incident No.: 61
Dam/Mine Name: Irelyakh
Mine Location: USSR
Ore/Tailings Type:
Dam Height (m): 10
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A Cause: ST
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam was being built as a starter dike for subsequent upstream embankment raising. Used initially to retain water, excessive seepage developed at the contact of the wooden spillway and the foundation. This was attributed to the poor quality of winter construction.
Source: Biyanov, 1976

DAM TYPE: WR INCIDENT TYPE: 2A
INCIDENT CAUSE: EQ

Incident No.: 26
Dam/Mine Name: Cerro Blanco de Polpaico
Mine Location: Chile
Ore/Tailings Type: limestone
Dam Height (m): 9
Dam Type: WR Dam Fill Material: R
Impoundment Volume (cu. m):
Incident Information:
Date: 03-28-1965
Incident Type: 2A Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The rockfill dam with slideslopes of about 1.5:1.0 was subjected to the M7-7 1/4 La Ligua earthquake. Damage consisted of shallow longitudinal cracking on the crest.
Source: Dobry and Alvarez, 1967

DAM TYPE: WR INCIDENT TYPE: 2A
INCIDENT CAUSE: ER

Incident No.: 95
Dam/Mine Name: Pinchi Lake
Mine Location: British Columbia, Canada
Ore/Tailings Type: mercury
Dam Height (m): 13
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 1971 Incident Type: 2A Cause: ER
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The tailings dam consisted of a homogeneous section of compacted glacial till. Water decanted from the impoundment flowed in an unlined channel parallel to the downstream toe of the dam. Erosion of the unlined channel produced downcutting of as much as 12 feet. This triggered cracking and deformation of the downstream embankment slope, with movements seated within lacustrine foundation sediments at a depth coincident with the eroded channel bottom. Movements were stabilized by placement of a berm and relocation of the channel.
Source: Brawner, 1979

DAM TYPE: WR INCIDENT TYPE: 2B
INCIDENT CAUSE: ER

Incident No.:198
Dam/Mine Name: Kojkovac
Mine Location: Majkovac, Montenegro.
Ore/Tailings Type: lead/zinc
Dam Height (m):
Dam Type: WR Dam Fill Material: E
Impoundment Volume (Cu. m): 3,500,000
Incident Information:
Date: 11-1992 Incident Type: 2B Cause: ER

Quantity of Tailings Released (cu. m): none
Tailings Travel Distance (m):
Incident Description:
Earthfill dam with plastic liner built about 1970 alongside Tara river. Dam toe eroded by flooded river, caused slip that halved thickness of crest, but no overtopping. River diverted and toe protected by gabions under UN emergency project to prevent pollution that would have affected Danube.
Source: UNDRO, Geneva.

DAM TYPE: WR INCIDENT TYPE: 3
INCIDENT CAUSE: NR

Incident No.: 98
Dam/Mine Name: Rain Starter Dam
Mine Location: Elko, NV, USA
Ore/Tailings Type: gold
Dam Height (m): 27
Dam Type: WR Dam Fill Material: ER
Impoundment Volume (cu. m): 1,500,000
Incident Information:
Date: 1988 Incident Type: 3 Cause:
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Unanticipated seepage occurred through the impoundment bottom that resulted in an effluent spring downstream from the dam discharging 5 gpm. Seepage was retained by a catchment dam and pumped back to the impoundment.
Source: Nevada Dept. of Conservation and Nat. Res., Div. Water Res.

Incident No.: 35
Dam/Mine Name: Dam No. 1
Mine Location: Elliot Lake Ontario, Canada
Ore/Tailings Type: uranium
Dam Height (m): 9
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 1979 Incident Type: 3 Cause:
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Measures to reduce seepage were undertaken in conjunction with abandonment and closure of the impoundment. These included an embankment buttress with an internal synthetic impervious membrane, and cement-bentonite grouting of selected zones of the rock foundation. Post-construction monitoring indicated seepage of less than 1 gpm.
Source: Reades, et al, 1981

Incident No.: 161
Dam/Mine Name: Unidentified
Mine Location: Green River, WY, USA
Ore/Tailings Type: trona
Dam Height (m): 18
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 1975 Incident Type: 3 Cause:
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam was constructed to retain tailings and provide evaporation of effluent from trona mined for soda ash processing. Foundation conditions consisted of highly fractured rock with open joints, and the dam initially incorporated a nominal cutoff. When seepage containing high salt concentrations emerged on the surface downstream from the dam, foundation grouting was performed but failed to stop the seepage. Subsequently, an interceptor trench was excavated at the downstream toe to depths up to 60 feet and backfilled with drainage material. Pumping from wells installed in the interceptor trench was effective in preventing further downstream seepage migration.
Source: Anecdotal

Incident No.: 155
Dam/Mine Name: Unidentified
Mine Location: WY, USA
Ore/Tailings Type: trona
Dam Height (m): 24
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 3 Cause:
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam was constructed to retain tailings and evaporate effluent from trona processed for soda ash, and included a cutoff to reduce seepage through highly fractured foundation bedrock. Approximately 150 gpm of underseepage occurred, somewhat greater than anticipated quantities and of environmental concern due to deposits it produced on the ground surface. No remedial measures were adopted due to the presence of a secondary pond downstream where seepage was collected and returned to the main impoundment.
Source: Robinson and Toland, 1979

Incident No.: 132
Dam/Mine Name: Unidentified
Mine Location:
Ore/Tailings Type: gold

Dam Height (m): 8
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 3 Cause:
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The impoundment was lined with a 40-mil PVC geomembrane without an overlying soil cover. During initial tailings deposition when tailings had accumulated over the liner to an average depth of less than one foot, an air bubble developed beneath the liner and lifted it about 20 feet over a 100-ft diameter area. The cause of bubble formation was not determined but may have been related to formation of water vapor from subgrade soil moisture. The liner over the bubble ruptured, allowing a small quantity of tailings and retained fluid to escape. The liner was repaired after decontamination and cleanup. Other smaller bubble areas were vented using a special apparatus.
Source: Anecdotal

Incident No.: 148
Dam/Mine Name: Unidentified
Mine Location: CO, USA
Ore/Tailings Type:
Dam Height (m):
Dam Type: WR Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 3 Cause:
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A soil-bentonite slurry cutoff was constructed to reduce unacceptable quantities of contaminated foundation underseepage. The cutoff was installed to the greatest depth possible without expensive rock excavation. Remaining zones of underseepage through fractured rock below the bottom of the cutoff were identified by piezometers and locally grouted, effectively stopping the seepage.
Source: Taylor and Achhorner, 1984

DAM TYPE: GRAVITY INCIDENT TYPE: 1A INCIDENT CAUSE: ST

Incident No.: 189
Dam/Mine Name:
Mine Location: Itabirito, Minas Gerais, Brazil
Ore/Tailings Type: Iron ore
Dam Height (m): 30
Dam Type: Gravity Dam Fill Material: Masonry
Impoundment Volume (cu. m):
Incident Information:
Date: Late 05-1986
Incident Type: lA Cause: ST
Quantity of Tailings Released (tonnes): 100,000
Tailings Travel Distance (m): 12,000
Incident Description:
Dam of masonry construction using bricks made from clay and iron ore tailings burst, it is said, due to saturation of the brickwork. Loss of mining equipment and infrastructure: 7 dead.
Source: Engineering News Record, 5th June 1986.

DAM TYPE: VALLEY SIDE INCIDENT TYPE: 1B INCIDENT CAUSE: OT

Incident No.: 191
Dam/Mine Name: Story's Creek
Mine Location: Tasmania
Ore/Tailings Type:
Dam Height (m): 17
Dam Type: Valley side Dam Fill Material:
Impoundment Volume (cu. m): 30,000
Incident Information:
Date: 05-16-1986
Incident Type: lB Cause: OT
Quantity of Tailings Released (Cu. m): minimal
Tailings Travel Distance (m):
Incident Description:
Dam built in 1931 in an uncontrolled manner, mainly of tailings, with crest width of 1m and downstream slope 1:1. Overtopped during 1 in 100 year flood. Dam failed, spillway shifted; slimes released; pipeline washed out causing further pollution of waterway.
Source: Inspector of Mines, Tasmania.

DAM TYPE: NR INCIDENT TYPE: 1A INCIDENT CAUSE: SI

Incident No.: 180
Dam/Mine Name: Western Nuclear
Mine Location: Jeffrey City, WY, USA
Ore/Tailings Type: Uranium
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 1977 Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m): 40
Tailings Travel Distance (m):
Incident Description:
The dam slopes were steeper than 3:1, and melting of snow incorporated into the dam fill caused sufficient slumping to allow overtopping to occur. About 2.3 million gallons of effluent was released along with a small quantity of tailings, but no offsite contamination occurred.

Source: Teknekron, Inc., 1978

Incident No.: 217
Dam/Mine Name:
Mine Location: Stoney Middleton, UK
Ore/Tailings Type:
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (Mt):
Incident Information:
Date: 02-08-1968
Incident Type: lA Cause: SI
Quantity of Tailings Released (tons):
Tailings Travel Distance (m):
Incident Description:
The retaining dam of a settling pond burst and there was damage to property and roads.
Source: Penman and Charles 1990

Incident No.: 15
Dam/Mine Name: Bilbao
Mine Location: Spain
Ore/Tailings Type:
Dam Height (m):
Dam Type: Dam Fill Material: R
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 1A Cause: SI
Quantity of Tailings Released (cu. m): 115,000
Tailings Travel Distance (m):
Incident Description:
Sloughing of the rockfill dam following heavy rains caused large strains in the saturated tailings deposit that induced liquefaction and tailings flowsliding, with major downstream damage and loss of life.
Source: Smith, 1969

DAM TYPE: NR INCIDENT TYPE: 1A INCIDENT CAUSE: SE

Incident No.: 119
Dam/Mine Name: Sweeney Tailings Dam
Mine Location: Longmont, CO, USA
Ore/Tailings Type: sand and gravel
Dam Height (m): 7
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 05-01-1980
Incident Type: 1A Cause: SE
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Dam breached due to piping around outlet conduit.
Source: MHSA

DAM TYPE: NR INCIDENT TYPE: 1A INCIDENT CAUSE: FN

Incident No.: 207
Dam/Mine Name: Golden Cross
Mine Location: Waitekauri Valley, New Zealand
Ore/Tailings Type: gold
Dam Height (m): 25-30
Dam Type: Dam Fill Material: R
Impoundment Volume (Mt): 3
Incident Information:
Date: 12-1995
Incident Type: lA Cause: FN
Quantity of Tailings Released (cu. m): none
Tailings Travel Distance (m):
Incident Description:
Dam was on valley side and slipped bodily on plane of weakness about 50m deep at junction between lava flows and later material on which the dam was founded. Cost of repair NZ$ 5,000,000.
Source: Mine Manager, Coeur Golden Cross.

Incident No.: 197
Dam/Mine Name: No 2 tailings pond
Mine Location: Padcal, Luzon, Philippines.
Ore/Tailings Type: copper
Dam Height (m):
Dam Type: Fill Material:
Impoundment Volume (Mt): 80
Incident Information:
Date: 01-1992
Incident Type: 1A Cause: FN
Quantity of Tailings Released (Mt): 80
Tailings Travel Distance (m):
Incident Description:
Dam collapsed. Thought to relate to earthquake in July 1990 (6 months beforehand)
Source: Philex Mining Corp.

Incident No.: 183
Dam/Mine Name: Williamthorpe
Mine Location: United Kingdom
Ore/Tailings Type: coal
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 1966 Incident Type: 1A Cause: FN
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The failure is thought to have been triggered by excess foundation pore pressures.
Source: Bishop, 1973

Incident No.: 103
Dam/Mine Name: Santander

Mine Location: Spain
Ore/Tailings Type:
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 1A Cause: FN
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam had been constructed on a foundation containing old tailings. Lateral spreading and foundation strains triggered liquefaction, and a tailings flowslide occurred.
Source: Smith, 1969

DAM TYPE: NR INCIDENT TYPE: 1A INCIDENT CAUSE: OT

Incident No.: 199
Dam/Mine Name: Itogon-Suyoc
Mine Location: Baguio gold district, Luzon, Philippines.
Ore/Tailings Type: Gold
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 07-26-1993
Incident Type: lA Cause: OT
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Diversion tunnel taking river around the impoundment, blocked, causing flood water to enter impoundment and overtop dam. Partial failure, i.e. collapse of part of the dam. Contributory cause; typhoon.
Source: Itogon-Suyoc Mines

Incident No.: 114
Dam/Mine Name: Spring Creek Plant
Mine Location: Borger, TX, USA
Ore/Tailings Type: sand and gravel
Dam Height (m): 5
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m): 30,000
Incident Information:
Date: 08-01-1986
Incident Type: 1A Cause: OT
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A seven-inch rainfall caused overtopping and breach of the dam due to inadequate spillway capacity.
Source: MHSA

Incident No.: 216
Dam/Mine Name: Williamthorpe
Mine Location: UK
Ore/Tailings Type: coal
Dam Height (m):
Dam Type: Dam Fill Material: MW
Impoundment Volume (Mt):
Incident Information:
Date: 03-24-1966
Incident Type: lA Cause: OT
Quantity of Tailings Released (tons):
Tailings Travel Distance (m):
Incident Description:
A slurry pond that had been built into the Old Dirt Tip collapsed, sending a flow of tailings over an adjacent road which was covered to a depth of 3m and remained closed for 10 days.
Source: Penman and Charles 1990.

Incident No.:125
Dam/Mine Name: Tymawr
Mine Location: Rhondda valley, South Wales, UK
Ore/Tailings Type: coal
Dam Height (m): 25
Dam Type: Dam Fill Material: MWImpoundment Volume (Mt):
Incident Information:
Date: 03-29-1965
Incident Type: lA Cause: OT
Quantity of Tailings Released (tons):
Tailings Travel Distance (m): 732
Incident Description:
Lagoon has been formed in heaps of colliery waste on mountain side. When tailings level reached 175m, the downslope bund breached and the released tailings flowed down towards the river and entered the colliery car park at elevation 65m where it smashed two or three cars. It could easily have gone down the shaft. Cost to the Coal Board about £20,000 (1966 prices).
Source: Aberfan Report, Her Majesty's Stationery Office, 1967.

Incident No.: 124
Dam/Mine Name: Tymawr colliery.
Mine Location: South Wales, UK.
Ore/Tailings Type: coal
Dam Height (m):
Dam Type: Dam Fill Material: MW
Impoundment Volume (Mt):
Incident Information:
Date: 12-1961
Incident Type: lA Cause: OT
Quantity of Tailings Released (tons):
Tailings Travel Distance (m): 643
Incident Description:

Lagoon had been formed in the toe of a pile of colliery waste on a valley side at an elevation of about 183m, and washery tailings pumped to it by pipeline. The downslope bund overtopped and breached, releasing tailings that flowed down to an elevation of 65m near the Rhondda River.
Source: Aberfan Report, Her Majesty's Stationery Office, 1967.

Incident No.: 170
Dam/Mine Name: Union Carbide
Mine Location: Green River, UT, USA
Ore/Tailings Type: uranium
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 08-19-1959
Incident Type: 1A Cause: OT
Quantity of Tailings Released (cu. m): 8,400
Tailings Travel Distance (m):
Incident Description:
The tailings dam failed during a flash flood, with tailings and mill effluent reaching a creek and river. No increase in dissolved radium was noted in the river.
Source: US AEC, 1974

Incident No.: 215
Dam/Mine Name: Forquilha Mine Location: Brazil
Ore/Tailings Type: iron
Dam Height (m):
Dam Type: Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: lA Cause: OT
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The lower impoundment of this disposal scheme was under construction, in a valley adjacent to the active upper one. A saddle between the two valleys had a small dam to prevent overflow. At a time of maximum water level in the upper impoundment, a piping failure occurred at the left end of the saddle dam, releasing water into the lower impoundment, causing overtopping of the tailings dam under construction, washing out a considerable amount of fill. The accident caused significant delay to the operation of the scheme.
Source: ICOLD Tailings Committee.

DAM TYPE: NR INCIDENT TYPE: 1A INCIDENT CAUSE: ST

Incident No.: 208
Dam/Mine Name: Marcopper Mine
Location: Marinduque Island, Philippines
Ore/Tailings Type: copper
Dam Height (m):
Dam Type: Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 03-24-1996
Incident Type: lA Cause: ST
Quantity of Tailings Released (Mt): 2.4
Tailings Travel Distance (m):
Incident Description:
Tailings being stored in a worked-out pit that had been drained through a 2,250m long tunnel to the river Makulapnit. This had been plugged with concrete and the plug failed. Flow of tailings started at 5 to 10 m^3/sec and continued for 4 days. Released tailings affected waterways downstream. Heavy sedimentation for 14 km, and some material washed to rivermouth *25* km from mine.
Source: Placer Dome Inc., Vancouver V7X 1P1, Canada.

Incident No.: 193
Dam/Mine Name: No 3 tailings pond
Mine Location: Mankayan, Luzon, Philippines
Ore/Tailings Type: copper
Dam Height (m):
Dam Type: Dam Fill Material: E
Impoundment Volume (cu. m):
Incident Information:
Date: 10-17-1986
Incident Type: lA Cause: ST
Quantity of Tailings Released (Cu. m):
Tailings Travel Distance (m):
Incident Description:
Dam was on an ancient slide area, and slopes of the dam too steep. After 8m extra height had been added, the dam failed. Decant tower was too close to the dam.
Source: Lepanto Consolidated Mining Co., Makati, Philippines.

Incident No.: 22
Dam/Mine Name: Carr Fork
Mine Location: Tooele, UT, USA
Ore/Tailings Type: copper
Dam Height (m): 10
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 02-01-1975
Incident Type: 1A Cause: ST
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:

The embankment breached due to overtopping when a slide blocked the spillway structure.
Source: MHSA

Incident No.: 181
Dam/Mine Name: Western Nuclear
Mine Location: Jeffrey City, WY, USA
Ore/Tailings Type: uranium
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 03-23-1971
Incident Type: 1A Cause: ST
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A break in the tailings discharge line caused the dike to breach and tailings to flow for a period of 2 hours. No offsite contamination occurred.
Source: US AEC, 1974

DAM TYPE: NR INCIDENT TYPE: 1A INCIDENT CAUSE: EQ

Incident No.: 176
Dam/Mine Name: Veta de Agua A
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 1981 Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A dam adjacent to Veta de Agua No. 1 (designated Dam A) is reported to have failed during an earthquake in 1981. No other details are available.
Source: Castro and Troncoso, 1989

Incident No.: 177
Dam/Mine Name: Veta de Agua B
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 1981 Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A dam adjacent to Veta de Agua No. 1 (designated Dam B) is reported to have failed during an earthquake in 1981. No other details are available.
Source: Castro and Troncoso, 1989

Incident No.: 99
Dam/Mine Name: Ramayana No. 1
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m): 5
Dam Type: US Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 03-28-1965
Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m): 150
Tailings Travel Distance (m):
Incident Description:
Two nearly identical upstream type dams were located on a 30 degree mountainside slope and used alternately. Dam No. 1 was breached during the M7-7 1/4 Ligua earthquake, releasing a small flowslide from the upper portion of the impounded tailings.
Source: Dobry and Alvarez, 1967

Incident No.: 135
Dam/Mine Name: Unidentified
Mine Location: Peru
Ore/Tailings Type:
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 1962 Incident Type: 1A Cause: EQ
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
An earthquake of M6-3/4 that occurred in northern Peru following 3 weeks of heavy rainfall caused liquefaction failure of a tailings embankment located in the vicinity of the epicenter.
Source: Smith, 1969

DAM TYPE: NOT REPORTED INCIDENT TYPE: 1A CAUSE: MS

Incident No.: 88
Dam/Mine Name: Mulfilira
Mine Location: Zambia
Ore/Tailings Type: copper
Dam Height (m): 50
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 1970 Incident Type: 1A Cause: MS
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):

Incident Description:
Some 1,000,000 tons of tailings liquefied and flowed within 15 minutes into underground mine workings where mining was in progress beneath the impoundment, resulting in death of 89 miners. It is believed that voids in the rock above the workings propagated upward to the tailings due to unequal extraction of ore or differential settlement of the caving rock. The tailings deposit was stabilized by dewatering and the mining method was changed.
Source: Brawner, 1979; Sandy, et. al., 1976; Lucia, 1981

Incident No.: 5
Dam/Mine Name: Atlas Consolidated
Mine Location: Philippines
Ore/Tailings Type:
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 1A Cause: MS
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Coarse, saturated tailings deposited in an abandoned open pit liquefied and flowed into underground mine workings beneath the impoundment where mining was in progress. It is believed that voids in the rock above the workings propagated upward to the impoundment due to unequal extraction of ore or differential settlement of the caving rock.
Source: Brawner, 1979

DAM TYPE: NOT REPORTED
INCIDENT TYPE: 1A CAUSE: ER

Incident No.: 192
Dam/Mine Name: Pico de Sao Luis
Mine Location: Minas Gerais, Brazil
Ore/Tailings Type:
Dam Height (m): 20
Dam Type: Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 10-02-1986
Incident Type: lA Cause: ER
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Water flowing over spillway eroded dam toe on soft clay foundation, causing failure of downstream slope. Tailings released.
Source: ICOLD Tailings Committee.

DAM TYPE: NOT REPORTED
INCIDENT TYPE: 1A CAUSE: U

Incident No.: 182
Dam/Mine Name: Williamsport Washer
Mine Location: Maury County, TN, USA
Ore/Tailings Type: phosphate
Dam Height (m): 21
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 1970 Incident Type: 1A Cause: U
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
No details provided.
Source: MHSA

Incident No.: 1
Dam/Mine Name: Agrico Chemical
Mine Location: Payne Creek, FL, USA
Ore/Tailings Type: phosphate
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 1968 Incident Type: 1A Cause: U
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Dam breach resulted in pollution of a nearby creek and the Peace River. No further details are available.
Source: Anecdotal

Incident No.: 83
Dam/Mine Name: Mobil Chemical
Mine Location: Ft. Meade, FL, USA
Ore/Tailings Type: phosphate
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 1967 Incident Type: 1A Cause: U
Quantity of Tailings Released (cu. m): 250,000
Tailings Travel Distance (m):
Incident Description:
Dam breach and release of phosphatic clay slimes resulted in pollution of the Peace River. No further details are available.
Source: Anecdotal

Incident No.: 33
Dam/Mine Name: Climax
Mine Location: Grand Junction, CO, USA
Ore/Tailings Type: uranium
Dam Height (m):
Dam Type: Dam Fill Material:

Impoundment Volume (cu. m):
Incident Information:
Date: 07-02-1967 Incident Type: 1A Cause: U
Quantity of Tailings Released (cu. m): 12,000
Tailings Travel Distance (m):
Incident Description:
Failure due to unreported causes released 58,000 gallons of effluent into an adjacent river. There was no indication that dissolved radium concentrations in the river exceeded regulatory standards.
Source: US AEC, 1974

Incident No.: 4
Dam/Mine Name: American Cyanamid
Mine Location: Lithia, FL, USA
Ore/Tailings Type: phosphate
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 1965 Incident Type: 1A Cause: U
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Release of impounded phosphatic clay slimes caused pollution of an adjacent creek and the Alafia River. No further details are available.
Source: Anecdotal

Incident No.: 3
Dam/Mine Name: American Cyanamid
Mine Location: Brewster, FL, USA
Ore/Tailings Type: gypsum
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 1962 Incident Type: 1A Cause: U
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Dam breach caused release of 10 million gallons of acid water and pollution of the Alafia River. No further details are available.
Source: Anecdotal

Incident No.: 80
Dam/Mine Name: Mines Development
Mine Location: Edgemont, SD, USA
Ore/Tailings Type: uranium
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 06-11-1962 Incident Type: 1A Cause: U
Quantity of Tailings Released (cu. m): 100
Tailings Travel Distance (m):
Incident Description:
The dam failed from unreported causes. Tailings released reached a creek and some were carried 25 miles to a reservoir downstream.
Source: US AEC, 1974

Incident No.: 171
Dam/Mine Name: Union Carbide
Mine Location: Maybell, CO, USA
Ore/Tailings Type:
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 12-06-1961 Incident Type: 1A Cause: U
Quantity of Tailings Released (cu. m):
280Tailings Travel Distance (m):
Incident Description:
The dam failed from unreported causes. No damage was reported, and effluent released did not reach any flowing stream.
Source: U.S. AEC, 1974

Incident No.: 20
Dam/Mine Name: Captains Flat Dump 3
Mine Location: Australia
Ore/Tailings Type: copper
Dam Height (m):
Dam Type: Dam Fill Material: T
Impoundment Volume (cu. m):
Incident Information:
Date: 1942 Incident Type: 1A Cause: U
Quantity of Tailings Released (cu. m): 40,000
Tailings Travel Distance (m):
Incident Description:
No details of the failure are available, except that the tailings liquefied and the resulting tailings flowslide reached a nearby river.
Source: Ash, 1976

Incident No.: 136
Dam/Mine Name: Unidentified
Mine Location: South Africa
Ore/Tailings Type: gold
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 1917 Incident Type: 1A Cause: U
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Mention is made of a failed tailings dam. No further details are provided.
Source: White, 1917

Incident No.: 142
Dam/Mine Name: Unidentified

Mine Location: United Kingdom
Ore/Tailings Type:
Dam Height (m): 12
Dam Type: Dam Fill Material: R
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 1A Cause: U
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
An old tailings dam constructed within a quarry breached and was subsequently repaired. The cause is not reported.
Source: Little and Beavan, 1976

Incident No.: 2
Dam/Mine Name: Alcoa
Mine Location: Point Comfort, TX, USA
Ore/Tailings Type: bauxite
Dam Height (m): 19
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m): 4,500,000
Incident Information:
Date: 10-01-1964 Incident Type: 1A Cause: U
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
Cause of the failure is not reported. Released material was contained in a downstream impoundment.
Source: MHSA

DAM TYPE: NOT REPORTED
INCIDENT TYPE: 1B CAUSE: OT

Incident No.: 76
Dam/Mine Name: Marga
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 1985 Incident Type: 1B Cause: OT
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The cross-valley abandoned dam had a decant structure but no abandonment spillway. Overtopping failure occurred due to insufficient decant capacity for routing streamflows through the impoundment.
Source: Troncoso, 1990

Incident No.: 175
Dam/Mine Name: Vallenar 1 and 2
Mine Location: Chile
Ore/Tailings Type: copper
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 1983 Incident Type: 1B Cause: OT
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The two abandoned dams contained cross-valley impoundments in series, and incorporated only decant structures with no abandonment spillways. Overtopping caused failure of the upper dam and cascade failure of the lower dam.
Source: Troncoso, 1990

Incident No.: 151
Dam/Mine Name: Unidentified
Mine Location: IN, USA
Ore/Tailings Type: coal
Dam Height (m): Dam Type:
Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 1B Cause: OT
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
An abandoned coal slurry impoundment was breached by overtopping during heavy rains. No other details are available.
Source: Wobber, et al, 1974

DAM TYPE: NOT REPORTED
INCIDENT TYPE: 1B CAUSE: U

Incident No.: 39
Dam/Mine Name: Dixie Mine
Mine Location: Clear Creek County, CO, USA
Ore/Tailings Type: gold
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 04-01-1981 Incident Type: 1B Cause: U
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
No details provided.
Source: MHSA

Incident No.: 65
Dam/Mine Name: Keystone Mine
Mine Location: Creste Butte, CO, USA
Ore/Tailings Type: molybdenum
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:

Date: 05-01-1975 Incident Type: 1B Cause: U
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
No details provided.
Source: MHSA

Incident No.: 50
Dam/Mine Name: Golden Gilpin Mine
Mine Location: Blackhawk, CO, USA
Ore/Tailings Type: gold
Dam Height (m): 12
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 11-01-1974 Incident Type: 1B Cause: U
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
No details provided.
Source: MHSA

DAMTYPE: NOT REPORTED
INCIDENT TYPE: 2A CAUSE: SI

Incident No.: 160
Dam/Mine Name: Unidentified
Mine Location: Eastern US, USA
Ore/Tailings Type: coal
Dam Height (m): 150
Dam Type: Dam Fill Material: MW
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 2A Cause: SI
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
An embankment of coal refuse constructed with uncompacted, train-dumped fill retained impounded coal tailings and water. Downstream slope movements of 20 m/yr, accompanied by a high phreatic surface and large quantities of seepage, resulted in severe cracking and deformation of the embankment slope.
Source: Wahler and Schlick, 1976

DAMTYPE: NOT REPORTED
INCIDENT TYPE: 2A CAUSE: OT

Incident No.: 67
Dam/Mine Name: Kyanite Mining
Mine Location: Prince Edward Co., VA, USA
Ore/Tailings Type: kyanite
Dam Height (m): 11
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m): 430,000
Incident Information:
Date: 1980 Incident Type: 2A Cause: OT
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam was overtopped, but no breach occurred and tailings were not released. The dam was placed back in service with minor repairs.
Source: Virginia Dept. Mines, Minerals and Energy, Div. Mined Land Reclamation

Incident No.: 174
Dam/Mine Name: Utah Construction
Mine Location: Riverton, WY, USA
Ore/Tailings Type: uranium
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: 06-16-1963
Incident Type: 2A Cause: OT
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
The dam was intentionally breached and a 2-foot depth of effluent was released to prevent uncontrolled release of the impoundment contents during heavy rain.
Source: US AEC, 1974

DAM TYPE: NOT REPORTED
INCIDENT TYPE: 3 CAUSE: NR

Incident No.: 8
Dam/Mine Name: Bancroft
Mine Location: Ontario, Canada
Ore/Tailings Type: uranium
Dam Height (m):
Dam Type: Dam Fill Material:
Impoundment Volume (cu. m):
Incident Information:
Date: Incident Type: 3 Cause:
Quantity of Tailings Released (cu. m):
Tailings Travel Distance (m):
Incident Description:
A grout curtain installed in alluvial deposits was used to control seepage containing radium-226 from an existing, reactivated tailings dam. It was constructed by injecting a mixture of clay, water, cement, bentonite and occasionally calcium chloride through slotted pipes installed in small diameter boreholes drilled through the alluvium and into bedrock. Pump tests conducted after completion of the grout curtain showed that a thin zone along the bedrock surface remained ungrouted and that seepage through only the upper part of the alluvium was retarded. It was observed that the total effect of the curtain in reducing seepage was minimal. Yet the content of the dissolved radium-226 in the ground water

was greatly lowered in passing through and beneath the grout curtain. The decrease in contaminant concentrations was attributed to ion exchange between the effluent seepage and chemicals in the grout curtain.
Source: Dodds, 1979

ISSN 0534-8293

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MINNESOTA CLIMATE AND HEALTH PROFILE REPORT **2015**

An Assessment of Climate Change Impacts on the Health & Well-Being of Minnesotans

Minnesota Department of Health

MINNESOTA CLIMATE & HEALTH PROGRAM, ENVIRONMENTAL IMPACTS ANALYSIS UNIT

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February 2015

Suggested citation:

Minnesota Climate & Health Profile Report. Minnesota Department of Health. St. Paul, MN. February 2015.

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Contents

I Executive Summary

Changes are occurring in Minnesota's climate with serious consequences for human health and well-being. Minnesota has become measurably warmer, particularly in the last few decades, and precipitation patterns have become more erratic, including heavier rainfall events. Climate projections for the state indicate that these trends are likely to continue well into the current century and may worsen, according to some scenarios.

The *Minnesota Climate & Health Profile Report (Profile Report)* provides a comprehensive assessment of climate change impacts and potential health burden for the state. In addition to describing climate trends and projections relevant to Minnesota, the *Profile Report* identifies how these climate changes are linked to health impacts along with opportunities and challenges for developing quantitative measures of health outcomes.

Minnesota's continental climate is characterized by seasonal variations in temperature and precipitation. In addition, given the size and geographic diversity of the state, temperatures can vary substantially depending on location. Air pollution, extreme heat, flooding, drought, and ecosystem threats were identified as hazards most likely to occur from a changing climate that are especially relevant to Minnesotans. A central theme of the *Profile Report* is that climate impacts are experienced both directly and indirectly and can affect a wide spectrum of social and environmental factors that are well-recognized determinants of health. Because Minnesota's climate varies significantly across the state, the types and scope of hazards likely to be experienced by a specific community will depend on local factors, thereby presenting a difficult challenge for planners and elected officials.

The following hazard-impact pathways are described in detail:

- Air pollution (ozone, particulate matter, pollen)
  - Direct health impacts: chronic obstructive pulmonary disease, lung cancer, cardiovascular disease, allergies and asthma
- Extreme heat
  - Direct health impacts: mortality, heat stress, and other conditions exacerbated by heat
  - Indirect health impacts: infrastructure failures, strain on essential services and disruption of key social networks
- Floods and drought
  - Direct health impacts: drowning and injuries, waterborne disease, and mental stress
  - Indirect health impacts: respiratory ailments, disruption of essential services, fiscal strain, loss of livelihood, and threat to community cohesion
- Ecosystem threats
  - Direct health impacts: West Nile virus, Lyme disease, harmful algal blooms

The *Profile Report* concludes with a brief discussion of next steps required for an effective response to the identified climate hazards. The overarching goal of providing this information is to highlight what is currently known about each climate hazard and potential health impacts, as well as data or information gaps and various system challenges that are making it difficult to advance our knowledge base and promote climate adaptation actions. It is hoped that the *Profile Report* will be used by public health officials, practitioners, and other stakeholders in their efforts to understand and

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prepare for climate change impacts on the health of individuals within their own communities.

ACKNOWLEDGEMENTS

The *Profile Report* was funded through a cooperative agreement with MDH from the Centers for Disease Control and Prevention (CDC) Building Resilience Against Climate Effects (BRACE) program (5H13EH001125-02). The BRACE framework was developed by the CDC as an approach for state and local public health departments to address climate change impacts. BRACE is a multi-step process that facilitates public health professionals, climate experts, and other agency colleagues with developing and implementing effective climate adaptation strategies specific to state and local jurisdictions.

II Introduction

Changes occurring in Minnesota's climate are affecting the health and well-being of the people who live, work and play within its borders. While residents are well aware of the erratic nature of Minnesota weather, extreme events have become even more frequent and precipitation patterns have become even less predictable. Already these climate changes are impacting Minnesota's agricultural and industrial economies, natural resources, public infrastructure and population health. The risks are especially high for people who lack "climate resilience" due to age, income, residence or numerous other vulnerability factors that influence whether an individual can thrive in a changing climate.

The Minnesota (MN) Climate & Health Program has established a set of six goals outlined in a strategic plan for the Minnesota Department of Health (MDH). These goals will guide priorities and funding decisions for activities to minimize climate change impacts on the health of all Minnesotans:

Goal 1: MDH will understand, research, monitor, track, and report on the public health impacts of climate change.

Goal 2: MDH will identify and develop potential mitigation and adaptation strategies and tools to address climate change and public health.

Goal 3: MDH will identify populations that are at risk of poor health outcomes and sources.

Goal 4: MDH will enhance planning and preparedness for emergency and disaster response and recovery to effectively protect the public's health against negative impacts associated with climate change-related disasters.

Goal 5: MDH will increase the public health system's capacity to respond to and adapt to the public health impacts of climate change.

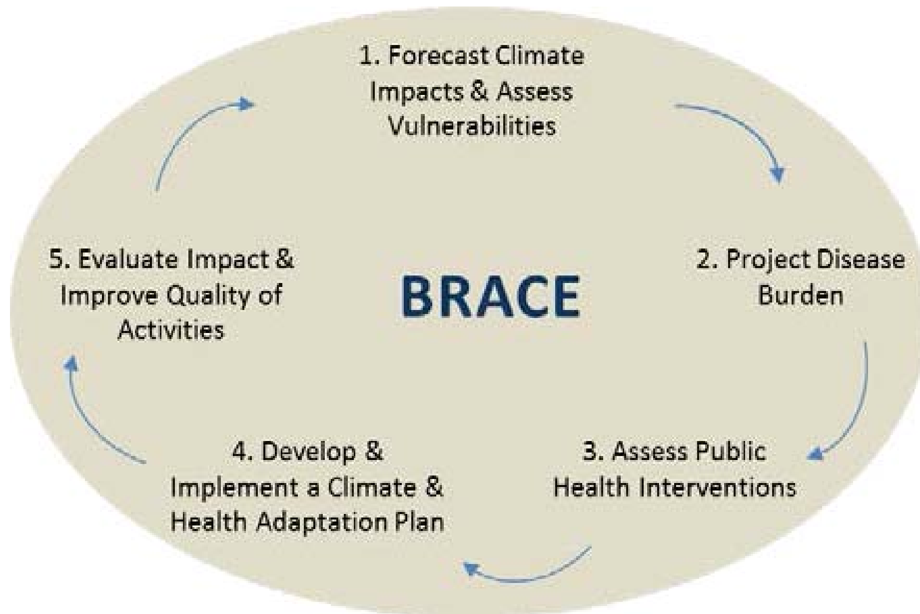
Goal 6: MDH will communicate and educate public health professionals, healthcare providers, state agency personnel, policy-makers, vulnerable populations and the general public on climate change's effects on human health.

In 2010 MDH's MN Climate & Health Program received funding from the Centers for Disease Control and Prevention (CDC) to conduct programmatic activities aimed at reducing health impacts of climate change through the Building Resilience Against Climate Effects (BRACE) framework (Figure 2.1). BRACE is a multi-step process that enables health departments to work with climate experts and other agency colleagues to incorporate the best available climate science into the development and implementation of a comprehensive climate and health adaptation strategy for their jurisdictions. The process results in lessons learned to improve future program efforts and ensure the best public health outcomes.

The *Minnesota Climate & Health Profile Report (Profile Report)* responds to the first step in the BRACE framework and a number of MDH programmatic goals by providing a comprehensive assessment of the climate impacts, population vulnerabilities, and potential health burden distinct to Minnesotans. The science and practice of public health are rooted in a view of health as a consequence of a wide range of factors (Figure 2.2). Some factors have to do with the individual, such as genetics, demographics, and lifestyle choices. Other factors have to do with the social and natural environments in which the individual lives, including the availability of clean air and water or access to secure employment, education and health care services.

When assessing the full range of health impacts from climate change, it is important to consider not only direct effects, such as heat wave deaths, but also indirect effects that are mediated by the environment. For example, extreme heat could indirectly affect many Minnesotans by hurting the state's agricultural economy, threatening the ability of farmers to maintain a self-supporting livelihood and consequently the long term viability of rural communities. A central theme of the *Profile Report* is that climate change impacts are experienced both directly and indirectly. These impacts can affect a wide range of social and environmental factors that are known determinants of health.

FIGURE 2.1. BUILDING RESILIENCE AGAINST CLIMATE EFFECTS (BRACE) FRAMEWORK. CDC, 2012.



Actions to reduce climate impacts will require economic and behavioral changes, bringing costs and benefits to different sectors of society. Decision-makers from these sectors will inevitably face challenges for support and resources to initiate changes. To provide a rational basis for prioritizing actions, decision-makers need an idea of the magnitude and distribution of health risks and related exposure factors connected to climate changes. Such estimates can be important for identifying the health outcomes and factors associated with the greatest burden of death, disease or distress in a population and those individuals that are most vulnerable. A number of leading health authorities and research institutions, such as the World Health Organization and the Institute for Health Metrics & Evaluation (IHME), have defined methods for quantifying health burdens relevant to climate change at the global or national level (Campbell-Lendrum et al., 2007; IHME, 2013). However, in order to apply these methods to the state or local level, a wide range of data is needed that is relevant to smaller scale populations. Furthermore, in many cases researchers and policy-makers are still working to understand the full range of health risks and exposure factors that are relevant to a particular climate change in a particular location, and may lack the resources to collect and analyze data for quantifying population-level health burdens.

The objective of the *Profile Report* is to explore existing datasets and identify gaps that could affect future efforts towards developing quantitative measures of health burden for Minnesotans impacted by climate change. Ultimately the goal of the *Profile Report* is to facilitate evidence-based approaches to climate change adaptation to protect public health. The *Profile Report* was written primarily for state and local public health staff, but also may prove useful for policy-makers, individuals, and organizations interested in the health and well-being of Minnesotans.

FIGURE 2.2. FACTORS THAT INFLUENCE HEALTH. IMAGE SOURCE: HIP, 2011.

Organization of the Report

The *Profile Report* is organized to reflect the basic components of risk assessment (Figure 2.3). In the context of public health, risk assessment is the process of characterizing the harmful effects to individuals or populations from certain hazards. Public health risk assessment includes a detailed analysis of the factors or pathways that lead to people being exposed to a hazard as well as specific health outcomes. The *Profile Report* is based on a view of climate change as a significant hazard to health that is likely to continue well into the future.

Section 3 of the *Profile Report* provides a brief overview of Minnesota's climate, geography and natural resources that will be impacted by climate change, with repercussions for the health of the population. Section 4 provides a review of climate science, both historical trends and future projections, as it pertains to Minnesota or the Midwest region. Section 5 links information on climate hazards to distinct health determinants or outcomes relevant to Minnesotans, with an emphasis on changes in the environment. For instance, increased storm severity is a climate change hazard affecting Minnesota. Minnesotans will be exposed to adverse impacts from increased storms through flooding, which can lead to outcomes such as reduced mental health, poor water quality, household instability, and damaged infrastructure.

The four distinct environmental changes described in Section 5 include alterations in air, weather, water and ecosystems. As a results of these changes, MDH considers the following hazards particularly compelling for the Minnesota population at this time: Air Pollution, Extreme Heat, Flood & Drought, and Ecosystem Threats.

Section 5 also includes summary tables briefly describing existing knowledge and data gaps pertaining to climate-mediated hazards, exposure pathways, and vulnerability factors that contribute to overall health risk. Vulnerability is an important factor in assessing risk given that certain demographic groups and environments are differentially impacted by climate changes. Vulnerability is a major focus for public health attention, and widely recognized as a determining factor in a person's ability to survive and thrive in a changing climate (IPCC, 2007). MDH's MN Climate & Health Program has conducted an extensive climate vulnerability assessment for areas of Minnesota, culminating in the Minnesota Climate Change Vulnerability Assessment (http://www.health.state.mn.us/divs/climatechange/data.html#ccva).

Section 6 provides a summary of key points from the *Profile Report* with an emphasis on areas where further information and research are required to support the preservation of health of Minnesotans against a changing climate.

FIGURE 2.3. HAZARD-RISK CONTINUUM FOR CLIMATE-MEDIATED HEALTH IMPACTS. THE RISK OF ADVERSE IMPACTS TO AN INDIVIDUAL'S HEALTH IS INFLUENCED BY THREE FACTORS: THE PRESENCE OF A HAZARD IN THE INDIVIDUAL'S ENVIRONMENT, THE OPPORTUNITY THAT EXISTS FOR THAT INDIVIDUAL TO BE EXPOSED TO THE HAZARD, AND CERTAIN CHARACTERISTICS OF THE PERSON (OR THEIR ENVIRONMENT) THAT MAY INCREASE THE LIKELIHOOD THAT ADVERSE IMPACTS WILL ARISE, I.E., VULNERABILITY.

CLIMATE HAZARD + EXPOSURE + VULNERABILITY = RISK

III Overview of Minnesota Geography & Climate

Climate change has already had observable effects on many resources in Minnesota that are crucial for our natural and built environments. Climate change also is impacting Minnesota's most important resource, its citizens. Before exploring in greater detail the links between climate changes in Minnesota and the health of the people who live, work and play here, the following provides a snapshot of what makes Minnesota a unique state with regard to geography and climate, and the resources that will be impacted as our climate continues to change.

The state of Minnesota covers nearly 80,000 square miles, making it the 12th largest state in the nation. Within these borders are resources that have supported human communities dating back past the native Anishinaabe and Dakota peoples. Minnesota's natural resources are rich and varied and have long braced the state's economy and framed its cultural identity.

Water. Nearly 21,000 square miles of the state is covered by water or wetlands, contributing to Minnesota's abundant water supply (Figure 3.1). There are nearly 12,000 lakes larger than 10 acres inside its borders, as well as more than 6,500 rivers and streams. In addition, there are approximately 11 million acres of wetlands within Minnesota, more than any other state except Alaska. Three counties contain 189 miles of shoreline and 82 beaches along Lake Superior. Minnesota also is considered to have abundant groundwater that supplies about 75 percent of the state's drinking water (DNR, 2014).

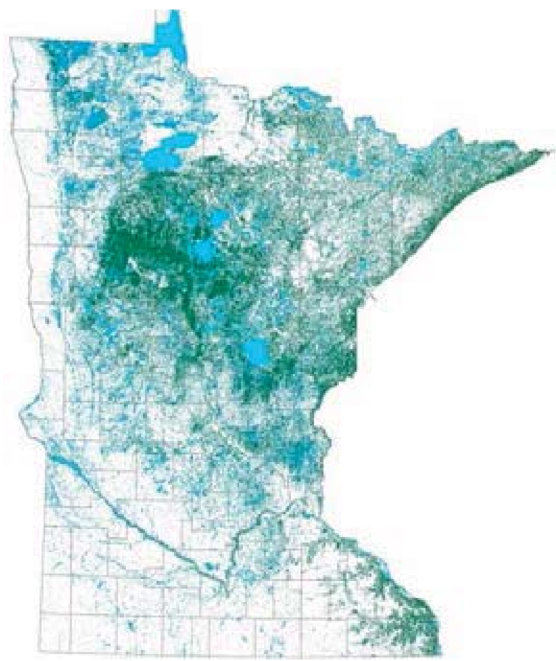
Forests. Minnesota is the 16th most forested state in the nation with more than 17 million acres of forested land and 52 native species of trees (Figure 3.1). The forest products industry provides an income to more than 40,000 Minnesotans and produces around seven billion dollars worth of timber-related products each year. However, the public owns most of Minnesota's forests, and people have access to nearly four million state-owned acres (Duffey & Hoff, 2008).

Wildlife. Minnesota hosts many varied wildlife species. There are 1,440 public wildlife management areas with nearly 1.3 million acres of habitat, from prairies and wetlands to forests and swamps. These areas not only sustain protected terrain for Minnesota birds, fish and animals, but provide recreation for hunters, fishers, hikers, bird-watchers, wildlife photographers, and other outdoor enthusiasts. Over 15 percent of Minnesotans hunt and 52 percent enjoy watching birds and other wildlife, the highest participation rate in the nation. Together these pursuits are a one billion dollar industry for the state. In addition, there are over one million licensed anglers in Minnesota utilizing 5,400 fishing lakes and over 15,000 miles of fishable rivers and streams, sustaining fishing as an important cross-generational sport (DNR, 2014).

Agriculture. Minnesota is the fifth largest agricultural producer in the nation, with nearly 81,000 farms covering 27 million acres and generating nearly 10 billion dollars in annual revenue (Figure 3.2). In 2011, Minnesota farmers harvested a combined 15 million acres of corn and soybeans, placing the state in the top five nationally for production of these two commodities. The economic contribution of Minnesota agriculture reaches beyond the farm with 80 percent of agricultural jobs located off the farm. There are 1,000 agricultural or food-related companies in the state, generating 55 billion dollars and supporting over 367,000 jobs (Ye, 2014).

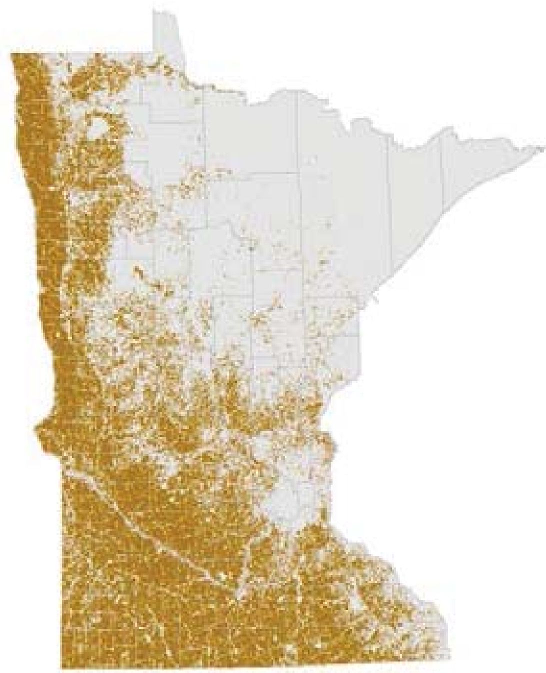
Air. Minnesotans also enjoy above average air quality. In 2011, nearly all areas of the state were in compliance with federal air standards. A 2014 report by the American Lung Association grouped Cass and St. Louis counties among the cleanest counties in the nation for particle pollution and Becker, Goodhue, Lake, Lyon, Mille Lacs, Olmsted, St. Louis, Stearns, and Wright for having the least ozone pollution. Rochester and Duluth were amongst the cleanest cities for ozone and particle pollution, respectively (ALA, 2014).

FIGURE 3.1. MINNESOTA SURFACE WATER AND WETLANDS (BLUE) AND FOREST AREAS (GREEN). USDA/NASS, 2014.



People. Minnesota is home to over five million people. Nearly 60 percent of the state's population lives in the seven-county Twin Cities metro area, making the Twin Cities the 13th most populous metro area in the U.S. The rate of population growth for the state as a whole is very close to the national average. It is projected that the population of Minnesota will grow 16 percent from 2013 to 2065, reaching 6.45 million by 2065 (MNDC, 2014).

FIGURE. 3.2. MINNESOTA AGRICULTURAL LAND (BROWN). USDA/NASS, 2014.

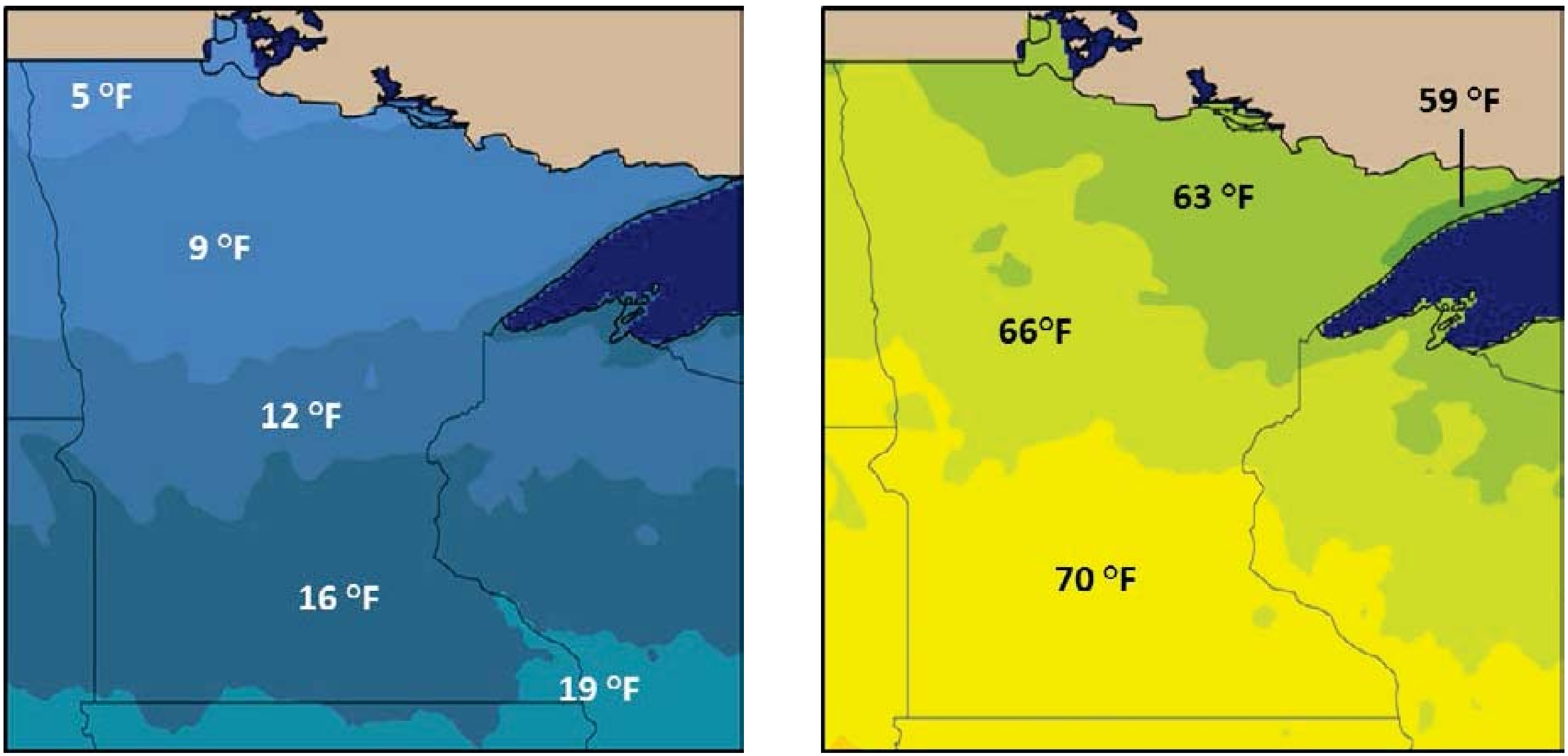


What is the difference between "weather" and "climate"?

The difference between weather and climate is a measure of time. Weather refers to conditions of the atmosphere over a short period of time, such as daily or weekly, while climate is the average daily weather over relatively long periods of time, such as by decade or century (NASA, 2014).

Minnesota's climate is classically continental, characterized by distinct seasonal variations in temperature and precipitation. Average statewide winter temperature, influenced by occasional bursts of Arctic polar air, is approximately 10°F. Average statewide summer temperature, affected by warm air pushing north from the Gulf of Mexico, is approximately 67°F (NOAA/NCDC, 2014). However, given the size and geographic diversity of the state, seasonal temperatures can vary substantially depending on location (Figure 3.3).

FIGURE 3.3. AVERAGE WINTER (DECEMBER – FEBRUARY) AND SUMMER (JUNE-AUGUST) TEMPERATURES ACROSS MINNESOTA BASED ON DATA FROM 1895-2012. DATA AND IMAGE SOURCE: CLIMATE REANALYZER (HTTP://CCI-REANALYZER.ORG), CLIMATE CHANGE INSTITUTE, UNIVERSITY OF MAINE.



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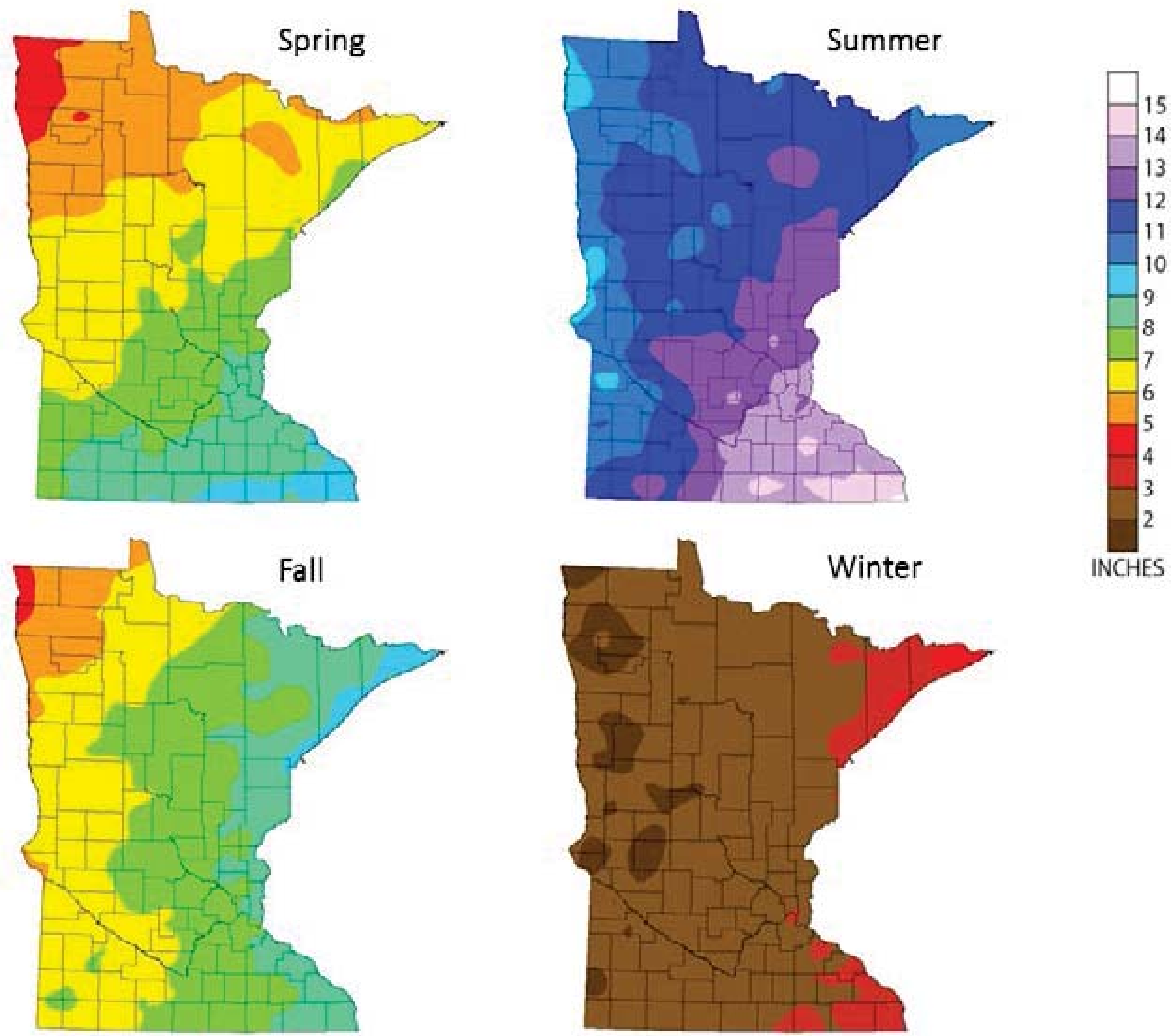
Precipitation is relatively moderate but also varies with season and location. Average annual precipitation (rainfall plus the water equivalent found in snowfall) can range from 32 inches in the southeast to 18 inches in the northwest portion of the state (DNR, 2014). Nearly two-thirds of Minnesota's precipitation falls as rain during the growing season, May through September (Figure 3.4).

The Minnesota State Climatology Office collects and manages a large amount of climate and weather data for Minnesota, some dating back to the early 19th century. A substantial amount of data is provided by participants in the National Weather Service (NWS) Cooperative Observer Program (COOP) (NWS, 2014). Minnesota's COOP network relies on numerous dedicated volunteers who collect vital weather and climate-related measurements from farms, urban and suburban areas, parks, and shorelines and report this information electronically to the NWS. These data not only allow for charting historical temperature and precipitation trends, but also facilitate climate projections and planning around floods, droughts, heat and cold waves, agriculture and construction projects. Along with agency data, COOP information plays a critical role in evaluating the extent of climate change from local to global scales and conversely, assists in understanding how climate changes will affect human health and the environment. A majority of climate data that exist for Minnesota are available on the Minnesota Climatology Working Group website (climate.umn.edu/), a collaboration between the State Climatology Office and the University of Minnesota.

FIGURE 3.4. AVERAGE SEASONAL PRECIPITATION (INCHES) ACROSS MINNESOTA BASED ON DATA FROM 1981-2010. IMAGE SOURCE: DNR, 2014.



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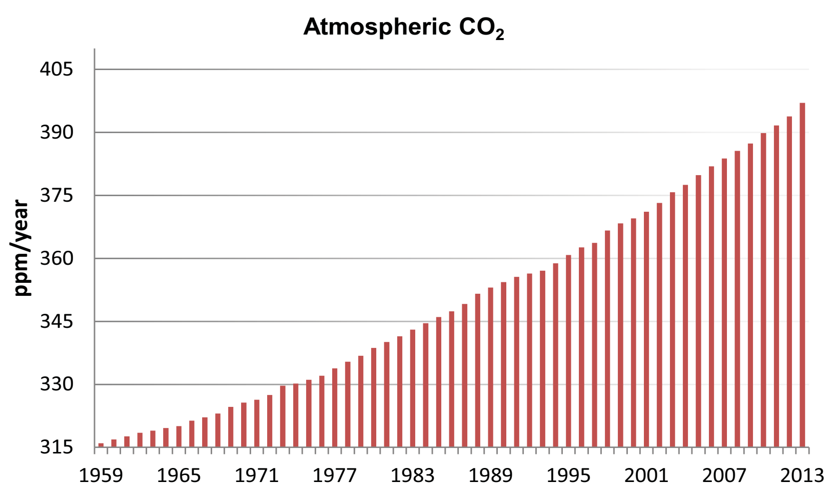
IV Climate Trends & Projections

Historical Trends

Human influences on the earth's climate have become increasingly apparent in recent decades. Since the onset of the industrial revolution, concentrations of carbon dioxide, nitrous oxide, and methane in the earth's atmosphere have increased significantly by approximately 40, 20 and 151 percent, respectively (IPCC, 2007). These gases are often referred to as greenhouse gases because of their ability to absorb and emit radiation both upwards to space and back down to the Earth's surface. The increase in greenhouse gases in the atmosphere, and resulting increase in solar energy retained by the planet, is the driving force behind climate change globally (IPCC, 2007).

FIGURE 4.1. ANNUAL AVERAGE ATMOSPHERIC CARBON DIOXIDE CONCENTRATION (GIVEN AS PARTS PER MILLION IN DRY AIR) MEASURED AT MAUNA LOA OBSERVATORY. DATA SOURCE: NOAA/ESRL, 2014.



The dramatic rise in atmospheric concentrations of these gases is driven by the burning of fossil fuels for transportation and energy, although land use changes (e.g. deforestation and agriculture) also contribute (Walsh et al., 2014). The emission of carbon dioxide (CO_2), the most common greenhouse gas released by human activity, has tripled since measurements began more than 50 years ago (NOAA/ESRL, 2014). For 800,000 years, the amount of CO_2 in the atmosphere fluctuated between 180 and 280 parts per million (ppm). However, recent levels have escalated to 400 ppm, and this upward trend appears likely to continue (Figure 4.1; NOAA/ESRL, 2014).

The effects of increasing greenhouse gas emissions, in particular CO_2, are observable in Minnesota's own climate. State experts in climatology have identified a number of climate trends affecting Minnesota, in particular (DNR, 2011 & 2013; Zandlo, 2008):

- Average annual temperature is rising with distinct daily and seasonal trends.
- Precipitation patterns are becoming more extreme with more heavy rainfall from storm activity.

In addition, although at this time there is insufficient data to identify a statistical trend in dew point measures, there is evidence that spikes in dew point (>70°F) have become higher and more frequent in recent decades. There also have been measurable impacts on Lake Superior.

TEMPERATURE

Minnesota has gotten noticeably warmer, especially over the last few decades. Since the beginning of the data record (1895), Minnesota's annual average temperature has increased by nearly 0.2°F per decade (equivalent to about 2.3°F per century), and over the last few decades this warming effect has accelerated. Data for the last half-century (1960-2013) show that the recent rate of warming for Minnesota has sped up substantially to 0.5°F per decade (5.3°F per century; Figure 4.2).

The increase is driving changes in the environment, affecting ice cover, soil moisture, bird migrations, insect behavior, and forest and plant growth. Some of these changes will impact the health of Minnesotans. For example, finding ways to cope with recent summer heat waves has made climate change less of an abstract concept for many residents, especially those who struggle to acclimate or adjust to these events, and the challenge will not likely ebb soon. Based on more than a century of data, 7 of the top 10 warmest years for Minnesota have occurred just within the last 15 years (Figure 4.3).

FIGURE 4.2. AVERAGE ANNUAL ALL-SEASON TEMPERATURE FOR MINNESOTA. BLUE (LEFT) AND RED (RIGHT) LINES HIGHLIGHT TRENDS FOR 1895-1959 AND 1960-2013, RESPECTIVELY. DATA SOURCE: NOAA/NCDC, 2014.

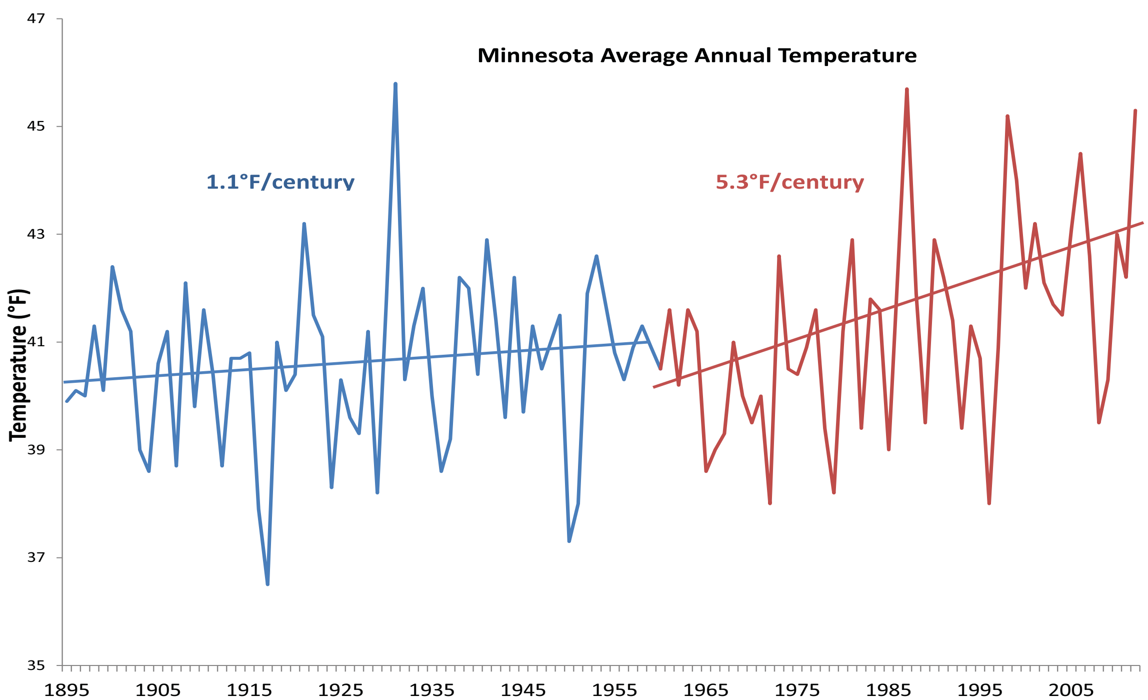


FIGURE 4.3. TOP TEN WARMEST YEARS FOR MINNESOTA BASED ON ANNUAL AVERAGE TEMPERATURE FOR YEARS 1895-2013. DATA SOURCE: NOAA/NCDC, 2014.

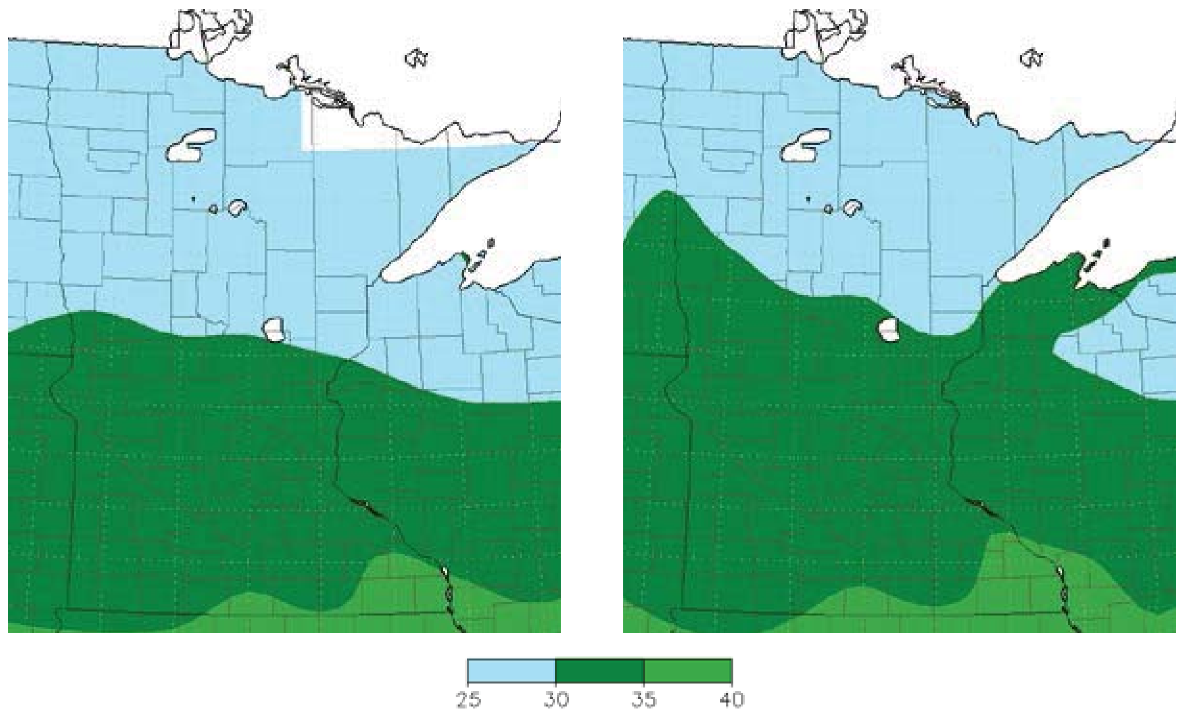
Top 10 Warmest Years for Minnesota, 1895-2013	
Year	Annual Average Temperature (°F)
2012	45.2
2010	42.9
2006	44.4
2005	43.1
2001	43.1
1999	43.7
1998	44.9
1987	45.3
1981	42.6
1931	45.0

Investigating climate change: What is a good baseline?

When analyzing climate anomalies, the National Oceanic and Atmospheric Administration (NOAA) uses the range 1901-1960 to represent baseline climate conditions when compared to current trends or conditions (Kunkel et al., 2013). NOAA explains that 1960 was selected as the end of the reference period because climate data display a pronounced acceleration of heating due to human influences after 1960. The current period 1960-2012, is used in this report to represent the effect of human activities associated with significantly increased greenhouse gas emissions. The increase of emissions is sometimes referred to as "radiative forcing" on the earth's climate.

In Minnesota, there are distinct daily, seasonal, and regional trends in temperature. Minimum temperatures, often referred to as "overnight lows", have increased at a faster rate than average daily temperatures taken as a whole. Since 1985, average annual overnight lows have been rising at a rate of 6.0°F per century (NOAA/NCDC, 2014). Over the last half century there has been a distinct spread of warmer lows into the northern part of Minnesota (Figure 4.4). There is no straightforward answer as to why nighttime lows are warming faster than daytime highs, but one likely factor is an increase in cloudiness that insulates land surface at night combined with reduced snowcover (Dai et al., 1999).

FIGURE 4.4. ANNUAL AVERAGE MINIMUM TEMPERATURE (°F) ACROSS MINNESOTA FOR 1900-1959 (LEFT) AND 1960-2013 (RIGHT). DATA AND IMAGE SOURCE: MRCC, 2014.



Visitors to Minnesota in January may find it hard to believe, but across the state winter season temperatures have been warming nearly twice as fast as annual average temperatures (Figure 4.5). The change in Minnesota's winter daily low temperatures drives the statewide trend in all-season annual temperatures (Zandlo, 2008). This trend has been indentified in Wisconsin (WICCI, 2011) and other Midwestern states as well (Figure 4.6).

FIGURE 4.5. MINNESOTA ANNUAL AVERAGE WINTER TEMPERATURES (DECEMBER – FEBRUARY). BLUE (LEFT) AND RED (RIGHT) LINES HIGHLIGHT TRENDS FOR 1895-1959 AND 1960-2013, RESPECTIVELY. DATA SOURCE: NOAA/NCDC, 2014.

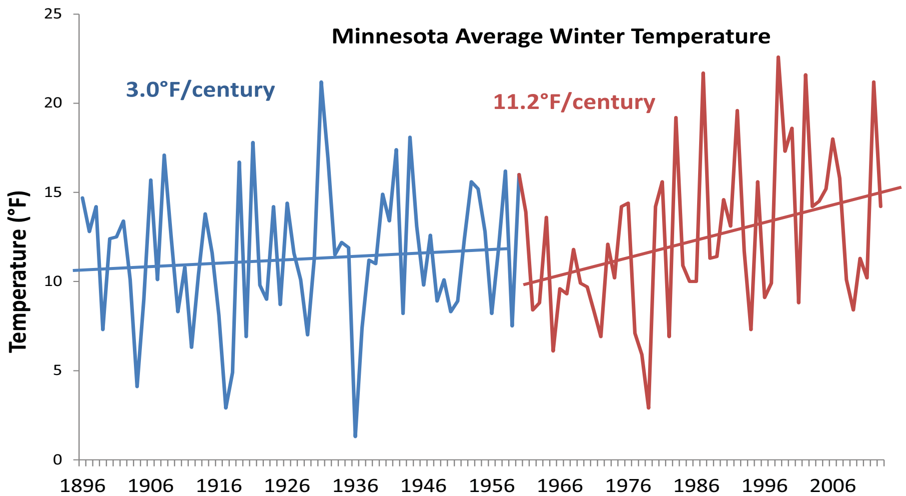
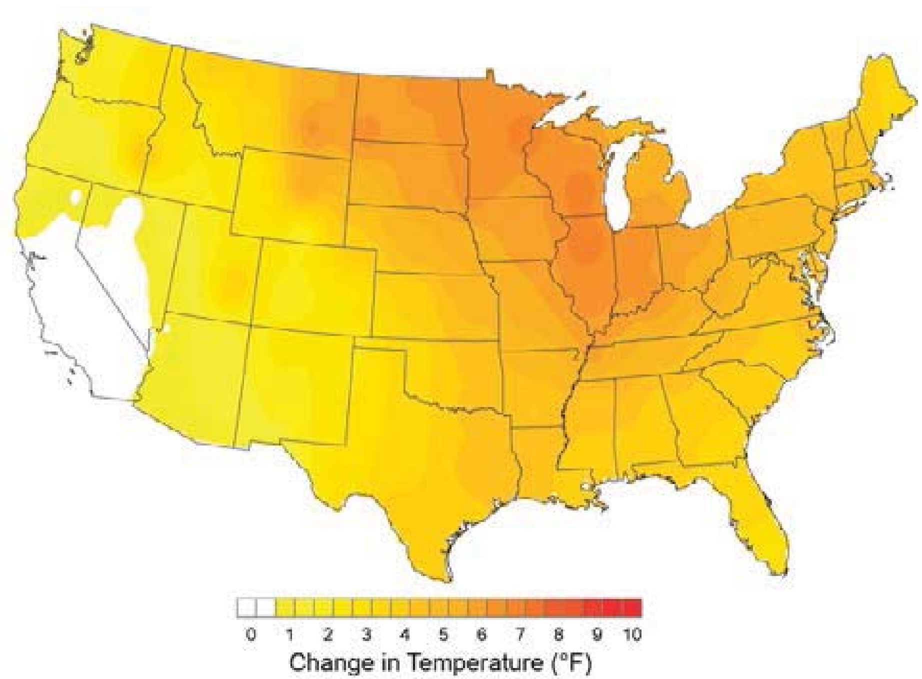


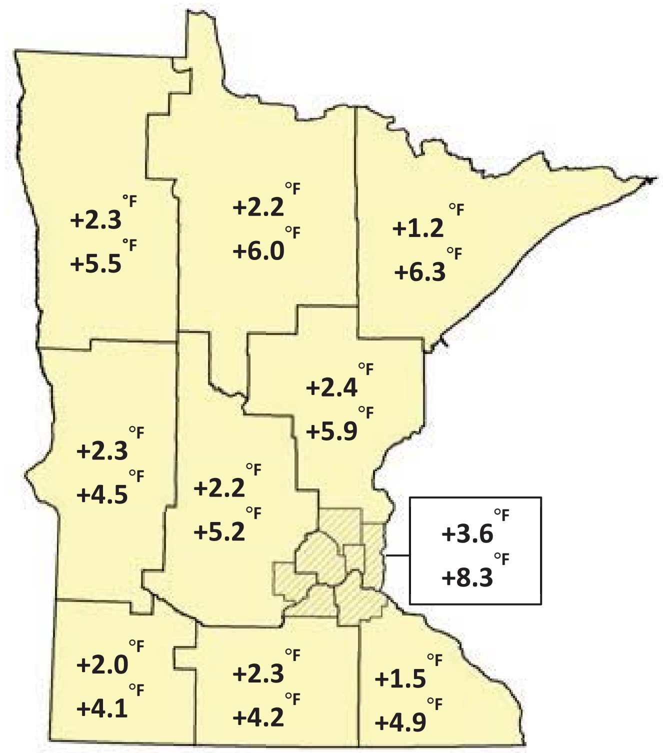
FIGURE 4.6. AVERAGE ANNUAL WINTER TEMPERATURES (°F) ACROSS THE U.S. FOR 1975-2007. TEMPERATURES ARE RISING FASTER IN WINTER THAN IN ANY OTHER SEASON, ESPECIALLY IN MINNESOTA AND THROUGHOUT THE MIDWEST REGION. FIGURE SOURCE: KARL ET AL., 2009.



Given the size and location of the state, it's not surprising that temperature varies within Minnesota's borders. The average annual temperature in the extreme northern area of the state is around 38°F compared to 47°F along the Mississippi River in the southeast (G. Spoden, personal communication, June 23, 2014). Areas adjacent to Lake Superior are affected by the moderating influence of this large body of water, in particular by summer season cooling.

Warming rates generally have been higher in northern areas of the state compared to southern areas, which is consistent with patterns seen across the Northern Hemisphere, where climate changes are occurring more rapidly at higher latitudes (IPCC, 2007). However, the Twin Cities area located in the southeastern part of the state is a notable exception. The highest calculated warming trend of recent years is associated with the Twin Cities Metropolitan Area (TCMA; Figure 4.7). The high rate of warming associated with the TCMA may be a result of the urban heat island effect (see inset on the next page). Trends for the TCMA, however, have greater uncertainty because less data are used compared to calculations of state or nationwide trends, which are based on much larger datasets. Regional measures are still very useful for informing the direction of ongoing investigations. For example, regional differences in climate trends may be strongly influenced by local land use changes (Zandlo, 2008).

FIGURE 4.7. COMPARISON OF AVERAGE TEMPERATURES (°F/CENTURY) FOR MINNESOTA CLIMATE DIVISIONS. THE "CLIMATE DIVISION" BOUNDARIES FOR EVERY STATE ARE ESTABLISHED BY NOAA TO ALLOW FOR LONG-TERM COMPARISONS ACROSS REGIONS. VALUES ARE BASED ON DATA FROM 1895-1959 (TOP) AND 1960-2013 (BOTTOM). THE TWIN CITIES METROPOLITAN AREA IS REPRESENTED BY THE SHADED AREA AND INCLUDES THE FOLLOWING SEVEN COUNTIES: ANOKA, CARVER, DAKOTA, HENNEPIN, RAMSEY, SCOTT, AND WASHINGTON. DATA SOURCE: MRCC, 2014.



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What is the difference between relative humidity and dew point?

Relative humidity is the ratio of water vapor in the air to the maximum amount of water vapor required for saturation at a particular temperature. The relative humidity indicates how close the air is to being saturated with water vapor. Dew point is an actual measure of moisture in the air. Dew point is the temperature at which the relative humidity reaches 100 percent. When relative humidity or dew point is high, sweat will not evaporate efficiently off the skin, compromising the body's primary cooling mechanism. Most people are comfortable with dew point temperatures up to 60°F. Above 70°F dew point is generally considered quite uncomfortable and above 75°F is oppressive.

Warmer air results in a greater water vapor capacity, so a rise in air temperature can lead to increases in dew point. Water vapor is considered a greenhouse gas, so an increase in the amount of water vapor in the atmosphere amplifies an initial rise in temperature. This is one of the strongest positive feedback loops in the climate system (Stocker et al., 2013). The number of days with high dew point temperatures may be increasing in Minnesota. Based on summer season data from the beginning of the 20th century to 2008, well over one-third of all record high dew point temperatures (measured from a Twin Cities location) were recorded in the last few decades (DNR, 2013). The average maximum dew point temperature for the Twin Cities summer season based on the entire data record is 74°F. Yet, in the last decade (2003-2013) annual maximum dew points have exceeded 74°F in 9 out of 10 years. The highest dew point temperature ever recorded in the Twin Cities (82°F) occurred in 2011, while the highest dew point ever recorded for the state (88°F) occurred on the same date in Moorhead.

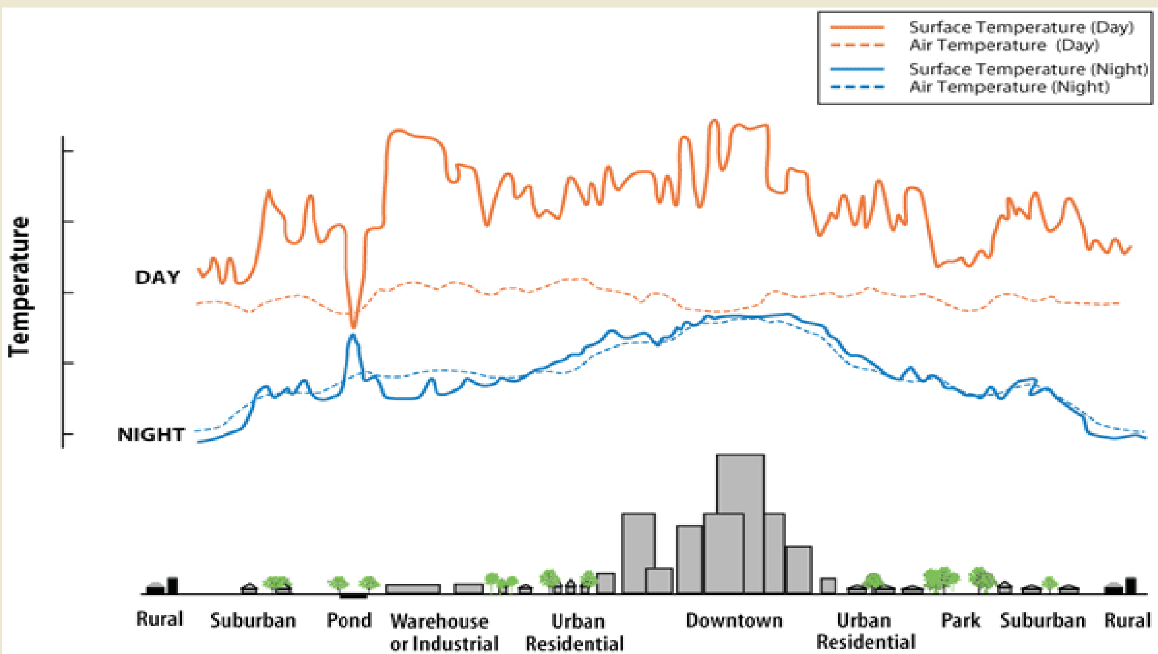

Figure source: EPA, 2013

What is the urban heat island effect?

The term "heat island" refers to built-up areas that are hotter than nearby less developed areas. More dry, impervious surfaces, tall buildings and reduced vegetation, lead to less shade and moisture as well as more heat absorption and retention in urban areas. The temperature difference between urban and rural areas can be in excess of 5°F during the daytime and as much as 22°F at night (EPA, n.d.). For more information on the urban heat island effect, see the University of Minnesota's Islands in the Sun website (islands.environment.umn.edu/).

PRECIPITATION

Patterns of precipitation in Minnesota are becoming more extreme. Based on national data from the past 50 years, the Midwest has experienced some of the greatest increases in heavy precipitation compared to other regions further west and south (Figure 4.8; Melillo, 2014). Trends based on available data for annual and summer precipitation are upward and statistically significant for the entire Midwest region. One estimate found that over the last century there was a 50 percent increase in the frequency of days with precipitation over four inches in the upper Midwest (Kunkel et al., 2008).

Minnesota in particular has gotten wetter in the past 50 years. For most of the first half of the 20th century (1895-1959), the trend in precipitation was slightly downward, at a loss of 0.15 inches per decade (1.5 inches per century), influenced by the Dust Bowl years of the 1930s. However, the rate of precipitation across the state has increased by nearly 0.35 inches per decade (3.5 inches per century) over the last half century, a 7% increase in annual average precipitation (Figure 4.9). The largest gains in precipitation appear to have occurred in the TCMA and southern regions of the state (Figure 4.10). For the TCMA, the trend went from a decline of 8.8 inches per century for 1895-1959 to an increase of 7.8 inches per century for 1960-2012, representing a change of 16.6 inches per century.

FIGURE 4.8. PERCENT CHANGE IN PRECIPITATION FALLING DURING VERY HEAVY EVENTS (DEFINED AS THE HEAVIEST 1% OF ALL DAILY EVENTS) FROM 1958 TO 2012. TRENDS ARE LARGER THAN NATURAL VARIATIONS FOR MANY REGIONS, PARTICULARLY THE NORTHEAST AND MIDWEST. FIGURE SOURCE: MELILLO ET AL., 2014.

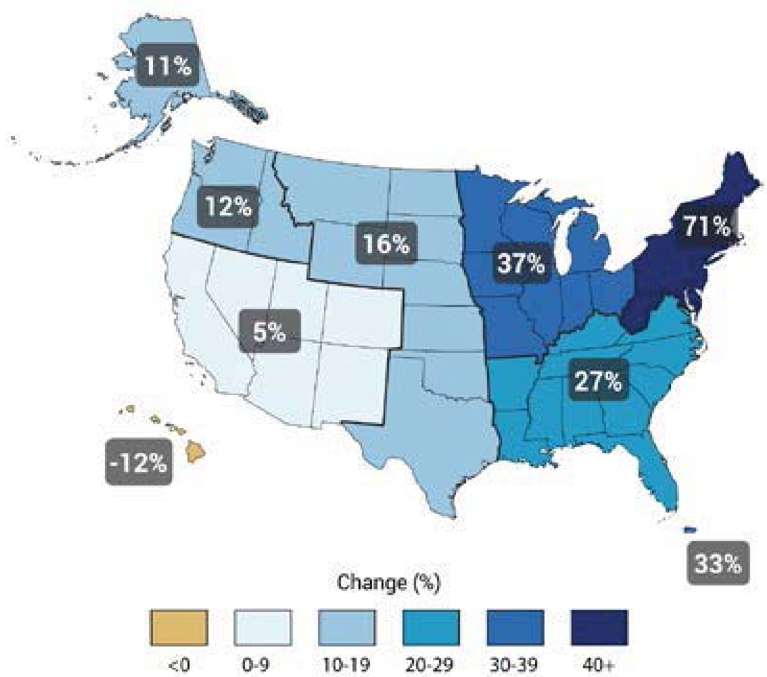

FIGURE 4.9. MINNESOTA ANNUAL AVERAGE PRECIPITATION (INCHES). BLUE (LEFT) AND RED (RIGHT) LINES HIGHLIGHT TRENDS FOR 1895-1959 AND 1960-2013, RESPECTIVELY. DATA SOURCE: NOAA/NCDC, 2014.

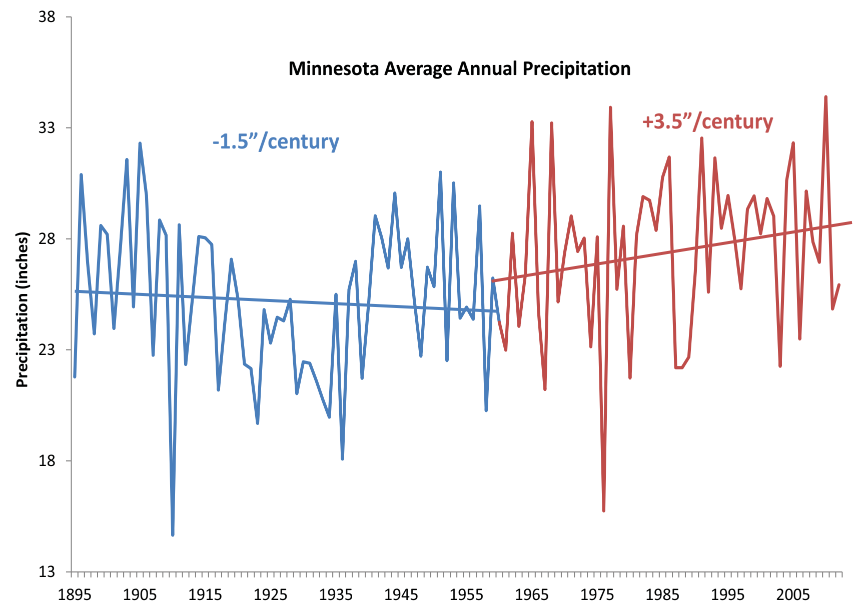

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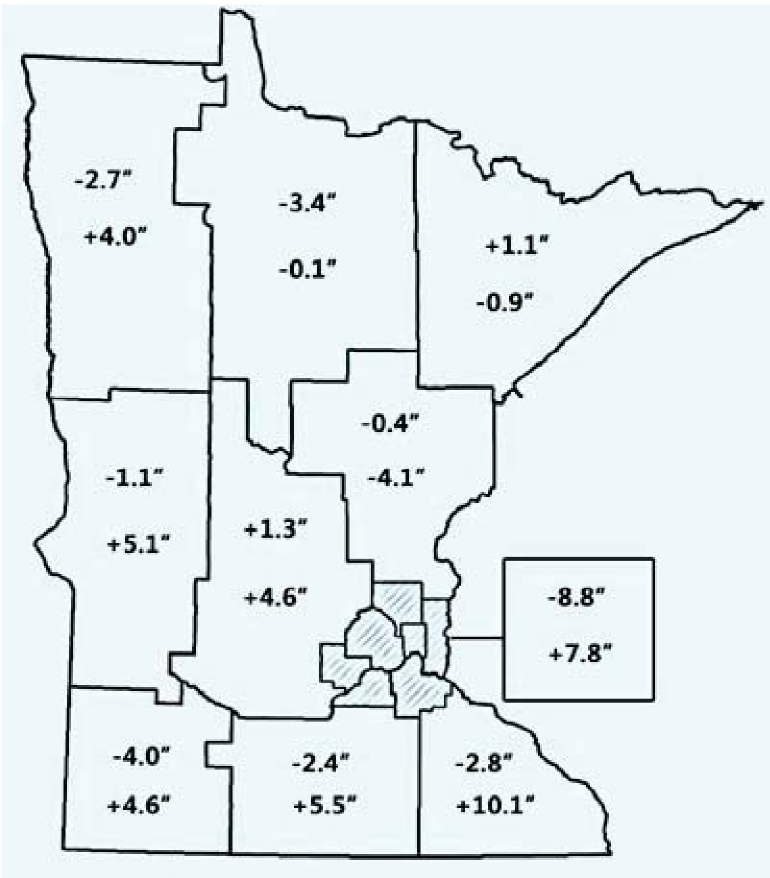
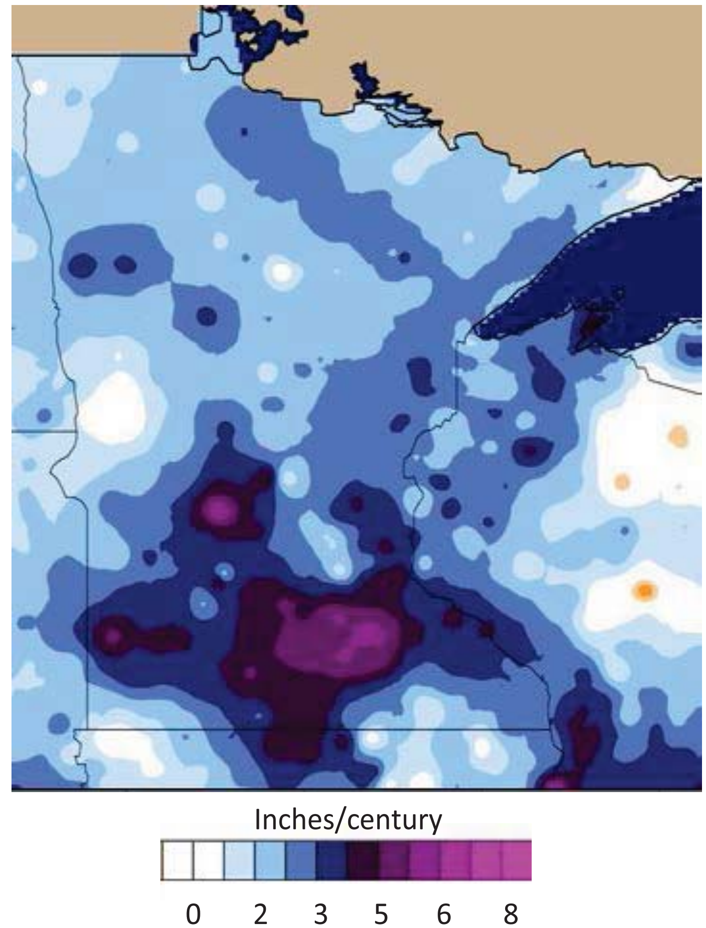
FIGURE 4.10. PRECIPITATION TRENDS FOR MINNESOTA. MAP ON THE LEFT SHOWS A COMPARISON OF AVERAGE PRECIPITATION (INCHES/CENTURY) FOR MINNESOTA CLIMATE DIVISIONS BASED ON DATA FROM 1895-1959 (TOP) AND 1960-2013 (BOTTOM). THE TWIN CITIES METROPOLITAN AREA (TCMA) IS REPRESENTED BY THE SHADED AREA. DATA SOURCE: MRCC, 2014. MAP ON THE RIGHT SHOWS THE DIFFERENCE BETWEEN AVERAGE ANNUAL PRECIPITATION TOTALS (INCHES) BETWEEN 1960-2012 AND 1895-1959. DATA AND IMAGE SOURCE: CLIMATE REANALYZER (CCI-REANALYZER.ORG), CLIMATE CHANGE INSTITUTE, UNIVERSITY OF MAINE.





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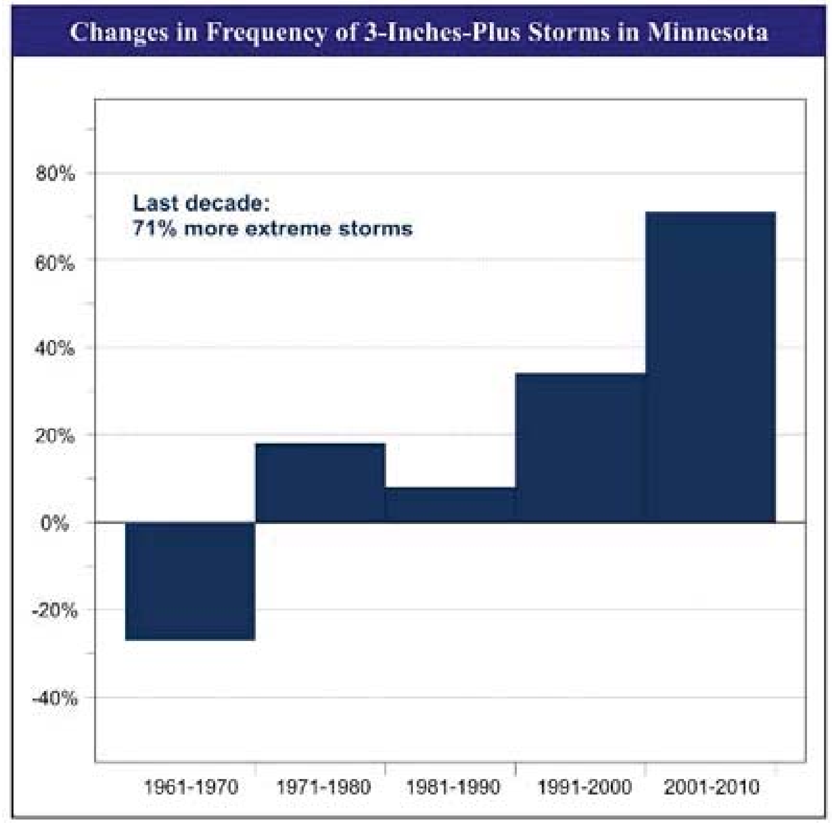
Given the economic reliance of the state on agriculture, the increase in precipitation could be a welcome occurrence in some areas. However, the pattern of precipitation has begun to favor extremes, which can be difficult to adjust for, not only within the agriculture sector but for industry and municipalities as well. Numerous measures demonstrate that the frequency and intensity of precipitation across the Midwest has increased, resulting in more storm events associated with flooding and longer intervening dry spells (Melillo et al., 2014). A 2012 analysis of Midwest storms by the Natural Resources Defense Council reports that the largest increases in extreme storm frequencies have been in states with relatively low average overall precipitation, including Minnesota (RMCO/NRDC, 2012). Drawing on data from the U.S. Historical Climatology Network, the analysis revealed that Minnesota experienced 71 percent more storms, discharging at least three inches of precipitation, in the decade 2001-2010 compared to previous data compiled for 1961-1990 (Figure 4.11).

FIGURE 4.11. CHANGES IN THE FREQUENCY OF STORMS WITH HEAVY PRECIPITATION. DATA FROM MINNESOTA WEATHER STATIONS INDICATE THAT THERE HAS BEEN A 71% INCREASE IN STORMS DISCHARGING 3 INCHES OR MORE RAINFALL WHEN COMPARING DECADES 2001-2010 TO 1961-1970. FIGURE SOURCE: RMCO/NRDC, 2012.



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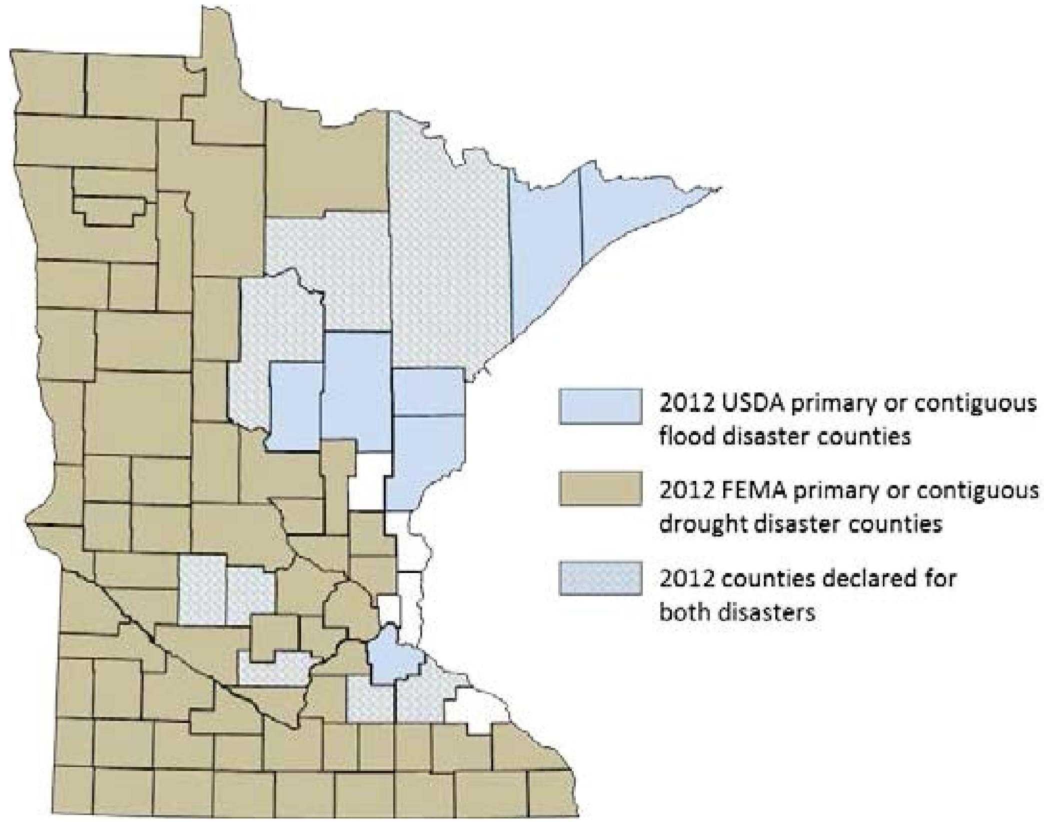
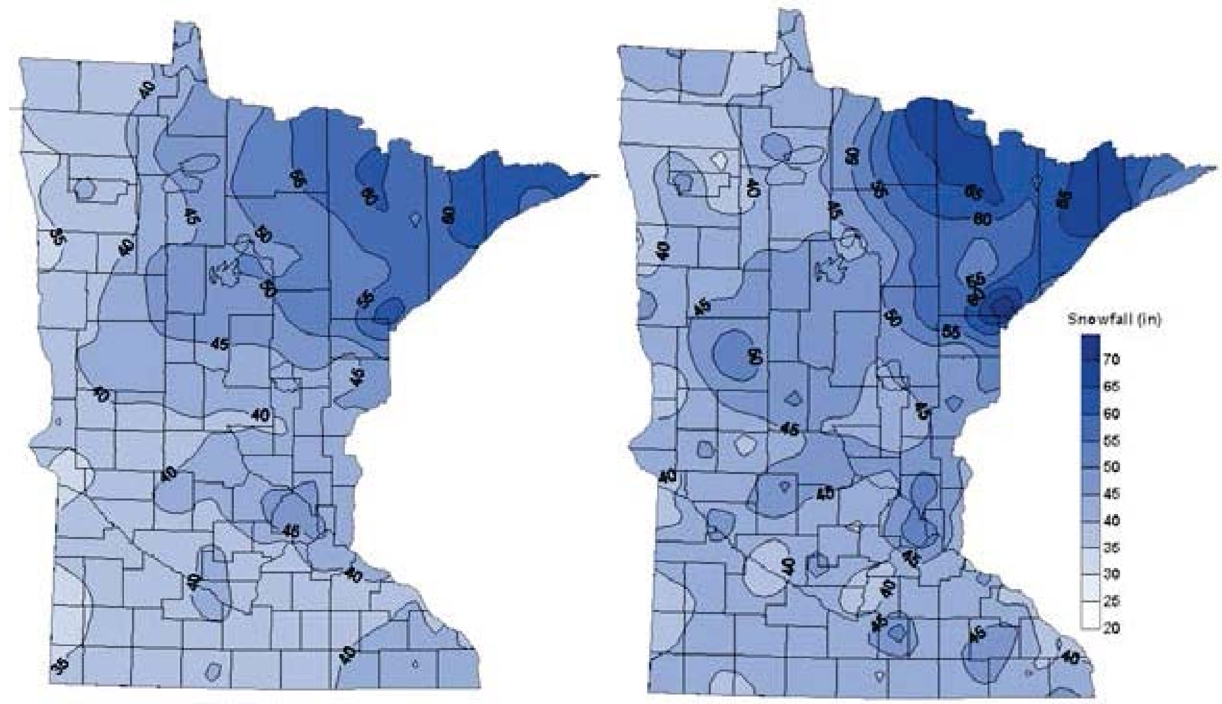
Recent years with long periods of little to no precipitation in certain areas have also raised concerns, especially amongst those in the agriculture sector, that Minnesota may be experiencing an increase in drought. In 2012, 75 counties were declared primary or contiguous disaster areas for drought (USDA, 2012). However, in that same year 15 counties and three tribal reservations were declared disaster areas for flooding (FEMA, 2012), with eight counties receiving disaster designation for both, underscoring the intensification of precipitation extremes in both directions (Figure 4.12). Based on available data for Minnesota from the U.S. Drought Monitor (2000-2013), the average percent area of the state that is classified as “abnormally dry” or drought-affected has increased over recent years, despite wide interannual variability (NDMC, 2014). However, such a small time frame cannot be considered representative of a larger drought trend for Minnesota. In addition, determining drought trends is challenging given that there are few direct measurements of drought-related variables (IPCC, 2013) and different definitions of what constitutes the occurrence of drought (Panu & Sharma, 2002). Current investigations of global drought trends suggest that drought may be decreasing in some areas in central North America (IPCC, 2013).

FIGURE 4.12. MINNESOTA 2012 DISASTER DECLARATIONS. COUNTIES IN BLUE WERE DECLARED DISASTER AREAS DUE TO FLOODING BY FEMA, WHILE COUNTIES IN BROWN WERE DECLARED PRIMARY OR CONTIGUOUS DISASTER AREAS DUE TO DROUGHT BY USDA. COUNTIES WITH CROSS-HATCHING RECEIVED BOTH DESIGNATIONS IN 2012. SOURCE DATA: USDA, 2012 AND FEMA, 2012.



While snow is an important part of Minnesota's hydrology the water found in snow contributes less than 20% of the total precipitation received annually (DNR, 2014). Generally, the average annual snowfall in Minnesota varies from 36 inches in the southwest to more than 70 inches along Lake Superior. The pattern of snowfall is driven by temperature and the supply of moisture. Regional analyses suggest that there has been an increase in snow storms in the Upper Midwest, although considerable decade-to-decade variations are present (Burnett et al., 2003). Data on average annual snow accumulation for Minnesota suggest a trend in greater snowfall spreading west across the state that has become more pronounced in the last few decades (Figure 4.13). However, there is some evidence that this trend will not continue into the future, due in part to rising winter temperatures.

FIGURE 4.13. AVERAGE ANNUAL SNOWFALL (INCHES) ACROSS MINNESOTA FOR YEARS 1890-2000 (LEFT) AND 1971-2000 (RIGHT). FIGURE SOURCE: SHULSKI & SEELEY, N.D.



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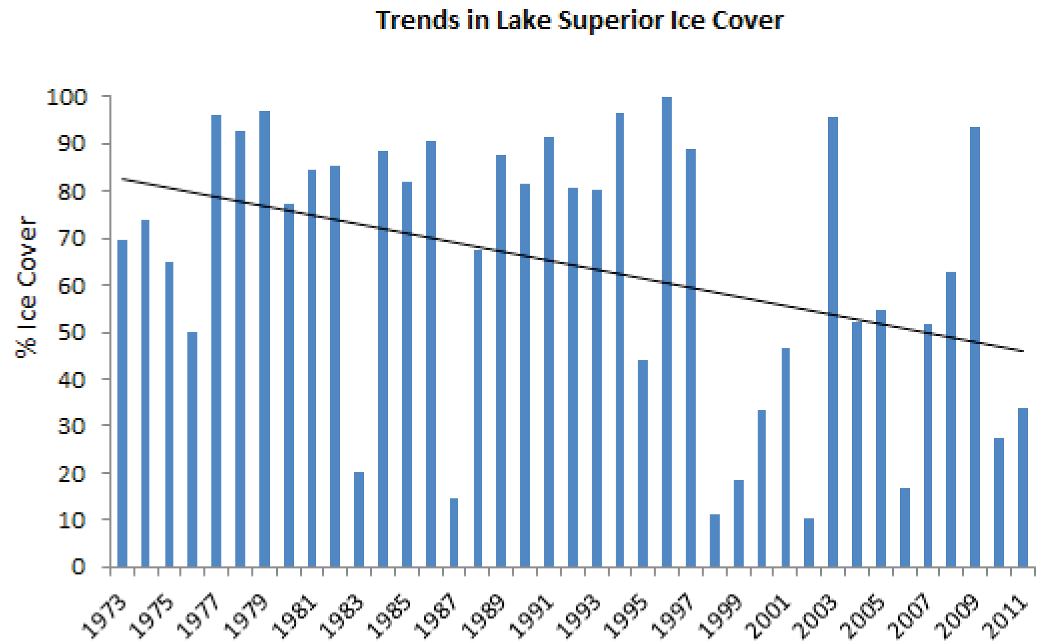
Rising winter temperatures may also be driving trends in ice cover. Lake Superior is the largest, deepest and coldest of the Great Lakes, yet total ice cover on the lake has shrunk by about 20 percent over the past 40 years (NOAA/GLERL, 2013). Changes to Lake Superior winter ice cover vary year to year but a diminishing trend is clear (Figure 4.14). Less ice cover leads to increases in evaporation and greater moisture in the air and contributes to heavier storm activity. A warming trend observed in Lake Superior surface temperature may be a sentinel indicator of increasing temperatures in other state surface waters that have already had noticeable effects on fish and other aquatic species (Austin & Colman, 2007; Huff & Thomas, 2014).

FIGURE 4.14. OBSERVED CHANGES IN LAKE SUPERIOR ICE COVER BASED ON SEASONAL MAXIMUM COVERAGE FOR YEARS 1973-2011. DESPITE LARGE INTER-ANNUAL VARIABILITY, THERE IS AN UNDERLYING DECREASING TREND IN THE EXTENT OF ICE COVER ON LAKE SUPERIOR. DATA SOURCE: NOAA/GLERL, 2013.



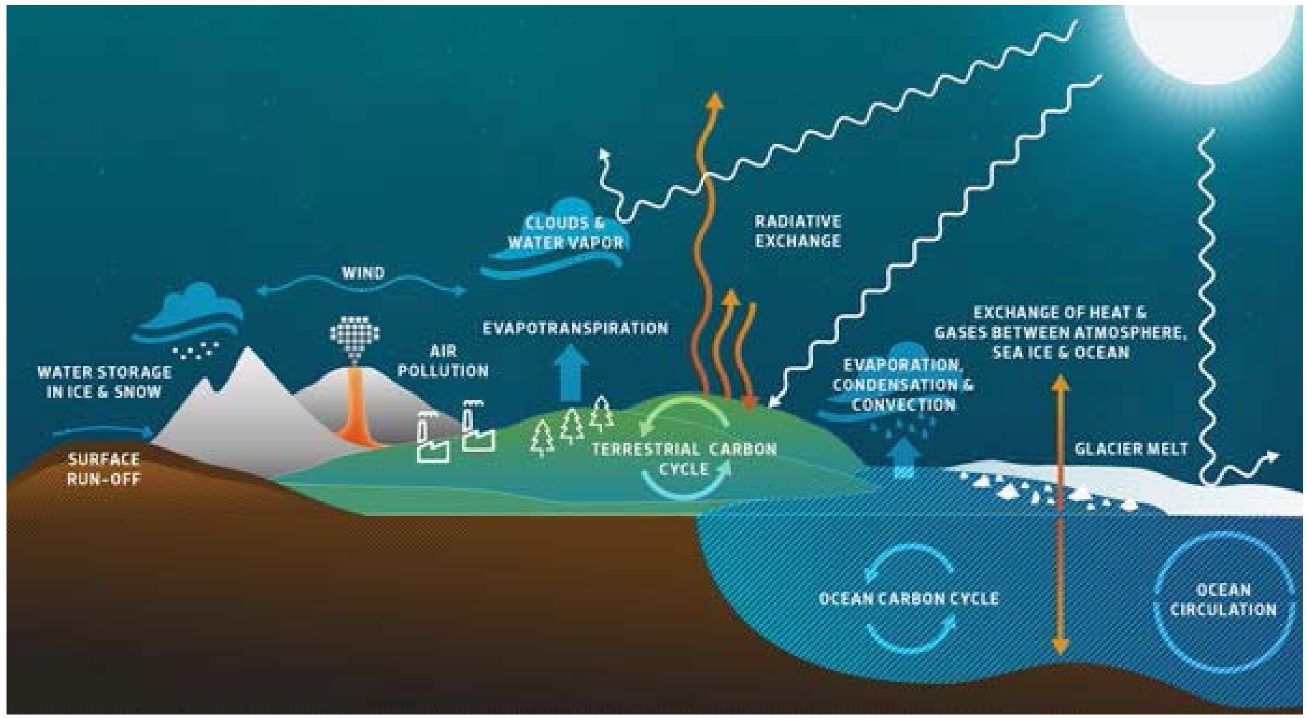
Future Projections

A climate projection is a statement about the likelihood that changes to the Earth's climate will happen sometime in the future (from several decades to centuries) given certain influential factors. Climate projections extending toward the end of the century use complex numerical models that account for changes in the flow of energy into and out of the Earth's climate system (Figure 4.15).

With any projection or modeling effort some uncertainty is unavoidable. For climate modeling, one of the greatest uncertainties relates to human behaviors that contribute to greenhouse gas emissions. To adjust for uncertainty, climate scientists use a range of "scenarios" to explore the consequences of various human decisions on climate. Each scenario includes different assumptions about population growth, economic activity, energy conservation, and land use, which lead to differences in projected annual greenhouse gas emissions (EPA, 2014).

The Intergovernmental Panel on Climate Change (IPCC) relies on a series of scenarios developed by the expert community for its research, and these have been described in the 2000 Special Report on Emissions Scenarios (SRES; Nakicenovic et al., 2000). The 2000 SRES scenarios cover from 1990 to 2100. There are 40 scenarios, grouped into four "families" (A1, A2, B1, B2), each with a storyline describing possible futures and combinations of driving forces (Figure 4.16). These scenarios are widely used as the basis for scientific studies and as a reference for political and societal discussions on climate change.

FIGURE 4.15. GLOBAL ENERGY FLOW. CLIMATE PROJECTIONS AND MODELS DRAW UPON MATHEMATICAL EQUATIONS BASED ON WIDELY ACCEPTED PRINCIPLES TO DEPICT THE BEHAVIOR AND INTERACTIONS OF CLIMATE AND EARTH PROCESSES. FIGURE SOURCE: EPA, 2014.

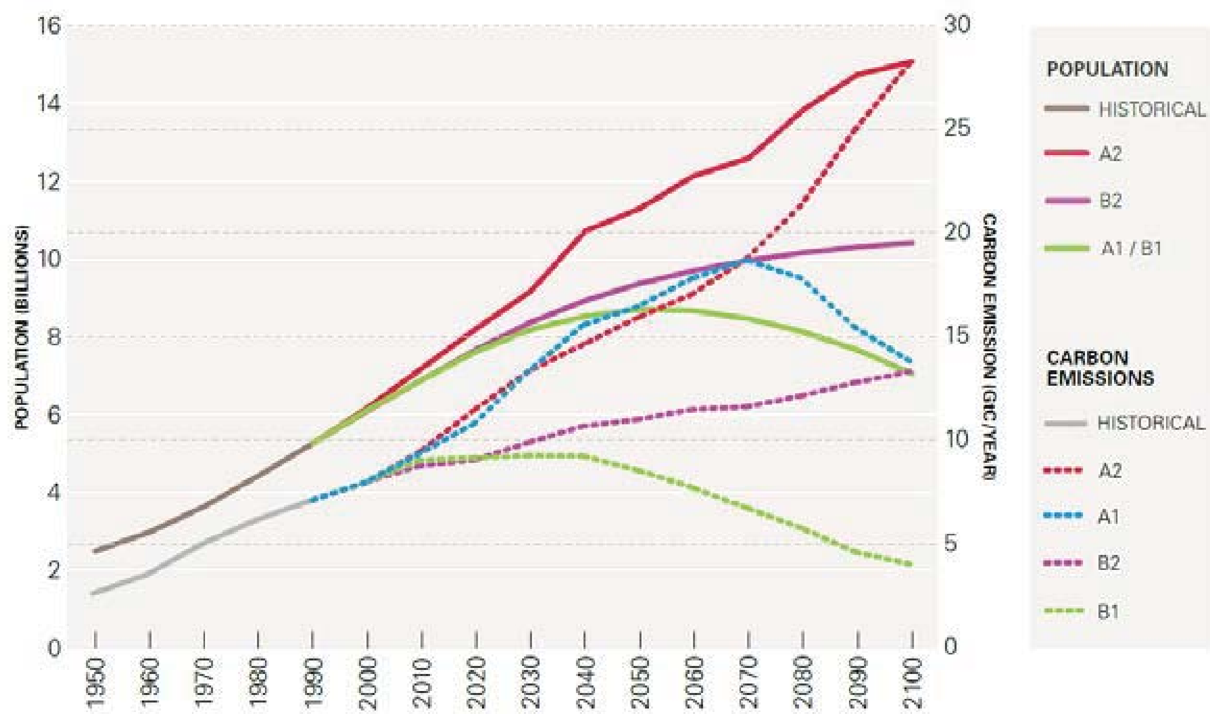


The A1 family of scenarios is characterized by low population growth and rapid economic growth. The A2 family assumes higher population growth, regional differences in development and slower, more fragmented economic growth. Emissions in the A2 family usually span the highest end of the SRES scenarios. The B1 family assumes a world of rapid economic change, with emphasis on information and service sectors, and extensive use of clean technologies and fuel sources. Emissions in the B1 family span the lowest end of the SRES scenarios. Finally, the B2 family illustrates a world that develops local solutions to energy needs, along with intermediate development, environmental protections, and modest technological change. However, global population continues to grow, although at a slower rate than A2 (Nakicenovic et al., 2000).

One of the most cited resources for U.S. region-specific climate impacts is the National Climate Assessment (NCA). The Global Change Research Act of 1990 mandates that national assessments of climate change be prepared not less frequently than every four years. The NCA report provides scientific information from multiple sources regarding climate change across the nation, establishes consistent methods for evaluating impacts, and informs national response priorities. The third edition of the NCA was released in May 2014 and is available on the GlobalChange.gov website (nca2014.globalchange.gov/).

Climate projections presented in the NCA are based on the SRES scenarios and include the A2 family (high emissions future) and B1 family (low emissions future). According to contributing authors of the Midwest section, the A2 and B1 emission scenarios were selected "because they incorporate much of the range of potential future impacts on the climate system and because there is a body of literature that uses climate and other scenarios based on them to evaluate potential impacts and adaptation options" (Kunkel et al., 2013). Under the A2 scenario, there is an acceleration in CO_2 concentrations, and by 2100 the estimated concentration is above 800 parts per million (ppm). Under the B1 scenario, the rate of increase gradually slows and concentrations level off at about 500 ppm by 2100. For reference, current atmospheric CO_2 concentration is already above 400 ppm.

FIGURE 4.16. POPULATION CHANGES AND CARBON EMISSIONS UNDER DIFFERENT SRES SCENARIOS COMPARED TO HISTORICAL TRENDS. FIGURE SOURCE: HOEPF YOUNG ET AL., 2009.



A number of figures on the following pages are taken from the third NCA report (Pryor et al., 2014). Other climate projection maps and figures relevant to Minnesota are provided to supplement the figures from the NCA report. (Information and data sources are provided in all figure captions.)

Some of this additional information was derived from the online data access tool, the Climate Reanalyzer (cci-reanalyzer.org/), which is produced by the Climate Change Institute at the University of Maine. The Climate Reanalyzer utilizes and provides access to existing climate datasets and models through a simple, user-friendly interface. The maps provided from the Climate Reanalyzer are based on the SRES A2 emission scenario.

Other figures were derived from the National Climate Change Viewer (NCCV) (www.usgs.gov/climate_landuse/clu_rd/nccv.asp), an online data access tool developed by the United States Geological Survey (USGS). The NCCV includes historical and future climate projections for two Representative Concentration Pathways (RCP; see inset). Figures included here represent the RCP emission scenario of 8.5 watts/m^2, which is indicative of a steep rise in emissions through the century.

What are Representative Concentration Pathways?

Global modeling efforts continue to evolve and improve. Recently, the IPCC has adopted four greenhouse gas concentration trajectories, called Representative Concentration Pathways (RCPs), which are increasingly used as inputs in climate modeling and research. The RCPs will form the basis of a new set of scenarios to replace the SRES 2000 scenarios in the IPCC Fifth Assessment Report that is being released in parts from 2013 through 2014. The four RCPs (RCP2.6, RCP4.5, RCP6, RCP8.5) represent a range of anthropogenic forcing values for the year 2100 relative to pre-industrial values. RCP categories are based on high and low forcing values found in the current scientific literature and are sufficiently separated (by about 2 watts/m^2) to provide distinguishable climate results (van Vuuren et al., 2011). Some recently released climate projections are already using RCP scenarios and results vary depending on the RCP scenario used. For example, RCP2.6 represents a “peak-and-decline” scenario where anthropogenic forcing peaks mid-century at 3.1 watts/m^2 and then returns to 2.6 watts/m2 by 2100, while RCP8.5 is characterized by greenhouse gas emissions that increase over time to values of 8.5 watts/m^2 or more.

TEMPERATURE

Most climate projections show that annual average temperature will continue to increase across the Midwest and Minnesota. Temperature increases in Minnesota may be more extreme compared to other Midwestern states, particularly in the northern part of the state where average annual temperatures may rise by well over 5°F compared to recent averages (Figure 4.17). By the end of the century, the rate of warming will be distinctly higher compared to the previous half century if emissions continue unabated. As seen with historical trends, warming rates, especially in higher latitudes, may be driven by increased winter season temperatures. However, average summer season temperatures are also expected to rise substantially, in part due to an increase frequency of extreme heat events (Figure 4.18). Climate projections suggest that western and southern Minnesota may experience 5-15 more days with a maximum temperature above 95°F by mid-century (Figure 4.19).

FIGURE 4.17. PROJECTED INCREASE IN ANNUAL AVERAGE TEMPERATURES ACROSS THE MIDWEST BY MID-CENTURY (2041-2070) AS COMPARED TO THE MORE RECENT 1971-2000 PERIOD. PROJECTIONS ARE FROM GLOBAL CLIMATE MODELS THAT ASSUME EMISSIONS OF GREENHOUSE GASES CONTINUE TO RISE (A2 SCENARIO). FIGURE SOURCE: PRYOR ET AL., 2014.

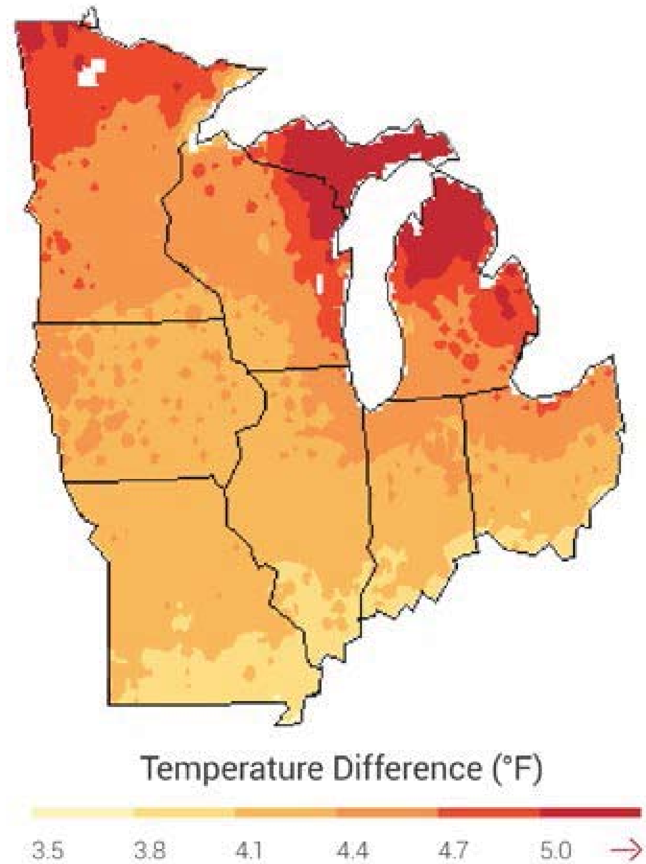


FIGURE 4.18. APPROXIMATE SEASONAL TEMPERATURES ACROSS MINNESOTA. MAPS ON THE TOP DISPLAY AVERAGE WINTER (DECEMBER-JANUARY) TEMPERATURES AND MAPS ON THE BOTTOM DISPLAY AVERAGE SUMMER (JUNE-AUGUST) TEMPERATURES. MAPS ON THE LEFT DISPLAY AVERAGE OBSERVED TEMPERATURES FOR 1870-1960 AND MAPS ON THE RIGHT DISPLAY AVERAGE PROJECTED TEMPERATURES FOR 2070-2099. MAPS ARE BASED ON THE A2 EMISSIONS SCENARIO. DATA AND IMAGE SOURCE: CLIMATE REANALYZER (HTTP://CCI-REANALYZER.ORG), CLIMATE CHANGE INSTITUTE, UNIVERSITY OF MAINE.

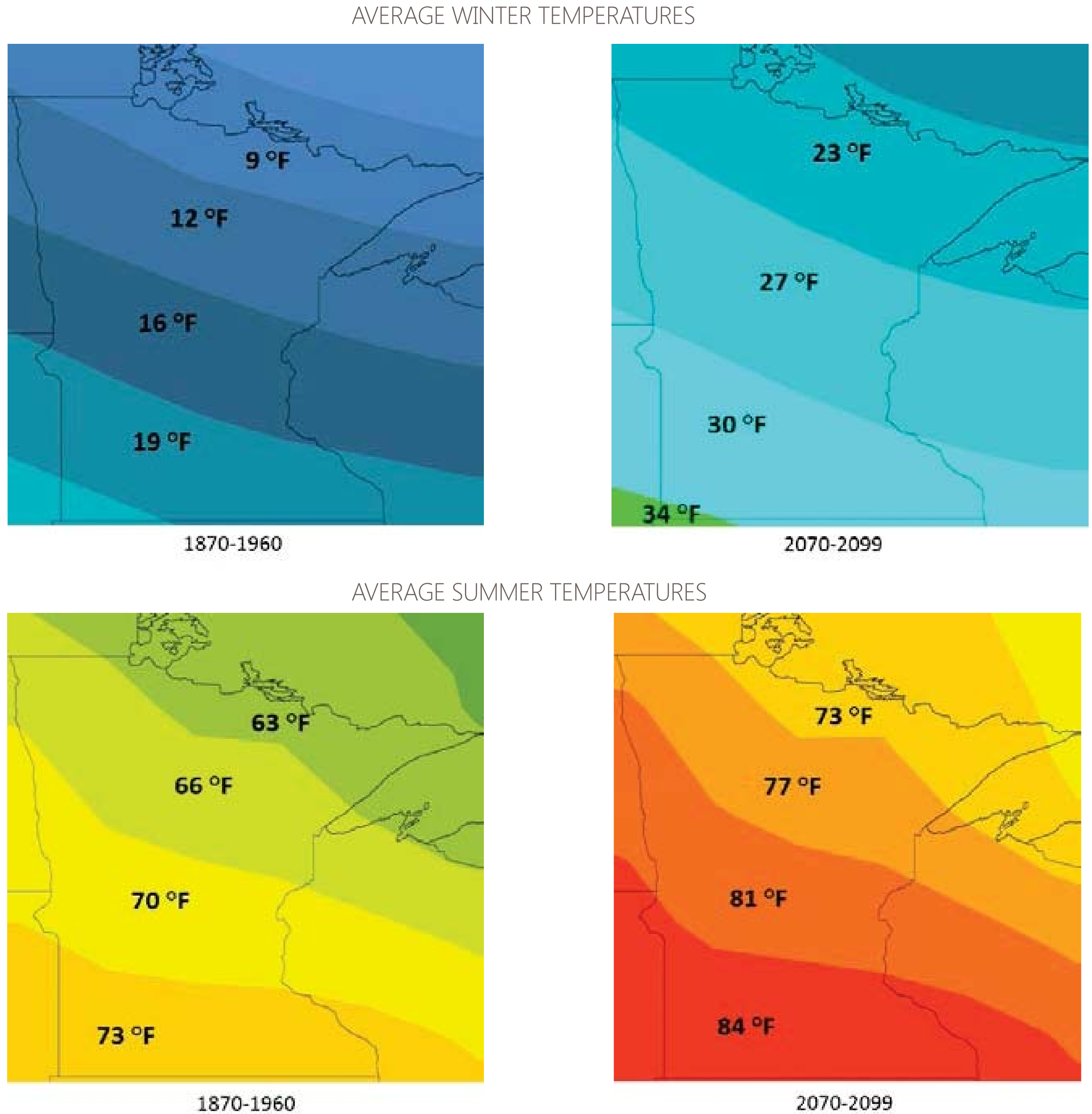
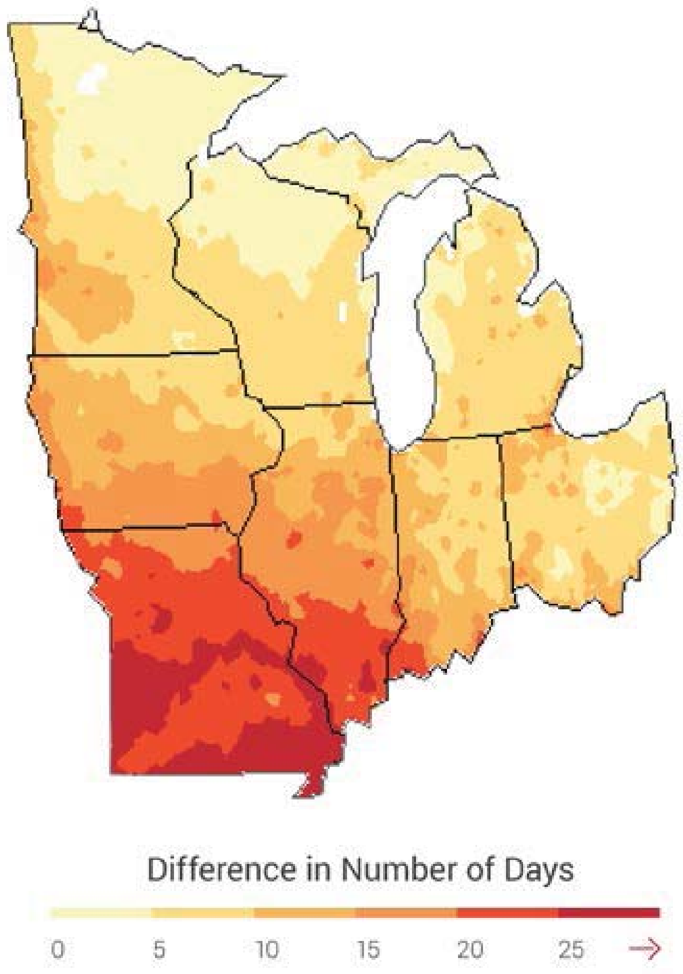


FIGURE 4.19. PROJECTED INCREASE IN THE NUMBER OF DAYS WITH A MAXIMUM TEMPERATURE ABOVE 95°F ACROSS THE MIDWEST BY MID-CENTURY (2041-2070) AS COMPARED TO THE MORE RECENT 1971-2000 PERIOD. PROJECTIONS ARE BASED ON THE A2 EMISSIONS SCENARIO. FIGURE SOURCE: PRYOR ET AL., 2014.



PRECIPITATION

Climate projections suggest that the total amount of precipitation will increase across the Midwest and Minnesota throughout the century. Certain parts of Minnesota, particularly in the central and southern areas may gain an additional three inches or more of annual precipitation (Figure 4.20). Yet, there will likely be some distinct seasonal variations. Some models predict that under a high emissions scenario, Minnesota may experience a decrease in summer season precipitation; however, there is a notable amount of uncertainty surrounding these projections (Figure 4.21; Pryor et al., 2014; Winkler et al., 2012). Historical trends of increased precipitation during the past century were mainly due to the intensification of the heaviest rainfall events, and this tendency towards precipitation extremes is projected to continue in the future (Pryor et al., 2014). In addition, projections indicate that intervening dry periods will become longer, a variable that has been used to indicate an increase in the chance of drought (Figure 4.22).

FIGURE 4.20. PROJECTED CHANGES IN TOTAL ANNUAL AVERAGE PRECIPITATION (INCHES) FOR THE MIDDLE OF THE CURRENT CENTURY (2041-2070) RELATIVE TO THE END OF THE LAST CENTURY (1971-2000) ACROSS THE MIDWEST UNDER A HIGH EMISSIONS (A2) SCENARIO. FIGURE SOURCE: PRYOR ET AL., 2014.

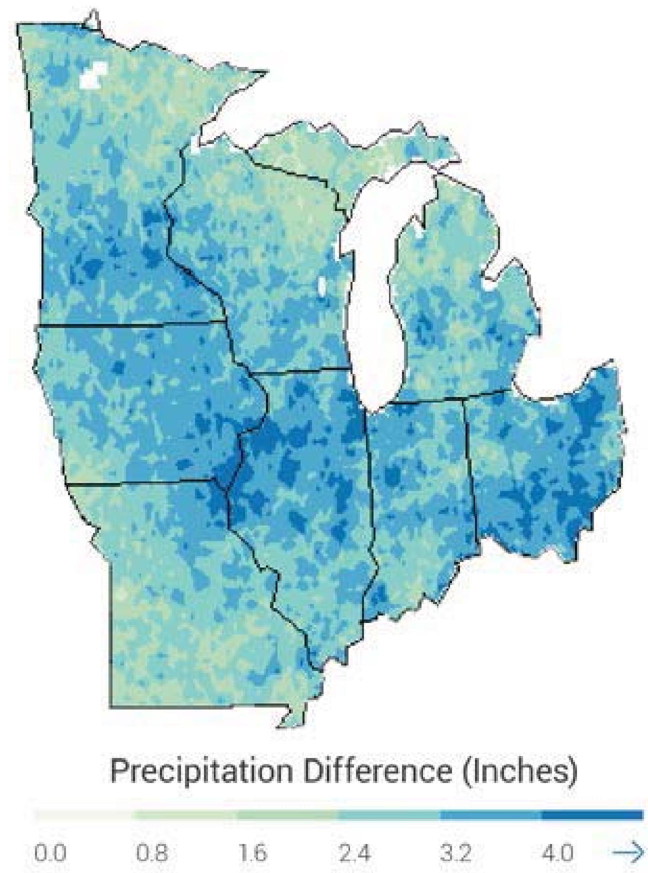


FIGURE 4.21. MONTHLY AVERAGES OF PRECIPITATION (INCHES) FOR MINNESOTA BASED ON RCP8.5 EMISSION SCENARIO OVER FOUR TIME PERIODS. EACH SOLID LINE REPRESENTS AN AVERAGE OF 30 DIFFERENT CLIMATE PROJECTION MODELS WHILE SHADED AREAS AROUND EACH LINE REPRESENT STANDARD DEVIATIONS. FIGURE SOURCE: NCCV (HTTP://WWW.USGS.GOV/CLIMATE_LANDUSE/CLU_RD/NCCV.ASP), USGS, USA.

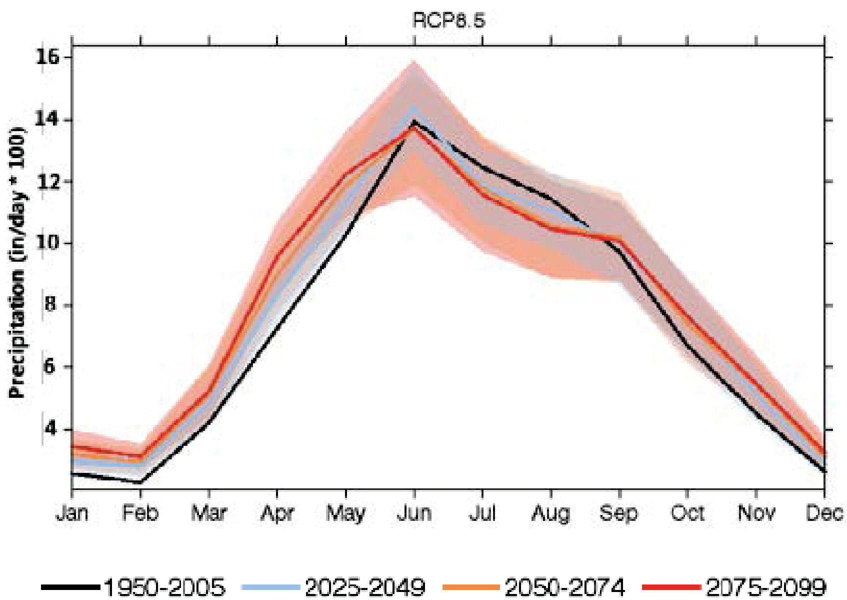
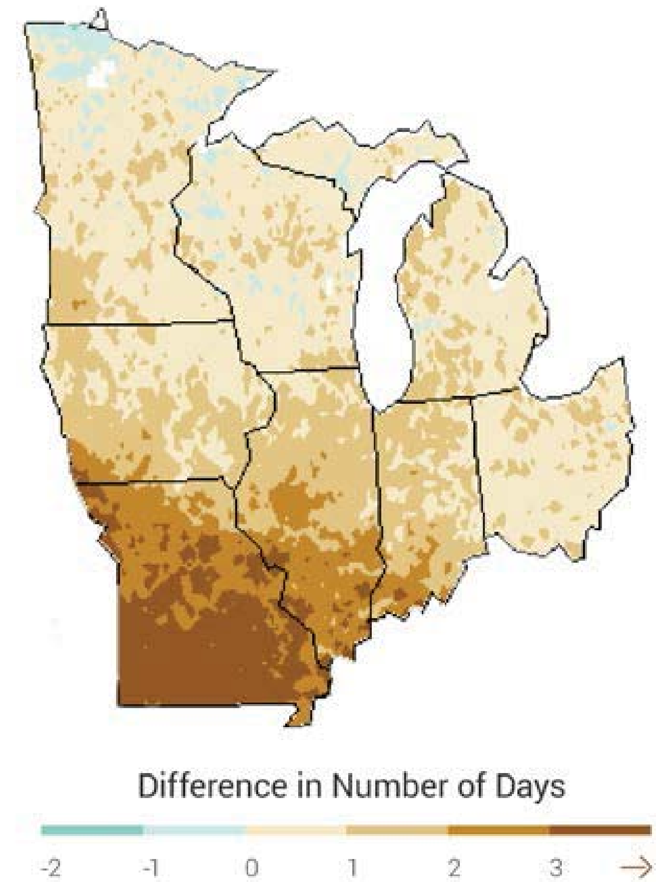


FIGURE 4.22. CHANGE IN AVERAGE MAXIMUM NUMBER OF CONSECUTIVE DAYS EACH YEAR WITH LESS THAN 0.01 INCHES OF PRECIPITATION FOR THE MIDDLE OF THE CURRENT CENTURY (2041-2070) RELATIVE TO THE END OF THE LAST CENTURY (1971-2000) ACROSS THE MIDWEST UNDER A HIGH EMISSIONS (A2) SCENARIO. AN INCREASE IN THIS VARIABLE HAS BEEN USED TO INDICATE AN INCREASE IN THE CHANCE OF DROUGHT IN THE FUTURE. FIGURE SOURCE: PRYOR ET AL., 2014.



The amount of precipitation occurring as snowfall in the winter is projected to decrease, with a larger proportion falling as rain, in part due to warming temperatures (Figure 4.23). Snowpack strongly influences seasonal runoff. Less snowpack coupled with earlier snow melt, is expected to change the timing and magnitude of surface runoff with implications for future flooding, which may occur earlier in the spring season (Figure 4.24).

FIGURE 4.23. MONTHLY AVERAGES OF SNOW WATER (INCHES) FOR MINNESOTA BASED ON RCP8.5 EMISSION SCENARIO OVER FOUR TIME PERIODS. EACH SOLID LINE REPRESENTS AN AVERAGE OF 30 DIFFERENT CLIMATE PROJECTION MODELS WHILE SHADED AREAS AROUND EACH LINE REPRESENT STANDARD DEVIATIONS. FIGURE SOURCE: NCCV (HTTP://WWW.USGS.GOV/CLIMATE_LANDUSE/CLU_RD/NCCV.ASP), USGS, USA.

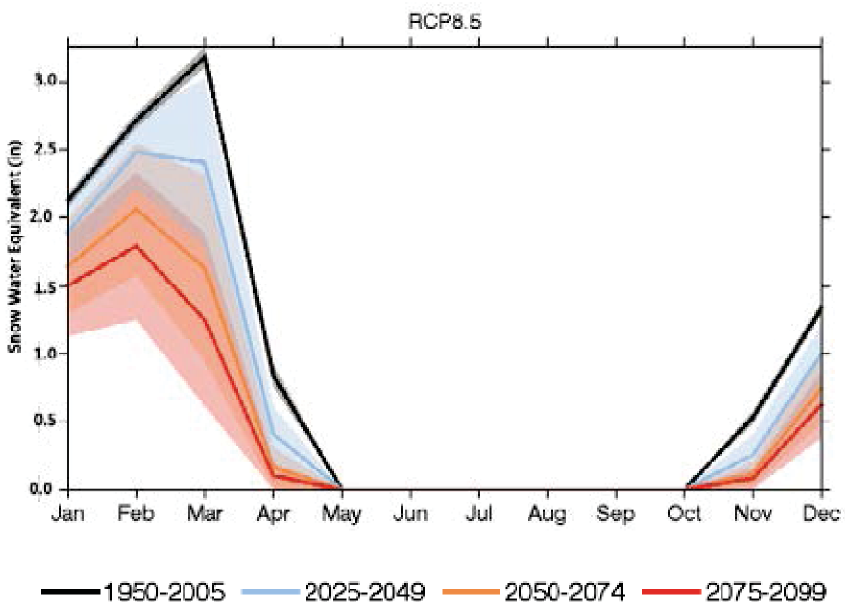
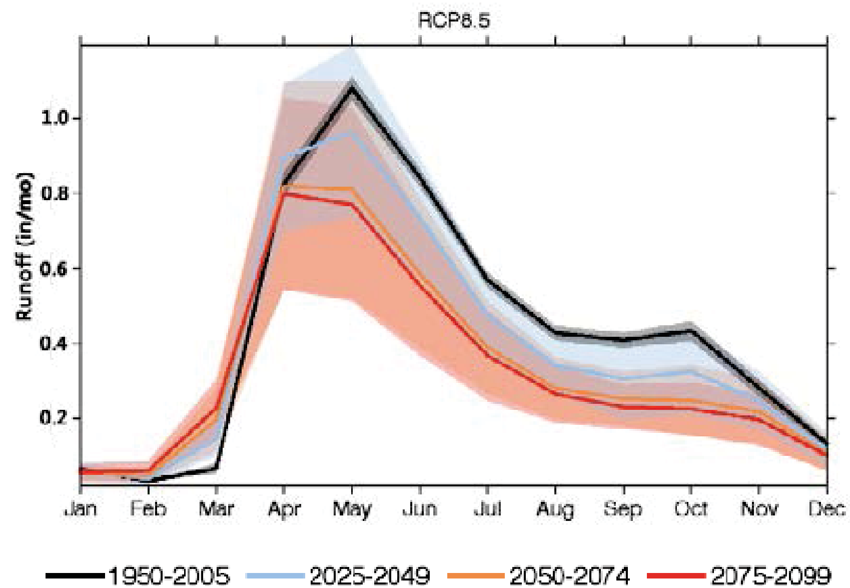


FIGURE 4.24. MONTHLY AVERAGES OF RUNOFF (INCHES PER MONTH) FOR MINNESOTA BASED ON RCP8.5 EMISSION SCENARIO OVER FOUR TIME PERIODS. RUNOFF IS DEFINED AS THE SUM OF DIRECT RUNOFF THAT OCCURS FROM PRECIPITATION AND SNOW MELT AND SURPLUS RUNOFF WHICH OCCURS WHEN SOIL MOISTURE IS AT 100% CAPACITY. EACH SOLID LINE REPRESENTS AN AVERAGE OF 30 DIFFERENT CLIMATE PROJECTION MODELS WHILE SHADED AREAS AROUND EACH LINE REPRESENT STANDARD DEVIATIONS. FIGURE SOURCE: NCCV (HTTP://WWW.USGS.GOV/CLIMATE_LANDUSE/CLU_RD/NCCV.ASP), USGS, USA.



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Projections of temperature and precipitation shifts suggest future climate changes are likely to have substantial impacts on Minnesota's ecosystems and agriculture. Plant hardiness zones provide a standard by which growers can determine which plants are most likely to thrive at a certain location. Plant hardiness zones in the Midwest have already changed significantly (Kart et al., 2009). In the future, hardiness zones for the winter season are projected to shift one-half to one full zone every 30 years across the Midwest (Figure 4.25). This may have repercussions for crop yields and viability of both native and invasive species. By the end of the 21st century, plants now associated with the Southeastern U.S. may become established throughout the Midwest, while some native plant and tree species may disappear altogether. For example, the range size of many exemplar tree species for Minnesota, such as paper birch, quaking aspen, balsam fir and black spruce, are projected to decline substantially across the northern Midwest, while species that are common farther south, like some oaks and pines, will move northward into the region (Figure 4.26).

FIGURE 4.25. OBSERVED AND PROJECTED CHANGES IN PLANT HARDINESS ZONES. NORTHWARD SHIFTS IN PLANT HARDINESS ZONES HAVE ALREADY OCCURRED ACROSS THE MIDWEST AND ARE PROJECTED TO CONTINUE. EACH ZONE REPRESENTS A 10°F RANGE IN THE LOWEST TEMPERATURE OF THE YEAR, WITH ZONE 3 REPRESENTING -40 TO -30°F AND ZONE 8 REPRESENTING 10 TO 20°F. FIGURE SOURCE: KARL ET AL., 2009.

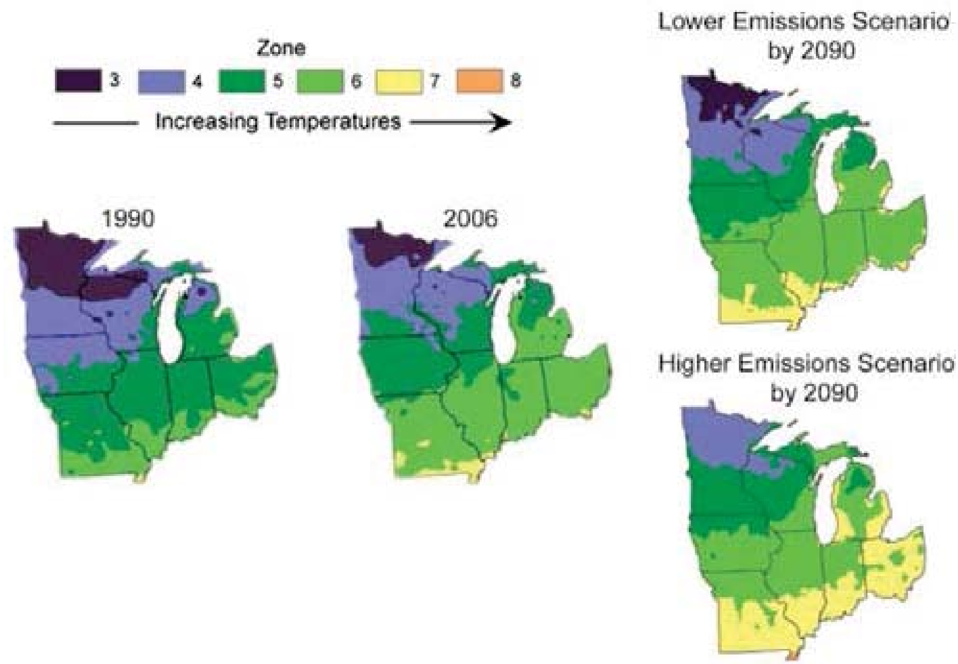
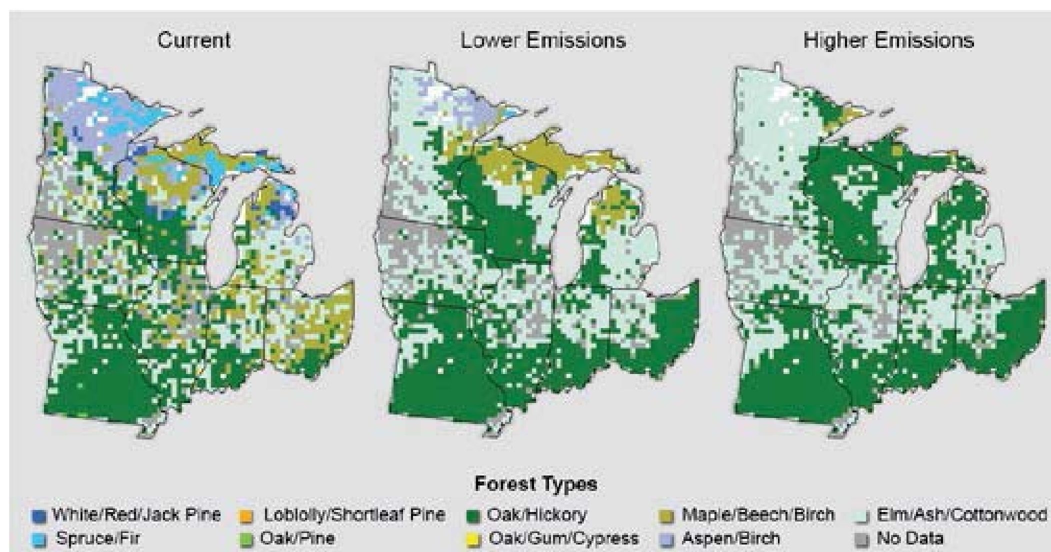


FIGURE 4.26. CURRENT AND PROJECTED DISTRIBUTION OF FOREST TYPES IN THE MIDWEST. PROJECTED FUTURE DISTRIBUTION MAPS ARE BASED ON SRES LOWER AND HIGHER GREENHOUSE GAS EMISSIONS SCENARIOS. FIGURE SOURCE: PRASAD ET AL., 2007.



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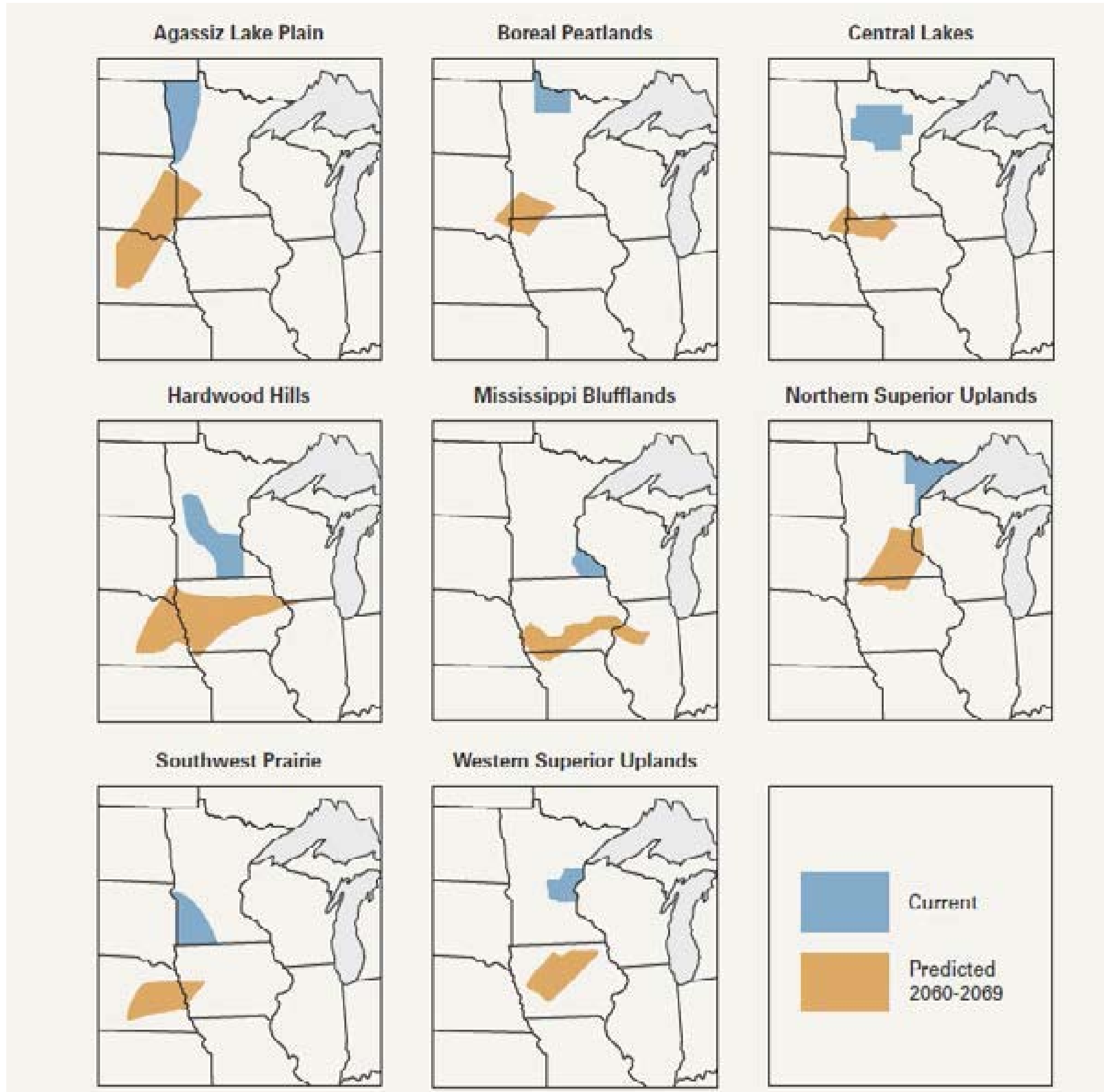
Studies by University of Minnesota researchers offer another perspective on the potential for substantial changes to Minnesota's ecosystems and natural resources. Galatowitsch and colleagues (2009) used climate projections to assess impacts for eight "landscape" regions in Minnesota with results indicating that each of these regions may be replaced by climate areas currently located 400-500 kilometers to the south or southwest (Figure 4.27). For example, projections suggest that by mid-century the current climate of Minnesota northern peatlands will be replaced by climate conditions representative of the far southwestern border of the state. Boreal peatlands are a unique and valuable ecosystem for the state and region given that peatlands effectively store carbon: one-third of the world's soil carbon pool is sequestered in peatlands. Mounting evidence suggests that warming trends are already disrupting the ability of Minnesota peatlands to capture and store carbon, which may potentially lead to larger releases of sequestered carbon to the atmosphere (Keller et al., 2004).

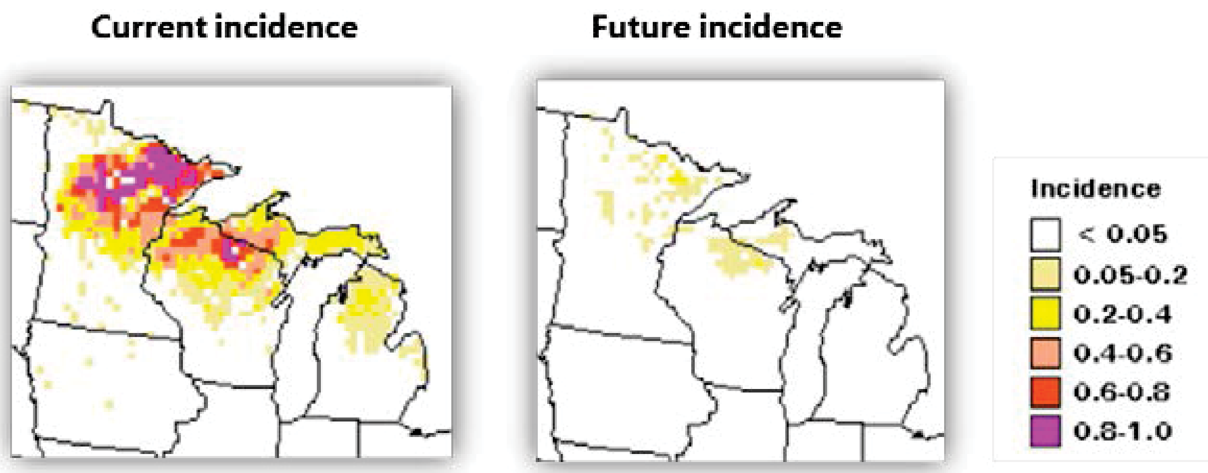
FIGURE 4.27. FUTURE CLIMATE ANALOGS FOR SELECT MINNESOTA LANDSCAPES. EIGHT UNIQUE ECOSYSTEMS IN MINNESOTA ARE IDENTIFIED IN BLUE. ASSOCIATED BROWN COLORED AREAS TO THE SOUTH REPRESENT CURRENT CLIMATES MOST RESEMBLING WHAT THE BLUE AREAS WILL BECOME BY MID-CENTURY (2060-2069) BASED ON HIGH EMISSIONS CLIMATE CHANGE PROJECTIONS. FIGURE BASED ON RESEARCH BY GALATOWITSCH ET AL., 2009. FIGURE SOURCE: DNR, 2011.



All major groups of animals, including birds, mammals, amphibians, reptiles, and insects, are likely to be affected by future impacts to Minnesota's ecosystems. Die-offs of moose and common loons, iconic animals for Minnesota, have already been documented, and projections suggest serious population declines in the future (Figure 4.28; Gardner, 2013; Robbins, 2013).

Climate projections also indicate substantial impacts to one of Minnesota's most prized ecosystems, Lake Superior. Minnesota's Great Lake contains nine percent of the world's surface fresh water and is considered the most pristine of the Great Lakes. Yet it faces a range of ongoing challenges—spread of invasive species, declining resident species, legacy and emerging pollutants, land development and hydropower dams—including climate change stressors. Climate stressors include higher water temperatures, falling lake levels (Figure 4.29) and reduced snow fall.

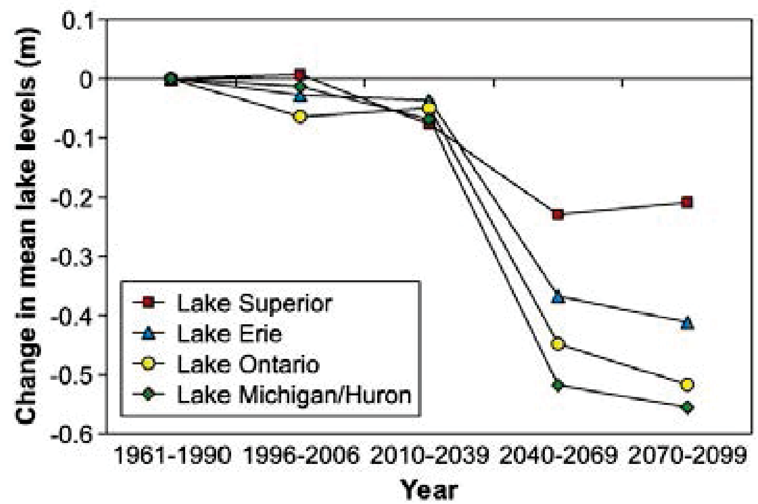
FIGURE 4.28. POTENTIAL FUTURE CHANGES IN THE INCIDENCE OF THE COMMON LOON. CURRENT INCIDENCE IS SHOWN ON THE LEFT AND PROJECTED INCIDENCE FOR 2099 UNDER A HIGH EMISSIONS SCENARIO IS SHOWN ON THE RIGHT. FIGURE SOURCE: HUFF & THOMAS, 2014.



In January 2014, the Lake Superior Binational Program (LSBP) released the comprehensive report, *Lake Superior Climate Change Impacts and Adaptation*, which details the following future climate impacts for the Lake Superior ecosystem (Huff & Thomas, 2014):

- While annual precipitation in the Lake Superior basin may only increase slightly by the end of the century (5 to 15 percent), more winter precipitation will fall as rain and less as snow.
- Ice cover will continue to decrease throughout the 21st century. Average February ice cover is expected to be only 2 to 11 percent for parts of the basin by 2090.
- In addition, the duration of ice cover will continue to decrease, perhaps by as much as 1 to 2 months by 2100.
- Water levels may decrease, beginning mid-century, on the order of 1.2 to 8.4 inches.
- Spring and summer are expected to begin earlier and the growing season to last longer in the Lake Superior basin through this century.
- The length of the frost-free season in the Midwest, including the southern Lake Superior basin, may increase by an additional 4 to 8 weeks through the end of this century. In addition, the last spring frost may arrive earlier by as much as 15 to 35 days and the last autumn frost may be delayed up to 35 days.

FIGURE 4.29. PROJECTED CHANGES IN ANNUAL AVERAGE WATER LEVELS (METERS) FOR THE GREAT LAKES, RELATIVE TO 1961-1990 AVERAGE VALUES, BASED ON A MODERATE EMISSIONS SCENARIO (SRES A1B). FIGURE SOURCE: HUFF & THOMAS, 2014, COURTESY OF HAYHOE ET AL., 2010.



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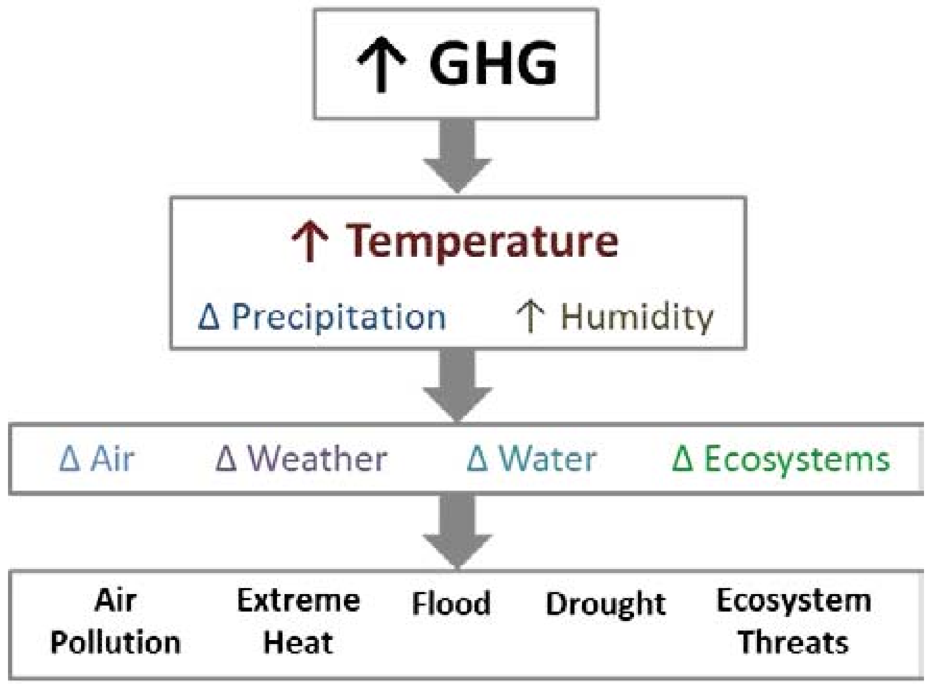
V Climate Hazards & Exposure Pathways

Recognizing the need for identifying research goals and gaps relating to health effects of climate change, leaders from numerous federal agencies, universities and institutes convened the Interagency Working Group on Climate Change and Health (IWGCCH) to develop the white paper, *A Human Health Perspective on Climate Change* (Portier, et al., 2010). This influential report provides a framework for public health researchers and decision-makers to organize their collective understanding of the wide range of health effects related to climate change and the most efficient approaches to climate change adaptation. The logic model presented here (Figure 5.1.), which summarizes the organization and priorities of the *Profile Report*, was based in part on a similar model from the IWGCCH paper and adapted for relevancy to Minnesota.

Increases in greenhouse gas (GHG) emissions, particularly carbon dioxide, are leading to dramatic increases in ambient temperatures, which in turn are leading to extremes in precipitation and humidity. All of these atmospheric influences (GHG emissions, temperature, precipitation, humidity) are directly or indirectly causing disruptions in four key aspects of the human environment—air, weather, water, and ecosystems. Changes in these areas are in turn leading to situations that threaten the health and vitality of human communities. For example, GHG emissions directly contribute to air pollution and inhalation exposures that are associated with poor health outcomes. GHG emissions also cause higher ambient temperatures which facilitate chemical reactions that lead to toxic by-products, like ground-level ozone, which can aggravate certain conditions like asthma. High ozone alert days reduce worker productivity, especially for outdoor occupations, and stifle outdoor recreation activities, which foster physical activity and community cohesion.

FIGURE 5.1. LINKS BETWEEN THE RISE IN ATMOSPHERIC GREENHOUSE GASES, CHANGES TO THE EARTH'S CLIMATE, IMPACTS ON KEY ASPECTS OF THE ENVIRONMENT, AND CLIMATE HAZARDS RELEVANT TO THE HEALTH OF MINNESOTANS.



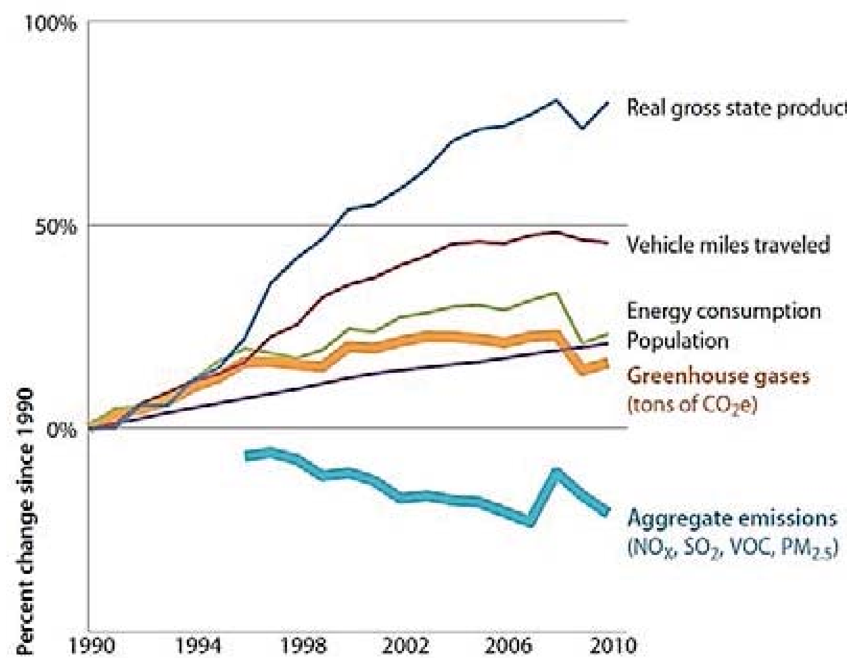
The following sections of the *Profile Report* will expand on five specific climate hazards and describe how they are likely to affect the health of Minnesotans, i.e., air pollution, extreme heat, flooding, drought, and ecosystem threats. Each section includes a brief review of the climate hazard, associated exposure pathways and vulnerability factors along with the corresponding health outcomes in order to facilitate ongoing discussions and analyses of the health risks associated with climate change in Minnesota. The overarching goal of providing this information is to highlight what is currently known about each climate hazard and potential health impacts, as well as data or information gaps and various system challenges that are making it difficult to improve our knowledge base and subsequent promotion of climate adaption actions.

Air Pollution

A number of factors can contribute to "bad air days", including unfavorable weather, air pollutants such as greenhouse gas emissions and particulate matter, and high pollen counts (EPA, 2008; Jacob & Winner, 2009). Hotter, more stagnant weather systems combined with rising levels of air contaminants may lead to longer periods of exposure to a greater amount of pollutants for many Americans (Fang et al., 2013; Norris et al., 2000). Already, nearly 5 in 10 people live where pollution levels are often dangerous to breathe (ALA, 2014). In addition, some emission pollutants can trigger climate conditions that add to and amplify existing contamination and health risks. For example, breathing in particulate matter (such as black carbon, a component of soot) for even short periods of time is associated with an increased risk of heart attack and other forms of heart disease (Fang et al., 2002; Zanobetti et al., 2014). In addition, black carbon particles absorb sunlight increasing atmospheric temperatures, which contribute to the formation of another major pollutant, ground-level ozone, which is also harmful to health (EPA, 2012).

Minnesota's overall air quality has improved over the past two decades (MPCA, 2013). This is a notable accomplishment given that population density, greenhouse gas emissions, and economic activity have increased (Figure 5.2).

FIGURE 5.2. COMPARISON OF GROWTH AREAS AND EMISSIONS IN MINNESOTA. MINNESOTA'S AIR QUALITY IS IMPROVING DESPITE INCREASES IN POPULATION AND ECONOMIC ACTIVITY. IMAGE SOURCE: MPCA, 2013.



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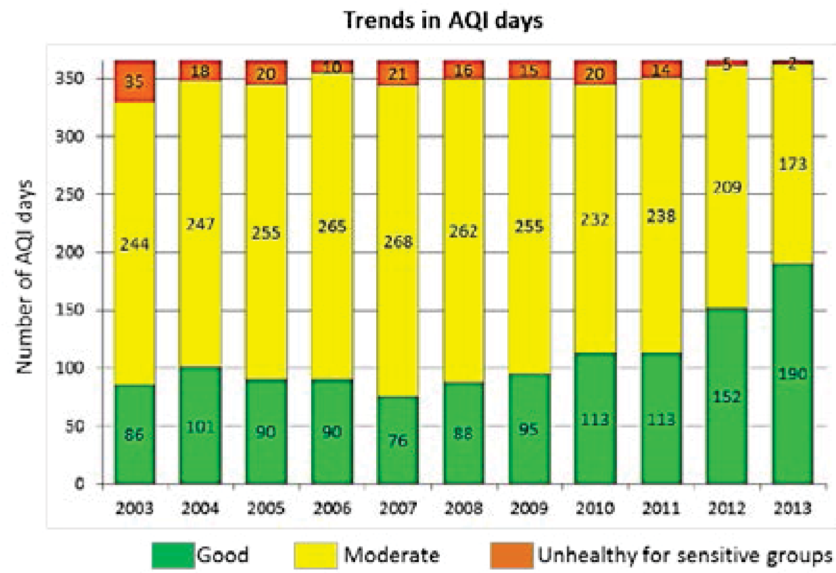
For over a decade, Minnesota air has met federal air quality limits for most major air pollutants. The Air Quality Index (AQI) is used to represent real-time air quality conditions and communicate current health risk to the public. Minnesota has experienced only a small number of days where the AQI exceeded the range for healthy air conditions (Figure 5.3), and generally the average annual AQI for the state has been consistently below the national average.

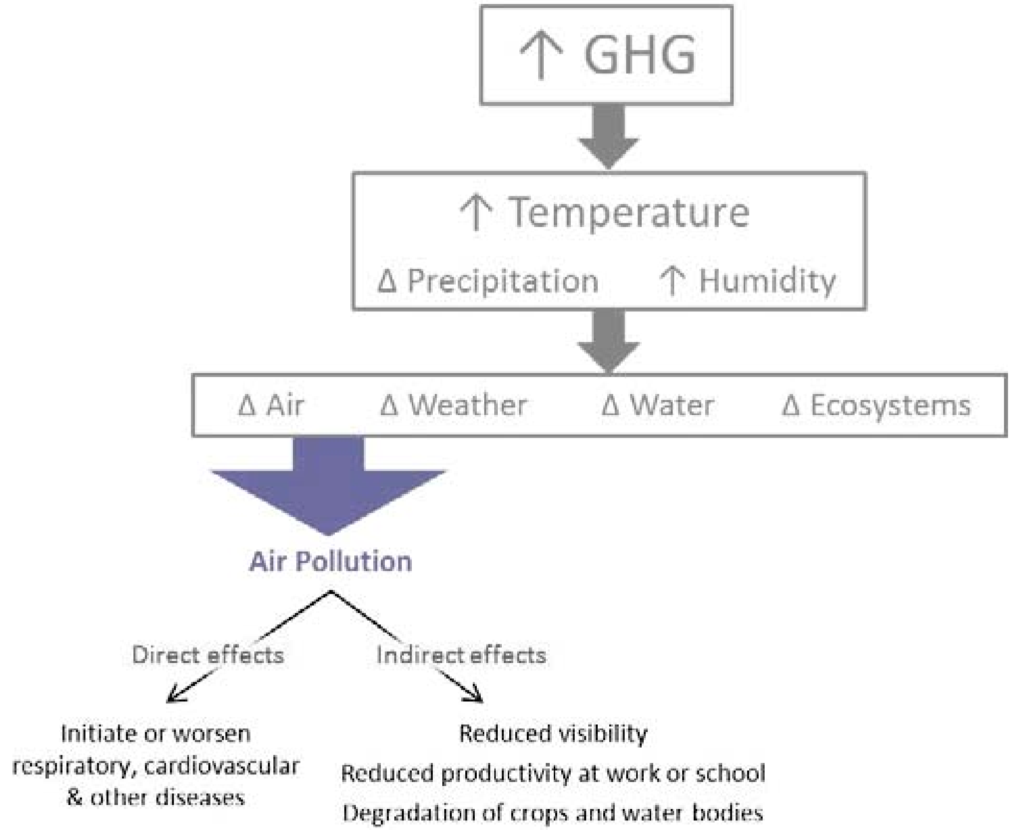
Yet, air quality issues are a concern for public health, and many sources of air pollution in the state are challenging to address. Small, widespread sources of air emissions—like cars, trucks and wood burning—are significant contributors to pollution in Minnesota and lack the same oversight as factories or power plants (MPCA, 2014). Air quality in Minnesota's urban centers, especially the Twin Cities metro area, is worse compared to other areas of the state, largely due to elevated levels of ozone and particulate matter (MPCA, 2013; 2014). Many air pollutants can travel long distances, especially on windy days. Thus, a pollutant like ozone generated in the densely populated Twin Cities area, where ozone pre-cursors (from traffic and power plant emissions) are plentiful, can be transported across the state impacting air quality in suburban and rural areas (MPCA, 2014). Conversely, rural and suburban areas with allergenic trees, plants and shrubs can negatively impact urban "airsheds." Pollen from a ragweed plant can travel up to 400 miles by wind (NIAID, 2012).

FIGURE 5.3. AIR QUALITY INDEX (AQI) TRENDS FOR MINNESOTA, 2003-2013. ASSIGNED TO EACH DAY OF THE YEAR, AN AQI VALUE CORRELATES TO ONE OF FIVE HEALTH-BASED CATEGORIES: GOOD, MODERATE, UNHEALTHY FOR SENSITIVE GROUPS, UNHEALTHY, AND VERY UNHEALTHY. THE DAILY AQI IS BASED ON THE HIGHEST POLLUTANT MEASURED ACROSS ALL MINNESOTA MONITORING SITES (OVER 30 STATIONS ACROSS THE STATE). THE STATEWIDE AQI TREND SHOWS IMPROVEMENTS IN AIR QUALITY OVER TIME. IMAGE ADAPTED FROM MPCA, 2014.



Minnesotans will experience a wide range of direct and indirect effects from increased air pollution resulting from climate change (Figure 5.4). The focus of this section will be on exposure to three air pollutants of most concern for public health (Jacob & Winner, 2009), particulate matter, ground-level ozone, and pollen, a "natural" hazard that is a growing problem for Minnesotans. Research has demonstrated a relationship between climate change and each of these hazards (EPA, 2008; Fang et al., 2013; Jacob & Winner, 2009). Major health issues related to air pollution include, chronic obstructive pulmonary disease (COPD), lung cancer, cardiovascular disease, allergies and asthma. Indirect effects from air pollutants on health, such as reduced visibility, reduced productivity and degradation of crops and water sources, can represent a serious health and economic burden to many communities but are not a substantial risk for the health of Minnesotans at this time. Therefore, these indirect effects will not be addressed in this report but may be added in future iterations.

FIGURE 5.4. LINKS BETWEEN THE RISE IN ATMOSPHERIC GREENHOUSE GASES, CHANGES TO MINNESOTA'S CLIMATE, AND DIRECT AND INDIRECT IMPACTS ON HEALTH FROM AIR POLLUTION.



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PARTICULATE MATTER

Particulate matter is a broad class of chemically and physically diverse material that exists over a wide range of sizes. Two sizes in particular are associated with adverse health effects: coarse particles with diameters less than or equal to 10 microns (PM_{10}) and fine particles with diameters less than or equal to 2.5 microns ($PM_{2.5}$). Particulate matter can be released directly from sources, such as forest fires or factories, or they can form when emissions from power plants and vehicles react in the air. Vehicle exhaust contributes one-third to one-half of fine particle concentrations in urban areas (MPCA, 2013). Particulate matter is both a product and cause of climate change (Ebi & McGregor, 2008; Griffin, 2013). According to experts, certain types of particulate matter are "extremely important climate forcers" (Dawson et al., 2014) because they can absorb and radiate energy from the sun, contributing toward global warming trends. Particulate matter also affects cloud formation and duration (Griffin, 2013). Current studies suggest that air stagnation and precipitation frequency may be important aspects of the relationship between climate change and particle pollution (Jacob & Winner, 2009). Levels of particulate matter in the air can fluctuate across seasons, but in general, it is viewed as a year-round air quality problem. Patterns of particulate matter are difficult to predict given the influence of local factors and sources coupled with the fact that particles can travel long distances.

The size of fine particles is directly related to their potential to harm human health. Very small particles can pass through the throat and nose, lodge deep in the lungs, and even pass into the bloodstream and move throughout the body. Studies have shown that fine particles can bind and transport other harmful toxins, like metals, thus increasing the hazards of inhalation (Aust et al., 2002). Exposure to particulate matter has been linked to numerous adverse health conditions, such as asthma, chronic bronchitis, reduced lung function, irregular heartbeat, heart attack, and premature death (EPA, 2014; MPCA, 2013; Sacks et al., 2011). Particulate matter also is associated with damage to forests and crops, acidification of lakes and streams, disruption to the nutrient balance in coastal waters and river basins, nutrient depletion in soils, and reductions in visibility (EPA, 2014).

The Minnesota Pollution Control Agency (MPCA) collects hourly measurements of fine particles, ozone, sulfur dioxide, and carbon monoxide at over 30 locations across the state and posts hourly AQI results on the MPCA website (www.pca.state.mn.us). Statewide levels of particulate matter (both $PM_{2.5}$ & PM_{10}) in Minnesota generally meet annual and daily standards and are below the national mean (MPCA, 2013). However, in 2012, the EPA strengthened the annual air standard for $PM_{2.5}$ from 15 micrograms per cubic meter of air (µg/m^3) to 12 µg/m^3, putting some urban areas of the state under pressure to remain in compliance (MPCA, 2013). MPCA has calculated estimates of the health benefits associated with improving air quality in Minnesota. For each reduction of one µg/m^3 in annual ambient $PM_{2.5}$ levels, there would be annual health benefits in 2020 of about two billion dollars (MPCA, 2013).

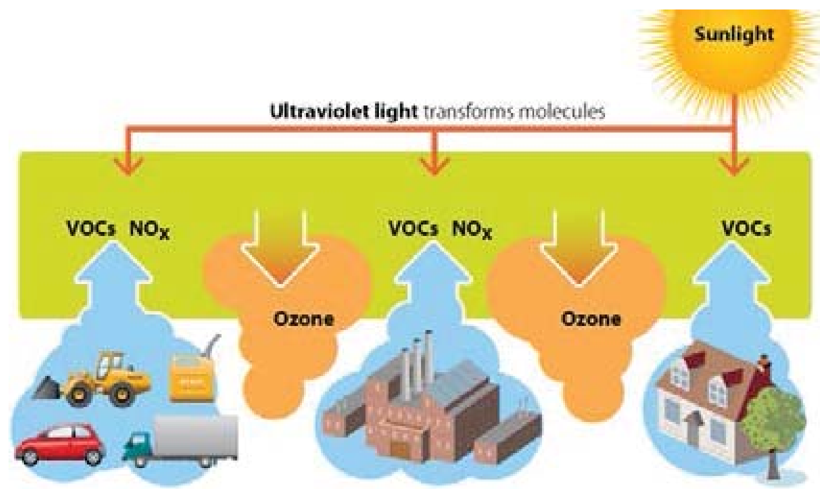
Photo courtesy of Wikimedia Commons.

OZONE

Ozone is a colorless gas produced naturally in the upper layer of the Earth's atmosphere where it performs the essential task of absorbing most of the harmful ultraviolet radiation from the sun. Ozone also is found in the lowest portion of the atmosphere, at the Earth's surface. Surface or ground-level ozone is not health-protective. In fact, it can be very dangerous to human health. Ground-level ozone forms when air pollutants, like carbon monoxide, nitrogen oxides and volatile organic compounds (by-products of fossil fuel combustion and other sources), are exposed to heat and sunlight (Figure 5.5). Results from several modeling studies suggest that climate-induced changes in temperature, cloud cover, and circulation patterns may increase ozone concentrations over large parts of the U.S. The EPA has concluded that climate change could lead to a 2-8 parts per billion (ppb) increase in the summertime average for ground-level ozone levels in many regions of the country, as well as an overall lengthening of the ozone season (EPA, 2009). On hot, sunny summer days, which are becoming more frequent, ozone concentrations can rise to unhealthy levels. Like particulate matter, winds may carry ozone long distances. Recent statewide measures of ambient ozone indicate that the highest levels are downwind of the Twin Cities urban core in the surrounding suburban areas (MPCA, 2013). All areas of the state generally meet federal standards for ozone, but Minnesota is at risk for being out of compliance in the near future. Similar to actions taken on particulate matter, EPA also has proposed new standards for ozone, reducing acceptable levels from 75 ppb to somewhere in the range of 60-70 ppb. Many areas of Minnesota would be out of compliance if the EPA revises the current standard to the lower end of this range (MPCA, 2013).

Breathing air containing ozone can reduce lung function and irritate airways, which can aggravate asthma and lead to respiratory disease. Ozone exposure also increases the risk of premature death from heart or lung disease (EPA, 2013). The MPCA has estimated that for each incremental reduction of one ppb in ozone concentration, there would be annual health benefits in 2020 of about 150 million dollars (MPCA. 2013). The impacts of high ozone levels are not limited to direct health effects but may indirectly impact crops and forests (EPA, 2013). Ground-level ozone can interfere with plants' ability to produce and store food, damaging leaves and reducing forest growth and crop yields.

FIGURE 5.5. HOW GROUND-LEVEL OZONE FORMS. VOLATILE ORGANIC COMPOUNDS (VOCS) AND NITROGEN OXIDES (NO_X) FROM VEHICLE AND FACTORY EMISSIONS, FUEL AND CHEMICAL VAPORS, OR HEATING SYSTEMS ARE TRANSFORMED BY ULTRAVIOLET LIGHT INTO GROUND-LEVEL OZONE. IMAGE SOURCE: MPCA, 2013.



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POLLEN

The study of climate change and plant physiology is a compelling area of research across many disciplines. Agricultural scientists want to know how elevated CO_2 and climate changes are likely to impact crop yields, while forest managers want to know the impacts for native forests, harvestable timber stands, and invasive species. In public health, increasing attention is being paid to the effects of CO_2 and climate change on pollen as an "aeroallergen", given the health consequences of increased pollen exposure on individuals with allergies, asthma or other respiratory ailments (EPA, 2008). Current climate research is examining how climate changes (e.g., increased CO_2 levels and other air pollutants, rising temperatures, and both increased and decreased regional precipitation) can alter the production, distribution, and potency of aeroallergens (EPA, 2014; 2008).

Research shows that pollen production is on the rise in most areas of the U.S., and especially in the Midwest (EPA, 2008). A recent study based on data collected from various Midwest locations for the period 1995-2013 revealed that the ragweed pollen season has increased by as much as 10-22 days for areas in and around Minnesota (Figure 5.6; Ziska et al., 2011). The authors report that the lengthening pollen season is strongly related to climate change characteristics, such as lengthening of the frost-free season and later timing of the first fall frost, changes that are most pronounced in northern areas of Minnesota. This trend is consistent with other observations showing that climate changes and impacts are occurring more rapidly at higher latitudes (IPCC, 2007).

Besides longer pollen production seasons (which equates to longer exposure times for humans), there is evidence that certain trees and plants are producing larger quantities of pollen (Beck et al., 2013; Ziello et al., 2012). By some estimates, if greenhouse gas emissions continue on their current trajectory, pollen production is projected to increase by 60 to 100 percent by 2085 from the CO_2 fertilization effect alone. Many allergenic plants are already highly effective pollen-producers; each ragweed plant is able to produce about one billion grains of pollen each season (NIAID, 2012). There also is growing evidence that pollen is becoming more allergenic. One study found that production of an allergenic protein in ragweed increased by 70 percent when CO_2 levels were increased to 600 ppm, the concentration expected by mid-century if emissions are not drastically reduced (Singer et al., 2005). Given that quantities and allergenicity of pollen are rising in tandem with levels of other damaging air pollutants (e.g., ozone, particulate matter, nitrogen dioxide), the associated health burden, especially on individuals with pre-existing respiratory problems, may be substantial (EPA, 2008).

FIGURE 5.6. INCREASE IN THE DURATION OF RAGWEED POLLEN SEASON FOR AREAS IN AND AROUND MINNESOTA. IMAGE SOURCE: EPA, 2014.

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DIRECT HEALTH IMPACTS

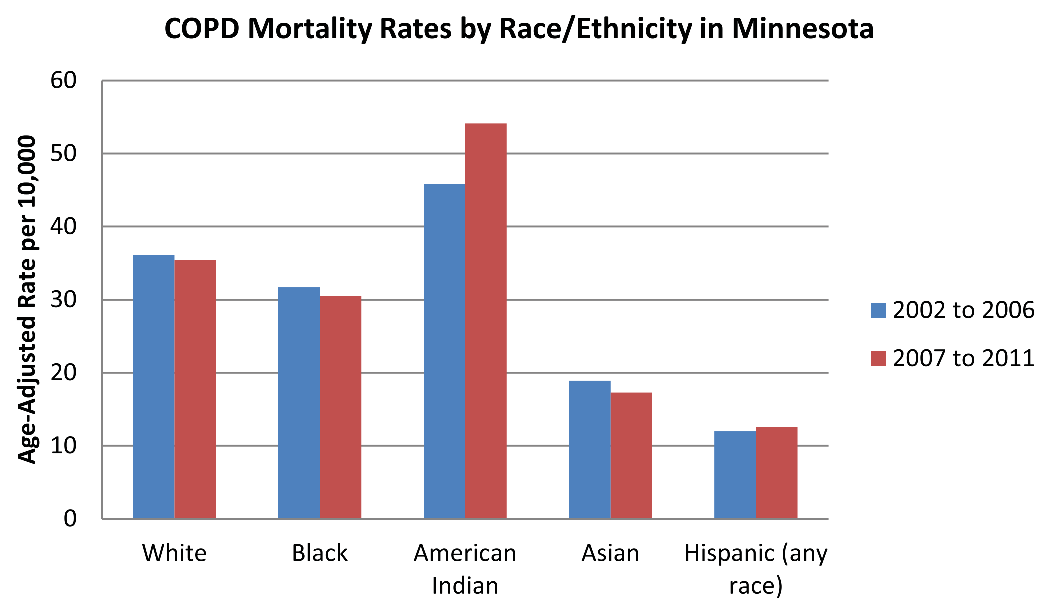
Health effects directly related to ozone, particulate matter and pollen include COPD, lung cancer, cardiovascular disease, asthma and allergies. Although there are many other health and quality-of-life determinants that are affected by air pollution, there is a large body of evidence substantiating links to these specific disease categories, and many Minnesotans are affected by them.

CHRONIC OBSTRUCTIVE PULMONARY DISEASE

COPD is a progressive inflammatory condition of the respiratory tract, which is projected to be the third leading cause of death and fifth leading cause of disability in the U.S. by 2020 (Mannino et al., 2002). While cigarette smoking is responsible for the vast majority of COPD, environmental factors are gaining attention. Evidence shows that exposure to air pollution, including ozone, particulate matter, and pollen, is associated with the development and progression of COPD (Faustini et al., 2012; Hanigan & Johnston, 2007; Li et al. 2013). This risk exists for both acute exposure of several days to high levels of air pollution and chronic exposure over a number of years to low levels (Andersen et al., 2011). The prevalence of COPD rises with age, and the risk is greater for people with other health-related conditions, such as diabetes, asthma or obesity.

According to a recent report released by the Minnesota chapter of the American Lung Association in partnership with MDH, over four percent of Minnesotans report living with COPD, or approximately 165,000 people (ALA, 2013). The disease is currently the fifth-leading cause of death in the state (MDH, n.d.). COPD negatively impacts employment status as well as health: One-fifth of Minnesota adults unable to work in 2011 reported having COPD, and two out of three adults living in the state with COPD report being diagnosed before the age of retirement (65 years). There also are significant race disparities in COPD risk as American Indians are burdened with the highest death rates attributed to COPD (Figure 5.7; ALA, 2013).

FIGURE 5.7. CHRONIC OBSTRUCTIVE PULMONARY DISEASE (COPD) DEATHS BY RACE AND ETHNICITY IN MINNESOTA. RATE IS ADJUSTED FOR AGE AND REPRESENTS NUMBER OF DEATHS PER 100,000 INDIVIDUALS. AMERICAN INDIANS IN THE STATE HAVE THE HIGHEST RATES OF DEATH FROM COPD COMPARED TO PEOPLE WHO ARE BLACK, WHITE, ASIAN OR HISPANIC. IMAGE SOURCE: MDH, N.D.



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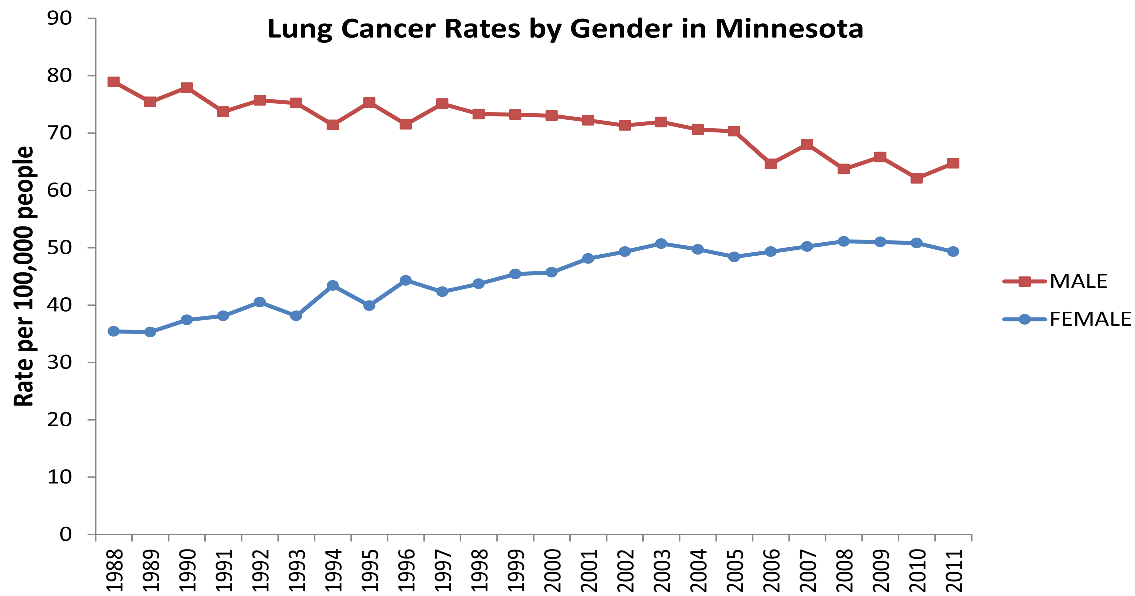
LUNG CANCER

Cigarette smoking is the number one cause of lung cancer; however, a large body of evidence shows that air pollution can contribute to lung cancer risk. The International Agency for Research on Cancer (IARC) has classified outdoor air pollution, on the whole, as a cancer-causing agent (Loomis et al., 2013). The IARC is part of the World Health Organization and a global leader in cancer research. According to lead IARC researcher Dr. Kurt Straif, "Outdoor air pollution is not only a major environmental risk to health in general, it is the most important environmental cancer killer due to the large number of people exposed." (ACS, 2013). In its evaluation, the IARC identified lung cancer as the primary cancer risk associated with air pollution, demonstrating an increasing risk of lung cancer with increasing levels of exposure to outdoor air pollution. The IARC also classified particulate matter as a carcinogen on its own, a decision that reflects a large body of epidemiological evidence showing that exposure to both fine and coarse particulate matter contributes to lung cancer risk (Ghassan et al., 2014). Further research is needed to clarify if any association exists between ozone and lung cancer (Hystad et al., 2013).

Lung cancer is the second most common cancer diagnosis in Minnesota, and it is the leading cause of cancer mortality in the state (ACS, 2014). Lung cancer kills more than twice as many men as prostate cancer and nearly twice as many women as breast cancer in Minnesota (MDH, n.d.). Over the last few decades, rates of lung cancer are falling for men but rising for women (Figure 5.8). A large difference also exists among different race/ethnicity categories, with American Indians leading the state in lung cancer incidence.

FIGURE 5.8. LUNG AND BRONCHUS CANCER INCIDENCE BY GENDER IN MINNESOTA. RATE IS ADJUSTED FOR AGE AND REPRESENTS NUMBER OF NEW CASES PER 100,000 INDIVIDUALS. SINCE 1988, INCIDENCE AMONG MALES HAS DECREASED BY ABOUT 20 PERCENT WHILE RISING AMONGST FEMALES BY ABOUT 30 PERCENT. IMAGE SOURCE: MDH, N.D.

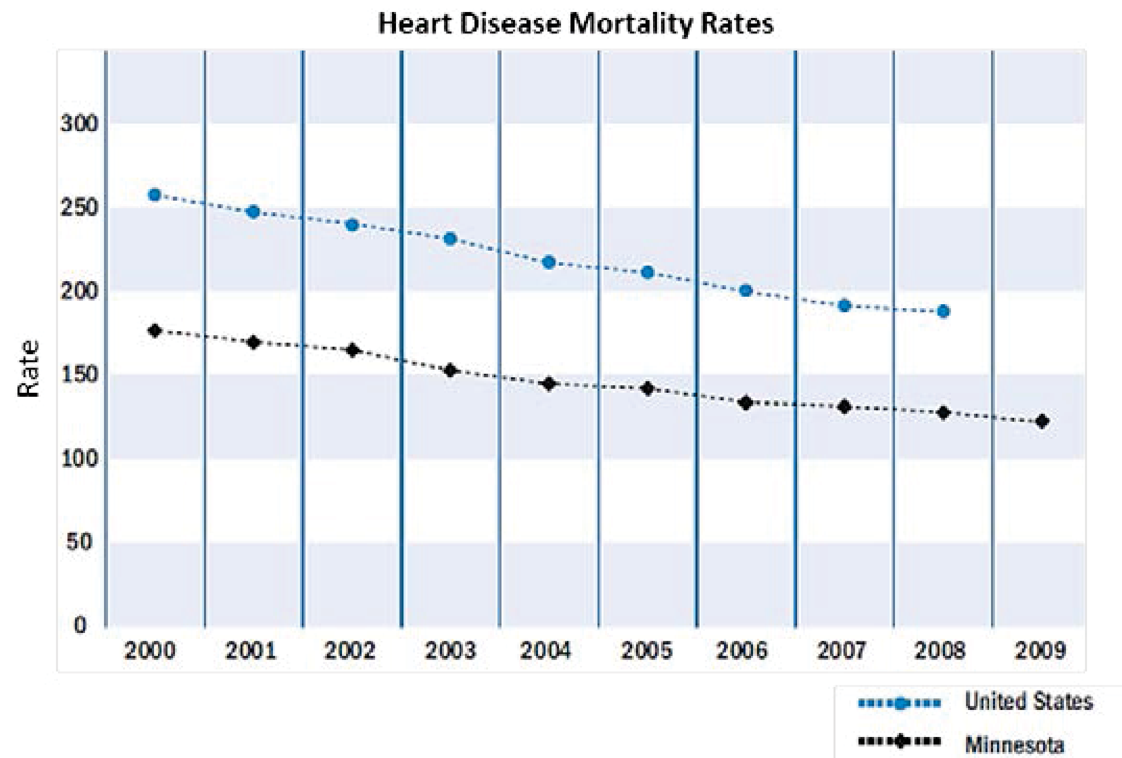


CARDIOVASCULAR DISEASE

Decades of research have shown that air pollution negatively impacts cardiovascular health and can trigger heart attacks, strokes, and irregular heart rhythms (EPA, 2014; Lee et al., 2014; Nawrot et al., 2011). Past evidence has focused on particulate matter as the primary contaminant of concern for cardiovascular health, but more attention is being paid to ozone and its role in vessel and heart-related disease. Large population studies have demonstrated that both long-term and short-term exposures to air pollution can lead to poor cardiovascular outcomes (Brook et al., 2004; Lee et al., 2014). Cardiovascular impacts are worse for individuals with pre-existing heart or blood vessel disease. A recent study of cardiac patients found that those living in high pollution areas were over 40 percent more likely to have a second heart attack or suffer congestive heart failure and 46 percent more likely to suffer a stroke (Koton et al., 2013).

According to a recent report by MDH, approximately 139,000 Minnesotans (3.5% of adults) have coronary heart disease, and over 90,000 (2.3% of adults) have had a stroke (Peacock & Shanedling, 2011). In 2009, heart disease and stroke were the second and fourth leading causes of death in the state, respectively. While Minnesotans have fewer risk factors related to cardiovascular disease (e.g., smoking, low physical activity, hypertension, diabetes), rates of overweight and obesity for the state have been consistently at or above the national median rate. However, Minnesota continues to have lower mortality rates due to heart disease and stroke (Figure 5.9). Unfortunately, the racial disparities seen in other health outcomes also are seen in the trends for cardiovascular disease: American Indians have consistently higher heart disease mortality compared to all other races (Peacock & Shanedling, 2011).

FIGURE 5.9. HEART DISEASE MORTALITY RATE FOR THE UNITED STATES (TOP BLUE LINE) AND MINNESOTA (BOTTOM BLACK LINE), INCLUDES ALL AGES, 2000-2009. RATE IS ADJUSTED FOR AGE AND REPRESENTS NUMBER OF DEATHS PER 100,000 INDIVIDUALS. IMAGE SOURCE: PEACOCK & SHANEDLING, 2011).



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ALLERGIES & ASTHMA

Allergies are becoming more prevalent in the U.S. and significantly impact health, quality-of-life, and the economy. More than 50 million Americans suffer from allergies each year (CDC, 2011). By some estimates, over 54% of people in the U.S. test positive for at least one allergen (Arbes et al., 2005). Allergy is the sixth leading chronic disease in the U.S. among all ages, collectively costing the health care system approximately 18 billion dollars every year (CDC, 2011). Allergic conditions are among the most common medical conditions affecting U.S. children (Jackson et al., 2013). Across all age groups, allergies account for more than 11 million outpatient office visits annually, primarily in the spring and fall (Blackwell et al., 2014). Hay fever in particular is a major cause of work absenteeism and reduced productivity, with associated at-work productivity loses ranging from 2.4 to 4.6 billion dollars (Crystal-Peters et al., 2000).

There are three main categories of pollen allergens: tree, weed, and grass. Trees account for the large majority of pollen produced (75-90 percent), followed by weeds (6-17 percent) and grasses (3-10 percent) (EPA, 2008). However, clinical studies show that exposure to allergens from grasses, followed by weeds and trees, are more prone to result in development of allergic diseases (EPA, 2008). The development of allergic diseases occurs through a two-stage process. First, an individual is exposed and sensitized to the allergen, resulting in the production of specific antibodies. Second, subsequent exposure to the allergen elicits disease symptoms due to the presence of the antibodies and the associated inflammatory response. It only takes a small amount of grass pollen (e.g., 4-12 grains/m^3 of air) to initiate allergic symptoms in a sensitized individual (EPA, 2008).

Hay fever symptoms are familiar to many in the Midwest: runny or stuffy nose, sneezing, and itchy eyes, nose and throat. For most sufferers, ragweed pollen is the primary trigger of fall hay fever. Rates of ragweed sensitization are especially high in Minnesota. Quest Diagnostics runs millions of blood tests for allergies every year and analyzed the results by city. Minneapolis/St. Paul ranked 10th among the nation's most populous cities for ragweed sensitivity. Nearly 22% of patients tested in the metro area for ragweed sensitization return positive results (Quest Diagnostics, 2011).

People with asthma and other respiratory ailments are especially vulnerable to aeroallergens, given that these conditions increase both sensitivity and allergic responses to exposure. Over 80% of people with asthma suffer from allergies and 10-40% of people with allergies have asthma (Bousquet et al., 2001). Asthma is a chronic respiratory disease characterized by episodes of airway constriction and inflammation and is one of the most common chronic diseases in the U.S. (Moorman et al., 2012). The prevalence of asthma in the U.S. has increased from approximately 3.1 percent in 1980 to 8.4 percent in 2010 (Moorman et al., 2012). Asthma also is a costly health burden. Over the years 2002-2007, the annual costs attributed to asthma in the U.S. from health care and lost productivity was approximately 56 billion dollars (Barnett et al., 2011).

Common Ragweed. Photo courtesy of Sue Sweeney, Wikimedia Commons.

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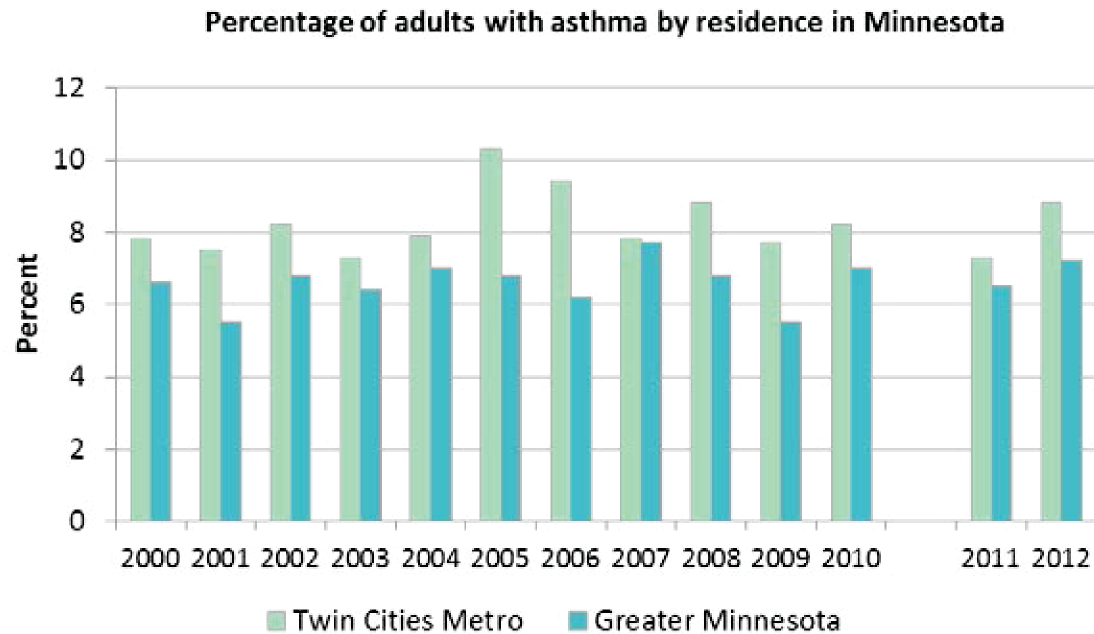
MDH maintains an asthma surveillance system to track and describe asthma prevalence in Minnesota. In 2012, eight percent, or 1 in 12, of Minnesota adults reported that they live with asthma compared to 8.9 percent, the median estimate for all U.S. states (W. Brunner, personal communication, July 22, 2014). While asthma prevalence has steadily increased across the U.S., rates for Minnesota have been relatively steady since 2006. However, rates of reported asthma for the Twin Cities metro area have been consistently higher compared to the rest of the state (Figure 5.10). In 2011, 7.1 percent of Minnesota children were reported to have asthma compared to the national estimate of 8.8 percent (MDH, 2012). However, there are distinct racial and ethnic disparities in asthma prevalence among Minnesota youth. Based on data from the 2013 Minnesota Student Survey, rates of asthma are higher in American Indian and African American youth compared to youth of other racial or ethnic groups. Impacts to Minnesota's economy from asthma are substantial. In 2010, asthma required approximately 544 million dollars in direct medical expenditures and 62 million in missed work days (CDC, 2014).

FIGURE 5.10. PERCENTAGE OF ADULTS WITH ASTHMA IN THE TWIN CITIES METRO AREA COMPARED TO THE REST OF MINNESOTA. THE GAP BETWEEN 2010 AND 2011 INDICATES A CHANGE IN METHODOLOGY. IMAGE SOURCE: W. BRUNNER, UNPUBLISHED DATA.



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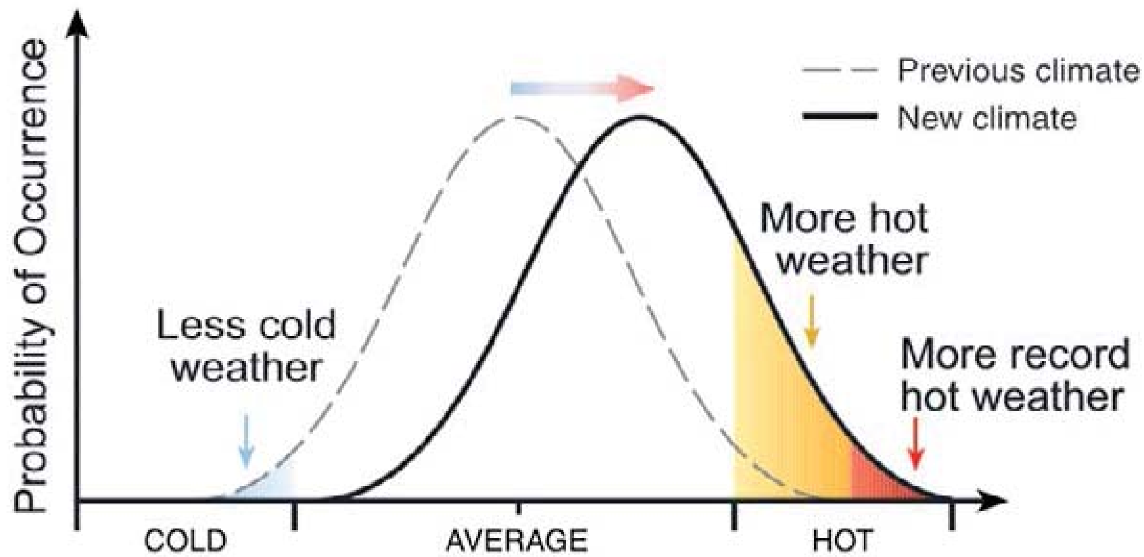
Assessment of Health Risk from Air Pollution	
Hazard	While air pollution in Minnesota is below the national average on measures of ozone and particulate matter, it still can be a hazard, particularly for populations within or downwind from the Twin Cities metropolitan area (TCMA). Pollen also is a widespread, increasing health hazard for many residents across the state, especially for the estimated 392,000 Minnesotans with asthma.
Exposure	People may be exposed to air pollution at any time, but the likelihood increases with proximity to emission sources, season, and weather. Current statewide ambient air monitoring efforts maintained by the MPCA provide a crucial means of characterizing exposure to major air contaminants of concern like ozone and particulate matter. However, there is very little monitoring for pollen. The American Academy of Allergy Asthma & Immunology maintains a single certified pollen counter in Minneapolis. Daily data on pollen levels are only publicly available for a limited time period.
Vulnerability	Environment: Air pollution can damage buildings, forests, and crops and acidify lakes and rivers, making water unsuitable for fish and other wildlife. In addition, air pollution can create haze and reduce visibility, leading to unsafe conditions for drivers and some occupations. Humans: Everyone is vulnerable to air pollution to some degree, but some individuals are more vulnerable than others, especially people with pre-existing cardiovascular or respiratory conditions, the elderly, children, and people who are active outdoors.
Risk	There is a large body of evidence demonstrating the close relationship between mortality and morbidity and exposure to high levels of air pollution. In addition, numerous studies on climate change and air quality suggest that climate changes will generate or amplify air contamination. Although much of greater Minnesota benefits from air quality that meets current federal health-based standards, conditions in the TCMA have been consistently worse compared to rural areas of the state. Given that more than half of the state's population lives in the TCMA (and population forecasts estimate that this percentage will increase), a substantial number of Minnesotans may be at risk for adverse health outcomes from air pollution, especially those with pre-existing medical conditions.

Extreme Heat

While the rise in greenhouse gas emissions is associated with gradual warming trends, it is also sparking more heat extremes. Experts predict that as average temperature increases, extreme heat events will become more frequent, longer lasting, and more severe (Figure 5.11). According to the United Nation's World Meteorological Organization, 13 of the 14 warmest years in recorded history have occurred in this century, with 2001-2010 being the warmest decade on record (WMO, 2014). Experts in the government study, *Global Climate Change Impacts in the United States*, report that high humidity heat waves have become both more frequent and more intense in the last 30 to 40 years (Karl et al., 2009). In the 1950s, record low temperatures were just as likely to occur as record highs. In contrast, over the later half of the twentieth century, the U.S. experienced twice as many record highs as record lows (Meehl et al., 2009). A 2014 study provides evidence of an upward trend in extreme heat worldwide with projections indicating further increases in record high temperatures and number of heat wave days (Seneviratne et al., 2014). By the end of this century, extremely high temperatures that currently occur once every 20 years could occur as often as every two to four years (CDC, 2013).

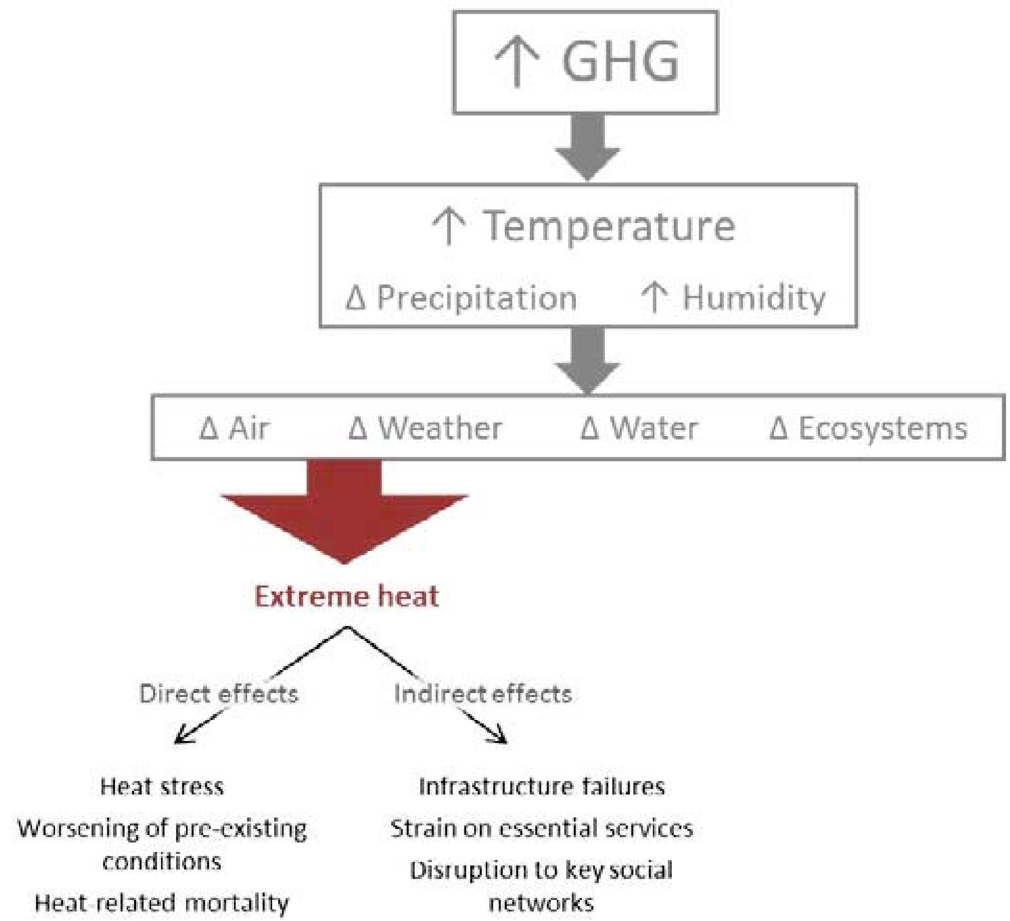
The spike in extreme heat events appears to be especially pronounced in the Midwest and has been observable in Minnesota (O'Neill & Ebi, 2009; Meehl & Tebaldi, 2004). A study led by the Union of Concerned Scientists investigated over 60 years of data on summer air masses affecting ten Midwestern cities, including Minneapolis (Perera et al., 2012). The authors found that on average summers in Minneapolis now have nearly five more days of the hottest and most humid weather compared to the mid-1940s. The city also gets less relief having lost on average nearly five cool, dry summer days. Nighttime temperatures on hot, humid nights have risen by 1.6°F along with an increase in dew point temperatures by 2.2°F. Minneapolis has gained an additional heat wave each summer, defined as three or more days of a dangerous hot air mass.

FIGURE 5.11. IMPACT OF INCREASED AVERAGE TEMPERATURES. AS AVERAGE TEMPERATURE INCREASES, EXTREME HEAT EVENTS WILL BECOME MORE FREQUENT AND SEVERE. FIGURE SOURCE: CDC, 2013.



Minnesotans will experience a wide range of direct and indirect health impacts from the increased frequency and severity of extreme heat events (Figure 5.12). Direct health effects include symptoms associated with heat stress, such as fatigue, cramps, headaches and nausea, or responses that are much more extreme, including heat stroke, organ failure, and even death. In addition, heat waves can exacerbate pre-existing medical conditions or diseases, such as diabetes, cardiovascular disease, chronic obstructive pulmonary disease, kidney ailments and mental or behavioral disorders. Indirect health effects include infrastructure failures like power outages; disruption of some occupations (especially those involving outdoor, strenuous labor), schooling, or major events, like athletic competitions or festivals; and a strain on emergency and health care services, in particular 911 response and emergency department operations. In addition, extreme heat contributes to other major climate impact areas, such as air contamination and drought, which in turn have direct and indirect effects on the health of Minnesotans.

FIGURE 5.12. LINKS BETWEEN THE RISE IN ATMOSPHERIC GREENHOUSE GASES, CHANGES TO THE EARTH'S CLIMATE, AND DIRECT AND INDIRECT IMPACTS ON HEALTH FROM EXTREME HEAT EVENTS.



The U.S. Centers for Disease Control and Prevention (CDC) forecasts that "extreme heat is a real danger to human health that will become worse with time" (Perera et al., 2012). The experience of heat is somewhat relative and place-based, given that humans can acclimate to a range of environmental conditions within physiological limits. An outdoor temperature that a person in coastal New England identifies as a heat wave may cause only mild discomfort for a native of west Texas. Therefore, it is difficult to develop absolute standards for defining an extreme heat event. Generally, most definitions will include a reference to a period of several days or more with weather that is substantially hotter than average for that location at that time of year.

The National Weather Service (NWS) bases its heat advisories in part on the Heat Index (HI). HI is a measure of how hot it feels to the average person when relative humidity is factored in with actual air temperature. The NWS will initiate heat alerts when HI is forecasted to exceed 105°-110°F (depending on average local conditions) for at least two consecutive days (NWS, 2014). However, it should be noted that HI values were devised for shady, light wind conditions. Exposure to full sun and strong winds could increase values up to 15°F. HI is commonly used as a proxy measure of heat exposure in health studies (Anderson et al., 2013) and is the basis of many emerging extreme heat surveillance systems (Hajat et al., 2010; Kent et al., 2014; Metzger et al., 2010).

Heat waves are the leading cause of weather-related mortality in the U.S. (Davis et al., 2003). Between 1999-2009, 7,233 heat-related deaths occurred across the U.S., an average of 658 per year (CDC, 2013b). In the large majority of these deaths, the primary underlying cause was exposure to excessive heat, while heat was a secondary or contributing factor in the remaining deaths. Fatalities were most frequent among males, adults over 65 years, and individuals without air conditioning. Almost all heat-related deaths occurred during the summer season (May–September), with the highest numbers reported during July and August (CDC, 2013b). In recent years there have been several notable heat waves that have caused a catastrophic number of deaths around the world, including a 2003 heat wave that caused over 70,000 deaths across Europe (Robine et al., 2007). Closer to home, more than 700 deaths have been attributed to the 1995 Chicago heat wave, a tragedy that garnered a great deal of awareness for extreme heat hazards amongst U.S. public health and emergency preparedness professionals (Palecki et al., 2001).

Yet, the complete number of deaths and illnesses from extreme heat is often underreported. Most states or municipalities do not have an official, real-time reporting system in place specifically dedicated to tracking morbidity or mortality cases related to extreme heat. Also, there is a wide range of heat exposure symptoms, and many individuals who are impacted do not seek professional medical attention. If they do, clinicians often do not code the visit or fatality as primarily heat-related. Some studies of heat-related hospitalizations and mortality have found that using primary diagnoses alone can underestimate actual incidence (Kilbourne, 1999; Semenza et al., 1999).

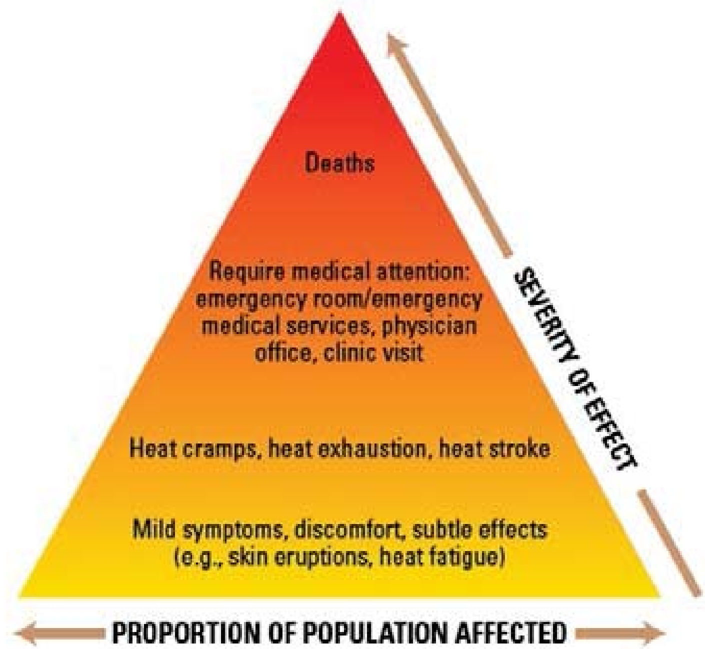
Photo courtesy of Wikimedia Commons.

DIRECT HEALTH IMPACTS

Even small temperature changes can have a dramatic effect on the human body (Figure 5.13). Humans can only survive when core body temperature stays within a narrow range around 98.6°F. If the body produces or absorbs more heat than it can remove through sweating or other cooling mechanisms, core temperature will rise. If it exceeds 100°F for several hours, symptoms like heat exhaustion and reduced mental and physical capacity are likely to occur as organ systems are increasingly stressed to maintain homeostasis. At 102°F core body temperature, heat stroke and loss of consciousness threatens, and beyond 107°F death will occur after a relatively short time (Berry et al., 2010; Parsons, 2003).

The risk of heat-related illness varies from person to person, depending on their general health and how well they are already adapted to heat. Individuals who are overweight, older, taking certain medications, have a poor level of physical fitness, or afflicted with pre-existing medical conditions may be more susceptible to feeling the extremes of heat and suffering worse outcomes (CDC, 2013; Gronlund et al., 2014). Individuals with conditions that affect the circulatory, metabolic, or respiratory systems are especially vulnerable to high heat given that these systems are essential for maintaining the body's internal thermostat. A large body of evidence exists demonstrating that heat-related hospitalizations and mortality rates are higher in populations with co-morbidities (Fletcher et al., 2012; Kosatsky, 2012; Lavigne et al., 2014; Li et al. 2012). In addition, a growing number of studies are including mental, emotional and behavioral disorders in their assessment of health outcomes related to heat exposure. A recent study based in Toronto found a strong association between emergency department visits for mental and behavioral disorders and mean daily temperatures above 80°F (Wang et al., 2014). A study based in Sydney found that hospital admissions were higher on hot days for numerous morbidity outcomes, included psychoses (Vaneckova & Bambrick, 2013). There is also evidence of an association between extreme heat events and measures of crime and violence (Hsiang et al., 2013; Mares, 2013; Talaei et al., 2014).

FIGURE 5.13. DISTRIBUTION OF HEALTH EFFECTS RELATED TO HEAT. EXTREME HEAT EVENTS CAN CAUSE A RANGE OF MILD TO LIFE-THREATENING HEALTH PROBLEMS AND MAKE OTHER HEALTH PROBLEMS WORSE. WHILE MILD EFFECTS, LIKE LETHARGY AND HEAT RASH ARE MORE COMMON, IN EXTREME CASES PEOPLE CAN DIE, ESPECIALLY THOSE WITH CERTAIN VULNERABILITIES, LIKE THE ELDERLY OR PEOPLE WITH PRE-EXISTING MEDICAL CONDITIONS. IMAGE SOURCE: CDC, 2013.

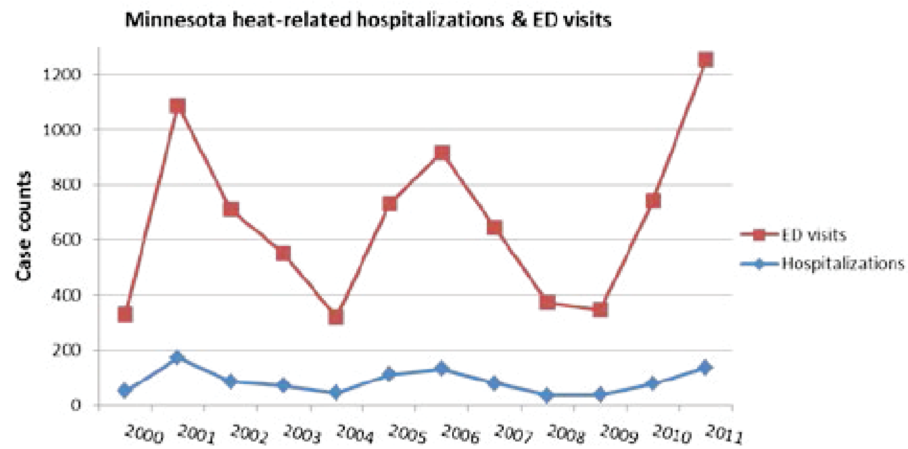


Some studies are suggesting that race and income may play a role in heat susceptibility. A New York State study found that hospitalizations for acute renal failure went up 9% for every 5°F in mean temperature, with black and Hispanic populations carrying twice the odds of whites for hospitalization. There also was some indication of higher risk among the economically disadvantaged (Fletcher et al., 2012). A study based in New York City concluded that extreme high temperatures drove up hospital admissions for cardiovascular and respiratory disorders, but rates were higher for Hispanic persons and the elderly (Lin et al., 2009). There is also some evidence that maternal exposure to high environmental temperatures during certain developmental windows may be linked to birth defects, such as congenital cataracts (Van Zutphen et al., 2012), although studies in this area are limited.

Data on the numbers of hospitalizations, emergency department (ED) visits and deaths directly attributed to heat (i.e., heat exposure is listed as the primary diagnosis or cause) are collected and provided to the public by MDH's Environmental Public Health Tracking program (EPHT). Between 2000-2011, over 1,000 hospitalizations, 8,000 ED visits, and nearly 40 deaths directly attributable to heat exposure were recorded in Minnesota (Figure 5.14). Similar to global patterns of lethality, nearly all heat-related deaths occurred in the summer months, mainly July and August. In most cases, the elderly were the most affected age group and males were more affected than females. However, younger males aged 15-34 had the highest rate of ED visits due to heat-related illness, while males age 65 years old and older had the highest rate of hospitalizations. This may reflect hospital intake practices (e.g., a bias toward hospitalizing older patients) and a higher prevalence of risky behaviors amongst younger males (e.g., engaging in strenuous, outdoor activities without taking necessary precautions) that may lead to an ED visit.

When interpreting EPHT data it is essential to keep in mind certain limitations. First, symptoms of heat-related illness can vary substantially depending on the individual and level of exposure. Since only people with the most severe signs of illness are hospitalized, visit the ED, or perish from exposure, EPHT data cannot be used to represent the total burden of extreme heat events on Minnesotans. Only cases where an individual seeks medical care and heat-related illness is explicitly listed as the primary cause will be represented in the EPHT dataset. Rarely is heat listed as a primary

FIGURE 5.14. MINNESOTA CASE COUNTS OF HEAT-RELATED HOSPITALIZATIONS AND EMERGENCY DEPARTMENT (ED) VISITS FROM 2000-2011. DATA SOURCE: MINNESOTA PUBLIC HEALTH DATA ACCESS PORTAL (WWW.HEALTH.STATE.MN.US/DIVS/HPCD/TRACKING/DATA/INDEX.HTML), MINNESOTA TRACKING, MINNESOTA DEPARTMENT OF HEALTH.



cause of death on death certificates, given that it is difficult to establish if environmental conditions are not witnessed directly by a physician or emergency care provider. Cases where heat was a significant casual factor but were not coded as directly attributable to heat will not be represented in the EPHT dataset. Second, in all cases, personal identifiers are stripped from the data, which comes from the Minnesota Mortality Database and the Minnesota Hospital Discharge Data. Without any identifiers it is not possible to determine if an individual is receiving care at more than one facility and therefore could potentially be represented more than once in the dataset. Finally, only Minnesota resident data are included in the dataset. This excludes out-of-state visitors who may nonetheless have been treated or admitted for heat-related illness associated with an extreme heat event in the state. Regardless of the limitations, EPHT data provide an essential snapshot of heat-related health effects in the state, which assists state and local public health departments with planning and evaluating response and prevention efforts.

INDIRECT HEALTH IMPACTS

While the most documented impact of heat events is their potential to directly degrade physical health, extreme heat also can have significant indirect consequences on the health and vitality of communities (PWC, 2011), including infrastructure failures, strain on essential services, and disruption to key social and economic networks.

INFRASTRUCTURE FAILURES

More frequent heat waves will mean more demand for energy, primarily for cooling industrial equipment and indoor spaces. At the same time, heat waves can significantly compromise energy supply by reducing the efficiency of power plants and transmission throughout the grid. Increased air and water temperatures challenge the cooling capacity of power plants, and accompanying drought conditions can reduce the amount of water available for power generation. Minnesota depends heavily on adequate water supplies for power generation. Ensuring the stability of the electric grid is difficult since, unlike natural gas and petroleum, electricity cannot be stored cost-effectively. At any given moment, there must be enough electric generation and transmission capacity available to ensure on-demand service. Especially when high temperatures persist overnight, the likelihood of power outages increases. According to some estimates, climate change could increase the need for additional electric generating capacity by 10-20% by mid-century, based on a 6 to 9°F gain in temperature (CCSP, 2007). For the Midwest, summer peak power demand is expected to increase at an average rate of 1.24 percent per year during 2010-2019 (DOC, 2012). The impact of outages, especially multiple outages across a city, can extend well beyond the energy sector, affecting communication systems, utilities, transportation, food safety, and essential health services.

Air conditioning use, with its substantial energy costs, is increasing alongside temperatures and heat waves, and can be as much a contributor to climate change as it is an adaptive measure. Given that 87 percent of households are equipped with air-conditioning, the U.S. already uses more energy for air-conditioning than all other nations combined (Sivak, 2013). However, the penalty for relying too heavily on cooling systems to escape the impact of prolonged heat waves is evident in the experience of developing nations that are rapidly adopting this technology. India's massive energy blackout during the summer of 2012 left 600 million people in the dark and was associated, in part, with the country's increasing use of air conditioning (Lundgren & Kjellstrom, 2013). Developed nations are also at risk for blackouts from high energy demand: A 2009 heat wave in Victoria, Australia led to rolling blackouts that left 500,000 people without power, mainly attributed to record demand for electricity for air conditioning and refrigeration (PWC, 2011).

Population growth in Minnesota will magnify current energy demand, especially in the Twin Cities region and other metro areas. Currently, Minnesota has below average electricity prices compared to the rest of the nation. However, if demand for energy continues to push against existing supply, rates are likely to go up. Low-income households will face tough decisions about budgeting for utility services alongside necessities like food, medicine, health care, and transportation. Individuals with access to air conditioning may decide not to run these systems because they are unaffordable, yet are proven to be life-saving, especially for the elderly (Ostro et al., 2010; Theocharis et al., 2013).

New York City Blackout, Post Hurricane Sandy, 2012
Source: Wikimedia Commons, David Shankbone

STRAIN ON ESSENTIAL SERVICES

During periods of high temperatures, there is increased demand for ambulance and hospital emergency services (Thornes et al. 2014; Kue & Dyer, 2013). Extensive documentation of the Chicago heat wave of 1995 showed that the capacity of emergency services to meet sudden increases in demand is often significantly compromised during an extreme heat event, resulting in extended response times, inadequate care, and a potential for increased morbidity and mortality (Klinenberg, 2002; Palecki, 2001). In addition, there is substantial evidence to suggest a link between heat and crime, such as homicide, suicide, sexual assault, and domestic abuse (Bushman et al., 2005; Hsiang et al., 2013; Lin et al., 2008; McLean, 2007; Mares et al., 2013). Increased crime and violence places additional strain on law enforcement and may lead to delayed response times by available officers.

Photo courtesy of Microsoft Clip Art, 2014.

DISRUPTION TO KEY SOCIAL AND ECONOMIC NETWORKS

Extreme heat events may disrupt certain sectors of employment and even compromise long-term viability of some occupations. Construction, agriculture, forestry, mining and other outdoor occupations are likely to be disrupted in the short term as employee productivity decreases and the risk of adverse health consequences increases (Houser et al., 2014). In addition, lost labor hours may have negative economic impacts for affected employees, while worker absenteeism may lead to interruptions in service or production sectors. In the long term, farming and other jobs dependent on agriculture, may become far less secure as high heat events and accompanying drought conditions lead to crop losses and threats to animal welfare (Walthall et al., 2012). Extreme heat events also lead to school and event cancellations, which may reduce opportunities for community cohesion and physical activity. For example, the August 2013 heat wave in Minneapolis led to classes and many athletic events being cancelled at more than 25 schools over multiple days (Littlefield, 2013).

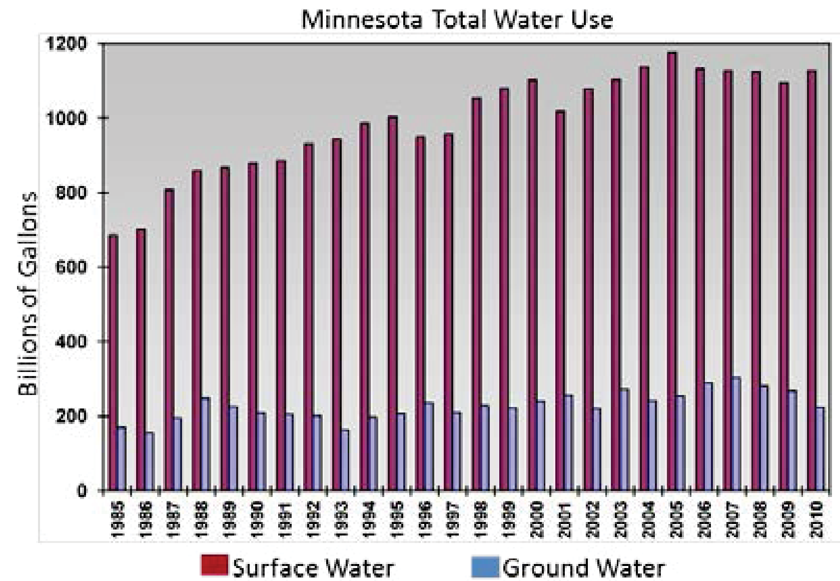
Assessment of Health Risk from Extreme Heat	
Hazard	There is evidence showing that extreme heat events have increased in Minnesota. In addition, climate projections suggest that the frequency, duration and severity of heat waves will increase over the current century. Both heat and humidity can adversely impact health.
Exposure	During an extreme heat event, any person outside of a temperature-controlled indoor environment is exposed. However, characteristics of exposure will vary widely depending on the individual. Those who work or play outdoors for long periods of time during a heat wave sustain longer exposures to heat. Exposure can be amplified with improper clothing or poor hydration. Prevalence data on individuals that have been exposed to high heat long enough to develop symptoms are estimated by Minnesota's EPHT system. However, it is likely that many people who are exposed to high heat and are adversely affected are missing from existing data collection or public health monitoring systems because they either do not seek medical attention, or heat is not included as a factor in diagnosis.
Vulnerability	Environment: Approximately half of Minnesota is farmland. Agriculture is a fundamental part of the state's economy, sustaining numerous rural communities. Heat waves can drastically cut down crop yields, stress or kill livestock, and require large amounts of water for irrigation and electricity for cooling. Thus, agricultural land is particularly vulnerable to the effects of extreme heat. In addition, aspects of the built environment, such as utilities that may shut down or roads that can buckle, may also be considered vulnerable to extreme heat. Humans: Because of a reduced capacity to physiologically adjust to rising core temperatures, children, the elderly and people with pre-existing medical conditions are particularly vulnerable to the effects of extreme heat, as are individuals without access to air conditioning or the ability to travel to cooling centers. In addition, due to long exposure times, otherwise healthy people who work or exercise outdoors during a heat wave are also vulnerable.
Risk	Both historical climate trends and future projections demonstrate that extreme heat will continue to be a significant health concern for Minnesotans. The consequences for individuals and communities are likely to be substantial and diverse, ranging from a rise in morbidity to reductions in crop yields, all of which have direct or indirect impacts on public health. Currently, there are only limited data available, mainly through Minnesota's EPHT system, to characterize the number of and manner in which people are affected annually by the rise in heat waves. However, given the many studies that have been conducted on heat waves both nationally and abroad, especially in the last couple decades, public health practitioners in Minnesota have a large amount of data to draw from to characterize climate-mediated risk from extreme heat and implement proactive measures.

Floods & Drought

Minnesota benefits from more freshwater than any other of the 48 contiguous U.S. states, serving the needs of many competing sectors. The viability of Minnesota's industries, farms, utilities, and municipalities hinge on adequate provision of clean water through controlled, reliable systems. Across all sectors, water consumption in the state is steadily increasing (Figure 5.15). Overall water use has risen from about 700 billion gallons per year in the mid-1980s (when electronic data tracking began) to well over one trillion gallons per year in 2010. Surface water provides nearly 80 percent of Minnesota's total water needs. However, the majority of public and private drinking water comes from groundwater sources. Minnesotans that rely on public water systems consume nearly 200 billion gallons of water every year (DNR, 2014). At this time, given the absence of state or federal monitoring, it is not possible to gauge the amount of water consumed through private wells.

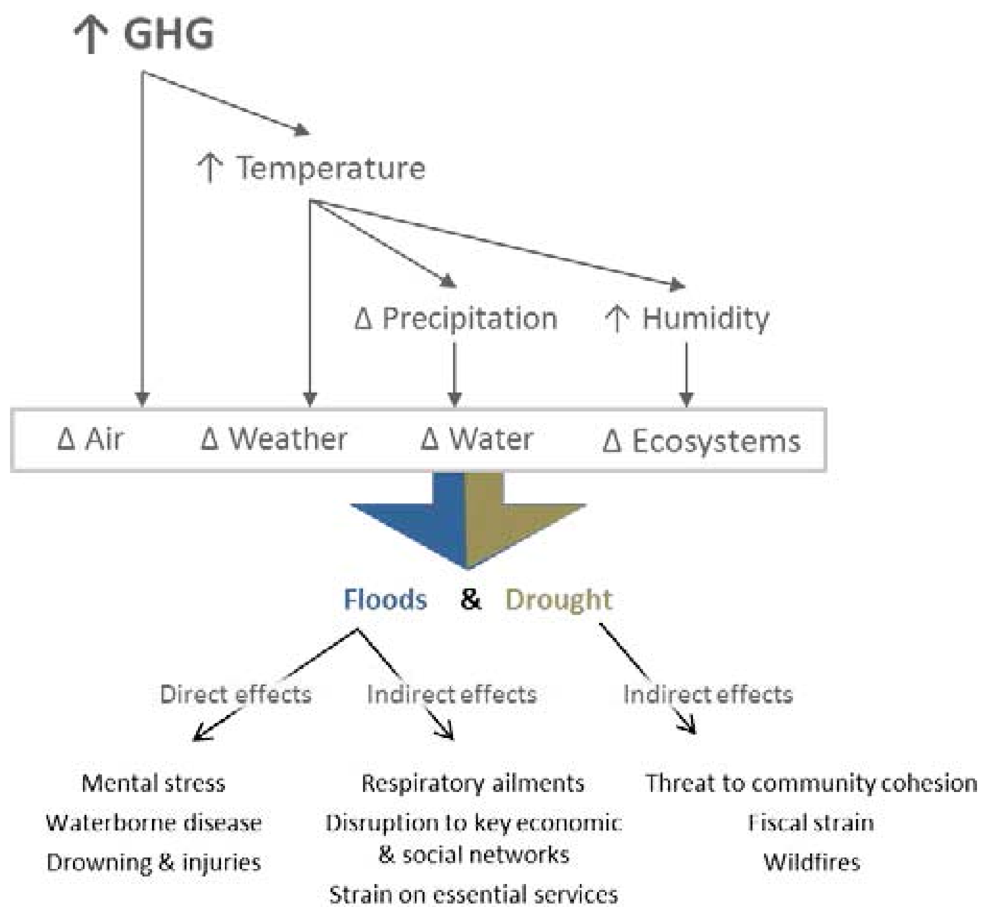
Climate-related changes in extreme weather and precipitation patterns will likely threaten existing water systems in Minnesota and significantly disrupt the hydrologic cycle. Climate change is expected to affect the frequency, intensity, and duration of extreme weather events such as excessive rainfall, storm surges and drought (Kunkel et al., 2013). Altered pressure and temperature patterns along with acceleration of atmospheric warming will shift the distribution of when and where extreme weather events occur.

FIGURE 5.15. MINNESOTA TOTAL WATER USE, 1985-2010. IMAGE SOURCE: DNR, 2014.



This section will focus on the health impacts related to both flood and drought (Figure 5.16), disparate consequences of climate change that share the same mechanism of effect, disruption of the hydrologic cycle that is essential for human life. In addition, floods and drought also contribute to other climate change impacts, such as air contamination and the spread of disease-carrying insects, which in turn have direct and indirect effects on the health of Minnesotans.

FIGURE 5.16. LINKS BETWEEN THE RISE IN ATMOSPHERIC GREENHOUSE GASES, CHANGES TO THE EARTH'S CLIMATE, AND DIRECT AND INDIRECT IMPACTS ON HEALTH FROM FLOODS AND DROUGHT. WILDFIRES WILL NOT BE ADDRESSED IN THIS REPORT BUT MAY BE ADDED IN FUTURE ITERATIONS.



MCEA Comments Ex. 20

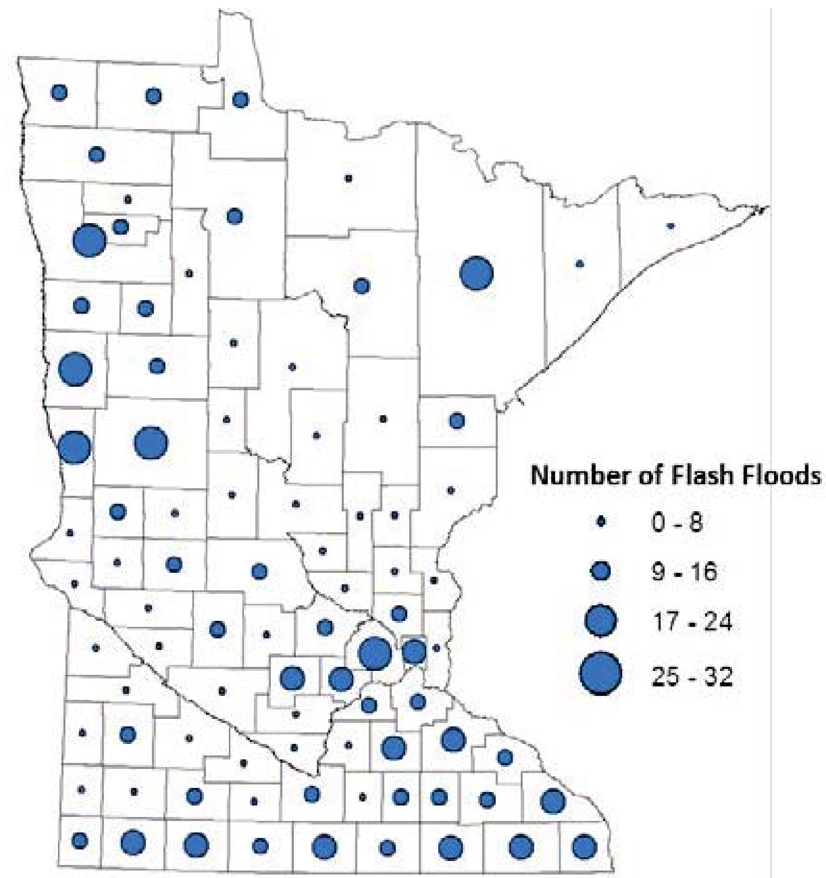
FLOODS

The number of large storms occurring across the Midwest has been increasing over the last half century and by some estimates are more frequent in the Midwest than other areas of the nation (Saunders et al., 2012). Extreme precipitation events that were previously rare, occurring once in 20 years, are projected to become more frequent in the future (Pryor et al., 2014). These storms will increase the risk of major flooding across many parts of Minnesota. Snowpack also is an important contributor to flood risk through earlier melt-off and changes in the rain-to-snow ratio (Peterson et al., 2013).

DROWNING & INJURIES

Floods are common, deadly, and expensive natural disasters (Alderman et al., 2012). In the U.S., floods have been directly responsible for nearly 750 deaths over the last ten years (NOAA/NWS, 2014). Flash floods, characterized by high-velocity flows and short warning times, are responsible for the majority of flood deaths in developed countries (Jonkman, 2005). According to the NOAA Storm Events Database, Minnesota has suffered over 860 flash flood events since 1996, the first year with data available (Figure 5.17). Nearly a third of these events have occurred in just the last few years (2010-2013) (NOAA/NCDC, 2014b).

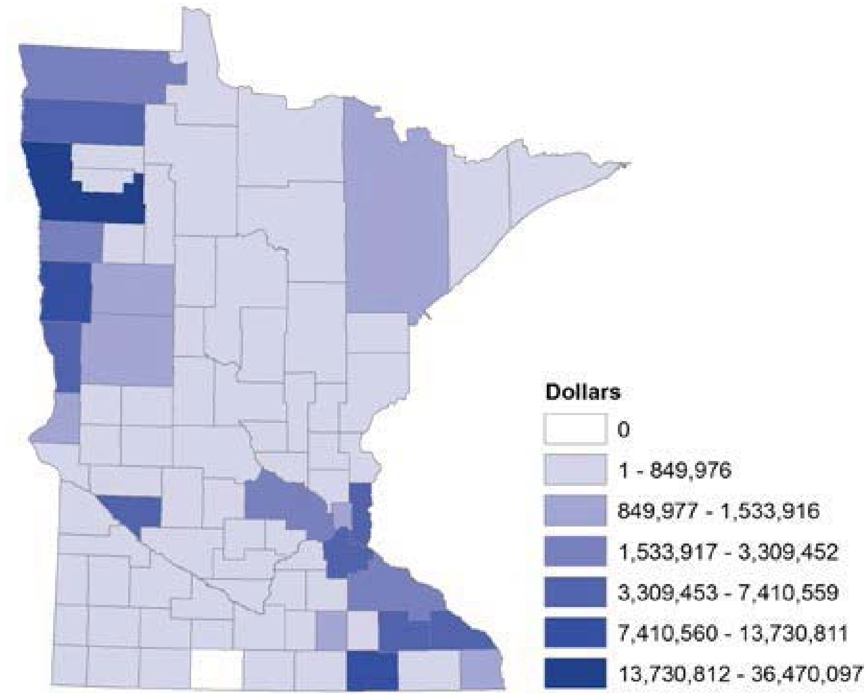
FIGURE 5.17. NUMBER OF FLASH FLOODS IN MINNESOTA PER COUNTY, 1996 – 2013. BLUE CIRCLES REPRESENTING FLOOD COUNTS ARE CENTRALLY LOCATED WITHIN EACH COUNTY AND DO NOT REPRESENT THE LOCATION OF THE AFFECTED TOWN OR CITY. DATA SOURCE: NOAA/NCDC, 2014B.



Like most other extreme weather events, the impact of a flood is typically measured in lives lost and the dollar value of damaged property (Perera et al., 2012). All together, Minnesota flash floods from 1996-2013 are directly responsible for 13 deaths, nearly 13 million dollars in crop losses, and over 314 million dollars in property damage (NOAA/NCDC, 2014b). NOAA does not provide estimates of damage costs to public property, such as roads and utilities, which can be substantial. Damage assessed to utilities, streets, parks and trails in the city of Duluth from a single severe flood event in 2012 was estimated at over 100 million dollars based on agency and industry estimates (Schwartz, 2012).

Assessing the full spectrum of costs associated with a flood is difficult, given that damage can be extensive and diverse. Data from the National Flood Insurance Program (NFIP), administered through the Federal Emergency Management Agency (FEMA), provides another means of characterizing the cost of floods in Minnesota. NFIP data show that between 1978-2013 nearly 11,000 flood claims were submitted from Minnesota property owners, placing Minnesota 11th among all landlocked states for total reported flood losses (FEMA/NFIP, 2014). More than 136 million dollars were paid out to affected Minnesotans from the NFIP, with over 50 percent of flood insurance payments going to losses in just four counties: Polk, Clay, Mower, and Wilkin (Figure 5.18).

FIGURE 5.18. NATIONAL FLOOD INSURANCE PROGRAM (NFIP) FLOOD LOSS PAYMENTS FOR INDIVIDUAL MINNESOTA COUNTIES, 1973-2013. DATA SOURCE: FEMA/NFIP, 2014.



MCEA Comments Ex. 20

Millions of American property owners, including many Minnesotans, get subsidized flood insurance from the federal government through the FNIP. At the end of 2012, the NFIP sponsored 5.5 million policies nationally for a total insured value of 1.3 trillion dollars (Akabas et al., 2014). Large government payouts in the wake of hurricanes Katrina and Sandy have caused the NFIP to cover huge property losses and as a result the program went into debt for approximately 24 billion dollars to the U.S. Treasury (Akabas et al., 2014). This is a concern given the number of people who depend on the FNIP to help with mediating the financial and emotional strain of flooding, coupled with predictions that flood events will become more frequent.

The cost of homeowners insurance has increased dramatically for Minnesota residents in the last two decades, in part due to repercussions from numerous storm and flooding events. For example, three major storms struck Minnesota in 1998, and insurers paid out more than 1.5 billion dollars in storm losses that year, more than was paid in the previous 40 years combined (FEMA/NFIP, 2014). In 2007 and 2008, Minnesota had the second and third highest disaster losses, respectively, in the nation (Johnson, 2012). From 1997 to 2011, average insurance premiums for Minnesota homeowners rose approximately 286 percent, well over the average percent increase for the nation as a whole (214 percent) (Figure 5.19).

FIGURE 5.19. A COMPARISON OF MINNESOTA'S AVERAGE HOMEOWNERS INSURANCE PREMIUM AND COMPARATIVE RANKING WITH THE NATIONAL AVERAGE, 1997-2011. DATA SOURCE: INSURANCE FEDERATION OF MINNESOTA, DEC. 2013.

Minnesota vs National Average for Homeowners Annual Insurance Premiums

Year	MN Average Premium	National Ranking	National Average
1997	$368	35th	$455
1998	*	*	*
1999	$390	37th	$487
2000	$420	35th	$508
2001	$464	28th	$536
2002	$590	18th	$593
2003	$733	10th	$668
2004	$767	17th	$729
2005	$790	18th	$764
2006	*	*	*
2007	*	*	*
2008	$845	14th	$791
2009	$919	14th	$880
2010	$981	14th	$909
2011	$1,056	14th	$978

*Data unavailable

There are a number of problems with relying solely on mortality statistics and damage estimates to fully characterize flood impacts to society (NOAA/NWS, 2014b). First, flood events associated with high levels of mortality, especially in developed countries, are rare (Doocy et al., 2013). Despite the increased frequency of flood events worldwide, flood trends show a decrease in the average number of deaths per event, but an increase in the size of affected populations (Doocy et al., 2013). Flood fatalities alone provide little detail of the full range of impacts from a flood event in a community. Second, flood damage estimates are reported in many different ways and are subject to a wide variety of errors. Currently, there is no one governmental agency that has specific responsibility for collecting and evaluating detailed flood loss information. Estimates can come from federal, state, county and city level officials (NOAA/NWS, 2014b), even the media. However, the amount and type of media coverage are not necessarily proportional to the size of a flood event (WHO, 2013). In addition, damages are often underreported, in part due to a lack of post-event follow up with affected individuals and the absence of a centralized system for collecting these data.

Finally, damage estimates are as much a reflection of extreme weather events as they are of land use decisions and wealth. Increased urbanization in areas where flooding may occur means that there are more homes, highways, utilities, and other infrastructure and property that can be damaged (Du et al., 2010). If flooding occurs in an undeveloped area, it is not likely to be associated with extensive damage costs. Some researchers make the distinction between hydrological floods (which occur in unpopulated areas and may not be linked to damages) and disasters (which occur in populated areas and can adversely impact a number of socioeconomic systems) (Barredo, 2009). With regard to private flood insurance, an individual with an expensive home and belongings will likely receive a larger damage estimate than an individual with less capital, even though the relative impact of the flood on their health, finances and well-being may be comparable, or even greater for the less advantaged person. A number of researchers argue that flood loss estimates are, in effect, a reflection of urbanization and assets and are more closely related to a population's standard of living than with flood severity, thus limiting their usefulness (Barredo, 2009; Pielke, 1999). Rarely do flood damage estimates take into account the costs incurred from impacts to human health (beyond mortality and limited injury statistics), ecosystem services (such as clean water and air), or quality of life (WHO, 2013).

Therefore, it's important to acknowledge that the full extent to which a flood impacts health is determined not just by the magnitude of the flood, but also by individual and societal factors, such as socio-economic and demographic characteristics, expanding development into flood-prone areas, neglecting maintenance to key infrastructure (e.g., roads, railways, drinking water and sewage systems), and modifications to waterways (Du et al., 2010; Lowe et al, 2013).

Specific health impacts of flooding can be divided into those associated with the immediate event (direct effects) and those arising in the aftermath (indirect effects). Immediate, direct effects are caused by primary exposure to floodwaters and the debris it contains (e.g. injury, mental stress, waterborne disease), but a flood continues to adversely impact health during the recovery and rebuilding process, which may continue for months to years. These effects are less easily identified and quantified with currently available data compared to fatalities and direct injuries. Examples of indirect health impacts associated with floods include respiratory distress or disease from exposure to indoor air contaminants, exacerbation of existing medical conditions, and disrupted health services. These impacts are likely to be relevant for Minnesotans and will be covered in more detail in the rest of the section.

Minnesota National Guard Soldiers drive through rushing water down Minnesota Highway 1 into Oslo on April 16, 2011. Source: Wikimedia Commons, Tech. Sgt. Erik Gudmundson.

MCEA Comments Ex. 20

WATERBORNE DISEASE

Flood events can threaten the safety and availability of drinking water by washing biological and chemical contamination into source water or by overwhelming the capacity of treatment systems to clean the water. Power shortages stemming from storm- and flood-damaged utility lines also can disable treatment systems rendering water unpotable. The repercussions can extend beyond the flood-afflicted area and impact the entire population served by the damaged systems.

The full extent of floodwater contamination depends on land use and associated infrastructure in the affected area. Examples of common floodwater contaminants include the following:

- pesticides, synthetic fertilizers, and manure from agricultural runoff;
- pathogens and pharmaceutical residues from human waste and other domestic chemicals in sewer overflow; and
- heavy metals, petroleum hydrocarbons and polycyclic aromatic hydrocarbons from roads, parking lots and other impervious surfaces.

In addition, floodwaters that mix with acid mine drainage, landfill leachate or releases from industrial plants and waste storage facilities can introduce highly toxic, long-term contamination to both surface and groundwater drinking water sources that is challenging to remediate and a health hazard at nearly any level of exposure. Naturally-occurring micro-organisms in soils or foreign pathogens introduced by manure or sewage can wash into drinking water. Floodwaters may contain over 100 types of disease-causing bacteria, viruses, and parasites, which can cause serious gastrointestinal illness or even death in highly vulnerable individuals (Perera et al., 2012). Extreme precipitation events and outbreaks of waterborne disease are strongly linked. Very heavy rainfall events (defined as those in the top seven percent) more than double the probability of a waterborne disease outbreak (Perera et al., 2012). One Wisconsin study based on children's emergency room visits found an 11 percent increase in gastrointestinal illness in the days immediately following intense rainfall (Drayna et al., 2010). Given that the observed increase in visits occurred in the absence of any reported outbreaks to public health officials underscores that the prevalence of illness associated with heavy rainfall is underestimated, an issue that has been widely acknowledged, but is difficult to address (Cann et al., 2012).

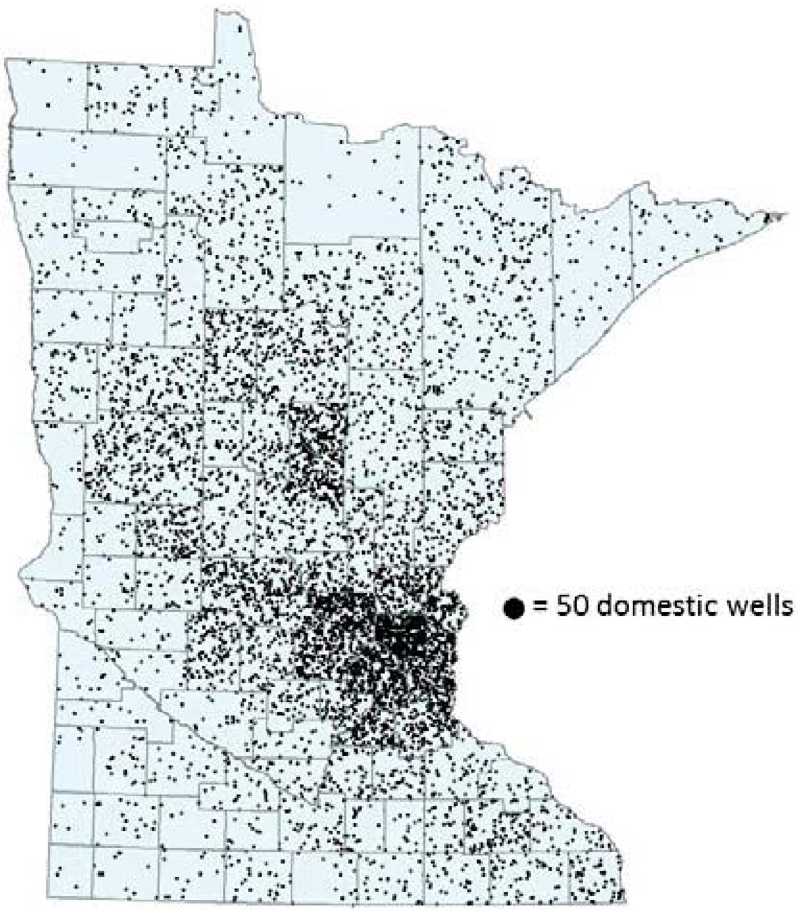
1993 Milwaukee Cryptosporidiosis outbreak

The largest epidemic of waterborne disease reported in U.S. history occurred in Milwaukee, Wisconsin and was linked to the heaviest rainfall in 50 years in area watersheds (Curriero et al., 2001). Rivers bloated by heavy spring rains and snow runoff transported *Crypotsporidum* oocysts into Lake Michigan. From there, these pathogens entered the intake of a major treatment plant that supplied water to residences and businesses in the city and nine surrounding municipalities. Potential sources of the oocysts included cattle along rivers that flow into the Milwaukee harbor, slaughterhouses, and human sewage from overflowing sewers. An estimated 403,000 residents and numerous visitors to the area were sickened by the pathogen and 58 residents died (Hoxie et al., 1997; Mac Kenzie et al., 1994). The effects were broad and lingering: *Cryptosporidum* outbreaks directly associated with Milwaukee were reported in other areas of the country as affected residents traveled on spring break and swam in pools where chlorine cannot kill the pathogen (Ellis, 2007). Serum samples of Milwaukee children confirm that cryptosporidiosis antibody rates had jumped from 10 percent before the outbreak to 80 percent afterwards, suggesting that there was extensive asymptomatic infection (Ellis, 2007). CDC researchers estimate that the total medical costs and productivity losses associated with the Milwaukee outbreak ranged from 75 to 118 million dollars (Corso et al., 2003).

Both public and private drinking water systems (such as a household well) are affected by flooding. While contaminated public water supplies carry the risk of poisoning large numbers of people, these systems benefit from regular monitoring. When contamination is detected warning networks can alert customers to use bottled or boiled water until the system is repaired. However, private wells are rarely monitored as closely as public systems since well owners are responsible for testing and treating their water as needed to maintain potability. A number of studies suggest that most well owners do not adequately test or treat their water as recommended by health-based guidelines (Flanagan et al., 2014; Roche et al., 2013). Private well owners therefore represent another vulnerable population for waterborne illness from flooding events. This is particularly relevant for Minnesota given that nearly one million residents rely on a private well for household water (Figure 5.20).

The structural integrity and power supply of a private well can be significantly damaged during a flood, and floodwaters can wash surface pollutants directly into the well, not only posing a hazard for the household but potentially contaminating the source aquifer (Wallender et al., 2013). Groundwater contamination can be very persistent, depending on the contaminant. Nitrate from agricultural fertilizers and manure can last for decades under the right conditions (Dubrovsky et al., 2010). Ample groundwater plays a major role in climate resilience by expanding potential water reserves but is increasingly threatened as farmers, municipalities, industry and other sectors scramble to adjust to abrupt strains of drought. Given the increasing reliance on aquifers to address the burgeoning demands for water in the state, groundwater contamination is a pressing concern for Minnesota, as it is for most states across the nation (Freshwater Society, 2013). Flooding therefore is a threat not only to the immediate potability of drinking water for affected Minnesotans but to the long-term availability of clean water for generations to come.

FIGURE 5.20. APPROXIMATE LOCATIONS OF MINNESOTA PRIVATE DRINKING WATER WELLS THAT PROVIDE HOUSEHOLD DRINKING WATER TO NEARLY ONE MILLION RESIDENTS. INDIVIDUAL DOTS REPRESENT APPROXIMATE LOCATION OF AREA WELLS. SOURCE DATA COURTESY OF THE MDH WELL MANAGEMENT PROGRAM.



MENTAL STRESS

According to a report by the Climate Institute, one in five people suffers from the effects of psychological stress after a severe weather event (Climate Institute, 2011). Some level of mental stress is common during and after a flood. Emotional responses like tearfulness, numbness, anger, or insomnia are common (WHO, 2013). However, these symptoms can persist and disrupt an individual's ability to fulfill their responsibilities and be present in their daily life. Post-traumatic stress disorder (PTSD) is the most common mental health disorder found in people affected by natural disasters like floods, followed by depression and anxiety (Heo et al., 2008; Liu et al., 2006; Mason et al., 2010). These conditions and the toll they exact on victims' lives can last for months or even years after a flood, as those affected continue to struggle with the loss of their belongings, damage to their homes, disruption to work, school and social involvement, or complete displacement from their community. Studies of flood impacts report prevalence of mental health disorders ranging from approximately nine percent (Liu et al., 2006) to 53 percent (Heo et al., 2008) in the first two years following floods. One study of psychosocial outcomes in those badly affected by floods showed that the prevalence of adverse mental health symptoms was two to five times higher among people who reported floodwater in their home than in people who did not (Paranjothy et al., 2011). A study on a large population of Hurricane Katrina flood victims in the U.S. showed significant racial and gender difference in psychological impacts, such as sleeplessness, anxiety and depression (Adeola, 2009).

In the aftermath of the Midwest floods of 1993, rates of domestic violence and alcohol abuse, manifestations of extreme stress and mental health problems, were all elevated in affected communities (Axelrod et al., 1994). A study on the 1997 floods in North Dakota suggested that impacted women experienced worse pregnancy outcomes post-disaster than pre-disaster, with the authors attributing this result to stress and anxiety associated with the event (Tong et al., 2011).

A resident carries some of the contents of her home. Rushford, MN, 2007. Photo courtesy of Wikimedia Commons.

The increased use of mental health services following a flood contributes to its overall economic cost, but is often not included in broad estimates of damage. Studies of severe flooding in the U.K. demonstrated that 90 percent of associated public health costs were devoted to mental health problems (UKEA, 2010). Flooding in France in 2002 resulted in a net increase in psychotropic drugs, with a cost to insurance of approximately 375,000 dollars (WHO, 2013).

Substantial evidence from the scientific literature indicates that a major factor in the experience and persistence of mental stress amongst flood victims is how people are treated by the organizations they interact with during the recovery phase (WHO, 2013). Examples of these organizations include builders, contractors, utility companies and insurance adjusters. In particular, a number of studies underscore that financial losses and frustrations with negotiating insurance payments for rebuilding are highly correlated with the mental stress experienced by flood victims (Bei et al., 2013; Paranjothy et al., 2011). A report from the UK on the experience of households flooded over the period 1998-2000 found that "problems with insurers" was a major predictor of psychological distress and PTSD at two to four years after flooding (DEFRA/UKEA).

RESPIRATORY AILMENTS

Flood events can indirectly lead to indoor inhalation exposures that can adversely impact health both immediately following the event as well as years after the waters have receded. Two major indoor air contaminants associated with flooding are described here, carbon monoxide and mold.

Carbon Monoxide

Power outages are common following a major flood event. Inappropriate use of portable generators or indoor stoves and grills during power outages increases the risk for carbon monoxide (CO) poisoning (Waite et al., 2014). If the area around the generator is not adequately ventilated, this odorless, colorless gas can accumulate without notice, leading to a range of health effects from fatigue and headache to cardiorespiratory failure, coma or even death. Even a generator located outside of a building but near an open window can pose a threat. A number of recent studies help to characterize the risk of CO exposure from the use of power generators following natural disasters, like floods. According to a CDC report, 263 CO exposures had been reported to poison centers in eight states following Hurricane Sandy, four of which were fatal (CDC, 2012). CDC states that this is likely an underestimation of fatalities, given that larger CO-related deaths were reported in the media. A review of the scientific literature found that between 1991-2009 there were reports of 1,888 cases of disaster-related CO poisoning cases in the U.S., including 75 fatalities (Iqbal et al., 2012). Generators were the primary exposure source for both fatal and nonfatal cases, and the majority of all cases occurred within three days of disaster onset. A review of hospital records from ten Florida hospitals following an active hurricane season with widespread power outages found that 167 people had been treated for nonfatal CO poisoning and six had died (Van Sickle et al. 2007). A portable, gasoline-powered generator was implicated in nearly all cases and were most often located outdoors.

Mold

A range of factors contribute to mold and microbial growth in buildings, yet dampness—which can persist for weeks to months—provides a near optimal breeding ground (Brandt et al., 2006). Numerous studies show that a significant growth of indoor mold occurs following floods (Alderman, 2012). People that continue to live in damp buildings after flood events, especially for long periods of time, are at risk for health problems (WHO, 2013). Adverse health effects associated with exposure to indoor mold are mainly respiratory, but also may include irritation of the eyes and skin (WHO, 2013). Occupants of mold-affected buildings are burdened with a higher prevalence of respiratory infections and a higher asthma risk (Quansah et al., 2012; Thorn et al., 2001). A recent comprehensive literature review found that upper respiratory tract symptoms, such as coughing or wheezing, and asthma exacerbation are associated with dampness and poor indoor air quality (Mendell et al. 2011). A New Orleans study reported respiratory symptoms in people living in water-damaged homes long after hurricanes flooded the area (Cummings et al., 2008). Symptom scores increased linearly with exposure. Findings from the study indicate that any exposure to water-damaged homes results in a greater risk for upper and lower respiratory tract symptoms and this risk persists for at least 6 months after a flood (Cummings et al., 2008). Of particular concern for children are recent studies supporting a casual link between dampness, mold and the development of asthma. For example, infants and children, particularly those of low socioeconomic status, exposed to mold before one year of age are up to four times more likely to develop asthma than unexposed peers (Mendell et al. 2011).

After flood waters recede, dangerous mold spores can begin to grow within 24 to 48 hours on a variety of household surfaces. Photo courtesy of Wikimedia Commons.

DISRUPTION OF ESSENTIAL SERVICES

Depending on the extent and severity, floods can hinder a person's ability to travel to the grocery store to buy food, the pharmacy to pick up required medications, and clinics for medical, lab or other therapy appointments. Flooding also can keep providers of essential services from reaching vulnerable individuals, such as in-home therapists or caregivers (along with in-home meal deliveries), paramedics, fire-fighters and police officers. Flooding and damage to hospitals, clinics, and key businesses also can mean that individuals are left without access to their necessary services, perhaps at a time when they are especially needed. Evacuation and displacement during flooding pull individuals out of their established support networks, and it can be challenging to recreate or repair those networks after recovery. Given the wide range of repercussions and how uniquely any one individual may be affected, the amount of data is limited, but growing, that can provide some insight into how seriously flood victims are impacted by disrupted essential services (WHO, 2013). For example, a study of dialysis patients in the aftermath of Hurricane Katrina flooding revealed that of 450 evacuated patients, half had missed one dialysis session, while nearly 17 percent had missed three or more (Anderson et al., 2009). A survey of 1,000 people affected by Katrina floods found reduced access to a physician was reported by 41 percent, reduced access to medications by 33 percent and transport problems by 23 percent (HKCAG & Kessler, 2007). A separate study found that loss of essential services was a major risk factor for poor mental health outcomes associated with a flood event, worsening mental health status two- to three-fold (Paranjothy et al., 2011).

The elderly and people with chronic disease conditions are especially vulnerable to disruptions of essential services and support networks, and evidence suggests this vulnerability lasts long after the floodwaters have subsided. One-third of all deaths in areas affected by hurricanes Katrina and Rita occurred in homes that were spared from floodwater (Jonkman et al., 2009). Those fatalities were due to "dehydration/heat stroke, heart attack/stroke, or other causes associated with lack of sustaining medical supplies". Inability to maintain a stable medication schedule was the main barrier to continuity of care for those suffering from chronic conditions during the floods. Similarly during Japan's flood of 2006, the elderly and those on long-term care were more likely to have medications interrupted as a result of flooding, and this interruption caused a four-fold risk of worse health outcomes as compared to patients with continued care (Tomio et al., 2010). Evacuation and displacement can be especially harmful for the elderly. A French study of mortality patterns in nursing home residents that were evacuated following a major flood event found that the number of deaths recorded in the month following was three times higher than the expected number, and two times higher during the second month, compared with facilities in the affected area that were not evacuated (Mantey et al., 2012).

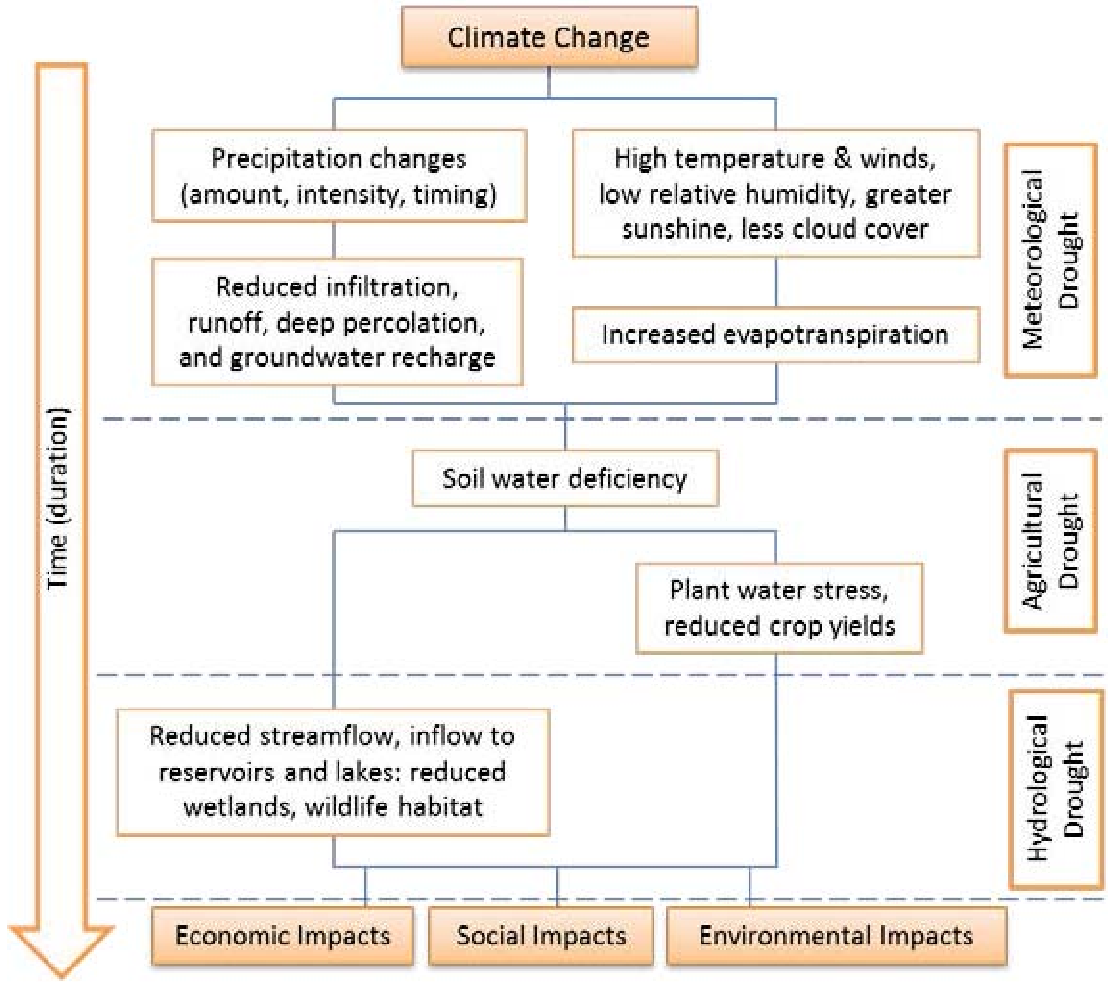
Damage to a road caused by flood water. Photo courtesy of Wikimedia Commons.

MCEA Comments Ex. 20

DROUGHT

While climate change is projected to increase overall precipitation in Minnesota, more of this precipitation is expected to occur during heavy rains and storm events, potentially increasing the rate and duration of intervening dry spells (Seeley, 2007). In addition, evapotranspiration, which is highly seasonal, is expected to increase in the upper Midwest along with warming temperatures, which may further exacerbate drought events (Jackson et al., 2001). In general, drought refers to a scarcity of water, that adversely affects various sectors of society, such as agriculture, energy, municipalities, or industry. However, when observed in more detail, drought means different things to different people, depending on their area of concern (Panu & Sharma, 2002). To the farmer, drought refers to inadequate moisture in the root zone of crops, while the meteorologist views it as precipitation shortfall. To the hydrologist, drought refers to below average water levels in lakes and rivers, while the economist sees it as a resource shortfall that can disrupt the established economy (Palmer, 1965). Wilhite & Glantz (1985) identify four types of drought that capture these different perspectives: meteorological, hydrological, agricultural and socio-economic. The first three types deal with ways to measure drought as a physical phenomenon (Figure 5.21). The last deals with drought in terms of supply and demand, tracking the effects of water shortfall as it impacts socioeconomic systems and is relevant to the major health determinants of concern related to drought for Minnesotans: fiscal strain, loss of livelihood, and threat to community cohesion.

FIGURE 5.21. RELATIONSHIP BETWEEN VARIOUS DROUGHT TYPES, ASSOCIATED FACTORS AND IMPACTS. ADAPTED FROM NDMC, 2014.



FISCAL STRAIN

A growing number of researchers and decision-makers acknowledge the fiscal strain that may impact individuals and communities having to compete with other influential sectors for constrained water resources. Discussing a recent NOAA study on stressed watersheds, co-author Dr. Kristen Averyt states, "By midcentury, we expect to see less reliable surface water supplies in several regions of the United States. This is likely to create growing challenges for agriculture, electrical suppliers and municipalities, as there may be more demand for water and less to go around" (CIRES, 2013). A 2010 modeling study on the future of Minnesota's energy and water resources funded by the Legislative Citizen Commission on Minnesota Resources (LCCMR) found that population growth and increasing demand on electric power generation are two primary factors that will drive increases in future water demand in Minnesota and will make the state significantly more vulnerable to late summer and late winter drought (Suh et al., 2010). Incorporating climate change factors (e.g., precipitation, temperature, humidity, wind speed and solar radiation) into the model framework revealed that water use will be strongly and differentially impacted across the state by climate change.

The cost of potable household drinking water is likely to rise, not only due to marketplace competition but also due to intensive treatment technologies that will be needed to address contamination in less desirable water sources tapped to sustain supply. The likelihood that water rates will increase with increasing demands and reduced source availability is confirmed by the experience of communities across the nation. According to a 2013 report by the Pacific Institute, between 1991 and 2006 California's average monthly charge for 1,500 cubic feet of water increased by more than 8.00 to 41.97 inflation-adjusted dollars. The report recognizes that efforts to adjust to climate changes, such as enlarging existing reservoirs, have been contributing to the rate hikes (Donnelly & Christian-Smith, 2013). Yet, the overall burden will likely be small for most household budgets, at least over the short term, because current water rates for Americans are relatively low. On average, tap water currently costs around two dollars per 1,000 gallons with 15 percent of the cost due to treatment (EPA, 2004). A 2009 study comparing household water bills across the U.S. listed the cost to Minnesotans on public systems as 280 dollars a year, the approximate median for all states considered in the analysis (FWW, 2009).

The more prominent fiscal strain that may be looming for Minnesotans related to increasing water demands and decreasing water supplies may stem from attendant increases in the costs of electricity. The water bill accounts for about 1.5 percent of median U.S. household budgets (Moore et al., 2011). In contrast, based on 2010 data, Minnesota families spent an average of 12 percent of their after-tax income on energy. The 413,000 Minnesota households with annual incomes 10,000 to 30,000 dollars—one-fifth of the state's population—spent an estimated 24 percent of their after-tax family budget on energy (Trisko, 2011). As power companies struggle to obtain adequate water for cooling and power generation, the cost to maintain the power supply likely will be passed on to consumers, posing difficult budget choices among energy and other basic necessities such as food and rent, especially for low income earners.

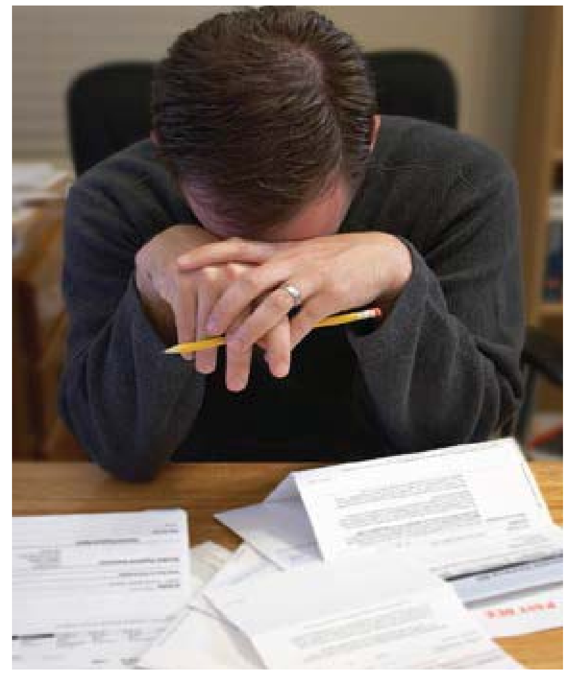
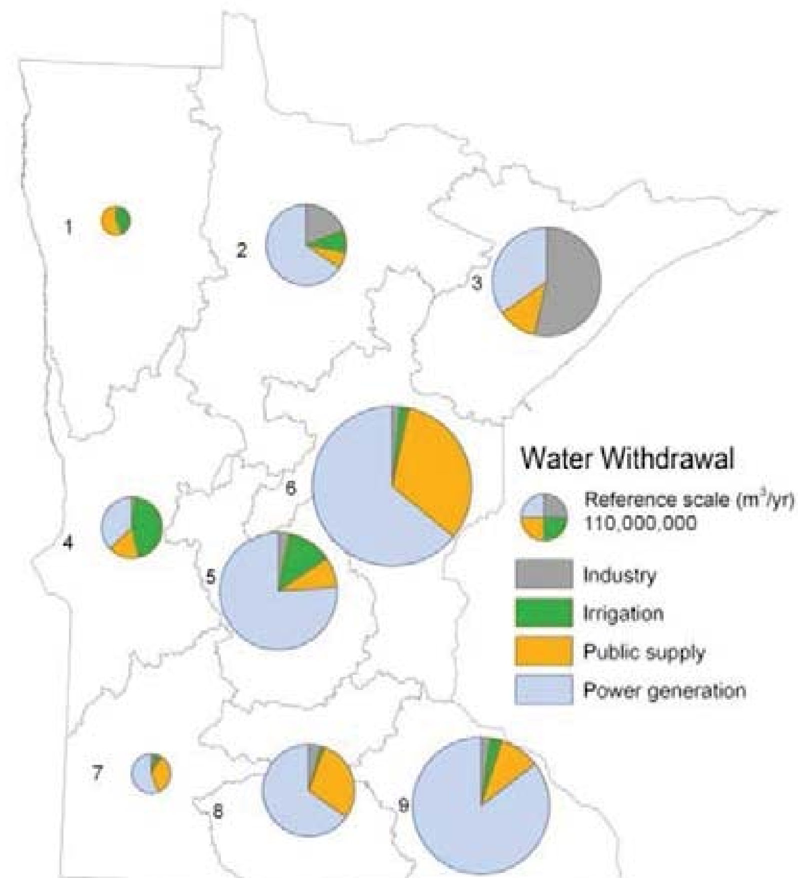


Photo courtesy of Microsoft Office.

By far, power generation has historically accounted for the largest percentage of water use in the state, mainly for cooling power plant equipment. Although power generation use is primarily non-consumptive (i.e., most of the water is returned after use), sufficient source water at a cool temperature still needs to be available to keep systems running. The Fourth Assessment Report of the Intergovernmental Panel on Climate Change synthesized several studies that suggested that future energy generation will be vulnerable in part to reduced availability of cooling water (IPCC, 2007). Modeling results from the LCCMR study show that future energy generation needs will increase the need for water withdrawals across Minnesota (Suh et al., 2010). Different regions of the state will be impacted differently, such that drought occurring in central or southeastern Minnesota may substantially disrupt power supply, given the power industries in these regions (Figure 5.22). There are few studies available that provide estimates for how much source water shortages could impact electricity prices. A study on the European Union found that projections of reduced river flows and higher temperatures associated with future climate change scenarios were associated with price hikes for electricity for over half of the countries studied, especially during the summer season (Van Vliet et al., 2012). Another European study found that models incorporating falling river levels and rising river temperatures implied that the price of electricity will increase by roughly one percent for every degree that river temperatures rise above a 25°C (77°F) threshold (McDermott, 2012). The extent to which electricity costs will rise for Minnesotans as a result of climate-related drought or uneven water supply remains uncertain.

FIGURE 5.22. WATER WITHDRAWAL BASED ON MAJOR CONSUMER CATEGORIES FOR MINNESOTA'S NINE CLIMATE DIVISIONS. IMAGE SOURCE: SUH ET AL., 2010.



LOSS OF LIVELIHOOD & THREAT TO COMMUNITY COHESION

By numerous measures, agriculture and rural life define Minnesota culture. Minnesota is the fifth largest agricultural producer in the nation and the fourth largest agricultural exporting state (Ye, 2014). Minnesota is among the nation's leaders in corn and soybean production, two of the world's most valued crop commodities. With the implementation of new bioenergy policies, Minnesota has become one of the top five bioethanol producers in the U.S. as well (Ye, 2012). Agriculture is the second largest employer in the state and fuels the viability of both state and local economies.

Crop production depends on a balance of chemistry, temperature and precipitation, and climate change threatens this balance. Atmospheric CO_2 loading and the associated impacts on climate, in particular extended dry spells coupled with rising temperatures and heat waves, may significantly compromise farming as a dependable means of earning a living. A recent report for the National Climate Assessment reviewing climate impacts on Midwest agriculture acknowledges that water availability is the dominant climatic factor causing significant decreases in crop yields observable in all Midwest states, including Minnesota (Pryor et al., 2014).

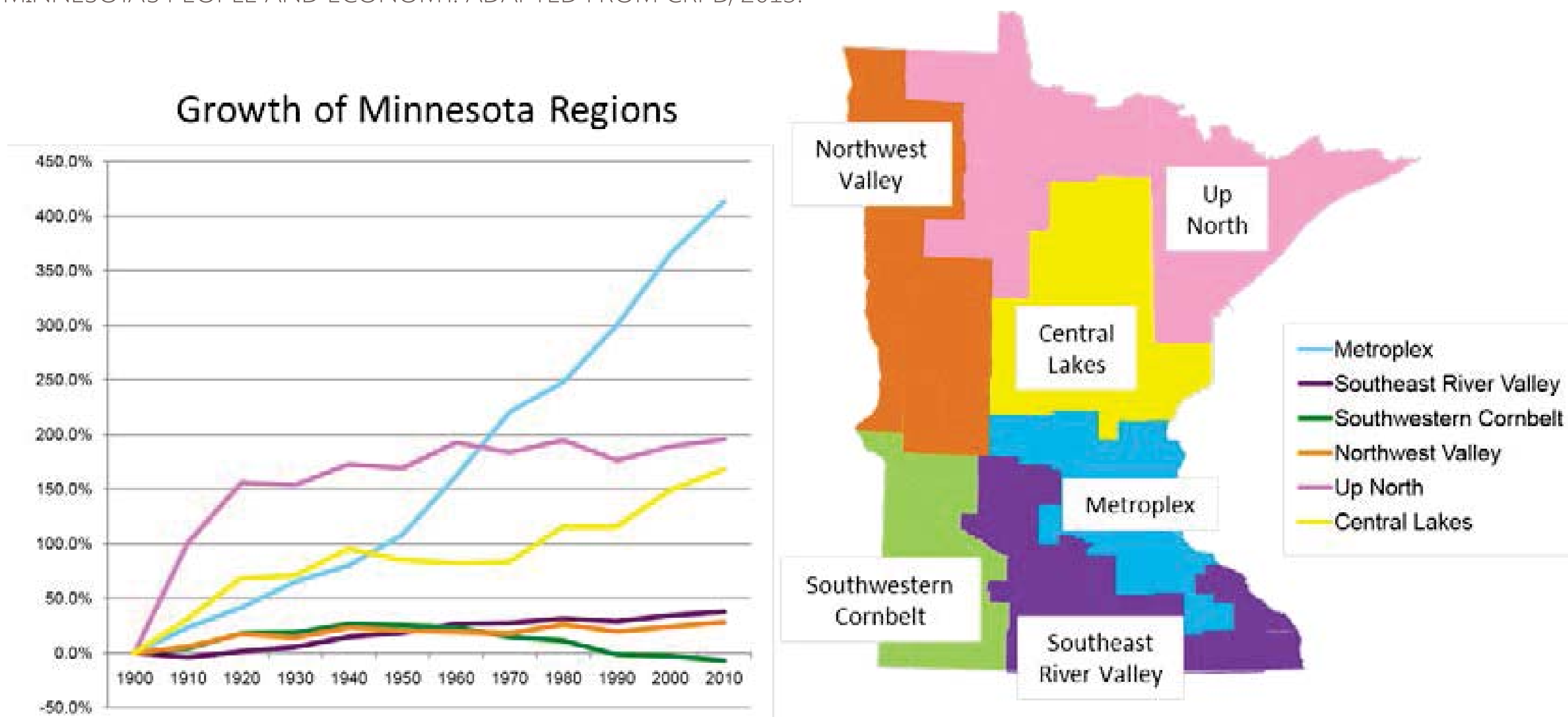
Significant changes to the agricultural landscape have already occurred. Some may seem advantageous, such as the northward shift in plant hardiness zones, which expands the possibilities for some growers. On average, the growing season for the entire Midwest has lengthened by almost two weeks since 1950 (Pryor et al., 2014). However, the long-term potential agricultural consequences of climate change impacts are complex and vary by crop. For corn, long-term temperature increases could shorten the duration of reproductive development, leading to yield declines even when offset by the fertilization effect of higher ambient CO_2 levels (Pryor et al., 2014). Researchers already have observed that changes in temperature from 1980 to 2008 have reduced corn yields by 3.8 percent despite yield gains from technology improvements and CO_2 fertilization (Lobell et al., 2011). For soybean, yields are likely to increase early in the current century due to CO_2fertilization, but these increases will be offset later by high temperature stress (Pryor et al., 2014). In addition, the shift in plant hardiness zones means native species are likely to face increasing threats from pests, diseases and invasive species migrating from the south.

Recent notable events demonstrate the dramatic toll of drought on the state's agricultural economy. Drought was the cause of a decline in Minnesota's 2013 soybean harvest, at a loss of 175 million dollars, due to the resulting crop damage (Steil, 2013). Even though rainfall was excessive early in the crop season, by August dry conditions returned with more than half the state designated in moderate to severe drought. Dairy farmers in the state were also hit hard losing nearly a million acres of alfalfa feed to the weakening effects of the drought. The previous year was perhaps even worse with some calling 2012 the "year of the drought" when corn yields came in well below average (Thiesse, 2012). To adjust to the extended dry spells, more farmers are applying for irrigation permits and tapping into groundwater aquifers (Freshwater Society, 2013). Irrigation wells, which currently supply water for only 3 percent of the state's cropland, on average pump more than 25 percent of the groundwater reported by high-capacity wells in the state. They are the second-biggest user of groundwater and by far the fastest-growing use (Freshwater Society, 2013). With nearly three-quarters of Minnesotan residents relying on groundwater for drinking water, the additional use for crop irrigation is not sustainable, a situation that is widely recognized by resource experts and decision-leaders (Freshwater Society, 2013).

Photo courtesy of Farm Industry News, May 24, 2012: http://bit.ly/1rxN7HC.

Rural Minnesota is already losing a large majority of the next generation of workers and residents to the Twin Cities and suburbs, and the additional uncertainties introduced by climate change may further discourage young people from committing to a livelihood dependent on farming. Over the last century, the majority of Minnesota counties that have experienced the slowest population growth or even losses are those where agriculture is the economic focus (CRPC, 2013; Figure 5.23). As the number of young families and workers dwindle and the percentage of retired elderly rises the damage to community cohesion in Minnesota's rural areas could be significant and difficult to reverse. A population with strong community cohesion sustains numerous opportunities for upward mobility and fosters a widespread sense of belonging and trust for its members. As the state's wealth and workforce shift to the Metro area, there are fewer services and resources (such as property tax revenue and skilled professionals) left in their absence. Schools may have to close or consolidate, a reduced tax base leads to reduced support services, and there are fewer volunteers for fire and rescue squads. The threat to community cohesion when large numbers of people lose access to a dependable livelihood and have to migrate outside of the area may adversely affect the health of those who remain.

FIGURE 5.23. POPULATION GROWTH OF MINNESOTA'S REGIONS, 1900 TO 2010. REGIONS ARE BASED ON COMMON CHARACTERISTICS AND WERE CREATED FOR THE CENTER FOR RURAL POLICY & DEVELOPMENT TO SHOW MAJOR CHARACTERISTICS AND TRENDS FOR MINNESOTA'S PEOPLE AND ECONOMY. ADAPTED FROM CRPD, 2013.



Assessment of Health Risk from Assessment of Health Risk from Floods & Drought	
Hazard	There is historical evidence demonstrating an increase in extreme precipitation events in Minnesota, including heavy storms that lead to flooding and longer periods of intervening dry spells. In addition, climate projections suggest that the frequency, duration and intensity of flooding will increase in the state over the current century. However, it is difficult to predict at this time if the prevalence of drought will also become more frequent and severe. Given that many experts anticipate more frequent and severe heat waves for the Midwest, it seems likely that drought will continue to be a threat to consider, although with increased precipitation some watersheds may benefit from "basin-scale memory", a reference to soil moisture retention.
Exposure	Floods: With nearly 21,000 square miles of the state covered by water, waterfront properties in Minnesota are popular and widespread. A national analysis of "Special Flood Hazard Areas" (areas with a one percent annual chance of flooding) conducted by the CDC in partnership with FEMA found that nearly 160,000 Minnesotans currently reside in a SFHA. This does not take into account seasonal residents, which may be a substantial population given the popularity of cabin properties in the state. While homeowners are the target population of concern, anyone traveling into or out of a flood zone can be exposed to the hazards of a flood. Drought: The focus here has been mainly on drought "exposure" as it impacts the livelihood of farmers and other agricultural workers. The number of people involved in agriculture in the state is substantial: There are nearly 81,000 farms in Minnesota, and an estimated 367,000 jobs associated with agricultural or food-related companies.
Vulnerability	Environment: Low-lying riparian areas are vulnerable to flooding. Results of the CDC-FEMA analysis found that there are over 3,200 square miles of SFHA land in Minnesota. However, urban areas can also be vulnerable to flooding if rainfall exceeds the capacity of existing stormwater infrastructure. With regard to drought, the focus of this section has been on the vulnerability of farmland. The assessment of which crop areas of the state may be especially sensitive to drought depends on a wide range of variables and their interactions. However, two of the "thirstiest" row crops dominate Minnesota agriculture—corn and soybeans—and together cover nearly 16 million acres of the state, which may increase the strain on water sources, especially during dry spells. Humans: The following characteristics have a strong influence on a person's vulnerability to flood impacts: ▪ Residing in a flood-prone area, ▪ Limited mobility or limited access to a car or other means of escaping flood, ▪ Lacking adequate flood insurance or income to address damage, ▪ Dependent on essential care services (e.g. dialysis, routine medications, counseling, in-home food delivery), ▪ Dependent on a private well for household water, and ▪ Tribal members and other state-licensed ricers reliant on wild rice harvest for food and income. With regard to drought, this section focused on farmers and agricultural workers as a uniquely vulnerable population, given that their livelihoods depend on productive cropland.

Assessment of Health Risk from Floods & Drought (continued)	
Risk	The evidence base for assessing health risk from floods is growing but is still small and consists mainly of mortality statistics. Epidemiological studies that exist on the health effects of flooding are often limited to small populations that are not widely generalizable. Flood risk in general depends on population density and distribution can vary widely between urban and rural areas. Ultimately, there is a lack of data available that could be used to characterize the full range of health impacts from floods in Minnesota, including severity, magnitude and timing. Health outcomes from flooding are not usually recorded in medical notes, so the association between the complaint and its cause are not explicitly made. At this time, data available is insufficient to fully characterize flood and drought risk.

Ecosystem Threats

Climate change already has had demonstrable effects on the distribution, composition and productivity of the Earth's ecosystems, and these effects are likely to continue, and intensify, in the coming decades (Grimm et al., 2013). Ranges of many species are moving northward to higher latitudes, forcing some native plants and animals into less hospitable areas and enticing invasive species from the south (EPA, 2013; Prasad et al., 2007). Warmer, wetter climate trends are supporting the spread of pathogens and parasites in non-endemic areas. For many species, climate changes are altering the timing of key life-cycle stages, leading to mismatches in migration, blooming, breeding, and food availability (EPA, 2013; DNR, 2011). Impacts on one species may have repercussions throughout the food web, amplifying consequences throughout the ecosystem, including human communities. Climate change not only impacts ecosystems and species directly, it also exacerbates human stressors placed on the environment (EPA, 2013). For example, heavy fertilizer use and subsequent nutrient run-off into surface waters coupled with warming temperatures and heavier precipitation are likely drivers behind the increased prevalence of harmful algal blooms across the U.S. (Rex, 2013). Forest fragmentation and associated loss of wildlife diversity due to urban sprawl have been implicated as a potential contributor in the spread of Lyme disease (LoGiudice et al., 2003).

Minnesota is home to a wide variety of aquatic, grassland, and forest ecosystems, which support numerous wildlife and plant species. In addition, with over five million residents, Minnesota has an extensive network of urban ecosystems within its many cities and towns. Natural areas and parkland are treasured by Minnesotans: Over 15 percent of residents hunt, 52 percent enjoy bird and wildlife-watching, and there are over one million licensed anglers (DNR, 2014). For two consecutive years (2013 & 2014), the Trust for Public Land ranked Minneapolis as having the best park system in the nation (TPL, n.d.). Yet, the impacts of climate change are threatening the state's natural areas in a myriad of ways and some have serious implications for human health.

This section will focus on the climate and health impacts related to ecosystems, specifically vector-borne illnesses, such as Lyme disease

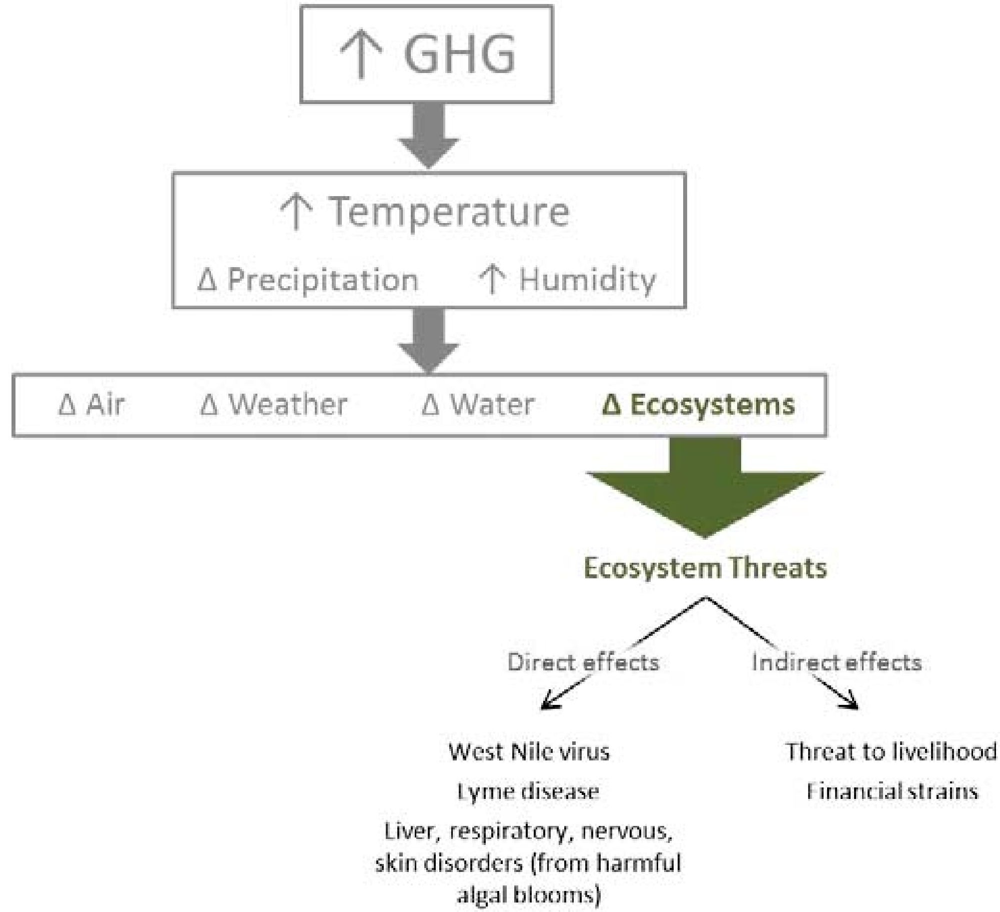
What is a vector-borne disease?

A vector-borne disease is an illness caused by a pathogen (i.e., virus, bacterium, or parasite) that is transmitted to people by mosquitoes, fleas, lice, biting flies, mites or ticks. Vectors typically become infected while feeding on infected birds, rodents, other larger animals, or humans, and then pass on the pathogen to a susceptible person or animal. Vector-borne diseases and their distribution patterns are very difficult to predict, prevent, or control, and only a few have vaccines. Mosquitoes and ticks, common disease vectors, are difficult to control and often develop resistance to insecticides (Brogden & McAllister, 1998).

Adult deer tick, *Ixodes scapularis.* Photo courtesy of Wikimedia Commons.

and West Nile virus, and exposure to toxins from harmful algal blooms (Figure 5.24). There are other health impacts associated with climate-mediated changes in Minnesota ecosystems, such as other vector-borne diseases (e.g., human anaplasmosis or babesiosis) or indirect impacts, such as threats to ecosystem-linked economies (e.g., fishing, hunting, logging). However, Lyme disease (Minnesota's primary tick-borne disease), West Nile virus (Minnesota's primary mosquito-borne disease), and exposure to harmful algal blooms represent a substantial amount of risk to the health of Minnesotans from climate change impacted ecosystems and are therefore the focus of this section.

FIGURE 5.24. LINKS BETWEEN THE RISE IN ATMOSPHERIC GREENHOUSE GASES, CHANGES TO THE EARTH'S CLIMATE, AND DIRECT AND INDIRECT IMPACTS ON HEALTH FROM ECOSYSTEM CHANGES. INDIRECT EFFECTS WILL NOT BE ADDRESSED IN THIS REPORT BUT MAY BE ADDED IN FUTURE ITERATIONS.



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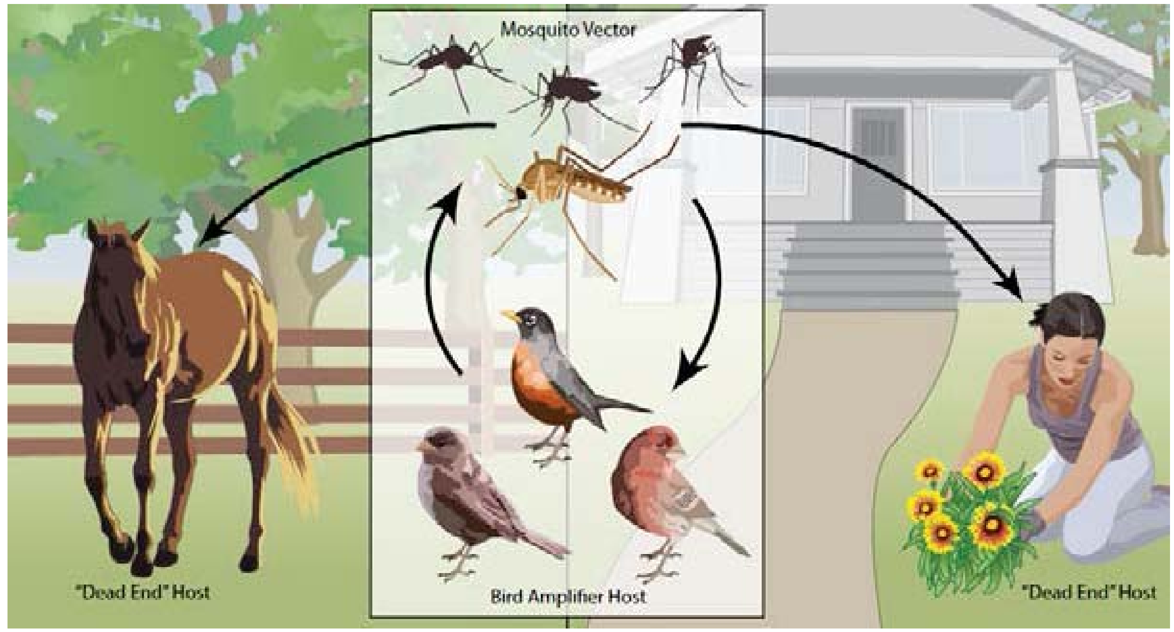
WEST NILE VIRUS

West Nile virus (WNV) was first recognized in North America in 1999. The virus is transmitted to humans mainly through the bite of an infected mosquito (genus *Culex*) that itself contracted the virus from an infected bird host (Figure 5.25). Mosquitoes that transmit WNV exist in rural and urban areas, reproducing in low-lying places with poor drainage, urban catch basins, roadside ditches, sewage treatment lagoons, and artificial containers around homes and other buildings—anywhere with favorable conditions for mosquitoes to lay their eggs. Because of its ability to establish and persist in a wide variety of ecosystems, WNV has spread rapidly throughout the continent, and it is now endemic in every U.S. state, except Alaska and Hawaii, and has produced two large nationwide epidemics (2003 and 2012). Evidence of the virus from infected humans, mosquitoes, birds, horses or other mammals has been reported from 96 percent of all U.S. counties (CDC, 2013). Since 1999, there have been approximately 40,000 reported WNV cases nationally, including 1,663 fatalities. In the U.S for 2013 alone, there were well over 2,400 reported WNV cases with 119 deaths (CDC, 2014).

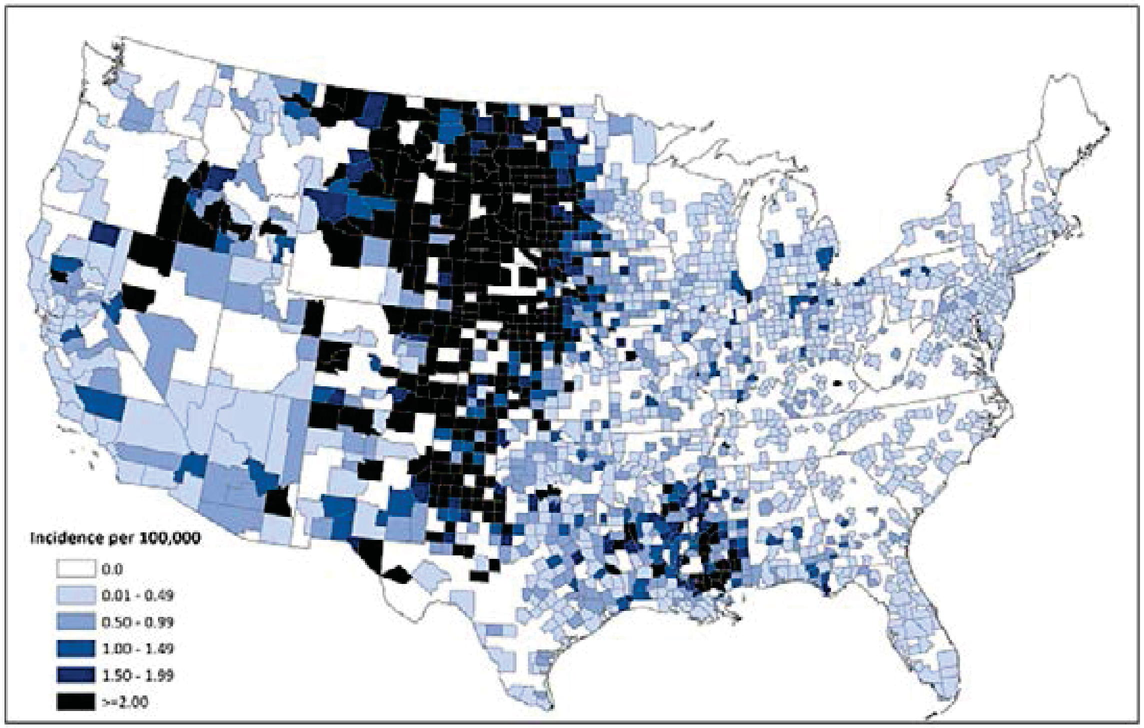
FIGURE 5.25 WEST NILE VIRUS TRANSMISSION CYCLE. INFECTED BIRDS CAN PASS THE VIRUS TO BITING MOSQUITOES. THOSE INFECTED MOSQUITOES THEN BITE AND INFECT PEOPLE, HORSES AND OTHER MAMMALS, WHICH ARE CONSIDERED "DEAD END" HOSTS BECAUSE THEY CANNOT PASS THE VIRUS ON TO OTHER BITING MOSQUITOES. HOWEVER, "DEAD END" HOSTS CAN BECOME SICK FROM THE INFECTION. IMAGE AND INFORMATION SOURCE: CDC, 2014.



Minnesota has documented 615 WNV cases since 2002 when the virus was first identified in the state (MDH, 2014). In 2013, 80 WNV cases (and three deaths) were reported in Minnesota, placing the state in the top 10 for highest annual WNV incidence (Figure 5.26). Most human infections (94 percent) have been reported during the months of July through September (CDC, 2013). Typical symptoms of WNV are similar to the flu, and the large majority of those infected will have only mild or no symptoms. However, in some individuals WNV can cause brain inflammation (encephalitis), and in severe cases, paralysis, coma or death. Additionally, a WNV epidemic can impose enormous costs on local economies (Wang et al., 2010). The estimated short-term cost incurred from the 2002 WNV epidemic in Louisiana was over 20 million dollars (Zohrabian et al., 2004).

Characterizing WNV risk depends on an understanding of the dynamic interactions between pathogen, vector, and hosts, and each with their environment. With the ability to affect these varied components in the WNV transmission cycle, climate change has been identified as an influential driver in the spread of the disease (Paz & Semenza, 2013; Morin & Comrie, 2013; Chen et al., 2013). A national study of WNV cases found that warmer temperatures and heavy precipitation (more than two inches rain in one day) increased the rate of WNV infection in the U.S. (Soverow et al., 2009). Other studies indicate that extended hot and dry spells also play a role in mosquito proliferation and the spread of WNV (Ruiz et al., 2010; Stanke et al., 2013; Wang et al., 2010). Even slight increases in ambient temperature may have a significant impact on transmission, by increasing mosquito feeding and egg laying, leading to more generations of mosquitoes per year. More importantly, warmer temperatures speed the rate at which the virus multiplies in an infected mosquito, increasing the likelihood that the pathogen will be passed on to a host from a bite (Johnson & Sukhdeo, 2013).

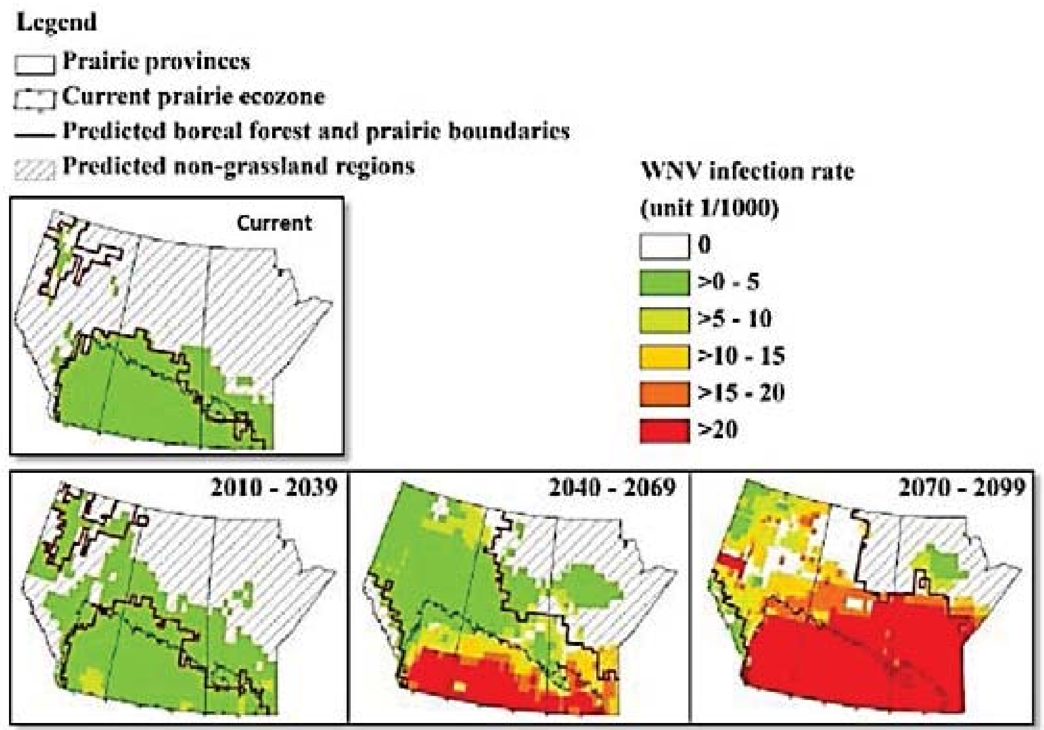
FIGURE 5.26. AVERAGE YEARLY INCIDENCE OF WEST NILE VIRUS DISEASE, 1999-2012. IMAGE SOURCE: CDC, 2013.



Given the wide array of ecological and population-level factors that contribute to a WNV outbreak, few models have been developed to provide long-term predictions of how and where these factors, particularly for Minnesota, will combine to spread the disease (CDC, 2013). At this time, areas of Minnesota with the highest incidence of WNV cases are stretched along the western border of the state (Figure 5.26), areas with prairie, grassland or agricultural land. Research in Canadian prairie provinces, where WNV cases are the highest in Canada, has demonstrated that the effects of climate change on grassland ecozones may facilitate the spread of WNV (Chen et al., 2013). Under extreme warming conditions, projections indicate that WNV infection rates for *Culex* tarsalis mosquitos in these Canadian prairie areas could be 30 times that of baseline levels by mid-century (Figure 5.27; Chen et al., 2013). If the same situation holds for Minnesota prairie areas, the likelihood of infection for people and domestic animals in those areas could be substantial. However, researchers have found that climate change effects on WNV factors are highly localized. Therefore, caution must be taken when generalizing spatially refined models from one locale to another (Morin & Comrie, 2013; Ruiz et al., 2013).

Tracking the incidence of WNV in humans is a powerful tool for understanding and mitigating the rise and spread of the infection across the state. WNV is a nationally notifiable condition, meaning that cases must be reported to the CDC by state and local health departments. WNV cases are collected through ArboNet, a national arboviral surveillance system managed by CDC and state health departments, including MDH. However, the ability of disease surveillance by itself to assess total disease burden and predict where outbreaks may occur is limited, especially given how much local environmental factors can influence epidemics and the lag time inherent in collection of case reports after diagnosis. Research shows that increases in WNV infection rates in mosquito populations can provide an indicator of developing outbreak conditions in advance of increases in human infections (CDC, 2013).

FIGURE 5.27. PROJECTED WEST NILE VIRUS INFECTION RATE IN CULEX TARSALIS MOSQUITOS ACROSS CANADIAN PRAIRIE PROVINCES UNDER "CURRENT" (1961-1990) AND FUTURE PROJECTION SCENARIOS. IMAGE SOURCE: CHEN ET AL., 2013.



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Assessment of Health Risk from West Nile Virus	
Hazard	West Nile virus (WNV) was identified in Minnesota in 2002 and WNV illness in humans has been reported every year since then. This along with findings in mosquitoes, birds and horses suggest that WNV is firmly established in the state.
Exposure	People may be exposed by living, playing or working in or near areas where WNV is present in human-feeding mosquitoes. Peak risk for WNV exposure occurs in late summer. Dusk and dawn is primarily when exposure occurs because this is when the mosquitoes feed.
Vulnerability	Anyone living in an area where WNV is present in mosquitoes is vulnerable to infection. People who work outside or participate in outdoor activities are more likely to be exposed, especially if they do not use repellents or wear protective clothing. The elderly and people with certain medical conditions, such as cancer, diabetes, hypertension and kidney disease, are especially vulnerable to the most severe manifestations of WNV.
Risk	The disease risk to Minnesotans will likely continue to be high in central and western Minnesota, where *Culex tarsalis* mosquitoes are most abundant. Population growth forecasts suggest that in the coming decades fewer people will be living and working in these areas of the state, but they are still likely to be destinations for visitor, especially for outdoor recreation. Current WNV cases in Minnesota tend to be farmers or other rural residents exposed at or near their home, suggesting that open agricultural areas may pose a place-based risk. Given the large areas of prime mosquito habitat across Minnesota and intensifying climate changes that facilitate transmission (e.g., rising temperatures and heavy rainfall events with interceding dry spells), it is likely that WNV will continue to be a health hazard for the state and possibly worsen in magnitude over the long term.

LYME DISEASE

Lyme disease is the most common tick-borne illness in the United States. Typical symptoms of Lyme disease include fever, headache, fatigue, and skin rash. If left untreated, the infection can spread to joints, heart and nervous system, resulting in significant, sometimes irreversible damage (CDC, n.d.).

Since being formally identified in the 1970s, the incidence of the disease has increased and expanded its geographic range, causing epidemics in the Eastern and Midwestern regions of the nation (Diuk-Wasser et al., 2012). The CDC estimates that approximately 300,000 Americans are diagnosed with Lyme disease every year (CDC, 2013). Minnesota is among the top states for Lyme disease incidence (Figure 5.28). From 1996 to 2013, over 18,000 cases of tick-borne diseases were reported in Minnesota, the majority of which were Lyme disease (MDH, 2014). In 2013 alone, there were over 1,400 confirmed cases of Lyme disease in Minnesota (Figure 5.29). However, due to reporting and attribution challenges, this is likely an underestimate of the true prevalence of the disease (CDC, 2013). Based on reported cases for Minnesota, the distribution of Lyme disease appears to be expanding northwest across most of the state; however cases are located according to the residence of the patient, which may be different than where infection may have occurred (Figure 5.30).

FIGURE 5.28. REPORTED CASES OF LYME DISEASE ACROSS THE U.S. FOR 2011. ONE DOT WAS PLACED RANDOMLY WITHIN THE COUNTY OF RESIDENCE FOR EACH CONFIRMED CASE. THOUGH LYME CASES HAVE BEEN REPORTED IN NEARLY EVERY STATE, CASES ARE REPORTED FROM THE INFECTED PERSON'S COUNTY OF RESIDENCE, NOT NECESSARILY THE PLACE WHERE THEY WERE BITTEN BY AN INFECTED TICK. IMAGE SOURCE: CDC, N.D.

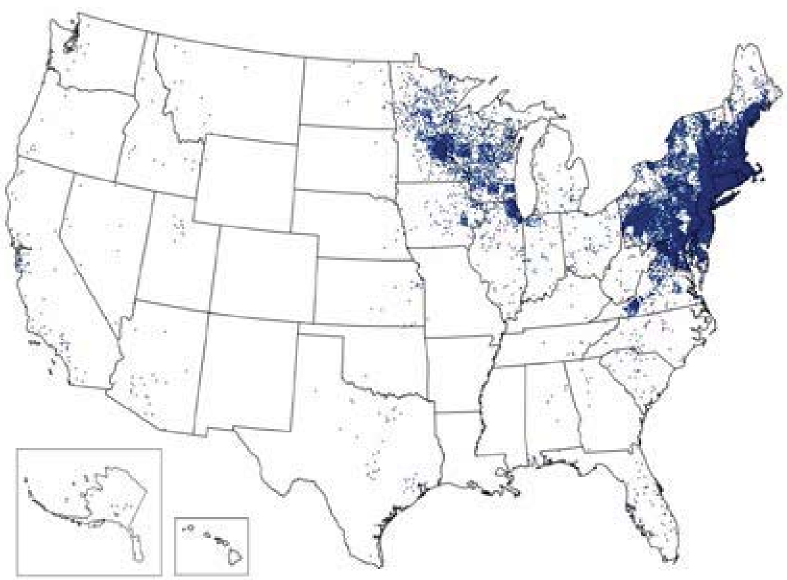

FIGURE 5.29. CONFIRMED CASES OF LYME DISEASE IN MINNESOTA, 1996-2013. DATA SOURCE: MDH, 2014.

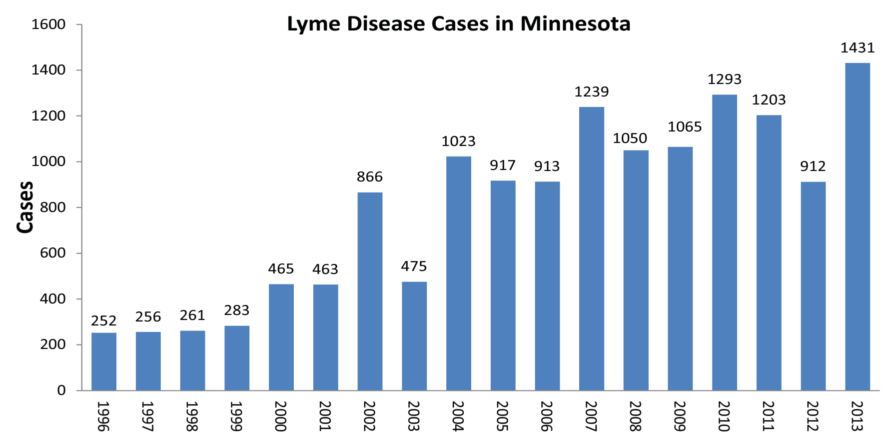

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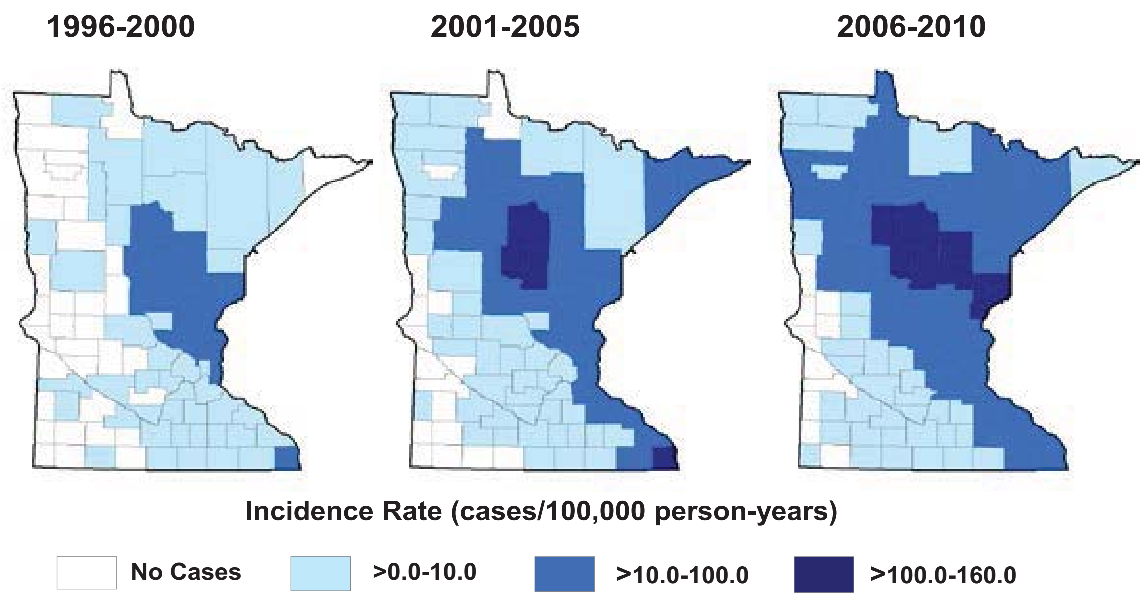
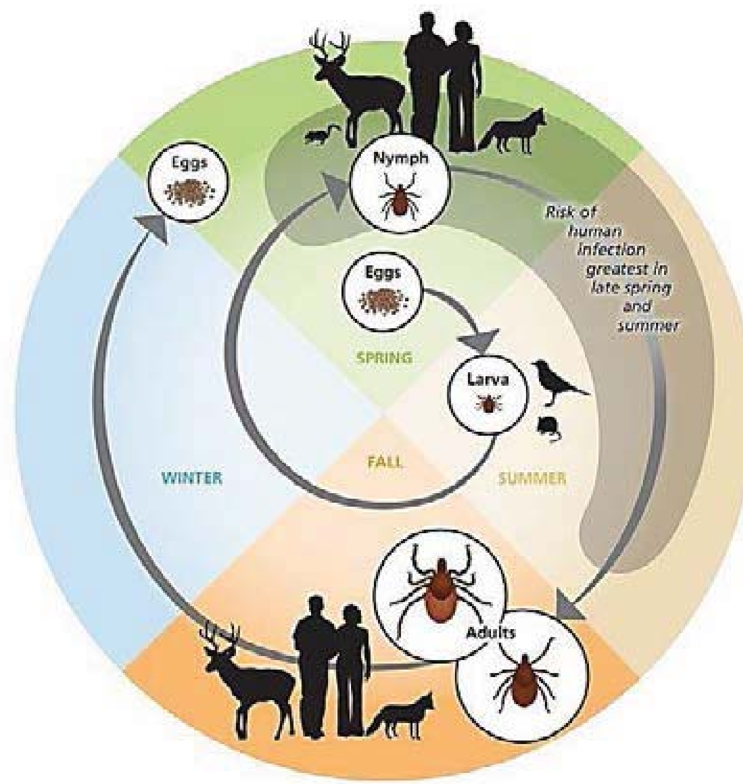
FIGURE 5.30. EXPANDING GEOGRAPHICAL DISTRIBUTION OF LYME DISEASE BY CASE COUNTY OF RESIDENCE, MN, 1996-2010. IMAGE SOURCE: UNPUBLISHED DATA, D. NEITZEL, JUNE 2014.



Lyme disease is caused by the bacterium, *Borrelia burgdorferi* (*Bb*). *Bb* is tethered to the life cycle of its tick vector, *Ixodes scapularis*, commonly known as the blacklegged or deer tick (Figure 5.31). *Bb* is spread through the bite of a blacklegged tick that itself has been infected by feeding on the blood of an infected host. In the past, the emergence of Lyme disease was linked to the abundant white-tailed deer populations, particularly in the Midwest, given that they are a common source of nourishment for the adult blacklegged tick. However, deer clear the bacteria from their blood after they are infected, and therefore are unable to serve as a reservoir for the disease. Instead, especially in Minnesota, white-footed mice are the most common reservoir host for *Bb*, recognized as a powerful “amplifier” of Lyme disease and a maintenance host (i.e., a host that establishes the pathogen in an area over time). When a tick feeds on an infected mouse, it can transmit *Bb* to the next host it feeds on, such as people and pets.

Climate change may be a factor in the rising incidence of Lyme disease given the influence of climate on the complex ecology of *Bb* (Brownstein et al., 2005; Gray et al., 2009; Ogden et al., 2010). By affecting the abundance, distribution, and behaviors of both vector and host, changes in seasonal temperatures and precipitation are leading to an expansion of Lyme risk. For example, blacklegged tick survival is highly dependent on climate patterns, with both water stress and temperature regulating mortality (Bertrand & Wilson, 1996; Needham & Teel, 1991). Northward shifts in the migration patterns of birds and other hosts responding to warming temperatures enable the spread of both ticks and *Bb* from south to north (Brinkerhoff et al., 2011; Ogden et al., 2010). In addition to climate factors, expansion of Lyme risk is affected by societal decisions regarding land use and recreation – as residential development encroaches on forested areas and people spend more time in the outdoors, the opportunity for exposure to infected ticks increases. However, recent research from Wisconsin demonstrating the spread of blacklegged ticks into metropolitan areas may weaken the assumption that urban and suburban dwellers are safe from tick exposure and *Bb* infection (Lee et al., 2013). In areas of the Midwest, including Minnesota, blacklegged ticks can transmit other disease agents in addition to Lyme, including anaplasmosis and babesiosis (CDC, n.d.).

FIGURE 5.31. LIFE CYCLE OF THE BLACKLEGGED TICK AND ASSOCIATION WITH LYME DISEASE RISK. BLACKLEGGED TICKS HAVE THREE LIFE STAGES, IN ADDITION TO THE “EGG”-LIKE STAGE—LARVAE, NYMPHS AND ADULTS—THAT ALL FEED ON ANIMAL OR HUMAN HOSTS. LARVAE HATCH FROM EGGS UNINFECTED WITH ***BORRELIA BURGDORFERI*** (LYME DISEASE BACTERIA) BUT CAN BECOME INFECTED WHEN THEY TAKE A BLOOD MEAL FROM AN INFECTED SMALL MAMMAL. NYMPHS ARE DIFFICULT TO DETECT AND ACTIVELY FEED DURING LATE SPRING AND EARLY SUMMER. PEOPLE ARE MOST AT RISK OF CONTRACTING LYME DISEASE FROM AN INFECTED NYMPH. ADULT TICKS CAN CLIMB HIGHER ON GRASS AND SHRUBS AND WILL GENERALLY ATTACH TO LARGER ANIMALS, IN PARTICULAR DEER THAT ARE THEIR PREFERRED SOURCE OF BLOOD. IMAGE SOURCE: ORENT, 2013.



Reducing the burden of Lyme disease and other tick-borne diseases in Minnesota requires two main strategies, early identification and treatment of currently infected individuals and prevention of *Bb* transmission, with the latter being the ultimate goal. Critical to any prevention strategy is an understanding of the tick, hosts, pathogen and the dynamic interplay among them. Accurate information on patterns of human Lyme risk is essential for making personal protection decisions, allocating public health resources, and assisting clinicians with diagnoses and use of prophylaxis or vaccines (Diuk-Wasser et al., 2012). However, it is difficult to get reliable data due to underreporting, misdiagnoses, and case definitions, and the variable interval between exposure and appearance of symptoms, which can confound determination of exposure location (Diuk-Wasser et al., 2012; Lee et al., 2013). Some regions also face issues with changes in surveillance methods. However, Minnesota has had relatively consistent surveillance since the mid-1990s.

Lyme disease is a nationally notifiable condition in the U.S. Cases are collected and verified by state and local health departments and then shared with the CDC through an infectious disease reporting system. Generally, surveillance data are captured at a coarse level, and not by county of exposure, but county of residence. Often, Lyme disease data are not available at local scales, nor linked to the point of disease transmission, two factors that are crucial for epidemiological studies and development of effective prevention strategies. In Minnesota, MDH is able to determine data at local scales and link cases to the point of disease transmission. MDH adds to surveillance efforts in Minnesota by following up with patients directly in order to collect information on potential locations of exposure and other relevant data. In addition, monitoring and data collection efforts are centralized within the MDH, which lends efficiency and consistency to surveillance methods.

Assessment of Health Risk from Lyme Disease	
Hazard	Lyme disease has been identified throughout Minnesota. Recent case counts suggest that the disease is expanding beyond its endemic range.
Exposure	Because Lyme disease is a nationally reportable condition, exposure to *Borrelia burgdorferi* that results in a diagnosis of Lyme disease is captured. However, Lyme disease cases diagnosed on the basis of the "bulls-eye" rash alone without laboratory tests are often not reported to MDH. Because the body's reaction to *Borrelia burgdorferi* can be mistaken for other ailments, it is possible that many cases are escaping detection altogether. Therefore, at this time we do not have an exact measure of the extent to which Minnesotans are being exposed to the pathogen through tick bites.
Vulnerability	Any person living, playing or working in or near areas that are endemic for Lyme disease is vulnerable. However, people who regularly work outside (e.g., in landscaping, farming, or forestry) or actively participate in outdoor activities (e.g., gardening, hiking, camping, hunting, or bird-watching) are vulnerable, especially if they do not use bug spray or wear protective clothing. Pets also are vulnerable to contracting Lyme disease.
Risk	Infectious disease reporting provides a means of estimating the annual number of Lyme disease cases in Minnesota, although there are probably many cases that are not counted due to misdiagnosis by a clinician or the individual never seeks medical care. The disease risk to Minnesotans will likely continue to be higher in areas where habitat exists to support a thriving tick population and people are regularly coming into contact with ticks, such as those working outside or participating in outdoor activities, especially through the spring, summer and fall seasons. Given that Lyme disease is endemic in most areas of the state and climate changes will facilitate transmission (e.g., rising temperatures and heavy rainfall events with interceding dry spells), it is likely that Lyme disease will continue to be a significant health hazard for the state, both in the near and long term.

HARMFUL ALGAL BLOOMS

Many species of algae are found in Minnesota lakes and rivers. Most algae are harmless and are essential to the aquatic ecosystem. However, blue-green algae (more correctly referred to as cyanobacteria), which are also found throughout Minnesota's lakes and rivers, are capable of releasing dangerous toxins into the water. Under the right environmental conditions, blue-green algae can grow quickly, forming harmful algal blooms (HABs). HABs are most likely to occur in warm, stagnant waters that are rich in nutrients from agricultural and urban sources or direct wastewater discharge. HABs usually float to the surface of the water and can exhibit many different appearances and colors and often accumulate on downwind shorelines (Paerl et al., 2001). People, pets, or livestock that swim in or drink water containing toxins from HABs may develop liver, digestive and skin diseases, respiratory problems, or neurological impairment. Fatalities are rare for humans but more common for dogs that ingest contaminated water (Hilborn et al., 2014; Lopez et al., 2008). Often the first sign that a HAB exists is a dog that becomes ill or dies after swimming in algae-filled waters. Worldwide, HABs are considered a major threat to the use of surface water for drinking water, irrigation, fishing and recreation (Lopez et al. 2008; Paerl et al., 2001). HABs are linked to fish kills and widespread loss of other animals and aquatic plants. HAB toxins can accumulate in fish or shellfish and pass the exposure onto fish consumers (Lopez et al., 2008; NOAA/NOS, 2014). In some areas of the nation, indigenous tribes were the first to draw attention to frequent and severe algae blooms in waters that were relied on for subsistence fishing. While not all blue-green algal blooms produce toxins, there is no way to determine by sight alone if toxins are present. In addition, the location and concentration of toxins within or around a HAB may linger after the algae bloom itself has disappeared (NOAA/NOS, 2014).

HABs cause environmental problems in all 50 states, and their incidence and intensity, as well as associated economic losses, have increased in recent decades (Oneil et al., 2012). Agricultural runoff, laden with nitrogen and phosphorus from fertilizers and manure, is considered to be a major cause of HAB proliferation. In particular, the explosion of corn cultivation, may add to the problem. Corn is the most widely planted field crop in the U.S. (and Minnesota) and requires the most fertilizer per acre (Ribaudo, 2011). The plant itself is "leaky", meaning that it absorbs less nitrogen per acre compared to other crops, leaving excess nutrients to be washed off the fields with rainfall or irrigation (Engelhaupt, 2007).

However, like most ecosystem disturbances, there are a number of factors that may be occurring simultaneously to influence the occurrence and detection of HABs, including wastewater treatment discharge, stormwater runoff, improvements in surveillance and detection methods and finally climate changes (Oneil et al., 2012). Rising ambient temperatures, changes in precipitation patterns and the associated impacts on the hydrologic cycle, can strongly affect the metabolism, growth and formation of HABs. For example, growth of harmful blue-green algae is optimized at relatively high temperatures, allowing these organisms to dominate over other non-toxic algal species (Oneil et al., 2012; Paerl et al., 2011). Larger and more intense rainfall events wash nutrients into surface waters, promoting growth, while protracted periods of drought provide for still, stagnant waters that support HAB expansion and the build-up of toxins. Finally, higher levels of atmospheric CO_2, itself a potent fertilizer for some blue-green algal species, also may be contributing to the increased occurrence of HABs (Oneil et al., 2012).

Harmful algal bloom in a freshwater pond. Photo courtesy of Wikimedia Commons.

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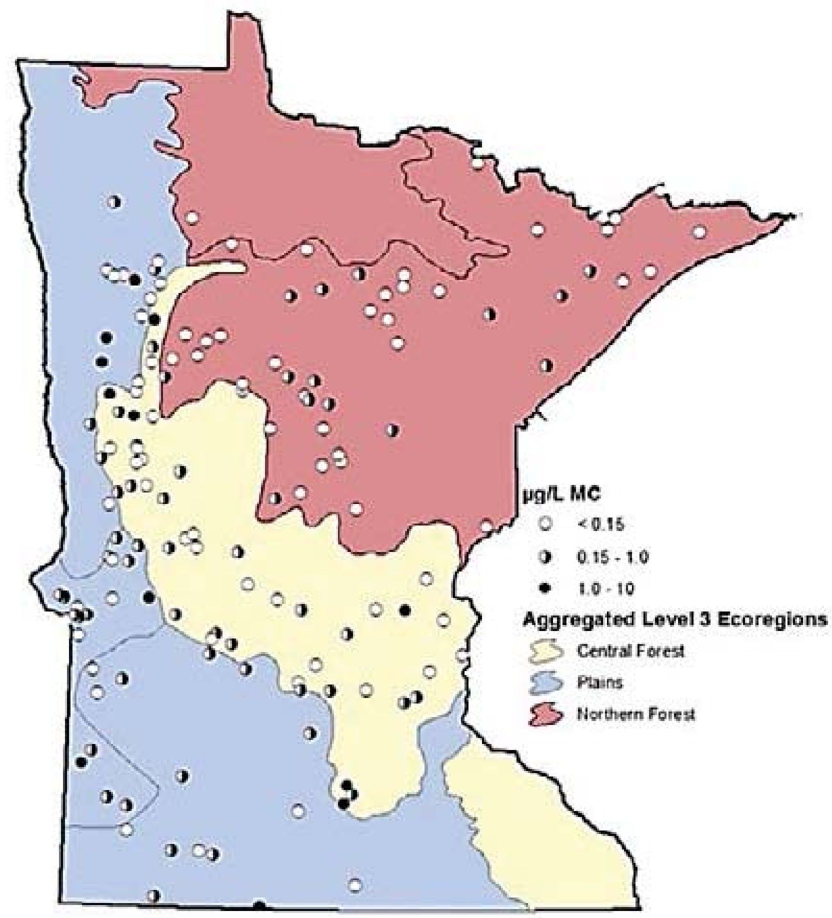
Blue-green algal toxicity is not a new issue in Minnesota. Documents dating back to the 1800s report on several incidences of HABs that led to cattle, horse and dog deaths (Lindon & Heiskary, 2009). A number of studies, led in part by the Minnesota Pollution Control Agency (MPCA), have identified HABs in Minnesota (Heiskary et al., 2014). A 2006 targeted study of 12 lakes in south central Minnesota examined the spatial and temporal variation in microcystin, one of the most frequent, well-studied and hazardous HAB toxins (Lindon & Heiskary, 2009). The 2007 and 2012 National Lakes Assessment Project included surveys of microcystin in 50 randomly selected Minnesota lakes (Heiskary et al., 2014; MPCA, 2008). These studies provide valuable information on the extent, magnitude, and frequency of microcystin in Minnesota lakes. A recent study by Heiskary et al. (2014) shows that microcystin has been measured in high and very high risk concentrations in sampled Minnesota lakes, particularly those that are in agriculturally dominant areas to the south and southwest (Figure 5.32). Given the substantial costs and challenges related to continuous sampling, currently there is no formal monitoring or testing program in Minnesota that ensures routine surveillance of all state surface waters. This lack of data makes it difficult to investigate and assess the true scope of affected or potentially affected waters and their impacts on health, particularly in the long term. However, MPCA staff do track reports of potential HABs, and MDH will investigate if HAB toxins are discovered. Both agencies work together to educate the public about HABs, and future actions are likely to include development of recreational and fish consumption exposure guidelines (Heiskary et al., 2014).

FIGURE 5.32. MICROCYSTIN (MC) CONCENTRATIONS (MICROGRAMS PER LITER) IN MINNESOTA LAKES FROM 2012 NATIONAL LAKE ASSESSMENT PROJECT. IMAGE SOURCE: HEISKARY ET AL., 2014.



Assessment of Health Risk from Harmful Algal Blooms	
Hazard	Harmful algal blooms (HABs) have been identified in Minnesota. However, with no established ongoing HABs monitoring program in the state it is difficult to estimate trends in microcystin production, nor the extent to which HABs are a current or future health hazard.
Exposure	People may be exposed by swimming in, drinking, or breathing in HAB toxins. However, characteristics of exposure are difficult to determine given the current lack of sampling data. At this time, it is not possible to identify the frequency, duration, magnitude or severity of HAB-toxin exposure to Minnesota residents or visitors. Yet, given the large number of potentially impacted ponds, lakes, and rivers, coupled with resident and visitor fondness for water recreation in the state, suggests that exposure could be frequent and impact a large number of people and pets. In addition, the body's reaction to HAB toxins can be mistaken for numerous other infections or ailments. Thus, cases of HAB poisonings may not always be accurately identified because the person doesn't seek treatment, or they are misdiagnosed.
Vulnerability	Environment: Fertilizer and manure runoff from cropland enriches surface waters with excess nutrients and is a leading cause of HABs in the rural portions of Minnesota. In urban areas, stormwater and wastewater discharge are prominent sources of excess nutrients to lake and rivers. Humans: Any person swimming in, drinking, or breathing in HAB toxins is vulnerable. However children, the elderly, individuals with pre-existing health problems, and frequent fish consumers (e.g., members of indigenous or other ethnic groups) may be especially vulnerable to adverse health impacts. People who rely on private wells also may be vulnerable if the well draws directly from surface water or groundwater that is recharged by contaminated surface water. Communities that rely on lake and river tourism traffic or commercial fishing employment may be vulnerable to impacts from HABs. Some communities may face increased water treatment costs to remediate HAB toxins in drinking water sources.
Risk	Given the lack of data on the occurrence and location of HABs coupled with the lack of a case reporting system on HAB poisonings, it is difficult at this time to estimate the current and future risk to population health from HABs in Minnesota. Yet, given the number of nutrient-affected waters in the state coupled with intensifying climate changes (i.e., rising temperatures and heavy rainfall events with interceding dry spells) it is possible that HAB poisonings could be a significant health hazard for the state, both in the near and long term.

VI Conclusion

Minnesota has gotten warmer, and precipitation patterns have become more unpredictable. According to climate projections for the state, these trends are likely to continue with wide-ranging repercussions for the health and well-being of the population. Minnesota's *Climate & Health Profile Report* provides a comprehensive assessment of the major health impacts that Minnesotans will face from climate change hazards, including extreme heat and flood events, rise in air pollution, spread of vector-borne diseases, and drought impacts on the agriculture sector.

The next steps for an effective response to these hazards are reflected in the CDC BRACE process and MDH's Minnesota Climate & Health Program goals and activities, and include the following:

- **Addressing data and research gaps.** Characterizing the broad spectrum of health threats from climate change requires an equally broad spectrum of quantitative and qualitative information. Public health professionals will have to expand their view of the types of data and information that are required for characterizing the true burden of disease or distress associated with climate hazards, as well as obtaining access to these data. For example, limited data exist to help characterize all impacts from flooding on Minnesotans, including financial, physical, and emotional influences on health and well-being. A complete assessment of this kind will require identifying and accessing datasets that may not be commonly used for public health investigations, or creating opportunities for gathering necessary information, e.g., interviews with affected individuals or town hall style forums.

- **Communicating direct and indirect effects of climate change.** Climate change impacts health both directly and indirectly. Disseminating information about and planning for indirect and "upstream" effects are as important as describing and preparing for the direct impacts. Public health practice is rooted in a view of health as a "downstream" outcome of numerous "upstream" influences. From this perspective, factors that diminish a person's access to secure employment, quality education, or a reliable support network are viewed as key health determinants. Underscoring these connections with agency colleagues, decision-makers, and the public will be instrumental toward garnering attention and action toward bolstering these indirect health determinants, and mitigating poor health outcomes well before they arise in a population.

- **Outreach to vulnerable populations.** Some Minnesotans will be more vulnerable to a particular climate hazard compared to the general population due to age, gender, education level, income, occupation, recreational interests, or current health status. Vulnerability is situational; a person who is vulnerable to one hazard may not necessarily be vulnerable to another. Also, climate vulnerability may be temporary along with the characteristic, e.g., pregnancy or homelessness. A wide view should be taken as to what groups are vulnerable to a climate hazard and when they are vulnerable. For instance, the elderly are often identified as vulnerable to extreme heat, but less often acknowledged are farmers who are also vulnerable, albeit indirectly due to crop or livestock loss, threat to livelihood and financial strains on access to basic necessities and essential services. In addition, while direct outreach to vulnerable populations is essential, tapping into networks that support these populations can be equally effective. For example, some jurisdictions work with local "Meals on Wheels" programs to provide some additional oversight of potentially isolated elderly during extreme heat events.

- **Expand cross-sector collaboration.** Characterizing climate change impacts on the health of Minnesotans will require teaming up with sectors and disciplines that may be relatively new to public health. In addition, adaptation efforts will need to happen on larger and faster scales compared to the past, which will necessitate that public health professionals ally with other state agency personnel, business leaders, policy-makers, vulnerable populations and the general public to obtain and analyze information and apply this knowledge toward fostering climate resiliency in the state. These cross-sector collaborations will have extensive co-benefits, strengthening public health alongside Minnesota's infrastructure and economy.

- **Identifying effective interventions.** Expanding access to data, widening our view of key health determinants and developing cross-sector partnerships will help to identify interventions or strategies for optimizing health and well-being in Minnesota's changing climate. Given that states and municipalities across the nation are undergoing similar efforts, numerous strategies will likely be identified and implemented to mitigate the effects of climate hazards. Minnesota public health professionals can learn from these experiences while drawing on resources and partnerships here in the state to select the most effective interventions for sustaining health. These interventions should be implemented with an evaluation plan in place so their efficacy can be measured and outcomes shared with the public and colleagues.

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Document Review: NorthMet Project Flotation Tailings Basin Management Plan Version 2 (DEC 2012)
Reviewers: Dam Safety (Jason Boyle, Dana Dostert); Lands and Minerals (LAM); ERM

When:
3=V3 (4/12)
4=V4 (July)
S=this sheet
P=permitting

#	Location (Doc, Page, Figure, or Table)	Section Number	Comment/Concern	Reviewer	When	PolyMet Response 04/10/2013
1	pg 5	2.1	In several places in the report, the report notes that the dam design can be modified or the construction sequence can be altered as needed to conform to tailings characteristics from full-scale operations. This might leave an outside reviewer with little confidence in the current design. Explanation of the observation method and how it relates to the EIS may be warranted.	Boyle	3	See Response to Comment #41. The existing Flotation Management Plan (FTMP) discussion on mitigation will be expanded.
2	pg 5	2.1	A general description of the tailings geochemistry should be provided here, not only using external references.	LAM	S	By design the document structure minimizes duplication of information and this has been agreed in principle. Geochemistry is discussed in NorthMet Project Waste Characterization Data Package - Version 10, March 7, 2014, Section 5 and 10.
3	pg 5	2.1.1	Flotation tailings samples were also ran in 2006.	LAM	3	It is true that a Flotation Process Optimization test was run in 2006. However, samples from that test were collected for environmental purposes (e.g., waste characterization, air quality, water balance, etc.) and not for geotechnical purposes. Only gradation testing was performed for geotechnical purposes. Therefore there is little significance of the 2006 test to the geotechnical design effort. The following edit will be made to Section 2.1.1: "Flotation Tailings samples were collected from the pilot-test runs in 2005, 2006, 2008, and 2009. "
4	pg 6	2.2.1	Please describe the emergency overflow systems either in this section or 2.2.4 or 7.4.	Boyle	3	The emergency overflow structure is described in FTMP Section 7.4 and the location and layout of the emergency overflow is provided in Drawings FTB-015 to FTB-018. Also see response to Comment #34. Please advise if more information is required.
5	pg 6	2.2.1	Is there an environmental risk to allowing "routine overflow via the emergency overflow outlets to the adjacent wetlands"?	Boyle	3	The reference to "routine overflow..." is after reclamation, when the pond water meets applicable water quality standards or the overflow can be treated by non-mechanical treatment systems. Prior to that time, the WWTP will treat water from the FTB South Surface Seepage Management System, FTB Containment System, HRF drainage water and the FTB pond as needed to prevent overflow. The WWTP will be maintained operable until the MDNR releases the company from mechanical water treatment requirements under the Permit to Mine. Operation of the WWTP during long-term closure is discussed in Section 4.2 of the NorthMet Project Adaptive Water Management Plan - Version 5, March 7, 2013. Text in 2.2.1 will be modified to reflect this.
6	pg 6	2.2.1	There seems to be a stability issue that conflicts with the water holding issue.	LAM	S	This comment is unclear; the stability analysis presented in Geotechnical Data Package - Vol. 1 demonstrates adequate stability of the Flotation Tailings Basin (FTB) during operations and at closure. Both conditions have been analyzed for stability with the proposed pond in place and slope stability safety factors meet MDNR requirements. Further, this topic has been discussed with the MDNR and it has already been agreed that the permanent pond is part of the mine waste management plan which is intended to minimize the amount of oxygen reaching the Flotation Tailings at closure. If more information is required, please clarify the comment and a follow-up response will be provided.
7	pg 6	2.2.1	How important is a well graded particle size distribution with minimal segregation? The statement in the third paragraph of section 2.2.1 contradicts reviewer's experience.	LAM	S	It is assumed that the statement in question is "The Flotation Tailings will have a small and fairly uniform grind size such that when deposited a fairly consistent particle size distribution will be achieved thereby minimizing segregation of coarse and fine portions (Section 3.2)". The "small and fairly uniform grind size" of the Flotation Tailings is a design outcome of the beneficiation process. The "segregation of coarse and fine portions" has been discussed and is supported by the Saint Anthony Falls Laboratory testing work (FTMP - Attachment B). The grind size and deposition of the Flotation Tailings and resulting hydraulic conductivity and shear strength result in acceptable slope stability safety factors when the overall stability of the FTB is analyzed as reported in NorthMet Project Geotechnical Data Package - Vol. 1 Flotation Tailings Basin Version 3, November 21, 2012.
8	pg 6	2.2.2	The statement that "The dam design can be modified, if necessary, based on results of performance and stability monitoring." This is true during years 0-20 +/-, but not during years 20-900. How are observed erosion rates incorporated in the 900 year design? Are erosion rates going to be measured in the future?	LAM	S	It appears that this comment relates to Section 2.2.1 rather than Section 2.2.2. The dam is most susceptible to erosion during and for several years after construction; prior to establishment of a dense vegetative cover and root mass. To account for this, inspection and maintenance of erosion is required by Sections 5.5, 5.6, 5.7 and 5.8 of the FTMP. Some variation of these inspections will be carried on after closure, during reclamation and continuing into long term care. Inspection and erosion repair will be included in financial assurance.
9	pg 7	2.23, 2nd para	Is Appendix G the right reference?	LAM	S	This comment is unclear; the FTMP does not include a Section 2.23 nor an Appendix G. If this is in reference to "Additional information on borrow material and preliminary specifications can be found in Attachment G" at the end of the 2nd paragraph of Section 2.2.4; Attachment G is the correct reference and will remain in Version 3.

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10	pg 7	2.2.4	In the data package, there was discussion about blends created from the various LTVSMC tailings for dam construction, but here you say there are sufficient coarse tailings to provide for all construction materials. Which is correct?	Dostert	3	Geotechnical Data Package - Volume 1 - Ver. 3 Section 4.4.2.5 "Permeability of LTVSMC Bulk Tailings" does not indicate that blends of tailings will be created for use in dam construction. Rather, it is acknowledged that the LTVSMC coarse tailings that will be used for dam construction may have occasional inclusions of fine tailings and slimes, and the purpose of testing blends of tailings is to understand the potential affect of these finer tailings on performance of the constructed dams. The statement is Section 5.0 of the Geotechnical Data Package indicating that "Suitable structural fill will be imported for dam construction if LTVSMC bulk tailings quantities are insufficient...." was added in response to previous agency comments but will be removed from Geotechnical Data Package - Vol. 1 - Ver. 4 to minimize potential confusion. Section 2.2.4 of the FTMP clearly states that 20,000,000 cubic yards of LTVSMC coarse tailings are available to meet the estimated 18,000,000 cubic yard dam construction quantity requirement, and the location of the LTVSMC coarse tailings borrow is provided on Drawing FTB-003. Please advise if more information is needed.
11	pg 8	2.2.4	Recommend adding a timeline chart of construction activities (buttressing, lift sequencing, closure, cover. etc.) similar to the HRF timeline. Table 2-1 is good for lift related information.	Dostert	4	Agreed. Timeline will be added to FTMP **in Version 4.**
12	pg 9	2.2.4	All buttress info needs updating.	Boyle	3	Agreed. Buttress information will be updated to reflect changes in buttress volume resulting from: 1) use of new and more accurate LIDAR based topographic data for the tailings basin toe of slope, and 2) the slight modification of the buttress design (raised by 4 feet) to provide a Factor of Safety of 1.11 for the fully liquefied baseline slope stability analysis case as presented in Geotechnical Data Package - Vol. 1 - Ver. 4.
13	pg 10	2.3	What will be the criteria for moving spigots?	Dostert	S	Spigotting is used to develop beaches to support subsequent dam lifts. Once sufficient beach has been developed at a spigotting location, spigotting will cease at that location and be moved to another location. Once sufficient beach has been developed to allow the next dam lift to be constructed, Flotation Tailings will be deposited subaqueously in the pond.
14			What will be the criteria used to switch between perimeter and subaqueous deposition?	Dostert	S	See response to Comment #13.
15	pg 13-14	3.1	Discussion of design requirements and going beyond what is required for USSA yield condition is unclear.	Boyle	S	The $USSA_{yield}$ condition is typically the condition with a higher probability of failure than either the ESSA or the $USSA_{liquefied}$ condition. For the $USSA_{yield}$ condition the dam as configured provides slope stability safety factors well above the requirement of FOS $\geq$ 1.3; safety factors for the $USSA_{yield}$ condition are all above 2.
16	pg 15	3.1	Change "final" to "interim" in "final lift conditions (Table 3.2)"	Boyle	S	Table 3-1 presents results for "final lift conditions" and has Lift 8 data (Lift 8 is the highest and final lift). Table 3-**2** presents results for "interim lifts" and has data for Lifts 2,4,6 which are at times the highest lift but not the final lift. Please clarify this comment and a follow-up response will be provided.
17	pg 16	3.1	Suggest adding liquefaction analysis and results to this section since it is referenced in the last two bullets on page 13.	Boyle	3	Complete liquefaction analysis is provided in Geotechnical Data Package - Vol. 1. A short summary of the liquefaction analysis and results will be added to Section 3.1.
18	pg 17	3.2	The St. Anthony study represents warm weather deposition. In the winter, it is probable that much of the fines will be caught in snow, ice and other sedimentary features and freeze in the beaches. What are the long term impacts if the perimeter tailings have a higher fines content (>30%) than expected?	Dostert	S	Please provide the analysis that supports the statement that "In the winter, it is probable that much of the fines......". The tailings discharge water will be warm (approximately 72 degrees F). As with other tailings basins with spiggotted discharges, the warm water at the discharge point melts snow and ice as the water and tailings follow a braided and meandering channel pattern to their ultimate point of deposition. As such, there is not anticipated to be preferential deposition of fine tailings on beaches during winter operations. In other words, freezing conditions do not materially interfere with delta formation, fines accumulation on beaches, or delivery of fines to the pond. If frigid conditions (e.g., -30 F) were to occur and tailings deposition issues were to develop, tailings deposition could be shifted to only directly discharge to the pond via the barge and treme diffuser system. Since it is possible that some winter pond freeze-up could occur, making barge and tremie diffuser operations more difficult, bubbler systems may be required to maintain open pond areas in targeted operating areas during extended stretches of frigid conditions. In any case, fines accumulation on the beaches (>30 percent) would not materially impact the Factor of Safety analysis because the critical slope stability failure surface intersects relatively little beach area and and potential strength reductions due to fines deposition in beach areas would be countered by coarser material zones elsewhere along the failure surface.
19	pg 17	3.2	What are the water impacts if a large volume of slurry goes into ice on the beaches and little water or fines/slimes reach the pond?	Dostert	S	See response to Comment #18. Given the tailings discharge temperature it seems improbable that the discharge would instantaneously freeze on the beaches without ever reaching the pond.
20	pg 17	3.3	Is there a breach pathway in the east to the other Trimble Creek and into Silver Mine Creek and the Lake? Can this outflow into Wyman Creek or will an overflow of Silver Mine Lake spill into the mine pit to the SE? Note that any flows that enter Wyman Creek may eventually enter Colby Lake.	Dostert	S	For purposes of this response it is assumed that the reference in the comment should be to Spring Mine Creek rather than to Silver Mine Creek. Spring Mine Creek flows from Spring Mine to the Area 5NW Pit. Wyman Creek flows from the Area 5SW Pit. Area 5N and Area 5W are not connected. A breach to the East would not reach Colby Lake. If the response does not correctly interpret the comment, then please clarify the comment and a follow-up response will be provided.

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23	pg 18	3.3, 2nd para	Please specify what is meant by "affect" by an FTB dam break.	LAM	S	"affected" means inundated as indicated by the estimated inundation areas referenced in the previous sentence and as shown in Figure 3 of Attachment H.
24	pg 19	4.2	Is there a contingency plan if there is insufficient water in the basin and Colby Lake water is not available due to low levels?	Dostert	3	Contingency mitigation for pond level management will be added in Section 6 (see response to Comment #41)
25	pg 20	4.2	Statement that during winter the "deposition will remain unchanged" doesn't align with first sentence on page 11 that describes subaqueous deposition being potentially more difficult in the winter.	Boyle	3	See response to Comment #18. Text will be updated to describe the additional operating activities (e.g., use of bubbler systems as an aid to maintaining open pond area) that may be implemented during winter operations that, while being routine methods of operation, would differ from summer operations.
26	pg 20	4.2	Is there a predicted operating range for pool elevation? (normal bounce?) What level will result in reducing inflow from external sources and what water level will result in removing water from the basin?	Dostert	4	It is assumed that the question refers to "When water elevation bounce occurs in the FTB as the result of large rain events or rapid snowmelt, the volume of make-up water from Colby Lake will be temporarily reduced until the water level in the FTB pond is reduced to the desired operating elevation". FTMP Version 4 will be updated to include a comprehensive response. The following is representative of the type of information that will be included in Version 4: A freeboard (distance from pond elevation to emergency overflow elevation which is where dam stability assessments have been made) will be an operational target. A large rain event or rapid snowmelt could raise the pond elevation by xx ft. The amount of water being added (and which could be reduced) varies over the life of the project from xx gpm to xx gpm. The conversion of this flow rate into a change in pond elevation depends on many factors but is roughly xx to xx inch per day. Water can be removed from the pond via the WWTP. The amount of water able to be removed varies over the life of the project from xx gpm to xx gpm. The conversion of this flow rate into a change in pond elevation depends on many factors but is roughly xx to xx inch per day. The two factors combined are included in project modeling and are expected to be able to manage the pond elevation.
27	pg 21	5.2	The instrumentation description is quite vague. There needs to be developed a specific plan showing locations of inclinometers, survey hubs, etc. At some point before a permit is issued, there needs to be more specifics provided.	LAM	P	This is permit level detail and will be developed in permitting and approved by the permitting agency.
28	pg 25	5.7	Would prefer that a "Special" or "non-Routine" inspection be implemented for unusual events as implementing a weekly inspection is unlikely to provide the detail needed for the event.	Dostert	3	Section 5.7 and Table 5-1 speak to "special" or "non-routine" events/observations that warrant a non-routine inspection. To be consistent with the intent of Section 5.7 as written, the word "weekly" in Section 5.7 will be replaced with the word "special". Please also see FTMP Attachment F Emergency Action Plan (to be renamed Contingency Action Plan) for further information on actions required if unusual events occur.
29	pg 28	6	Please also include Dam Safety in review of any changes that could impact reclamation as those changes could impact the closure pond.	Dostert	3	Agreed - text will be revised accordingly.
30	pg 29	7.2	The bentonite on the beaches will be subject to penetration by plant roots, desiccation cracking during droughts and riling and erosion. How will you prevent these types of damages?	Dostert	S	The bentonite-amended layer, at 30" below the surface, is below the root zone for many plant types - even deep rooted plants. As such, root penetration is not anticipated to be of signifant concern but is targeted for evaluation during facility inspections (see response to Comment #8). Once tailings deposition on the beaches has been terminated, as the tailings drain to their natural moisture content, dessication cracking may occur. However, after this drainage is complete, the tailings should be less susceptible to dessication cracking. Further, the depth of cracking will be limited by the presence of the bentonite amended layer, which will provide a natural barrier to tailings dessication at greater depth. Finally, while there is the potential for some erosion to occur before vegetation becomes well established, the beach slopes will be relatively flat (so low surface water runoff velocities) and the occurance of erosion after vegetation becomes established can be expected to be quite limited.
31	pg 30	7.2	The bentonite layer in the pond will be subject to ice push so it is unlikely that the layer will remain intact for an extended period of time. How will you prevent the damage from ice push?	Dostert	S	The meaning of "ice push" within the context of the FTB is unclear. After **reclamation** the pond in the FTP is expected to freeze in the winter in a manner experienced on a small lake. Since the pond surface will be completely frozen and/or the fetch would be too small to produce the massive wave action necessary to break up the ice and force large blocks of ice on shore, concerns about impacts to the rip-rap protected shoreline are unclear. Please clarify the comment and a follow-up response will be provided.

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32	pg 30	7.2	Literature goes both ways on bentonite being a barrier to root penetration. It works best as a root barrier when the material below the bentonite is high acid and works the poorest where bentonite overlies a ph neutral material. How will you determine when and where the bentonite layer has failed?	Dostert	3	The first complete paragraph on Page 72 (beginning with "Long term performance….") describes the inspections that will be performed and the actions that will be taken if evidence is found that the bentonite amended tailings layer is being detrimentally impacted by settlement, desiccation, erosion or root penetration. In the case of root penetration, root depths for regional plant types are generally known and will affect the plant types selected for vegetation of the beach and dam areas. Additional text will be added to the FTMP to add clarity regarding root depth. If additional concerns exist, please clarify the concerns and a follow-up response will be provided.
33	pg 30	7.2	How will you repair the bentonite when rills and erosion channels form in it on the side slopes? Dense vegetation will be impacted by time of year, wet or dry climate, fires, etc., and while mostly successful, will still be subject to failure.	Dostert	3	The FTB slopes, at 4.5H:1V are fairly flat such that the potential for development of uncontrolled erosion channels is somewhat reduced as compared to steeper slopes at the **existing** tailings basin and other tailings basins state-wide. If erosion does occur into or through bentonite amended zone, the appropriate segments of the eroded area will be backfilled with a soil-bentonite mix, will be covered and revegetated. Text of the FTMP will be amended to incorporate this comment response.
34	pg 31	7.4	Reference drawings show operations phase emergency spillway, where is emergency spillway after closure? Please update text to describe the location and any differences between the two.	Boyle	3	Drawings provided as Attachment A to the FTMP provide emergency spillway information for operations and after closure. FTMP text will be updated to create a stronger connection between the information provided on the drawings and the written content of the FTMP.
35	Appendix D		Will cold weather deposition result in different grain size distributions and different sedimentations in the basin?	Dostert	S	See response to Comment #18.
36	Instrumentation	2.1.1	There should also be a piezometer installed into the filter layer between LTVSMC and FTB Tailings due to the critical nature of this filter.	Dostert	P	This is permit level detail and will be developed in permitting and approved by the permitting agency.
37	General	General	The plan is primarily focused on management during mine operation and not 50 to 500 years later, which is a concern related to long term instability of the embankments due to erosion.	LAM	S	Section 7.5 of the FTMP discusses long term monitoring and erosion repair and Geotechnical Data Package - Vol. 1 - Ver. 3 shows that slope stability is adequate even in the event of a significant erosion event in the perimeter dam. Also see response to Comment #8. Please clarify the comment to more specifically state the basis for the concern and a follow-up response will be provided.
38	General	General	There seems to be needed explanations for the conclusions on whether the tailings will segregate by size (believe that coarse sizes near the discharge and the finer sizes further away) and a contradiction on water balance in that the existing tailings have a negative balance, yet the well-drained PolyMet tailings will result in a positive water balance.	LAM	S	See response to Comment #7 regarding particle size segregation and work at the SAFL previously reported to the MDNR. Water balance information for the FTB is contained in the NorthMet Project Water Modeling Data Package -Vol. 2. - Ver. 9, Sections 6.1.2 and 6.1.5
39	General	General	The long-term infiltration number of 6.5 in/year (or 5 x 10-7 cm/sec.) used in the AWMP and other PolyMet reports, appears to be very low for the bentonite-amended FTB pond bottom (specially given the uncertainties associated with accurate bentonite placement). This long-term infiltration value needs to be clearly justified.	LAM	S	The basis for the 6.5 in/year infiltration rate is reported in the NorthMet Project Adaptive Water Management Plan Version 5, March 7, 2013 which has already been reviewed and accepted.
40	General	General	General, subjective statements need to be made more definitive and quantitative for final/permitting efforts, specially when it comes to monitoring.	LAM	P	This is permit level detail and will be developed in permitting in coordination with and approved by the permitting agency.
41	General	General	We are anticipating that version 3 of the management plan will include discussion of plans for monitoring and mitigation.	ERM	3	Monitoring is discussed in Section 5 of the FTMP. Adaptive Management with Contingency Mitigation is added to Section 6 and/or to Attachment F Emergency Action Plan (renamed Contingency Action Plan).

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Remaining issues as of August 27, 2014.

Text is marked bold where I noted items that would be updated, modified, or where additional clarification was sought from the Co-lead agencies.

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1	pg 5	2.1	In several places in the report, the report notes that the dam design can be modified or the construction sequence can be altered as needed to conform to tailings characteristics from full-scale operations. This might leave an outside reviewer with little confidence in the current design. Explanation of the observation method and how it relates to the EIS may be warranted.	Boyle	3	See Response to Comment #41. **The existing Flotation Management Plan (FTMP) discussion on mitigation will be expanded**.
3	pg 5	2.1.1	Flotation tailings samples were also ran in 2006.	LAM	3	It is true that a Flotation Process Optimization test was run in 2006. However, samples from that test were collected for environmental purposes (e.g., waste characterization, air quality, water balance, etc.) and not for geotechnical purposes. Only gradation testing was performed for geotechnical purposes. Therefore there is little significance of the 2006 test to the geotechnical design effort. **The following edit will be made to Section 2.1.1: "Flotation Tailings samples were collected from the pilot-test runs in 2005, 2006, 2008, and 2009. "**
4	pg 6	2.2.1	Please describe the emergency overflow systems either in this section or 2.2.4 or 7.4.	Boyle	3	The emergency overflow structure is described in FTMP Section 7.4 and the location and layout of the emergency overflow is provided in Drawings FTB-015 to FTB-018. Also see response to Comment #34. **Please advise if more information is required.**
5	pg 6	2.2.1	Is there an environmental risk to allowing "routine overflow via the emergency overflow outlets to the adjacent wetlands"?	Boyle	3	The reference to "routine overflow..." is after reclamation, when the pond water meets applicable water quality standards or the overflow can be treated by non-mechanical treatment systems. Prior to that time, the WWTP will treat water from the FTB South Surface Seepage Management System, FTB Containment System, HRF drainage water and the FTB pond as needed to prevent overflow. The WWTP will be maintained operable until the MDNR releases the company from mechanical water treatment requirements under the Permit to Mine. **Operation of the WWTP during long-term closure is discussed in Section 4.2 of the NorthMet Project Adaptive Water Management Plan - Version 5, March 7, 2013. Text in 2.2.1 will be modified to reflect this.**
6	pg 6	2.2.1	There seems to be a stability issue that conflicts with the water holding issue.	LAM	S	This **comment is unclear**; the stability analysis presented in Geotechnical Data Package - Vol. 1 demonstrates adequate stability of the Flotation Tailings Basin (FTB) during operations and at closure. Both conditions have been analyzed for stability with the proposed pond in place and slope stability safety factors meet MDNR requirements. Further, this topic has been discussed with the MDNR and it has already been agreed that the permanent pond is part of the mine waste management plan which is intended to minimize the amount of oxygen reaching the Flotation Tailings at closure. **If more information is required, please clarify the comment and a follow-up response will be provided.**
8	pg 6	2.2.2	The statement that "The dam design can be modified, if necessary, based on results of performance and stability monitoring." This is true during years 0-20 +/-, but not during years 20-900. How are observed erosion rates incorporated in the 900 year design? Are erosion rates going to be measured in the future?	LAM	S	It appears that this comment relates to Section 2.2.1 rather than Section 2.2.2. The dam is most susceptible to erosion during and for several years after construction; prior to establishment of a dense vegetative cover and root mass. To account for this, inspection and maintenance of erosion is required by Sections 5.5, 5.6, 5.7 and 5.8 of the FTMP. Some variation of these inspections will be carried on after closure, during reclamation and continuing into long term care. **Inspection and erosion repair will be included in financial assurance.**
9	pg 7	2.23, 2nd para	Is Appendix G the right reference?	LAM	S	**This comment is unclear**; the FTMP does not include a Section 2.23 nor an Appendix G. If this is in reference to "Additional information on borrow material and preliminary specifications can be found in Attachment G" at the end of the 2nd paragraph of Section 2.2.4; Attachment G is the correct reference and will remain in Version 3.

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26	pg 20	4.2	Is there a predicted operating range for pool elevation? (normal bounce?) What level will result in reducing inflow from external sources and what water level will result in removing water from the basin?	Dostert	4	It is assumed that the question refers to "When water elevation bounce occurs in the FTB as the result of large rain events or rapid snowmelt, the volume of make-up water from Colby Lake will be temporarily reduced until the water level in the FTB pond is reduced to the desired operating elevation". **FTMP Version 4 will be updated to include a comprehensive response.** The following is representative of the type of information that will be included in Version 4: A freeboard (distance from pond elevation to emergency overflow elevation which is where dam stability assessments have been made) will be an operational target. A large rain event or rapid snowmelt could raise the pond elevation by xx ft. The amount of water being added (and which could be reduced) varies over the life of the project from xx gpm to xx gpm. The conversion of this flow rate into a change in pond elevation depends on many factors but is roughly xx to xx inch per day. Water can be removed from the pond via the WWTP. The amount of water able to be removed varies over the life of the project from xx gpm to xx gpm. The conversion of this flow rate into a change in pond elevation depends on many factors but is roughly xx to xx inch per day. The two factors combined are included in project modeling and are expected to be able to manage the pond elevation.	See Response to Comment 4 on "Comments on Ver 4" Tab
27	pg 21	5.2	The instrumentation description is quite vague. There needs to be developed a specific plan showing locations of inclinometers, survey hubs, etc. At some point before a permit is issued, there needs to be more specifics provided.	LAM	P	**This is permit level detail and will be developed in permitting and approved by the permitting agency.**	
28	pg 25	5.7	Would prefer that a "Special" or "non-Routine" inspection be implemented for unusual events as implementing a weekly inspection is unlikely to provide the detail needed for the event.	Dostert	3	Section 5.7 and Table 5-1 speak to "special" or "non-routine" events/observations that warrant a non-routine inspection. To be consistent with the intent of Section 5.7 as written, **the word "weekly" in Section 5.7 will be replaced with the word "special"**. Please also see FTMP Attachment F Emergency Action Plan (to be renamed Contingency Action Plan) for further information on actions required if unusual events occur.	OK
29	pg 28	6	Please also include Dam Safety in review of any changes that could impact reclamation as those changes could impact the closure pond.	Dostert	3	Agreed - **text will be revised accordingly.**	OK
31	pg 30	7.2	The bentonite layer in the pond will be subject to ice push so it is unlikely that the layer will remain intact for an extended period of time. How will you prevent the damage from ice push? RESPONSE By ice push, I mean the pond freezing, the bottom of the ice at depths up to 5 feet will be sitting on the bentonite. As the ice expands, It will scour the bentonite and push it towards shore, eventaully making bentonite beaches and leaving scoured areas between3 to 5 feet deep. (Beaches in the context of a normal water body, not tailings beach).	Dostert	S	The meaning of "ice push" within the context of the FTB is unclear. After reclamation the pond in the FTP is expected to freeze in the winter in a manner experienced on a small lake. Since the pond surface will be completely frozen and/or the fetch would be too small to produce the massive wave action necessary to break up the ice and force large blocks of ice on shore, concerns about impacts to the rip-rap protected shoreline are unclear. **Please clarify the comment and a follow-up response will be provided.**	see response
32	pg 30	7.2	Literature goes both ways on bentonite being a barrier to root penetration. It works best as a root barrier when the material below the bentonite is high acid and works the poorest where bentonite overlies a ph neutral material. How will you determine when and where the bentonite layer has failed? RESPONSE: How will you keep undeireable plants from piercing the bentonite. Dandelions can grow to 15' deep.	Dostert	3	The first complete paragraph on Page 72 (beginning with "Long term performance….") describes the inspections that will be performed and the actions that will be taken if evidence is found that the bentonite amended tailings layer is being detrimentally impacted by settlement, desiccation, erosion or root penetration. In the case of root penetration, root depths for regional plant types are generally known and will affect the plant types selected for vegetation of the beach and dam areas. **Additional text will be added to the FTMP to add clarity regarding root depth. If additional concerns exist, please clarify the concerns and a follow-up response will be provided.**	see response
33	pg 30	7.2	How will you repair the bentonite when rills and erosion channels form in it on the side slopes? Dense vegetation will be impacted by time of year, wet or dry climate, fires, etc., and while mostly successful, will still be subject to failure.	Dostert	3	The FTB slopes, at 4.5H:1V are fairly flat such that the potential for development of uncontrolled erosion channels is somewhat reduced as compared to steeper slopes at the existing tailings basin and other tailings basins state-wide. If erosion does occur into or through bentonite amended zone, the appropriate segments of the eroded area will be backfilled with a soil-bentonite mix, will be covered and revegetated. **Text of the FTMP will be amended to incorporate this comment response.**	OK

34	pg 31	7.4	Reference drawings show operations phase emergency spillway, where is emergency spillway after closure? Please update text to describe the location and any differences between the two.	Boyle	3	Drawings provided as Attachment A to the FTMP provide emergency spillway information for operations and after closure. **FTMP text will be updated to create a stronger connection between the information provided on the drawings and the written content of the FTMP.**	
36	Instrumentation	2.1.1	There should also be a piezometer installed into the filter layer between LTVSMC and FTB Tailings due to the critical nature of this filter.	Dostert	P	**This is permit level detail and will be developed in permitting and approved by the permitting agency.**	OK
37	General	General	The plan is primarily focused on management during mine operation and not 50 to 500 years later, which is a concern related to long term instability of the embankments due to erosion.	LAM	S	Section 7.5 of the FTMP discusses long term monitoring and erosion repair and Geotechnical Data Package - Vol. 1 - Ver. 3 shows that slope stability is adequate even in the event of a significant erosion event in the perimeter dam. Also see response to Comment #8. **Please clarify the comment to more specifically state the basis for the concern and a follow-up response will be provided.**	
40	General	General	General, subjective statements need to be made more definitive and quantitative for final/permitting efforts, specially when it comes to monitoring.	LAM	P	**This is permit level detail and will be developed in permitting in coordination with and approved by the permitting agency.**	

Document Review & Verification: NorthMet Project Flotation Tailings Basin Management Plan Version 5 (MARCH 2015)

Reviewers: DNR Dam Safety (Boyle, Dostert); DNR LAM (Kunz); KP (Coffin)

Comment Number	Location (Doc, Page, Figure, or Table)	Section Number	Reviewer	Comment	Proposer Response 03/04/2015	Agency Review & Verification 03/04/2015
1	page 7	2.2.3	DNR Dam Safety - Dostert	Am uncomfortable with the return periods listed for the PMP (100 million to 10 billion years). PMP's are controversial due to the methods and accuracy of the computations used to make them. Unlike the PMF, no recurrence interval is suggested with a PMP. It is simply the Probable Maximum Precipitation that could occur. Recommend removing all discussions related to a recurrence intervals, other than it is a very rare event but possible during the 20 years of basin operations, and during the 1000 year recovery period.	It is agreed that the PMP does not have an assigned return period and that hydrologist's opinions vary as to what return period should be assigned to a PMP. Section 2.3.3 text has been adjusted accordingly.	
2	page 16	3.1	DNR Dam Safety - Boyle	Text below table 3-2 mixes references to table 3-1 and 3-2	Text has been updated in Flotation Tailings Management Plan - Version 5.	
3	Drawings		DNR Dam Safety - Boyle	Drawings use term "bulk" tailings for embankments, while Management Plan uses term "coarse" tailings.	Drawings have been updated to use the term "course" tailings for embankments.	
4			DNR Dam Safety	See orange highlighted cell on remaining issues tab (response to a review comment on a previous version)	WWTP design capacity will be finalized in conjunction with facility permitting, at which time flow rate data relative to pond bounce management can be provided. For FTMP – Version 4, Section 4.2 has been updated to provide additional context for future definition of pond elevation targets for routine operating conditions. More generally and on a preliminary basis, for Cells 1E and 2E combined, each 1-inch rainfall event will add on the order of 40,000,000 gallons of water to the tailings basin (neglecting consideration of evaporation and infiltration). Assuming a 2,000 gpm water withdrawal rate, to the WWTP and from make-up water reductions, the time required for draw-down to pre-event elevations would be on the order of 14 days. However, a 1-inch pond bounce is inconsequential relative to beach length; representing an 8-foot reduction in beach length at a beach slope of 1%. Therefore, a substantially longer time period could be utilized to draw down the pond (i.e., use of substantially lower withdrawal rates) without affecting overall basin stability and operating conditions.	
5	Page	5	KP	Suggest changing "The current ore processing plant… does not change the characteristics of the tailings…" to "The current ore processing plant…. Is not anticipated to significantly change the characteristics of the tailings…"	This suggested edit has been incorporated in Section 2.1.	
6	Page	8	KP	Suggest adding text regarding the degree to which the bentonite amendment will limit oxygen infiltration.	Detailed discussion of the Geochemical Parameters for the Flotation Tailings Basin, including bentonite-amended tailings, is presented in the Waste Characterization Data Package - Version 12 and therefore is not presented in nor proposed to be duplicated in the Flotation Tailings Management Plan.	
7	Page	9	KP	Will the shear walls be extended through the peat and into the competent soils, or just placed within the slimes and fine tailings? Will they be placed within zones of coarse tailings?	Location and configuration of the Cement Deep Soil Mix (CDSM) shear walls are described in Geotechnical Data Package - Volume 1 - Version 6. Minor text edits regarding shear wall location have been incorporated into Flotation Tailings Management Plan - Version 5.	

8	Page	11	KP	Suggest adding text regarding a field trial to validate design concept of shear walls. Additionally, suggest adding QA/QC procedures during construction pertaining to the shear wall placement.	Section 2.2.4 text has been updated to address concept validation and testing. Details of concept validation and testing are beyond the scope of the FTMP and will be prepared as part of future Dam Safety permitting activities.	
9	Page	15-16	KP	The term "above and beyond" would suggest that the buttress and shear walls are unnecessary to achieve required minimum values. Suggest modifying text accordingly.	The phrase "above and beyond" refers to the fact that computed safety factor values for the FTB dams often exceed the minimums required. However, the phrase "above and beyond" has been removed from Section 3.1 text.	
10	Page	24	KP	Are current water transport and treatment systems (plant, pipes, pumps, etc.) sized to accommodate a higher flow rate than current design parameters? Suggest adding text regarding ability to transport and treat additional waters if available or other mitigations (i.e., additional temporary pumps and pipes). Will this higher throughput impact the 'water side' of the project?	Management of additional water will be primarily by two methods; increasing the rate of discharge through the plant site Waste Water Treatment Plant (WWTP), and by reducing the rate of make-up water addition to the system. Details of the sizing of the WWTP are beyond the scope of the FTMP but are being planned relative to Flotation Tailings Basin operating requirements. Pumping and piping systems have been preliminarily sized. Final design details will be developed as part of final system design. Use of temporary pumping and piping systems is a routine approach to transporting additional water when required and such an approach would be used at this facility as needed.	
11	red-line version - pg 5	Sec 2.1.1, 2nd sentence	DNR - LAM	Is this no longer a true statement? Because removing it, does not flow with the next sentence, starting with "Additional" and following sentences explaining the various pilot test years (now wording omits first year - 2005)?	The sentence in question was deleted from v4 because the results of the 2005 pilot test are no longer fully representative of the expected characteristics of the Flotation Tailings. However, for v5 the sentence has been reinstated with other wording changes to clarify the differences between the pilot tests in the various years.	
12	red-line version - pg 14	last sentence	DNR - LAM	Has something changed in water balance to revise from "will be overflow" to "may be occasional overflow"?	This edit will be reversed; because there is a net positive water balance in the region it is anticipated that there will be occasional overflow after FTB final closure.	
13	red-line version pgs 16-19	Tables 3-1 thru 3-5	DNR - LAM	Updates to come after completion of Geotech Data Package - Vol 1, Ver 5.	Updates have been incorporated into Version 5.	

CENTER for SCIENCE in PUBLIC PARTICIPATION

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"Technical Support for Grassroots Public Interest Groups"

March 14, 2014

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RE: Comments on NorthMet Mining Project and Land Exchange Supplemental Draft Environmental Impact Statement, prepared by Minnesota Department of Natural Resources, United States Army Corps of Engineers, United States Forest Service, November, 2013

The Center for Science in Public Participation provides technical advice to public interest groups, non-governmental organizations, regulatory agencies, mining companies, and indigenous communities on the environmental impacts of mining. CSP2 specializes in hard rock mining, especially with those issues related to water quality impacts and reclamation bonding.

Dr. David Chambers has 37 years of experience in mineral exploration and development – 15 years of technical and management experience in the mineral exploration industry, and for the past 22 years he has served as an advisor on the environmental effects of mining projects both nationally and internationally. He has Professional Engineering Degree in Physics from the Colorado School of Mines, a Master of Science Degree in Engineering from the University of California at Berkeley, a Ph.D. in Environmental Planning from Berkeley, and is a registered professional geophysicist in California (# GP 972).

Stuart M. Levit has a Masters Degree in Land Rehabilitation from Montana State University in 1989, where he focused on natural resources issues, mining, and environmental regulation, and a law degree from the University of Montana in 1994. He worked for the Montana Abandoned Mines Reclamation Bureau where he designed reclamation projects, and was oversaw site evaluation, engineering, and construction of these projects. Stu has worked with the Center for Science in Public Participation since 2000, concentrating on reclamation planning and bonding, the adequacy of environmental impact statements, and water pollution regulatory issues.

SUMMARY

The NorthMet Project Supplemental Draft EIS is based on a plethora of information, but there are still several important areas that need additional work and/or revisions:

(1) Probably the most glaring omission is that there is only the most scant analysis of the financial surety that will be needed for this project. As is discussed in more detail to follow, the financial surety for this project could be in excess of $400 million. This is very significant potential impact not only to the financial requirements of the mine owners, but also to the citizens of Minnesota, who are ultimately accountable should the mine operator go bankrupt without an adequate financial surety to

close the mine and treat waste water. If the mine operator were to go bankrupt without an adequate financial surety public funds for closure would either need to be provided, or the public would bear the environmental consequences of not properly closing the mine. Either way, the public would pay, probably for centuries.

(2) Every Draft EIS should contain a Draft Reclamation Plan which includes a detailed analysis of the financial surety, so that it is demonstrated before the project is allowed to proceed that project closure can be accomplished using demonstrated reclamation techniques, and at a cost that is affordable. Lack of a viable reclamation plan and closure cost estimate is a major flaw in the SDEIS.

Many sections of the Reclamation Plan (and its referenced documents, such as the Mine Plan, Water Management Plan, Wetlands Management Plan are more accurately plans to plan reclamation than actual reclamation plans. As described in these comments, the Reclamation Plan often lack detail and specific goals and methods necessary to evaluate the likely success of reclamation and provide comment about how to improve the chances of success.

These are the most critical issues, but there are a number of other important points to be made in the Section-Specific Comments below.

SECTION-SPECIFIC COMMENTS:

2.6 Financial Assurance

This section, which should be one of the major points of analysis and public disclosure, is a mere ½ page in length. It is stated:

> *"The level of engineering design and planning required to calculate detailed financial assurance amounts is typically made available during the permitting process." (SDEIS, p 2-10)*

This statement is most probably not correct. The engineering design and planning to calculate detailed financial assurance is available at the point a project reaches the Draft EIS stage. For a company to reach this stage without having this information available, and without having performed this calculation internally, would be fiscally irresponsible to the company's shareholders and board of directors. Management could not go forward with a proposal to the company board to spend the hundreds of millions/billions of dollars involved in constructing a mine of this type without knowing what the likely closure costs will be, since those costs are also likely in the hundreds of millions of dollars range.

Likewise, the public is the final resting point for any financial surety. If there is a bankruptcy or other applicable failure to perform that requires the use of the financial surety, and this surety is lacking the public either makes up the deficit with tax dollars, or bears the environmental costs of not performing the reclamation, water treatment, or any other costs related to the deficit in the financial surety. With this much money at stake, regulatory agencies should want the most thorough review possible before allowing a project to go forward.

Recommendation: A detailed financial surety calculation, based on the project as proposed by the applicant, should be presented in the SDEIS.

3.1.1 NorthMet Project Proposed Action Overview

3.1.1.7 Project Closure Overview

A very important component of the financial assurance, long term water treatment, is mentioned in this section:

> *"Mechanical water treatment is part of the modeled NorthMet Project Proposed Action for the duration of the simulations (200 years at the Mine Site and 500 years at the Plant Site). The duration of the simulations was determined based on capturing the highest predicted concentrations of the modeled NorthMet Project Proposed Action. It is uncertain how long the NorthMet Project Proposed Action would require water treatment, but it is expected to be long term;* ***actual treatment requirements would be based on measured, rather than modeled****, NorthMet Project water quality performance, as determined through monitoring requirements." (SDEIS, p 3-5,* ***emphasis added****)*

It is appropriate to base models like this on worst-case predictions, because there are too many examples of situations where conditions turned out to be worse than the worst-case prediction. It is also appropriate to eventually base water quality predictions and the need for water quality treatment on actual conditions, once a mine is in operation. However, for a new mine the initial calculations of a financial surety must be based the best information available, because the mine could go bankrupt before a sufficient amount of data has been collected to establish an operational baseline. An initial calculation of a financial surety must necessarily be based on worst-case water quality predictions – until operational data can prove conditions to the contrary.

In effect, from a worst case standpoint any prediction of poor water quality from a model that extends more than 5 years beyond the closure of a mine must essentially be treated as requiring water treatment in perpetuity. (For example, it might take 5 years for tailings pond seepage to stabilize post-closure.) This is because no geochemical model today is accurate enough to predict how much time it will take for water quality to improve enough to terminate water treatment required after mine closure.

Recommendation: The initial calculation of the financial surety should be based on worst-case water quality predictions, and on the project as proposed by the applicant.

3.2.2.1.6 Haulage, Storage, and Transport of Ore

Ore will be transported from the mine to the mill in side-dump rail cars. (see Figure 3.2-21: Side Dump Railroad Cars, SDEIS, p 3-85): at the Rail Transfer Hopper haul trucks dump run-of-mine ore into the hopper, where it is loaded by conveyor onto the rail cars. Each 100-ton rail car can be loaded in one minute (SDEIS, p 3-43). There is no mention of an enclosed building, so it is assumed that this is being done in the open. Since this is run-of-mine ore there will be a fraction of the ore that is dust-sized material. The high transfer rate both with the haul trucks dumping to the hopper, and the conveyor loading the rail cars, will generate a significant amount of dust in the vicinity, and in the predominant downwind direction of the Rail Transfer Hopper. And, since the rail cars are open-top, there will be an accumulation of dust along the rail corridor between the mine and mill.

Ore-dust contains a number of potential contaminants which, even though a low concentrations, could accumulate over years of use.

Recommendation: The Rail Transfer Hopper and rail car loading conveyor and platform should be in an enclosed structure, and the rail cars fitted with a top that would limit the loss of ore-dust along the rail line between the mine and mill.

3.2.2.1.8 Engineered Water Controls

Category 1 Stockpile Water Containment System and Cover

Waste Rock Liner and Cover Systems

A cutoff wall would be constructed to enclose the Category 1 waste rock pile to prevent metals leaching contaminants from reaching groundwater.

> *"The cutoff wall would be constructed by excavating a trench down to bedrock and backfilling it with a compacted soil material or by placing a manufactured **geosynthetic clay barrier** in the trench. Compacted soil material would have a **hydraulic conductivity specification of no more than 1x10^{-5} centimeters per second** (cm/sec)." (SDEIS, p 3-46, **emphasis added**)*

At a permeability 1x10^{-5} cm/sec, a fluid will move thorough this material at a rate of slightly more than 10 feet/year. For a 3 foot wide cutoff wall that means 3+ pore volumes would pass through the wall each year.

If the permeability is lowered to 1x10^{-6} cm/sec, a fluid will move through the wall at a rate of 1 foot/year. For a 3 foot wide cutoff wall this means that it would take one pore volume 3 years to pass through the wall.

Recommendation: A permeability of 1x10^{-6} or less should be the goal of the cutoff walls for the Category 1 waste rock and the tailings cutoff wall.

Geosynthetic clay barriers can significantly increase the effectiveness of a cutoff wall, but it is not clear exactly what would trigger the use of this barrier. And, if installed, the barrier is likely to decrease the permeability of the section of the cutoff wall significantly over sections that do not have the barrier. It would be more consistent to use a barrier for the entire wall, or not at all.

Recommendation: More definition needs to be provided in when a geosynthetic clay barrier would be employed.

Category 2/3 and 4 Stockpiles and Ore Surge Pile Liners

In Table 3.2-9 Summary of the Stockpile Liners and Covers (SDEIS, p 3-51) the liners that will be employed for the 3 categories of waste rock and the ore surge pile are described. The Category 2/3 Stockpile liner: "12-inch compacted (1x10^{-5} cm/s) subgrade overlaid by 80-mil LLDPE geomembrane, covered by a 24-inch overliner drainage layer." The other liners have (1x10^{-6} cm/s) subgrades. Even though the effective permeability of the total liner system is governed by the permeability of the 80-mil LLDPE geomembrane, since 1x10^{-6} cm/s is essentially the minimum standard for a liner, for sake of standardization the Category 2/3 Stockpile liner should also have a 1x10^{-6} cm/s subgrade.

Recommendation: For standardization, both on the project and with most other liners, all liners made of natural materials should have a permeability of 1x10^{-6} or lower.

3.2.2.3.9 Transport of Consumables and Products

> *"Nickel and cobalt hydroxide and precious metal precipitate products would be shipped in sealed bulk bags or sealed containers. Copper and nickel concentrates would be shipped in solid-bottom rail cars with weather-tight covers. Cars would be checked before loading and any debris would be removed and holes plugged. Loading operations would be conducted in a building via a conveyor system. Car exteriors would be inspected before leaving the buildings and any concentrate on the car exterior would be recovered and returned to storage. The concentrate is expected to be 8 percent to*

> *10 percent moisture,* ***which is not expected to generate dust during loading****." (SDEIS, p 3-116,* ***emphasis added)***

This is all good except for the last phrase (**emphasized above**). Inevitably there will, at the very least, be differences in the moisture content of the concentrate due to the amount of time it has been stored, or other factors. Concentrate ALWAYS creates dust at some point. If the rail cars are loaded in an enclosed building, this should limit the escape of dust to the environment, but to assume that dust will not occur, or that there will no leakage of small amounts of concentrate along the transportation corridor, would be an oversight.

In a related comment from Table 8-1 Major Differences of Opinion:

> *"GLIFWC disagrees that the amount of ore that could escape from rail cars would be small because the rail cars proposed for use are not sealed. GLIFWC states that, given the design and current condition of rail cars proposed for transport, an ecologically significant amount of spillage could occur into streams, wetlands, and their watersheds. GLIFWC believes that fugitive dust escaping through gaps in the rail cars is also a concern. GLIFWC does not believe that the method described to segregate fines in the center of the rail car, away from the gaps, is realistic. Further, GLIFWC does not believe that monitoring of the creeks along the rail line will be effective in preventing or minimizing impacts because once detected in monitoring, the impact will have already occurred. GLIFWC states that cleanup of ore dust in an aquatic environment is a long and difficult process." (SDEIS, p 8-18)*

This mirrors a CSP2 comment on the SDEIS, and we echo that comment, and support the GLIFWC comment above.

3.2.2.4 Financial Assurance

3.2.2.4.1 Cost Coverage and Estimation

The financial burden associated with closing the mine is largely incurred in the first few years of the mine's life. When a mine is developed much of the disturbance takes place in the first few years of operation. As the mine operates the pit will get deeper, and waste piles larger, but the plant area, infrastructure, and tailings impoundment will stay largely the same.

Mine closure and post-closure costs represent a significant potential impact to the public that should be discussed as a part of the process. This cost estimate would be used to establish the amount of the post-closure financial surety, until it is amended by subsequent estimates that are based on actual operating costs and pre-closure reclamation experience at the site. However, it is stated:

> *"The level of engineering design and planning required to calculate detailed financial assurance amounts is typically made available during the permitting process and was not available at the time that this SDEIS was prepared." (SDEIS, p 3-136)*

It would be equally correct to say 'The level of engineering design and planning required to calculate detailed financial assurance amounts is typically available during the EIS process.' The basic reason is that the operator must understand what the costs of reclamation and closure will be (and how that will influence the economics of the mining operation). Since the amount of money involved in reclamation and closure can be tens to hundreds of millions of dollars, management that does not forecast these expenditures for its own internal cost analysis would not be proposing a project that is fiscally defensible to its board of directors.

An example of a reclamation plan with an appropriate level of planning detail and cost estimates is the Pogo Project Reclamation and Closure Plan, December 2002. The document is available digitally through the Alaska Department of Natural Resources, Anchorage, AK, and was is titled the Final Environmental Impact Statement, Pogo Gold Mine Project, USEPA, Region 10, September, 2003.

A "preliminary" reclamation closure cost estimate is presented in SDEIS Table 3.2-15 Preliminary Cost Estimate for Closure (SDEIS, p 3-138). However, this "preliminary closure cost estimate" is not supported by calculations and/or detailed information. One of the support documents "Proposed SDEIS Financial Assurance Language" (Foth, 2013) does discuss mine closure, but again does not provide sufficient documentation to substantiate this closure cost estimate. Foth, 2013, does contain the table identical to SDEIS Table 3.2-15.

Table 1
Closure Cost Estimate
(Chapter 6132.1200 Financial Assurance)

	Year of Closure (end of year)			
	Year 1	Year 11	Year 20	Steady State Long-Term Care
TOTAL				
Estimated Range	\$50,000,000 - \$90,000,000	\$160,000,000 - \$200,000,000	\$120,000,000 - \$170,000,000	\$3,500,000 - \$6,000,000

Note: Costs based on preliminary reclamation estimates prepared by PolyMet. There are uncertainties involved in calculating Financial Assurance estimates before completing the final project design, operating practices, and permit requirements. The final financial assurance estimate completed under the Permit to Mine application may fall outside the estimate ranges above.

Prepared by: CED1
Checked by: KKB

Foth, 2013, also contains this explanation:

> *"**In addition to the costs of the closure reclamation plan**, PolyMet will also incur ongoing annual monitoring costs after water management as the project site reaches steady state. Depending on the year of closure, those costs are estimated to range from \$3,500,000 to \$6,000,000. Once again, these estimated costs include a 10% contingency factor." (Foth, 2013, p 7, **emphasis added**)*

From Table 1, above, the maximum liability for reclamation only would be incurred in Year 11 - \$200 million.

And, according to the wording of the paragraph cited above, there is additional long-term care that would amount to \$3.5 - \$6.0 million/year. The long-term care cost is assumed to be operating and maintenance costs (lacking further explanation).

In order to develop a ballpark estimate of the financial surety that might be required for the NorthMet project, CSP2 developed a Net Present Value (NPV) spreadsheet to estimate the NPV of the long-term costs.

Table 2: Assumptions for NorthMet Net Present Value Calculations, contains a list of the basic assumptions made for this model. These assumptions include the direct costs of constructing the Wastewater Treatment Facility, the Central Pumping Station, and the Treated Water Pipeline. These

capital cost estimates are taken from Updated NI 43-101 Technical Report on the NorthMet Deposit, AGP Mining Consultants Inc., January 14, 2013, Table 21-4: Initial Capital Costs, p. 21-8.

The indirect costs include: plan of operations scope contingency, contracting bid contingency, mobilization/demobilization, engineering design, performance & payment bonds, estimated sales tax, contractor profit & overhead, agency project & contract management, and annual inflation. These cost categories come from Training Guide for Reclamation Bond Estimation and Administration, for Mineral Plans of Operation Authorized and Administered under 36 CFR 228A, USDA Forest Service, Minerals and Geology Management, April 2004. The Forest Service Training Guide suggests the range of indirect costs for a project might range from 39% to 128%, depending on site-specific circumstances. The indirect costs assumed for the NPV spreadsheet total 52%, which is in the lower range of that suggested by the Forest Service Training Guide.

Table 2: ASSUMPTIONS FOR NORTHMET NET PRESENT VALUE CALCULATIONS

BASIC ASSUMPTIONS:		
	Plan of Operations Scope Contingency*	10%
	Contracting Bid Contingency*	15%
	Mobilization/Demobilization (%)*	1%
	Engineering Design (%)*	2%
	Performance & Payment Bonds (%)*	3%
	Estimated Sales Tax (%)*	0%
	Contractor Profit & Overhead (%)*	15%
	Agency Project & Contract Management (%)*	6%
	Inflation*	2.0%
	Real Discount Rate	5.0%
	Nominal Discount Rate	5.1%
	{Total Indirect Costs (%)}	54%
	Capital Equipment Replacement Interval (yrs)	50
	Replacement Plant Cost (% of original cost)	100%
CAPITAL COSTS:		
	Waste Water Treatment Facility	$4,553,000
	Central Pumping Station	$1,781,000
	Treated Water Pipeline	$2,303,000
ANNUAL OPERATIONS & MAINTENANCE COSTS	(millions)	$3.5 - $6.0

For the spreadsheet it was assumed that the water treatment facilities would need to be replaced every 50 years, and that the replacement plant cost is the same as the original (in present day dollars). This essentially means that the replacement plant and support facilities will be of the same capacity as the existing design.

The annual operating costs of the treatment plants come from PolyMet's Table 1, Closure Cost Estimate, and for the spreadsheet it is assumed that the operating cost estimates already include indirect costs (agency management, profit & overhead).

NPV Spreadsheet Results and Sensitivity

The most informative way to view the results of the NPV spreadsheet is to look first as the Base Case assumptions, then to perform some sensitivity runs changing the variables to see which of them have the most pronounced effect on the net present value of the financial assurance.

In Table 3: NorthMet Closure Financial Surety Calculation - Sensitivity Analysis, for the Base Cases it was assumed that the Real Interest Rate, the difference between the discount rate (5%) and inflation (2%), was 3 percent. This is a fiscally conservative, but realistic, assumption.

NorthMet Base Case Operating Costs

For the Base Case runs, assuming operating cost of $3.5 million/yr leads to a financial surety of about $137.4 million (line 1), and if the operating costs are $6 million/yr the financial surety would be $222.2 million (line 2) – for water treatment alone.

Capital Cost and Replacement Interval for the Water Treatment Plant

The NPVs are relatively insensitive to the capital replacement cost interval, as can be seen by comparing the NPVs in line 1 to line 3, and line 2 to line 4 of the Table, which vary by less than 3%. Going to a 25-year replacement schedule does make a 6.2% difference (comparing line 1 to line 5), but a 25-year replacement interval is too short to be realistic. The NPVs are also relatively insensitive to the capital cost of the treatment plant. If the capital cost is increased from $14.5 million (line 2) to $25 million, the increase in cost (72% cost increase, see line 6) results in only a 6% increase in the NPV; and, a capital cost increase to $50 million (245% cost increase, see line 7) results in only a 21% increase in the NPV.

Real Interest Rate – the difference between the rate of return and inflation

The spreadsheet is most sensitive to the Real Interest Rate. If the Real Interest Rate goes from 3% to 4%, the NPV of the financial surety goes down from $222.2 (line 2) to $171.4 million (line 8). On the other hand, if the Real Interest Rate goes from 3% to 2%, the NPV goes from $222.2 (line 2) to $323.1 million (line 9). If the Real Interest Rate goes from 3% to 1%, the NPV of the financial surety goes from $222.2 (line 2) to $590 million (line 10). The spreadsheet calculations are most sensitive to variation in the Real Interest Rate. This means that long term management of the financial surety corpus is very important, and that uncontrollable variations in the Real Interest Rate could significantly impact the corpus if it remains low for a significant period of time.

Variations in Water Treatment Plant Operating Costs

The NPV is marginally sensitive to change in operating costs. Increasing the operating costs from $3.5 million to $4.0 million per year, the NPV goes from $137.4 million (line 1) to $154.4 million (14% increase in operating cost, 12% increase in NPV – line 12). An annual operating cost of $5.0 million increases the NPV to $188.2 million (43% increase in operating cost, 37% increase in NPV – line 13).

The interest and return on investment rates are purely a choice that government agencies will make, and are not dependent on a mine plan. The operating costs do require a detailed calculation, but again the design for water treatment in the water management plan are going to be estimates no matter the final mine plan. A calculation of water treatment based on PolyMet's proposed alternative would provide a very reasonable base for calculating operating costs for discussion in the SDEIS.

Recommendation: Because the interest and return on investment rates, and the operating costs, can make hundreds of millions of dollars difference in the financial surety required for the NorthMet mine, these assumptions/calculations should be made available for comment in the SDEIS.

Table 3: NorthMet Closure Financial Surety Calculation - Sensitivity Analysis

Line Number	Governing Assumption	Plant Capital Cost	Plant Replacement Interval (yrs)	Annual Operating Cost	Inflation	Real Discount Rate	NPV @ 200 years	NPV @ 300 years	NPV @ 500 years
	NorthMet Base Case Operating Costs								
1	Low Operating Cost Estimate	$14,423,790	50	**$3,500,000**	2.0%	5.0%	$137,070,730	$137,355,404	$137,370,377
2	High Operating Cost Estimate	$14,423,790	50	**$6,000,000**	2.0%	5.0%	$221,622,414	$222,103,131	$222,128,416
	Capital Cost and Replacement Interval								
3	100 yr Replacement Interval	$14,423,790	**100**	$3,500,000	2.0%	5.0%	$133,577,872	$133,854,200	$133,868,735
4	100 yr Replacement Interval	$14,423,790	**100**	$6,000,000	2.0%	5.0%	$218,129,555	$218,601,927	$218,626,773
5	25 yr Replacement Interval	$14,423,790	**25**	$3,500,001	2.0%	5.0%	$146,106,338	$146,412,600	$146,428,629
6	High Plant Capital Cost	**$25,000,000**	50	$6,000,000	2.0%	5.0%	$235,332,951	$235,821,158	$235,846,836
7	High Plant Capital Cost	**$50,000,000**	50	$6,000,000	2.0%	5.0%	$267,741,862	$268,247,770	$268,274,380
	Real Interest Rate Difference								
8	4% Real Interest Rate	$14,423,790	50	$6,000,000	**2.0%**	**6.0%**	$171,316,733	$171,370,566	$171,371,612
9	2% Real Interest Rate	$14,423,790	50	$6,000,000	**2.0%**	**4.0%**	$317,754,522	$322,385,295	$323,082,092
10	1% Real Interest Rate	$14,423,790	50	$6,000,000	**3.0%**	**4.0%**	$526,756,421	$570,231,596	$589,971,976
11	5% Real Interest Rate	$14,423,790	50	$6,000,000	**2.0%**	**7.0%**	$140,901,604	$140,908,023	$140,908,070
	Variations in Operating Costs								
12	Intermediate Operating Cost	$14,423,790	50	**$4,000,000**	2.0%	5.0%	$153,981,067	$154,304,949	$154,321,985
13	Intermediate Operating Cost	$14,423,790	50	**$5,000,000**	2.0%	5.0%	$187,801,740	$188,204,040	$188,225,200

4.2.14.2.2 Development of the Existing LTVSMC Tailings Basin

In a discussion of the 2W tailings cell, it is noted:

> *"…in Cell 2W, rapid construction in later years of development resulted in oversteepened dams on all sides of Cell 2W. Some seepage has occurred from the dam in this and other areas along the dam embankments. Other points along the dam embankments have been subject to erosion of the perimeter dam due to the leaking and failure of LTVSMC discharge pipes, and from the natural geomorphological processes such as melting snow, precipitation runoff, soil creep, wind erosion and others." (SDEIS, p 4-371)*

There is no indication that this issue will be addressed as a part of reclamation.

Recommendation: A plan to address the oversteepening of the embankments of Cell 2W should be included in the closure/reclamation plan.

5.2.7 Air Quality

5.2.7.1.3 Proposed Action Emissions

Dust Suppression

The SDEIS identifies that "water and other dust suppressants" will or may be used - but does not specify when and where (*see e.g*. section 5.2.7.1.3 Proposed Action Emissions, SDEIS p. 5-401). Water alone can help suppress dust but for roads and similar flat wind-erosive surfaces it is recommended that calcium or magnesium chloride or similar suppression enhancer be required in the water used for dust suppression to extend the durability of the treatments. The Air Quality Management Plan similarly calls for "water and/or MPCA approved dust suppressants"

Recommendation: Unless there is a compelling environmental or safety reason, water (or snow) should not be used alone as a dust suppressant and instead magnesium chloride or similar suppression enhancer should be added to water to enhance dust suppression.

Recommendation: The mine should investigate and, where appropriate, mitigate reports of airborne dust that come from any entity, whether a mine employee or observer off the mine site.

In SDEIS Section 5.2.7.1.3, Proposed Action Emissions, PolyMet voluntarily agrees to:

> *"… accept emission limits below the major source threshold (stationary sources less than 250 tpy for criteria pollutants and 100,000 tpy for GHGs) so as to be classified as a synthetic minor PSD source and therefore not be subject to PSD requirements including modeling attainment with PSD increments for permitting purposes*." (SDEIS, p 5-402)

This lower threshold (higher permitting stringency) is described as based on the assertion that:

> *"NorthMet Project Proposed Action does not have projected controlled emissions above major PSD thresholds **on an annual basis**." (SDEIS, p 5-402 [**emphasis added**]).*

Annual averages are convenient for permitting purposes but it will be critical as part of the air quality permitting process and during mine monitoring to ensure that short-term air quality criteria are not exceeded.

Recommendation: The mine's air quality permit and air monitoring protocol should ensure that short term and peak air quality conditions are measured, representative, and reported to the public and regulatory agencies to ensure that short-term peaks, whether or not they violate standards, are considered as part of air quality degradation.

5.2.7.2 NorthMet Project Proposed Action

5.2.7.2.1 NAAQS and Prevention of Significant Deterioration Increment Impact Analysis

Mine Site Receptors Analysis

The waste rock pile is anticipated to present significant contaminating potential. The plan anticipates that dormant portions of the pile will be covered when not active (while being loaded before being covered and later while being transported to the pit for final disposal). Because the waste rock is highly reactive it stands to reason that its dust, notably fugitive dust, may be highly reactive. Therefore, the Mine Plan should review and analyze the reactivity of fugitive dust from waste rock sources (piles, handling, transport, etc.). If this review and analysis indicates the potential for degraded air quality or the potential for fugitive dust to contaminate lands or waters it travels to, then the regulatory agencies should take appropriate steps to halt such fugitive dust.

Recommendation: Waste Rock fugitive dust should be reviewed and analyzed to ensure that it does not pose a special hazard or threat to human health or the environment (beyond the typical opacity or other visual considerations).

5.2.7.4.3 Voluntary Mitigation Measures

The SDEIS discusses a voluntary anti-idle program (section 5.2.7.4.3, Voluntary Mitigation Measures, p. 5-434). The mine should commit to this program, even with the caveats described in that section, to help ensure that it is doing all that it can to reduce air pollution. Anti-idling will most impact sulfur- and nitrogen emissions, which come from trucks, locomotives, and mining equipment. The company should further ensure that its truck deliveries comply with the program. It is good for the SDEIS to describe PolyMet's consideration, but it is hollow to describe unless the company commits to it.

Recommendation: The SDEIS should detail PolyMet's actual commitment to an anti-idling program that maximizes reduction of vehicle-based NOx and SO2 emissions while accommodating mine-based requirements for productivity and safety.

5.2.7.2.5 Mercury Deposition Impact Analysis

Autoclaves, which are the main processing node of the hydrometallurgical processing, can be a significant source of airborne mercury. Autoclaves typically operate at temperatures in the hundreds of degrees. They use injected pure oxygen to actually 'burn' pyrite and other sulfides to produce this heat. As a result, any mercury that is in the ore is typically volatilized, and must be removed from the off-gas stream from the autoclave. PolyMet is proposing to use first stage and second stage scrubbers to remove mercury volatilized in the off-gas from the autoclaves (Barr 2012r, p 26). Third stage controls are available, but have been judged to be uneconomical by Barr/PolyMet (Barr 2012r, p 27). There is only a brief discussion of the mercury air emission control processes that will be employed at NorthMet (SDEIS, p 5-431). It is mentioned several places in the SDEIS that annual emissions from the autoclaves will be 4.1 pounds of mercury, after first and second stage controls, but there is no discussion in either the SDEIS or the primary technical support document (Barr 2012r) of the total amount of mercury in the process stream (bulk tailings, hydrometallurgical tailings, or autoclave scrubber waste).

It is important to know how much mercury is in the system, where it is going, and how stable it will be in its final form. This information should be disclosed in the SDEIS, and not buried in technical documents.

The scrubber mercury waste will be placed in the hydrometallurgical tailings facility, but this is not mentioned in SDEIS, only in the reference documents. This is significant process step and should be discussed in the SDEIS.

Recommendation: There should be a discussion in the SDEIS of the total amount of mercury in the process stream (bulk tailings, hydrometallurgical tailings, or autoclave scrubber waste), and where the autoclave scrubber waste will be disposed.

One of the major concerns with these capture systems is ensuring that they are performing as planned. Under USEPA standards mercury emissions from autoclaves are monitored only once a year. The performance of some high temperature components in the processing stream can be based only on manufacturer's specifications with no monitoring. Once a year measurements will not provide enough data to ensure statistically reliable measurements of the efficiency of the mercury capture systems.

GAO reported that equipment to monitor mercury air emissions on a continuous basis was installed on 16 boilers at coal-fired power plants and used for monitoring operations and compliance reporting. On average these systems operated about 90 percent of the time.[1] The technology for measuring mercury economically and often enough to provide meaningful data is available.[2]

Recommendation: In order to ensure that the mercury capture systems on the autoclaves are functioning as designed, a monitoring scheme should be required that will provide statistically reliable data on the autoclave mercury emissions.

5.2.8 Noise and Vibration

As part of the SDEIS' noise impact methodology the regulatory agencies should require noise contour maps, notably regarding the Boundary Waters recreational areas, so that the public can fully assess the noise impacts that could occur offsite, particularly to federal and state recreational lands.

Recommendation: The SDEIS should include noise contour maps depicting offsite noise contours from mine activities (including transportation to and within the site) and cumulative noise impacts.

The SDEIS fails to discuss specific types of noise, which could impact people off-site differently than just measuring impacts by straight decibel levels. The SDEIS focuses primarily on volume but noise impacts are also impacted by four primary factors:

1. Tonality. Tonal noises which have a narrow sound frequency, such as the whine of an electric motor or an electric saw. A tonal audibility or annoyance factor may be calculated by comparing the tone level to the level of the surrounding spectral components.[3]

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[1] Mercury Control Technologies at Coal-Fired Power Plants Have Achieved Substantial Emissions Reductions, Report to the Chairman, Subcommittee on Clean Air and Nuclear Safety, Committee on Environment and Public Works, U.S. Senate, United States Government Accountability Office, October 2009, GAO-10-47 Clean Air Act, p. 26.
[2] Mercury and Modern Gold Mining in Nevada, Greg Jones, Glenn Miller, Dept. of Natural Resources and Environmental Sciences, Final Report to U.S. Environmental Protection Agency Region IX, October 11, 2005, p. 26.
[3] *See* Breul and Kjaer. 2000. *Environmental Noise Handbook,* p. 25. (http://www.macavsat.org/pdf_files/misc_reports/bk.pdf)

2. Low Frequencies. Low frequency noise emits from machinery, all forms of transport and turbulence, turbines, exhaust gas, compressors, etc., and can travel greater distance than audible noises. Low frequency noise may cause notable human disturbances even when the decibel level (the sound pressure level) is below 30 dBA.[4]
3. Fluctuating Noise. Fluctuating or intermittent sounds are inconsistent in time and/or duration. Examples include generators or machinery operated in cycles, etc. Fluctuating noise has been shown to increase the annoying aspects/annoyance factor of the noise (notably when compared to average sound levels).[5]
4. Impulsive sounds. Impulsive sounds are brief, abrupt noises that can cause startling effects that cause greater annoyance levels than may be expected from just measuring the sound level.[6] An impulsive sound at mine sites would be blasting noises, but could also include metal on metal, rock on rock, or rock on metal noise (such as dumping rock from a loader onto a transport truck, railcar, or rock pile).

The SDEIS should further consider types of noise, rather than simply volume level (decibels) or the conclusion that applicable standards will not be violated.

Recommendation:* *In addition to decibel (volume) impacts, the SDEIS should consider the potential for impacts from tonality, low frequency noises, fluctuating noise, and impulsive sounds. The focus should consider the Boundary Waters recreation areas but could also consider other land uses around the mine site.

5.2.14 Geotechnical Stability

5.2.14.2.2 Tailings Basin

In discussing the construction of the new tailings facility, it is noted:

> *"The Tailings Basin would be constructed using the upstream method, whereby NorthMet dam embankments would be constructed using preferentially borrowed LTVSMC tailings on top of the existing LTVSMC tailings embankment and on the spigotted tailings adjacent to the perimeter embankment." (SDEIS, p 5-561)*

Upstream construction poses the highest risk for seismic and static failure of tailings dams. Most tailings dam failures have been associated with upstream dam construction.

A significant concern with upstream tailings dam construction is its susceptibility to failure during earthquakes. If the tailings upon which the dam is constructed are saturated with water, the tailings do not form a stable foundation for the dam under seismic loading.

Tailings are placed in a saturated state. Tailings materials are relatively uniform in their size and shape, and typically have very low permeability, a fact often cited by mining engineers to argue that liners are not needed for tailings facilities. As a result, it will be difficult to consistently drain the water from all the tailings under the proposed dam expansion.

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[4] Berglund, B., Lindvall, T. and Schwela, D. 1999. *Guidelines for Community Noise*. Page 46. World Health Organization.
[5] Leventhall, G. 2003. A Review of Published Research on Low Frequency Noise and its Effects. Prepared for Department for Environmental Foods and Rural Affairs. (United Kingdom). p. 11.
[6] Breul and Kjaer. 2000, p. 14.

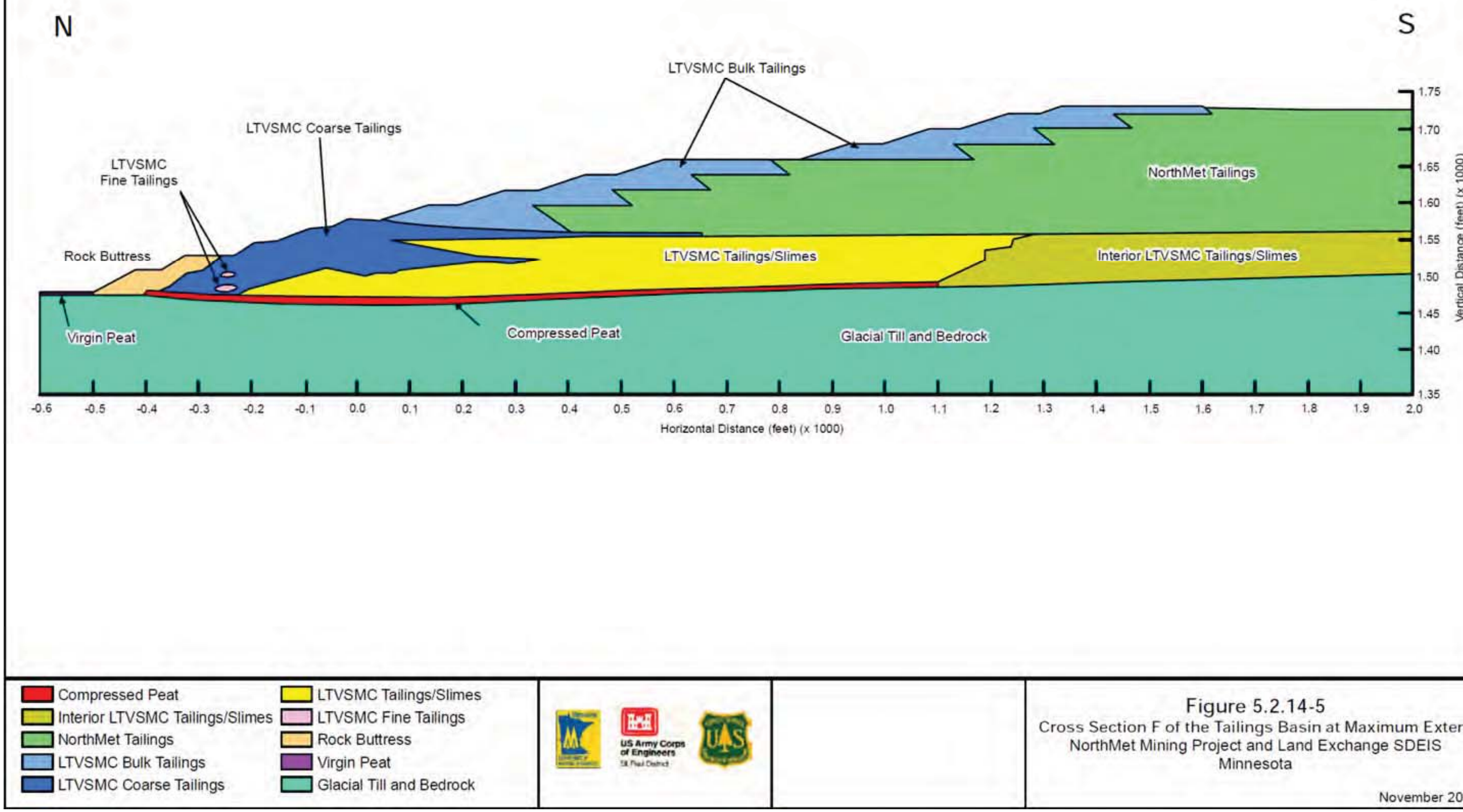

Figure 5.2.14-5
Cross Section F of the Tailings Basin at Maximum Extent
NorthMet Mining Project and Land Exchange SDEIS
Minnesota

November 2013

Continuing to use upstream-type dam construction methods to increase the capacity of the tailings at the NorthMet tailings facility is the least expensive dam construction approach, but poses the most risk to long term seismic stability.

PolyMet picked a "critical" cross section, noting:

> *"Geotechnical conditions along the length of existing LTVSMC Tailings Basin dams have varying layers of coarse, fine, and slime tailings. Cross Section F, which intersects the northern dam of Cell 2E, as shown in Figure 5.2.14-4, was selected to represent the critical cross section for stability analysis purposes as it is the maximum section and some layers of the weaker fine and slime tailings extend close to the dam embankment, and the dam embankment is underlain by peat." (SDEIS, p 5-565)*

(see Figure 5.2.14-5: Cross Section F of the Tailings Basin at Maximum Extent)

The dark blue segment in Figure 5.2.14-5 is the existing tailings dam, and the lighter blue would be the new upstream raises. This figure illustrates very well the importance of the stability of the tailings as a base for the upstream dam.

It is then noted in the SDEIS:

> *"The results reported in Geotechnical Data Package Volume 1 Version 4 indicate that the proposed design of the Tailings Basin would meet all respective Factors of Safety as required (PolyMet 2013n)." (SDEIS, p 5-565)*

There are several problems with the otherwise good work in Geotechnical Data Package Volume 1 Version 4:

1) The Probabilistic Seismic Hazard Analysis (PSHA) considers the 2,475-year return seismic event to be the largest earthquake the dam will experience. The PSHA should have used the Maximum Credible Earthquake (MCE) as the design earthquake.

   The design earthquake should represent the ground motions or fault movements from the most severe earthquake considered at the site. Since a tailings dam must stand in perpetuity, the design earthquake should be equivalent to the Maximum Credible Earthquake.

   The estimated largest earthquake that could occur at any given location is called the Maximum Credible Earthquake. The MCE is defined as the greatest earthquake that reasonably could be generated by a specific seismic source, based on seismological and geologic evidence and interpretations. The Maximum Credible Earthquake is most often associated with a recurrence interval of 10,000 years.[7]

   If the MCE/10,000-year event is used for the analysis of the 2,475-year event, the horizontal acceleration (horizontal g-force the dam is subject to) will increase significantly.

2) The mean distance to the nearfield earthquake is 100 miles. Probabilistic determination for the size of the largest earthquake is appropriate, but the assumption of 100 miles for nearfield is going to make the horizontal acceleration used to design the dam lower than what it should be.

   The further away the tailings dam is from the location of the earthquake, the less energy the tailings dam will need to withstand in order to maintain its structural integrity. The closer the location of the

[7] Large Dams the First Structures Designed Systematically Against Earthquakes, Martin Wieland, ICOLD, The 14th World Conference on Earthquake Engineering, Beijing, China, October 12-17, 2008

earthquake to the tailings dam, the higher the cost of building the dam, because the closer the earthquake the more energy the dam will have to withstand.

Seismologists know that there are many active faults that have not been mapped or have been mapped inaccurately, that some faults believed to be inactive may actually be active, and that there are many inactive faults that may become active again. Because of these considerations, probabilistic methods are the more conservative way to determine the magnitude of a Maximum Credible Earthquake for dam analysis.

For tailings dams the most conservative choice for the location of the Maximum Credible Earthquake would be what is sometimes referred to as a 'floating earthquake' on an undiscovered fault that passes very near the site of the dam. This is a way of recognizing that we do not know the present, future, and even the past locations of significant faulting, and associated earthquakes.[8] The conservative choice for a Maximum Design Earthquake would be a Maximum Credible Earthquake that ruptures the ground surface on which the dam is built.

3) The evaluation for dam stability does not employ dynamic modeling.

Polymet did not perform dynamic modeling for the tailings dams.

*"Results of the seismic liquefaction screening evaluation (Section 6.5.3.3) indicate that seismic triggering will not occur. As the **seismic design event (2,475-year return period)** would not trigger liquefaction in any FTB materials, per the Work Plan (Attachment A), **no additional seismic triggering analyses were necessary**." (Geotechnical Data Package Volume 1 – Flotation Tailings Basin Version 4, PolyMet Mining, April 12, 2013, p 92, **emphasis added**)*

PolyMet performed what might be termed a pseudostatic analysis. Today, most US regulatory agencies will not accept pseudostatic methods for seismic design of new dam projects. Dynamic analysis of seismic loading for most new dams is required if the maximum credible earthquake produces a peak ground acceleration of more than 0.1 g at the site.[9]

A pseudostatic analysis (sometimes called seismic coefficient analysis) should only be considered as an index of the seismic resistance available in a structure not subject to build-up of pore pressure from shaking. It is not possible to predict failure by pseudostatic analysis, and other types of analysis are generally required to provide a more reliable basis for evaluating field performance.[10]

An example of a government agency which happens to focus on dam safety and that will not accept pseudostatic analysis is the Federal Energy Management Agency (FEMA). FEMA practice previously allowed the use of the pseudostatic method of analysis in areas of low or negligible seismicity. FEMA does not recommend the pseudostatic analysis to judge the seismic stability of embankment dams.[11]

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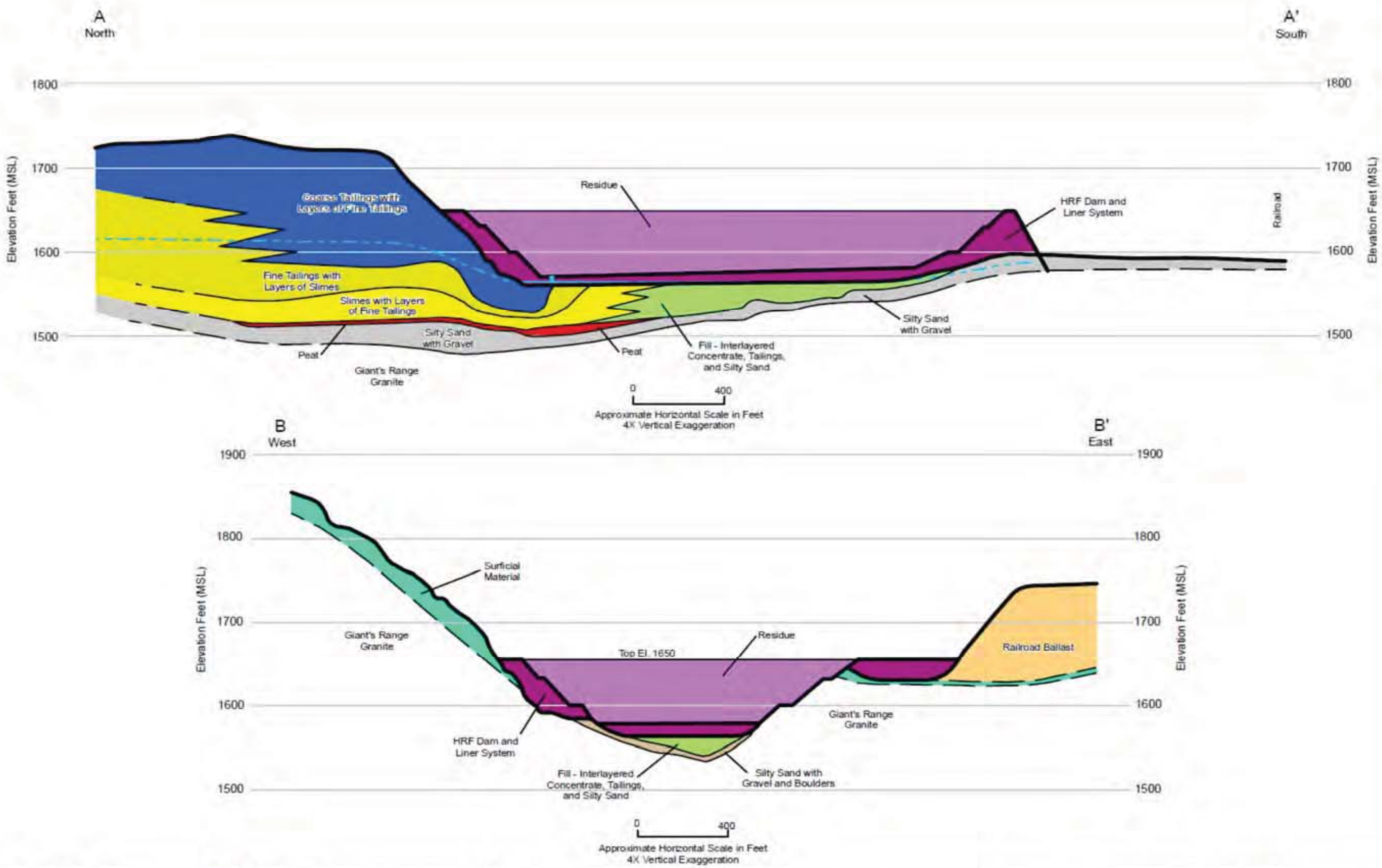
[8] Safety of Dams, Flood and Earthquake Criteria, Committee on Safety Criteria for Dams, Water Science and Technology Board, Commission on Engineering and Technical Systems, National Research Council, National Academies Press, Washington, D. C. 1985

[9] http://www.meadhunt.com/documents/newsletters/persp_water3.pdf, downloaded on 14Jan10

[10] Federal Guidelines for Dam Safety Earthquake Analyses and Design of Dams, Federal Energy Management Agency, May 2005, p. 35

[11] Federal Guidelines for Dam Safety Earthquake Analyses and Design of Dams, Federal Energy Management Agency, May 2005, p. 38



Coarse Tailings with Layers of Fine Tailings
Fill - Interlayered Concentrate, Tailings, and Silty Sand
Fine Tailings with Layers of Slimes
Slimes with Layers of Fine Tailings
Peat
Railroad Ballast
Residue
Silty Sand with Gravel
Silty Sand with Gravel and Boulders
HRF Dam and Liner System
Surficial Material

US Army Corps of Engineers
St. Paul District

Figure 5.2.14-6
Cross Sections A and B of the Hydrometallurgical Residue Facility at Year 20
NorthMet Mining Project and Land Exchange SDEIS
Minnesota

November 2013

Dynamic analysis is the most rigorous method of evaluating dam survivability under seismic loading. Typically a dynamic analysis will use finite element or finite difference programs such as TARA (Finn et al 1986), FLAC (Itasca Group 2002), or PLAXIS (PlaxisBV 2002) in which dynamic response, pore-pressure development, and deformations can be fully coupled.[12]

These tailings dams must contain this material in perpetuity. If not, the cost of collecting spillage due to an earthquake-related failure, and rebuilding the containment structure, would be many millions of today's dollars. This is not a risk, or cost, that should be passed on to future generations. If these containment structures are going to be built, the assumptions used to check the design should be conservative, and the models the best available.

5.2.14 Geotechnical Stability

5.2.14.2.3 Hydrometallurgical Residue Facility

Global Slope Stability

As described above with the tailings basin geotechnical design, similarly there was no dynamic modeling for the hydrometallurgical facility.

> *"Liquefaction analysis was not applicable and not performed because the material proposed in the constructed dams would be well-compacted and the Hydrometallurgical Residue Facility liner system would limit leakage through the dams." (SDEIS, p 5-575)*

Even though the construction of the hydrometallurgical facility dam is downstream, the safest type of dam construction, the material that this facility holds is potentially very dangerous to both human health and the environment – if it were to be released. As a result, the geotechnical analysis of the dam should be conservative, and as with the bulk tailings dam, dynamic modeling should be performed.

In addition, it is proposed that the hydrometallurgical facility be placed on a residual layer of taconite tailings.

According to SDEIS Figure 5.2. 14-6 above a large portion of the hydrometallurgical facility (and liner system) will lie on: "Coarse Tailings with Layers of Fine Tailings, Fine Tailings with Layers of Slimes; Slimes with Layers of Fine Tailings; Fill – Interlayered Concentrate, Tailings, and Silty Sand; Silty Sand with Gravel; and, Peat."

Even if this material will be "well-compacted" it would be safer to remove the original peat and silty sand/gravel, and the taconite tailings and slimes, and replacing this material with compacted fill, so that the hydrometallurgical facility is built on a well prepared and verifiably stable base. This is the conservative approach.

Recommendation: The underlying original ground and the taconite waste should be removed from underneath the hydrometallurgical tailings facility, an engineered stable base installed, and dynamic modeling performed on the hydrometallurgical dam.

[12] See Federal Guidelines for Dam Safety Earthquake Analyses and Design of Dams, Federal Energy Management Agency, May 2005, p. 32

APPENDIX B: Underground Mining Alternative Assessment for the NorthMet Mining Project and Land Exchange Environmental Impact Statement

2.0 Screening of the Underground Mining Alternative

The underground option was judged to be feasible technically, but not economically, based on the information in Table 2 Economic Assessment of a Sample of Underground Mining Scenarios Considered.

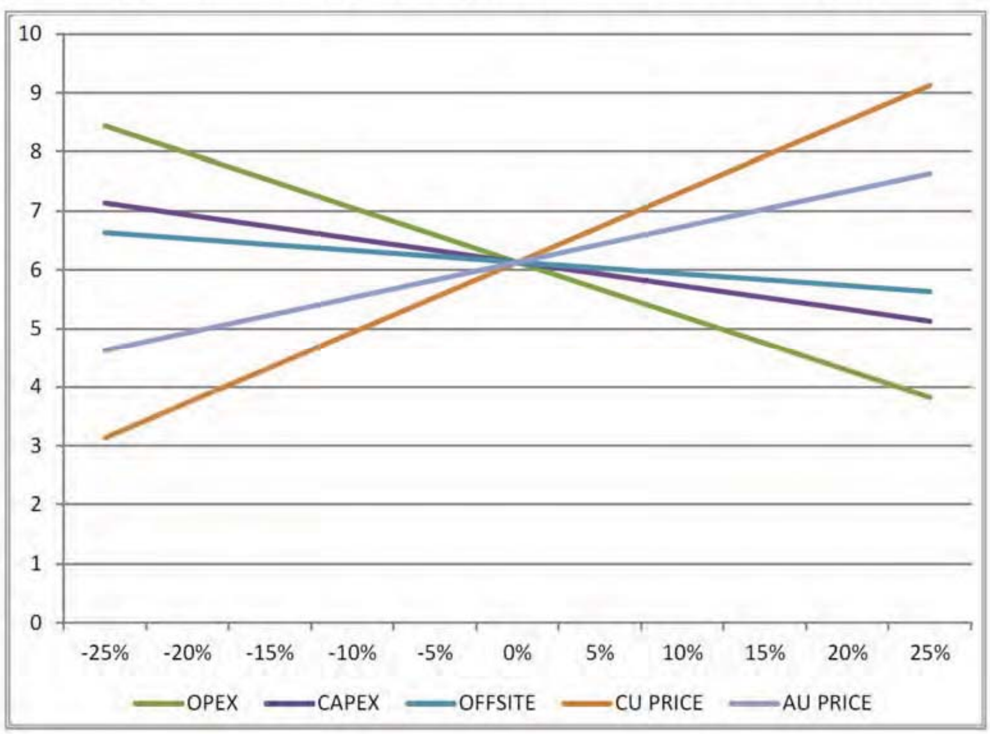
The table is fine as far as it goes, but there is no discussion of the sensitivity to change in metals prices. For example, what if the price of gold increased $500/oz. or copper increased $0.50/lb (both reasonable assumptions), how would that affect the net extracted metal value? Would this make the underground mine economic? If so, then an evaluation of an underground mine as an alternative would be warranted.

A typical technical analysis for a mine proposal, like the public securities documents and information filed by public companies in the SEDAR filing system.

The analysis in Appendix B does a partial job of assessing sensitivity, in this case the variables are limited to tonnage produced and operating cost, but it ignores variations in metal prices.

PolyMet's NI 43-101 report does contain a limited cost sensitivity analysis,[13] but a better and more typical example is Figure 18.8.5 below,[14] where the sensitivity is also displayed graphically. Underground mining might be economical in the future with an increase in metals prices, and when the processing technology proposed for the NorthMet operation has been proven enough to clearly quantify the costs.

Figure 18.8.5 Pre-Tax NPV_7 Sensitivity to Inputs – 45-Year Reference Case ($B)



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[13] Updated NI 43-101 Technical Report on the NorthMet Deposit, AGP Mining Consultants Inc., January 14, 2013
[14] Preliminary Assessment of the Pebble Project, Southwest Alaska, Ghaffari et al., Wardrop-Northern Dynasty Mines, February 17, 2011

The environmental benefits of an underground mine would be significant – far less waste rock, and the potential to backfill most or all of the waste rock. It might also be possible under this scenario to limit or avoid long term water treatment, which would be a significant economic benefit that was not analyzed in the economics of the underground mine scenario.

Recommendation: ***A metals cost sensitivity analysis should be added to the Underground Mining Alternative Assessment to verify that the underground option is not economical with higher metals prices.***

RECLAMATION PLAN
(PolyMet 2013a, NorthMet Project Reclamation Plan, Version 3. January 22, 2013)

2.1.2 Rail Transfer Hopper (RTH)

The use of the East Pit as a repository (such as for sediment from the Rail Transfer Hopper (RTH) and process water pond (including ore remaining in the RTH, the OSP, or along the railroad tracks between them (PolyMet 2013a, section 2.1.2, p. 9)) underscores the importance of characterizing the wastes being placed there and then monitoring water that may pool/collect in the East Pit and waters (surface and ground) that may be hydrologically connected to the Pit.

Recommendation: ***All wastes that could be disposed in any pit, and the pits themselves, should be characterized to ensure that there will not be regular or seasonal contamination directly or indirectly to surface or ground waters.***

2.1.5 Pipelines and Power Lines

The Reclamation Plan states: "*Underground pipelines will be abandoned in place*." (PolyMet 2013a, section 2.1.5, p. 11). In general, abandonment may not be a problem but no pipeline should be allowed to remain if it could cause water to flow between two places where it should not, such as diverting water from a creek or connecting surface to ground water or connecting hydrologic units that are isolated from each other.

Recommendation: ***All pipelines proposed for abandonment in place should be evaluated to ensure that their residual contents will not cause contamination and that their presence cannot facilitate connectivity between otherwise isolated bodies of surface and/or ground water, whether constant or intermittent.***

2.1.6 Tanks

The discussion about aboveground tank disposal appears reasonable. That section states: "*Insulation and coverings will be removed and disposed appropriately*." (PolyMet 2013a, section 2.1.6, page 12) This statement implies compliance with applicable asbestos or other hazardous materials laws - but it would be more appropriate to expressly state that hazardous, toxic, nuclear and related materials will be disposed of in full compliance with applicable federal and state laws and regulations.

This comment applies to the entire Reclamation Plan, including this section and other sections that directly or indirectly include materials disposal (e.g. Sections 2.3 Special Material Disposal, 2.4 Product Disposal, etc.).

Recommendation: ***The Reclamation Plan should expressly state the applicable federal and state laws, regulations, and rules that apply and that will be complied with in the testing, handling, storage, and disposal of all hazardous, toxic, nuclear, and related materials and wastes.***

Recommendation: ***The Reclamation Plan should require a tracking system that demonstrates that materials are actually disposed-of according to the Reclamation Plan and applicable laws. The tracking system should include not only affirmation but documentation that certifies proper materials disposal.***

3.0 Mine Site Reclamation and Long-Term Closure

3.1 Mine Pits

The Reclamation Plan describes that pit reclamation details are contained in PolyMet 2012t: 2012 Mine Plan (NorthMet Project Mine Plan (v2). December 2012 ("Mine Plan").

Section 6.2.1 of the Mine Plan states that:

> *"...the water pipes between the WWTF and the East Pit could be used during reclamation to convey treated water to the East Pit if insufficient water was otherwise available to maintain water levels or to convey East Pit water to the WWTF for treatment."*
>
> *"... the water pipes between the West Pit and the WWTF will be used in reclamation to convey treated water from the WWTF to the West Pit if insufficient water was otherwise available to maintain water levels and to convey West Pit water to the WWTF for treatment." (PolyMet 2012t, section 6.2.1, p. 24).*

The Reclamation Plan should more clearly predict the water balance desired in the East and West Pits and the conditions that could impair it from being achieved. The Mine Plan later describes that both pits will achieve overflow status at some point but it is unclear what is intended and needed before full-pool/overflow is reached. (*See also* PolyMet 2012t, section 6.2.6 Water Management During Reclamation (PolyMet 2012t, p. 26)).

If insufficient - or too much - water in the pits could be a problem then the Reclamation Plan should better explain the water management goals and develop a better approach, or at least better predictability and an explanation of the conditions and goals. If a steady-state is not achievable without long term, post closure monitoring and management then the mine plan should be modified to ensure that a natural steady-state will develop with reasonably minimal initial modification and no long-term rebalancing.

Recommendation: ***The Reclamation Plan should ensure that water management in both the East and West pits do not require long term volumetric management until they reach their steady-state/full/overflow condition. The Reclamation Plan should ensure that specific goals for pit pools are established and achieved.***

The Mine Plan's description for final reclamation lacks the detail necessary to evaluate it as part of the Reclamation Plan. PolyMet 2012t, section 6.2, Final Reclamation, states that:

> *"The following paragraphs describe the reclamation of the mine pits once operations cease.*
>
> *As part of wetland impact mitigation, a wetland may be constructed on the backfilled East Pit as described in Section 2.2 of Reference (9), and the East Pit will be flooded." (PolyMet 2012t, p 25)*

Section 2.2 of Reference (9) is the NorthMet Project Wetland Management Plan (v3). January 2013, which is discussed in more detail below, and which does not contain the necessary detail or information to assess the mine's plan in this area.

Recommendation: The Reclamation Plan should provide sufficient detail about wetlands construction and associated decision-making processes to allow regulatory agencies and the public to reasonably understand what is likely to happen and what criteria will be employed to determine reclamation success, and what steps will be taken in the even that reclamation does not achieve established criteria.

PolyMet 2012t, Section 6.2.3, Pit Perimeter Barrier, (PolyMet 2012t, p. 25) generally summarizes the types and placement of barriers around the pits. It does not, however, establish specific goals that will be collectively achieved by the barriers. The Reclamation Plan should specify that fencing will achieve an appropriate, reasonably measured barrier to access by people, including children, and wildlife. Some of this may be achieved by the referenced approval of the St. Louis County Mine Inspector but the mine company, and the Reclamation Plan bear the responsibility of establishing the reclamation criteria that will be achieved.

Recommendation: The Reclamation Plan should establish specific fencing goals to ensure that the barriers proposed in the Mine Plan will achieve an appropriate, reasonably achievable barrier to access by people, including children, and wildlife.

PolyMet 2012t, Section 6.2.5, Mine Pit Lake Level Management (PolyMet 2012t, p. 25-26), generally describes the pits and the excavated channel that will connect the East and West pits. The descriptions do not adequately ensure that the final plan will protect wildlife (that invariably will manage to enter the site in spite of the proposed barriers) that enters either pit or the excavated channel between them is able to reasonably escape.

Recommendation: The Reclamation Plan should plan for measures to allow animals to expressly rescue themselves should they fall into the pits or excavated channel between them. This should also include predicting the impacts and express protections for both terrestrial wildlife and birds that may land on contaminated waters. If contaminated waters present a reasonable threat to animals then additional measures should be committed-to and employed (such as hazing or surface netting for birds) to ensure that the pits do not become wildlife death traps.

Erosion

The Mine Plan's Reclamation Maintenance section (6.3.1) describes that:

> *"Reclaimed mine overburden slope erosion will be corrected and re-vegetated as needed. In areas where excess erosion is a repetitive problem, channels and/or outfall structures will be designed for those specific locations." (PolyMet 2012t, p. 26).*

Neither the Reclamation Plan nor the Mine Plan describes adequate preventative reclamation methods to reduce overburden slope erosion or measures to ensure that erosion does not contaminate surface waters. The Reclamation Plan should describe detailed methods to be employed to prevent erosion. Should those methods prove inadequate then further planning and implementation should be employed. Methods to prevent erosion may include, but not be limited to, dozer basins, terraces, rock and rip-rap placement, etc. What matters is to ensure that prevention takes primacy over responses to failure.

Where erosion does occur there should be a clear commitment to not only correct the cause/problem but to employ further preventative measures.

The Reclamation Plan should establish specific goals for erosion - the failure of which will trigger specified responses. Given that erosion may occur many years after successful revegetation (such as after a drought year stresses erosion-protecting plants or a particularly wet year or piping causes new or increased erosion) it is important for the Reclamation Plan to develop these goals and commitments.

Recommendation: ***The Reclamation Plan should establish clear, measurable erosion goals including success criteria(such as less than x-feet of rilling per y-area and no erosion wider or deeper than z-inches) and responses to failure to meet those reclamation criteria, including but not limited to treatment protocols; long-term protection from post-reclamation disturbances; timeframes over which success will be measured and how criteria failure or re-treatment activities will re-start timeframes, etc.***

Stockpile reclamation is included in NorthMet Project Rock and Overburden Management Plan Version 5 (December 28, 2012), section 7.0 Reclamation and Long-Term Closure.

Wetlands

PolyMet 2012s, the Rock and Overburden Management Plan (v5, December 2012) section 7.1.2.2, Reclamation of Footprint (mitigation wetlands) states that:

> *"Once the waste rock and overburden are completely relocated from the temporary stockpiles to the East Pit, the stockpile bases, which include the overliner drainage system, liner system, underdrain system, if required, and portions of the foundation, will be disassembled for reclamation of the footprint. Generally, pipes, liners, and pumps, will be removed and the footprint of the stockpile will be reclaimed.*
>
> *For the Category 2/3 Waste Rock Stockpile, wetlands will be restored or cultivated where the hydrology and soil conditions exist to support their development. Approximately 60 acres of wetlands have been identified within the Category 2/3 Waste Rock Stockpile footprint.*
>
> *Wetland mitigation is expected to occur in areas that were wetlands prior to the start of stockpile development, as well as in additional areas where the stockpile load has depressed the soils enough that wetland hydrology can be established from prior upland areas. The plan for development of wetlands within these areas will likely include grading, the addition of soils as needed, and wetland plant propagation. The ultimate goal in restoration and development of wetlands within the former stockpile footprint will be to restore the original flow patterns that existed prior to mining and to establish an area of wetlands equal to or greater than existed prior to mining. See Section 2.2 of Reference (11) for more information on wetland mitigation at the Mine Site. For portions of the footprint that cannot be converted to wetlands, the surface will be scarified or soil will be placed over the reclaimed foundation, if needed, followed by seeding." (PolyMet 2012s, p. 43).*

The Reclamation Plan should make substantive analysis and establish specific goals regarding waste rock footprint reclamation. As written, the reclamation results could yield uncertain results with uncertain responses to failures if those results fail to achieve what should be pre-established goals. Given the importance of wetlands to the ecosystem and ecosystem function, their replacement should be paramount in the Reclamation Plan. Off-site purchase/protected wetlands is an excellent commitment - but does not replace the need for functioning wetlands in the local (on-site, post reclamation) ecosystem.

The Reclamation Plan should predict the likely compaction caused by the waste rock and overburden piles and thereby calculate the necessary treatments needed to return them to productivity. The reference to scarification may not reach the depth of compaction without additional measures. The Reclamation Plan therefore should model/predict compaction rates and depths and commit to ensuring that reclamation activities reverse this compaction.

The Reclamation Plan should further commit to adequate testing and removal of contaminated soils to a 'clean' depth (depth at which contamination no longer exists) below the waste rock and overburden piles. This will be particularly important for the Class 1 waste rock but should apply to all materials that contain contaminants.

Finally the Reclamation Plan should establish wetland function goals that will be achieved. The commitment to try to re-establish the buried wetlands (under the waste rock and overburden piles) is good - but needs to be substantive and measurable, not simply a commitment to try.

The Rock and Overburden Management Plan refers to the Wetlands Management Plan v. 3 (*Section 2.2 of Reference (11))*. Version 3 was dated January 2013; Version 4 was dated March 2013 - therefore the latter is referenced here because it is more recent.

PolyMet 2013h, Wetlands Management Plan v. 4[15] states:

> *2.2.1 General Mine Area Wetlands.*
>
> *"Upon reclamation, approximately 72 acres of wetlands may be created at the temporary mine stockpile areas after removal of the Category 2/3 Waste Rock Stockpile and the OSLA as described in Section 7 of Reference (5) (Large Figure 6). <u>Because it may not be feasible to construct wetlands on the entire footprint of these temporary areas, it was assumed that only the area equivalent to the directly impacted wetlands within the footprints will be viable for wetland mitigation</u> (Reference (1)).* ***Design of wetland mitigation areas will be further evaluated in the detailed reclamation design in Section 7 of Reference (5). The design will include the preservation of upland buffer around the perimeter of the wetland mitigation areas."***
>
> *(PolyMet 2013h., pages 9-10 [**emphasis added**]).*

The basis is unclear for the Plan to "assume" that any viable wetlands construction will be feasible, let alone the entire or partial footprint of the stockpile sites. The Reclamation Plan (and ACOE 404 permit analysis) should not count on what is currently a very limited and imprecise technology (wetlands construction) as an offset for 20 years of loss (during mining) and an assumed success of even a wetland footprint equivalent to what the mine will destroy. After 20 years of mining impacts he post-mining site will effectively be a "new" site. The constructed wetlands are not being **re**constructed but rather are being constructed from scratch in a new hydrologic regimen (particularly with pit impacts to ground water quality, quantity, and flows).

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[15] Version 3 is largely unchanged, except for updating values:

> *"Upon reclamation, approximately 71 acres of wetlands may be created at the temporary mine stockpile areas after removal of the Category 2/3 Waste Rock Stockpile and the OSLA as described in Section 7 of Reference (5) (Large Figure 6). Because it may not be feasible to construct wetlands on the entire footprint of these temporary areas, it was assumed that only the area equivalent to the directly impacted wetlands within the footprints will be viable for wetland mitigation (Reference (1)). Design of wetland mitigation areas will be further evaluated in the detailed reclamation design in Section 7 of Reference (5). The design will include the preservation of upland buffer around the perimeter of the wetland mitigation areas." (Wetlands Management Plan v. 3, p. 6).*

Recommendation:* *Constructed wetlands should be not be presumed successful for any footprint until they are constructed and successfully functioning, at the desired type and level, for at least 5-10 years. The Reclamation Plan should establish wetland size/volume and function goals that will be achieved. The commitment to try to re-establish buried wetlands should to be substantive and measurable, not simply a 'commitment to try.'

3.0 Wetland Mitigation Outcomes

"This section documents the implementation of the Wetland Mitigation Plan. Wetland restoration construction progress will be tracked along with compliance with permit conditions. On- and off-site wetland mitigation monitoring and as-built reports will be summarized along with monitoring reports to document indirect wetland impacts in Section 4.0. Actual wetland impacts and compensatory mitigation will be tracked in this section as the Project progresses."

There is a "PLACEHOLDER" in PolyMet 2013h for this section on Mitigation Outcomes, so it is not known what the mitigation outcomes will be.

Sections 4 (Monitoring) and 5 (Reporting) describe more about the wetlands but still do not establish the on-site goals and triggers necessary to ensure that reasonably predict on-site actions and successes - or the conditions necessary to determine that on-site wetlands cannot be restored, meaning that the sites will be simply revegetated.

The Wetlands Management Plan v.4 refers to a detailed wetland reclamation plan in "Section 7 of Reference (5)." This is:

5. —. NorthMet Project Rock and Overburden Management Plan (v5). December 2012.

Section 7 of the Rock and Overburden Management Plan is:

7.0 Reclamation and Long-Term Closure

It appears that taken together, the Reclamation Plan refers to the Rock and Overburden Management Plan, which refers to the Wetlands Management Plan, which refers back to the Rock and Overburden Management Plan. Collectively they seem to lack substantive detail and commitment necessary for the regulatory agencies and public to ensure that the Reclamation Plan will adequately reclaim the waste rock and overburden footprints, most importantly regarding wetlands.

The construction methods and measurements for wetlands success should be described in detail to ensure that the mine cannot simply spend years ostensibly constructing wetlands only to fail and "decide" to place cover material and revegetate - thus sacrificing onsite wetlands goals and functions (notably without clear criteria, steps to be taken, and decision process.

Recommendation:* *The Reclamation Plan should specify in detail the wetlands reclamation and restoration (and wetlands 'replacement') activities that will occur. It should further establish specific goals and standards that will be met, and establish specific responses to goals that are not met. Finally, it should establish the criteria to be used to determine that it is not possible or practicable to restore on-site wetlands so that the regulatory agencies and public can know in advance just what will happen and how decisions will be made.

Section 7.1.2.2 of the Rock and Overburden Management Plan, Reclamation of Footprint (mitigation wetlands), states that:

> *"Once the waste rock and overburden are completely relocated from the temporary stockpiles to the East Pit, the stockpile bases, which include the overliner drainage system, liner system, underdrain system, if required, and portions of the foundation, will be disassembled for reclamation of the footprint. Generally, pipes, liners, and pumps, will be removed and the footprint of the stockpile will be reclaimed." (PolyMet 2012s, page 43).*

The Reclamation Plan should specify that the entire footprint of waste rock and overburden piles is removed and the entire site is reclaimed. As written, the above language commits only to generalities that collectively do not guarantee that the rock pile foundations and contamination will be fully removed.

Recommendation: The Reclamation Plan should commit to removing all waste rock and overburden pile facilities and all underlying materials that are contaminated or contain contaminated materials.

Monitoring

Rock and Overburden Management Plan section 7.3, Long-Term Closure, states that:

> *"After the reclamation process is complete, monitoring and maintenance of reclaimed areas will be done, as needed, in the spring and fall and as required by the PTM. If any of the sites have been damaged by erosion or suffered or experienced plant failure and need additional work, a plan will be created and implemented to repair the damage. This responsibility will continue until the release or partial release of PolyMet from the PTM responsibility. Of the areas at the Mine Site discussed in Section 7.1 and 7.2, the Category 1 Waste Rock Stockpile cover is the area that may require further maintenance in the long-term closure period.*
>
> *However, monitoring of reclaimed surfaces will continue until the partial release or full release of these areas from the PTM responsibilities is granted. Long-term closure monitoring* of *reclamation wetlands is discussed in Section 4.2 of Reference (11)." (PolyMet 2012s, page 44-5).*

The Reclamation Plan should contain specific commitments to monitoring (as reported in the PTM) but further identify specific goals that must be met before monitoring is reduced (or bond is released), the criteria used to measure those goals, and specific responses to failures to meet those goals. It is insufficient for the Reclamation Plan (or its referenced sections) to require only generalized commitments and nebulous responses to the failure to achieve those reclamation commitments.

Recommendation: All reclamation plans should be subject to full public and regulatory review prior to permitting and establish clear success criteria and responses to failure to meet those reclamation criteria. This should include, but not be limited to, plan-specific goals, objective criteria for each goal, timeframes over which success will be measured and how criteria failure or re-treatment activities should re-start timeframes, etc.

Recommendation: The reclamation plans should be reviewed and updated on a regular schedule, such as every 5 years, which will allow regulatory agencies and the public to monitor and predict reclamation success and issues and also allow for appropriate bond recalculation.

For detailed information about water management system reclamation, the Reclamation Plan references PolyMet 2013e: the NorthMet Project Water Management Plan – Mine, Version 2 (January 9, 2013). The Water Management Plan section 7.1, Incremental Reclamation, provides that:

> *"Once reclamation in these areas is complete, the haul roads to these areas will also be scarified and seeded to allow continued access by small vehicles only for long-term monitoring." (Water Management Plan, page 47)*

It seems that this sentence mixes two different issues. One is revegetation and one is access. Revegetation plans, goals, and methods should stand on their own, details of which are discussed elsewhere in these comments. Separate from revegetation is road reclamation. It would be more clear and appropriate for the Reclamation Plan to comprehensively identify which roads will be maintained and how (e.g. revegetated but allowing small vehicle access, maintained large-access road, etc).

Recommendation: The Reclamation Plan should identify clear revegetation goals and methods, including revegetation standards, timetables, etc.

Recommendation: The Reclamation Plan should comprehensively identify all roads and clearly state how they will be reclaimed, including their short term and long term uses (or lack thereof).

Revegetation Criteria

Section 3.4 of the Reclamation Plan describes the cover material to be used for revegetation. It states:

> *"3.4 Building Areas, Roads and Parking Lots. After demolition of Mine Site buildings and parking areas, 2 feet of overburden material suitable for vegetation will be placed over the facility's former footprint. Mine roads that are deemed not necessary for access by the MDNR Commissioner will be scarified and vegetated." (PolyMet 2013a, p. 16)*

The Reclamation Plan does not describe or commit to criteria for revegetation. The following criteria are therefore recommended for ALL sites to be reclaimed pursuant to the Reclamation Plan.

Topsoil Salvage and Placement

The higher the quality of topsoil (soil growth media) then the better the likelihood of successfully establishing durable vegetative covers. It is not clear what overburden material will be employed but rather than generalized names the Reclamation Plan should establish specific requirements for soils. This includes, but should not be limited to, pH, fertility, microbial biota, ratios of: sand/silt/clay, and nutrient cycles, such as for nitrogen and organic matter. Prior to mining, and based on pre-mine conditions, the Reclamation Plan should characterize the existing soils to guide the standards for post-mine soils. Specific topsoil requirements should be established to ensure that all growth media is suitable and will maximize the potential for revegetation success.

The greater the depth/quantity of topsoil (soil growth media) then the greater the chances of revegetation success. Long-term vegetation success will depend on greater soil depths compared to short-term vegetation success. Greater soil depth may not benefit revegetation success in the 5-year period of revegetation monitoring but greater soil depths is highly likely to benefit longer-term revegetation success. It would be a waste - and potentially impair long-term revegetation success to not salvage, preserve, and re-use all topsoil resources. Further it could impair long term revegetation success to not ensure that all sites have sufficient growth media.

The benefits of high quality and high volume topsoil are different from the mine and the public. For the mining company, the extra benefit may not be realized because the company seeks the return of its bond and then it will leave the site forever. For the public - increasing revegetation success is highly valuable - and it is the public that will ultimately be responsible for the site when the company leaves. Therefore, it is important to ensure that all soils materials are salvaged and effectively used for reclamation and that where the original materials are deficient that alternative, quality topsoil is secured.

Recommendation: All soil material should be salvaged, stored, accounted for, and distributed to maximize revegetation potential.

Where multiple materials will be used, such as topsoil placed over non-contaminating rock (waste rock or otherwise) it is important to ensure that the Reclamation Plan accounts for size/fractionation differences between the different materials. Quarried or 'clean' overburden/waste rock can be very course compared to the other materials. If fine materials, such as topsoil are placed over course materials such that the material sizes are very different, the smaller materials placed on top of the larger materials can form a layer that appears stable but over time (ranging from weeks or months to many years) may form pipes (piping) or simply infiltrate (fall) into the larger material. For this reason, the Reclamation Plan should ensure that operators and inspectors are aware of the problems associated with disparate size fractions when materials are being placed. This is particularly important for topsoil, which can be particularly susceptible to infiltrating/falling into spaces below it during storm events, snowmelt, and freeze/thaw cycles.

Recommendation: The Reclamation Plan should establish general criteria and guidance to ensure that materials placement where topsoil is replaced does not allow small size materials to be placed on materials that have much larger size particles. Where this could happen, an interlayer of mid-size materials should be placed between them.

The Plan and permit should require salvaging all topsoil and subsoil from areas disturbed by mining activities - regardless of location or volume. Post-mine plant growth and establishment benefit substantially from maximizing plant growth media (soils), particularly where agriculture is a proposed post-mine land use. The more soil, the better the post-mine revegetation success, particularly in the first five years.

The best reclamation practice would be for the company to salvage existing soil materials in two lifts - the first being A and B horizons and the second lift being sub-B-horizon. During reclamation (re)placement, the lower horizons should then be placed as the first step of replacing cover material, upon which the upper (A and B horizons) would be placed. The net effect is more cover material that will better support plants and more quickly further develop soils than just the A and B horizons placed on top of sand, waste rock, liners, etc.

The topsoil salvage piles will stand unused for years. As a result the soils quality will degrade during mine operations and the soil value will be reduced from when it was salvaged compared to when it is replaced. To preserve soil integrity (including organic materials, microbes such as mycorrhizae, promote aeration, reduce weed introduction, and reduce erosion, the Reclamation Plan should identify specific steps that it commits to employ to establishing 'nurse' crops on the topsoil salvage piles. These plants should be consistent with, and not compete, with the planned postmine revegetation, especially agricultural seeding/planting.

The Reclamation Plan should analyze and the company should commit to characterizing stored topsoil resources (one or two years prior to starting reclamation) to identify basic physical and chemical characteristic. These results can then be used to modify the reclamation plan and determine what, if any, amendments are necessary and appropriate to enhance and ensure revegetation success. Criteria should include material size fractions, nutrients, pH, microbial condition (such as mycorrhizae), and organic content. Sampling should be done at the surface and deep in the piles. This will ensure that the replaced soil and subsoil materials/horizons are best able to support post-mine agricultural goals. By sampling and evaluating the materials before they are disturbed, the mine can mix-in organics and other materials/amendments that may be necessary to ensure they are fully integrated into the replaced soils (as compared to simply added as top-dressing).

Recommendation: The Plan should develop detailed topsoil salvage and storage plans to ensure that the maximum amount of materials is salvaged for reclamation. These materials should be stored to maximize soil health and reclamation efforts. To ensure that all viable growth media is salvaged, characterization of materials should include field observation and not solely rely on a 'standardized' depth measurement.

Reclamation Maintenance

Reclamation Plan section 6.2, Reclamation Maintenance, provides that:

> *"Monitoring and maintenance of all reclaimed areas, including mine slopes, Mine Site stockpiles, Area 5 stockpiles, Plant Site Building Areas, the FTB and the HRF,* ***will be inspected at least twice per year, as necessary****, or as required by Minnesota Rules , part 6132.5200. Any areas that have been damaged by erosion or that have lost vegetation will be identified and plans to make repairs or reseed will be developed and implemented.*
>
> *Inspection and repair will continue until the MDNR determines that the reclamation is stable and self-sustaining and issues a release of the permittee as outlined in Minnesota Rules, part 6132.1400 and 6132.4800. See Large Figure 2 for Plant Site Building Areas and Large Figure 16 for locations of reclaimed areas at Area 5...." (PolyMet 2013a, p. 26 [****emphasis added****])*

Necessity can be indicated by an existing problem or by hindsight. It is unclear in this context how the mine may interpret "as necessary" but it is suggested that inspection be based on a decreasing schedule based on need. In the first two years after construction, reclaimed sites should be monitored (at least) monthly to ensure that problems are detected early-on. Where no problems are evident for one year those inspections may be reduced to quarterly. Where no problems are evident after two years of quarterly inspections then the inspection frequency may be reduced to twice per year. The timing should be reasonably based on capturing problems early-on and should be approved by the state regulatory agencies. If a problem is detected that requires remedial action then the inspection schedule should restart for that site.

Recommendation: Inspection of reclamation at all sites should be based on a decreasing schedule of frequency that begins with a monthly or every-other-month schedule and reduces to quarterly after one year and semi-annually after two years of each inspection schedule without the need for remedial actions. If remedial actions are required then the inspection schedule should re-start for that site.

Revegetation Plan

Reclamation Plan Attachment A, Reclamation Seeding and Mulching Procedure, describes many important features for seeding and mulching. However, it fails to establish a quantifiable revegetation plan or criteria for revegetation success. The Reclamation Plan should, at a minimum, establish basic revegetation plans and criteria that can be applied as a basis to all sites to be revegetated (where modification is necessary it can be so-described in the particular section of the Reclamation Plan.

The Reclamation Plan should establish specific goals for essential revegetation features and not just generalized, conceptual goals. There should be clear noxious weed criteria, based on basal and aerial cover, which should be used to trigger treatment and/or retreatment.

Vegetation cover goals should be established. Further, the percentage cover should be required to persist for at least 5 consecutive years prior to bond reduction or release. Plant growth (germination and early growth) is not as important as long-term establishment.

Because post-mine land uses will not be homogenous, it will be important to establish criteria for both alpha and beta diversity. Such criteria should make clear both aerial and basal cover-percent and further identify criteria for success and failure for both alpha and beta diversity. Without these standards revegetation could achieve some goal or required percent coverage but not establish, or even provide a reasonable ecological basis for future establishment of the diverse vegetative cover that will persist and support post-mine land uses. These standards should roughly mimic the pre-mine alpha and beta diversity numbers for the mine, broken down into appropriate sub-regions. The goal should be to ensure that both species numbers and richness are established - which is necessary to achieve post-mine land use goals.

The 5-year period described s for bond reduction/release should re-start whenever revegetation activities are taken to enhance revegetation. The goal of any minimum period should be reasonably demonstrating that plants have established and are self-sustaining. If supplemental activities are taken (such as watering, adding amendments, fixing erosion or subsidence, recontouring, reseeding, planting, weed control, etc.) then the clock should re-start to ensure that vegetation is actually surviving on its own. The 5-year period should demonstrate the site's ability to sustain itself - not demonstrate that with various treatments the company can keep the site growing.

Recommendation: Establish clear noxious species/weed criteria, including the lowest amount of weeds that will trigger treatment and the highest allowable percentage of noxious weeds that will be allowed for bond reduction/release.

Recommendation: Establish minimum percentage vegetative cover goals of at least 50% after three years and 80% for five years before determining "success" or allowing relevant bond release.

Recommendation: Establish clear alpha and beta diversity requirements for vegetative cover.

Recommendation: Revegetation success should be measured no sooner than five years after revegetation goals have been met - without additional treatments or activities. If additional treatments or activities are undertaken, the 5-year clock should restart to ensure that revegetation and long-term plant establishment has actually occurred.

Weed Plan

The Reclamation Plan does not establish a detailed weed control plan, but weeds could significantly threaten the post-mine land uses. Weed problems can begin during the first stages of mining, particularly during topsoil salvage operations and establishing nurse crops, when weeds can begin to take hold.

Recommendation.** **A weed-prevention program should be developed and implemented. At a minimum, this plan should include, but not necessarily be limited to:

Certification of weed-free seed;

Processes to prevent weed introduction (such as washing vehicles entering the site);

Weed-response plan identifying how weeds will be controlled if they do come to the site.

Commitment to Reporting

It is important that the public be able to participate in all phases of mine permitting, operations, closure, and post-closure activities. To support this need, monitoring and discharge reports, including reporting on contamination of surface and ground water, should be made publicly available in a timely manner.

The mine should immediately notify the public of leaks, contamination, etc., and develop a system for such timely notification in a way that is broadly accessible to all affected parties. This is essential for trust and to develop a working relationship with the public, especially affected communities. Adequate monitoring is the only way to determine spills and their impacts. Unknown leaks, or leaks that employees fail to report or attempt to hide will remain undiscovered and their contamination will continue or disperse unless monitoring is in place to detect them. Adequate monitoring before, during, and following mining also protects the company, because it allows all involved to determine what is caused by the mine versus other sources/causes.

Before permit issuance, actual monitoring points for all monitoring should be clearly identified in terms of location and times of sampling. Moreover, monitoring points should be representative and be close to the discharge, to prevent long mixing zones that may become essentially sacrifice zones.

Recommendation. Contaminant release and incident reporting structures should require that the company provide environmental data and reports to the public. There should be full transparency and the company should commit to informing the public and government about any unplanned or unpermitted releases as soon as it becomes known - not just during the regular document/reporting cycle. Annual or even quarterly reports do not adequately address the public's right to know about problems at the mine. These are essential for good operating procedures and public trust.

Thank you for the opportunity to comment on this Supplemental Draft EIS.

Sincerely;

David M. Chambers, Ph.D., P. Geop.

Stuart M. Levit, M.S., J.D.

References Supplied as a part of these Comments:[16]

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Updated NI 43-101 Technical Report on the NorthMet Deposit, AGP Mining Consultants Inc., January 14, 2013

[16] The PolyMet references are not provided because they are references from the SDEIS. One reference, a book, Safety of Dams, Flood and Earthquake Criteria, is available online for reading but not copying.

SPECTRUM ENGINEERING

To:	Jennifer Engstrom, MDNR
From:	Donald Sutton
Date:	February 24, 2012
Subject:	HydroMet and Stockpiles - review of Barr responses to comments

PolyMet Geotechnical Modeling Work Plan comments

Flotation Tailings Basin Geotechnical Model for SDEIS, FEIS and Permitting:

1. Item 3a. The bentonite seal is a hail Mary type of concept in my opinion. I believe it will exacerbate erosion and slope failure and will eventually fail, so I recommend that the stability analysis should assume the bentonite doesn't prevent seepage so far as stability is concerned.
    a. What is the stability of the side slopes if a layer of tailings is placed above the bentonite and it becomes saturated? Will the bentonite slope fail?
    b. What if the bentonite slope is saturated and there is an earthquake or a thunderstorm?
2. I am concerned about long term climate change and how it will affect the water balance in the tailings facility because this can affect the water level, the water head, the saturation, and most importantly, erosion.

    Erosion can cause the shape of the embankments to change over time, it can cause erosion and gullying that can create a pathway for water to escape from the pond. This bothers me, because the proposed design is temporary and will fail unless it is perpetually maintained. I am surprised that the Minnesota statutes allow a temporary impoundment structure to be permitted permanently. This wouldn't be allowed in other jurisdictions. I realize that this will be addressed during the permitting and financial assurance review. Estimating the liabilities will be contentious.

Stockpile Geotechnical Models for SDEIS, FEIS and Permitting:

1. Is there a schedule for collecting the foundation data? How does it relate to the EIS if the data isn't obtained in time?
    a. As a practical matter, I think the stockpile geotechnical designs pose no geotechnical EIS related concerns provided they honor the statutes. My concerns are related to water management, erosion control and geochemical issues.
    b. There is a potential minor leak risk with the liners, but this is minimal and has been addressed. I would like to see the sub-base designed so that any leakage can be collected or directed to the pit pumping system. At some point, someone needs to check this, but it isn't a geotechnical item.

1413 4th Avenue North • Billings, Montana 59101 • 406/534-4660 • Fax 406/259-1456 • sutton@spectrum-eng.com

Message

From:	Liljegren, Michael W (DNR) [/O=MMS/OU=EXCHANGE ADMINISTRATIVE GROUP (FYDIBOHF23SPDLT)/CN=RECIPIENTS/CN=LILJEGREN, MICHAEL 9858C687-9028-4AE2-AFB5-484CE02FE190]
Sent:	11/6/2012 9:24:52 AM
To:	Kunz, Michael (DNR) [michael.kunz@state.mn.us]
Subject:	RE: PolyMet Hydro questions

I will try to get to them today. Sorry for the delay.

Mike

From: Kunz, Michael (DNR)
Sent: Monday, November 05, 2012 1:14 PM
To: Liljegren, Michael W (DNR)
Subject: FW: PolyMet Hydro questions

Hey there – following up this again – I would like to let Cecilio know if his hydro related questions below (5 of them) have been "explored" resolved thru the process – over the life of the project/history…and you have history!!

Thanks

Mike

From: Kunz, Michael (DNR)
Sent: Thursday, October 25, 2012 3:57 PM
To: Liljegren, Michael W (DNR)
Subject: RE: PolyMet Hydro questions

Hey Mike – I know your out – but I am following up with items…one is this on the hydro related comments from Cecilio listed below.

Thanks

Mike

From: Kunz, Michael (DNR)
Sent: Monday, October 15, 2012 3:21 PM
To: Liljegren, Michael W (DNR)
Subject: PolyMet Hydro questions

Hey Mike – below are some comments on Hydro that came from EOR and we thought it best to past through you to discuss/provide input before going to PolyMet/Barr (if need to)…were a number of these already hashed out through the process…

Thanks

Mike

#	To pic	V er	Location (Doc, Page, Figure, or Table)	Section Number	Comment/Concern	Reviewer	PolyMet Response
5			page 33	2.4.1.2	SCS methodology does not seem the most appropriate to estimate annual runoff, evapotranspiration and soil retention since it does not account for in-between storm events changes in hydraulic conductivity and soil moisture conditions. Further more, if the recorded daily precipitation is equally distributed during the day (as it seems to be the case with the HELP model), upper soils moisture retention ability and evapotranspiration could be significantly overestimated, reducing the amount of water in contact with the geomembrane and eventually having the ability to percolate.	CO	
6			Large Table 3		In line with the comment above, the evapotranspiration numbers in table 3 (around 18 in/year) seem high for the type and configuration of the reclamation proposed (i.e. more runoff and lateral drainage off geomembrane could be expected)	CO	
7			Large Table 2		The saturated hydraulic conductivities for the vertical percolation layers 1 and 2 are very low for the type of materials described (i.e. inorganic silts with fine sands, etc). For reference, typical Type C hydrologic soils (fairly low general permeability) are calibrated at about 3.5×10^{-4} cm/sec. A value of 1.9×10^{-5} cm/sec. for vertical percolation layer 2, seems totally out of range.	CO	
8			Large Table 3		The comment above will be a non-issue if the saturated hydraulic conductivity for ALL the layers was increased to 1×10^{-3} cm/sec. Was that the case, including vertical percolation layer 2?	CO	
11			page 105	8.4.3	The long-term infiltration number of 6.5 in/year (or 5×10^{-7} cm/sec.) appears to be very low for the bentonite-amended FTB pond bottom (specially given the uncertainties associated with this methodology). Can you provide some justification for this value?	CO	

Michael R. Kunz, PG, CHMM
Principal Planner
DNR – Land and Minerals

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MINE TAILINGS DAMS:
WHEN THINGS GO WRONG

Michael Davies, Todd Martin and Peter Lighthall
AGRA Earth & Environmental Limited, Burnaby, BC

Abstract

Mine tailings impoundment failures continue to occur at unacceptable rates. The worldwide mining industry has experienced roughly one significant impoundment failure per year over the past 30 years. Many of these failure events have resulted in massive damage, severe economical impact and, in several cases, loss of life.

A tailings impoundment failure case history database has been developed. In addition to an overview of this database, the basic features of a number of specific case histories are presented that provide valuable lessons to the industry. From the overall database, failure modes, failure impacts, and failure frequency are identified. The review of failure modes shows that most events can be attributed to easily preventable causes - a disappointing conclusion but one that offers a readily identifiable solution. The review of failure impacts indicates the large scale of immediate economic losses and expensive longer-term harm resulting from tailings dam failures.

The paper shows there are clear trends that arise from objectively reviewing tailings dam failure case histories. Understanding these trends greatly assists in enhancing design, construction, operation and closure stewardship of mine tailings facilities. As demonstrated by a review of case histories, an ignorance of past failure events and the lessons offered by these events can be highly contributory to subsequent failures.

The mining industry is at a crossroads with tailings impoundment performance - is the relatively constant failure frequency trend for the past 30 years going to continue or decrease as we enter this new century?

Introduction and Perspective

Dams have been used for water supply and/or flood control purposes for thousands of years. More recently, dams have been developed for both hydroelectric power generation and the retention of industrial byproducts such as mine tailings. Mine tailings dams, which really became recognized as "structures" near the beginning of the 20th century, rival or in many cases exceed the scale of conventional water supply, flood control or hydroelectric dams. Despite their size, and despite tailings impoundments representing some of the largest man-made structures, tailings dams have only gained recognition as "dams" in the last few decades.

Conventional dams have had a generally good safety record although catastrophic failures have occurred. Examples from each of the past three centuries include:

- The 46-m high Estrocho de Rientes dam in Spain breached in April 1802 following first filling of the reservoir. The town of Lorca was inundated and approximately 600 people lost their lives.
- On May 31, 1889, the 22-m high South Fork dam in Pennsylvania initially overtopped and, within three hours, fully breached. The flood damage included 2209 fatalities.
- On October 9, 1963, an overtopping event of the 266 m high Vaiont Dam in Italy occurred as a result of a reservoir landslide. The resulting landslide induced wave passed over the dam roughly 250 m above the crest and swept down more than 500 m into the valley below killing about 2500 people in the villages of Longarone, Pirago, Villanova, Rivalta and Fae. The actual dam structure was essentially undamaged by the overtopping event.

Conventional dams continue to be constructed to greater heights with greater storage volumes. However, the safety record of conventional dams has been steadily improving over the past 40 years to the point that the probability of a conventional dam failure in any given year is roughly 1 in 10,000. As will be shown in this paper, this safety trend is not the case for mine tailings dams which appear to be failing at a rate at least ten times higher than that for conventional dams. Some make a different argument (e.g. Bruce et al., 1997), implying that tailings dams are equally "safe" as conventional dams and that both are being built to at least the same "state-of-the-art" practice. This latter interpretation of the statistical database is common and worrisome as it can lead to a complacent attitude. It also does not appear to account for the fact that tailings dams can undergo environmental failures while maintaining physical integrity - an issue not readily associated with conventional dams.

The authors support efforts to show the mining industry in a good light with respect to the tailings dam performance history. Recent trends and initiatives in tailings dam stewardship, spearheaded by the mining industry, are extremely positive and encouraging (Martin and Davies, 2000), though these initiatives tend to get ignored by a relatively biased news media. However, an objective evaluation of the tailings dam failure database illustrates that many tailings dams are not being designed, constructed and/or operated to adequate standards. Moreover, the safety record of tailings dams cannot be considered acceptable given the tremendous damage to the overall mining industry that every new failure provides.

Tailings dams currently have a higher profile in the mining process than at any previous period. There has been a dramatic increase over the past ten years in the number of regulatory agencies involved in setting prescriptive and/or rigid guidelines. The number of mining companies with internal programs aimed specifically at assessing current and planned tailings dams likely outnumbers those who do not have such programs; at least for medium to large sized organizations. An increasing number of undergraduate programs offer at least some form of training in the basics of tailings dam design and the number of graduate theses published on tailings dams has roughly doubled over the past decade. Design professionals have an increasing number of technical forums to update their skills and compare design competency with their peers.

So why do failures of tailings dams continue to occur? The failures are not just of older facilities constructed without formal designs, but include facilities designed and commissioned in the past 5 to 20 years - supposedly the "modern age" of tailings dam engineering.

The first step in evaluating the reasons for continued tailings dam failures comes from recognizing the uniqueness of mine tailings dams. The unique attributes include:

- Tailings impoundments are among the largest manmade structures with several approaching 1 x 10^9 tonnes of stored slurried tailings;
- Tailings dams are built on a continuous basis by mine operators; and
- Tailings dams are a cost to the mining process - they do not generate a revenue stream akin to a hydroelectric dam.

Mining companies typically do not have in-house geotechnical expertise, instead there is reliance on periodic design and perhaps construction monitoring from consulting engineers. Most large-scale water supply and/or hydroelectric agencies more often than not have very capable dam designers and surveillance engineers/technicians in-house. Owners of large conventional dams also typically retain an independent board of eminent consultants to provide expert third-party review. This is not a typical practice in mining at this time.

Are the unique features of tailings dams the reason for the failure trends? The authors suggest that a combination of factors including a lack of input from appropriate external consultants and/or the reliance on third-party consultants without adequate review of their work are highly contributory to the failure trends. As noted by Davies and Martin (2000), there are basic requirements for a designer working in tailings dam engineering and these requirements need to be followed.

This paper examines the phenomenon of tailings dam failure, or, “when things go wrong." The paper is not geared at assigning blame for dam failures but takes the approach that most, if not all, of the failures that have occurred fit into a very consistent set of trends. This consistency is emphasized with the clear premise that if one becomes familiar enough with these trends, future failure events will not arise from an ignorance of the lessons offered by the failure database. If the mining industry collectively embraces the lessons from these trends, the current profile surrounding tailings dams can perhaps wane considerably as the safety record for tailings dams improves to the standard demanded by those who are so quick to criticize the industry.

Definition of Failure

When tailings dams go wrong, it is to say that they have failed. Websters' dictionary offers the following for defining failure: falling short, weakening, breakdown in operation, neglect, not succeeding, becoming bankrupt. All of these have some appropriateness with tailings dam incidents. Leonards (1982) in his Terzaghi lecture defines failure as "…an unacceptable difference between expected and observed performance".

The authors suggest the terminology offered by Leonards (1982) captures what failure means in the context of tailings dams. Failures need not be catastrophic flow failures for those who wish to learn the most from the errors of others. In fact, there are dramatically more "mundane" failures to learn from (e.g. compare the USEPA "failure" case histories, USEPA, 1997, with the USCOLD, 1994, "failure incident" summary document). While the more catastrophic failures gather the most attention and certainly dominate the typical failure databases that get developed, the same trends and lessons are available from the lesser failures (also called "upsets" by many in the industry). As the lesser failures tend to get very little publicity, and almost never any technical publication, practitioners of tailings dam design should keep their own database developed from observations obtained from reviews, audits and the like.

Tailings Dam Failure Database

There is a very poor database of the world's tailings dam inventory. From an extensive literature review and discussions with regulatory officials worldwide, it is estimated that there are somewhat more than 3500 tailings dams worldwide. This total is made up of contributions that include the following where relatively good inventory lists exist: 350 in Western Australia, 65 in Quebec, 130 in British Columbia, 400 in South Africa and 500 in Zimbabwe.

As far as performance of these dams, there are a number of publications that summarize portions of the worldwide tailings dam failure incident database. These include the four most often referenced:

1. 1994 USCOLD database of tailings dam failure incidents
2. 1996 UNEP database on mine waste incidents
3. 1997 USEPA summary of relatively recent tailings dam incidents largely focusing on non-compliant events and limited to certain jurisdictions of the United States.
4. WISE Internet site.

The authors, through reviews and similar assignments, have been made aware of a significant number of failure case histories not captured by any of the above documents, but which occurred within the timeframes and jurisdictions reviewed in each case. This does not condemn any of the above efforts - these summary documents are of tremendous value. The point illustrated is that these publications do not offer the entire suite of information available on tailings dam failures. A great many failures (and the valuable lessons associated with them) go unpublished due to sensitivity and legal implications.

The developed database includes a compilation of available case histories published as single events or in compilations such as those noted above. The database has been further augmented with largely unpublished information gathered by the authors over time. From the overall developed database, it can be concluded that for the past 30 years, there have been approximately 2 to 5 "major" tailings dam failure incidents per year. During no year were there less than two events (1970-1999, inclusive). If one assumes a worldwide inventory of 3500 tailings dams (a tenuous extrapolation at best), then 2 to 5 failures per year equates to an annual probability of between 1 in 700 to 1 in 1750. This rate of failure does not offer a favorable comparison with the 1 in 10,000 figure that appears representative for conventional dams. The comparison is even more unfavorable if less "spectacular" tailings dam failures are considered.

Public Perception and Tailings Dam Failures

The public has high expectations for the mining industry in stewarding mine tailings. There are "fringe" groups who appear opposed to mining of any sort that have either not thought out their position with any real effort or advocate a return to a Paleolithic lifestyle. Given that society, at least implicitly by consumption patterns, places a high value on mined products, public perception of the industry should be commensurate with the value of the industry. Tailings dams, particularly the well-publicized failure events, are lightening rods for public scrutiny of the industry. However, as summarized below, this is not as new a public sentiment as many would believe:

> *"The strongest argument of the detractors of mining is that the fields are devastated by mining operations…further, when the ores are washed, the water used poisons the brooks and streams, and either destroys the fish or drives them away…thus it is said, it is clear to all that there is greater detriment from mining than the values of the metals which the mining produces"*
>
> Agricola - 1556

The public now has instantaneous access to tailings dam events (see discussion on the recent Baia Mare event later in this paper). Given the relatively constant frequency of tailings dam failures over the past thirty years, the public perception is that such events are on the rise due to the increase in publicity each successive event receives. The influence public sentiment can have on the viability of a proposed or existing mining project has never been higher. Public, and some regulatory, perception considerations are now largely driving project design decisions, as opposed to appropriate experience and technical logic.

Failures of tailings dams tend to get viewed as events caused by the collective mining industry. It is naïve to assume that an individual corporation or regulatory jurisdiction is not affected by the dam failures of others. Whether the industry deserves the situation, each failure incident "raises the bar" with both the public and regulatory bodies for the "next" project.

Tailings Dams Failure Impacts

Tailings dam failures can have any or all of the following impacts:

- Extended production interruption
- Loss of life
- Environmental damage
- Damage to company and industry image
- Economic consequences company, and even industry, wide
- Legal responsibility for company officers

For the mining company, the most tangible impact after ensuring public safety is the immediate and longer-term financial impact. Table 1 presents some approximate costs of recent tailings impoundment failures (note that Marcopper was not a dam incident).

Table 1 - Approximate Costs Associated with Tailings Impoundment Failures[1] (all costs x 10^6 US Currency)

Category/ Tailings Facility	Omai	Golden Cross	Marcopper	Los Frailes
Direct Expenses	~$14	>$10	~$80	≥$34
Deferred Cash Flow	~$16	N/A	N/A	~$10
Loss of Asset Value	-	$53	N/A	N/A
Drop in Share Price	25%	N/A	$43 impact in 1996	~50%

[1]Data partially from Vick, 1997.

Tailings dam failures have also resulted in loss of life during extreme events. Table 2 presents a list of the case histories involving fatalities. There are several other incidents, several in the former Soviet Union, where fatalities have occurred but the details of the event and/or the actual number of fatalities are difficult to ascertain.

Table 2 - Examples of Fatalities from Tailings Impoundment Failures

Date	Name	Location	Ore	Dam Type	Failure Cause	Fatalities
1928	Barahona	Chile	Cu	upstream	earthquake	54
1937	Dos Estrellas	Mexico	Au	upstream	slope instability	70
1965	El Cobre	Chile	Cu	upstream	earthquake	>300
1966	Mir	Bulgaria	Pb/Zn	upstream	unknown	(>10)
1970	Mufulira	Zambia	Cu	-	tailings into mine collapse	89
1974	Bafokeng	South Africa	Pt	upstream	seepage	12
1985	Stava	Italy	F	upstream	slope instability	269
1986	Huangmeishan	China	Fe	upstream	seepage/slope instability	19
1988	Jinduicheng	China	Mo	-	dam breach (spillway blockage)	~20
1993	Marsa	Peru	Au	upstream	overtopping	6
1994	Harmony (Merriespruit)	South Africa	Au	upstream	overtopping/slope instability	17
1995	Surigao del Norte	Philippines	Au	upstream	foundation failure	12
					Total Fatalities	**>878**

Finally, as society becomes more litigious, there are increasing legal ramifications for company owners and, in some recent cases, their design consultants. These legal considerations can be more than purely financial as criminal charges were considered in at least two tailings dam failures in the past fifteen years (e.g. involuntary manslaughter). There is no reason to expect this litigious trend to subside in at least the foreseeable future.

Tailings Dam Failure Modes - Example Case Histories

To better illustrate the nature of tailings dam failures, and hence their impacts, a few examples are briefly introduced. In each case, the likely cause of the failure is suggested along with information indicating factual versus perceived impact and lessons that can be learned from the event.

Aurui Gold Plant - 2000

On January 30, 2000, a spill of cyanide laden supernatant tailings water was released from a tailings dam at the Aurui SA Tailings Retreatment Plant in Baia Mare, Romania. The overflow was caused by a build up of surface water and occurred over an approximately 25-m length of the more than 3800 m long tailings dam. The period of December 1999 and January 2000 included abundant snowfall and subsequent melting combined with heavy rainfall during January 26-30, 2000. The overflow from the tailings dam entered the adjacent Lapus River 5.2kms away, then entered the Somes River which flows 75kms to the border of Romania and Hungary. This failure event did not include a massive structural failure of the tailings dam and no tailings were released in the event. The public outcry was immediate and worldwide. Television, radio, newsprint media and the Internet were quick to condemn the 50% owners from Australia and, in the two weeks following the failure to the time of finalizing this paper, the following headlines could be found from "reputable" news agencies during the week ending February 11, 2000:

- **(United Press)** A 100,000-gallon spill of cyanide-contaminated water from an Australian-owned gold mine has caused an ecological disaster in Romania, the British newspaper Guardian reported Friday. The EU and national environmental officials say there is a major threat to the purity of drinking water supplies for at least 2.5 million people.
- **(ABC Internet)** A massive cyanide spill has occurred from an Australian-owned gold operation in Romania, killing fish and rendering water undrinkable. The 100,000 cubic metres of cyanide solution has entered River Szamos, a river which flows from Romania to Hungary. The cyanide concentration is between 325 and 700 times the legal limit.
- **(The Guardian - London)** February 10, 2000. The spill from the Aurul gold mine, near Baia Mare in the north of the country, began 10 days ago. The affected water is reported to be headed downriver toward the Danube and has already reached Yugoslavia. Some sections of the river Tisza, up to 80 percent of the fish have been killed. The Perth, Australia company Esmeralda Exploration, owns 50 percent of the mine and 45 percent is owned by the Romanian government. The remaining 5 percent is owned by foreign investors. Esmeralda Exploration denies responsibility for any spill and claims reports of a disaster were "grossly exaggerated".
- **(CNN Internet)** ABC Radio reports from Hungary - suggestions that 95% of marine life dead in upper section of river. Hungary's thirst prolonged by environmental disaster.
- (**ABC National Radio)** - Gold firm suspended from ASX trade, company plays down damage.

- **(Esmeralda Exploration WebSite)** - Director resigns to pursue other opportunities, February 11, 2000.

The authors are not fully aware of all of the facts of this case history that occurred literally during final edits to this paper. Experience would tend to indicate that the environmental impacts have been overstated and cyanide degradation will be rapid with little, if any, long-term impact to receiving waters. However, the nature of the news reports indicates the climate in which the mining industry finds itself. Any failure, anywhere in the world, can cause immediate and devastating damage to the mining company and its shareholders. The failure also serves to graphically illustrate the need to maintain adequate flood storage volume in tailings impoundments - storage that is based upon appropriate design criteria.

Sullivan Mine, Canada - 1948 and 1991

Davies et al. (1998) describe the static liquefaction event that occurred to the Active Iron Pond tailings impoundment at the Sullivan Mine in August of 1991. The event resulted in a flowslide but, fortunately, another tailings dyke contained the flow and no off-site impact was experienced. The dam had been built on a foundation of older tailings that were placed as beach below water (BBW) material. The failure occurred to the upstream constructed facility by the initiation of shear stresses in the foundation tailings in excess of their shear strength. As the material strained, the pore pressures rose and drainage was impeded leading to liquefaction event. The downstream slope of the dyke was roughly 3H:1V, imposing stresses in excess of the collapse surface for the foundation tailings in an extensive stress path and near to the collapse surface in compressive shear. The Sullivan tailings facility had been under the design and monitoring stewardship of a recognized consulting organization. This event served to demonstrate that "a well intentioned corporation employing apparently well-qualified consultants is not adequate insurance against serious incidents" (Morgenstern, 1998).

Ironically, the 1991 event was similar in nature to a dyke failure that occurred in 1948. The passage of more than forty years should not have been enough to induce the designers into TDA (Tailings Dam Amnesia). As defined by Martin and Davies (2000), TDA refers to a state of tailings dam design or stewardship where lessons available at that very site are ignored in spite of ample available information on-site, visual evidence of previous event occurrence and/or published accounts of incidents on a given project.

Merriespruit, South Africa - 1994

TDA struck again but in a slightly different form with the Harmony mine adjacent to Merriespruit, South Africa. The Bafokeng tailings dam in South Africa, also a paddock upstream facility, failed in almost the same manner (static liquefaction involved) with a similar result; e.g. downstream fatalities. At Bafokeng in 1974, seepage/piping introduced the retrogressive liquefaction flowslide whereas at Merriespruit, overtopping due to inadequate freeboard was ample trigger for liquefaction once enough toe material was eroded away.

The Merriespruit failure occurred on February 22, 1994 in the evening. A massive failure of the north wall occurred following a heavy rainstorm. Over 600,000 m^3 of tailings and 90,000 m^3 of water were released. The slurry traveled about 2 km covering nearly 500,000 m^2. Given the downstream population, it is fortunate that not more than 17 people lost their lives in this tragedy.

Stava, Italy - 1985

Perhaps the most tragic tailings dam failure to date occurred on July 19, 1985. A flourite mine, located near Stava in Northern Italy, had both of its tailings dams fail suddenly and release approximately 240,000 m^3 of liquefied tailings. The liquefied mass moved up to speeds of 60 km/h obliterating everything in its path for a stretch of some 4-km. The flowslide destroyed the village of Stava and also caused considerable damage at Tesero, at the junction of Stava Creek and the Avisio River at the 4 km point from the mine.

The tailings dams were both nearly 25 m high with one directly upstream of the other. The failure mechanism began with failure of the upper dam that in turn overtopped and failed the lower dam as well. The dams were upstream constructed with outer slopes from 1.2 to 1.5 horizontal to 1 vertical. Based upon the likely state of the in-situ tailings, the soil mechanics curiosity with this failure is that the dams could attain such a height prior to failure. There is no question that the design of these dams was not consistent with even the most elementary of engineering principals available at the time. There are a number of "rules" for upstream tailings dam engineering (Davies and Martin, 2000) that were understood for many years prior to the Stava failure. The Stava dams both broke far more of these rules than they followed.

Los Frailes, Spain - 1997

Possibly the most publicized tailings dam failure in history was the 1997 Los Frailes event in Spain. A shallow foundation failure led to the release more than 3 x 10^6 m^3 of process water and tailings from one of two adjacent ponds within an overall impoundment. For this failure, a lack of understanding of the prevailing foundation conditions was directly attributable to a design that was contraindicated by site conditions.

The Los Frailes incident, besides demonstrating the immense power of the media to bring tailings dam failure events to a worldwide audience in a matter of hours, allows a candid assessment of how such incidents can have immediate, and dramatic, impact on a mining company's finances. While other events were certainly at play in 1998, the failure triggered an immediate negative market response. The event occurred at only one of a number of mines for a relatively major mining company. The dramatic share devaluation in 1998 demonstrated the collective impact a single tailings failure event can have on at least medium-term investment confidence in a given corporation.

Omai, Guyana - 1994

Another highly publicized event, the internal erosion failure of the Omai mine's tailings dam, involved a dam breach and the release of cyanide-laden water to the Omai River and then to the much larger Essequibo River. This event caused debatable environmental damage with reports of downstream devastation far outstripping the ability of the dilute contamination to ever accomplish.

The failure was likely the first incident with worldwide outrage. However, the technical debate that was part of the aftermath of this failure was as unique as the degree of public outcry in comparison with the actual damage to the environment. Following extensive post-failure investigations, representatives of the original design consultant and the post-incident Dam Review Team strongly disagreed on relatively basic engineering issues involved in both the original design and the ultimate failure mechanism(s) (Haile, 1997 and Vick, 1997, respectively).

Trends from the Failure Database - Lessons to Learn

By combining published accounts of dam failures and those available through reviews, industry contacts and similar sources, several trends from the tailings dam failure database are evident:

- active dams are more susceptible to failure - this trend may diminish over time if the current trend advocated by some to flood all tailings impoundments upon closure gains momentum
- upstream constructed dams = more incidents - not quite fair as there are more upstream dams, however upstream dams are more susceptible to liquefaction flow events and are solely responsible for all major static liquefaction events
- slope instability/earthquakes for 2/3 of all upstream dam incidents
- seepage related phenomena (e.g. piping due to poor filter design such as was evident in the Omai dam failure) is the main failure mode for non-upstream tailings dams
- earthquakes are of little consequence for most non-upstream dams
- for inactive dams, overtopping is cited as the primary failure mode in nearly 1/2 of the incidents

The list of trends from the database can be continued and has been presented in the past by many others. However reviewers of the case histories seldom make the most important conclusion; that is that there have been no unexplained failure events. If one becomes a student of tailings dam failure case histories, and all designers and regulators should indeed do just that, a single conclusion arises. These failures, each and every one, were entirely predictable in hindsight. There are no unknown loading causes, no mysterious soil mechanics, no "substantially different material behaviour" and definitely no acceptable failures. In all of the cases of the past thirty years, the necessary knowledge existing to prevent the failure at either the design and/or operating stage. There is lack of design ability, poor stewardship (construction, operating or closure) or a combination of the two, in each and every case history. If basic design and construction considerations are ignored, a tailings dam's candidacy as a potential failure case history is immediate.

Table 3 summarizes the main contributory failure mode(s) for a few examples from the tailings dam failure database. In each case, and for all other significant failures in the database, elementary engineering issues and/or basic operating issues have been involved. As shown by the examples in Table 3, there is no need for exotic explanations for the failures and no need to question the fundamental principles of engineering mechanics/hydraulics as the latter have governed in each failure case but were seemingly lost along the way.

Table 3 - Examples of Tailings Dam Failure Causes

Case	Reasons for Failure
Baia Mare	Pond water management - maintenance of adequate freeboard
Los Frailes	Non-recognition of brittle foundation conditions and limitations of surveillance for such conditions
Stava	Water management / dam design with no soil mechanics
El Cobre	Earthquake / dam design incompatible with seismic loading
Sullivan	Foundation / static liquefaction
Merriespriut	Pond water management / static liquefaction
Omai	Filter design/construction issues
Nasca	Earthquake / dam design incompatible with seismic loading

Tailings Dam Failure Axiom - Tailings dam failures are a result of design and/or construction/operation management flaws - not "acts of god".

As a positive corollary to the axiom, if the reasons for tailings dam failures are readily identifiable, there is the potential to essentially eliminate such events with an industry-wide commitment to correct design and stewardship practices. The necessary knowledge exists; there just has to be used.

Concluding Remarks

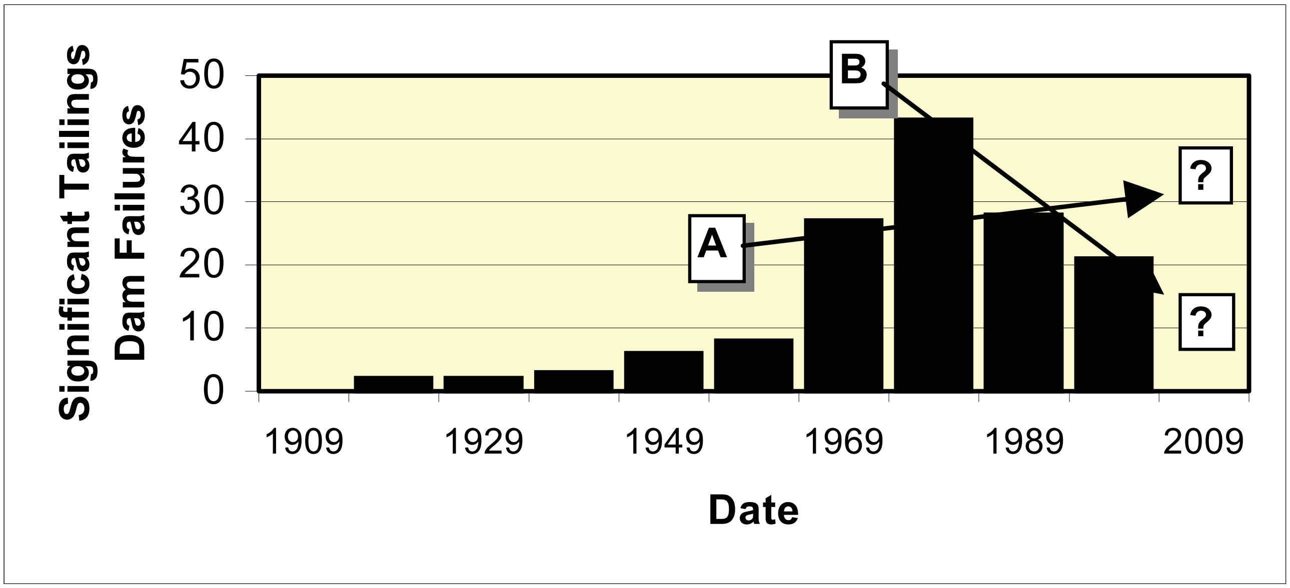
Figure 1 presents a summary of sufficiently well documented "significant" tailings dam failures over the 20th century. From the summarized information in Figure 1, two possible trends are shown and are labeled A and B. Using Figure 1 as a barometer, what is the likely future for tailings dam performance? Is the trend to be like A; either remaining at roughly 2 or more significant failures per year with a gradual increase and perhaps also having the occasional particularly "bad" decade (like the 1970's)? Alternatively, will the trend of an apparent decrease in failures since the 1970's suggested by line B continue into this new century?

An optimistic response, e.g. a B trend, is possible with a commitment from the entire industry to an adherence to fundamentally sound design and operating concepts; the authors are cautiously optimistic, as this commitment appears to be growing. The optimism would be further increased if those in the industry who believe there has not been a significant problem from tailings dam failures would take the time to review and acknowledge the less than perfect history. These individuals should also understand that the current scrutiny under which the industry currently finds itself is largely a result of this history.

This conference is a unique event with the representation of owners, designers and regulators. The authors suggest some minimum expectations for each of the four main participants in the tailings dam life cycle to provide the best opportunity for an improvement in the tailings dam performance record. These participants are:

1. Owners
2. Designers
3. Regulators
4. Public individuals or collectives

Figure 1 - Historical Trend of Significant Tailings Dam Failure Incidents



Owners – only retain design assistance from reputable designers with track records that can be verified. Have submitted designs checked by independent professionals. Give serious consideration to retaining third-party review as part of a periodic audit process. During operations, have a qualified person charged with tailings dam stewardship and provide that individual with the authority to retain professional assistance as deemed necessary. For older operations, be diligent in assessing the history of the operation - look for forgotten "incidents" involving tailings dam management.

Designers – do not work out of your area of competence and/or experience. This includes not using “off the shelf” designs that may have been successful for you in the past but are possibly woefully inappropriate for the climatic/tectonic/foundation conditions for the project at hand. Welcome independent review - do not view such as an attack on your design and/or competency but a benefit to you as much as your client.

Regulators - establish/maintain a database on all tailings dams, operating and otherwise, within your jurisdiction. Maintain candid assessments of the performance records of owners and designers and share such details with other regulators as appropriate. Facilitate developments where the owner presents an independently reviewed design that is consistent with standard design criteria. Work to repeal regulations that are incompatible with common sense.

Public Participants - continue to expect responsible stewardship of the environment by this necessary industry. However, acknowledge that the vast majority of mining industry operators and operations deserve praise for their efforts. Concentrate on factual accounts of incidents to develop and maintain credibility. Avoid supporting non-government organizations that endorse actions against corporations committed to a high degree of environmental stewardship and who operate their mines accordingly.

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"Technical Support for Grassroots Public Interest Groups"

April 30, 2015

To: Betsy Daub
Policy Director
Friends of the Boundary Waters Wilderness
401 N. Third Street, Suite 290
Minneapolis, MN 55401
betsy@friends-bwca.org

Re: Comments on the Geotechnical Stability of the Proposed NorthMet Tailings Basin and Hydrometallurgical Residue Facility in light of the Failure of the Mt Polley Tailings Storage Facility.

On March 14, 2014, the Center for Science in Public Participation (CSP2) submitted comments on the Supplemental Draft Environmental Impact Statement to Lisa Fay, EIS Project Manager, MDNR Division of Ecological and Water Resources, Environmental Review Unit, that included comments on EIS section 5.2.14 Geotechnical Stability, subsections 5.2.14.2.2 Tailings Basin, and 5.2.14.2.3 Hydrometallurgical Residue Facility.

The failure of the Mt Polley Tailings Storage Facility (TSF) has bearing on several features of the proposed NorthMet TSF and Hydrometallurgical Residue Facility. In this report CSP2 will augment its comments of March 14, 2014, to include reflections of the factors involved in the Mt Polley TSF failure that might also come into play at NorthMet.

Also included in this report as Appendix 1 is "A Review of the Report on Mount Polley Tailings Storage Facility Breach, Independent Expert Engineering Investigation and Review Panel."[1]

Sincerely;

David M Chamber, Ph.D., P. Geop.

[1] The original report is: "Report on Mount Polley 2015. Dr. Norbert R. Morgenstern (Chair), CM, AOE, FRSC, FCAE, Ph.D., P.Eng.; Mr. Steven G. Vick, M.Sc., P.E.; and, Dr. Dirk Van Zyl, Ph.D., P.E., P.Eng. Report on Mount Polley Tailings Storage Facility Breach, Independent Expert Engineering Investigation and Review Panel, Province of British Columbia, January 30, 2015"

EXECUTIVE SUMMARY

One of the driving conclusions of the Expert Panel on the Mt Polley tailings dam failure is that the Panel "*… firmly rejects any notion that business as usual can continue*." *(Report on Mount Polley 2015, p. 118)*

They went on to recommend:

> *"For new tailings facilities – BAT* (Best Available Technologies) *should be actively encouraged for new tailings facilities at existing and proposed mines. Safety attributes should be evaluated separately from economic considerations, and cost should not be the determining factor." (Report on Mount Polley 2015, p. 125)*

If taken at face value, as they should be, the recommendations of the Expert Panel clearly say there is a crisis occurring with tailings dam construction and management today. This can also be seen in comparing failure rates of tailings dam to that of conventional water supply reservoir dams – tailings dams fail at rate that is approximately ten times higher than that of water supply reservoir dams (Davies, M.P., 2002, p. 32). There is no engineering reason for this phenomenon to take place, and it is probably the prime indicator that something is wrong with the way tailings dams are designed, constructed, and/or operated.

The primary implications of catastrophic tailings dam failures are:

- public safety (fatalities that result from dam failures);
- economic losses (loss of revenue from business impacted by the accident, as well as the cost of cleanup which is typically borne by public/taxpayer); and
- environmental degradation (even if the tailings released can be cleaned up, complete recovery is impossible and some level of long-term environmental degradation results).

The implications of the recommendations from the Mt Polley Expert Panel should have a direct impact on at least two aspects of the proposed NorthMet Tailings Basin.

First, regulators should reconsider use of old taconite tailings basins/dams for the addition of the non-acid generating NorthMet rougher tailings. The existing taconite tailings basin was constructed using the upstream-type dam construction method. Upstream-type dam construction is statistically the least safe of the three methods of tailings dam construction, and NorthMet will not only be using this same type of dam construction for its future dam expansion, but will also need to depend in part on the safety of the design and construction of the old NorthMet dams, and the underlying geology and nature of the taconite tailings for support of the tailings dam extensions. Neither the existing tailings dam facilities, nor the expansions designed by NorthMet, are designed to be "dry closure" facilities as recommended by Mt Polley Expert Panel. Extending a risky design on top of an old design that itself poses higher risk, against the recommendation of the Mt Polley Expert Panel for dry closure, for a facility that has not yet received regulatory approval, would not be recognizing the long-term risks being posed to the public.

Second, regulators should reconsider whether to construct a wet tailings basin to hold the acid-generating NorthMet tailings. The design of the Hydrometallurgical Residue Facility would also be contrary to the dry closure recommendation of the Mt Polley Expert Panel. In this instance the Panel's recommendation would probably be best met by a dry stack closure design for the Hydrometallurgical Residue Facility. Again, since the mine proposal is still in the draft stage, the design of the Hydrometallurgical Residue Facility should be reconsidered in light of the Mt Polley Expert Panel recommendations.

COMMENTS ON SDEIS SECTIONS

SDEIS Comments from 14Mar14:

5.2.14 Geotechnical Stability

5.2.14.2.2 Tailings Basin

In discussing the construction of the new tailings facility, it is noted:

> *"The Tailings Basin would be constructed using the upstream method, whereby NorthMet dam embankments would be constructed using preferentially borrowed LTVSMC tailings on top of the existing LTVSMC tailings embankment and on the spigotted tailings adjacent to the perimeter embankment." (SDEIS, p 5-561)*

Upstream construction poses the highest risk for seismic and static failure of tailings dams. Most tailings dam failures have been associated with upstream dam construction.

A significant concern with upstream tailings dam construction is its susceptibility to failure during earthquakes. If the tailings upon which the dam is constructed are saturated with water, the tailings do not form a stable foundation for the dam under seismic loading.

Tailings are placed in a saturated state. Tailings materials are relatively uniform in their size and shape, and typically have very low permeability, a fact often cited by mining engineers to argue that liners are not needed for tailings facilities. As a result, it will be difficult to consistently drain the water from all the tailings under the proposed dam expansion.

Continuing to use upstream-type dam construction methods to increase the capacity of the tailings at the NorthMet tailings facility is the least expensive dam construction approach, but poses the most risk to long term seismic stability.

PolyMet picked a "critical" cross section, noting:

"Geotechnical conditions along the length of existing LTVSMC Tailings Basin dams have varying layers of coarse, fine, and slime tailings. Cross Section F, which intersects the northern dam of Cell 2E, as shown in Figure 5.2.14-4, was selected to represent the critical cross section for stability analysis purposes as it is the maximum section and some layers of the weaker fine and slime tailings extend close to the dam embankment, and the dam embankment is underlain by peat." (SDEIS, p 5-565)

(see Figure 5.2.14-5: Cross Section F of the Tailings Basin at Maximum Extent)

The dark blue segment in Figure 5.2.14-5 is the existing tailings dam, and the lighter blue would be the new upstream raises. This figure illustrates very well the importance of the stability of the tailings as a base for the upstream dam.

Mt Polley Implications:

Failure to detect a clays layer beneath the portion of the dam that failed is the primary cause of the Mt Polley accident. This type of failure can occur even absent seismic activity, as the Mt. Polley accident demonstrates. As can be seen from Figure 2 the upstream-type dams will be built on "LTVSMC tailings slimes." These slimes are of a consistency and similar behavior to clays. One of main issues here is be the variability in the consistency of the slimes beneath the upstream tailings dams.

The drill holes bored for the Mt Polley dam foundation sampling were either not spaced close enough, or deep enough, to detect the clay layer that caused the dam to fail. If detected, both the physical properties and thickness of the material in question become significant for predicting how the dam will behave under both static and seismic loading. At Mt Polley the dam designers knew there were clay layers associated with old glacial lakes in the area. The one they knew about was deeper than the one they didn't know about, and deeper glacial lake clays response to the increased pressures from the weight of both the tailings dam and tailings/water themselves was better than that of the shallower (~ 8-10 meters) glacial lake clay they didn't detect.

Many dam-response models assume that the physical properties of each vertical layer are uniform. In fact these properties probably vary in three dimensions. This is one reason why full dynamic modeling should be required for all large tailings dams (like NorthMet), instead of pseudo-static modeling. But even full dynamic modeling is not able to account for all of the complexities of the real geology.

The construction of a new tailings disposal facility on top of an existing tailings facility with problematic features, namely existing upstream construction and slimes as a foundation material, encompasses two of the factors that led to the Mt Polley TSF failure.

As noted above upstream dam construction is the least stable dam construction type. In its recommendations, the Mt Polley Review Panel clearly recommends:

- *For existing tailings impoundments. Constructing filtered tailings facilities on existing conventional impoundments poses several technical hurdles. Chief among them is undrained shear failure in the underlying saturated tailings, similar to what caused the Mount Polley incident. Attempting to retrofit existing conventional tailings impoundments is therefore not recommended, with reliance instead on best practices during their remaining active life.*
- *For new tailings facilities. BAT* (Best Available Technology) *should be actively encouraged for new tailings facilities at existing and proposed mines. Safety attributes should be evaluated separately from economic considerations, and cost should not be the determining factor.*
- *For closure. BAT principles should be applied to closure of active impoundments so that they are progressively removed from the inventory by attrition. Where applicable, alternatives to water covers should be aggressively pursued. (Report on Mount Polley 2015, p. 125)*

In its recommendations for existing tailings impoundments the Panel is warning that "*Chief among them* (technical hurdles) *is undrained shear failure in the underlying saturated tailings…"* Undrained shear strength of the slimes under the proposed upstream tailings impoundment expansion is an issue, given the revelations of the Panel about the frequency of the drillholes. Coupled with the lack of associated lab work for the drillholes at Mt Polley, it is not clear that these issues are adequately addressed at NorthMet (Existing drillholes for NorthMet are plotted on Figure B-1, Historic and Current Geotechnical Test Locations – Barr Engineering 22Sep11). An independent review panel, as recommended by the Panel Report for Mt Polley, should be convened to review the adequacy of the long term storage design for NorthMet.

For new TSFs, the recommended direction of the Independent Expert Engineering Investigation and Review Panel is clear – dry tailings, underground tailings disposal, or other non-wet alternatives. Reason would say that since "*… cost should not be the determining factor" (Report on Mount Polley 2015, p. 125)* all new impoundments should be dry, but economics is still the strongest driving factor in any mine proposal.

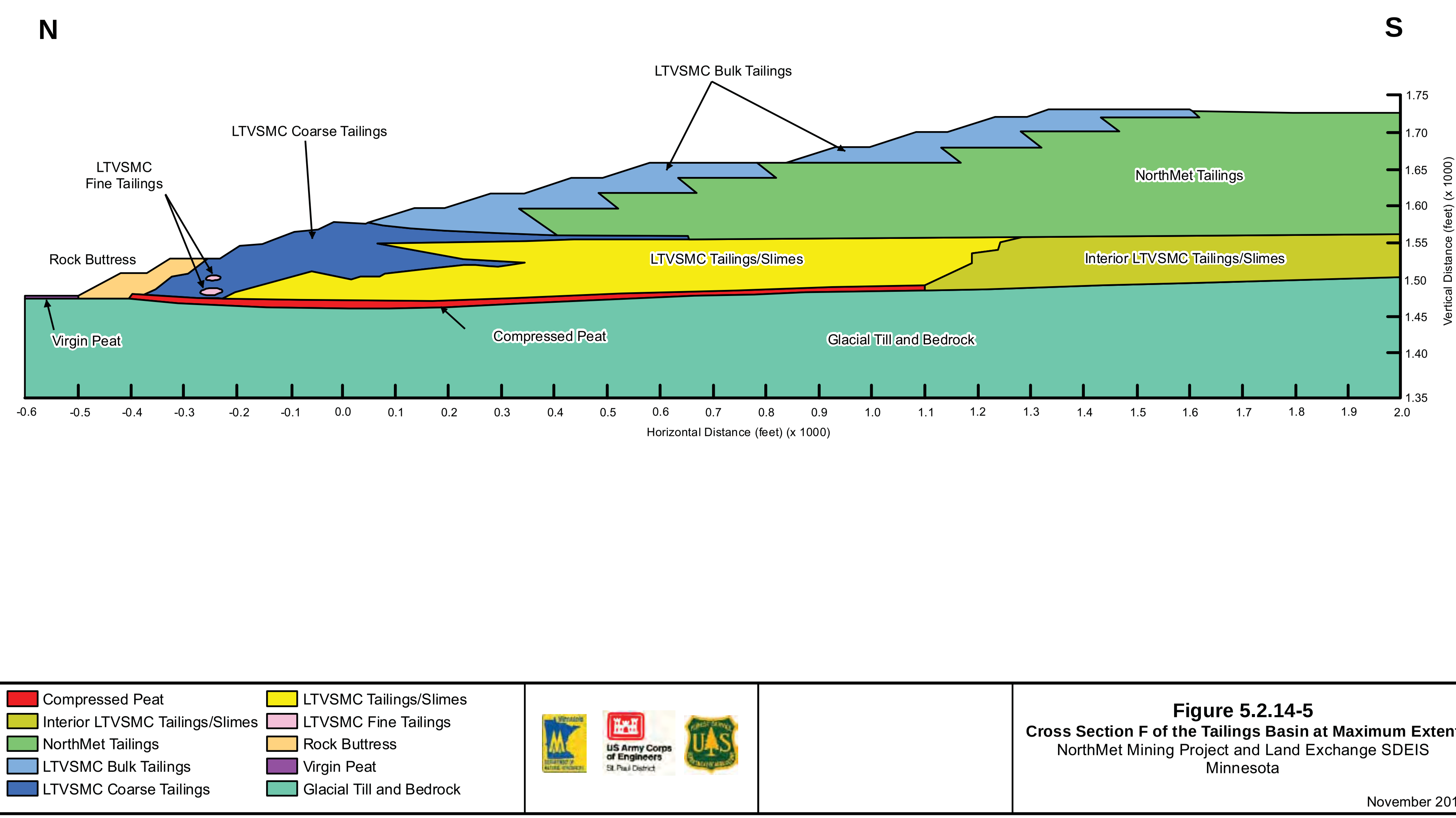
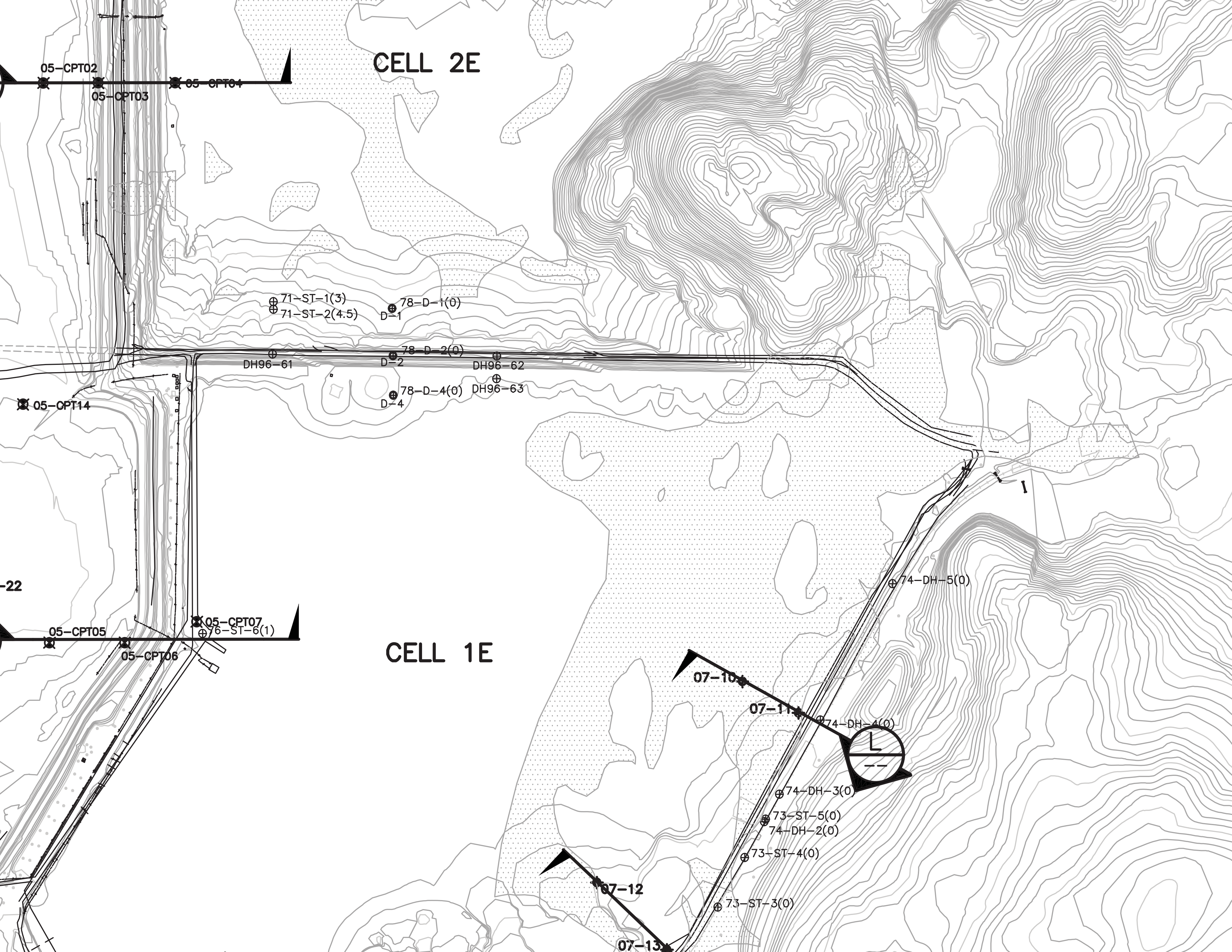


Figure 5.2.14-5
Cross Section F of the Tailings Basin at Maximum Extent
NorthMet Mining Project and Land Exchange SDEIS
Minnesota

November 2013

MCEA Comments Ex. 26


CELL 2E
05-CPT02
05-CPT03
05-CPT04
71-ST-1(3)
71-ST-2(4.5)
78-D-1(0)
D-1
DH96-61
78-D-2(0)
D-2
DH96-62
DH96-63
78-D-4(0)
D-4
05-CPT14
-22
05-CPT07
76-ST-6(1)
05-CPT05
05-CPT06
CELL 1E
74-DH-5(0)
07-10
07-11
74-DH-4(0)
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74-DH-3(0)
73-ST-5(0)
74-DH-2(0)
73-ST-4(0)
07-12
73-ST-3(0)
07-13

The Panel also observed: "*The Panel firmly rejects any notion that business as usual can continue*." *(Report on Mount Polley 2015, p. 118)* The Panel is saying safety, not cost, should be the determining factor in waste impoundment design. The use of the existing tailings pond, and the choice of upstream-type dam construction, are clearly driven by economic considerations.

Before Mt Polley, the engineering companies and the regulatory agencies regularly took the position that a Mt Polley-type failure— the failure of a dam designed and monitored by a reputable engineering company, regulated by an agency in an economically-developed country—could not happen. Not that it was not likely to happen, but that it could not happen. But it did. That was the surprise at Mt Polley, that the system of engineering design and oversight was not robust enough to detect that failure before it happened. This is why the Independent Expert Engineering Investigation and Review Panel "… *firmly rejects any notion that business as usual can continue*."

Returning to the SDEIS Comments from 14Mar14:

It is then noted in the SDEIS:

> *"The results reported in Geotechnical Data Package Volume 1 Version 4 indicate that the proposed design of the Tailings Basin would meet all respective Factors of Safety as required (PolyMet 2013n)." (SDEIS, p 5-565)*

There are several problems with the otherwise good work in Geotechnical Data Package Volume 1 Version 4:

1) The Probabilistic Seismic Hazard Analysis (PSHA) considers the 2,475-year return seismic event to be the largest earthquake the dam will experience. The PSHA should have used the Maximum Credible Earthquake (MCE) as the design earthquake.

   The design earthquake should represent the ground motions or fault movements from the most severe earthquake considered at the site. Since a tailings dam must stand in perpetuity, the design earthquake should be equivalent to the Maximum Credible Earthquake.

   The estimated largest earthquake that could occur at any given location is called the Maximum Credible Earthquake. The MCE is defined as the greatest earthquake that reasonably could be generated by a specific seismic source, based on seismological and geologic evidence and interpretations. The Maximum Credible Earthquake is most often associated with a recurrence interval of 10,000 years.[2]

   If the MCE/10,000-year event is used for the analysis of the 2,475-year event, the horizontal acceleration (horizontal g-force the dam is subject to) will increase significantly.

2) The mean distance to the nearfield earthquake is 100 miles. Probabilistic determination for the size of the largest earthquake is appropriate, but the assumption of 100 miles for nearfield is going to make the horizontal acceleration used to design the dam lower than what it should be.

   The further away the tailings dam is from the location of the earthquake, the less energy the tailings dam will need to withstand in order to maintain its structural integrity. The closer the location of the earthquake to the tailings dam, the higher the cost of building the dam, because the closer the earthquake the more energy the dam will have to withstand.

   Seismologists know that there are many active faults that have not been mapped or have been mapped inaccurately, that some faults believed to be inactive may actually be active, and that there are many inactive faults that may become active again. Because of these considerations, probabilistic methods

[2] Large Dams the First Structures Designed Systematically Against Earthquakes, Martin Wieland, ICOLD, The 14th World Conference on Earthquake Engineering, Beijing, China, October 12-17, 2008

are the more conservative way to determine the magnitude of a Maximum Credible Earthquake for dam analysis.

For tailings dams the most conservative choice for the location of the Maximum Credible Earthquake would be what is sometimes referred to as a 'floating earthquake' on an undiscovered fault that passes very near the site of the dam. This is a way of recognizing that we do not know the present, future, and even the past locations of significant faulting, and associated earthquakes.[3] The conservative choice for a Maximum Design Earthquake would be a Maximum Credible Earthquake that ruptures the ground surface on which the dam is built.

3) The evaluation for dam stability does not employ dynamic modeling.

Polymet did not perform dynamic modeling for the tailings dams.

"Results of the seismic liquefaction screening evaluation (Section 6.5.3.3) indicate that seismic triggering will not occur. As the ***seismic design event (2,475-year return period)*** *would not trigger liquefaction in any FTB materials, per the Work Plan (Attachment A),* ***no additional seismic triggering analyses were necessary****." (Geotechnical Data Package Volume 1 – Flotation Tailings Basin Version 4, PolyMet Mining, April 12, 2013, p 92,* ***emphasis added****)*

PolyMet performed what might be termed a pseudostatic analysis. Today, most US regulatory agencies will not accept pseudostatic methods for seismic design of new dam projects. Dynamic analysis of seismic loading for most new dams is required if the maximum credible earthquake produces a peak ground acceleration of more than 0.1 g at the site.[4]

A pseudostatic analysis (sometimes called seismic coefficient analysis) should only be considered as an index of the seismic resistance available in a structure not subject to build-up of pore pressure from shaking. It is not possible to predict failure by pseudostatic analysis, and other types of analysis are generally required to provide a more reliable basis for evaluating field performance.[5]

An example of a government agency which happens to focus on dam safety and that will not accept pseudostatic analysis is the Federal Energy Management Agency (FEMA). FEMA practice previously allowed the use of the pseudostatic method of analysis in areas of low or negligible seismicity. FEMA does not recommend the pseudostatic analysis to judge the seismic stability of embankment dams.[6]

Dynamic analysis is the most rigorous method of evaluating dam survivability under seismic loading. Typically a dynamic analysis will use finite element or finite difference programs such as TARA (Finn et al 1986), FLAC (Itasca Group 2002), or PLAXIS (PlaxisBV 2002) in which dynamic response, pore-pressure development, and deformations can be fully coupled.[7]

These tailings dams must contain this material in perpetuity. If not, the cost of collecting spillage due to an earthquake-related failure, and rebuilding the containment structure, would be many millions of today's dollars. This is not a risk, or cost, that should be passed on to future generations. If these

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[3] Safety of Dams, Flood and Earthquake Criteria, Committee on Safety Criteria for Dams, Water Science and Technology Board, Commission on Engineering and Technical Systems, National Research Council, National Academies Press, Washington, D. C. 1985

[4] http://www.meadhunt.com/documents/newsletters/persp_water3.pdf, downloaded on 14Jan10

[5] Federal Guidelines for Dam Safety Earthquake Analyses and Design of Dams, Federal Energy Management Agency, May 2005, p. 35

[6] Federal Guidelines for Dam Safety Earthquake Analyses and Design of Dams, Federal Energy Management Agency, May 2005, p. 38

[7] See Federal Guidelines for Dam Safety Earthquake Analyses and Design of Dams, Federal Energy Management Agency, May 2005, p. 32

containment structures are going to be built, the assumptions used to check the design should be conservative, and the models the best available.

Mt Polley Implications:

If there had been an earthquake at Mt Polley, it would likely have triggered a failure as well. But the dam failure at Mt Polley is what is called a static failure – the dam failed under its own weight. In addition to the failure to detect the glacial lake (clay layer) under the dam, there was another significant contributing factor – the dam was not being constructed according to its original design.

The plans for the dam originally called for a downstream slope of 2.0 horizontal to 1.0 vertical. Early on in the construction of the dam, which at the time of the failure had occurred in nine separately approved construction events, a decision had been made to build the dam at a steeper slope (1.3 horizontal to 1.0 vertical) until enough construction rock became available to fill in the downstream "buttress" of the dam. The result was that that the steeper-sloped dam put more pressure on a smaller area, causing it to fail. As noted by the panel, if the dam had been constructed as designed, with a downstream slope of 2.0 horizontal to 1.0 vertical, the pressure from the dam and tailings would have been distributed over a greater area, and the dam would not have failed (Report on Mount Polley 2015, p.108).

While this is a significant contributing factor, the basic cause of the dam failure (failure to detect the glacial lake sediments) remains the same. It is unlikely that a lack of construction material would lead to a similar problem at NorthMet, since the dam proposed would be constructed of existing tailings.

However, another factor that was being used at Mt Polley, which could come into play at NorthMet is also of concern – use of the "Observational Method" for dam construction. The Mt. Polley Panel noted that the Observational Method is a "*commonly accepted approach*" (Report on Mount Polley 2015, p.77) in managing dams. A more candid observation is the Observational Method is another way of saying there will be deviations from the original plan for construction/operation. The Observational Method was invoked for the Mt Polley tailings dam because sufficient quantities of waste rock were not available to build the downstream slope (buttress) of the dam out at a 2H:1V slope as called for in the original dam plans. Instead the downstream slope was built at a steeper 1.3H:1V. The dam designers thought this would be safe, but they didn't know about the glacial lake clays. Had they know about the glacial lake clays, they would have known that building the dam this steep, even temporarily, was not safe. Ironically, if they had built the dam to its original 2H:1V specification, even with the undetected glacial lake clays the dam would have held.

The tailings pond was also being operated with much more water in it than had been planned, again under the auspices of the Observational Method.

Managing mine water was an issue because the water balance predictions were not accurate. The water balance model included the site-specific information to the date of analysis, and future conditions were based on average climatic conditions. They did not account for specific wet year conditions. This is an issue that should have been apparent to both regulators and mine designers, but was either missed or ignored.

The mine had received permission to discharge treated water to resolve this problem, and a treatment plant was scheduled to begin operation in September, 2014. The accident happened on August 4, 2014. Earlier in 2014 the tailings pond faced a potentially catastrophic situation when water reached the top of the dam, and began to overflow. If this had continued, it too would have caused a catastrophic dam failure with concurrent release of tailings and contaminated water, much like the August accident.

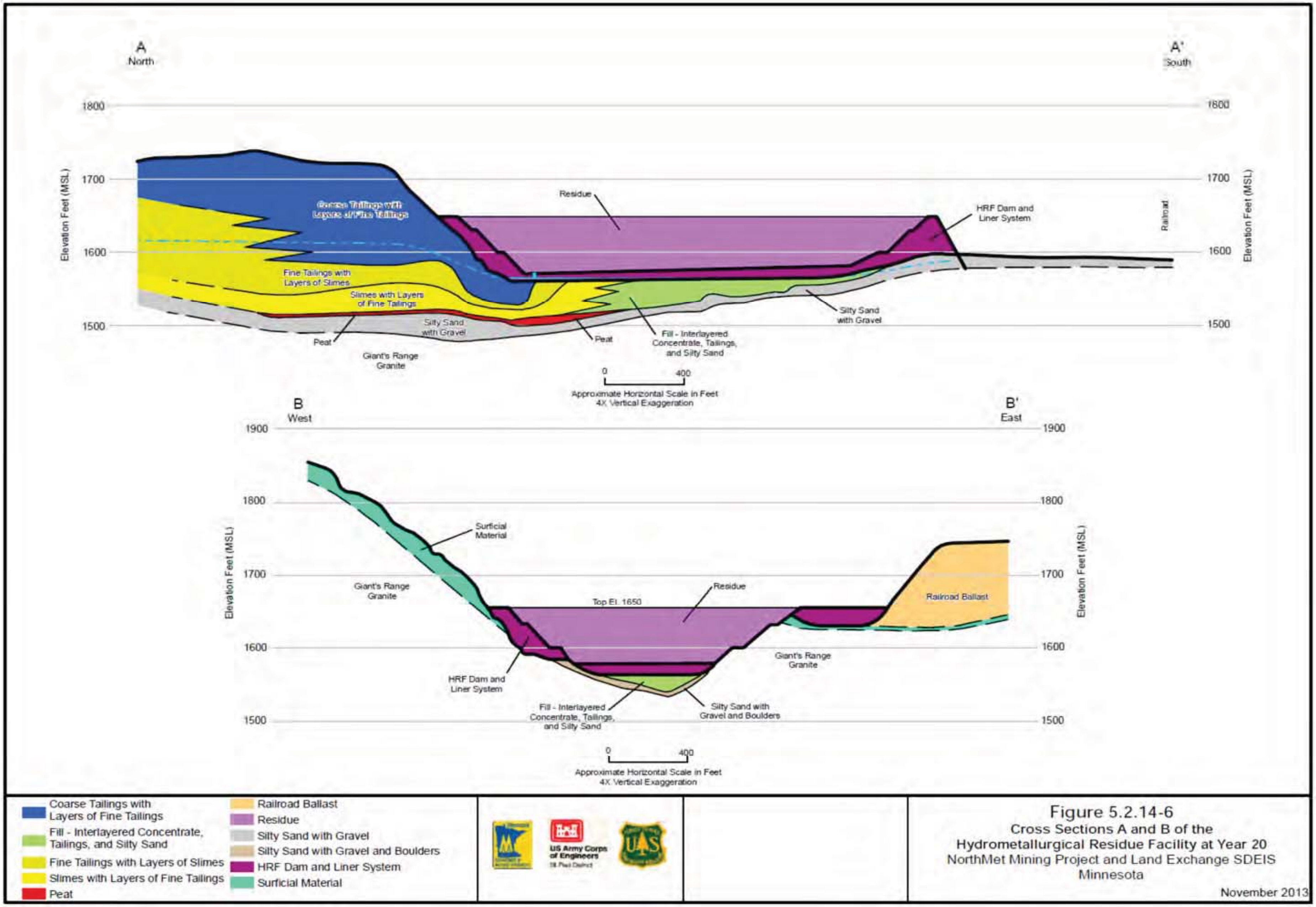

Figure 5.2.14-6
Cross Sections A and B of the
Hydrometallurgical Residue Facility at Year 20
NorthMet Mining Project and Land Exchange SDEIS
Minnesota
November 2013

The overflow of water due to the high water level in the tailings pond caused the mass release of tailings and contaminated water. There would have been a dam breach at Mt Polley even absent the water, but with no water there would have been little tailings release. There would probably have been minimal or no tailings release if the tailings pond were at normal levels – but it wasn't, and the tailings pond full of water led to the large release of tailings downstream.

In the view of the Panel, the Operational Method was misapplied at Mt Polley (Report on Mount Polley 2015, p.136). But more succinctly, the Operational Method was probably invoked at Mt Polley in order to keep mine operation on schedule. Invoking the Operational Method eventually led to the dam failure.

There appears to be no regulatory guidelines as to when the Operational Method can be invoked, or what should be done to put a dam operated under the Operational Method back on its planned track. This is a concern that is appropriate for consideration at NorthMet.

Returning to the EIS Comments from 14Mar14:

5.2.14 Geotechnical Stability

5.2.14.2.3 Hydrometallurgical Residue Facility

Global Slope Stability

As described above with the tailings basin geotechnical design, similarly there was no dynamic modeling for the hydrometallurgical facility.

> *"Liquefaction analysis was not applicable and not performed because the material proposed in the constructed dams would be well-compacted and the Hydrometallurgical Residue Facility liner system would limit leakage through the dams." (SDEIS, p 5-575)*

Even though the construction of the hydrometallurgical facility dam is downstream, the safest type of dam construction, the material that this facility holds is potentially very dangerous to both human health and the environment – if it were to be released. As a result, the geotechnical analysis of the dam should be conservative, and as with the bulk tailings dam, dynamic modeling should be performed.

In addition, it is proposed that the hydrometallurgical facility be placed on a residual layer of taconite tailings.

According to SDEIS Figure 5.2. 14-6 above a large portion of the hydrometallurgical facility (and liner system) will lie on: "Coarse Tailings with Layers of Fine Tailings, Fine Tailings with Layers of Slimes; Slimes with Layers of Fine Tailings; Fill – Interlayered Concentrate, Tailings, and Silty Sand; Silty Sand with Gravel; and, Peat."

Even if this material will be "well-compacted" it would be safer to remove the original peat and silty sand/gravel, and the taconite tailings and slimes, and replacing this material with compacted fill, so that the hydrometallurgical facility is built on a well prepared and verifiably stable base. This is the conservative approach.

Recommendation: The underlying original ground and the taconite waste should be removed from underneath the hydrometallurgical tailings facility, an engineered stable base installed, and dynamic modeling performed on the hydrometallurgical dam.

Mt. Polley Implications:

As noted above, even though downstream-type construction will be used for the hydrometallurgical facility dam, it would be built on a thick layer of taconite tailings. And a large portion of the hydrometallurgical facility (and liner system) will lie on: "Coarse Tailings with Layers of Fine Tailings, Fine Tailings with Layers of Slimes; Slimes with Layers of Fine Tailings; Fill – Interlayered Concentrate, Tailings, and Silty Sand; Silty Sand with Gravel; and, Peat."

According to the presentation on SDEIS Figure 5.2. 14-6, this material is over 50 feet deep under the north embankment of the hydrometallurgical facility. It is not practical to mechanically compact this material without removing and reapplying it, so the statement that it is "well-compacted" (see quote in Global Slope Stability above) will depend on its in situ conditions.

In short, the recommendation of the Panel, that a waste facility should be "*…independent of the integrity of any containment structures" (Report on Mount Polley 2015, p.121)* should be seriously investigated.

The Panel did not make this recommendation lightly. It also acknowledged *"The Panel recognizes that creating dry tailings may increase the amount of water requiring treatment or storage." (Report on Mount Polley 2015, p.122)*

References Supplied as a part of these Comments:[8]

Davies, M.P., 2002, "Tailings Impoundment Failures: Are Geotechnical Engineers Listening?" Michael P. Davies, Geotechnical News, September 2002, pp. 31-36.

Federal Guidelines for Dam Safety, Earthquake Analyses and Design of Dams, Federal Emergency Management Agency, May 2005

Large Dams the First Structures Designed Systematically Against Earthquakes, Martin Wieland, ICOLD, The 14th World Conference on Earthquake Engineering, Beijing, China, October 12-17, 2008

PolyMet 2013e, NorthMet Project Water Management Plan – Mine, Version 2. January 9, 2013

PolyMet 2013n, NorthMet Project Geotechnical Data Package - Flotation Tailings Basin, Volume 1, Version 4. April 12, 2013.

Report on Mount Polley 2015. Report on Mount Polley Tailings Storage Facility Breach, Independent Expert Engineering Investigation and Review Panel, Dr. Norbert R. Morgenstern (Chair), CM, AOE, FRSC, FCAE, Ph.D., P.Eng.; Mr. Steven G. Vick, M.Sc., P.E.; and, Dr. Dirk Van Zyl, Ph.D., P.E., P.Eng.; Province of British Columbia, January 30, 2015

Safety of Dams, Flood and Earthquake Criteria, Committee on Safety Criteria for Dams, Water Science and Technology Board, Commission on Engineering and Technical Systems, National Research Council, National Academies Press, Washington, D. C. 1985 (294 pages, available for reading but not copying online at http://www.nap.edu/catalog.php?record_id=288)

[8] The PolyMet references are not provided because they are references from the SDEIS. One reference, a book, Safety of Dams, Flood and Earthquake Criteria, is available online for reading but not copying.

Appendix 1

A Review of the
Report on Mount Polley Tailings Storage Facility Breach, Independent Expert Engineering Investigation and Review Panel

David M Chambers
Center for Science in Public Participation
February, 2015

A Review of the
Report on Mount Polley Tailings Storage Facility Breach, Independent Expert Engineering Investigation and Review Panel

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Early on August 4, 2014, the Perimeter Embankment at the Mt Polley copper mine near Likely, south-central British Columbia, failed catastrophically. The loss of containment was sudden, with no warning. That failure, which released at least 25 million cubic meters of mine tailings and mine effluent mixed with stormwater into Polley Lake, Hazeltine Creek and finally stopped when it reached Quesnel Lake, a large salmon-spawning fjord-type lake.

The Cariboo Regional District declared a local state of emergency in several nearby communities, the Interior Health Authority ordered drinking water bans, and the Department of Fisheries and Oceans closed the recreational salmon fishery on the Quesnel and Cariboo Rivers. Fortunately, there were no human fatalities or injuries.

Why did the Mt Polley TSF Fail?

The failure of the Mt Polley Tailings Storage Facility (TSF) was reviewed shortly after the accident by an expert panel of three engineers.[1] The words of the panel itself succinctly describes what happened, why it happened, and what we should be doing to avoid similar TSF failures in the future.

> *The Panel concluded that the dominant contribution to the failure resides in the design. The design did not take into account the complexity of the sub-glacial and pre-glacial geological environment associated with the Perimeter Embankment foundation. As a result, foundation investigations and associated site characterization failed to identify a continuous GLU (Glasciolacustrine Unit) layer in the vicinity of the breach and to recognize that it was susceptible to undrained failure when subject to the stresses associated with the embankment.*

The tailings dam was built on top of an old, relatively small, glacial lake that contained mainly clays. The builders of the dam, Knight-Piesold Ltd., made several assumptions that led to this problem. They assumed that the extent of the clay was less widespread that it in fact was, and that the clay constituting the lake sediment (called the Upper Glasciolacustrine Unit – GLU) would not loose shear strength as the sediment was loaded by the weight of the dam, tailings, and water. These proved to be both flawed and ultimately fatal assumptions for the dam.

Figure 1 (from the Report) maps the resulting failure on top of an aerial photo of the failed dam. The increasing load due to the ongoing construction of the dam, and the load of tailings and water behind the dam, finally caused the glacial clay lake-layer to break and slide, rupturing the dam. There were no precursor warnings to the failure. The failed piece of the dam rotated down and out, letting water spill over the top of the failed segment, and in a short time washed that piece of dam away, carrying in its wake almost the entire contents of the tailings basin.

[1]Dr. Norbert R. Morgenstern (Chair), CM, AOE, FRSC, FCAE, Ph.D., P.Eng.; Mr. Steven G. Vick, M.Sc., P.E.; and, Dr. Dirk Van Zyl, Ph.D., P.E., P.Eng. Report on Mount Polley Tailings Storage Facility Breach, Independent Expert Engineering Investigation and Review Panel, Province of British Columbia, January 30, 2015

Figure 1: Plan Showing Direction and Extent of Mass Movements

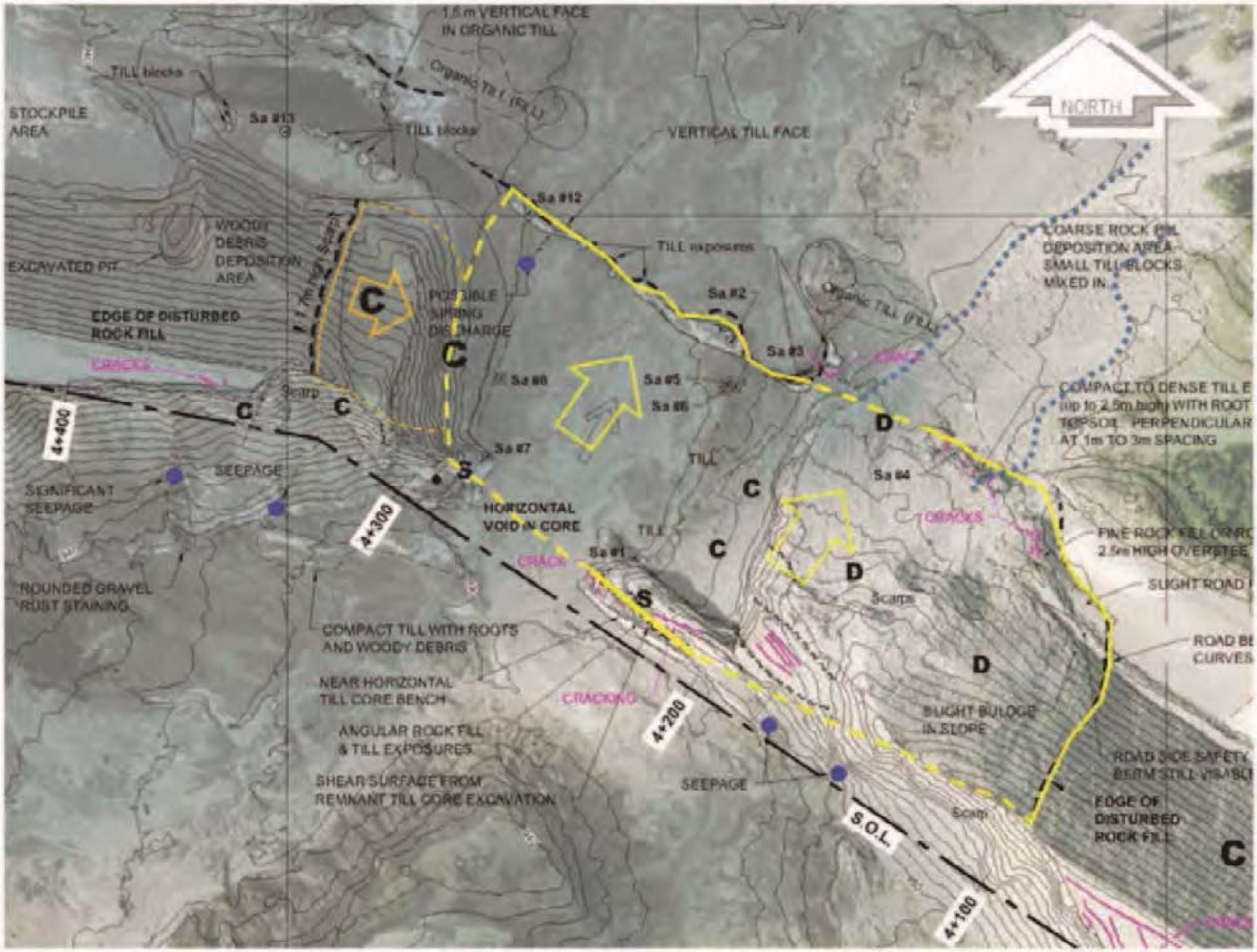
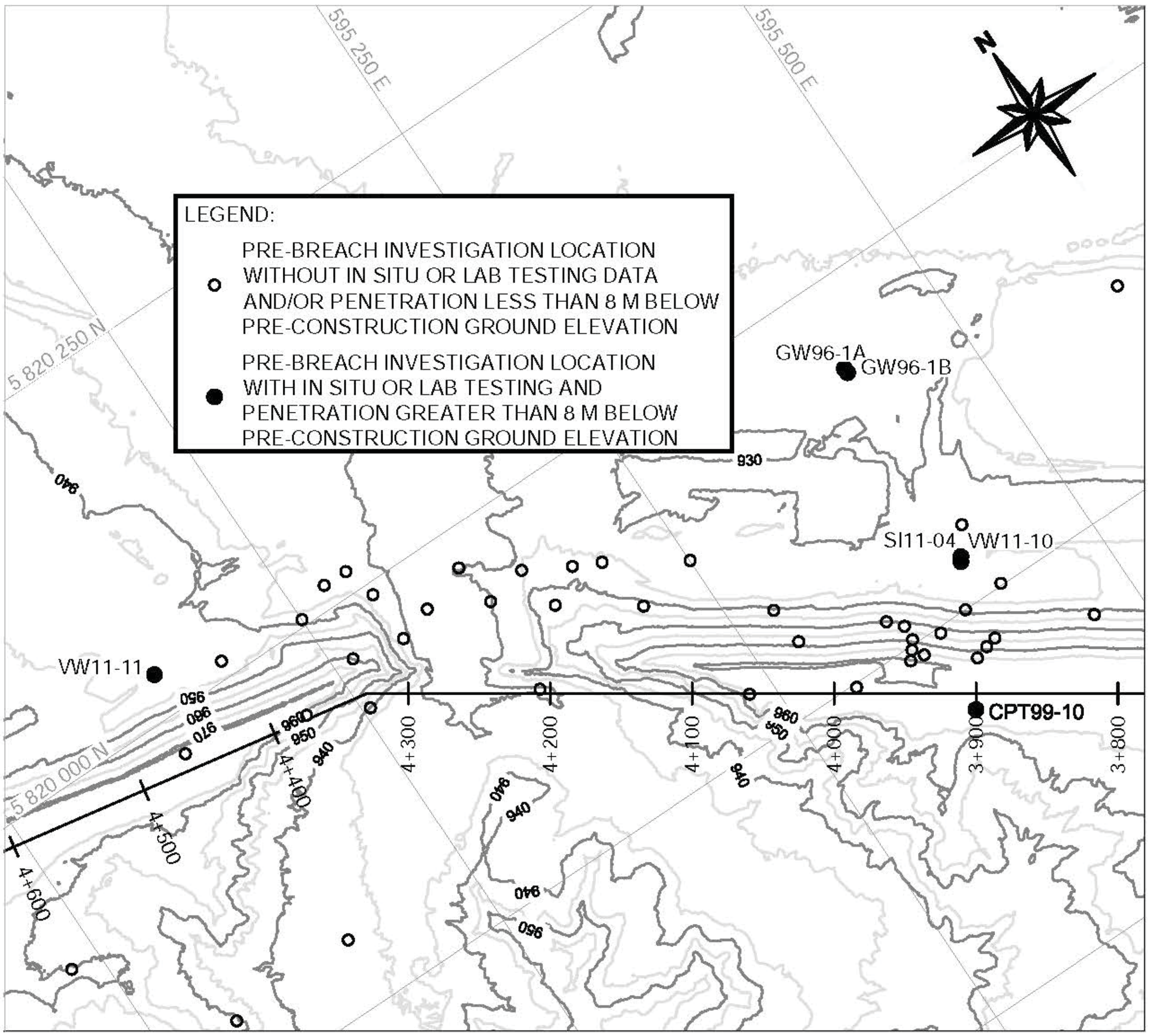


Figure 2 shows the drillholes made before the dam was built. The pre-failure drillholes, depicted as solid circles in Figure 2, were drilled deep enough to intersect the Upper Glasciolacustrine Unit, and clays intersected in those holes were lab tested for shear strength. The pre-failure drillholes depicted as open circles were not drilled deep enough to intersect the Upper GLU.

As can be seen in Figure 2, there are only shallow drillholes (open circles) in the area of the failed dam segment. There are no drillholes in the area of the dam failure that intersected the Upper GLU or that were lab tested for shear strength.

Figure 2: Pre-Failure Site Investigation Drillhole Locations in Breach Area



Post-failure drilling in the area of the failure, Figure 3, did intersect the Upper GLU, and lab testing of these clays clearly determined that the clay of the Upper GLU would fail under the increased pressures of the dam and tailings.

Figure 3: Joint and Panel Site Investigation Drillhole Locations

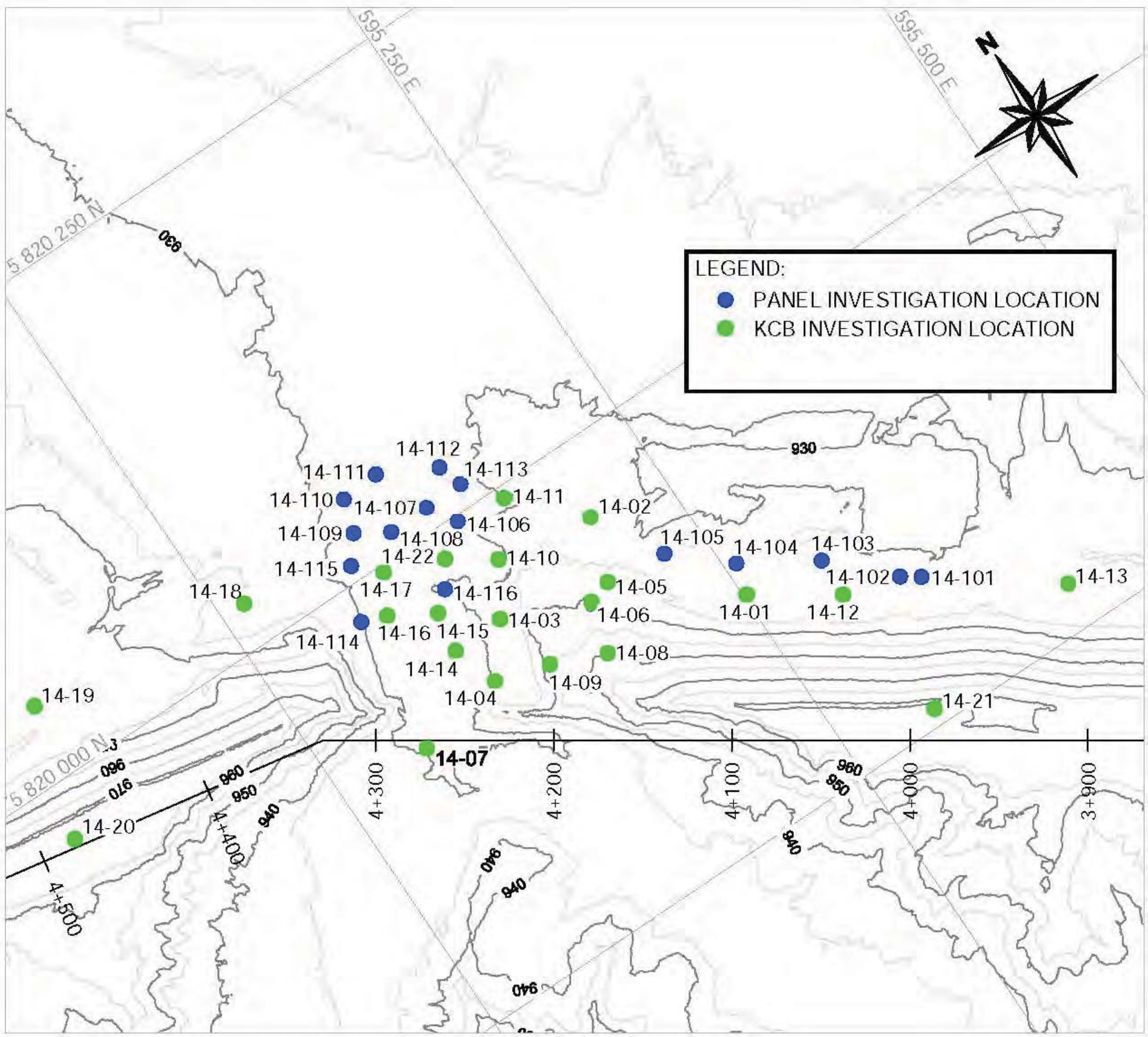
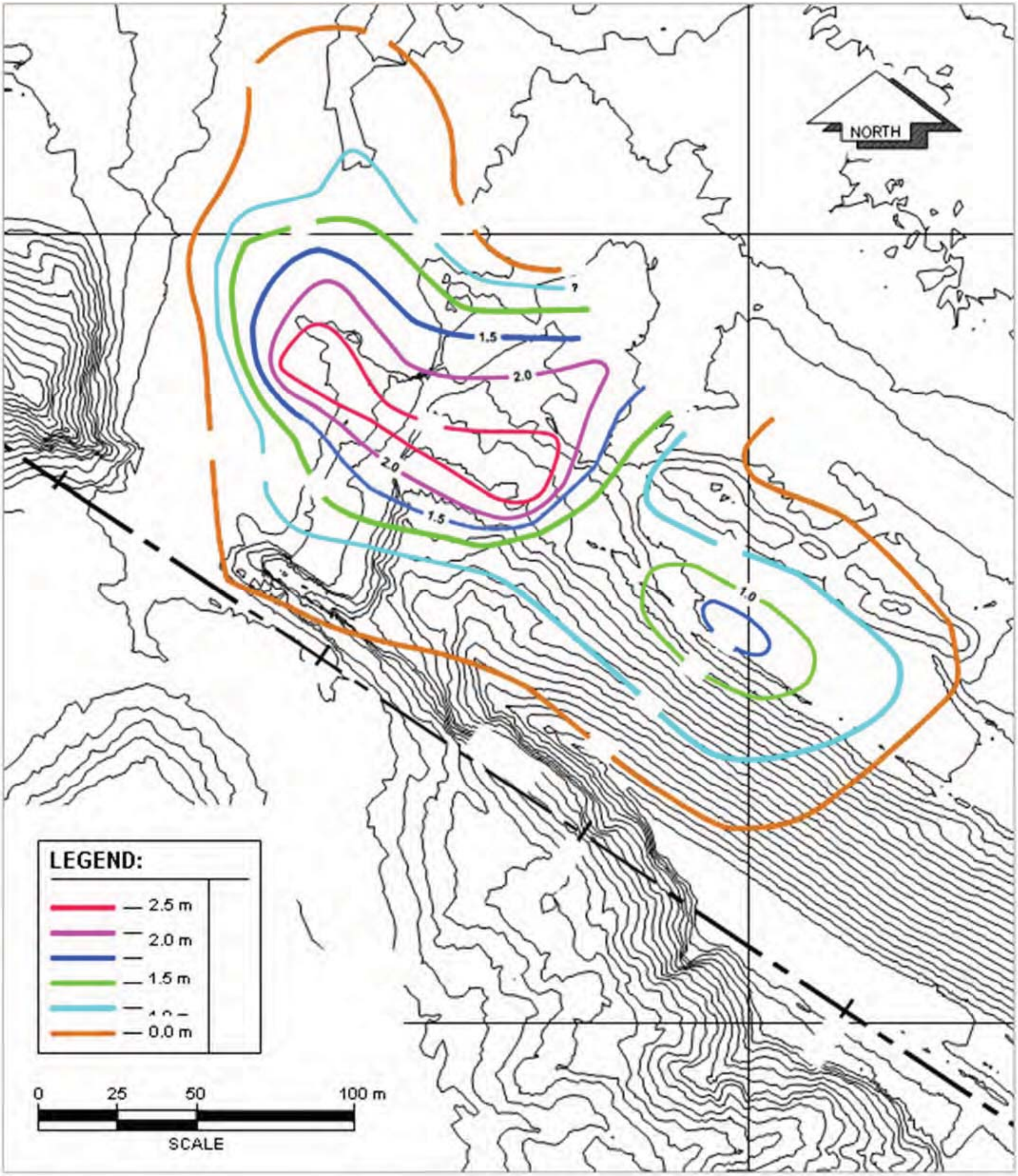


Figure 4 shows the extent and thickness of the Upper GLU – just small enough to have avoided the original deeper drillholes – but large enough to cause the catastrophe.

Figure 4: Contours of Upper GLU Thickness in Breach Area



The factors that contributed to either the dam failure, or that significantly increased the impact of the dam failure, were a bit more complex than just the inability to detect the Upper GLU. The environmental damage due to the outflow of tailings and effluent were heavily influenced by several of these other factors.

Oversteepening of the Downstream Rockfill Zone

> *The specifics of the failure were triggered by the construction of the downstream rockfill zone at a steep slope of 1.3 horizontal to 1.0 vertical. Had the downstream slope in recent years been flattened to 2.0 horizontal to 1.0 vertical, as proposed in the original design, failure would have been avoided. The slope was on the way to being flattened to meet its ultimate design criteria at the time of the incident.*

The plans for the dam originally called for a downstream slope of 2.0 horizontal to 1.0 vertical. Early on in the construction of the dam, which at the time of the failure had occurred in nine separately-approved construction events, a decision had been made to build the dam at a steeper slope (1.3 horizontal to 1.0 vertical) until enough construction rock became available to fill in the downstream "buttress" of the dam. The result was that that the steeper-sloped dam put more pressure on a smaller area, causing it to fail. As noted by the panel, if the dam had been constructed as designed, with a downstream slope of 2.0 horizontal to 1.0 vertical, the pressure from the dam and tailings would have been distributed over a greater area, and the dam would not have failed.

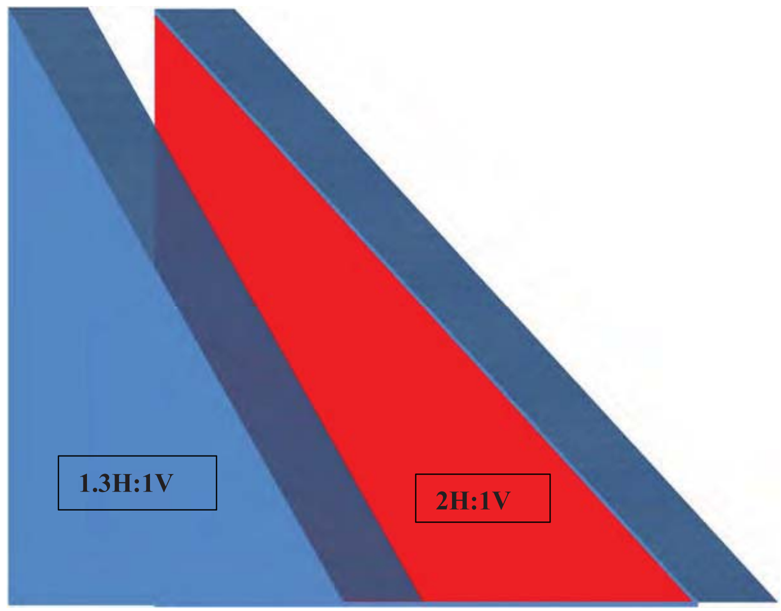


Figure 5: Tailings Dam Downstream Slopes

The panel's overall conclusion was:

> *The dominant contribution to the failure resides in its design. The design did not take into account the complexity of the sub-glacial and pre-glacial geological environment associated with the Perimeter Embankment foundation. ... Hence, the omissions associated with site characterization may be likened to creating a loaded gun. Notwithstanding the large number of experienced geotechnical engineers associated with the TSF over the years, the existence of this loaded gun remained undetected.*

and;

> *If constructing unknowingly on the Upper GLU...constituted loading the gun, building with a 1.3H:1V angle of repose slope over this stratum pulled the trigger.*

and;

> *The design was caught between the rising water and the Mine plan, between the imperative of raising the dam and the scarcity of materials for building it. Something had to give, and the result was oversteepened dam slopes, deferred buttressing, and the seemingly ad hoc nature of dam expansion that so often ended up constructing something different from what had originally been designed.*

Not knowing about, and accounting for, the glacial lake clay "loaded the gun" in the panel's words, and building the dam steeper than the design called for "pulled the trigger."

Other Complicitous Factors

There were a number of other factors that turned up during the course of the investigation of the dam failure that contributed materially to the fundamental cause of the accident itself. However, one factor made the accident significantly worse, and two others could eventually have led to a dam failure on their own.

(1) Tailings Pond Water Level

At the time of the dam failure the water level in the tailings pond was just below the maximum level allowed. For some time the mine has been forced to manage water in the tailings pond at emergency levels due to higher than predicted precipitation

> *The high water level was the final link in the chain of failure events. Immediately before the failure, the water was about 2.3 m below the dam core. The Panel's excavation of the failure surface showed that the crest dropped at least 3.3 m, which allowed overflow to begin and breaching to initiate. Had the water level been even a metre lower and the tailings beach commensurately wider, this last link might have held until dawn the next morning, allowing timely intervention and potentially turning a fatal condition into something survivable.*

The overflow of water due to the high water level in the tailings pond caused the mass release of tailings and contaminated water. There would have been a dam breach even absent the water, but with no water there would have been little tailings release. There would probably have been minimal or no tailings release if the tailings pond were at normal levels – but it wasn't, and the tailings pond full of water led to the large release of tailings downstream.

Managing mine water was an issue because the water balance predictions were not accurate.

> *The water balance model included the site-specific information to the date of analysis, and future conditions were based on average climatic conditions. They did not account for specific wet year conditions.*

This is an issue that should have been apparent to both regulators and mine designers, but was either missed or ignored.

The mine had received permission to discharge treated water to resolve this problem, and a treatment plant was scheduled to begin operation in September, 2014. The accident happened on August 4, 2014.

However, earlier in 2014 the tailings pond faced a potentially catastrophic situation when water reached the top of the dam, and began to overflow. If this had continued, it too would have caused a catastrophic dam failure with concurrent release of tailings and contaminated water, much like the August accident.

Again, in order to stress the severity of the issue, here are the words of Panel:

> *For years, dam raising had managed to stay one step ahead of the rising water. But on May 24, 2014, the water caught up. With Stage 9 nearing completion, what was described as "seepage flow" was observed over the dam core. Intensive surveillance and construction activity over the following days and weeks succeeded in raising low areas around the embankment perimeter, restoring containment integrity, and saving the dam from overtopping failure.*

The problems with the water level demonstrates the multitude of threats at this site. The water level in the tailings pond did not cause the tailings dam to fail, though it caused the damage to be far worse once it did fail. But dam failure due to overtopping by water in the tailings pond was a real risk, and that almost happened on May 24, 2014.

(2) Dam Filter Material

Figure 6: Void in left abutment (note geo-pick for size)

The duty of the filter zone in the dam is to collect any seepage coming through the core and to prevent fines from migrating out of the core. In order for the dam to drain properly internally, the core, filter, and transition (to the buttress) zones must be carefully constructed. Much of the as-placed filter material at Mt Polley failed to meet applicable filter criteria and requirements for internal stability of its grading.

> *... in a sampling of as-placed Zone S filter gradations, the Panel found that 30% were too coarse to meet the ... filter criterion ... with only about 25% satisfying both filter and internal stability requirements.*

If the filter material is too course, it does not act as filter, but more like a drain. This can lead to voids in the core of the dam. This was essentially the cause of the Omai tailings dam failure. Had this situation been widespread, it too could have led to dam failure at Mt Polley.

And, in fact, during the field work associated with the dam failure, a serious void was discovered (Figure 6), but there was no evidence of further voids discovered during the investigation. The quality control function of dam construction was obviously not working satisfactorily. This reflects poorly on both those who constructed the dam, those who were supervising the construction (this should have been an independent party), and on the standards set by regulators, which were not tight enough to detect these errors.

(3) Inoperative Piezometers

A piezometer is a general term used for a well drilled into the dam to measure water level and pressure. Installed in the dam were 116 piezometers. Piezometers were installed in the dam foundation, in various embankment components, such as the upstream fill, core, and downstream transition zone, in drains located in the embankment and foundation, and in the tailings upstream from the embankment.

Piezometers, even if properly located and operating, would probably not been able to detect this type of failure. The piezometers at the Perimeter Embankment were located too far beyond the dam toe to provide critical data, and too far in between to cover the area where the breach occurred, so they were not able to supply information on the dam failure. However, normally they can provide an early warning that the core of the dam is compromised, and can provide warning of impending dam failures.

As early as 2009 the functionality of these piezometers had been an issue.[2] Yet as of August 2014, there were a total of 64 operating piezometers and 52 non-operating piezometers in the dam. There were nine operating and 13 non-operating piezometers along the section of the Perimeter Embankment that failed.

[2] Knight Piésold Consulting, Tailings Storage Facility Report on 2009 Annual Inspection (Ref No VA101-1/27-1, 2011), at 7.

Allowing nearly 50% of the piezometers to be non-operational should not be acceptable either to the dam operator or the dam regulators. Non-operational piezometers take a significant safety tool away from all dam observers.

(4) TSF Management and the "Observational Method"

According to the Panel:

> *The Observational Method is a powerful tool to manage uncertainty in geotechnical practice. However, it relies on recognition of the potential failure modes, an acceptable design to deal with them, and practical contingency plans to execute in the event observations lead to conditions that require mitigation. The lack of recognition of the critical undrained failure mode that prevailed reduced the Observational Method to mere trial and error.*
>
> *The Observational Method was invoked early on as the basis for design. This commonly accepted approach uses observed performance from instrumentation data for implementing preplanned design features or actions in response.*

However;

> *The Observational Method relies on measuring the right things in the right places.*

Interpreting from the Report, invoking the Observational Method allowed the dam operators, designers, and regulators to depart from implementing the planned design of the dam, most notably the allowing the Factor of Safety[3] to go from the planned 1.5 down to 1.3, by not constructing the dam buttressing on the planned schedule.

To make the Operational Method work mine designers would have to have known about the clay layer beneath the dam, but they didn't. They should have had extensive instrumentation to monitor the dam, but the instrumentation present at the mine site was not only in the wrong places, but much of it was not working.

In the view of the Panel, the Operational Method was misapplied at Mt Polly. But more succinctly, the Operational Method was probably invoked at Mt Polley in order to keep mine operation on schedule. Invoking the Operational Method eventually led to the dam failure. There appears to be no regulatory guidelines as to when the Operational Method can be invoked, or what should be done to put a dam operated under the Operational Method back on its planned track.

A (But Not Necessarily The) Way Forward

The Panel opened its recommendations by saying flatly:

> *The Panel **firmly rejects any notion that business as usual can continue. (emphasis added)***

The Panel goes on to explain what this means before rendering specific recommendations:

> *In risk-based dam safety practice for conventional water dams, some particular level of tolerable risk is often specified that, in turn, implies some tolerable failure rate. The Panel does not accept the concept of a tolerable failure rate for tailings dams. To do so, no matter how small, would institutionalize failure. First Nations will not accept this, the public will not permit it, government will not allow it, and the mining industry will not survive it. ... Tailings dams are complex systems that have evolved over the years. They are also unforgiving systems, in terms of the number of things that have to go right. Their reliability is contingent on **consistently flawless execution** in planning, in subsurface investigation, in analysis and design, in construction quality, in operational diligence, in*

[3] Factor of Safety is the ratio of available strength to the strength required for equilibrium.

monitoring, in regulatory actions, and in risk management at every level. All of these activities are subject to human error. ***(emphasis added)***

...

Improving technology to ensure against failures requires eliminating water both on and in the tailings: water on the surface, and water contained in the interparticle voids. Only this can provide the kind of failsafe redundancy that prevents releases no matter what. ... ***Simply put, dam failures are reduced by reducing the number of dams that can fail. (emphasis added)***

Thus, the path to zero leads to best practices, then continues on to best technology.

The "*path to zero*" should not be interpreted literally to mean the Panel believes that achieving zero tailings dam failures is attainable for tailings dams or even tailings impoundments. It does mean the "goal" should be zero failures, and that in order to move toward this goal tailings impoundments need to be designed such that their stability does not depend on the structural integrity of a tailings dam.

Best Available Tailings Technology (BAT)

The goal of BAT for tailings management is to assure physical stability of the tailings deposit. This is achieved by preventing release of impoundment contents, independent of the integrity of any containment structures.

The implication of the statement "*... preventing the release of impoundment contents independent of ... containment structures.*" are significant. This explicitly says that the tailings must have structural integrity that is independent of a containment structure.

Tailings that are saturated with water do not have any structural integrity. The Panel recommends pursuing tailings disposal methods like dry tailings and underground tailings disposal, as well as the development of new disposal technologies, the possibilities for which the Panel considers "ripe" if the right incentives are put in place.

This recommendation from the Panel is nothing short of profound. While it stops short of saying explicitly saying no more tailings dams, it couldn't get any closer without saying it. The 'physical stability of the tailings must be independent of the containment structures.' While it might be argued that a deposit of wet tailings could be made free-draining after deposition, and therefore have some structural stability, tailings are not noted for being free-draining (in fact it is often argued they are self-sealing, that is do not leak pore water into groundwater underneath an unlined impoundment). And even if the tailings were free-draining, the portion of the tailings next to the dam would still depend on the dam for some stability.

The Panel specifically notes that water covers (i.e. maintaining saturated and water-covered tailings in perpetuity) should be avoided, even for potentially acid generating material, because the long-term risk of dam failure is too great. The Panel prefers to see potentially acid generating material stored in a dry manner, even if that means a concomitant increase in the need for (perpetual) water treatment. To the Panel, more water treatment is preferable to long-term wet storage. This is sobering.

In terms of how to apply BAT, the Panel made the following recommendations:

Implementation of BAT is best carried out using a phased approach that applies differently to tailings impoundments in various stages of their life cycle.

- *For existing tailings impoundments. Constructing filtered tailings facilities on existing conventional impoundments poses several technical hurdles. Chief among them is undrained shear failure in the underlying saturated tailings, similar to what caused the Mount Polley* *incident. Attempting to retrofit existing conventional tailings impoundments is therefore not recommended, with reliance instead on best practices during their remaining active life.*

- *For new tailings facilities. BAT should be actively encouraged for new tailings facilities at existing and proposed mines. Safety attributes should be evaluated separately from economic considerations, and cost should not be the determining factor.*
- *For closure. BAT principles should be applied to closure of active impoundments so that they are progressively removed from the inventory by attrition. Where applicable, alternatives to water covers should be aggressively pursued.*

Interpreting, the Panel is saying:

❖ For existing impoundments – apply Best Applicable Practices (discussed below)

❖ For new TSFs, the recommended direction is clear – dry tailings, underground tailings disposal, or other non-wet alternatives. This raises the question of how to treat mines that are already in the proposal process, but which have not yet received regulatory approval. Reason would dictate that since "… *cost should not be the determining factor"* all new impoundments should be dry. But unfortunately, economic considerations are still the strongest driving factor in any mine proposal. This is probably the most cogent issue associated with Panel's observation that "*The Panel firmly rejects any notion that business as usual can continue*." The Panel is saying safety, not cost, should be the determining factor in waste impoundment design.

❖ For closure of existing impoundments – for existing impoundments, all closure plans should be for dry closure, not for water covers, even if this means increased and perpetual water treatment.

Best Applicable Practices (BAP)

Best Available Practices are more complex and detailed than Best Available Technologies. The Panel describes the situation thusly:

> *The safety of any dam, water or tailings, relies on multiple levels of defence. The Panel was disconcerted to find that, while the Mount Polley Tailings Dam failed because of an undetected weakness in the foundation, it could have failed by overtopping, which it almost did in May 2014. Or it could have failed by internal erosion, for which some evidence was discovered. Clearly, multiple failure modes were in progress, and they differed mainly in how far they had progressed down their respective failure pathways.*

The Panel makes a number of detailed recommendation for BAP that would impact dam designers, mine operators, and regulators. The BAP recommendation of most note is to implement Independent Tailings Review Boards (ITRB) for all large tailings dams, and that the effectiveness of an ITRB depends on the following:

- *That it not be used exclusively as a means for obtaining regulatory approval.*
- *That it not be used for transfer of corporate liability by requesting indemnification from Board members.*
- *That it be free from external influence or conflict of interest.*
- *That there be means to assure that its recommendations are acted upon.*

The Panel believes that it is essential that the reports of the ITRB "... *go to senior corporate management and Regulators.*" The Panel does not include the public as one if its suggested parties to be informed. Whether this is an intentional omission, or whether the Panel assumed that since the reports would go to regulators they would then become public records, is not clear.

The Panel made a number of very insightful observations on Best Available Practices, including:

> *The Panel anticipates that this* (adopting guidelines) *will result in more prescriptive requirements for site investigation, failure mode recognition, selection of design properties, and specification of factors of safety.*

Here the Panel is saying that more prescriptive requirements are needed to provide guidance to tailings impoundment designers and operators. This is not a recommendation that says 'less regulation,' or 'self-regulation', but a recommendation that clearly says more 'guidance' is needed from regulators.

With a broader view, the Panel also noted:

> *… future BAP require considerations that go beyond stability calculations. It is important that safety be enhanced by providing for robust outcomes in dam design, construction and operations.*

By focusing on "…*providing for robust outcomes in dam design, construction and operations.*" the Panel is saying that tailings dam design and operation must do more than just provide "*stability calculations*". Here the Panel is again demonstrating its focus on safety (in placing emphasis on determining *robust outcomes*) over cost (merely focusing on *stability calculations* for the structures that the project can afford).

The Panel notes that in its 'revised costing' approach

> *The chief reason for the limited industry adoption of filtered tailings to date is economic. Comparisons of capital and operating costs alone invariably favour conventional methods. But this takes a limited view. Cost estimates for conventional tailings dams do not include the risk costs, either direct or indirect, associated with failure potential. ... Nor do standard costing procedures consider externalities, like added costs that accrue to the industry as a whole, some of them difficult or impossible to quantify. Full consideration of life cycle costs including closure, environmental liabilities, and other externalities will provide a more complete economic picture. While economic factors cannot be neglected, neither can they continue to pre-empt best technology.*

If "*business as usual*" is to change, then a goal of zero failures which places a priority on conservative assumptions in dam/disposal design must take precedence. Safety in operation must take priority over mine production. From a project standpoint waste disposal costs must be driven by safety considerations, not by 'what the project can afford'.

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[MPARS] DNR Request for Comments - Dam Safety - Construction - St. Louis County - Applications 2016-1383 and 2016-1380

2016-1383

Fisheries' concern about the Hydrometallurgical Facility (HRF) is because it will hold waste from the metallurgical plant and will be dependent on a liner underneath and a wick drain system. This seems like it would function appropriately during operation of the mine and certainly our engineers and hydrologists have this covered for the review.

My concern is about far into the future. How long does such a liner last and what happens when it inevitably degrades as nothing lasts forever? Even if it takes 200 years, the waste will still be there and in its location would be very susceptible to leaching into nearby wetlands and groundwater. There is no mention of the expected longevity of the liner and leakage system in the long term closure description. There is mention of a monitoring plan but no mention of how the liner could be maintained or repaired or replaced. (Section 7.3, Residue Management Plan v.5). I don't understand how a liner could be replaced, or even repaired, under a 97 acre site with 50 feet of fill on top. The site is to be capped for long-term closure, but I don’t know if that means there would be no leaching concerns long-term. I would think not as long as this location has groundwater movement. The Hydrometallurgical Residue Facility is a concern to Fisheries because of its potential impact on water quality as the system ages - on the very long term scale.

2016-1380

The permit for reinforcing and building up the legacy tailings pits is less concerning to fisheries and water quality concerns, I think because the waste being discharged to them would be more dilute and of a different composition. In addition these pits have a storage history and they are designed with a treatment system in place to catch leachate. They are not dependent on a manufactured liner either. As long as our engineers and hydrologists have confidence in the design, I would hope fisheries and water quality are protected.

Edie Evarts | Area Fisheries Supervisor

MN Department of Natural Resources

650 Highway 169, Tower, MN 55790

To: Dostert, Dana M (DNR)[dana.dostert@state.mn.us]
From: Boyle, Jason (DNR)
Sent: Fri 10/1/2010 2:05:56 PM
Subject: RE_ Slope Stability Analysis.pdf
Polymet geotechnical factor of safety.msg

From:
To:
Boyle, Jason (DNR)
Subject:
Date:
Attachments:
Dostert, Dana M (DNR)
RE: Slope Stability Analysis
Friday, October 01, 2010 1:05:56 PM
Polymet geotechnical factor of safety.msg
Let's discuss Polymet today when you get back.
I'll be gone Monday and Stuart wants to meet on Monday regarding how much detail we need in the EIS.
I thought we were counting on the rock buttress as the means to add stability and increase the factors of safety should the factors of safety during construction be less than anticipated, since we aren't that crazy about the conservatism of the design to date. (see attached email)
Why is the Corps now involved?
Jason
-----Original Message----From:
Schwanz, Neil T MVP [mailto:Neil.T.Schwanz@usace.army.mil]
Sent: Friday, October 01, 2010 11:07 AM
To: Dostert, Dana M (DNR)
Cc: Boyle, Jason (DNR); al.trippel@erm.com; Ahlness, Jon K MVP; Augustin, Ralph J MVP
Subject: RE: Slope Stability Analysis
Hi Dana,
I understand your concern and I'm not an expert on tailings dam designs. I'm pretty comfortable with stability analyses and shear strength of soils though but lack the actual experience with tailings materials so I really need to get better versed in these properties. USACE doesn't have any criteria for tailings dams so I look at this in the context of our embankment dams.
According to ER 1110-2-1806, "Earthquake Design and Evaluation for Civil Works Projects," I expect Polymet would fall into either the Significant or High hazard category (see hazard table attached).
Minnesota and Wisconsin are in Seismic Zone 0 of the Uniform Building Code but all areas have some seismic potential so we probably wouldn't think that 0 is the right answer for us on a potential high hazard dam. I'm used to thinking in terms of seismic coefficients and the 1991 USGS map of 5% damped,
0.3 second pseudo-acceleration response for 50-yr, 10% chance of exceedance shows acceleration between 0.02g and 0.05g for where I spotted Hoyt Lakes.
For a 250-yr occurrence this increases to about 0.15g. In terms of effect on typical projects I'm familiar with, this is pretty minimal but for these liquefiable materials this may very likely not be the case. I expect there are some checks we can do to verify liquefaction potential for these soils under this response. But even if analyses would show no liquefaction potential under seismic loading we still would consider static liquefaction.
In terms of static loading a 14-in rainfall would add a uniform surcharge of
73 psf and we probably know that loading would not be uniform so some areas would see greater loads.
I haven't seen the borings yet but you mention existing tailings as being in a semi-liquid state so I'm thinking of these existing slimes (silts/clays?) as being under-consolidated with a loose grain structure and possessing some undrained shear strength under static conditions but with time would consolidate further and gain additional strength. This is unlike a viscous slurry that would flow if not confined. The coarse tailings might be fully drained but still in a very loose grain arrangement that may collapse upon

some triggering event. Does this seem right?
I'm expecting that we won't get everything we need from the Barr stability report to make our decisions

on this issue so we might consider sitting down with Barr in your office and go over the seismic analyses in some detail. I don't know if you've reviewed the report yet but I haven't seen it other than glancing through Tom's copy at the meeting. In the meantime I'll try to get a little more info from our folks.
Regards,
Neil
-----Original Message----From:
Dostert, Dana M (DNR) [mailto:Dana.Dostert@state.mn.us]
Sent: Friday, October 01, 2010 9:05 AM
To: Schwanz, Neil T MVP
Cc: Boyle, Jason (DNR); al.trippel@erm.com
Subject: RE: Slope Stability Analysis
Neil,
Thanks for the response. I am over my head in this area, so any guidance and analysis you can provide is greatly appreciated. As I stated in the meeting, Minnesota experiences a magnitude 4 to 4.5 earthquake about every 20 years.
With Polymet being so close to the Wisconsin, Michigan and Ontario borders, and rock more characteristic of those states, I will also look into earthquake frequency of those states.
It is my understanding that the LTV tailings at depth are already in a semi liquefied state. I remain concerned that if another 150 feet of tailings are added, and then experience a major precipitation event such as last week's 14 inches in south central Minnesota, the loading will lead to liquefaction and an embankment failure. With EPA requiring cleanup of any spilled tailings, the cost to Polymet or the State of Minnesota would be in the tens of Millions of dollars. Further complicating this issue is the use of upstream construction, which I don't think is a good method for a dam that is required to last for centuries.
Thank you very much!
Dana Dostert PE, PG
Senior Engineer - Dam Safety
MN Department of Natural Resources
500 Lafayette Road
St. Paul, MN 55155-4032
(651)-259-5663
dana.dostert@state.mn.us

From: Schwanz, Neil T MVP [mailto:Neil.T.Schwanz@usace.army.mil]
Sent: Friday, October 01, 2010 7:44 AM
To: Ahlness, Jon K MVP
Cc: Augustin, Ralph J MVP; Jorgenson, Terrance L MVP; Dostert, Dana M (DNR)
Subject: Slope Stability Analysis
Hi Jon,
It was nice of Tom to share this but it's really too basic to address the stability concerns at Polymet.
Steve Hoffman indicated that we (DNR and
USACE) need to be in agreement on slope stability criteria. My understanding is that this needs to satisfy State criteria but as a co-lead agency USACE needs to be happy as well. As mentioned during the meeting in Hoyt Lakes the criteria for the effective stress and total stress analyses are in accordance with USACE criteria for dams but I really need to get a copy of the stability report to see how the analyses were completed to say they were performed in accordance with Corps procedures (factors of safety are tied to method of analysis, sampling, testing, selection of shear strength). As described in the meeting the controlling load case is the static liquefaction. I have not yet found current (there is old) Corps criteria for earthquake analyses using a seismic coefficient method (we seem to have migrated to more rigorous methods where we look at displacement), but will check with others.
Dana questioned the use of the FS=1.05 for liquefied soils. Rightfully so, t <<image001.jpg>> his factor is so close to instability that minor error in selection of strength could lead to failure. In that

regard it was suggested we try to better understand the conservatism inherent in the analyses performed. Again we need the report to start to assess this point. One note on this point is that the old criteria for earthquake loading required a minimum FS=1.0. I did find one paper that mentioned that rainfall could initiate liquefaction but it was mentioned along with other initiating events and not discussed in detail. I can only assume the writers were thinking that a large rainfall provides a load over a significant part of the tailings surface that might cause some deformation and shear that would lead to collapse of the tailings grain structure initiating liquefaction. If the non-liquefied static stability is high enough I'm really not sure there would be enough displacement for this to happen but again I'll try to get more info on this from our earthquake guys at ERDC.
I did find a calculator for computing flow extent for tailing assuming a breach on the internet but haven't dug into this at all. I really don't have a lot of time for review but believe we should spend a couple of days digging into stability questions starting from the Barr stability report. Can you get a copy of this report?
Thanks,
Neil
-----Original Message----From:
Tom Radue [mailto:TRadue@barr.com]
Sent: Thursday, September 30, 2010 8:58 AM
To: 'dana.dostert@dnr.state.mn.us'; Schwanz, Neil T MVP
Cc: 'Jason.Boyle@dnr.state.mn.us'
Subject: Slope Stability Analysis
Dana and Neil - it occurred to me after our meeting in Hoyt Lakes last week that Jed Greenwood and Bethany Erfourth of Barr had previously developed a Soil Mechanics and Slope Stability Analysis Primer document for use in a short course a number of years ago. The Primer represents Jed's and Bethany's attempt to compile information and concepts normally taught over a number of college semesters and routinely contained in a variety of soil mechanics and geotechnical engineering textbooks. I know that the content of the Primer will be mostly familiar to each of you but thought a copy may be of some value as you work on various projects in the future that involve slope stability analysis.

□If you have an opportunity to review or use the attached document and develop any questions along the way, please feel free to contact me and I'll do the best I can to provide follow-up information.
Best regards,
Tom
Tom Radue
Vice President
Senior Geotechnical Engineer
Minneapolis office: 952.832.2871
cell: 952.240.4051
tradue@barr.com <mailto:tradue@barr.com> www.barr.com <http://www.barr.com>
resourceful. naturally. <http://www.barr.com/>

Donald G. Sutton P.E.
Senior Principal Mining Engineer

Areas of Expertise

- Coal Contract Negotiations
- Mine Design & Optimization
- Mine Auditing & Economics
- Mine Reclamation/Remediation
- Computer Modeling & Programming (Survcadd, ArcView, Excel)
- Arbitration/Mediation

Education

B. Ap. Science Mining and Mineral Processing, University of British Columbia (1969)

2 years postgraduate studies at Penn State in Mineral Engineering Management (similar to MBA)

Professional History

Professional Engineer in Montana

1984 – Present. Senior Principal Engineer
Spectrum Engineering and Environmental LLC, Billings, MT
1984 – 2015. President
Spectrum Engineering Inc., Billings, MT
1978 – 1984. Director of Engineering
Billings office of IntraSearch
1974 – 1978. Manager of Exploration
Consolidation Coal (Alberta and Pennsylvania)
1973 – 1974. Penn State Graduate School
1969 – 1973. Mine Engineer
Cominco (Northwest Territories and British Columbia)

BACKGROUND

Mr. Sutton's career includes a broad experience in mining, mine reclamation, mine and smelter environmental remediation, economics, accounting, exploration, permitting, management consulting, commercial contract negotiation and litigation support.

After graduation, he held a variety of positions at Cominco's open pit lead-zinc mines in Pine Point, NWT, such as mine planning, maintenance planning, pit shift boss, and mill foreman. His experience with Cominco included engineering and contract administration in underground lead and zinc mines, designing a large open pit copper mine, implementing a computerized mine accounting and MIS system, drilling and blasting engineer at Fording Coal, and assistant to the vice president of operations. He also developed a computer program for the lead and zinc smelter in Trail, BC to process custom ore.

He returned to graduate school at Penn State where he enjoyed 2 years studying management, accounting, finance and advanced coal mining techniques. As a graduate assistant, he helped write a computer program for modeling coal geology, and taught surveying to undergraduates.

BACKGROUND (cont.)

After graduate school, he worked for Consolidation Coal in a variety of positions that focused primarily on exploration, property management, development of new properties, and geology and rock mechanics problems at existing mines. He was manager of exploration for many of the eastern states when he decided to become a consultant in 1978, and moved to Billings to assist in the coal mining development of the Powder River Basin (PRB), which is now the dominant coal mining area in the USA.

As a consultant, he managed the design and permitting of several PRB coal mines, during the period when new coal mine reclamation laws and regulations were being established. He has continued to help his clients in the areas of coal supply agreements, coal contract negotiation, mine permitting, and mine design and reclamation. This work involves considerable economic and 3-D geological modeling, and has involved him in a variety of commercial litigations and arbitrations. He has acted as an arbitrator and mediator for some of these cases.

Spectrum Engineering's expertise in mine reclamation and cost accounting led to our involvement in some superfund remediation, and eventually in taking over some bankrupt gold mines and managing the reclamation and environmental remediation for agencies such as the BLM, EPA, and the State of Montana DEQ.

REPRESENTATIVE EXPERIENCE

Mine Design

Mr. Sutton worked as a miner and a mine planning engineer in underground and open pit hardrock and surface coal mines before becoming a consultant. As a consultant, he has designed a variety of open pit coal and phosphate mines, and fine tuned or optimized many others. For example:

- Mr. Sutton oversaw the design and equipment selection of 3 open pit phosphate mines in southern Israel for Rotem Amfert Negev. He changed the maintenance system and the mining methods. Mr. Sutton trained the mine managers, engineers, and production staff. This involved trips to the mines, and bringing the managers to the USA to learn how maintenance, safety, production, et cetera are practiced here.
- He designed and permitted the North Antelope and Rochelle Coal Mines in the Southern Powder River Basin for Powder River Coal (Peabody). Tasks included long range plans, optimizing the overburden placement, short term plans, PMT design, and mining methods analysis.
- He prepared geologic models, mine plans, and detailed economics for an oil shale mine in southern Israel for PAMA. Then, he prepared detailed short term development plans for the 4 million tpy pilot plant.

Reclamation Design

Mr. Sutton has helped many clients save millions of dollars by showing them how to reduce their coal mine reclamation costs by intelligently optimizing and coordinating the mining and reclamation activities. He has successfully performed this work for mining companies and on behalf of utilities with cost plus contracts. Clients include:

- Powder River Coal, North Antelope and Rochelle Mines
- Peabody Coal, Big Sky Mine

REPRESENTATIVE EXPERIENCE (cont.)

- Detroit Edison, East and West Decker Mines
- Commonwealth Edison, Decker, Big Horn, and Black Butte Mines
- Northern States Power, Absaloka and Rosebud Mines

Operational Audits

- Southern California Edison, Black Mesa Mine
- Idaho Power, Jim Bridger Mine
- Northern States Power, Absaloka and Rosebud Mines
- Northern Indiana Public Service, Carbon Co. Coal, Medicine Bow Mine
- El Cerrejon coal mine in Columbia. Reviewed operational practices and feasibility of using small dragline for Exxon
- Arizona Public Service, Four Corners Mines

Fuel Supply Auditing

Mr. Sutton helps utilities ensure that they are paying the correct amount for their fuel, and that the contracts prices are being properly computed. He uses his experience in mine accounting, auditing and mining operations to ensure that all costs are properly booked, and that pass through costs are reasonable and fair. He has performed this work as an advocate for the utilities, and as a arbitrator. His goal is to be fair and to try to establish a win/win outcome.

- Southern California Edison, Black Mesa Mine
- Northern States Power, Absaloka and Rosebud Mines
- Otter Tail Power, Beulah Mine
- Wisconsin P&L, Absaloka Mine
- Arizona Public Service, Navajo Mine
- For Detroit Edison, LCRA, and Commonwealth Edison, audited East and West Decker Mines
- For Commonwealth Edison, audited Big Horn and Black Butte Mines
- For Northern Indiana Public Service, audited Medicine Bow Mine Contract

Geology and Resource Evaluation

- As exploration manager for Consolidation Coal, managed several large coal exploration and property acquisition projects in Alberta, West Virginia, Ohio, and Pennsylvania.
- As a consultant, managed coal exploration projects in Montana for Wesco and Montco.
- Provided consultation, economic analysis, technical advice, audit services and testimony for the Lower Colorado River Authority in litigation with Decker Coal Company.
- Provided economic evaluation and testimony for plaintiff in a condemnation of some coal lands by a pipeline.
- Created complex geologic computer models of 3 multi-seam phosphate deposits in Israel. Trained the local engineers and geologists to perform geologic investigations and computer modeling.
- Created complex 3-D economic computer models of phosphate deposits for Rotem Amfert Negev to identify the most economic areas considering different types of fertilizer being produced, and the cost of sulfuric acid and other raw materials required to manufacture fertilizer. Models also included the mining costs, the % P_2O_5, the haul distance to the plant, and the acid consumption of each type of phosphate.

REPRESENTATIVE EXPERIENCE (cont.)

- Created complex 3-D model of an underground coal mine and associated mine fire in Utah.
- As a consultant, evaluated numerous coal reserves and resources for confidential utility clients. The tasks involved ensuring that the bidders could prove that they could produce the quantity and quality of coal represented in long term coal supply agreement proposals.

Abandoned Mine Land Reclamation

- Since 1984, Spectrum Engineering has worked for the States of Montana, Wyoming, and Utah inventorying, designing the closures, and, in many cases, supervising the reclamation of the abandoned mine sites. Mr. Sutton completed the engineering and project management on some of these sites.
- Provided detailed engineering for the Upper Blackfoot Mike Horse Mine remediation project.

Superfund Remediation

Mr. Sutton provided engineering design services and economic evaluations on portions of:

- The Bunker Hill Superfund site in Kellogg, Idaho. Spectrum Engineering provided engineering services to the State of Idaho DEQ as a subcontractor to Terragraphics Environmental Engineering. Work included determining methods to remove mine tailings from the South Fork of the Coeur D'Alene River and disposing them in a repository.
- Silver Bow Creek Operable Unit portion of the Butte Anaconda Superfund Site. Retained by ARCO to provide construction engineering oversight (prior to Montana DEQ involvement) and economic evaluations for the removal of mine tailings from over 20 miles of stream channel.
- Acid Brook Superfund Site, Rariton River, NJ for DuPont. Prepared computer models and plans for removing metal contamination and rebuilding the river.
- Prepared an Engineering Evaluation and Cost Analysis for the BLM regarding mine reclamation and water treatment issues for the Zortman and Landusky Mines in Montana.

Management Consulting/Litigation Support

Mr. Sutton has assisted clients in cases involving long term coal supply agreement disputes, construction disputes and property valuation disputes. He provided technical expertise, discovery expertise, economic evaluations and strategy development.

- Testified on behalf of the U.S. Army Corps of Engineers in a construction dispute involving the excavation of the Tom Bigbee Waterway.
- Provided consultation, economic analysis, discovery advice and testimony for Commonwealth Edison in several cases involving force majeure and other economic issues with a variety of long term coal contracts.
- Provided consultation, economic analysis, technical advice, audit services and testimony for Lower Colorado River Authority in litigation with Decker Coal Company.
- Provided economic evaluation and testimony for plaintiff in a condemnation of some coal lands by a pipeline company.

TRAINING AND CONTINUING EDUCATION

- OSHA 29CFR1910.120 40-hour hazardous material handling training (HazMat) and yearly 8-hour refresher training.
- OSHA 29CFR1910.120 8-hour OSHA supervisor training.
- OSHA 10-Hour Outreach Training Program - Construction (360training.com)
- MSHA 32-hour safety training and yearly 8-hour refresher training.
- Land Reclamation, Geomorphology.
- Managerial and Financial Accounting.
- Computer Modeling (geology, mining, GIS).
- Corporate Ethics Plan, Code of Business Conduct, Training.
- Harassment Prevention Training.

AFFILIATIONS

Society of Mining Engineers

Message

From:	Arkley, Stuart (DNR) [/O=STATE OF MN/OU=EXCHANGE ADMINISTRATIVE GROUP (FYDIBOHF23SPDLT)/CN=RECIPIENTS/CN=STUART.ARKLEY]
Sent:	6/15/2011 2:09:22 PM
To:	Donald Sutton [sutton@spectrum-eng.com]
CC:	Engstrom, Jennifer N (DNR) [jennifer.engstrom@state.mn.us]; 'Al Trippel' [Al.Trippel@erm.com]; Johnson, Bill H (DNR) [bill.johnson@state.mn.us]; Boyle, Jason (DNR) [jason.boyle@state.mn.us]
Subject:	RE: followup to Geotechnical Work Plan review meeting

Don,
Thanks for your comments. After conferring with Jennifer and others yesterday we will be proceeding with approving the Barr work plan, recalling that the work plan is intended to capture a path forward designed through the Impact Assessment Planning (IAP) process and through numerous conversations involving USEPA, Barr and Knight Piesold experts.

We do not want to lose sight of the questions you have raised, which I suspect are largely a mix of questions we have already considered and made decisions upon, and questions that we have traditionally thought of as being more at the "permitting level of detail" (the distinction from permitting is not as clear as it once was so this input is important now, and will continue to be). The DNR, primarily through Jennifer, will work with you to get up to speed on past Lead Agency decisions and to address your questions and concerns. The goal is to do this as efficiently as possible so that your input can be incorporated in Barr's modeling efforts, where appropriate.

Note that the work plan is not intended to cover all issues – it focuses on those raised in the IAP process. Similarly we have found in the geochemistry IAP / work plan team that there are both IAP and separate, permitting-focused questions that need to be addressed in the next round of modeling. Our goal is to work closely with Barr while the modeling moves forward, rather than approving a work plan and waiting for results. This is a dynamic process that allows for some tailoring to address new questions and/or to understand PolyMet's construction design at a deeper level.

(Jennifer – I hope this adequately reflects what we decided yesterday. Let me know if any clarifications are needed).

Stuart

From: Donald Sutton [mailto:sutton@spectrum-eng.com]
Sent: Wednesday, June 15, 2011 12:48 PM
To: Engstrom, Jennifer N (DNR)
Cc: 'Greg Graske'; 'Beth Nixon'; Arkley, Stuart (DNR); Johnson, Bill H (DNR); Boyle, Jason (DNR); Dostert, Dana M (DNR)
Subject: RE: followup to Geotechnical Work Plan review meeting

Jennifer:
My impression is that Barr is adequately addressing the classical geotechnical issues in the work plan, however, the potential for foundation creep failure concerns me. I am not an expert in predicting long term creep in soils or peat, so this is a subject that I would like Barr and Knight Piesold to address sufficiently to mitigate my concern. Does the peat layer need to be excavated from beneath the buttress? What about long term creep in the interior under the PolyMet embankments?
I had some concerns about the long term weathering of the waste rock. When acid conditions exist, we have seen some rock types (usually high feldspar content) rapidly weather into sand and clay, totally changing the geotechnical properties. I discussed this with Shannon Shaw. Based on the rock mineralogy, she didn't think this should be a concern here. Has anyone looked into the nearby old waste dumps to observe their condition?

The long term water management uncertainties concern me because I don't know what they are yet.

I don't have any additional comments on the proposed geotechnical work plan. I do however have some concerns about the overall concept and the long-term liability issues.

Apparently the peat layer in the foundation is weak and was the source of the block style failure modeled on section F. The planned remedy is to buttress the slopes with mine waste and to use mine waste to build upstream embankments on top of the saturated tailings. Manipulating this geometry will probably increase the factor of safety to at least 1.5 in the short term, but my concern is the long-term stability of the entire system. Over a period of several decades or centuries will there be any creep in the peat layer in the foundation? If creep does occur how does this affect the long-term stability of a water body on top of saturated (weak) tailings? How is the geotechnical stability affected in the long term if any of the water management systems don't perform exactly as designed 50 or 500 years later? How sensitive is the long-term geotechnical stability to long-term erosion on the slopes? Will seeps or piping ever become an issue and how will they be repaired? What systems will be operational 100 or 500 years from now to repair erosion on the embankment slopes? I think this design will require perpetual long-term maintenance. Who will do this? Does the design need to be even more robust to accommodate long term uncertainties that can't be easily modeled today?

A significant percentage of mine tailings dam failures are caused by water problems and/or poor long-term operating and management practices. These are not necessarily geotechnical design issues, but they will affect long-term liability for the state of Minnesota and need to be considered. How sensitive is the stability if different components of the water management fail or don't function properly? I haven't found the details yet, but water management will be important. Would the failure risks be reduced if redundant water management systems are constructed?

If there is an embankment failure, say 25 or 100 years after mining ceases, what would the consequences be? How much would the cleanup cost? What type of repairs would be necessary to prevent this from occurring again? How much money must be placed in a fund today to cover these indefinite contingencies? How does this cost compare with drying the tailings before they are placed in a more stable repository, or building a different type of containment system? How do we really quantify the risk and who should bear the cost of the risk? Is 1.5 factor of safety appropriate for this type of facility that must remain functional forever? Have all the possible failure modes and mechanisms been addressed? How sensitive is the safety factor to the geotechnical data range for each type of material? How sensitive is the stability to construction management ensuring that the embankment material meets specifications and that the compaction is correct. How will this be scheduled? Will waste rock come directly from the mine or will it be rehandled from the dumps? What type of testing will be performed during construction? Can snow or ice be incorporated into the embankments? Can compaction be obtained during freezing conditions? There a numerous construction details that need to be considered here.

PolyMet is proposing to build the tailings disposal system that has the lowest initial cost, but has more long term risks than other tailings disposal methods. There is risk associated with perching a lake full of saturated tailings on top of the existing tailings. It is difficult to quantify the probability of failure over a long time frame, but I think you can consider the consequences of a failure and estimate the cost of cleaning up the failure, and then add the cost of operating the repaired facility forever. This cost can be compared to the additional cost of building a more stable facility initially. The choice of tailing storage method may be related to the size of the fund that must be set aside to pay for O&M and possible failure. The next step will be determining how much money PolyMet must set aside for this, and whether the state will accept a bond (perpetual) or real money. In my opinion, the state needs money in a trust account, and the amount of money in the account needs to adequately protect the state from the risk and place the risk on PolyMet.

I am going to be out of touch most of Thursday. If you need to contact me, try my cell phone (406-670-7270).

Donald Sutton P.E
Spectrum Engineering
1413 4th Ave N
Billings, MT 59101
406-534-4660

From: Engstrom, Jennifer N (DNR) [mailto:jennifer.engstrom@state.mn.us]
Sent: Tuesday, June 14, 2011 8:00 PM
To: Donald Sutton
Cc: Greg Graske; Beth Nixon; Arkley, Stuart (DNR); Johnson, Bill H (DNR); Boyle, Jason (DNR); Dostert, Dana M (DNR)
Subject: FW: followup to Geotechnical Work Plan review meeting

Don,

Folks are planning to submit comments tomorrow on the geotech workplan for Barr (PolyMet's consultant) to start the design and evaluation of the basin, stockpiles and hydromet facility.

I wanted to see if you had developed any comments at this point. I know that you are working your way through the documents that we had forwarded along. But I did want to check in and see where you are.

I am on the road and in an offsite meeting all day tomorrow. If you think you want to submit comments, let the group know (I am not expecting email access tomorrow) and we may have to send them in after tomorrow. If not, let them know that as well.

Thanks,
Jennifer

Please don't print this e-mail unless you really need to.

From: Boyle, Jason (DNR)
Sent: Tuesday, June 14, 2011 4:12 PM
To: Al Trippel; Dostert, Dana M (DNR); Engstrom, Jennifer N (DNR); Arkley, Stuart (DNR); Johnson, Bill H (DNR)
Subject: RE: followup to Geotechnical Work Plan review meeting

I heard from Dana and Stuart and made those changes to the document (attached). I will send to Barr at close of business tomorrow, Wednesday June 15.

Thanks,
Jason

From: Boyle, Jason (DNR)
Sent: Thursday, June 09, 2011 1:24 PM
To: 'Al Trippel'; Dostert, Dana M (DNR); Engstrom, Jennifer N (DNR); Arkley, Stuart (DNR); Johnson, Bill H (DNR); 'sutton@spectrum-eng.com'
Subject: followup to Geotechnical Work Plan review meeting

I have attempted to incorporate the comments we discussed at our meeting on June 8. Please review these comments for accuracy and reply with any additions or changes by June 14. I will then send our comments to Barr, they will incorporate the comments, and then we will finalize the Work Plan.

ps
The loading rate is addressed in Section 4.4 of the Preliminary Geotechnical Evaluation – March 2009.

Jason Boyle

This document is made available electronically by the Minnesota Legislative Reference Library as part of an ongoing digital archiving project. http://www.leg.state.mn.us/lrl/sonar/sonar.asp

MINNESOTA DEPARTMENT OF NATURAL RESOURCES

Nonferrous Metallic Mineral Mineland Reclamation Rules

STATEMENT OF NEED

INTRODUCTION

Minnesota Statutes, sections 93.44 to 93.51, entitled the Mineland Reclamation Act, authorize the commissioner of Natural Resources to adopt rules providing for the reclamation of lands disturbed by the mining of metallic minerals and peat. The law declares that it is the policy of the state, through mineland reclamation, to control the adverse environmental effects of mining, to preserve natural resources, and to encourage land use planning. The law further declares, as policy, the promotion of the orderly development of mining, the encouragement of good mining practices, and the recognition and identification of the beneficial aspects of mining.

The Mineland Reclamation Act, originally passed in 1969, represented a legislative response to Minnesota's nearly 100 year old iron mining industry (the only metallic mineral mining that had occurred in Minnesota). That industry has had enormous economic, social, and environmental impacts on this state and its residents. Because of these impacts, it was determined that the iron mining industry, and any other metallic mineral mining industry that might possibly develop, were in need of regulation. The non-metallic mineral resources, such as, sand, gravel, dimension stone, clay, and other industrial minerals, that had mining histories equally as long as that of iron, were viewed differently, because these were intentionally placed outside the scope of the law, by the legislature. Since the original passage of the Act, the only broadening of the scope was in 1983 when peat was included, by amendment of the statutes.

As a result of the original passage the Act, and the subsequent amendment, a set of metallic mineral reclamation rules (dealing exclusively with iron ore and taconite) was promulgated in 1980, followed by peat reclamation rules in 1985. Coincident with each of these events, a resource-specific reclamation program was developed by the department to deal with permitting issues.

Although the actual permitting process did not begin until 1980, the department had an extensive reclamation research program in place since well before that date. As early as 1975, the department was strongly focused on reclamation research. That research has been extensive and very divergent, including:

1) studies associated exclusively with iron ore and taconite reclamation, such as sloping, erosion control, and revegetation;

2) peatland resource studies that eventually led to the inclusion of peat into the Reclamation Act, and ultimately to the development of rules; and

3) studies associated with nonferrous metallic mineral characterization, drainage water evaluation, and the analysis and development of specialized nonferrous reclamation techniques and practices, that are in part the basis for these proposed rules.

Currently, as in 1969, the only metallic mineral mining that has been conducted, or that is currently proposed for development, in Minnesota, is related to iron. However, over the last three decades, a considerable annual investment of money and effort has been put into the exploration for nonferrous metallic minerals. Although there is presently no reason to believe that nonferrous mining is imminent, there has been a desire on the part of the department, the mineral industry, and the environmental community, to develop regulations so there can be a clear understanding of the obligations associated with nonferrous mineral development, should it occur in Minnesota.

The Mineland Reclamation Act directs the Commissioner of Natural Resources to conduct a comprehensive study and survey to determine the extent to which regulation of mining areas is necessary and in the interest of the general welfare. Upon conducting this study and survey, the department has identified the major

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problems associated with nonferrous mining. The rules have been prepared by the department to specifically address these problems. The implementation of these rules will provide for the reclamation of lands disturbed by nonferrous mining activities, returning them to a safe, productive, and environmentally sound condition.

The Minnesota Administrative Procedures Act and other associated statutes, direct the commissioner to consider a number of issues during the process of rule development and adoption. Three such issues have been considered by the department during this rulemaking process. These include:

1) impact of the rules on small businesses, as required by Minn. Stat., sec. 14.115;

2) impact of the rules on expenditure of public money by local public bodies, as required by Minn. Stat., sec 14.11, subd. 1; and

3) impact of the rules on agricultural land, as required by Minn. Stat., sec. 14.11, subd. 2, and secs. 17.80 to 17.84.

Other statutes that are specifically identified in the Administrative Procedures Act, including: Minn. Stat., sec. 115.43, subd. 1; Minn. Stat., sec. 116.07, subd.6; and Minn. Stat., sec. 144A.29, subd. 4, were reviewed and determined to be not applicable to the proposed rule.

The Department of Natural Resources has determined that complying with the provisions of Minn. Stat., sec. 14.115, relating to methods for reducing the impact of the rule on small businesses, would be contrary to the statutory objectives that are the basis for the proposed rulemaking. Based on site visits, conducted by the department, to nonferrous metallic mining operations located outside Minnesota, and discussions with regulators in other states and Canada, the department has determined that the potential for environmental concern is independent of the size of the business conducting the mining operation. Regulators from other states have reported numerous environmental problems at small, often under-capitalized, mining operations. These regulators have further indicated that it is often more difficult to resolve problems when dealing with small businesses than when deal with large businesses. If such situations were to occur in Minnesota, the intent of the reclamation statute, that requires the maintenance of satisfactory environmental control, could be defeated if the rules incorporate less stringent compliance standards or schedules that would lessen the requirements for mining operations conducted by small businesses.

Minnesota Statutes, section 14.115 specifically requires the commissioner to address the following:

1) **the establishment of less stringent compliance or reporting requirements for small businesses** - The commissioner has, in part, based the reclamation requirements on the extent of efforts necessary to protect natural resources. The extent of reclamation that may be necessary at a given site has no direct relationship with the size of the business conducting the mining operation, therefore the commissioner has not proposed less stringent compliance standards for small businesses. The commissioner requires the submission of an annual report, that coincides with a statutory requirement of an annual review of an operator's financial assurance, and also requires immediate reporting, in the event the permittee determines there is a permit violation. The commissioner finds both these reporting requirements to be necessary and reasonable, and is therefore not proposing a less stringent requirement.

2) **the establishment of less stringent schedules or deadlines for compliance or reporting requirements for small businesses** - The commissioner requires that the permittee be in compliance at all times, in order, in part, to protect natural resources. In the event that a permittee violates the permit, the commissioner will determine the allowable deadline for reestablishing compliance, based on the extent of the violation and the reasonable time necessary to correct it. These decisions are not dependent on the size of the business, and therefore the commissioner is not proposing a less stringent standard.

3) **the consolidation or simplification of compliance or reporting requirements for small businesses** - The commissioner has already taken steps to accomplish this for all permittees, regardless of size.

4) **the establishment of performance standards for small businesses to replace design or operational standards required in the rule** - The commissioner's rules are already basically established based upon performance standards. Each site and specific operation, without regard to size, will be analyzed individually to determine the extent of regulation needed to protect natural resources.

5) **the exemption of small businesses from any or all requirements of the rule** - If the commissioner

were to implement either of these options the intent of the reclamation statute, that requires the maintenance of satisfactory environmental control, could be defeated.

Minnesota Statutes, section 14.115 also requires the commissioner to make efforts to include small businesses in the rulemaking process. This has been accomplished for the proposed rule by:

1) distributing and taking comment on three separate major drafts of the proposed rules;

2) conducting group meetings with nonferrous metallic mineral explorers and miners;

3) the publication of notice of the proposed rule in Skillings' Mining Review, a minerals trade publication; and

4) direct notification of nonferrous metallic mineral explorers and miners.

In accordance with the provisions of Minn. Stat., sec. 14.11, subd. 1, the commissioner has concluded that adoption of the proposed rule is not anticipated to result in the significant expenditure of public money by local public bodies. The rules do establish the requirement that a public information meeting be held shortly after a potential mine operator decides to proceed with an application for a permit to mine. This meeting requirement does include the voluntary involvement of the local governmental units, whose involvement may merely extend to supplying a place to conduct the meeting and having local officials available to express local concerns. If a mining operation is ultimately conducted, the local involvement, associated with planning and zoning decisions that are currently the obligations of local public bodies, might be extensive. However, such obligations, and associated expenses, would not be a direct result of the adoption of the proposed reclamation rules.

In accordance with Minn. Stat., sec 14.11, subd. 2, the commissioner has determined that adoption of the proposed rule is not anticipated to have a direct and substantial adverse impact on agricultural lands in Minnesota. It is possible that at some future date a person may apply for a permit to mine that involves agricultural lands, however such decision is not a direct result of the adoption of these rules. In addition, because one of the purposes of these rules is to ensure that utility of the land is restored when mining is complete, these rules will help minimize any long-term adverse impacts on agricultural lands, that might possibly be affected by mining.

GENERAL PROVISIONS

6132.0100 Definitions

Subpart 1. This section contains two types of definitions; those terms which appear and are defined in the statute, and those terms which are used in the proposed rules, which may not be generally recognized and accepted, and to which special and specific meanings are attached for the purposes of specifically understanding the rules. It is reasonable to define any word that is meant to have a specific meaning, not normally associated with it, or when the word is a term of art used by a specific industry. When a word or term is used in the proposed rules, and does not appear in this section, it shall be assumed to have the definition that is found in commonly used dictionaries. Some commenters have asked that certain words, that were meant by the department to be used as they are commonly defined, be included in this section. In at least one instance when the department tried to comply with such a request, the revisor of statutes removed the word and its commonly used definition from this section.

Subp. 2. "Acceptable research" is defined to require approval by the commissioner, with its parameters dictated by the site and its measure of success based on the standards set forth in the rules. Because such research may be used to develop an alternative method of reclamation, it is reasonable to require that the research for an alternative method should be designed with the intention of striving to achieve the goals of these rules, which is the method followed by the commissioner in establishing the stated standards of the rules.

Subp. 3. "Adversely impact natural resources" is defined because it is one of the basic criteria used by the commissioner to establish several of the goal statements contained in the reclamation standards portion of the proposed rules. In addition, it is used as one of the factors for identifying generally acceptable locations for those mining facilities that have flexibility siting. As such, the definition is purposely devised to broadly and generally include any possible harmful effect on natural resources, that might potentially occur as a result of a mining operation, while recognizing that the effect is not likely to be specifically identifiable until an exact mine site is identified and the specific operating plans are selected and approved. Because the commissioner has, through numerous statutes, been given specific authority to protect the state's natural resources, it is reasonable that each of the department's rules, on any specific topic, should, at a minimum, give general considerations to impacts on all natural resources. This term is used in those parts of the rules that describe the basic philosophy of the rules, and as noted, for some general siting decisions. Because the term is used only to provide general guidance, it is reasonable that the definition be broad and general. The actual mandatory requirements of the rules, with which the permittee must comply, are necessarily very specific.

Subp. 4. "Auxiliary facilities" is defined to include every alteration to the natural environment associated with the mining operation because all such areas should be subject to reclamation. Common carrier facilities are not so included because their usefullness is not directly related to only a single mining operation nor will they necessarily terminate operations upon termination of the permit to mine.

Subp. 5. "Beneficiating plants" is defined in order to clarify that the applicability of these proposed rules extends to processing and fabrication facilities, and because the term is included in the definition of "Mining area" contained in the statute, and therefore relates to the scope of the rules.

Subp. 6. "Closure" is defined to establish the period of time when activities at a particular site change from being mining related to being reclamation and post mining end-use related. Increased inspection to determine adherence with specific reclamation standards will occur during this period to determine if ultimate release from the permit to mine is possible. Because compliance is in part dictated by when closure occurs, it is reasonable to have it defined.

Subp. 7. "Commissioner" is defined to extend the powers of the commissioner of natural resources to duly authorized field and office staff assigned to make determinations regarding the adequacy of reclamation

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activities associated with a permit to mine. Since it is more likely that field and office staff, rather than the commissioner will be directly dealing with the permittee and the public, it is reasonable that the rule acknowledge that such staff are authorized to act on behalf of and with the same force and effect as the commissioner.

Subp. 8. "Goals" is defined to provide a brief statement of policy, defining the rationale used in the development of specific requirements contained in Reclamation Standards portion of the rules. While it is recognized, by the department, that all the goals in the proposed rule may not be fully attainable, they provide needed targets for achievement and a framework within which reasonably effective and attainable requirements have been developed, and they will provide guidance and a measurement of success by which any requests for variance from stated requirements can in part be judged.

Subp. 9. "Heap and dump leaching" is defined because it is a method of ore processing typically used by nonferrous metallic operations. Traditionally, heap leaching has been used to treat precious metal ores placed in piles upon a prepared pad, while dump leaching is most often associated with base and heavy metals extracted from lean ores stockpiled on compacted foundations. In both cases, chemical solutions appropriated to ionize metals are applied to the mineralized material. The metal ions enter the solution, drain through the heaps and dumps, and are conveyed along the pad or foundation to a collection point. From the collection point, usually a pond, the metal bearing solution is directed to a facility where the metal ions are removed from the solution and are eventually converted into a solid metallic form. These processes, which are viewed by some as being different, have been combined into one definition to emphasize that, in Minnesota, from a reclamation standpoint the same requirements will be applied without regard to the type of metal produced, the chemical solution used, or the type of pad or foundation that is developed. It is reasonable and necessary to define this term because specific rules have been prepared to deal with reclamation of heap and dump leaching activities, and the term is one commonly used in the mining industry but not by the general public.

Subp. 10. "Heap and dump leaching facilities" is defined to include every source of alteration to the natural environment associated with heap or dump leaching because all such areas should be subject to reclamation.

Subp. 11. "In-situ leaching" is defined because it is a method of ore processing that has been utilized by some nonferrous metallic mining operations, where a unique combination of geology and hydrology have made the practice feasible. The proposed rules direct that this process not be allowed to be included in an operation for which a permit to mine is issued, until the environmental effects of the process can be determined and appropriate rules developed. Because a permit to mine will be denied if these activities are included in a mining and reclamation plan, it is reasonable to define specifically what it is so that confusion with other acceptable processes does not occur.

Subp. 12. "Leached ore" is defined because it is a specific type of mine waste that may need to be reclaimed in a manner differently than other mine wastes and is therefore reasonable to specifically define. Some commentors suggested that certain tailings may be considered to fit within this definition. This however was not the department's position. Leached ore is only that mine waste produced as part of a heap or dump leaching process. Tailings from a beneficiating process that includes a leaching procedure are to be considered as tailings, not leached ore.

Subp. 13. "leaching solutions" is defined because it is a specialized type of mineral processing liquid that is typically encountered at some nonferrous metallic operations. Such solutions require specialized handling and containment in order to ensure satisfactory reclamation of a mine site where they are utilized, and it is therefore reasonable to define this term.

Subp. 14. "Lean ore" is defined because it is a specific type of mine waste that because of its future economic potential, may require special siting and handling considerations to allow it to be readily available in the event favorable economic conditions arise and is therefore reasonable to specifically define.

Subp. 15. "Metallic mineral" is defined because it is one of the bases established in statute for determining whether reclamation rules are to be developed and applied to a particular mining operation, it is therefore reasonable and necessary to define this term.

Subp. 16. "Mine waste" is defined by the statute.

Subp. 17. "Minimize to the extent practicable" is defined because it is a term used frequently in the rules to identify specific requirements of the reclamation standards section. The term incorporates reasonability by requiring the use of existing technologies, practices, guidelines, standards, or engineering safety standards developed for and commonly used by mining or reasonably similar activities. The definition requires that the commissioner determine whether the technologies or practices are the most effective and workable means of achieving reclamation, that are available. The definition is reasonable and necessary because until a specific assessment can be completed, and available practices analyzed, the most appropriated mitigation can not be determined. It is further reasonable that the techniques and practices be approved by the commissioner since the selection of the practices, in effect, has the same force and effect as rule. The commissioner should not be bound from seeking advice from whatever sources are determined necessary by the commissioner, in arriving at the appropriate technologies and practices.

Subp. 18. "Mining" has been defined to include all activities directly associated with the production of nonferrous metallic minerals which could cause sufficient damage to require reclamation. Such definition is consistent with the statute since all such activities should be subject to reclamation.

Subp. 19. "Mining area or area subjected to mining" is defined by the statute.

Subp. 20. "Mining operation" is defined as it is to assure that projects which are indeed related are considered and acted upon concurrently so the ultimate decision can properly consider all factors related to the operations.

Subp. 21. "Natural resources" is defined in similar fashion as it is in M.S. Chapter 116B.02, Subd. 4. It was determined necessary to define natural resources because of the confusion expressed by commentors to the draft rules as to what the term included.

Subp. 22. "Nonferrous metallic mineral" is defined to clarify the minerals to which this set of regulations shall apply. Since rules dealing with the reclamation of iron ore and taconite already exist, under Minnesota Rules Chapter 6130, it is reasonable to excluded minerals from which iron is extracted from Chapter 6132.

Subp. 23. "Passive reclamation methods" is defined because the rules encourage the use of reclamation which can in effect sustain itself without substantial and perpetual maintenance. This is reasonable because the corporate lives of mine operators are limited, while the area subjected to mining will exist in perpetuity, thereby elevating the desireability of reclamation practices that do not require active maintenance.

Subp. 24. "Permit to mine" is defined because it represents the legal instrument which prescribes the terms and conditions under which a mining operation may be conducted, and constitutes the authorization by the commissioner to conduct such a mining operation.

Subp. 25. "Person" is defined because the term is used throughout the rules, and the statute, to describe those entities which must receive a permit to mine prior to initiation of a mining operation. The listing of types of businesses, contained in the definition, are those that currently exist and operate iron ore and taconite mines in Minnesota and it is therefore reasonable to assume that they also exist for the operation of nonferrous mining activities.

Subp. 26. "Post closure maintenance" is defined to describe reclamation activities for which the permittee will retain responsibility after cessation of the mining operation. Since continued maintenance of the mine area, though not desireable, may be inevitable it is reasonable to address this time period.

Subp. 27. "Progressive reclamation" is defined because it is reasonable to require that when mining activity is completed on a particular area, even though operations may not have ceased, that the area be stabilized through the application of reclamation practices to prevent adverse impacts on surrounding natural resources.

Subp. 28. "Reactive mine waste" is defined because nonferrous metallic mining often generates mine wastes with characteristics that can cause water that might contact such waste to assume an unacceptable quality due to contamination. Since such waste will have to reclaimed in a manner different from that without such characteristics it is reasonable to require its identification.

Subp. 29. "Reclamation" is defined because it provides the basis for determining the degree to which the terms and conditions of the Permit to mine have been met.

Subp. 30. "Reference area" is defined because it provides a reasonable means by which the extent of vegetation required by a Permit to Mine, and therefore compliance with vegetation standards can be measured.

Subp. 31. "Storage pile" is defined because it serves to collectively identify a number of different types of landforms containing physically similar types of material, created as a result of mining. It excludes from the definition other similar landforms which are temporary in nature, thereby not requiring reclamation, and tailings basins which have unique characteristics that require the application of different reclamation practices. Because specific requirements have been proposed for storage piles, that are different than the requirements for other types of mining facilities, it is reasonable to provide for their definition. In the iron ore and taconite rules these landforms were referred to as stockpiles. However, this term seemed to be confusing to some commentors to the draft nonferrous rules, and was therefore changed to storage piles.

Subp. 32. "Surface overburden" is defined because it is a specific type of mine waste that is often susceptible to erosion, but also often has the characteristic of readily supporting vegetation, thus making it a material needing reclamation attention, but also providing aid in the promotion of reclamation. Because specific requirements have been proposed for surface overburden, that are different than the requirements for other types of mine waste, it is reasonable to provide for its definition.

Subp. 33. "Tailings" is defined because it is a specific type of mine waste that has unique physical and sometimes chemical characteristics that cause its disposal and reclamation to be conducted in a manner completely different than any other waste encountered during mining. Because specific requirements have been proposed for tailings, that are different than the requirements for other types of mine waste, it is reasonable to provide for its definition.

Subp. 34. "Waste rock" is defined because it is a specific type of mine waste that because of its chemical content may require specialized storage and reclamation practices. Because specific requirements have been proposed for waste rock, that are different than the requirements for other types of mine waste, it is reasonable to provide for its definition.

6132.0200 Purpose and Policy

This section cites the enacting legislation for these proposed rules, that contains a number of statements regarding the purpose for which the statute and these rules were written. The statement of purpose lists those considerations required by the statute of the commissioner in determining the extent and nature of these proposed rules. It is necessary and reasonable to include these statements of statutory policy because they provide the rationale for many of the specific requirements in the rules. Comments received on the draft copy of the rules have indicated either a misunderstanding of what is included in statutory policy or perhaps even a disagreement with what that policy is.

The section also contains a statement of the Department's policy that describes the main factors it believes are essential, for a mining operation to accomplish, in order for the purpose expressed in the statute to be met. There is also an expression of the policy that the best way of ensuring permanent reclamation is to promote the use of practices that will require little of no maintenance, but if such maintenace is necessary it must be the responsibility of the mine operator. It is necessary and reasonable to include these statements of statutory policy because they provide the rationale for many of the specific requirements in the rules.

Finally the section concludes that the rules are designed to act as a framework within which specific permit requirements are to be developed to address the unique problems anticipated to exist at each individual mine site. The actual reclamation, conducted at a given mine, will have to be custom designed to account for each site and operation's uniquely specific characteristics. In order to make the proposed rules workable, it is necessary and reasonable to build in enough flexibility, while still providing basic direction on how reclamation can be achieved.

6132.0300 Scope

Subpart 1. The requirement to obtain a permit to mine prior to initiation of a mining activity is a requirement of Minn. Stat. 93.481. This rule specifically requires that the permittee be a person that both supplies capital and makes decisions on how the funds will be expended. It is reasonable for the commissioner to identify who has the ability to make decisions involving reclamation activities and therefore who will be held accountable for compliance with requirements of the permit. Persons whose only involvement is the supply of capital, such as financial institutions or stockholders will not be required to become permittees.

Subp. 2. This requirement acknowledges that often more that one person is engaged in or carries out a mining operation. Where this occurs a permit will be issued on a joint basis requiring each person to be jointly and severally liable responsible for compliance with the reclamation requirements. It is reasonable to require that each joint permittee be jointly and severally responsible, in order to avoid a situation where one, or all, might claim that the others are responsible but not themselves.

Subp. 3. The term of the permit, often referred to as "life of the mine," is established in statute as a requirement of Minn. Stat. Chapter 93.481. To determine the extent of this life, the commissioner requires the submittal of information related to the ore body and the permittee's plans for development.

Subp. 4. This section describes the types of mining activities to which these proposed rules will and will not apply. The rule clarifies that where iron is the predominant metal extracted during the mining of a metallic mineral, Chapter 6130 and not these proposed rules will be applicable. This is reasonable since there are already rules in use dealing specifically with mining from which iron is the predominant metal extracted, and this set of proposed rules is not intended to amend those existing rules.

The proposed rule states that the rules apply only to portions initiated after promulgation of the rules. This is so stated in order to be in compliance with the language of the statute. However, since no nonferrous mines exist in Minnesota, the proposed rules will only apply to new operations, or to the reactivation of these new operations at some time in the future.

The proposed rule identifies two types of mining activities for which no substantive effort has been made to determine the degree of reclamation regulation that might be required before adequate rulemaking can occur. These include any activity associated with radioactive ore development, and activities related to an ore beneficiating process called in-situ leaching. Until such time as adequate studies related to these activities can be completed, it would be unreasonable for the commissioner to allow such activites to occur. Some commentors have suggested that the mineral processing technique of heap and dump leaching should also be included in this category. Those commentors feel that this relatively new technology has not yet been shown to be practically applicable in Minnesota. The department has studied this technology extensively and has concluded that in particular, because of the ease with which fluids flow through the heaps and dumps, fluids that will neutralize the residual leaching chemicals can easily be introduced into the piles, and render the leached ore non-reactive. Since this is one of the major requirements associated with the reclamation of reactive mine wastes, such as leached ore, it would be unreasonable to preclude the use of this technology, from the perspective of reclamation alone.

Subp. 5. All rules, statutes, and ordinances combine to make up the compendium of law applicable to a certain activity. The commissioner does not intend, nor have the authority, to prevent the enforcement of laws which are applicable or which are more restrictive in a specific instance. In the same light, these proposed rules should apply regardless of the existence of a less stringent standard adopted by another unit of government.

PERMIT REQUIREMENTS

6132.1000 Mine Waste Characterization

Subpart 1. As explained earlier in this statement, relative to the Purpose and Policy section, "...the rules are designed to act as a framework within which specific permit requirements are to be developed to address the unique problems anticipated to exist at each individual mine site." One of the main factors that is likely to determine whether unique problems will exist is the chemical and mineralogical composition of the mine wastes produced during the process of mining. In order for the commissioner to determine the degree of reclamation that needs to be required at a particular site it is necessary and reasonable to characterize the wastes that will be generated by the proposed mining operation.

This entire section is designed to identify those constituents that exist within the various mine wastes that have the potential to adversely affect natural resources. This would include wastes that generate a low pH drainage or that release unacceptable levels of metals. Since the results of the characterizations will have a significant impact on how, or even whether, mining can be allowed, it is reasonable that the studies be conducted in advance of the submittal of a permit application. In addition since the commissioner has been significantly involved with characterizations of various potential Minnesota mine wastes, it is reasonable for an applicant to meet with the commissioner to determine the extent of necessary analyses and tests.

Subp. 2. This section identifies the types of analyses and tests that are to be utilized in conducting the mine waste characterization. In addition it identifies sources of test material that are normally collected during the process of exploration and mineral deposit evaluation. The specific evaluations that are proposed, have been determined to be necessary and reasonable based upon the extensive mine waste characterization studies that the department has been conducting since the mid 1970's. These efforts have been recognized both nationally and internationally as pioneering the evaluation procedures that are currently being utilized worldwide.

The types of analyses that constitute the mine waste characterization include those typically conducted by mine operators to evaluate ores. Tests for describing acid generation and dissolved solids release, though less common, are being more routinely conducted.

Similarly the proposed rules require analyses of all reagents added during mineral beneficiation and to the subsequent mine wastes created. By applying such analyses to the reagents and mine wastes it is expected that consituents having the potential for creating adverse impacts can be identified.

When the analyses reveal the existence of constituents known to have the potential for creating adverse impacts, the proposed rules require that additional analyses and tests be completed. These subsequent analyses and tests, listed in the proposed rules, are designed to identify the extent, or scale, of the potential impact. Such information is essential in determining whether mining can be allowed, or what type and degree of reclamation might be necessary to protect natural resources.

Subp. 3. This section identifies specific points when mine waste chaacterization results must be presented to the commissioner. It is necessary and reasonable to require submission of results at the designated time periods since these correspond to points when the commissioner is normally evaluating the issuance of, or determining compliance with, a permit to mine.

In addition this section requires submittal of the mine waste characterization to regulatory agencies that establish water quality standards. Because such standards are often designed with consideration given to impacts on natural resources, it is reasonable for the commissioner to share this information.

6132.1100 Permit Applications

Subpart 1. The purpose of this section is to initiate a dialogue between a potential permit applicant and the commissioner at the time when the applicant is nearing a decision on proceeding with mine development. This meeting will also give the commissioner an opportunity to visit the sites of potential development. The results of the mine waste characterization conducted up to this point shall be reviewed, and the potential applicant shall explain any preliminary mine development and siting thoughts, including any options or alternatives that have been evaluated. This type of process normally would occur, even without it being required by rule. However, it is reasonable and necessary to formalize an initiation of the permitting process in order to ensure that all requirements of these and other rules are addressed in a timely manner.

If, upon conclusion of the conference, the potential applicant wishes to proceed with the development of an application, the commissioner and the applicant shall jointly conduct an informational meeting to appraise the public concerning the possibility of mineral development. Other regulators will also be encouraged to participate in this meeting so that the issues, that must be addressed through the permit to mine, can be dicussed in terms of how they relate to other requirements that may be placed upon the mining operation.

This section outlines procedures for noticing the informational meeting that are designed to reach a wide variety of people who may be affected by, or interested in such a mine development. If it is deemed practical, and would not result in a substantial delay, the commissioner may consider combining the meeting required by these proposed rules with other required meetings, such as those associated with environmental review.

Subp. 2. requires that submittals be made in duplicate for use in the DNR's Hibbing and St. Paul offices.

Subp. 3. The first three pieces of information required by this section are dictated by the statute. The affidavit of advertisement is necessary because it ensures that the public, who may be affected by the operation, have been appraised of the proposed operation. The certificate of authority to transact business is reasonable to require because it indicates that the applicant has met all legal requirements necessary to operate in Minnesota. The proof of insurance satisfies the statutory desire that compensation for injury to persons and property caused by the mining operation has been obtained.

The fourth requirement of this section, documents relating to financial assurance, are those deemed necessary by the commissioner to protect against the potential for expenditure of public funds to cover the costs of reclamation or corrective action in the event the permittee goes out of business or becomes insolvent before the mine site is satisfactorily reclaimed.

Subp. 4. This section is necessary to describe the applicant(s), and how each joint applicant relates to the proposed project and to each other.

Subp. 5. This section is necessary so the commissioner can have a clear understanding of the geographic, geologic, and environmental setting in which the proposed project will be placed. The specific information requested will allow the commissioner to determine impacts on natural resources that might result due to the operation, and also to evaluate whether special restrictions on mine facility siting or design are necessary.

It is reasonable to require that environmental reports and impact statements be included as part of the permit application since these reports are prepared to aide regulators in decisions related to permit issuance.

The requirement of overlays to 7-1/2 minute quadrangle maps was chosen, and is reasonable, because this is an effective way to present diverse types of information relating to a single area, at a scale which is readily available and useable.

Presentation of the geology is necessary for an understanding of the type of mine waste that may be encountered. Delineation of the mining area is required to inform the commissioner of the magnitude and location of lands which will be used for mining.

Delineation of the locations of surface waters that may be affected by mining is necessary to determine the type of reclamation that may be required with respect to siting, facility design, closure, and continued maintenace, since decisions concerning to these factors are closely related to maintaining the integrity of such water resources. This information is essential for determining water appropriations and discharge which are major considerations of a mining operation.

Knowledge about watershed boundaries is necessary to ensure that watershed modifications are minimized and that runoff is managed properly within watersheds.

Hydrogeologic information is required to present an understanding of the existence of groundwater resources in the mining area. The information is necessary to determine the impact on groundwater in the area, especially with regard to the influence that drawdown of the groundwater surrounding an open pit or underground mine may have on surrounding water appropriations. Some of the information related to this resource may more reasonably be presented in tabular form, in conjunction with the map overlay.

Knowledge about the location of monitoring sites and water quality requirements is necessary for the commissioner to determine if conditions exist that would impact decisions related to the issuance of the permit to mine.

The soil inventory will be used to determine the integrity of foundation soil and to estimate the types and amounts of surface overburden that must be stockpiled. Published information is available but is limited; however much of the required information is collected by the mining operator in order to determine ore stripping ratios.

The locations of rare, endangered, and threatened species, identified as part of the environmental review process, will influence the siting of mine facilities and is therefore reasonable to require.

Because the siting of mining facilities on already disturbed locations as, opposed to the use of undisturbed sites, may result in less impact on natural resources, it has been determined that identification and evaluation of such locations is reasonable.

The locations of archeological or historic sites, identified as part of the environmental review process, will influence the siting of mine facilities and is therefore reasonable to require.

Buried structures and utilities may be affected by the siting of mine facilities, and are therefore reasonable to identify.

Because certain areas have been identified in the proposed rules as having restrictions on their use, relative to mining, it is reasonable to require their identification in the permit application.

The identification of surface and mineral ownership is required because Minn. Stat. 93.481, Subd. 1(d) requires publication of the ownership of the mining area, and use of an overlay is a reasonable way of presenting the data.

Subp. 6. The information required by this section, in conjunction with data presented on the maps required by the next subpart, will constitute a complete explanation of how mining and reclamation is proposed to occur. It is reasonable to require that information developed, as a result of mine waste characterization, be incorporated into the plans.

Information related to the operating life of the mine is necessary for the commissioner to determine the extent of the term of the permit.

Descriptions of how ores will be mined and processed, and how wastes will be stockpiled or otherwise disposed of, is necessary for an understanding of the scope and extent of the proposed operation and the associated reclamation efforts that will be necessary.

In order that the commisioner have a clear understanding of the rationale behind the selection of each reclamation procedure or technique, for which approval is being sought, it is reasonable to require a presentation of all engineering designs, methods, sequences, and schedules, and an appraisal of how these meet the reclamation requirements. Similarly, since reclamation research may lead to requests for variances from stated requirements, it is reasonable for the commissioner to have an understanding of any anticipated research.

Subp 7. As stated above, this section is necessary in order for the commissioner to have a clear understanding of the mining and reclamation activities proposed by the applicant. The use of maps and cross sections is required because based on the experience of the commissioner, in the performance of resource management responsibilities, it is the easiest and clearest way of presenting and analyzing much of the information that is required. Information normally found on U.S.G.S. quandrandle maps is required because this is basic information which will help relate these maps to the environmental setting maps (see 6132.1100 Subp. 5). A specific scale for the maps is not specified because it seems reasonable to allow each applicant to use a map scale that best fulfills the applicant's needs.

The depiction of the orebody is required because the commissioner must know where and what will be mined under the authorization of the permit to mine in order to impose and enforce a reasonable and effective reclamation plan.

Identification of potential vegetative reference areas is required to give the applicant an opportunity to suggest areas which might be utilized to measure the success of revegetation activities. This is reasonable because the applicant should have the best working knowledge of the surrounding area where suitable reference areas might exist, and of sites where the applicant may have successfully performed revegetation of disturbed lands.

The location of drainage patterns of waters contacting reactive mine wastes is necessary because such waters have the potential of adversely impacting natural resouces and therefore may need to be collected and treated.

The depiction of the status of:

1) mining;
2) watershed and hydrogeologic modifications; and
3) reclamation

at each mining landform or facility, throughout the proposed life of the operation, is required to inform the commissioner of the sequencing of such activities. This information is necessary and reasonable in order for the commissioner to determine:

1) if reclamation will be proceeding in a progressive manner;
2) when compliance monitoring of reclamation activities, by the commissioner, must be conducted;

and

3) when mining areas can be released and utilized for other purposes.

Subp. 8. This section requires submittal of a detailed operating plan for the first year of the mining operation. The contents of the plan are identified in detail under 6132.1300 of this statement of need and reasonableness. It is reasonable and necessary to require a detailed description of the first year's mining activities, since it is this information that will allow the commissioner to determine if the permittee is in compliance with the overall mining and reclamation plan.

6132.1200 Financial Assurance

The overall rationale behind this section is that there should be no possibility that public funds will have to be expended to correct accidents, reclaim lands, or rectify adverse effects resulting from a mining operation. To ensure the protection of public funds, a part of these proposed rules require that financial assurances be provided by the permittee prior to permit issuance. This requirement is becoming a common practice in most states. In fact, financial assurance is a major component of proposed federal guidelines for the regulation of mine wastes.

Subpart 1. This section addresses the possibility of an operator being unable to perform reclamation obligations. Two circumstances have been identified that might lead to the commissioner actually becoming responsible for performing reclamation. In each case it is reasonable for the commissioner ensure that arrangements exist to provide necessary financial assurance. The first circumstance necessitating and need for financial assurance involves the costs of any unfulfilled reclamation obligations, remaining in the event that a permittee unexpectedly ceases operations or becomes insolvent. Obviously such arrangements would have to be in place prior to the cessation of the operations. Therefore, such financial assurance would have to be in place from the first day of operations until all reclamation is complete.

The second situation that might necessitate the requirement of financial assurance would involve costs associated with a substantial non-compliance with the permit to mine. In the event it is determined that a permit violation exists, those actions needed to bring the operation back into compliance with the permit will have to be identified and estimates made of their costs. In the event the commissioner determines that existing financial assurance coverage would be insufficient to both correct the violation and to perform reclamation, that might be necessitated by a cessation of operations, then the commissioner will have to order the operator to provide additional assurance until permit compliance is achieved.

Subp. 2. Since it is possible that a mining operation could cease activities even during initial start up stages, a cost estimate of any reclamation activities that would need to be conducted is reasonable for the commissioner to require.

Because the initial estimate may only be accurate for a short period, it is also reasonable to require its update, a process that should be repeated on a regular basis, to maintain a reasonable level of assurance. An annual update and adjustment is required to coincide with other reporting requirements of the permit application submitted in an annual report.

It is very likely that the commissioner will not have the required staff, equipment, or management ability to perform the actual reclamation work. Therefore the cost estimates must consider that the commissioner will have to contract with others to perform the reclamation activities. The cost estimate must therefore reflect this, and also reflect administrative costs as they exist at the time of estimate preparation. It is additionally reasonable to require documentation of the source of estimated costs.

Because it may be necessary for the commissioner to react quickly to provided needed reclamation, it is reasonable to require that the financial assurance be liquid. Delays caused by waiting for the sale of assets, combined with inflating costs may jeopardize reclamation success.

Subp. 3. In the event that a permit violation occurs and the commissioner determines that the financial assurance that exists for the operation is not capable of covering all appropriate costs, it is reasonable that the commissioner order the preparation of a cost estimate of any activities necessary to bring the operation into compliance.

In the event that corrective action is extensive, taking a considerable time to complete, the initial estimate may only be accurate for a short period, it is therefore reasonable to require its update, a process that should be repeated on a regular basis, to maintain a reasonable level of assurance. An annual update is required to coincide with other reporting requirements of the permit application submitted in an annual report.

It is very likely that the commissioner will not have the required staff, equipment, or management ability to perform the actual corrective action work. Therefore the cost estimates must consider that the commissioner will have to contract with others to perform the activities. The cost estimate must therefore reflect this, and also reflect salaries and overhead costs as they exist at the time of estimate preparation. It is additionally reasonable to require documentation of the source of estimated costs.

Subp. 4. Because of the importance associated with ensuring that funding will always be available in the proper amount, this section is necessary to ensure:

1) that appropriate initial evaluation and annual updates of cost estimates occur;

2) that adequate assurance mechanisms are established;

3) that initial mechanisms, or their replacements, remain in existence until all reclamation is completed; and

4) that the assurances will be continually maintain even in the event that the permittee changes.

It is anticipated that the cost estimate information, submitted by the operator, may be beyond the capability of the commissioner to evaluate without the aide of experts in the fields of construction, building demolition, waste disposal, and several other specialized areas where accurate, up to date knowledge of costs, is only available through actual, day in and day out, hands on, working experience. It is therefore reasonable for the commissioner to seek such expertise. Because access to such expertise will expedite the commissioner's decisions, regarding acceptability of the operator's estimates, it is not anticipated that the operator will object to paying reasonable costs of such an evaluation.

This section requires the maintenance of continuous financial assurance, acceptable to the commissioner, in the amount equal to the estimated cost of implementing the contingency reclamation plan. It is reasonable to require that arrangements for financial assurance, in the amount equal to the initial contingency reclamation cost estimate, occur prior to permit approval. It is similarly reasonable to require that adjustments, corresponding to annual changes in the contingency reclamation cost estimates be made either upward or downward as necessary.

This section also requires the establishment and maintenance of financial assurance, acceptable to the commissioner, in the amount equal to the estimated cost of correcting a situation that has resulted in a substantial non-compliance with the permit to mine. It is reasonable to require that arrangements for financial assurance, in the amount equal to the initial corrective action cost estimate, be made when plans to correct the situation are approved by the commissioner. It is similarly reasonable to require that adjustments, corresponding to annual changes in the corrective action cost estimates be made either upward or downward as necessary.

This section contemplates that the permittee may at some time during the operation wish to cancel certain financial assurance mechanisms and replace these with others. This section allows this based upon approval of the commissioner.

To ensure that financial assurance is continually maintained, it is reasonable to require that alternative assurance, acceptable to the commissioner, be acquired prior to the point when an existing provider cancels the permittee's financial asssurance mechanism.

Because the statute recognized the possibility that the permittee may change throughout the life of a mining operation, the commissioner was given the ability to approve assignment of the permit. It is reasonable that the commissioner's approval of an assignment be contingent upon the continual maintenance of financial assurance, to ensure that public funds will not have to be spent if the assignee is unable to perform.

At some point in time it is anticipated that the operation will cease. If all reclamation requirements have been met and no post closure maintenance is required, the permittee will be released from responsibility. At that point it is excepted that financial assurance will no longer be necessary. This section therefore describes the circumstances under which maintenance of the assurance will cease.

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Subp. 5. This section contains a list of factors which if met will constitute an acceptable financial assurance mechanism.

Because the commissioner may be responsible for conducting reclamation or corrective action, in the event the permittee is unable or unwilling to do so, it is reasonable for the commissioner to require that the assurance be large enough to cover the costs.

It is also reasonable to require that the funds be available to the commissioner when needed, and not necessitate burdensome procedural or legal remedies to acquire.

The assurance must be binding, otherwise it could not be considered adequate.

Since one of the main conditions that could lead to a permittee being unable to perform reclamation or corrective action, is bankruptcy. Therefore it is reasonable to require that the financial assurance be available especially if bankruptcy should occur.

It is anticipated that financial assurance proposals, submitted by the operator, may be beyond the capability of the commissioner to evaluate without the aide of experts in the fields of insurance, banking, surety bonding, or any of several other specialized areas where accurate, up to date knowledge is only available through actual, day in and day out, hands on, working experience. It is therefore reasonable for the commissioner to seek such expertise. Because access to such expertise will expedite the commissioner's decisions, regarding acceptability of the operator's financial assurance, it is not anticipated that the operator will object to paying reasonable costs of such an evaluation.

Subp. 6. This section identifies the general procedures the commissioner will follow in the event it is determined that the assurance may need to be forfeited. Specific detailed language defining the grounds for what determines the forfeiture, and the exact procedures for transferring the funds to the commissioner, will be contained in great detail in the documents that create the financial assurance. The purpose of this section is to provide a final means of correcting a situation that could lead to forfeiture, prior to the actual forfeiture. It is reasonable to try to resolve the problem before the financial assurance is actually forfeited.

The process of initiating a forfeiture will begin with a warning notice of an impending forfeiture, with a description of what might be done to correct the situation.

Whether the actual forfeiture occurs will depend on whether the permittee takes corrective measures or not. If appropriate corrective actions are not taken in a timely manner the commissioner will take actions necessary to forfeit the financial assurance.

Subp. 7. In the event of failure by the permittee to acquire or to maintain financial assurance in an appropriate manner, it is reasonable for the commissioner to have the ability to take other actions, provided by statute, to help stop further non-compliance.

6132.1300 Annual Report

Subpart 1. The requirements of this section, which will be updated each year throughout the life of the operation, are necessary to provide specific details of the exact activities that occurred during the preceeding 12 months and those that will be conducted during the upcoming year. Because of the dynamic nature of mining it is reasonable to assume that at some time throughout the life of the operation there might be the necessity to deviate from the plans developed in the permit application. The annual update will allow both the permittee and the commissioner to evaluate whether there may be deviations from the permitted mining and reclamation plan. Such an evaluation is required to determine whether amendments to the permit to mine are necessary, thereby ensuring that permit conditions continue to be current and reflect changes in the mining plan. Duplicate copies are necessary for both the St. Paul and the Hibbing offices' uses. March 31st was selected as the due date, to ensure that the information and plans could be reviewed before the beginning of each upcoming spring reclamation planting season.

Subp 2. This section requires the permittee to provide an auditing of the mining and reclamation activities performed by the operator. The information supplied in this report will be used to monitor compliance with the rules, and the permit to mine, and is therefore necessary and reasonable to request. The specific information requirements parallel those of the mining and reclamation plans, and associated maps, provided in the permit application. The rationale for the requirements is the same as presented elsewhere in this statement, describing the need for the mining and reclamation plan.

Subp. 3. This section will provide the commissioner with detailed specifications on how mining and reclamation is planned to be conducted during the upcoming year. This information is reasonable to require to ensure that the commissioner will be made aware of any proposed modifications, dictated by changing technology, environmental conditions, economics, or other factors. The specific information requirements parallel those of the mining and reclamation plans, and associated maps, provided in the permit application. The rationale for the requirements is the same as presented elsewhere in this statement, describing the need for the mining and reclamation plan.

Subp. 4. Section 6132.1200 of the proposed rule, requires that there be no possibility of public funds being expended to reclaim mine lands. The purpose of this section is to establish the basis from which to estimate the size of the financial assurance that will be needed. The contingency reclamation plan identifies all reclamation activities that would need to be conducted at a mine site, in the event that the mining operation were to become insolvent, or is forced to cease activities. The plan must be periodically updated to compensate for the opening of new areas and the reclamation of completed sites. Because Minnesota Statutes, section 93.49 requires the commissioner to annually review the extent of each operator's financial assurance, it is reasonable that the contingency plan also be annually reviewed.

Subp. 5. Section 6132.1200 of the proposed rule, requires that there be no possibility of public funds being expended to take corrective actions that may be necessary to bring an operation into compliance with the rules. The purpose of this section is to establish the basis from which to estimate the size of the financial assurance that must be maintained. The corrective action plan identifies the status of activities that may be required, in order to reestablish compliance with the permit to mine, in the event that a violation occurs. Because Minnesota Statutes, section 93.49 requires the commissioner to annually review the extent of each operator's financial assurance, it is reasonable that the corrective action plan also be annually reviewed.

Subp. 6. Maps are required to help supplement the information presented in accordance with the preceeding parts of this section. Maps often provide the most clear method of presenting such data and are therefore reasonable to require.

6132.1400 Request of Release from Permit

Subpart 1. This section requires information which will be utilized by the commissioner in determining whether the permittee has complied with the requirements of these rules and the permit to mine, thereby allowing the commissioner to determine whether the permittee should be released, from all, or a portion of the mining area.

Subp. 2. Since this section provides a method for the permittee to be released from responsibility, it is reasonable to require that the permittee demonstrate that all requirements of the reclamation standards section of the rules and of the permit to mine have been met.

Ownership of the mining area and any remaining structures is required, in order to facilitate contact with the owner regarding the owner's understanding and acceptance of liability responsibilities associated with the area. It has been the experience of the commissioner, with mined lands on the iron range, that when management responsibilities are not well understood, lands are often either not properly maintained or are allowed to become tax forfeited, thereby shifting the responsibility to the public. Because one of the reasons for requiring reclamation, is to prevent the public from having to accept liability for abandoned mined lands, it is reasonable and necessary to ensure that the persons taking on that responsibility, after mining ceases, fully understand their responsibilities.

Because the possiblility that post closure maintenance may be necessary it is reasonable that the location, type, and schedule of maintenance be identified.

Since the proposed rules do not allow the release of areas requiring post closure maintenance, it is necessary that a complete discussion concerning the management of areas, on which such maintenance activities are required, be included in the request for release. This is reasonable, in order to eliminate the possibility of misunderstandings concerning post closure maintenance responsibilities.

To minimize the possibility that subsequent non-mining land use activities could adversely affect reclamation efforts, the rules require that notice be given to future land owners. It is reasonable that such a notice, stating that the lands have been mined and subsequently reclaimed, be affixed to the land records permanently maintained by the county recorder.

Finally, this section requires the submission of a map, to clearly show how the area has been reclaimed. This map, depicting the specific information detailed in this section, presents the as-built record of the mining and reclamation that was conducted, and is therefore a necessary and reasonable document to require in order to have a complete record of the mining and reclamation activities that occurred on the mine site.

6132.2000 SITING

Subpart 1. This section expresses the policy of these rules, that the process of site selection shall be utilized to minimize impacts of mining. It is reasonable that with judicious site selection environmental impacts from mining can be reduced. Some who have commented on the draft of the proposed rules have suggested that when mine facilities are proposed to be located on sites possessing unusual foundation conditions such as floodplains, active seismic zones, Karst topography, and wetlands, that the rules should merely exclude that site from use for mining purposes. The commissioner has determined that this suggestion is based mainly on a reasonable concern of the commenters, regarding the storage of material having the potential for creating water quality problems. The siting section, however is designed to address all mine siting, not just those facilities that might contain reactive mine waste. As a result, the commissioner is not proposing such a prohibition. Instead the commissioner, who shares the concerns of the commenters with regard to reactive waste storage, has determined that the General Siting Criteria, of subpart 5, in conjunction with the individual requirements of appropriate engineering design, as contained in the sections that deal with: Reactive Mine Waste; Storage Pile Design; Tailings Basins; and Heap and Dump Leaching Facilities, are adequate to regulate construction of mine facilities that might be located on areas possessing unusual foundation conditions.

Subp. 2. This section contains a listing of areas in which no mining shall be conducted unless authorized by statute. In these areas, that have been formally designated by state or federal legislative actions, mining is prohibited because such activities are either specifically prohibited by the enabling legislation or the legislative directive on how the lands are to be used and managed is so restrictive that the disruptive nature of mining is incompatible. Moreover, the reclamation act directs the commissioner to identify areas or types of areas which cannot be satisfactorily reclaimed under the rules, and further, prohibits the commissioner from issuing permits to mine such unreclaimable areas.

Minnesota statute, section 84.523 subd. 3, prohibits mining within the BWCA except in the event of a national emergency declared by congress and approved by the legislature.

Minnesota statute, section 84B.03, subd. 1, prohibits mining within Voyaguers National Park.

Minnesota statute, section 86A.05, subd. 6, prohibits the use of minerals within state wilderness areas.

Federally designated wilderness areas are managed in a manner that would preclude the development of mining.

Minnesota statute, section 86A.05, subd. 5, directs that state scientific and natural areas be specifically managed for research, educational, or interpretive purposes and be protected from unnatural influences. On the basis of this directive it is reasonable to prohibit mining.

Minnesota statute, section 84.035 establishes a special type of scientific and natural area, called peatland scientific and natural area. The statute requires that, if as a result of metallic mineral removal, there will be a significant alteration or modification of the peatland water levels or flows, peatland water chemistry, plant or animal species or communities, or other natural features of the peatland scientific and natural area, all restrictions otherwise applicable to scientific and natural areas shall apply. On the basis of this directive, the commissioner could allow the siting of a mine, within a peatland scientific and natural area, only if the ore removal will not significantly impact the peatland. Minnesota statute, section 84.035 also provides for relaxation of restrictions in the event of a national emergency declared by Congress.

Minnesota statute, section 103G.223 requires the commissioner to identify a specific type of wetland, called a calcareous fen, that may not be drained, filled, or otherwise degraded, wholly or partially, by any activity, unless the commissioner, under an approved management plan, decides some alteration is necessary.

State parks are major recreational units, managed by the commissioner. The statutory directive, for park management, requires physical development within the park to be limited to facilities that complement natural features or promote use and enjoyment of recreational resources. Except for those parks developed because of their relation to mining, the siting of mining facilities within a state park is not in keeping with the stated legislative intent.

Subp. 3. This section contains a list of areas that either possess important natural and human resource values, or that surround those specially designated areas listed in subpart 2. Each of the areas listed in this subpart, is likely to involve the presence of people, therefore specified zones of non-disturbance between such areas and mining activities are required. These zones are meant to act merely as separations, serving as a buffer between inconsistent land uses, and reducing the intrusive effects of mining onto adjacent lands. The commissioner has concluded that there is no way to completely eliminate all evidence of the existence of mining, but that the separation of non-compatible uses, by means of the stated separations, is a reasonable alternative. Some commenters on the draft of the proposed rules have alleged that the distances required by the proposed rules would not be sufficient to meet air, water, and noise standards. Those standards, which are regulated by others, are beyond the authority of the commissioner. However, if it can be demonstrated that the only means of achieving such standards is by the use of separations greater than are proposed by this subpart, then the commissioner would support the separations required by such other regulators.

The "Minnesota Department of Natural Resources B.W.C.A.W. Mineral Management Corridor" was selected for inclusion in this subpart because it is an area that was designed by the Department to provide a reasonable separation between the BWCAW and lands upon which the state is willing to offer mineral leases. This corridor, in addition to providing separations for increased public health and safety and a desired land use compatibility, was also designed to prevent the direct overland flow of runoff water, that forms within the corridor area, from entering the BWCAW. This last feature of the cooridor provides increased environmental protection to the waters of the BWCAW. It is reasonable to include this cooridor because it provides the type of protection to the BWCAW that this section of the rules was designed to provide.

Subitems B. through G. of this subpart contain the remainder of the specially designated areas listed in subpart 2. The 1/4 mile distance from these areas, forming the no surface disturbance zone, was selected to be consistent with existing management areas adjoining other similarly important natural resources already afforded state and federal protection, most noteably the national and state wild, scenic, and recreational rivers and their land use management zones. A 1/4 mile separation around peatland scientific and natural areas was not included within this section. Instead a different separation, as is listed in 6132.2000 Subp. 4., item C. will be required. The restrictions established for that zone of separation are more consistent with the provisions of Minnesota Statutes, section 84.035, which in some site specific cases (as are specified by 6132.2000 Subp. 2., item F.) could allow mining and some surface construction within the actual peatland scientific and natural area.

Historic sites, both national and state designated, have been identified as requiring the protection afforded by this subpart. It is reasonable that such sites will be visited by the public and therefore safety and compatiblity are important factors.

As already mentioned state and national wild, scenic, and recreational rivers and their associated management areas are the types of natural resources for which this section was designed. The public is encouraged, by the designation, to utilize these sites, therefore public protection takes on added importance. It is reasonable to afford the same type of protection to that portion of the Upper Mississippi River for which the described management plan has been developed, since in accordance with Minnesota statute 93.47 Subd. 4., the rules shall conform to local land use planning.

Local units of government have also organized to produce a plan for how the North Shore of Lake Superior will be regulated. It is reasonable, that these rules acknowledge the public process used to develop this plan, and further that the proposed rules afford to the public who utilize the North Shore, the protections offered by this subpart.

This section also includes areas characterized by substantial human activity such as commercial and residential areas, public transportation networks, and cemeteries. It is reasonable, because of the extreme disruptive nature of mining, to separate surface related mining activities, from the uses listed in this subitem. The distances selected are based on several factors, including existing experience along the iron range, the use of increased security around mining facilities located in close proximity to populated areas, and the numerous reclamation standards including: Buffers, Vegetation, Dust Suppression, Air Overpressure and Ground Vibrations from Blasting, Subsidence, and the various engineering requirements for specific mine facility designs that have been developed to address impacts on the public.

Subp. 4. This section contains a list of areas in which mining is prohibited, except when no feasible and prudent alternative to mining in such areas exists. The rationale for protecting these areas is that they possess high resource value, which should be protected from inconsistent development. However, from a natural resource standpoint, they are not as significant or rare as those areas listed under "Mining excluded" or "Surface disturbance prohibited," and as such, the prohibition of mining within these areas is not absolute. It is reasonable for the commissioner to review each proposal affecting such areas and base approval of the site on the extent of the impacts that will result. Further it is reasonable for the commissioner to require compensation for any resources lost.

Subp. 5. This section is included to require that those portions of a mining operation for which alternative locations exist, be sited in locations so as to minimize, to the extent that it is practical to do so, certain adverse environmental effects and possible injury and damage, particularly on certain areas identified as having special resource value. One method of achieving this is to site portions of a mining operation in areas previously used for mining purposes. If all other impacts are equal, it is more reasonable to redisturb a previously mined area than it is to disturb a new area. It is prudent, when a choice exists, to locate mine facilities where adverse environmental impacts and potential injury will be minimized.

Subp. 6. This section is included to require that when it is necessary to conduct mining operations that will result in the draining or filling of wetlands, regulated by the Wetland Conservation Act of 1991, that a replacement plan must first be approved pursuant to part 6132.5300, and that such replacement plan must consider measures that avoid and mitigate the drainage or filling. This requirement is necessary in order to comply with Minn. Stat., 103G.2242.

6132.2100 BUFFERS

Subpart 1. A mining operation is essentially a land use which is often incompatible with other, non-mining types of land use. The incompatibility is attributable in large part to the scale on which a mining operation is conducted. Often, thousands of tons of ore is mined per day, requiring the removal, transport, and deposition of even greater amounts of waste material. The noise generated as a result of material movement is substantial. The landforms created as a result of disposal or storage of the waste material can be uncharacteristic in size and form, relative to the surrounding natural topography. the physical plant necessary to process the mined materials generally consists of various buildings which are of enormous size, even when compared to other industrial facilities. The purpose of adopting standards relating to the construction of buffers, as the goal statement describes, is to reduce or eliminate the obtrusive and nuisance-type impacts a mining operation can often have on surrounding land uses.

Subp. 2. Natural topography and vegetation, or a vegetated constructed landform similar to that occurring in nature, may be used as buffers. The requirement to use natural terrain or vegetated berms is required in order that the mining area will blend in with the surrounding landscape, thereby minimizing its visual and aesthetic impacts on adjacent areas. The use of natural topography and vegetation is preferable, because it requires no disturbance of land and represents no cost to the operator.

Initiation of the buffering is required prior to beginning operations. This is especially important since it may take some time for planted vegetation to become fully effective. The goals would not be served if mining preceded initiation of necessary buffering. Construction of buffers within a separation zone required by 6132.2000 Subp. 3. M. (to be maintained between mining surface disturbances and commercial and residential areas, cemeteries, and transportation networks) is allowed because such buffer construction is designed to resemble the natural landscape, and there is no need, from an aesthetic standpoint to maintain a setback between such buffer and the adjacent land use area.

6132.2200 REACTIVE MINE WASTE

Subpart 1. As stated in its definition, reactive mine waste is material that has been shown through waste characterization studies to release substances that adversely impact natural resources. It is reasonable that the commissioner, who is responsible for managing natural resources, has as a goal, the prevention of such impacts.

Subp. 2. A. This section requires that a continuous program of waste characterization be conducted before, and during all phases of mining in order to identify all material that might have the potential of adversely impacting resources. This requirement is reasonable because nonferrous mineralized formations can have erratic mineralization patterns that can vary from one area to another. As a result of this situation, on-going analysis of the character of wastes being produced and stored on the surface is a necessity.

Subp. 2. B. This section requires that any facility constructed to contain reactive mine wastes be designed by professional engineers registered in Minnesota. This requirement is necessary and reasonable since it limits the designing to those persons whom the state has determined, through examination, are capable of performing engineering designs. The further requirements that the person be proficient in the design, construction, operation, and reclamation of reactive mine waste storage facilities, further limits the number of designers that would be acceptable to the commissioner. These requirements will ensure that only the most competent people will be designing these critical structures in Minnesota.

This section provides two requirements for dealing with reactive mine waste. To meet the first requirement, measures would have to be taken to prevent substances, that adversely impact natural resources, from forming within the mine waste. If no such substances are allowed to form, it can reasonably be expected that no impact will occur. In the event it is not be possible to prevent the formation of unacceptable substances, a design must be presented that: 1) prevents substantially all water from contacting unacceptable substances within the mine waste; and 2) provides for the collection and treatment of water that is contaminated, because it can not be kept away. Both these methods have been used in the mining industry, in various degrees, to control adverse impacts on natural resources, and are therefore reasonable to require. Another method, that consists of merely collecting contact water and treating it in order the meet water quality discharge standards, without a substantial effort to minimize the amount of water contacting the waste, has been rejected. While this method may provide acceptable results during active operations, when the permittee is present, the potential for longterm failure of such a system, when the operator is no longer available to correct the situation, is too great. Because of the necessity to provide a permanent solution to the water quality concerns related to reactive mine wastes, the two required methods of storing these wastes are the only reasonable methods currently available.

Subp. 2. C. This section contains a list of design specifications that the commissioner has determined must be provided in order to assess the adequacy of the design. It is reasonable for the commissioner to require descriptions of all materials that will be used, and explanations of any special construction or operating practices that must be employed, since this information is essential to determining if the plan can be approved. The commissioner is also requiring the designer to maintain a presence beyond design preparation, by mandating scheduled inspections to be conducted by the designer throughout the permitted life of the facility, in order to ensure continued design compliance. This requirement is essential, since the designer will be most familiar with the rationale for the particular design, and should be able to assess whether the facility is performing successfully. To further ensure compliance, the commissioner is requiring that the designer propose monitoring locations at which information can be collected to accurately measurer success of the facility.

Subp. 2. D. This section recognizes that there may be parts of these proposed rules, that if implemented at a particular reactive mine waste facility, may not be completely compatible with the design. This rule merely states that, if this were the case, the design rather than the incompatible rule shall be given precedence.

6132.2300 OVERBURDEN PORTION OF PITWALLS

Subpart 1. When orebodies are not deeply buried, the usual means of ore removal is by surface mining methods. This type of mining is safer for the mine workers, and is usually more cost effective, than conducting underground mining. However, it does leave large open pits, that because of their size and depth create safety concerns equal only to the most rugged and hazardous of natural landforms. The purpose of this section is to require that the near vertical walls, in the upper portion of such a pit (that occupied by the surface overburden), are sloped to a moderate angle, as early in the operation as possible. In addition to providing a surface that can be readily stabilized, this requirement provides a zone, that through selective vegetation, and fencing or signing, will help provide a warning to those entering the area.

Subp. 2. A. This section contains a series of requirements that more or less create a recipe for an acceptable pitwall slope. The specific heights, widths, and angles listed, are the outcome of studies and testimony presented at the public hearings on the iron ore and taconite reclamation rules that were conducted in 1979. However the most persuasive reason for the proposed rule is the experience of the department, gained as a result of administering the iron ore and taconite reclamation program over the last 12 years. The pitwall sloping that has been conducted at iron mines over the last twelve years has adequately met reclamation expectations.

Subp. 2. B. Because of the extreme diversity of surface overburden, it is possible that other sloping specifications could provide acceptable reclamation. This requirement reasonably allows the mine operators to explore other means, while providing the commissioner with the information necessary to make future decisions.

6132.2400 STORAGE PILE DESIGN

Subpart 1. A mining operation entails the substantial disruption of the physical environment of the mining area. In addition to the removal and movement of huge quantities of materials, there is construction of large landforms, referred to as "dumps," "stockpiles," or as in these rules, "storage piles." These structures contain the waste soil and rock encountered during the ore removal process, and those by-products of the ore beneficiating process (such as leached ore) not normally disposed of in a tailings basin. These landforms can often occupy areas as large as the mine opening itself. With any such land disturbance, the potential for environmental impacts and land use conflicts is high. The surface hydrology and drainage patterns of the mining area are often significantly disrupted. Erodable surfaces are created, leading to both air and water pollution. The terrain is drastically modified, and landforms of a scale and form uncharacteristic of the surrounding natural topography are created. This section of the rules gives direction on the factors that should be incorporated into storage pile design and construction, in order to minimize effects on natural resources, and facilitate effective reclamation of these structures.

Subp. 2. A. This section addresses specific standards that are to be applied to storage piles, without regard as to whether they will contain broken rock or surface overburden. If the stucture will be placed upon areas with foundation characteristics that might adversely affect an improperly designed structure, this rule requires that a professional engineer registered in Minnesota evaluate the foundation and design the structure. The requirement that the storage pile be designed by a registered professional engineer is reasonable since it limits the designing to those persons whom the state has determined, through examination, are capable of performing engineering designs. The further requirements that the person be proficient in the design, construction, operation, and reclamation of facilities on unstable foundations, further limits the number of designers that would be acceptable to the commissioner. These requirements will ensure that storage piles will be appropriately designed.

The rules also require the use of techniques commonly applied at construction sites to limit erosion and sedimentation. Such practices have been identified as "best management practices" by agencies such as the U. S. Agriculture Department's, Soil Conservation Service, for use on drastically disturbed areas. Because the techniques have been found to be effective and economical to incorporate into an operation, they are reasonable to require for erosion control.

This subpart recognizes the experiences observed by the department at mine sites and other drastically disturbed areas in Minnesota, where in spite of the creation of well designed and constructed artificial drainage structures, nature often develops its own means of providing drainage. When this inevitably occurs this rule requires that such flow paths be stabilized.

Finally, this section cautions that storage piles containing reactive mines wastes may require other design criteria to be developed in accordance with the provisions of 6132.2200.

Subp. 2. B. This section contains a series of requirements that more or less create a recipe for acceptable rock storage pile construction. The specific heights, widths, and angles listed, were selected based on the outcome of studies and testimony presented at the public hearings on the iron ore and taconite reclamation rules, conducted in 1979. Those rules specifically addressed stockpiled materials that are expected to be similar to the wastes that will be encountered at nonferrous metallic mining operations. Because of this similarity it is reasonable to again apply these rules. Additionally, the most persuasive support for the proposed rule is the experience of the department, gained as a result of administering the iron ore and taconite reclamation program over the last 12 years. The specifications for constructing rock stockpiles, have been demonstrated in the field to be reasonably attainable and to provide acceptable reclamation.

Subp. 2. C. This section deals with requirements specifically associated with surface overburden. The first part of the rule requires that if surface overburden is generated near completed benches and tops of rock, lean ore, or leached ore stockpiles, such surface overburden shall be placed upon those locations. The rationale for this requirement is that the surface overburden has been shown to enhance the ability of these locations to support the vegetation that is required to be established there. However, it has also been demonstrated to the commissioner that on certain types of rock, the vegetation requirements of these rules can be met with very little, and in some cases no application of surface overburden. Therefore this section allows the consideration of economics in determining whether the overburden must be placed upon rock piles, and does not include the mandate for a specific depth of surface overburden.

This section also contains a series of requirements that more or less create a recipe for acceptable surface overburden pile construction. The specific heights, widths, and angles listed, were selected based on the outcome of studies and testimony presented at the public hearings on the iron ore and taconite reclamation rules, conducted in 1979. Those rules specifically addressed surface overburden that is expected to be similar to that encountered at nonferrous metallic mining operations. Because of this similarity it is reasonable to again apply these same rules. Additionally, the most persuasive support for the proposed rule is the experience of the department, gained as a result of administering the iron ore and taconite reclamation program over the last 12 years. The specifications for constructing surface overburden piles, have been demonstrated in the field to be reasonably attainable and to provide acceptable reclamation.

Subp. 2. D. This section prohibits the covering of surface overburden with material that would inhibit the establishment of vegetation, or the future utilization of the surface overburden storage pile from future subsequent land uses.

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Subp. 2. E. Because of the extreme diversity in the types of mine wastes that may be contained within a storage pile, it is possible that specifications other than those included in this section could provide acceptable reclamation. This requirement allows the mine operator to explore other means of designing and constructing storage piles, while providing the commissioner with the information necessary to make future decisions.

6132.2500 TAILINGS BASINS

Subpart 1. Tailings are produced during the process of ore beneficiation. The ore is usually ground into the consistency of a fine sand, or powder, and the economic mineral portion is separated from the uneconomic mineral portion. The economic minerals are usually referred to as concentrate, while the remainder is called tailings. During the grinding of the ore, water is added to the process, to control dust, and to form a slurry that acts as a transport media to move the ground ore through the beneficiating process.

Most often the tailings are very fine and must be disposed of within a special disposal area called a tailings basin. The tailings basin not only provides a place for storing the waste, but also provides a quiescent location for clarifying the tailings water, so it can be recirculated back to the beneficiating plant for subsequent use. Sometimes a portion of the tailings can be course enough to be transpored by mechanical means, such as by truck or conveyor belt. These tailings are sometimes stockpiled in storage piles, similar to the waste rock, lean ore, and leached ore regulated in accordance with 6132.2400. However, most often these coarse tailings are used to construct dams to form the tailings basin within which the fine tailings are stored.

With nonferrous mining, the portion of the ore that will become tailings usually far exceeds the volume of concentrate produced. The result is that the tailings basin often becomes the largest structure associated with a mining area, sometimes covering square miles of area. This section of the rules gives direction on the factors that should be incorporated into tailings basin design and construction, in order to facilitate effective reclamation.

Subp. 2. A. Because of the immense size of the tailings basin and the inherent risks associated with impounding great volumes of water and fine solids, the commissioner has determined that it is reasonable and necessary to require that tailings basin structures be designed and constructed only under the supervision of qualified experts. This rule therefore, requires that a professional engineer registered in Minnesota design tailings basins. This requirement is reasonable since it limits the designing to those persons whom the state has determined, through examination, are capable of performing engineering designs. The further requirements that the person be proficient in the design, construction, operation, and reclamation of tailings basins, further limits the number of designers that would be acceptable to the commissioner. These requirements will ensure that only the most competent people will be designing these critical structures in Minnesota.

Subp. 2. B. This section contains a list of design specifications that the commissioner has determined must be provided in order to ensure reclamation, and assess the adequacy of the design. The topographic, hydrologic, and foundation conditions at the site of the tailings basin will essentially dictate the specific techniques and practices that will be incorporated into the ultimate design. It is therefore reasonable for the commissioner to require a submission of the rationale for site selection in order to make informed decisions regarding design approval.

Similarly, it is also reasonable for the commissioner to require descriptions of all materials that will be used, and explanations of any special construction or operating practices that must be employed. This information is essential to determining if the plan can be approved, and will be necessary for the commissioner to have, in order to adequately inspected the structure to ensure compliance.

Because the tailings basin forms an impoundment of water and tailings, it is imperative that an appropriate

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volume be maintained (often characterized by the "freeboard," a height between the free water elevation and the elevation of the top of the dam) within the basin, to hold waters that could result from excessive precipitation events. Some reviewers of the draft of these proposed rules have suggested that a specfic design precipitation event, called the "Probable Maximum Precipitation" (PMP) event, be utilized in establishing the required freeboard for the basin. In most cases, this design event will be the most severe and therefore the best selection, however there may be other events such as extended wet periods that may present even more concern, especially if there are restrictions on the discharge of water from the basin. It is reasonable that the commissioner avoid establishing a specific design precipitation event through rule, but to judge the adequacy of the the tailings basin design based on the specific conditions of the site, and the results of applying several hydrologic evaluations to the ultimate design.

Because of the importance of maintaining the structure in a safe condition it is necessary and reasonable that the designer be required to suggest the most effective means of providing reclamation of the tailings basin and the tailings dams.

The commissioner is also requiring the designer to maintain a presence beyond design preparation, by mandating scheduled inspections, to be conducted by the designer, throughout the permitted life of the facility, to ensure continued design compliance. This requirement is essential, since the designer will be most familiar with the rationale for the particular design, and should be able to assess whether the facility is performing successfully. To further ensure compliance, the commissioner is requiring that the designer propose monitoring locations at which information can be collected to accurately measure success of the facility.

This section also cautions that tailings basins containing reactive mines wastes may require other design criteria to be developed in accordance with the provisions of 6132.2200.

Subp. 2. C. This section recognizes that tailings basins are often segmented, with individual segments operated intermittantly. While segmenting the basin can be beneficial in providing progressive reclamation, it can also provide large areas of open tailings during periods of inactivity. Such areas can be sources of dust, therefore this rule is necessary to ensure that the generation of dust is controlled. The practices that are proposed for use are those currently practiced at other mine sites in Minnesota.

6132.2600 HEAP AND DUMP LEACHING FACILITIES

Subpart 1. Leaching is a hydrometallurgical process that ionizes metals contained within certain minerals. Once the leaching solutions, have liberated the metal ions from the mineral matrix, they transport the ions way from the remaining mineral constituents. The metallized leaching solutions are collected in ponds, then pumped to beneficiating plants where the metallic ions are eventually converted into solid metal.

The process of leaching metals from mineralized material has been conducted for decades. Initially the process was confined to leaching metals from ore concentrates. This practice is still conducted within ore beneficiating and treatment facilities, at many locations. However, the leaching of unprocessed ore, piled into "Heaps" or "Dumps," placed upon the surface, is becoming an increasingly popular method of metal extraction.

The terms heap leaching and dump leaching have at times been used almost interchangeably. The main difference seems to be, that with heap leaching the material to be treated was intentionally placed in a particular location for treatment, whereas dump leaching has often been conducted on stockpiled material that had earlier been considered uneconomic to process, and the leaching was performed more as an afterthought.

During the late 1970's techniques for processing low grade gold ore, using heap leaching techniques, were developed by the U. S. Bureau of Mines. This technology has now been applied at hundreds of locations worldwide, making the term heap leaching almost synonymous with gold ore processing. Both heap and dump leaching are however also utilized in the production of base metals. The leaching solutions vary depending on the metallic content of the ore: acids being used to produce base metals; and a variety of chemical solutions (but most commonly cyanide), for precious metals.

For the purpose of these rules, no differentiation has been made between heap and dump leaching because the impacts of either on the environment, if not properly controlled, would be devastating. This section of the rules gives direction on the factors that should be incorporated into heap and dump leaching designs and construction, in order to facilitate effective reclamation.

Subp. 2. A. Because heap and dump leaching requires the coordination of dams, ditches, high steeply sloped saturated stockpiles, impoundments of toxic and dangerous chemical solutions, pipes, pumps, and various pieces of highly technical equipment, all of which must be constructed atop an impermeable foundation to prevent spills from being released, the commissioner has determined that it is reasonable to require that heap and dump leaching facilities be designed and constructed only under the supervision of qualified experts. This rule therefore, requires that a professional engineer registered in Minnesota design heap and dump leaching facilities. This requirement is reasonable since it limits the designing to those persons whom the state has determined, through examination, are capable of performing engineering designs. The requirements that the person be proficient in the design, construction, operation, and reclamation of heap and dump leaching facilities, further limits the number of designers that would be acceptable to the commissioner. These requirements will ensure that only the most competent people will be designing these critical structures in Minnesota.

Subp. 2. B. This section contains a list of design specifications that the commissioner has determined must be provided in order to ensure reclamation and assess the adequacy of the design. The topographic, hydrologic, and foundation conditions at the site of a heap and dump leaching facility will essentially dictate the specific techniques and practices that will be incorporated into the ultimate design. It is therefore reasonable for the commissioner to require a submission of the rationale for site selection in order to make informed decisions regarding design approval.

The chemicals that make up the leaching solutions can be toxic and highly corrosive. Their impact on the environment could be devastating if they were released in an uncontrolled and untreated manner. This rule is therefore reasonable because it demands that techniques and practices be incorporated within the facility design to safely handle and appropriately dispose of the leaching solutions.

Heap and dump leaching facilities are usually designed with an impermeable membrane, often a synthetic (plastic) sheet, located directly beneath the mineralized ore that is being leached. The main purpose of this membrane is to facilitate an efficient and rapid collection of the leaching solutions that contain the metallic ions. A secondary purpose of the membrane is to help, along with other parts of the foundation, to keep the leaching solutions from leaving the facility. However, synthetic membranes are not totally impermeable, therefore the foundation beneath the membrane must prevent leaching solutions from reaching the natural environment.

It is reasonable for the commissioner to require descriptions of all materials that will be used, and explanations of any special construction or operating practices that must be employed, since this information is essential to determining if the plan can be approved, and will be necessary for the commissioner to have, in order to adequately inspected the structure to ensure compliance.

Because heap and dump leaching facilities include ponds to contain the leaching solutions, it is imperative that an appropriate volume be maintained (often characterized by the "freeboard," a height between the free water elevation an the elevation of the top of the dam) within the ponds, to hold the solutions and all additional waters that could result from excessive precipitation events. Some reviewers of the draft of these proposed rules have suggested that a specfic design precipitation event called the "Probable maximum precipitation" (PMP) event be utilized in establishing the required freeboard for the basin. In most cases this design event will be the most severe and therefore the best selection, however there may be other events such as extended wet periods that may present even more concern, especially since there are restrictions on the discharges from heap and dump leaching facilities. It is reasonable that the commissioner avoid establishing a specific design precipitation event through rule, but to judge the adequacy of the the heap and dump leaching design based on the specific conditions of the site, and the results of applying several hydrologic evaluations to the ultimate design.

Because of the toxic and corrosive nature of the chemicals used in leaching, it is essential that treatment to remove or render harmless all such chemicals occur when the process of active leaching has been completed. This rule ensures that the techniques and practices for accomplishing this requirement are incorporated into the design before the facilities are ever used.

The commissioner is also requiring the designer to maintain a presence beyond design preparation, by mandating scheduled inspections, to be conducted by the designer, throughout the permitted life of the facility, to ensure continued design compliance. This requirement is essential, since the designer will be most familiar with the rationale for the particular design, and should be able to assess whether the facility is performing successfully. To further ensure compliance, the commissioner is requiring that the designer propose monitoring locations at which information can be collected to accurately measurer success of the facility.

Subp. 2. C. This section cautions that if the neutralized and detoxified leached ore still constitutes a reactive mine waste, other design criteria may be required to be developed in accordance with the provisions of 6132.2200.

6132.2700 VEGETATION

Subpart 1. The result of mining activities is the creation of landforms and disturbed areas devoid of vegetation, which in turn is the potential cause of a number of adverse environmental consequences. The creation of unvegetated areas often causes air and water erosion, both of which have been, and occasionally continue to be in evidence near mining areas. Dry and unvegetated surfaces of tailings basins are easily erodable, and conditions similar to dust storms can periodically be observed on windy days in mining areas. Airborne particulates can cause several environmental and public health problems. Surfaces of slopes, such as storage piles and dams, are susceptible to water erosion, which can cause stream sedimentation and other water quality impacts downstream from the mining area. Water erosion can also endanger the structural stability of the facility being eroded. Natural vegetation provides wildlife habitat, and its removal from an area drasticallly alters the characteristics of the habitat. The removal or absence of vegetation limits the productivity of the area, from a land use standpoint.

The establishment of vegetation in on a drastically disturbed area returns stability to the land by helping to hold mineral soils in place. In addition, the initial planting of vegetation begins a soil building process that is essential to the ultimate survial of the more permanent vegetation that will be established on the site. The revegetation process also greatly expands land use options and enhances the aesthetic appeal of an area. This is particularly true in Minnesota, where unvegetated mining areas would be a major contrast to surrounding undisturbed areas. Forest growth, timber or crop production, and wildlife populations are functions of the type and quantity of vegetation.

The revegetation of created and disturbed areas serves two functions: The prevention or reduction of adverse environmental impacts; and the inherently beneficial aspects of vegetation itself, such as timber production, expended land use opertions, and aesthetic appeal. The goals stated in this subpart are merely a response to the functions served by requiring the vegetation of disturbed and created surfaces.

Subp. 2. A. This section contains a list of the areas that have been identified by the commissioner as needing vegetation. This list was developed based on observations by the department at various mine sites, created over the last century, along Minnesota's iron ranges. The commissioner's experience with enforcing reclamation rules at taconite mining operations over the past 12 years has demonstrated that the goals of this section can be achieved by providing revegetation at the specific sites listed in this subpart.

Subp. 2. B. It is reasonable to initiate vegetation as soon as is practical in order to receive its benefit as early as possible. The department's experience with enforcing the iron ore and taconite reclamation rules over the past 12 years has shown this requirement to be effective and has not placed an undue burden upon the operator.

Subp. 2. C. This section provides two uniquely different standards by which revegetation success will be measured. The first is directed at the initial establishment of vegetation designed to quickly stabilize the surface and begin the process of soil building on the site. The requirements that the area achieve a 90% cover and be free of gullies, is based on recommendations originally proposed by the Federal Office of Surface Mining, however, actual observations at Minnesota's taconite mining operations over the last 12 years also support the reasonableness of these requirements. The second standard is directed at the long term success of the vegetation. This requirement has, over the past 12 years of hands on experience, also proved to provide acceptable reclamation, while not unduly burdening the industry. Ample examples of successful revegetation, established at existing mine sites, are available to use as reference areas, by nonferrous mine operators.

6132.2800 DUST SUPPRESSION

Subpart 1. Dust is generated during a number of phases and in a number of locations of a mining operation, such as blastingwithin the mine pit, truck traffic on haul roads, and from the surfaces of storage piles and tailings basins. For numerous reasons such as: Protecting mining equipment from excessive wear; the need to meet increasingly stringent air quality emission standards; and being good neighbors to nearby residents, Minnesota's taconite industry has been employing a number of dust control techniques at their operations.

Subp. 2. The requirements of this section identify acceptable techniques in controlling dust emissions. To the extent that the manner in which dust is controlled in a mining area affects the overall reclamation and possibly the subsequent use of the mining area, it is reasonable that certain control techniques are preferable over others. All of the techniques identified have been variously and commonly used by mining companies to effectively control dust emmissions. They have been used with little or no environmental degradation as a consequence. On this basis, it is reasonable to require their use at those locations in a mining area where dust must be controlled. This list is not exclusive however, recognizing that other, perhaps as yet untested, techniques could be used as acceptable means of controlling dust.

6132.2900 AIR OVERPRESSURE AND GROUND VIBRATIONS FROM BLASTING

Subpart 1. Before rock and ore can be removed from the ground, they must be fractured and broken into pieces of a manageable size. This is accomplished by blasting. Blasting is an recurring procedure, the frequency of which depends on the quantities of material being mined, as well as, the size of area blasted, strength of the explosives used, the mining sequence, and other variables, and therefore varies from mining operation to mining operation.

Blasting at Minnesota's iron ore and taconite mining operations, when done in close proximity to residential areas, has created problems and conflict in the past. Property has occasionally been damaged as a result of blasting at mining operations in Minnesota, generally in the form of cracked foundations, plaster, or windows.

Some who have reviewed the draft of the proposed rules have questioned why an ephemeral procedure, such as blasting, is included in a rule that deals with reclamation. The reasons that it has been included are: 1) blasting has been demonstrated to have ocassionally caused structural damage, and such damage is likely to have a permanent impact, 2) in 1979 requests were made by local units of government for the commissioner to consider the inclusion of rules for blasting, and 3) Minnesota Statutes, section 93.47 Subd. 3, in part requires, "To the greatest extent possible, within the authority possessed by the commissioner, the rules so promulgated shall substantially comply with or exceed any minimum, mineland reclamation requirements which may be established pursuant to a federal mineland reclamation act." Regarding the last point, the federal government currently has no reclamation regulations that would apply to nonferrous metallic mining. However, when the iron ore and taconite rules were being adopted the federal government was in the process of adopting rules to direct the reclamation of coal mining in this country. These federal rules were carefully reviewed in 1979, and those parts which the commissioner felt might have applicability, in the event that the federal government were ever to regulate metal mines, were included in the state's reclamation rules. One such section was the regulation of blasting. In fact the standards that the commissioner ultimately adopted are exactly the same as that included in the federal rules.

There are two fundamental concerns related to blasting at mining operations: Air overpressure, the effects of which are expressed as airborne shock waves, including the actual sound of the blast; and ground vibration, the shock waves that travel through the ground. Both air overpressure and ground vibration can cause property damage.

This section expresses the prudent concept that injuries to the public should not be allowed to take place. Additionally, it is reasonable to regulate blasting activities to prevent damage to property.

Subp. 2. A. (1) is identical to the corresponding standard of the federal rules relating to the reclamation of surface coal mine lands. The upper limit of 130 decibels is based on experimentation conducted and field data collected by the United States Bureau of Mines, and represents the point above which structural damage begins to occur. The scale frequency band sensitivity range of 6 to 200 cycled per second is also based on U.S. Bureau of Mines data. Six cycles per second represents the reliable lower limit of measuring equipment. Energy above 200 cycles per second, resulting from blasting operations, does not cause structural damage

Subp. 2. A. (2) requires open pit mine operators to monitor their blasting. This is necessary to determine compliance with the blasting standards. The requirement to place blast monitoring stations adjacent to structures located nearest the blast, is based on the reasonable assumption that if structural damage were to take place, it would, in most cases, take place at such structure. It is also reasonable for the commissioner to require additional monitoring, beyond such location, if complaints are received.

Subp. 2. A. (3) requiring an open pit mine operator to keep a blasting log is necessary if complaints relating to blasting are received and must be investigated. The informational requirements would provide data to allow the reason for the condition causing the complaint to be determined, and would further provide the basis for any necessary modification of the blasting. Since mining companies routinely maintain a record containing similar information, this requirement is not onerous or unreasonable. The basis for requiring that the log be kept for at least six years is that if complaints are received, this is a reasonably long period for which the records can be evaluated to determine the existence and extent of any past problems and whether the permittee has been conducting blasting in an appropriate manner.

Subp. 2. A. (4) is designed to prohibit open pit blasting when a condition exists such that blasting could endanger the public health and welfare. Since such conditions would exist only temporarily, it is reasonable to prohibit blasting until the condition passes. This is also standard current mining practice.

Subp. 2. A. (5) limits open pit blasting to daylight hours, again, a standard mining practice today. This is designed so that the period when most of the human population is sleeping is not disrupted by blasting.

Subp. 2. B. (1) is identical to the corresponding standard of the federal rules relating to the reclamation of surface coal mine lands. The upper limit for the peak particle velocity, of one inch per second, is based on experimentation conducted and field data collected by the United States Bureau of Mines, and represents the point above which structural damage begins to occur.

Subp. 2. B. (2) The requirement that production blasts be monitored using a seismograph, that measures three mutually perpendicular peak particle velocities, is required because the use of such an instrument is the only way to directly determine the magnitude of the ground vibrations resulting from a blast.

Subp. 2. B. (3) This part requires mine operators to monitor their blasting to determine the extent of ground vibration. This is necessary to determine compliance with the blasting standards. The requirement to place blast monitoring stations adjacent to structures located nearest the blast, is based on the assumption that if structural damage were to take place, it would, in most cases, take place at the nearest structure. It is also reasonable for the commissioner to require additional monitoring, beyond such location, if complaints are received.

Subp. 2. B. (4) The blasting that is conducted at an underground mine is significantly different than that conducted at an open pit operation. Air overpressure will be confined to the underground working and will not be perceptible at the surface, except perhaps right at the mine opening. The blasts are much smaller, not providing the energy that would generally cause ground vibration concerns, however to verify this, the proposed rules do require ground vibration monitoring in the event of complaints. Because of the differences between surface and underground mining the necessity that an underground mine operator maintain a blasting log is not required unless the commissioner determines, based on experience at the site, that it is necessary, to investigate complaints. If complaints are received, the informational requirements of the blasting log would provide data to allow the reason for the condition causing the complaint to be determined, and would further provide the basis for any necessary modification of the blasting.

Subp. 2. C. This requirement provides for the availabilty of blast monitoring data during reasonably long period for which the records can be evaluated to determine the existence and extent of any past problems and to determine whether the permittee has been conducting blasting in an appropriate manner. The data supplied through such monitoring is important in determining the degree to which blasting may have caused damages, and is therefore important and reasonable to require.

6132.3000 SUBSIDENCE

Subpart 1. This section addresses surface displacements resulting from slumping and subsidence. Slumping is generally defined as the failure or collapse of the surface of a slope, such as along the crest of an open pit. Subsidence is the sinking, gradual or sudden, of the ground surface overlying an underground mine. These ground movements can create very hazardous conditions and cause significant property damage. Minimizing such conditions is prudent.

Subp. 2. This section lists requirements that must be taken to minimize the impacts caused by subsidence, through utilization of techniques and practices that reduce the potential for subsidence. This is accomplished by requiring mine openings and supporting structures to be designed in a manner that considers the strength and ability of the rock to sustain such openings. In the event that subsidence can not be avoided, in spite of the use of appropriate designs, it is reasonable to require that monitoring be performed to determine the location and degree to which subsidence is occuring, in order to determine its extent and the degree to which mitigation must be conducted. The mitigation practices that are listed in this section are those normally taken in such cases to protect the public and natural resouces.

6132.3100 CORRECTIVE ACTION

Subpart 1. Because of the many requirements and complexities of these rules, operators may occasionally be out of compliance. This section addresses this situation, describing what is expected of a permittee, and giving direction on how to accomplish corrections. It is reasonable to require that as soon as a violation is observed, appropriate notices be given and planning of the correction, or actual corrective action, be undertaken.

Subp. 2. A. The purpose of this section is to help maintain a dialogue between the commissioner and the permittee, during which any problems that would lead to a non-compliance, or an actual violation, can be discussed and corrective action taken before the problem increases in magnitude. The commissioner will make frequent inspections and promptly notify the permittee of any violations, so that corrections can be made in a timely manner.

Subp. 2. B. At the point when a violation is identified, it is reasonable to assume it will be corrected or that appropriate plans are made to effectuate the correction. Immediate corrective action is reasonable to expect for violations that that do not require substantial planning or that are not dependent upon special weather conditions, such as an appropriate planting season. It is conceivable that corrective actions may be complicated and require engineering review and planning. This section requires that such planning not be delayed. The two week requirement is considered by the commissioner to be an appropriate period to have an initial review of the situation completed and corrective action plans either developed or a schedule for their development identified. This time period closely corresponds to and complements a period, established by Minnesota Statute, section 93.481 subd. 4, that allows the permittee at least 15 days to take actual corrective action, in the event of a breach of the permit to mine or the law, before the commissioner can take actions to revoke or modify the permit. The specific details of the plan are necessary in order for the commissioner to assess viability of the corrective action, and to ensure that appropriate funding will be available in the event the permittee is not able to complete the corrections.

Subp. 2. C. When there is an immediate threat to human safety and natural resouces it is reasonable to require that corrective actions be taken immediately to minimize the threat, and then report to the commissioner, as soon as possible.

Subp. 2. D. It is reasonable for the commissioner to identify procedures that are available to the commissioner to ensure reclamation success. This rule emphasizes that the commissioner will use all procedures provided by statute to keep the mine area in compliance with these rules and the permit to mine.

6132.3200 CLOSURE AND POST CLOSURE MAINTENANCE

Subpart 1. This section recognizes that at some point mining activities will cease, either temporarily or permanently, and there will be a significant reduction in the presence of the permittee at the site. Therefore, it is reasonable that the rules identify the reclamation activities and accomplishments that must be completed and maintained, in order to ensure that the area remains in compliance with the rules. Freedom from maintenance, is an overall goal of these rules. Past experience and observation by the commissioner, at former mining operations in Minnesota, has indicated that the permittee will want to have as little ongoing involvement with the site as practical, once the operations cease. Likewise, the commissioner wants to limit any future public responsibility in the event that the lands revert to the state through tax forfeiture, as has been the past experience with some mine sites.

Subp. 2. A. This section requires that the commissioner be informed when operations will cease, either temporarily or permanently. This requirement is reasonable because many of the reclamation requirements are triggered by cessation of the ongoing operations.

Subp. 2. B. Several pieces of information must be submitted when an operator plans to temporarily cease operations. Information regarding the reasons for shutdown, anticipated length, and plans for how the mine site will be maintained, are necessary in order for the commissioner to determine whether it is reasonable to believe that the shutdown is temporary, or if the operator should be required to implement plans to permanently reclaim the area. The remainder of the subpart details the requirements that must be implemented under a temporary shutdown. These requirements are needed to ensure compliance with the permit, in the event the temporary shutdown becomes permanent.

Subp. 2. C. This section outlines the options available to the commissioner after reviewing a proposal made by an operator to temporarily cease operations. Because the commissioner is responsible for ensuring that reclamation of a mine site is accomplished, the determination of whether a shutdown is indeed temporary, or has the potential to become permanent, is important and must be made with a full understanding of the situation.

Subp. 2. D. Extension of the temporary shutdown approval may be sought if the factors that led to the initial shutdown are not resolved within the time anticipated. If this were to occur the commissioner would have to determine if an extension is reasonable or whether the operator should be required to implement permanent closure plans for the mining area. The information required in this subpart will help evaluate the physical status of the site to determine if it has deteriorated during the temporary shutdown and whether it can be maintained during the period for which the extension is being sought.

Subp. 2. E. When the operator determines that operations will permanently cease, the contingency reclamation plan which describes the activities necessary to bring the operation into compliance with the reclamation rules and the permit to mine, must be implemented. The remainder of this subpart establishes the time limits for accomplishing the various reclamation activities in order to provide adequate protection of the public and to ensure a prompt closure of the site.

Barriers and other safety precautions must be established at underground openings and around open pits, very quickly after shutdown, to prevent attractive nuisances and hazards to the public.

Debris and mobile equipment are generally easy to remove. If these are not necessary for the support of reclamation activities, they should be removed in a short period, so that these too do not become attractive nuisances.

A longer period has been allowed for the removal of the remaining stationery structures within the mine site. Based upon the commissioner's experience with the closure of other mine areas, if closure is allowed to linger beyond a three year period, it is difficult to bring closure to a conclusion.

Three years is also the allowable period to integrate basins back into the existing watershed. This has proven to be reasonable with the closure of other mine areas in Minnesota, since it often takes a season or two of experience, after the original contouring of a site, in order to fine tune the various water control structures that may be necessary.

This section also addresses the subject of continued maintenance. Although this is not the most desirable means of providing for the reclamation of a mining area, it may be the only means available to ensure that the reclamation requirements will continue to be met after operations cease. When continued maintenance of an area is necessary, it is reasonable that the commissioner be provided with the information necessary to evaluate the proposal, to appraise the ongoing success of the activities that are proposed, and to determine if appropriate levels of funding will be available to support the efforts.

It is reasonable that in the event continued maintenance is necessary, the responsibility remain with the permittee until such time as the maintenance is determined to be no longer necessary. This rule ensures that such responsibility is maintained by withholding release from the permit, on those parts of the mine area that require continued maintenance.

6132.4000 PROCEDURES FOR OBTAINING A PERMIT TO MINE

Subpart 1. Minnesota Statutes, section 93.491 requires that anyone wishing to engage in mining must first obtain a permit to mine. This section requires that a preapplication conference and site visit be conducted in order for the commissioner and the applicant to discuss the proposed project; to determine the degree of detail for the information that will be required for application submission; to outline procedures that must be followed relative to the permit to mine, as well as other permits, authorizations, and data submissions that are required; and to identify the best means of informing and involving the public in the decisions that will be made regarding the proposed operation. Based on this conference, a permit application in sufficient detail to allow adequate consideration, must be filed with the commissioner. To properly satisfy the statutory requirement of publication, and to insure that the commissioner is well informed and prepared for public inquiries, the application should be submitted and approved as adequate prior to publication. A copy of the advertisement should then be submitted to the commissioner along with affidavits of publication within seven days following the final publication, a reasonable time to allow the commissioner to make a determination of compliance with statutory requirements as early in the process as possible.

Subp. 2. A. Minnesota Statutes, section 93.481, Subd. 2, provides for an optional hearing to be held upon receipt of objections from certain specified parties. This section identifies the period, as specified in the statute, during which the objections will be accepted.

Subp. 2. B. The information required will allow the commissioner to determine if the person objecting meets the statutory requirements of Minnesota Statutes, section 93.481, Subd. 2, or raises an issue relating to the proposal over which the commissioner has jurisdiction, and whether holding a hearing might potentially resolve the objection.

Subp. 2. C. This section identifies the procedure that the commissioner will follow upon receipt of objections. If the individual filing the objection meets either of the first two criteria listed in this section, the statute dictates that a hearing be conducted. The commissioner has determined that an additional criterion be added to those listed in statute, in order to aid in determining if factors exist, over which the commissioner has jurisdiction, that have not been adequately addressed through the permit application. It is reasonable to add the last criteria to ensure that all factors related to the proposal will be addressed.

Subp. 2. D. This section identifies the process the commissioner will follow in trying to resolve the issues identified by persons meeting the criteria established in subpart 2. C. The commissioner has determined that it is reasonable and may be more effective to try to resolve the objections directly between the applicant and the objector before arranging for a hearing. This may save time and expense, and still reasonably resolve the problem. If differences can not be so resolved, a hearing will be conducted in accordance with subpart 3.

Subp. 2. E. This section deals with the eventuality that the individual objecting is unable to show how the mining operation would impact them or can not identify facts about the proposal over which the commissioner has jurisdiction. In this event it is reasonable for the commissioner to inform the individual of this decision, and proceed with processing the permit application.

Subp. 3. A. This section identifies three circumstances under which a hearing would be held. The first is when valid objections are received, and the commissioner is unable to directly resolve the differences between the applicant and the objector.

The second reason that the commissioner would hold a hearing would be if it was determined that special conditions should be incorporated into the permit in order to satisfy problems or conditions that were not resolved to the commissioner's satisfaction in the permit application. Since discussions regarding such problems and conditions will certainly be discussed well before a final submission of the application, this situation would only arise if the applicant and the commissioner could not resolve the situation. In such a case it is reasonable to hold a hearing in order to resolve the differences.

The final reason for conducting a hearing would be that the commissioner determines a permit can not be issued. It is more reasonable and efficient for the commissioner to order a hearing before denying the permit.

Subp. 3. B. This section contains the specifications that the commissioner will follow to initiate the hearing process. The process will be conducted by an administrative law judge assigned by the Office of Administrative Hearings, following the schedules and notification requirements of Minnesota Rules Chapter 1400.5600. Specific details have been incorporated into the proposed rules to compensate for differences between the Chapter 1400.5600 rules and the requirements of Minnesota Statutes, sections 93.44 through 93.51.

The requirement of mailing individual notices to the applicant, any objectors to the permit application, and the local units of government is reasonable, to ensure that these all have an opportunity to be heard.

An advertisement, of the notice of the hearing, is required to be published in the same local newspaper that was earlier used to publish the notice that an application for a permit to mine was being sought. This requirement maintains consistency in order to keep local individuals, who may be affected by the operation, informed about the status of the mining and permitting activities.

Subp. 3. C. This requirement maintains consistency with the statutory requirement that the commissioner conclude deliberations on a permit application within 120 days after holding a hearing. The administrative law judge is required to report findings to the commissioner within 30 days following close of a hearing, therefore allowing 90 days for the commissioner to make a final decision will reasonably allow the statutory time limits to be met.

Subp. 4. This section provides a process for approving a permit, within the time limit established by statute, when the conditions that would otherwise mandate a hearing do not exist.

Subp. 5. This section addresses the commissioner's review of the annual report submitted by the permittee, describing progress on activities conducted under a permit to mine. The document will contain a report of the preceding year's activities and a projection of how operations will be conducted during the upcoming year. Since mining is a very dynamic activity it is possible that actual mining may proceed differently than may have originally been planned. It is reasonable for the commissioner to compare what has, or will soon be occuring, with plans on which permit approval was based. This section identifies possible actions available to the commissioner to either acknowledge when the report shows compliance, or to direct the permittee on how to remain in, or reestablish compliance.

6132.4100 VARIANCES

Subpart 1. Pursuant to Minnesota Statutes, section 93.48, variances from the rules may be allowed, upon application therefor, if the variance is consistent with the general welfare. The commissioner has determined that the general welfare is reasonably served if the proposed alternative to the rule is directed toward the attainment of the goals expressed in the rules, and it can be demonstrated that the alternative will perform as well or better than the prescribed rule. In addition, it is reasonable to require the applicant to demonstrate that the prescribed rule is burdensome, especially if the proposed alternative is not superior to the rule it is designed to replace.

Subp. 2. To initiate the variance process, the commissioner must be provided with an application containing sufficient information to make the statutory determination. It is conceivable that some requests for variance could have a significant impact on natural resources and the public. If this were the case it would be reasonable for the commissioner to require that the applicant for such a variance, publish a notice in the local newspaper in the same manner as is required when there is an application for a permit to mine. However, for those variance applications which propose minor deviations from the rules or which are unlikely to cause great impacts, such public notice is not reasonable or necessary.

To resolve the decision of whether or not to require publication of a notice, this rule states that the commissioner, upon receipt of a variance request, will first determine whether the proposal constitutes a substantial change from the requirements of these rules. If the changes are substantial, the request will be treated in the same manner as if it were an application for a permit, including the mandatory notice publication and the possibility of holding a public hearing. When the changes are not substantial the commissioner will make the decision on the acceptability of the variance without notice, or need for hearing.

Some commenters who reviewed drafts of these proposed rules have questioned what constitutes a substantial change, and whether the commissioner should have the power to make such decisions without some sort of public involvement. The concept of allowing the commissioner to make decisions, based on substantial change, has its basis in Minnesota Statute, section 93.481 Subd. 3, which specifically addresses the issue of publishing notices, when amendments to a permit to mine are requested by a permittee. Since variances and amendments both constitute a deviation from the norm, it is reasonable that both should be treated in the same manner, and that both follow the process established by statute.

No clear direction was given by the statute on what constitutes a substantial change. It would be reasonable however to include: 1) proposals that would result in a major departure from what would normally occur, if the rules were followed; 2) proposals that would result in impacts occurring at greatly different times or places than would otherwise normally be expected; and 3) proposals where the commissioner feels that the professional expertise in the various disciplines held by the Department of Natural Resources' staff could be augmented by input of the public.

Subp. 3. The proposed rules acknowledge the possibility that a variance may be submitted at the same time as an application for a permit to mine. This rule requires that information about the variance request be inserted into the notice publication of the permit to mine, in order that the public is aware of the variance request.

6132.4200 Amendments

Subpart 1. Pursuant to Minnesota Statutes, section 93.481, subd. 3., an amendment to the permit to mine may be allowed upon application therefor, if the commissioner determines that all lawful requirements are met.

Subp. 2. Minnesota Statutes, section 93.481, subd. 3. also requires the commissioner to make a determination regarding whether the proposed amendment constitutes a substantial change to the permit to mine. In accordance with the statute, when the change is substantial, the person proposing the amendment must publish a notice of the amendment in the same manner as for a new permit. For the purposes of this section a substantial change shall be considered to occur when the proposed amendment: 1) would result in a major departure from what would normally occur, if the approved permit were followed; 2) would result in impacts occurring at greatly different times or places than would otherwise be expected; and 3) requires expert analysis, that the commissioner feels is unavailable within the department, and input of the public would be helpful.

When the change is not substantial and the commissioner determines the all lawful requirements will be met the amendment will be granted.

6132.4300 Modification of Permit to Mine

Pursuant to Minn. Stat., sec. 93.481, subd. 4., modification of the permit to mine is a procedure available to the commissioner, when it becomes necessary to change the permit to mine in order to resolve a previously unforeseen problem or situation, and all other means of cooperatively working with the permittee have not resolved the disagreement. It is necessary for the commissioner to have the ability to require changes in order to protect persons or property from damage as a result of the permittee's unwillingness to rectify a problem. It is reasonable that the commissioner utilize a process that includes a public hearing, in order to establish the facts that will support the commissioner's findings and decision on the content of the ultimate order for modification.

6132.4400 Cancellation

Pursuant to Minn. Stat. sec. 93.481, subd. 4., cancellation is a procedure where the commissioner and the permittee mutually agree that a permit to mine will no longer be valid. This process will only be utilized if the permittee is unable to begin a project for which a permit has been issued, and any development work that may have begun has not resulted in the need for reclamation, or any necessary reclamation has been completed. This procedure is reasonable because it will bring to conclusion a permit to mine when both the permittee and the commissioner agree that the permit is no longer necessary and that all obligations of the permit have been satisfied.

6132.4500 Suspension of Permit to Mine

Pursuant to Minn. Stat. sec. 93.481, subd. 4., suspension is a procedure where the commissioner requires the cessation of mining in order to prevent the operation from causing or enhancing a situation that endangers humans, natural resources, or property, and all other means of cooperatively working with the permittee to resolve the conditions causing the danger have failed. It is necessary for the commissioner to have the ability to require cessation of the operation, in order to prevent damage or injury that may occur as a result of the permittee's inability or unwillingness to otherwise rectify a problem. It is reasonable, in an emergency, that the commissioner have the ability to promptly suspend an operation for a short period of time, without the need for a public hearing. However, if the commissioner determines that it would be necessary to extend the suspension of the permit for a longer period, it is reasonable that a public hearing be conducted in order to establish the facts that will support the commissioner's decision to require an extended suspension.

6132.4600 Revocation of Permit to Mine

Pursuant to Minn. Stat. sec. 93.481, subd. 4., revocation of the permit to mine is a procedure available to the commissioner when it becomes necessary to end a permit to mine, in order to resolve a previously unforeseen problem or situation, and all other means of cooperatively working with the permittee have not resolved the disagreement. It is necessary for the commissioner to have the ability to revoke a permit in order to protect persons or property from damage that may occur as a result of the permittee's unwillingness to rectify a problem. It is reasonable that the commissioner utilize a process that includes a public hearing, in order to establish the facts that will support the commissioner's decision.

6132.4700 Assignment

Pursuant to Minn. Stat. sec. 93.481, subd. 5., assignment of the permit to mine is a procedure available to the commissioner when it is reasonable to transfer the permit to mine from one person to another. Before allowing the transfer, it is reasonable and necessary that the commissioner be assurred that all reclamation requirements will be met.

6132.4800 Release of Permittee

Release is a process by which a permittee's obligations under the permit can be ended. It is reasonable, that at some point following the cessation of mining, permit obligations can be ended if it can be satisfactorily demonstrated that all requirements of the permit have been met. This section recognizes the possibility that the commissioner and the permittee may not agree on the facts regarding release, and therefore establishes a process by which a public hearing can be held to determine the facts.

6132.4900 Publication

This section describes the process by which the public is to be officially informed of the various procedures, associated with a permit to mine, that require decision making by the commissioner. It is reasonable that the commissioner require the publication of the information listed in the section, in order that the public be informed of the extent of the activities proposed, and the rights that the public possesses with regard to its role in the decisionmaking process. The time limits and number of publications of notice that are contained in the proposed rule are consistent with the requirements of Minn. Stat., section 93.481, subdivisions 1 and 2.

6132.5000 Hearing Procedures

Minnesota Statutes, section 93.50 provides that any person aggrieved by any order, ruling, or decision of the commissioner may appeal such order, ruling, or decision in the manner provided in chapter 14., the portion dealing with contested case hearings The proposed rule identifies those portions of the Minnesota Rules, governing the activities of the Office of Administrative Hearings, that will be used to conduct the contested case hearing. The proposed rule recognizes that protions of parts 1400.5100 to 1400.8500, dealing with public notice time limits, will be dictated by Minn. Stat., secs. 93.44 to 93.51 rather than parts 1400.5100 to 1400.8500.

6132.5100 Civil Penalties

If any person violates the statutes, rules or any permit condition, Minn. Stat., sec. 93.51 allows the commissioner to assess fines of up to $1,000 per day for any such violation after notice and a period of not less that 15 days for correction. Such fines should apply to each separate occurance of each violation since such violation could by so numerous and widespread that their economic benefit would outweigh a single such fine. Fines are assessed to deter future violations especially those which are particularly aggregious. Thus the commissioner, in determining the amount up to the $1,000 statutory maximum, should consider the severity of the violation and the magnitude of potential or actual economic gains resulting therefrom so that the fine will act as a deterent.

6132.5200 Inspection of Mining Area

In order to properly enforce these rules and the law, it is reasonable that the commissioner must have the authority to inspect those portions of the mining operations and necessary records relevant to such enforcement.

6132.5300 Wetland Mitigation and Replacement Procedures

This section is inserted into these rules as a result of the requirements of Minn. Stat., sec. 103G.222, that prohibits any person, conducting activities under the authorization of a permit to mine, from draining or filling wetlands without first obtaining approval of the commissioner. In accordance with this statute, approval is to be based upon a mining and reclamation plan that incorporates the same principles and standards for wetland impact mitigation and replacement, found in the rules for wetland value replacement plans, promulgated pursuant to Minn. Stat., sec. 103G.2242.be affected by mining.

EXPERT WITNESSES APPEARING ON BEHALF OF THE DEPARTMENT

The department will supplement testimony by it own professinal staff with expert testimony by the following individuals:

Victoria J. Bryan
Reclamation Bonding Specialist

Department of the Interior
Office of Surface Mining
1020 15th Street
Denver CO 80202

Victoria Bryan will supplement the department's testimony on financial assurance for mining. In this capacity, she will discuss her own experience with administering the financial assurance program for coal mining in the context of the requirements of the department's Nonferrous Metallic Mineral Mineland Reclamation Rules.

Andrew MacG. Robertson, Ph.D., P. Eng.
Principal

Steffen, Robertson and Kirsten
Consulting Engineers
Suite 800
580 Hornby Street
Vancouver, B.C. V6C 3B6
CANADA

Dr. Robertson will supplement the department's testimony on mitigation of impacts associated with base and precious metal mining, particularly of sulfide ores. In this capacity, he will discuss his own experiences with mitigation of acid mine drainage in the context of the requirements of the department's Nonferrous Metallic Mineral Mineland Reclamation Rules.

Richard Lawrence, Ph.D.
Chair in Mining and Environment

Department of Mining and Mineral Process Engineering
University of British Columbia
6350 Stores Road
Room 517
Vancouver, B.C. V6T 1Z4
CANADA

Dr. Lawrence will supplement the department's testimony on mine waste characterization and prediction of mine waste drainage quality. In this capacity, he will discuss his own experience with the prediction of waste material leachate quality in the context of the requirements of the department's Nonferrous Metallic Mineral Mineland Reclamation Rules.

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Terry Mudder, Ph.D.
Corporate Consultant

Times Limited
15311 N.E. 90th Street
Redmond, WA 98052

Terry Mudder will present testimony on the use and management of cyanide in the mineral mining and processing industry. In this capacity, he will draw upon his own experience as consultant and project manager for waste water treatment systems developed for the mining industry of the U.S. and Canada.

POLYMET
MINING

Updated NI 43-101 Technical Report on the NorthMet Deposit

Minnesota, USA

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Report Date:	October 12, 2012
Amended Report Date:	January 14, 2013
Resource Data Cut-off Date:	October 13, 2007
Resource Effective Date:	February 20, 2008
Reserve Effective Date:	September 26, 2007

AGP
Mining Consultants Inc.

IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report for PolyMet Mining Corp. (PolyMet) by AGP Mining Consultants Inc. (AGP). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in AGP's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by PolyMet subject to the terms and conditions of its contract with AGP. This contract permits PolyMet to file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk.

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GLOSSARY

Abbreviations, Symbols, and Acronyms

AGP Mining Consultants Inc. .... AGP
Anglesite .... $PbSO_4$
Argentite .... Ag_2S
Canadian Institute of Mining .... CIM
Cerargyrite .... AgCl
Cerussite .... $PbCO_3$
Chalcopyrite .... $CuFeS_2$
Defiance Silver Corp. .... Defiance
Federal Official Gazette .... FOG
Foreign Investment Law .... FIL
Freibergite .... $(Ag,Cu,Fe)_{12}(Sb,As)_4S_{13}$
Galena .... PbS
Gold Equivalent .... Au
Ground Penetrating Radar .... GPR
Microsoft Excel spreadsheets .... XLS
Native Silver .... Ag
Proustite .... Ag_3AsS_3
Quality Assurance/Quality Control .... QA/QC
Polymet Mining Corp. .... Polymet
Silver Equivalent .... AgEq
Specific Gravity .... SG
Sphalerite .... (Zn,Fe)S
Standard Reference Material .... SRM

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Standard Reference Materials ....SRM
Standard Resources Inc...Silver Standard
Sterling Mining Company of Idaho ....Sterling
Two Dimensional...2D
Three Dimensional 3D
Transient Electromagnetic Method ....TEM

UNITS OF MEASURE

Above mean sea level	amsl
Acre	ac
Ampere	A
Annum (year)	a
Billion	B
Billion tonnes	Bt
Billion years ago	Ga
British thermal unit	BTU
Centimetre	cm
Cubic centimetre	cm3
Cubic feet per minute	cfm
Cubic feet per second	ft3/s
Cubic foot	ft3
Cubic inch	in3
Cubic metre	m3
Cubic yard	yd3
Coefficients of Variation	CVs
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Decibel adjusted	dBa
Decibel	dB
Degree	°
Degrees Celsius	°C
Diameter	ø
Dollar (American)	US$
Dollar (Canadian)	C$
Dry metric ton	dmt
Foot	ft

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Gallon	gal
Gallons per minute (US)	gpm
Gigajoule	GJ
Gigapascal	GPa
Gigawatt	GW
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m2)	ha
Hertz	Hz
Horsepower	hp
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Inch	"
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre	kg/m3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m2
Kilometre	km
Kilometres per hour	km/h
Kilopascal	kPa
Kilotonne	kt
Kilovolt	kV
Kilovolt-ampere	kVA
Kilovolts	kV
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Less than	<
Litre	L
Litres per minute	L/min
Megabytes per second	Mb/sec
Megapascal	MPa

Megavolt-ampere	MVA
Megawatt	MW
Metre	m
Metres above sea level	m asl
Metres Baltic sea level	m bsl
Metres per minute	m/min
Metres per second	m/s
Metric ton (tonne)	t
Microns	µm
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million	M
Million bank cubic metres	Mbm3
Million tonnes	Mt
Minute (plane angle)	'
Minute (time)	min
Month	mo
Ounce	oz
Pascal	Pa
Centipoise	mPa·s
Parts per million	ppm
Parts per billion	ppb
Percent	%
Pound(s)	lb
Pounds per square inch	psi
Revolutions per minute	rpm
Second (plane angle)	"
Second (time)	sec
Specific gravity	SG
Square centimetre	cm2
Square foot	ft2
Square inch	in2
Square kilometre	km2
Square metre	m2
Thousand tonnes	kt
Three Dimensional	3D

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Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Tonnes seconds per hour metre cubed	ts/hm3
Total	T
Volt	V
Week	wk
Weight/weight	w/w
Wet metric ton	wmt

ABBREVIATIONS AND ACRONYMS

Absolute Relative Difference	ABRD
Acid Base Accounting	ABA
Acid Rock Drainage	ARD
Alpine Tundra	AT
Atomic Absorption Spectrophotometer	AAS
Atomic Absorption	AA
British Columbia Environmental Assessment Act	BCEAA
British Columbia Environmental Assessment Office	BCEAO
British Columbia Environmental Assessment	BCEA
British Columbia	BC
Canadian Dam Association	CDA
Canadian Environmental Assessment Act	CEA Act
Canadian Environmental Assessment Agency	CEA Agency
Canadian Institute of Mining, Metallurgy, and Petroleum	CIM
Canadian National Railway	CNR
Carbon-in-leach	CIL
Caterpillar's® Fleet Production and Cost Analysis software	FPC
Closed-circuit Television	CCTV
Coefficient of Variation	CV
Copper equivalent	CuEq
Counter-current decantation	CCD
Cyanide Soluble	CN
Digital Elevation Model	DEM
Direct leach	DL
Distributed Control System	DCS
Drilling and Blasting	D&B

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Environmental Management System	EMS
Flocculant	floc
Free Carrier	FCA
Gemcom International Inc.	Gemcom
General and administration	G&A
Gold equivalent	AuEq
Heating, Ventilating, and Air Conditioning	HVAC
High Pressure Grinding Rolls	HPGR
Indicator Kriging	IK
Inductively Coupled Plasma Atomic Emission Spectroscopy	ICP-AES
Inductively Coupled Plasma	ICP
Inspectorate America Corp.	Inspectorate
Interior Cedar – Hemlock	ICH
Internal rate of return	IRR
International Congress on Large Dams	ICOLD
Inverse Distance Cubed	ID3
Land and Resource Management Plan	LRMP
Lerchs-Grossman	LG
Life-of-mine	LOM
Load-haul-dump	LHD
Locked cycle tests	LCTs
Loss on Ignition	LOI
Metal Mining Effluent Regulations	MMER
Methyl Isobutyl Carbinol	MIBC
Metres East	mE
Metres North	mN
Mineral Deposits Research Unit	MDRU
Mineral Titles Online	MTO
National Instrument 43-101	NI 43-101
Nearest Neighbour	NN
Net Invoice Value	NIV
Net Present Value	NPV
Net Smelter Prices	NSP
Net Smelter Return	NSR
Neutralization Potential	NP
Northwest Transmission Line	NTL
Official Community Plans	OCPs
Operator Interface Station	OIS

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1 SUMMARY

This report describes the results of a mineral resource estimation update of the NorthMet Project, which includes the NorthMet polymetallic copper-nickel-cobalt-platinum group element (Cu-Ni-Co-PGE) Deposit (the "NorthMet Deposit") leased by PolyMet Mining, Inc., and the Erie Plant, both owned by a wholly-owned subsidiary of PolyMet Mining Corp. (together with PolyMet Mining, Inc. "PolyMet"), a Canadian corporation. This revision and update of the 2007 National Instrument 43-101 (NI 43-101) compliant Resource report (Wardrop, September 2007) and the 2006 NI 43-101 compliant Feasibility report (Hunter, 2006) is based on the inclusion of results from 31 additional diamond drill holes completed between March 2007 and July 2007.

This report is updated from earlier reports, namely Wardrop September 2007, Hellman 2005 and 2006, and Hunter, 2006, all of which made extensive reference to Hammond, 2005, and Patelke and Geerts, 2006. All references to resource evaluation are based on current PolyMet data; reference herein to historical information is updated from these earlier reports.

This report has been prepared in order to incorporate reserves previously reported by PolyMet, complete the resource estimation, and comply with revised form of NI 43-101. Once PolyMet has finalized detailed engineering that will be set out in the NorthMet Enviromental Impact Statement (EIS), the company plans to issue an updated Technical Report, which will incorporate capital and operating costs, as well as current metal markets.

This new resource estimate by AGP Mining Consultants Inc. (AGP) incorporates the 2007 drilling results that were available as of October 15, 2007, this includes all drilling done through the end of July 2007, specifically, through hole 07-570C. The block model matrix dimension and the interpolation parameters remained the same as the September 2007 report, which included an extension of the block model matrix down to the 0.00 ft elevation. A smaller block size was used than in the definitive feasibility study (DFS) based upon a selective mining unit determination.

Since the 2007 mineral resource and reserve calculations, PolyMet has made two changes to the operating plans.

First, in May 2008 PolyMet revised the plans to include:

- The sale of concentrate during the construction and commissioning of new metallurgical facilities resulting in a shorter pre-production construction period (under twelve months) and reduced capital costs prior to first revenues ($312 million versus $380 million).
- Mine plans reflect the increase in reserves and decrease in stripping ratio reported on September 26, 2007, the use of 240-ton trucks, and owner versus contract mine operations.

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- On an equivalent basis, capital costs increase 36% to $517 million. In addition, the revised plan included an additional $85 million in measures to protect the environment, increasing the total capital to $602 million.
- Staged construction reduces pre-production capital costs to $312.3 million (including the additional environmental measures) with most of the additional $289.6 million for construction of the metallurgical facilities expected to be funded from operating cash flow.

In February 2011, PolyMet reported a further simplification whereby it would build the Project in two phases:

- Phase I: produce and market concentrates containing copper, nickel, cobalt, and precious metals.
- Phase II: process the nickel concentrate through a single autoclave, resulting in production and sale of high grade copper concentrate, value added nickel-cobalt hydroxide, and precious metals precipitate products.

The changes reflected continued metallurgical process and other project improvements as well as improved environmental controls that are being incorporated into the Supplemental Draft Environmental Impact Statement (SDEIS). The advantages, compared with the earlier plan, include a better return on capital investment, reduced financial risk, lower energy consumption, and reduced waste disposal and emissions at site.

Compared with the May 2008, of the total $602 million capital costs, approximately $127 million was attributed to the second autoclave and the copper circuit.

The SDEIS will also incorporate modifications to the detailed operating plan including mine scheduling and waste handling that will reduce the environmental impact of the proposed project. These details are being finalized at the time of writing as part of the SDEIS preparation. The mine plan set out in this report reflects plans reported as part of the DFS Update in May 2008.

These changes are referenced in the appropriate sections covering process flowsheet, capital and operating costs.

PolyMet plans to complete a full project update, which will be summarized in a 43-101 Technical Report, once the details have been finalized in the environmental review process.

1.1 Location and Ownership

The NorthMet Deposit is situated on a mineral lease located in St Louis County in northeastern Minnesota, USA, at approximately Latitude 47° 36’ north, Longitude 91° 58’ west, about 70 miles north of the City of Duluth and 6.5 miles south of the town of Babbitt.

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The NorthMet Project comprises two elements: the NorthMet Deposit and the nearby Erie Plant. PolyMet leases the mineral rights to the NorthMet Deposit under a perpetually renewable lease and owns the Erie Plant through a contract for deed with Cliffs Natural Resources (Cliffs), which will be satisfied when the State of Minnesota transfers existing operating permits to PolyMet.

1.2 Geology and Mineralization

The NorthMet Deposit is part of the Duluth Complex in northeastern Minnesota, which is a large, composite, grossly layered, tholeiitic mafic intrusion that was emplaced into comagmatic flood basalts along a portion of the Mesoproterozoic Mid-continent Rift System. NorthMet is one of eleven known copper-nickel deposits that occur along the western edge of the Duluth Complex and within the Partridge River (PRI) and South Kawishiwi (SKI) intrusions. The NorthMet Deposit is hosted within the PRI, which consists of varied troctolitic and (minor) gabbroic rock types that have been subdivided into seven igneous stratigraphic units based on drill core logging.

The metals of interest at NorthMet are copper, nickel, cobalt, platinum, palladium and gold. Minor amounts of rhodium and ruthenium are also present though these are considered to have no economic significance. In general, with the exception of cobalt, the metals have strong positive correlations with copper mineralization. Cobalt is well correlated with nickel and reasonably correlated with copper.

Mineralization occurs in four broadly defined zones throughout the NorthMet property. Three of these laterally continuous zones occur dominantly within basal Unit 1. The thickness of each of the three Unit 1 enriched zones varies from 5 ft to more than 200 ft. Unit 1 mineralization is found throughout the base of the Deposit. The definition of the Unit 1 mineralized domain (DOM1) includes a portion of localized mineralization in the overlying Unit 2, which is merged into the top of Unit 1 for estimation purposes. A less extensive mineralized zone (Magenta Zone), slightly enriched with platinum group elements, is found in Units 4, 5, and 6 in the western part of the Deposit. This is defined as a separate mineralized domain within units that are mainly barren.

1.3 Exploration and Sampling

Drill hole spacing averages between 190 and 200 ft in the area of the resource model. This excludes holes drilled for metallurgical or geotechnical purposes. Distance studies show that 50% of the drillhole intercepts within Unit 1 will be within a 197-ft distance from another hole. In the Magenta Zone, 50% of the drillhole intercepts will be within a 190-ft distance from another hole. Fourteen percent (14%) of the assayed footage is by Reverse Circulation (six inch) drilling, with the remainder by diamond coring (BQ, NQ2, NTW, PQ and four inch).

The assay and geological database was thoroughly checked, validated and updated by PolyMet in order to provide the basis for the resource estimates reported in July 2005 (Hellman, 2005). The 2005 estimate involved the re-evaluation of historical data and the addition of several thousand new assays since the 2001 estimate. Examination of check assay data from pre-2005 assay programs as well as from newly received data suggest that nickel and cobalt from previous drill programs (pre-2005) are likely to have been understated by between 5% and 15% due to the use of an analytical method that

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resulted in an incomplete digestion (aqua regia digestion). All assaying of samples since the 2005 drilling and sampling campaign is based on the more appropriate total digestion four acid method. The data added since the 2005 drilling and sampling campaign is well validated through both formal quality control methods and extensive review of all compiled data.

A comprehensive Quality Assurance/Quality Control (QA/QC) program involving the use of coarse blanks, standards and duplicates has been instigated under the direction of Hellman and Schofield (H&S) and Lynda Bloom of Analytical Solutions Ltd., Toronto (ASL). This process consisted of the production of three matrix-matched standards from the NorthMet Deposit, sample preparation and homogenization, homogeneity testing, formulation of recommended values based on a round robin and routine insertion of standards on an anonymous basis. The three standards have copper concentrations in the approximate range 0.15 to 0.60% and nickel from 0.1 to 0.2%. Homogeneity of pulps, as determined by coefficients of variation from 20 replicate assays, is excellent with, for example, values less than 2% for copper and nickel and less than 5% for palladium.

During February and March 2005, nearly 14,000 ft of four inch and PQ (3.3 inch) diameter core holes were drilled for metallurgical sample collection while, approximately, a further 16,000 ft of NTW and NQ2 drill core (21 holes) were completed for resource in-fill and geotechnical evaluation purposes. Sixty-one additional core holes (NQ2 and NTW diameter), totaling approximately 47,500 ft were drilled from September through December 2005, for resource definition, in-fill and geotechnical assessment purposes. Sampling and data compilation for this drilling as well as continued sampling of historic US Steel core continued into March 2006. In 2007, an additional 61 in-fill holes were drilled during the spring and summer months.

1.4 Mineral Resources & Reserves

In October 2006, PolyMet published a report titled "Technical Report on the NorthMet Project" authored by D.J. Hunter. The resource statement in the report was sourced from Dr. P.L. Hellman of Hellman & Schofield dated July 2006. The resource figures were based on a block model with dimensions of 100 ft on strike by 100 ft perpendicular to strike by 20 ft vertically and interpolated using ordinary kriging with data available as of July 2006. Hellman & Schofield elected to interpolate the resource model from surface to the 500 ft elevation based on a pit floor assumption at the 560 ft elevation. The pit floor elevation was obtained from a Whittle pit optimization conducted on an earlier model by mining engineering consultants Australian Mine Design & Development Pty Ltd (AMDAD). The resource was reported at a Net Metal Value (NMV) cut-off of US$7.42 per short ton.

AGP interpolated the June 2007 model using a new block size of 50 ft on strike by 50 ft perpendicular to strike by 20 ft vertically using ordinary kriging with inverse distance and nearest neighbour check models. The block size was reduced to 50 ft by 50 ft by 20 ft (from 100 ft by 100 ft by 20 ft) after an evaluation into the selective mining unit that is required to eventually mine the Deposit. The model was interpolated to the 0 ft elevation to allow further detailed mining engineering study to evaluate incorporating resources at depth.

AGP updated the resource model in December 2007 to include the assays that were pending from the spring 2007 drill campaign along with results from 14 new holes from the summer 2007 drilling

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campaign. Interpolation methodology remained essentially the same as the June 2007 model with updated parameters.

Based on the review of the QA/QC, data validation and statistical analysis of the data, AGP draws the following conclusions:

- AGP has reviewed the methods and procedures to collect and compile geological and assaying information for the NorthMet Deposit and found them meeting accepted industry standards and suitable for the style of mineralization found on the property.
- A mix of data type was used to generate the resource on the property. Fourteen percent (14%) of the assayed footage is by Reverse Circulation (six inch) drilling; with the remainder by diamond coring. The resource also includes historical drill results gathered while the property was under the ownership of US Steel. PolyMet validated the RC drill results against twin (or near twin) drill hole and found them to be satisfactory. AGP's Principal Resource Geologist visited the site, reviewed some of the historical drill core and interviewed PolyMet staff. AGP believes that the information supplied for the resource estimate and used in this report is accurate.
- A QA/QC program comprising industry standard blank, standard and duplicate samples has been used on the Project since the 2005 drill program. QA/QC submission rates meet industry-accepted standards.
- Data verification was performed by AGP through site visits, collection of independent character samples and a database audit prior to mineral resource estimation. AGP found the database to be exceptionally well maintained and error free and usable in mineral resource estimation.
- The specific gravity determinations are representative of the in-situ bulk density of the rock types.
- Sampling and analysis programs using standard practices provided acceptable results. AGP believes that the resulting data can effectively be used in the estimation of resources.
- Core handling, core storage, and chain of custody are consistent with industry standards.
- In AGP's opinion the current drill hole database is adequate for interpolating grade models for use in resource estimation.
- Mineral resources were classified using logic consistent with the CIM definitions referred to in NI 43-101.

Results including all data available as of October 15, 2007 indicate the NorthMet resources (above a US$7.42 NMV cut-off) contain 694.2 million short tons (629.8 million tonnes) in the Measured and Indicated categories grading at 0.265% copper, 0.077% nickel, 68 parts per billion (ppb) platinum, 239 ppb palladium, 35 ppb gold and 71 parts per million (ppm) cobalt. The Inferred category (above a

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US$7.42 NMV cut-off) totals 229.7 million short tons (208.4 million tonnes) grading at 0.273% copper, 0.079% nickel, 73 ppb platinum, 263 ppb palladium, 37 ppb gold and 56 ppm cobalt.

The NMV formula used and described in Section 17.2.12 of this report includes the gross metal price multiplied by the processing recovery minus refining, insurance and transportation charges and is the same formula used in the Hunter 2006 report.

Above the 0.2% copper cut-off the NorthMet Deposit contains 442.1 million short tons (401.0 million tonnes) in the Measured and Indicated categories grading at 0.325% copper, 0.089% nickel, 81 ppb platinum, 292 ppb palladium, 41 ppb gold and 73 ppm cobalt. The Inferred category totals 158.7 million short tons (144.0 million tonnes) grading at 0.329% copper, 0.088% nickel, 86 ppb platinum, 315 ppb palladium, 43 ppb gold and 55 ppm cobalt.

Comparing the AGP model with the previously published estimate, Table 17.23 of the Wardrop, September 2007 report, results show an increase of 15.5 million short tons (14.1 million tonnes) in the Measured category and 40.5 million short tons (36.7 million tonnes) in the Indicated category for a total of 56 million short tons (50.8 million tonnes) or 8.1% increase in the Measured plus Indicated category. The Inferred Resource tonnage dropped by 21.9 million short tons (26.4 million tonnes) or 9.5%. The comparison includes resources above a US$7.42 Net Metal Value cut-off from surface down to the 0 ft elevation level.

Compared with the Wardrop September 2006 estimate, grades in the Measured and Indicated categories dropped slightly for copper and nickel and increased slightly for platinum, palladium, gold, and cobalt grade elements. Copper changed by -0.3%, nickel by -0.5%, platinum by +2.1%, palladium by +1.8%, gold by +2.1% and cobalt by +0.1%. However, the contained metal value increased for all elements by about 10% in the Measured and Indicated categories. Copper increased by 8.5%, nickel by 8.2%, platinum by 11.1%, palladium by 10.8%, gold by 11.0% and cobalt by 8.9%.

The work carried out during the summer 2007 drill program met the primary objectives relating to the in-fill drilling.

Mineral Reserves are reported at commodity prices of:

- Copper = $1.25 / lb
- Nickel = $5.60 / lb
- Platinum = $800.00 / troy ounce
- Palladium = $210.00 / troy ounce
- Gold = $400.00 / troy ounce
- Cobalt = $15.25 / lb

These prices were used to generate the DFS pit shell, within which the reserves were contained. This pit shell is the same design as outlined in the DFS study published October 2006 and developed by Australian Mine Design & Development Pty Ltd. (AMDAD). This pit shell was applied to the updated resource model.

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A mining cutoff was used by AGP that was determined on a block by block basis with the following formula:

Block Value ($) = Gross Metal Value – Mining Cost – Processing cost – G&A.

Where:

- Block Value = net value of the block in dollars
- Gross Metal Value = value of metals considering price, recovery and downstream costs
- Mining Cost = cost to mine ore and waste adjusted for haulage path
- Processing Cost = cost to process ore tonnes
- G&A = anticipated General and Administrative costs

The block value was stored in each block and a cutoff where the block value was greater than or equal to $0.01. This implies that the block would make $0.01 or greater of net revenue (not considering capital) to mine the block and process it for the contained metal. Blocks with a value of $0.00 or less were deemed to be waste material.

Table 1-1 Updated Reserve Estimate – September 2007

Class	Tonnage (Mst)	Grades (Diluted)					
		Copper (%)	Nickel (%)	Platinum (ppb)	Palladium (ppb)	Gold (ppb)	Cobalt (ppm)
Proven	118.1	0.30	0.09	75	275	38	75
Probable	156.5	0.27	0.08	75	248	37	72
Total	274.7	0.28	0.08	75	260	37	73

The following notes should be read in conjunction with Table 1-1:
Rounding as required by reporting guidelines may result in apparent summation differences between tons, grade and contained metal.
Tonnage and grade measurements are in Imperial units.
The reserves are bound within the DFS pit shell.

1.5 Mining and Processing

The NorthMet Deposit will be developed as an open pit mine, starting at the East Pit, then both the East Pit and the larger West Pit, and finally after the East Pit has been completed, some waste from the West Pit will be backfilled into the East Pit.

Run of mine (ROM) rock will be delivered to a loading system, loaded onto rail cars which will deliver the rock to Erie Plant by private railroad.

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The Erie Plant operated from 1957 to 2001, processing taconite (low-grade iron ore), and was shut down in the bankruptcy of its owner, LTV Steel Mining Company (LTVSMC).

The exiting Erie Plant has a historic capacity of approximately 100,000 tons per day, comprising four-stage crushing and 34 mill lines, each comprising a rod mill and a ball mill. PolyMet's plans use one of the two primary crushers, and approximately one-third of the rest of the crushing and milling circuit.

The discharge from the ball mills will be processed through a flotation circuit to produce separate copper and nickel concentrates. In the initial phase of operation, PolyMet will sell both of these concentrates to Glencore International (Glencore) under a long-term marketing agreement.

PolyMet will then build a hydrometallurgical circuit to process the nickel concentrate, which will produce a nickel-cobalt hydroxide and a precious metals precipitate, which will be sold to Glencore.

Tailings from the flotation will be deposed of in the existing tailings basin, which is partially filled with taconite tailings, but has more than sufficient capacity for the planned operations.

1.6 Environmental

The NorthMet Project is located within the established mining corridor of existing and now disused iron ore mines, including the Peter Mitchell pit of the NorthShore operations of Cliffs immediately north of the NorthMet Deposit. The Erie Plant is an existing facility with all of the supporting infrastructure already in place.

Minnesota has very stringent environmental standards and environmental review process. The NorthMet environmental review process involves the Minnesota Department of Natural Resources (DNR) the United States Army Corp. of Engineers (USACE) and the United States Forest Service (USFS) as "Lead Agencies". The United States Enviromental Protection Agency (EPA) and tribal authorities are cooperating agencies and the Minnesota Pollution Agency (PCA) is taking part in the process as a permitting agency.

The biggest area of attention is water quality – NorthMet is in the headwaters of the St Louis River, which flows into Lake Superior and is therefore governed by Great Lakes standards. It is important to note that NorthMet is across the Laurentian Divide from the Boundary Waters Canoe Area wilderness and Voyagers National Park and therefore any water discharge will not affect those areas.

The Lead Agencies are currently preparing a detailed EIS that will consider the impact of the Project as it is planned to be built and operated. An earlier Draft EIS published in 2009 considered a range of alternative plans, did not include key mitigation plans that have been developed during the past three years, and did not recommend a preferred project plan. The Supplemental Draft EIS will address these concerns and demonstrate that the NorthMet project meets all state and federal standards.

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1.7 Economics

The economic summary reflects the 2008 DFS Update. Key economic metrics include earnings before interest, tax, depreciation, and amortization (EBITDA) which is projected to be $217.3 million on average over the first five years of operations. The net present value of future cash flow (after tax) discounted at 7.5% is estimated to be $649.4 million compared, and the after tax internal rate of return is estimated at 30.6%. Table 1-2 also sets out the affect on EBITDA of a 10% change in each metal price. The figures show a comparison with the NI 43-101 filed with the completion of the DFS in 2006.

Table 1-2: Key Economic Highlights

		Update May-08	DFS Sep-06
Operating plan			
Proven and probable reserves	million t	274.7	181.7
Ore mined - life of operation	million t	224.0	181.7
Overburden removed (capitalized under site preparation)	million t	18.5	-
Waste	million t	285.3	302.3
Operating costs per ton processed			
Mining and delivery to plant	$/t	4.31	3.80
Processing	$/t	8.07	6.75
G&A	$/t	0.94	0.46
Total	$/t	13.33	11.02
Metal price assumptions (SEC-standard)			
Copper	$/lb	2.90	2.25
Nickel	$/lb	12.20	7.80
Cobalt	$/lb	23.50	16.34
Palladium	$/oz	320	274
Platinum	$/oz	1,230	1,040
Gold	$/oz	635	540
Economic summary			
Annual earnings before interest, tax, depreciation and amortization (EBITDA) - average first five years	$ million	217.3	175.3
Net present value of future after tax cash flow discounted at 7.5%	$ million	649.4	595.4
Internal rate of return (after tax)		30.6%	26.7%
Sensitivity: 10% ± price = $Δ million in EBITDA			
Copper	$ million	18.6	15.7
Nickel	$ million	13.3	9.3
Cobalt	$ million	0.9	0.9
Palladium	$ million	1.7	2.0
Platinum	$ million	1.7	2.1
Gold	$ million	0.3	0.5
Copper costs			
cash - co-product method	$/lb	1.05	0.81
cash - by-product method	$/lb	(0.28)	0.06

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Table 1-3: Metal Prices

		DFS Base Case	DFS Market Case	DFS Update	06/30/12 3-year trailing average
Metal Price					
Copper	$/lb	1.50	2.25	2.90	3.56
Nickel	$/lb	6.50	7.80	12.20	9.47
Cobalt	$/lb	15.25	16.34	23.50	17.69
Palladium	$/oz	225	274	320	684
Platinum	$/oz	900	1,040	1,230	1,689
Gold	$/oz	450	540	635	1,485
After tax:					
Internal rate of return	%	13.4%	26.7%	30.6%	
PV dicounted at 7.5%	$ millions	161.9	595.4	649.4	

PolyMet did not report detailed economic impact of the 2011 project changes but the impact will have been positive owing to reduced capital and operating costs. This analysis will be included in the full project update once all of the details of environmental mitigation measures have been finalized in the Supplemental Draft EIS.

1.8 Conclusions and Recommendations

AGP offers the following recommendations.

PolyMet should proceed with final design engineering and construction of the NorthMet Project as soon as permitting allows. Prior to construction, PolyMet should:

- Review and update the scope of the Project design to reflect any changes resulting from the environmental review process and other project enhancements.
- Update the capital and operating cost estimates based on the scope review and current prices.
  - Continue to review and reassess core drilled by US Steel with particular reference to skeletonised holes within or near the current 20-year pit shell.

Prior to detailed, pre-production planning a limited program of close-spaced drilling is recommended. This program will have two objectives;

- To determine the optimum blast-hole spacing for grade control and scheduling and,
- To increase confidence in grade affecting the initial open pit production.
  - Budget for 625 large diameter (5 ½") reverse circulation drillholes averaging 30 ft for a total of 19,050 ft is estimated at $40 /ft for an all in cost of $782,000 including a $20,000 mobilization charge. Cost is less if using a 3 ½" diameter.

The total for all of these items is in PolyMet's budgets for activities before the start of construction, for a total of approximately $3.0 million.

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Various recommendations for further work resulted from the Updated DFS. Some of this work has been completed as of October 2012.

1) Development of a low-grade recovery relationship for copper and nickel and the other metals
   Development of a low-grade recovery relationship for copper, nickel and the other metals needs to be completed on low grade samples using a consistent metallurgical protocol. As the cutoff grade is dropped, the impact of lower grades becomes greater and also its impact on overall project economics. This work has been completed.

2) Updating of metal payment pricing and terms
   Metal prices and terms for mining planning purposes have not been updated since the DFS. With the introduction of concentrate sales, long-term marketing with Glencore, and changes to metal markets, the current cut-off is likely to exclude mineralization that would be economic to mine and process.

3) Stockpiling options possible to increase initial mill feed grade
   Current low grade ore stockpile limit is for 5 million tons of material. If the limit is increased to a higher value, the initial years mill feed grade can be increased improving overall project economics.

4) Potential for daily mine ore production increase
   The NorthMet resource base and the geometry of the deposits could allow an increase in ore tonnage.

2 INTRODUCTION

This report describes the results of a mineral resource estimation update of the NorthMet Deposit, which is controlled by PolyMet. The original report was prepared at the request of Mr. Don Hunter, who at the time was the Area Manager-Mining, NorthMet Project, following a drilling program that commenced in February 2007 and completed in July 2007. This updated report was prepared at the request of Mr. Douglas Newby, Chief Financial Officer of PolyMet Mining, in response to a request from the British Columbia Securities Commission in June 2012 for inclusion of the reserves announced in 2007. The 2007 program was instigated primarily to provide additional grade and confidence information and importantly, to provide greater, more extensive definition to the Magenta Zone which had been recognized in earlier drilling. This report is concerned with the drilling results available to PolyMet as at October 15, 2007, including results from all previous drilling.

Information, conclusions, and recommendations contained herein are based on a field examination, including a study of relevant and available data and discussions with Polymet site geologists Richard Patelke and Steve Geerts. Pierre Desautels, Principal Resource Geologist for AGP Mining Consultants Inc. and senior author of this report visited the Project area for a total of five days in March 2007 and August 2007.

2.1 Terms of Reference

The NorthMet resource estimates described herein were completed by AGP at the request of PolyMet in order to provide input to ongoing pit optimization studies and are reported in compliance with the Canadian Securities Administrators NI 43-101 under the direct supervision of:

Pierre Desautels P.Geo. Principal Resource Geologist with AGP Mining Consultants Inc. He directed the review of the 2007 digital data as well as the estimation of the resource for the NorthMet Deposit and is the qualified person (QP) responsible for the report. Mr. Desautels visited the NorthMet site from March 21-23, 2007 and again from August 27-29, 2007 to gather the necessary data used in the resource estimate, review drill core logging and sampling procedures, collect representative check samples and verify drillhole collars locations.

Richard Patelke P.Geo. Former Project Geologist with Poly Met Mining, Inc., now deceased. He was responsible for historical and background information on the NorthMet Deposit. Mr. Patelke resided in Minnesota and was a Registered Professional Geologist of good standing with the State of Minnesota at the time of the estimate. Mr. Patelke was involved in fieldwork at NorthMet, several of the adjacent copper-nickel deposits, detailed outcrop mapping projects, and other mine development projects in the region in a period covering seventeen years. He worked on logging and sampling of drill core recovered from the NorthMet Deposit and others during previous drilling and sampling campaigns. Pierre Desautels will now assume responsibility as the QP for the sections that were authored by Mr. Richard Patelke.

Gordon Zurowski P.Eng. Principal Mine Engineer with AGP Mining Consultants Inc. He completed the mining plans as well as compiled mine capital and operating costs. Mr. Zurowski is the qualified person

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(QP) responsible for the reserve statement. Mr. Zurowski visited the site on October 9th to 11th, 2007 to review the overall site layout, infrastructure and proposed rail sidings.

All units used in this report are imperial unless otherwise stated; grid references are based on the Minnesota State Plane Grid (North Zone, NAD83, NAVD 88).

2.2 Effective Dates

The data cut-off date and resource effective dates is October 15th, 2007. No additional work has been conducted on the property by PolyMet and as such, the QP considers the resource estimate to be current.

Reference is made to subsequent revisions to the process flow sheet, reported by PolyMet in May 2008 and February 2011. . In addition, reference is made to changes to the mine plan that is bbeing incorporated into the current environmental review where the absense of such reference could be misleading.

2.3 Previous Technical Reports

Much of the text in this report was sourced from the following technical reports and edited as required:

- Report titled "Technical Report on the NorthMet Deposit, Minnesota, USA" by Wardrop Engineering Inc. this report is author by Desautels, P., Patelke, R. and dated September 2007. This report is available on SEDAR.
- Report titled "Mineral Resource Update, NorthMet Poly-Metallic Deposit, Minnesota, USA" by Hellman & Schofield Pty Ltd. author by Hellman, P.L., PhD, FAIG. and dated August 2006. This report is available on SEDAR.
- Report titled "Technical Report on the NorthMet Project" author by Hunter, D.J., C.Eng, CP (Mining) and dated October 2006. This report has a sub-titled " Technical Report on the Results of a Definitive Feasibility Study of the NorthMet Project"
- NI43-101 Report titled "Mineral resource update, NorthMet poly-metallic Deposit, Minnesota, USA." authored by Hellman & Schofield Pty Ltd., and dated 2005. This report is available on SEDAR.

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3 RELIANCE ON OTHER EXPERTS

AGP has followed standard professional procedures in preparing the content of this resource estimation report. Data used in this report has been verified where possible and this report is based upon information believed to be accurate at the time of completion.

AGP has not verified the legal status or legal title to any claims and has not verified the legality of any underlying agreements for the subject properties and relied on the information provided by Richard Patelke and Mr. Don Hunter.

The writers have also relied on several sources of information on the property, including technical reports by consultants to PolyMet, digital geological and assay data, and geological interpretations by PolyMet. Therefore, in writing this report the senior author relies on the truth and accuracy as presented in various sources listed in the References section of this report.

Other Qualified Person contributing authors responsible for producing this report include: Karl Everett, David Dreisinger, and William Murray. Items of responsibility for each of the contributing authors are identified in Table 3-1.

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Table 3-1: Qualified Persons Table of Responsibility

Name	Site Visit	QP	Independent of the issuer	Responsibility
Pierre Desautels P.Geo. of AGP Mining Consultants	March 21-23, 2007 August 27-29, 2007	Yes	Yes	Sections 1.2, 1.3, the resource portion of Section 1.4 and the geology, exploration and resource portion of Section 1.8, and complete Sections 2, 3, 4.1 and complete Sections 5 through 12, section 14, 23, 24 and the portions of Section 25 and 26 related to geology, exploration and resources
Gordon Zurowski P.Eng. of AGP Mining Consultants	October 9-11, 2007	Yes	Yes	Reserve portion of Section 1.4, the mining portion of Section 1.5, the reserves and mining portions of Section 1.8, the complete Sections 15 and 16 and the portions of Sections 25 and 26 related to mining and reserves
Karl Everett, P.E. of Foth Infrastructure & Environment, LLC	Numerous, most recently April 19, 2012	Yes	Yes	Sections 1.6, 4.7, 4.8, and the complete Section 20
David Dreisinger	Numerous, most recently January 21, 2009	Yes	No	Mineral processing portion of Section 1.5, and the complete Sections 13 and 17
William Murray	Numerous, most recently October 25-27, 2011	Yes	No	Sections 1.1, 1.7, 4.2 through 4.6, 4.9, 4.10, and complete Sections 18, 19, 21, and 22

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4 PROPERTY DESCRIPTION AND LOCATION

4.1 Project Location

The NorthMet Project comprises two key elements: the NorthMet Deposit and the Erie Plant. The NorthMet Deposit is situated on a mineral lease located in St Louis County in northeastern Minnesota at Latitude 47° 36′ north, Longitude 91° 58′ west, about 70 miles north of the City of Duluth and 6.5 miles south of the town of Babbitt (Figure 4-1). The Erie Plant is approximately six miles west of the NorthMet Deposit.

The NorthMet Deposit site totals approximately 4,300 acres and the Erie Plant site, including the existing tailings basin, covers approximately 12,300 acres.

The NorthMet project is located immediately south of the eastern end of the historic Mesabi Iron Range and is in proximity to a number of existing iron ore mines including the Peter Mitchell open pit mine located approximately two miles to the north of the NorthMet Deposit. NorthMet is one of several known mineral deposits that have been identified within the 30-mile length of the Duluth Complex, a well-known geological formation containing copper, nickel, cobalt, platinum group metals and gold.

The NorthMet Deposit is connected to the Erie Plant by a transportation and utilities corridor that will comprise an existing private railroad that will primarily be used to transport ore, a segment of the existing private Dunka Road that will be upgraded to provide vehicle access, and new water pipelines and electrical power network for the NorthMet mine site

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Figure 4-1: Property Location Map

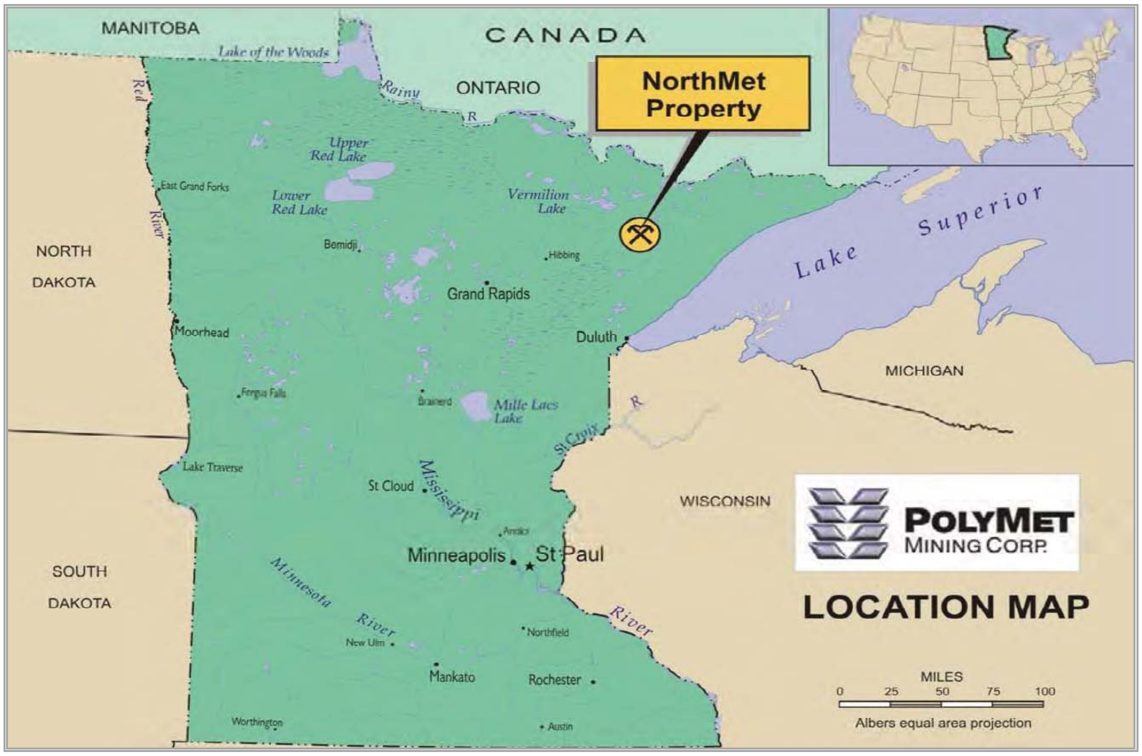
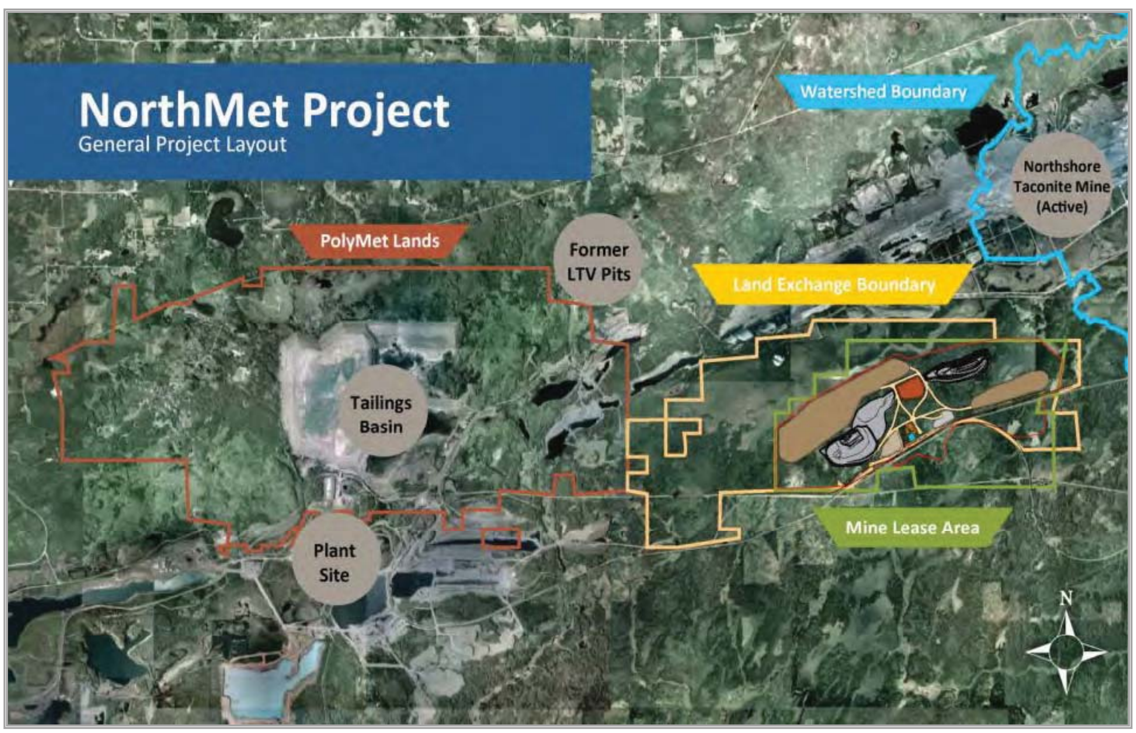


Figure 4-2: Property Layout Map



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4.2 Project Ownership

PolyMet Mining Corp. owns 100% of Poly Met Mining, Inc. (PolyMet US), a Minnesota corporation.

PolyMet US controls 100% of the NorthMet Project. The mineral rights covering 4,282 acres or 6.5 square miles at the NorthMet orebody are held through two mineral leases:

- The US Steel Lease dated January 4, 1989, subsequently amended and assigned, covers 4,162 acres originally leased from US Steel Corporation (US Steel), which subsequently sold the underlying mineral rights to RGGS Land & Minerals Ltd., L.P. (RGGS). PolyMet can and has extended the lease indefinitely by making $150,000 annual lease payments on each successive anniversary date. The lease payments are advance royalty payments and will be deducted from future production royalties payable to RGGS, which range from 3% to 5% based on the net smelter return, subject to minimum payments of $150,000 per annum.
- On December 1, 2008, PolyMet entered into an agreement with LMC Minerals ("LMC") whereby PolyMet leases 120 acres that are encircled by the RGGS property. The initial term of the renewable lease is 20 years with minimum annual lease payments of $3,000 on each successive anniversary date until the earlier of NorthMet commencing commercial production or for the first four years, after which the minimum annual lease payment increases to $30,000. The initial term may be extended for up to four additional five-year periods on the same terms. The lease payments are advance royalty payments and will be deducted from future royalties payable to LMC, which range from 3% to 5% based on the net smelter return, subject to a minimum payment of $30,000 per annum.

The surface rights are held by the USFS - see Section 4.4.

PolyMet US owns 100% of the Erie Plant, which covers approximately 12,400 acres, or 19.4 square miles, through contracts for deed with Cliffs. Further details can be found in Section 4.6.

4.3 Mineral Tenure

The NorthMet Project lies within the lands ceded by the Chippewa of Lake Superior to the United States in 1854, known as the "1854 Ceded Territory."

In the 1940s, copper and nickel were discovered near Ely, Minnesota, following which, in the 1960s, US Steel drilled what is now the NorthMet Deposit. US Steel investigated the NorthMet Deposit as a high-grade, underground copper-nickel resource, but considered it to be uneconomic based on its inability to produce separate, clean nickel and copper concentrates with the metallurgical processes available at that time. In addition, prior to the development of the autocatalyst market in the 1970s, there was little market for platinum group metals (PGMs) and there was no economic and reliable method to assay for low grades of these metals.

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In 1987, the Minnesota Natural Resources Research Institute ("NRRI") published data suggesting the possibility of a large resource of PGMs in the base of the Duluth Complex.

PolyMet, as Fleck Resources, acquired a 20-year renewable mineral rights lease to the NorthMet Deposit in 1989 from US Steel. The lease is subject to yearly lease payments before production and then to a sliding scale Net Smelter Return (NSR) royalty ranging from 3% to 5% with lease payments made before production considered as advance royalties and credited to the production royalty. PolyMet leases an additional 120 acres of mineral rights underlying 120 acres from LMC.

Mineral and surface rights have been severed, with the USFS owning the surface rights within most of the lease area. US Steel retained the mineral rights and certain rights to explore and mine on the site under the original documents that ceded surface title to the USFS.

4.4 Surface Rights

Surface rights at the NorthMet Deposit are held by the USFS. The United States acquired the surface rights from US Steel in 1938 under provisions of the Weeks Act of 1911. US Steel retained certain mining rights, which PolyMet secured under the US Steel Lease, along with the mineral rights.

PolyMet proposes to complete a land exchange with the USFS whereby the USFS will transfer its surface rights to PolyMet in exchange for two tracts of land totalling approximately 5,300 acres of forests, wetlands, and lakes with high recreational value that PolyMet has acquired. These lands are subject to a $4 million mortgage from the Iron Range Resources and Rehabilitation Board (IRRRB), an economic development agency with no regulatory oversight for mine permitting activities.

The proposed land exchange complies with the 2004 Superior National Forest Land and Resource Management Plan (Forest Plan) and will: provide and sustain benefits to the American people; conserve open space; sustain and enhance outdoor recreation opportunities; and maintain basic management capabilities of the Forest Service by reducing land lines and mineral conflicts.

The Superior National Forest will decide in a Record of Decision whether to proceed with the proposed land exchange, based on the Final EIS for the NorthMet Project.

4.5 Royalties and Encumbrances

The NorthMet Deposit mineral rights carry variable royalties of 3% to 5% based on the net metal value per ton of ore mined. For a net metal value of under $30 per ton, the royalty is 3%, for $30-35 per ton it is 4%, and above $35 per ton it is 5%. Both the US Steel Lease and the LMC Lease carry advance royalties which can be recouped from future royalty payments, subject to minimum payments in any year.

4.6 Environmental Liabilities

Federal, state and local laws and regulations concerning environmental protection affect PolyMet's operations. Under current regulations, PolyMet is contracted to indemnify Cliff's requirement to meet

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performance standards to minimize environmental impact from operations and to perform site restoration and other closure activities. PolyMet's provisions for future site closure and reclamation costs are based on known requirements. It is not currently possible to estimate the impact on operating results, if any, of future legislative or regulatory developments. PolyMet's estimate of the present value of the obligation to reclaim the NorthMet Project is based upon existing reclamation standards at July 31, 2012. Once PolyMet obtains permits to mine, the environmental and reclamation obligations will be transferred to PolyMet from Cliffs.

The Company's best estimate of the total environmental rehabilitation at July 31, 2012 was $25.8 million.

In April 2010, Cliffs entered into a consent decree with the Minnesota Pollution Control Agency (MPCA) relating to alleged violations on the Cliffs Erie Property. This consent decree required submission of Field Study Plan Outlines and Short Term Mitigation Plans, which have been approved by the MPCA. In April 2012, long-term mitigation plans were submitted to the MPCA for its review and approval, such approval remains outstanding to date. As part of its prior transactions with Cliffs, PolyMet has agreed to indemnify Cliffs for certain ongoing site environmental liabilities.

There is uncertainty related to the engineering scope and cost of mitigation required to meet applicable water standards, and responsibility for the financial liability. As such, the Company is unable to estimate its potential liability for the Long Term Mitigation Plan.

4.7 Permits

Cliffs holds certain permits that provide for the maintenance of the site, which is carried out by PolyMet at PolyMet's expense under the terms of the contracts for deed. PolyMet is not currently carrying out any exploration at the NorthMet mine site but would require permits from the USFS for any additional work prior to the completion of the land exchange.

Prior to construction and operation of the NorthMet Project, PolyMet will require several permits from federal and state agencies – see section 20.4.

4.8 Social License

The environmental review process is described on Section 20. The federal, state and local government permits needed for PolyMet to construct and operate the NorthMet Project are described in Section 20.4.

PolyMet has maintained an active community outreach program for many years. The focus of the program has been to provide information about the Project, its likely impact in the environment, and the socio-economic benefits. The local communities are supportive of the Project. PolyMet has received letters of support from U.S. Senators Klobuchar and Franken and U.S. Representative Cravaack is publicly and actively seeking ways to help the Project move forward.

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Bois Forte Band of Chippewa (Bois Forte), Grand Portage Band of Chippewa (Grand Portage), and the Fond du Lac Band of Lake Superior Chippewa (Fond du Lac) are cooperating agencies in preparation of the EIS. Fond du Lac has expressed the strongest opposition, primarily related to cultural heritage issues and seeking to ensure that water quality is protected.

The most active environmental groups in the area are focused on protecting the Boundary Waters Canoe Area Wilderness, which is located approximately 25 miles northeast of the NorthMet site, in a different watershed.

4.9 Significant Risk Factors

Permitting is the most significant risk factor for the Project. The NorthMet Project is the first copper-nickel project in Minnesota to seek permits for construction and operation and, as such, requires state regulators to interpret established regulations.

Permitting risk falls into two primary categories: permits may be denied or legally challenged, or operating requirements imposed by the permits could be financially so burdensome that the Project is unable to proceed.

These risks are mitigated by completing a thorough environmental review and, in the case of the NorthMet Project, the existence of the Erie Plant and associated infrastructure.

4.10 Comments on Section 4

Mineral and property tenure is secure. Completion of the environmental review and permitting is the biggest challenge, but the Lead and Co-operating Agencies are on track to finish this complex process.

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5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

The Project site is situated immediately south of the eastern part of the historically important Mesabi Iron Range, a world class mining district that produces approximately 42 million tons per year of taconite pellets and iron ore concentrate. There are six producing iron ore mines on the Range, of which the nearby Northshore open pit mine owned and operated by Cliffs is one of the largest. The Northshore pit is located approximately two miles north of the NorthMet Deposit.

5.1 Accessibility

Access to the NorthMet project is by a combination of good quality asphalt and gravel roads via the Erie Plant site. The nearest center of population is the town of Hoyt Lakes, which has a population of about 2,500 people. There are a number of similarly sized communities in the vicinity, all of which are well serviced, provide ready accommodation, and have been, or still are, directly associated with the region's extensive taconite mining industry. The road network in the area is well developed, though not heavily trafficked, and there is an extensive railroad network, which serves the taconite mining industry across the entire Range. There is access by ocean shipping via the ports at Taconite Harbor and Duluth/Superior (on the western end of Lake Superior) and the St. Lawrence Seaway.

5.2 Climate

Climate is continental and characterized by wide temperature variations and significant precipitation. The temperature in the town of Babbitt, about 6.5 miles north of the NorthMet Deposit, averages four degrees Fahrenheit (°F) in January and 66°F in July. During short periods in summer, temperatures may reach as high as 90°F with high humidity. Average annual precipitation is about 28 inches with about 30% of this falling mostly as snow between November and April. Annual snow fall is typically about 60 inches with 24 to 36 inches on the ground at any one time. The local taconite mines operate year round and it is rare for snow or inclement weather to cause production disruption.

5.3 Local Resources and Infrastructure

The area has been economically dependent on the mining industry for many years and while there is an abundance of skilled labour and local mining expertise, the closure in 2001 of the LTVSMC open pit mines and taconite processing facility has had a significant negative impact on the local economy and population growth. There are, however, a number of other operating mines in other parts of the Iron Range. Hence the mining support industries and industrial infrastructure remains well developed and of a high standard.

The Erie Plant site is connected to the electrical power supply grid and a main HV electrical power line (138 kV) runs parallel to the road and railroad that traverse the southern part of the mining lease area. PolyMet has a long-term power contract with Minnesota Power.

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There are plentiful local sources of fresh water. While electrical power and water is available nearby, the author is not qualified to comment as to the adequacy of these resources to support an open pit mining operation, but notes previous operations at 100,000 tons per day, or three times PolyMet's plans.

5.4 Physiography

The Iron Range forms an extensive and prominent regional topographic feature. The Project site is located on the southern flank of the eastern Range where the surrounding countryside is characterized as being gently undulating. Elevation at the Project site is about 1,600 ft above sea level (1,000 ft above Lake Superior). Much of the region is poorly drained and the predominant vegetation comprises wetlands and boreal forest. Forestry is a major local industry and the Project site and much of the surrounding area has been repeatedly logged. Relief across the site is approximately 100 ft.

5.5 Sufficiency of Surface Rights

Tenure of surface rights is described in some detail in Section 4.4. In summary, surface rights are held by the USFS. Exchange of these rights for other land owned by PolyMet is part of the Supplemental Draft EIS and PolyMet expects the exchange to occur following the Record of Decision related to the Final EIS.

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6 HISTORY

There has been no prior mineral production from the NorthMet Deposit though it has been subject to several episodes of exploration and drilling since its discovery in 1969 by US Steel. Table 6-1 summarizes the exploration drilling activities since 1969 and the amount of assay data.

US Steel held mineral and surface rights over much of the region, including the NorthMet lease, until the 1930s when, for political and land management reasons, surface title was ceded to the US Forest Service. In negotiating the deeds that separated the titles, US Steel retained the mineral rights and the rights to explore and mine any mineral or group of minerals on the site, effectively removing the possibility of veto of such activities by the USFS, provided they are carried out in a responsible manner.

In 1989, Fleck Resources Ltd. (Fleck), a company registered in British Columbia, Canada, acquired a 20-year renewable mineral rights lease to the NorthMet Deposit from US Steel and undertook exploration of the NorthMet Deposit. Fleck developed joint ventures with NERCO Inc. in 1991 and Argosy Mining Corp. in 1995 in order to progress exploration.

In June 1998, Fleck Resources Ltd. changed its name to PolyMet Mining Corporation. In 2000, there was a short-lived joint venture with North Mining Inc. that was terminated by PolyMet when North Mining Inc. was bought by Rio Tinto plc. With the exception of a hiatus between 2001 and 2003, PolyMet has continued exploration and evaluation of the NorthMet Deposit until 2007, since when it has been focused on completing the environmental review and permitting process, and enhancing the process design.

In 2000, PolyMet commissioned Independent Mining Consultants, Inc. of Tucson, Arizona (IMC) to carry out a Pre-feasibility Study. The report was published in 2001 and filed on SEDAR (IMC, 2000). One of the conclusions of the IMC Pre-feasibility Study report was that proceeding to the preparation of a full Feasibility Study was warranted.

In 2004, US Steel sold much of its real estate and mineral rights in the region, including the NorthMet Deposit, to a private company, RGGS of Houston Texas. PolyMet's US Steel lease was transferred to RGGS at that time without any change in conditions.

US Steel took at least three bulk samples from NorthMet in 1970 and 1971 (Patelke and Severson, 2006). The three samples weighed approximately 9 tons, 300 tons and 20 tons respectively. The samples came from mineralization in Units 3 and 1 (see descriptions of these units in Section 7 of the report).

Table 6-1: Summary of NorthMet Exploration Activity Since 1969

Company	Date of Drilling	Date of Assaying	No. of Drill Holes	Total Footage for Group	Number of Assay Intervals **used in "Accepted Values"** Tables	Assayed Footage used in Final Database	Assay Labs
US Steel	1969-1974	1969-1974 1989-1991 1999-2001 2005-2006	112	113,716	9,475	56,525	US Steel, ACME, ALS-Chemex
US Steel	1971-1972	Three surface bulk samples for metallurgical testing taken from two locations					
NERCO	1991	1991	2 (4)	842	165	822	ACME
NERCO	1991	Bulk metallurgical sample from large size (PQ) core used for tests of CUPREX hydrometallurgical process (842 ft)					
PolyMet Reverse Circulation Drilling	1998-2000	1998-2000	52	24,650	4,765	23,767	ACME
PolyMet Core Drilling	1999-2000	2000-2001	32	22,156	4,058	20,727	ALS-Chemex
PolyMet RC Drilling Deepened with AQ Core Trail	2000	2000	3	2,696	524	2,610	ALS-Chemex
PolyMet	1998 & 2000	Two flotation pilot plant campaigns and variability testing used about 60 tons of sample derived from RC drilling programs					
PolyMet Core Drilling	2005	2005-2006	109	77,166	11,656	71,896	ALS-Chemex
PolyMet	2005	Samples from four inch and PQ core processed for pilot flotation and metal production, three composites of average 0.3%, and 0.4% Cu, 10, 20, and 10 tons respectively					
PolyMet Core	Winter, 2007	2007	47	19,102.5	2,801	18,174	ALS-Chemex

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Company	Date of Drilling	Date of Assaying	No. of Drill Holes	Total Footage for Group	Number of Assay Intervals **used in "Accepted Values"** Tables	Assayed Footage used in Final Database	Assay Labs
Drilling							
PolyMet Core Drilling	Summer, 2007	2007	14	5,427.5	748	5,515.7	ALS-Chemex
Totals for Exploration Drilling			371	285,756	34,192	199,672.7	
US Steel Stratigraphic Holes*	1970s?	None used	6	9,647	None used	None used	
INCO*	1956	None used	3	2,015	None used	None used	
Humble Oil Exxon*	1968-1969	None used	3	9,912	None used	None used	
Bear Creek/AMAX*	1967-1977	None used	11	8,893	None used	None used	
PolyMet/Barr Engineering (Hydrologic Testing)	2005	None used	21	3,459	None used	None used	

Notes: The number of assays used in the PolyMet database reflects numerous generations of sampling duplication. See Section 14 for the assay history.

Stratigraphic holes in the area from other projects (not necessarily drilled for this project) used to help define edges of the geologic model and provided important stratigraphic information. Note that assays, especially those for the US Steel drilling, were not all completed at the time of the original drilling.

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6.1 Historical Resource Estimates

Numerous historical resource estimates by US Steel, Fleck and NERCO were quoted by Peatfield (1999) who regarded these as preliminary in nature and lacking detailed documentation. Details on cut-off grades used in this early work are mostly absent though appear to be from 0.1 to 0.2% copper (Peatfield, 1999).

A 1970s US Steel report (in Patelke & Severson, 2006) provides a preliminary estimate of 109 million short tons of material containing 0.77% copper and 0.24% nickel which was considered to be potentially mineable by underground methods. Although not conforming to the definition of a Mineral Reserve, it was estimated at that time that the amount of this potentially mineable material could be doubled if the average combined cut-off grade was dropped by 0.2%. It is unclear how US Steel planned to process the ore.

During 2001, IMC completed mining studies and reported Measured, Indicated and Inferred categories within a pit design to 200 ft elevation (approximate final pit depth of 1,400 ft below surface) (IMC, 2001).

Resource estimate carried out by Hellman & Schofield Pty Ltd. in 2006 saw the introduction of a US$7.42 NMV cut-off, which was, according to Hellman and Schofield, roughly equivalent to a lower cut-off of 0.2% copper and 0.06% nickel.

The most recent resource estimate was carried out by Wardrop Engineering dated September 2007, which included an extension of the block model matrix down to the 0 ft elevation, a smaller block size based upon a selective mining unit determination, a new interpolation plan that honoured the geological features and statistical characteristics of the NorthMet Deposit and a new classification model.

Table 6-2 lists the historical resource estimates for the NorthMet Deposit.

PolyMet does not treat the historical estimates as current mineral resources or reserves. These estimates are historical in nature and, with the exception of Hellman & Schofield and Wardrop September 2007, pre-date and are non-compliant with NI 43-101. They are reproduced in Table 6-2 purely for a record. These estimates are no longer relevant as they are being replaced by the NI 43-101 resource estimated presented in this report.

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Table 6-2: NorthMet Historical Resource Estimate

Origin	Cut-off	Tonage (M st)	Cu%	Ni*%	Ag* (ppm)	Au* (ppm)	Pt* (ppm)	Pd* (ppm)	Co (ppm)	Notes
US Steel	Unknown	272	0.5	0.16	-	-	-	-	-	Geological resources
US Steel	Unknown	99	0.77	0.24	-	-	-	-	-	to 200 ft elevation
Fleck? (1989)	Unknown	75	0.57	0.13	2.1	0.069	0.171	0.274	-	to 800 ft elevation
Fleck (1989)	Unknown	157	0.47	0.11	-	-	-	-	-	in pit, undiluted
Fleck (1989)	Unknown	173	0.43	0.1	-	-	-	-	-	"Diluted", to 800 ft
Fleck (1990)	Unknown	154	0.48	0.11	1.7	0.068	0.133	0.454	-	in pit, undiluted
Fleck (1990)	Unknown	179	0.42	0.09	1.5	0.06	0.117	0.399	-	"Diluted", to 800 ft
NERCO (1991)	0.1% Cu	1419	0.4	0.009	1.3	0.061	0.118	0.445	-	"Global"
NERCO (1991)		808	0.43	0.11	1.5	0.061	0.116	0.437	-	In Pit
IMC 2001 Resource	0.1% Cu	362	0.301	0.084	-	0.04	0.078	0.286	66	Measured
		303	0.328	0.085	-	0.047	0.09	0.324	62	Indicated
		340	0.336	0.085	-	0.048	0.093	0.341	59	Inferred
IMC 2001 Resource	0.2% Cu	290	0.336	0.091	-	0.045	0.087	0.323	67	Measured
		255	0.359	0.091	-	0.052	0.1	0.361	62	Indicated
		275	0.379	0.094	-	0.055	0.107	0.396	60	Inferred
IMC 2001 Mineable	0.1% Cu	489	0.3	0.08	-	0.042	0.083	0.285	66	Total "Ore"
		406								Measured + Indicated
IMC 2001 Mineable	0.2% Cu	340	0.336	0.085	-	0.048	0.093	0.341	59	Total "Ore"
		290								Measured + Indicated
H&S 2006 Resource	US$7.42 NMV	133.7	0.298	0.087		0.035	0.067	0.269	77	Measured (To 500 ft elev.)
		288.4	0.266	0.078		0.033	0.066	0.231	72	Indicated (To 500 ft elev.)
		120.6	0.247	0.074		0.033	0.065	0.217	70	Inferred (To 500 ft elev.)
Wardrop Sept 2007	US$7.42 NMV	187.0	0.287	0.084	-	0.035	0.068	0.256	73	Measured (To 0.00 ft elev.)
		451.1	0.256	0.075	-	0.034	0.065	0.226	70	Indicated (To 0.00 ft elev.)
		251.6	0.275	0.079	-	0.037	0.076	0.272	56	Inferred (To 0.00 ft elev.)

Note: Cu=copper Ni=nickel Ag = silver Pd = palladium Au = gold Pt = platinum Co = cobalt

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7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The NorthMet Deposit is situated in the Duluth Complex of northeastern Minnesota. This is a large, composite, grossly layered, tholeiitic mafic intrusion that was emplaced into comagmatic flood basalts along a portion of the Mesoproterozoic (Geerts, 1994) Mid-continent Rift System. Along the western edge of the Duluth Complex, and within the Partridge River and South Kawishiwi intrusions, there are eleven known copper-nickel deposits, some of which contain platinum group elements (Figure 7-1). The NorthMet Deposit is situated within the Partridge River Intrusion, which consists of varied troctolitic and (minor) gabbroic rock types that have been subdivided into seven igneous stratigraphic units based on drill core logging. On the footwall is the Paleoproterozoic Virginia Formation, comprised of contact-metamorphosed graywackes and siltstones.

The regional and local geology are well known (Geerts et al., 1990; Geerts, 1991, 1994; Severson, 1988; Severson and Hauck, 1990, 1997; Severson and Zanko, 1996; Severson and Miller, 1999; Severson et al., 2000; Hauck et al., 1997; Miller et al., 2001, 2002). There are over 1,100 exploration drill holes on this part of the Complex, and nearly 1,000,000 ft of core have been logged or re-logged in the past fifteen years by a small group of company and university research geologists (see Patelke, 2003).

All of these igneous units, which are described in the sub section below from bottom to top, exhibit shallow dips (10°-25°) to the south-southeast. The NorthMet Deposit and the contact between the Duluth Complex and the Virginia Formation strike 56° approximately east-northeast.

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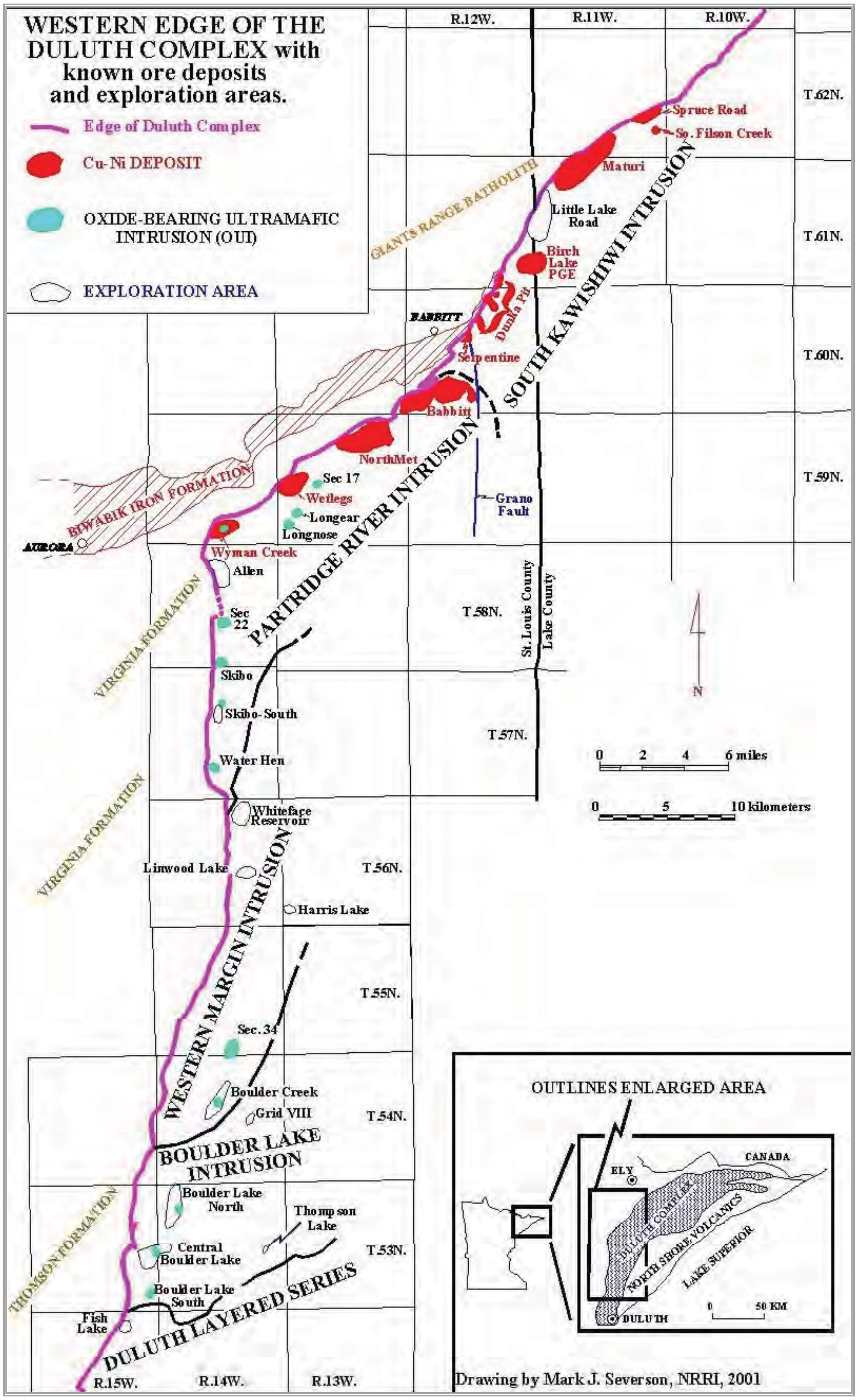
Figure 7-1: Copper-Nickel Deposits in the Duluth Complex (after Severson)



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7.1.1 Project Geology

Geology at NorthMet is well constrained by outcrop mapping (Severson and Zanko, 1996) and drill core logging on the US Steel holes, mostly by Geerts (Geerts et al., 1990, Geerts 1991, 1994), Severson (Severson et al., 2000) and Patelke (2001). This has been rather detailed logging which provided the framework for the more production oriented logging done by PolyMet during 1998-2000 (by various geologists trained by Severson) and the 2005 and 2007 (mostly by Severson and Geerts) drilling programs.

A summary of the general stratigraphy of the NorthMet Deposit shown in Figure 7-2 is outlined in the text below. Rock units and formations are listed in descending order, as would be observed from top to bottom in drill hole. NorthMet units are labeled as Units 1 through 7, bottom to top. Unit 3 is the oldest, the intrusion sequence of the other units is not clear.

The broad picture is of a regular stratigraphy of troctolitic to anorthositic rock units, dipping southeast at 20° to 25°, with basal ultramafic units commonly defining the boundaries of these units. The basal ultramafic zones tend to have diffuse tops, sharp bases, and are commonly serpentinized and foliated. Geologists have generally picked the unit boundaries at the base of these ultramafics though there are local exceptions. Economic sulfide mineralization is ubiquitous in the basal igneous unit (Unit 1) and is locally present, but restricted, in the upper units (i.e., Magenta Zone). There is no economic mineralization in the footwall rocks.

Geological domains for resource modelling are: Virginia Formation footwall rocks; a domain including the upper, higher grade parts of Unit 1, locally merged with the higher grade zones at the base of Unit 2; the remainder (lower part) of Unit 1; the Magenta Zone in Units 4, 5 and 6 in the western part of the Deposit; and the remaining, less mineralized, parts of Units 2 through 7.

Note that in the geologic solids model, Units 2 and 3 are combined as Unit 3, and Units 4 and 5 are combined as Unit 5. In both cases the combined units have more consistent thicknesses than the single units. Unit 2 and 3 may or may not be a single igneous package; there is evidence for both scenarios, while Units 4 and 5 are clearly one package with an arbitrary pick based on gradual changes in grain size and overall texture defining the unit boundaries.

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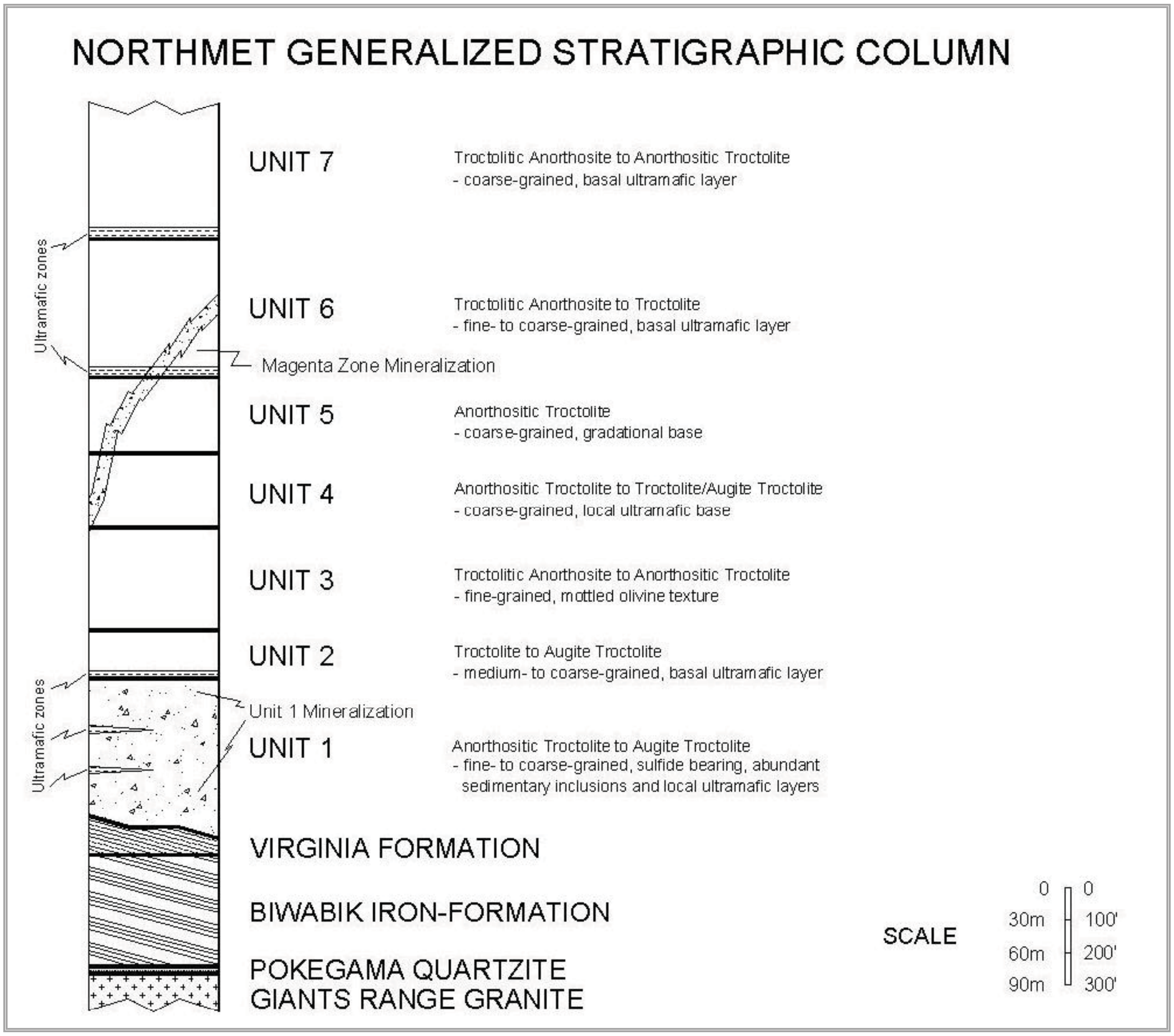
Figure 7-2: NorthMet Stratigraphic Column (after Geerts, 1994)

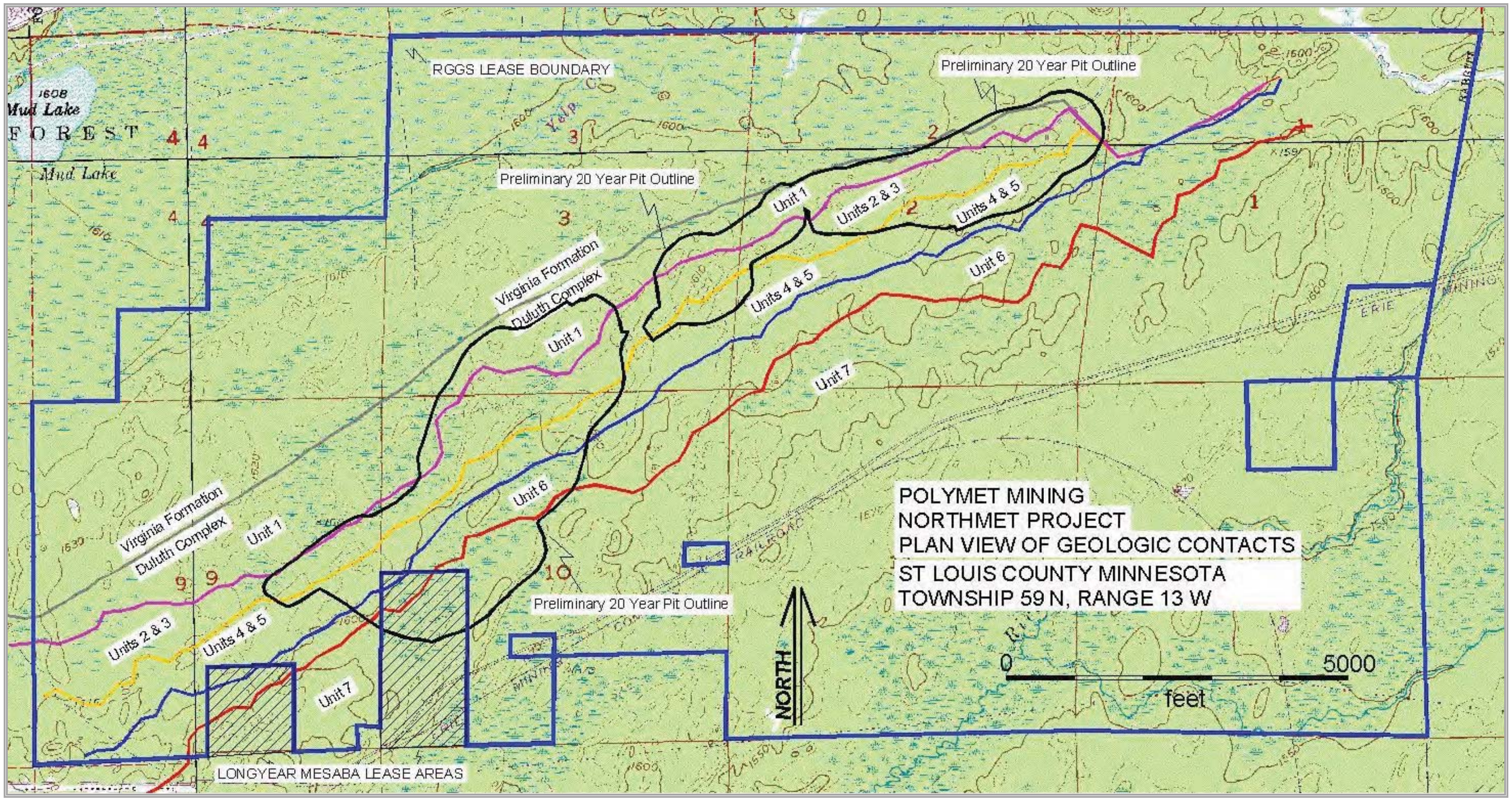


7.1.2 Rock Type and Unit Classification

Igneous rock types in the Complex are classified at NorthMet by visually estimating the modal percentages of plagioclase, olivine, and pyroxene. Due to subtle changes in the percentages of these minerals, a variation in the defined rock types within the rock units may be present from interval to interval or hole to hole. This is especially true for Unit 1.

Unit definitions are based on: overall texture of a rock type package; mineralogy; sulphide content; and context with respect to bounding surfaces (i.e., ultramafic horizons, oxide-rich horizons). Unit definitions are not always immediately clear in logging, but usually clarified when drill holes are plotted on cross-sections. In other words, to correctly identify a particular igneous stratigraphic unit, the context of the units directly above and below must also be considered. Figure 7-3 shows a plan view of the NorthMet geological contacts within the mining lease area.

Figure 7-3: NorthMet Geological Contacts



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Based on drill hole logging, the generalized rock type distribution at NorthMet is about 83% troctolitic, 6% anorthositic, 4% ultramafic, 4% sedimentary inclusions, 2% noritic and gabbroic rocks, and the rest as pegmatites, breccia, basalt inclusions and others

7.1.3 Unit Definitions and Descriptions

Unit 7

Unit 7 is the uppermost unit intersected in drill holes at the NorthMet Deposit. It consists predominantly of homogeneous, coarse-grained, anorthositic troctolite and troctolitic anorthosite. The unit is characterized by a continuous basal ultramafic sub-unit that averages 20 ft thick. The ultramafic consists of fine- to medium-grained melatroctolite to peridotite and minor dunite. The average thickness of Unit 7 is unknown due to the truncation by erosion on the surface exposure.

Unit 6

Very similar to Unit 7, Unit 6 is composed of homogeneous, fine- to coarse-grained, troctolitic anorthosite to troctolite. It averages 400 ft thick and has a continuous basal ultramafic sub-unit that averages 15 ft thick. Overall, sulphide mineralization is generally minimal, although a number of drillholes in the south-western portion of the NorthMet Deposit contain significant copper sulphides and associated elevated PGEs (Geerts 1991, 1994). Sulphides within Unit 6 generally occur as disseminated chalcopyrite/cubanite with minimal pyrrhotite. This mineralized occurrence (the Magenta Zone) is discussed in greater detail in the following sections.

Unit 5

Unit 5 exhibits an average thickness of 250 ft and is composed primarily of homogeneous, equigranular-textured, coarse-grained anorthositic troctolite. Anorthositic troctolite is the predominant rock type, but can locally grade into troctolite and augite troctolite towards the base of the unit. The lower contact of Unit 5 is gradational and lacks any ultramafic sub-unit; therefore the transition into Unit 4 is a somewhat arbitrary pick. Due to the ambiguity of this contact, thicknesses of both units vary dramatically. However, when Units 5 and 4 are combined, the thickness is fairly consistent deposit-wide.

Unit 4

Being somewhat more mafic than Unit 5, Unit 4 is characterized by homogeneous, coarse-grained, ophitic augite troctolite with some anorthosite troctolitic. Unit 4 averages about 250 ft thick. At its base, Unit 4 may contain a discontinuous, local, thin (usually no more than six inches) ultramafic layer or oxide-rich zone. The lower contact with Unit 3 is generally sharp. Overall, outside of the Magenta Zone, sulphides only occur in trace amounts within Unit 4 as finely disseminated grains of chalcopyrite and pyrrhotite.

Unit 3

Unit 3 is used as the major "marker bed" in determining stratigraphic position in drill core. It is composed of fine- to medium-grained, poikilitic and/or ophitic, troctolitic anorthosite to anorthositic troctolite. Characteristic poikilitic olivine gives the rock an overall mottled appearance. On average, Unit 3 is 300 ft thick. The lower contact of Unit 3 can be disrupted, with multiple "false starts" into

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typical Unit 2 homogenous rocks, only to go back to mottled Unit 3 with depth. The alternating sequence is common in the southwestern portion of the NorthMet Deposit and can span for many tens of ft along core before finally settling into definitive Unit 2. This most likely indicates that Unit 3 is broken up in this area and intruded by Unit 2 near the base of Unit 3. As with Units 4 and 5, the thickness of Units 2 and 3 tend to be highly variable, whereas if combined into one unit, it is more consistent deposit-wide (though not as consistent as Units 4 and 5).

Unit 3 can contain both footwall meta-sedimentary (Virginia Formation) and hanging wall basalt inclusions, which seems to indicate earliest emplacement within the intrusive sequence of the NorthMet Deposit. This exemplified by the fact that few sedimentary inclusions are found above Unit 3 and few basalt inclusions are found below it, as if Unit 3 was initially intruded between these units and eventually formed a barrier between them.

Unit 2

Unit 2 is characterized by homogeneous, medium- to coarse-grained troctolite and pyroxene troctolite with a consistent basal ultramafic sub-unit. The continuity of the basal ultramafic sub-unit, in addition to the relatively uniform grain size and homogeneity of the troctolite, makes this unit distinguishable from Units 1 and 3. Unit 2 has an average thickness of 100 ft. The ultramafic sub-unit at the base of Unit 2 is the lowermost continuous basal ultramafic horizon at the NorthMet Deposit, averages 25 ft thick, and is composed of melatroctolite to peridotite and minor dunite.

In some ways the characteristics of Unit 2 and how it fits into the igneous stratigraphy and the sequence of intrusion are ambiguous; it can be interpreted as the lower part of Unit 3, the upper part of Unit 1, or a separate unit. Based on continuity of the ultramafic boundary it seems to be a lower, more mafic, counterpart to Unit 3. The general lack of footwall inclusions in Unit 2 would argue against Unit 2 being older than Unit 1 and would indicate an intrusion sequence of 3, 1 then 2. Though Unit 2 has been historically described as barren, in the western part of the NorthMet Deposit it has mineralization grossly continuous with that at the top of Unit 1.

Unit 1

Of the seven igneous rock units represented within the NorthMet Deposit, Unit 1 is the only unit that contains significant deposit-wide sulphide mineralization. Sulphides occur primarily as disseminated interstitial grains between a dominant silicate framework and are chalcopyrite > pyrrhotite > cubanite > pentlandite. Unit 1 is also the most complex unit, with internal ultramafic sub-units, increasing and decreasing quantities of mineralization, complex textural relations and varying grain sizes, and abundant metasedimentary inclusions. It averages 450 ft thick, but is locally 1,000 ft thick and is characterized lithologically by fine- to coarse-grained heterogeneous rock ranging from anorthositic troctolite (more abundant in the upper half of Unit 1) to augite troctolite with lesser amounts of gabbro-norite and norite (becoming increasingly more abundant towards the basal contact) and numerous metasedimentary inclusions. By far the dominant rock type in Unit 1 is medium-grained ophitic augite troctolite, but the textures can vary wildly. Two internal ultramafic sub-units occur in drillholes in the southwest, and have an average thickness of 10 ft.

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Footwall: Animikie Group and Archean Rocks

The footwall rocks of the NorthMet Deposit consist of Paleoproterozoic (meta) sedimentary rocks of the Animikie Group. These rocks are represented by the following three formations, listed from youngest to oldest: the Virginia Formation; the Biwabik Iron Formation; and the Pokegama Quartzite. They are generally underlain by Archean granite of the Giants Range Batholith, but there are Archean basalts and metasediments mapped in outcrop near the Project area. The Duluth Complex is only in contact with the Virginia Formation at the NorthMet site.

Intrusion of the Complex metamorphosed the Virginia. Non-metamorphosed Virginia Formation (as found to the north of the site) consists of a thinly-bedded sequence of argillite and Greywacke, with lesser amounts of siltstone, carbonaceous-sulphidic argillite/mudstone, cherty-limey layers, and possibly some tuffaceous material. However, in proximity to the Duluth Complex, the grade of metamorphism (and associated local deformation) progressively increases, and several metamorphic varieties and textures are superimposed on the original sedimentary package at an angle to the original stratigraphy. At least four distinctive Virginia Formation varieties are present at NorthMet and informally referred to as Cordieritic Metasediments; Disrupted Unit; Recrystallized Unit; and Graphitic Argillite (often with pyrrhotite laminae). These sub-units are fully described in Severson et al., 2000.

Inclusions in the Duluth Complex

Two broad populations of inclusions occur at NorthMet: hanging wall basalts (Keweenawan) and footwall meta-sedimentary rocks. Basalts are fine-grained, generally gabbroic, with no apparent relation to any mineralization. Footwall inclusions may carry substantial sulphide (pyrrhotite) and often appear to contribute to the local sulphur content. Footwall inclusions are all Virginia Formation, no iron-formation, Pokegama Quartzite, or older granitic rock has been recognized as an inclusion at NorthMet.

Sedimentary inclusions make up about 4% of the logged rock types, and basalt inclusions sum to less than 1% of the drilling footage.

Generally, hanging wall inclusions are restricted to Unit 3 and the units above, while footwall inclusions are most abundant in Unit 1.

7.2 Mineralization

The metals of interest at NorthMet are copper, nickel, cobalt, platinum, palladium and gold. Minor amounts of rhodium and ruthenium are present though these are considered to have no economic significance. In general, with the exception of cobalt and gold, the metals are positively correlated with copper mineralization. Cobalt is well correlated with nickel.

Mineralization occurs in four broadly defined horizons or zones throughout the NorthMet property. Three of these horizons are within basal Unit 1, though they likely will not be discriminated in mining. The upper horizon locally extends upward into the base of Unit 2. The thickness of each of the three Unit 1 enriched horizons varies from 5 ft to more than 200 ft. Unit 1 mineralization is found throughout the base of the NorthMet Deposit. A less extensive (the copper-rich, sulphur-poor

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Magenta Zone) mineralized zone is found in Units 4, 5 and 6, in the western part of the NorthMet Deposit.

Mineralization occurs in two broad forms. Firstly, sulphides may be disseminated in heterogeneous troctolitic rocks (mainly Unit 1) in which the grain sizes of both silicates and sulphides widely vary. The occurrence and amount of this mineralization within drill holes can be unpredictable over the scale of 20 to 30 ft though mineralization is relatively constant in some horizons (i.e., top of Unit 1). Secondly, economic concentrations of sulphides in the upper units tend to be coarser grained and copper-rich (Units 2 to 7, particularly the Magenta Zone).

Sulphide mineralization consists of chalcopyrite and cubanite, pyrrhotite and pentlandite, with minor bornite, violarite, pyrite, sphalerite, galena, talnakhite, mackinawite and valerite. Sulphide minerals occur mainly as blebs interstitial to plagioclase, olivine and augite grains, but also may occur within plagioclase and augite grains, as intergrowths with silicates, or as fine veinlets. Small globular aggregates of sulphides (less than two centimetres) have been observed in core and in the small test pit on the site. The percentage of sulphide varies from trace to about 5%, but is rarely greater than 3%. Local massive sulphide is present, but rare. Platinum, palladium, and gold are associated with the sulphides as well as in tellurides and bismuthides.

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8 DEPOSIT TYPES

The NorthMet Deposit is a large-tonnage, disseminated accumulation of sulphide in mafic rocks, with rare massive sulphides. Copper to nickel ratios generally range from 3:1 to 4:1. Primary mineralization is probably magmatic, though the possibility of structurally controlled re-mobilization of the mineralization (especially PGEs) has not been excluded. Sulphur source is both local and magmatic (Theriault et al., 2000). Extensive detailed logging has shown no definitive relation between specific rock type and the quantity or grade quality of sulphide mineralization in the Unit 1 mineralized zone or in other units, though the localized noritic to gabbronoritic rocks (related to footwall assimilation) tend to be of poorer PGE grade and higher in sulphur.

Footwall faults are inferred from bedding dips in the underlying sedimentary rocks, considering the possibility that Keweenawan syn-rift normal faults may affect these underlying units and show less movement, or indeed no effect on the igneous units. Nonetheless, without faults, the footwall or igneous unit dips do not reconcile perfectly with the overall slope of the footwall. There are some apparent offsets in the igneous units, but definitive and continuous fault zones have not been identified. So far, no apparent local relation between the inferred location of faults and mineralization has been delineated.

Outcrop mapping (Severson and Zanko, 1996) shows apparent unit relations that require faults for perfect reconciliation. However, as with information derived from drill core, neither igneous stratigraphic unit recognition, nor outcrop density, is sufficiently definitive to establish exact fault locations without other evidence.

There is a wealth of regional (and some local) geophysical data available, though the resolution of core logging and field mapping is probably better than that of the geophysics, hence while the geophysical data is interesting, it has not yet been useful at delineating the structural geology of the site nor proved to be a guide to mineralization.

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9 EXPLORATION

Exploration history is outlined in Section 6. In general, the early drilling by US Steel is widely spaced but comparatively regularly distributed (approximately 600 ft by 600 ft), with some omissions that left substantial undrilled areas, especially down-dip. Subsequent programs by PolyMet were first focused on extracting metallurgical samples and on proving the up-dip and more readily accessible parts of the NorthMet Deposit. Besides extensive in-fill drilling since 2005, PolyMet has also expanded the definition of the mineralized zones to the west and southwest. In particular, it has become evident that the Magenta Zone, located in the upper units in the western part of the NorthMet Deposit, is much more robust than previously thought.

Those parts of the NorthMet Deposit at greater depth largely continue to have the original US Steel drill-hole spacing, which, in the eastern half of the NorthMet Deposit, is approximately 600 ft by 1,200 ft.

Drill spacing in the deepest known section of the NorthMet Deposit is approximately 1,200 ft by 1,200 ft. The Deposit is definitely open at depth and along strike. The deeper parts of the NorthMet Deposit (below about 1,600 ft from surface) may be of interest in the future, but they are considered to fall outside the scope of the current evaluation.

Drill hole spacing averages between 190 and 200 ft in the area of the resource model. This excludes holes drilled for metallurgical or geotechnical purposes. Distance studies show that 50% of the drillhole intercepts within Unit 1 will be within a 197 ft distance from another hole. In the Magenta Zone, 50% of the drillhole intercepts will be within a 190 ft distance from another hole. The best drilled area is in the vicinity of the preliminary optimum pit. This area also contains near-surface mineralization and is drilled at a spacing of about 150 ft (excluding geotechnical and metallurgical holes) from 171 holes.

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10 DRILLING

There have been four major (and one minor) drilling campaigns on the property as shown in Figure 10-1. This discussion is largely taken from Patelke and Geerts (2006).

In all cases, drilling has shown a basal mineralized zone (Unit 1) in heterogeneous troctolitic rocks with the highest values at its top and with grades generally diminishing with vertical depth along drill holes. Grade appears to increase down dip, but as depth increases less information is available. The main ore zone is from 200 to 1,000 ft thick, averaging about 450 ft. Mineralization sub-crops at the north edge of the NorthMet Deposit and continues to depths of greater than 2,500 ft. Sampling on the longest holes is sparse, with little in-fill work done since the original US Steel sampling (PolyMet took about 700 samples from these longer holes in spring of 2006, these data are included in the drilling database)

While the concept of some structural control on mineralization is valid (i.e., proximity to a vent system or re-mobilization of some metals) no evidence collected to date fully supports this view. More likely, this is a magmatic sulphide system which was then contaminated by sulphur from locally assimilated footwall rocks and modified to some extent by (late magmatic?) hydrothermal action.

Core recovery (Table 10-1) is reported by PolyMet to be upwards of 99% with rare zones of poor recovery. Rock quality designation (RQD) is also very high, upward of 85% for all units except in the Iron formation. Experience in the Duluth Complex indicates that core drilling has no difficulty in producing samples that are representative of the rock mass. Rock is fresh and competent and the common types of alteration (sausserization, uralization, serpentinization and chloritization) in the NorthMet Deposit are not those that affect recovery. Core recovery was recorded by US Steel and PolyMet in its earlier work and for the smaller diameter (NQ2 and NTW) drilling in since 2005. There is no readily apparent relation of recovery to sulphur content or rock type. Values in excess of 100 may arise from errors associated with assembling broken core or from core runs that are slightly longer than the core barrel. AGP comment that the core recovery appears very good in the holes that were inspected during the site visits.

In short-range detail, the NorthMet Deposit geology is subtle and complex. However, mineralogical and textural variation occurs within narrow ranges and at the mining scale, the overriding lithology will be troctolite to augite-troctolite (plagioclase>olivine>>pyroxene with biotite and minor ilmenite). The known ultramafic horizons are thin enough, and metasedimentary inclusions small enough, that material handling will homogenize the plant feed, as accounted for in the bulk samples. In general, rocks are medium- to coarse-grained, fresh, and competent.

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Figure 10-1: Drillhole Collar Location by Campaign

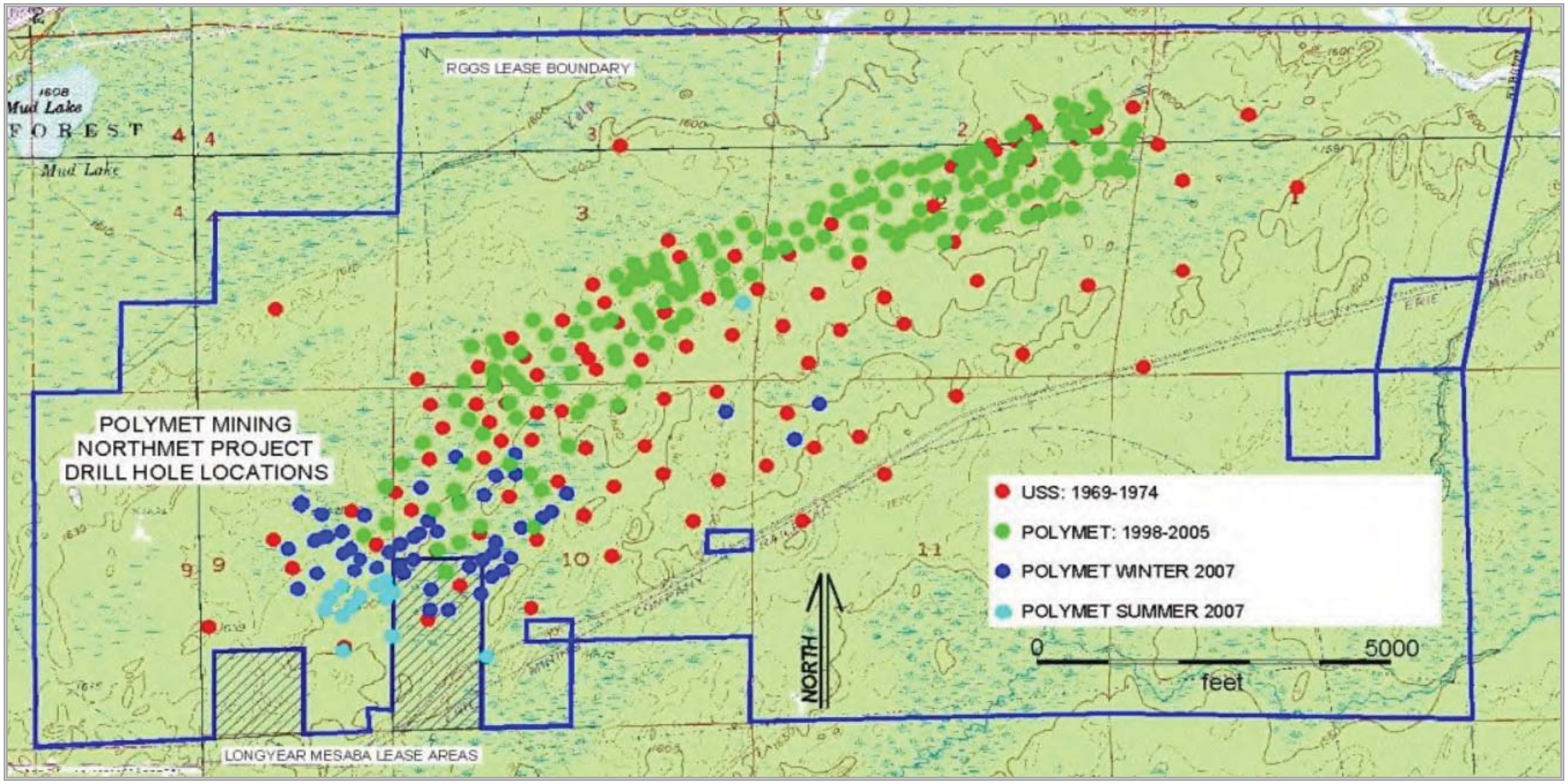

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Table 10-1: Summary of Core Recoveries and RQD Measurements (includes all drilling through summer 2007)

Unit	Recovery Count	Recovery Percentage (%)	RQD Count	RQD Percent
1	8,906	99.9	4,194	91.8
2	1,879	99.5	968	90.3
3	4,374	100	2,632	93.5
4	2,160	100	1,063	96.4
5	1,901	100	838	94.3
6	2,262	100	1,041	94.7
7	951	99.3	396	87.4
Virginia Formation	2,095	99.7	1,069	87.6
Inclusions	62	98.1	57	86.6
Biwabik Iron Formation	381	100	60	79.8
Duluth Complex Average		99.96		92.82

10.1 Drilling Campaign

10.1.1 US Steel Drilling, 1969-1974

From 1969 to 1974, US Steel drilled 112 holes across the property. Drilling began in an attempt to intersect a geophysical conductor (virtually all of the deposits in the area were originally drilled on geophysical targets) and the first hole hit three ft of massive sulphide with 4.8% copper, 115 ft from the surface. Drilling continued, without discovery of any more such dramatic results and eventually defined a broad zone of low-grade copper-nickel sulphide mineralization. Further drilling indicated that the original geophysical target was graphitic argillite in the footwall, rather than any mineralization in the Duluth Complex.

US Steel assayed only about 22,000 ft of the 133,000 ft they drilled, generally on 10 ft intervals. Their focus was on developing an underground reserve and sampling was limited to zones of continuous "higher grade" mineralization. As in many exploration projects, sampling focused on the expected main ore body, not more scattered intervals or assumed waste rock. US Steel was aware of the PGE value from the assaying of concentrates derived from bench work and test pits, but did no assaying for these metals on drill core. Nearly all core was BQ size, and only 14 of the holes were angled (all to the northwest, grid north). Hole depths ranged from 162 ft to 2,647 ft, averaging 1,193 ft. Five holes were over 2,500 ft in length.

US Steel drilling was by Longyear. Virtually all of the core from this program exists, is properly stored, and is available for further sampling. Seventeen US Steel holes were "skeletonised" after assaying, with only a ft kept for each five or ten ft "un-mineralized" and unsampled run. Core was split by US Steel using a manual core splitter. Samples submitted for assay were half core. US Steel assays were done at their own laboratories; most of these have since been re-assayed by ACME Laboratories (ACME) or ALS Chemex (Chemex). Drilling by PolyMet near some of the locations of skeletonised holes

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has indicated the possibility that some mineralized intervals may have been missed and disposed of in the skeletonising process.

The US Steel geologists logged all their holes, but neither recognized nor documented any comprehensive igneous stratigraphy. Mark Severson of the Natural Resources Research Institute (NRRI), Duluth, Minnesota began re-logging these holes in the late 1980's as part of a Partridge River intrusion geochemistry project. He quickly recognized Unit 3 as a marker horizon, which led to reliable correlations among the other units.

Steve Geerts, working for the NRRI with Fleck Resources (PolyMet precursor), refined the geologic model for the NorthMet Deposit in light of this igneous stratigraphy. This basic model is still considered by PolyMet to be valid and currently guides the interpretation of the NorthMet Deposit (Severson 1988, Severson and Hauck 1990, Geerts et al. 1990, Geerts 1991, 1994).

10.1.2 NERCO Drilling, 1991

NERCO conducted a minor drilling campaign in 1991—four holes at two sites. At each site a BQ sized core hole (1.43 inches) was drilled and sampled from collar to bottom of hole. A PQ (3.3 inch) hole twinned each of these two holes and was sent in its entirety for metallurgical work on the assumption that the assays on the smaller diameter core would represent the larger diameter core. Both sets of holes twinned existing US Steel holes (Pancoast, 1991).

One-hundred and sixty-five assays were taken from the smaller diameter cores and processed at ACME.

10.1.3 PolyMet Drilling, 1998-2000, Reverse Circulation Holes

PolyMet drilled 52 vertical reverse circulation (RC) holes to supply material for a bulk sample in 1998 to 2000. These holes twinned some US Steel holes and others served as in-fill for parts of the NorthMet Deposit. The drilling was done by a contractor from Duluth with extensive RC experience and was carried out in both summer and winter. The type of bit and extraction system used (cross-over sub or face-sampling) is not known. Available recorded sample weights indicate a recovery of at least 85%. Metallurgical core drilling in February and March 2005 approximately twinned some of these RC holes.

The PolyMet drilling in 1998 to 2000 targeted the up-dip portions of the NorthMet Deposit and was essentially in-fill drilling. Reverse circulation holes averaged 474 ft in length with a minimum of 65 ft and a maximum depth of 745 ft. Core holes averaged 692 ft in length with a minimum of 229 ft and a maximum depth of 1,192 ft (this does not include the three RC holes completed with AQ core).

The RC holes were assayed on five ft intervals. Six inch reverse circulation drilling produced about 135 lb to 150 lb of sample for every five ft of drilling. This material was split using a riffle splitter into two samples and placed in plastic bags and stored underwater in five gallon plastic buckets. A 1/16th sample was taken by rotary splitter from each five ft of chip sample and assayed. The assay values were used to develop a composite pilot plant sample from bucket samples. Actual compositing was

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done after samples had been shipped to Lakefield (Patelke and Severson, 2006). A second 1/16th sample was sent to the Minnesota Department of Natural Resources for their archive.

Chip samples were collected and later logged at the PolyMet office. PolyMet retains these samples in their warehouse. Logging is obviously not as precise as that for core, but the major silicate and sulfide minerals can be recognized and location of marker horizons derived. The underlying metasedimentary rocks (Virginia Formation) are easily recognized and finding the bottom of the NorthMet Deposit is relatively straightforward. Where rock recognition is difficult, the higher zinc content of the footwall rocks can help define the contact.

10.1.4 PolyMet Drilling, 1999 to 2000, Diamond Core Holes

The first PolyMet core-drilling program was carried out during the later parts of the RC program, with three holes drilled late in 1999 and the remainder in early 2000. There were seventeen BTW (1.65 inch) and fifteen NTW (2.2 inch) holes all of which were vertical. Three RC holes were re-entered and deepened with AQ core.

These holes were assayed from top to bottom (with rare exception) on five ft lengths. Samples were half core. Cutting was done at the PolyMet field office in Aurora, Minnesota.

Core logging was done at the PolyMet office by a variety of geologists, all trained in recognition of the units and the subtleties of the mineralogy and textures by Mark Severson of the NRRI.

10.1.5 PolyMet Drilling, 2005, Diamond Core Holes

PolyMet's 2005 drilling program had four distinct goals: collection of metallurgical sample; continued in-fill drilling for resource estimation; drilling outward from the margins of the well drilled area to expand resource; and collection of geotechnical data through core logging and recovery of oriented cores. The program covered 109 holes for 77,165 ft. These included:

- 54 one inch diameter holes for metallurgical sample (6,974 ft) drilled by Boart-Longyear of Salt Lake City in February-March 2005.
- 12 PQ sized holes (core diameter 3.3 inches) for 6,897 ft, mostly used for bulk sample material, but with a few holes intended as in-fill. The PQ holes were also all drilled in February-March of 2005.
- 52 NTW sized holes (2.2 inches) totalling 41,403 ft for resource definition.
- 30 NQ2 sized holes (2.0 inches) totalling 21,892 ft for resource definition and geotechnical purposes. The NTW and NQ2 size core was drilled in February-March and September-December of 2005.

About 11,650 multi-element assays were collected from the 2005 drilling program. Another 1,790 assays were performed on previously drilled US Steel and PolyMet core during that time. All assaying was by ALS-Chemex.

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Of the 109 holes drilled in 2005, 93 were angled, generally to grid north at dips of -60° to -75°. Sixteen NQ2 sized holes were drilled and marked as oriented core, ten to grid south and six to grid north, at varying dips, for geotechnical assessment across the NorthMet Deposit. These holes targeted expected positions of pit walls as defined by Whittle pit shells developed by mining consultants AMDAD and available in January 2005. These locations have proved to be reasonable for more recent iterations of pit design.

Besides extensive assaying for "pay" elements during this program, about 900 core intervals were analyzed for "whole rock" oxides, about 300 samples were analyzed for Rare Earth Elements (REE), and thousands of density measurements were taken. This data is used to support resource evaluation as well as waste characterization efforts for permitting.

Separately, about 100 samples from previously drilled and analyzed core were submitted for humidity cell testing. These samples represented a broad cross-section of Units, rock-types, metals content, and sulphur content. In addition, these humidity cell samples were all re-assayed, analyzed for whole rock and assessed in thin-section and by micro-probe.

10.1.6 PolyMet Drilling, 2007, Diamond Core Holes

In 2007, PolyMet conducted two drilling programs, a winter program for 47 holes over 19,102.5 ft and a summer program for 14 holes over 5,437.5 ft. The first 16 winter holes were NTW sized, the rest from both programs were NQ2 sized core. Most of these holes were angled to north-northeast (azimuth 326°).

For the 2007 holes the minimum length was 148 ft, the maximum length was 768.5 ft and the average length was 402 ft.

During the site visit, AGP noted that the drill core handling procedure carried out by Polymet met or exceed industry standard. Drill hole orientation and dip results in intersection with the lithological units that are more or less normal to the main structural trend and are appropriate for the style of mineralization present on the NorthMet Deposit.

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11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Sections 11.1 and 11.2 were extracted from the Hellman 2005 report.

11.1 Sampling Methods

Original US Steel sampling, generally on 10 ft intervals, honoured some, but not all, of the geological boundaries that were encountered. The PolyMet RC sampling transgressed boundaries, though the five ft chip samples diminish the opportunity for this to be of any consequence in a bulk mining (15 to 20 ft bench or greater) scenario.

Sampling of US Steel core by Geerts, Severson, and Pateke of NRRI at various times usually was on five ft samples and seldom crossed any significant geologic boundaries. Core sampling by PolyMet in 1999 and 2000 was usually on five ft intervals and crossed unit boundaries, as with the RC samples, the short sample length negates any major effect from this sampling choice. Sampling by PolyMet on the US Steel core in 2005 was generally on 10 ft intervals, but did not cross any major geologic boundaries and included some shorter intervals. Sampling of in-fill (NTW and NQ2) core in 2005 and 2007 used five ft samples in the main mineralized zone and 10 ft in the upper zones. This was adjusted to use smaller intervals in the upper parts with visible mineralization and did not cross-geologic boundaries.

Large diameter core collected for metallurgical sample was sampled and assayed by the box with the goal of minimizing re-handling during the preparation and compositing of the bulk sample. Four-inch core was sampled on an average interval of 3.45 ft, and PQ core was sampled on an average interval of 4.47 ft.

Table 11-1 shows average length of samples in Unit 1 and all other units for holes used in the resource model. Approximately 90.5% of Unit 1 and about 55.5% of the other units have been sampled project-wide. About 70% of the total exploration drilling by US Steel and PolyMet has been sampled across the property. Over 97% of the drilling intercepting the anticipated 20-year pit has been sampled.

Table 11-1: Sample Lengths (includes all drilling through summer 2007)

	Average Sample Length in Unit 1 (ft)	Average Sample Length in Other Units (ft)
US Steel Original Core	6.3	7.1
PolyMet RC Drilling	5.0	5.0
PolyMet Core Drilling	4.8	7.2
All Drilling	5.3	7.0

Sampling in Unit 1 (the main mineralized zone) is mostly continuous through the zone for all generations of drilling. The older PolyMet RC and core holes have continuous sample through the upper waste zones (which do have some intercepts of economic mineralization). Work in 2005 and 2006 essentially completed the sampling of historic US Steel core within the area likely to be mined. This broad sampling limits the possibility of bias in the sample set. The 2005 and 2007 sampling has been continuous along the drill hole. There is some US Steel core below the current block model to be sampled in the future. The overall effect on the resource should be minimal and is expected to be positive.

11.2 Preparation and Analysis

11.2.1 Sample Preparation Pre-2000

Bright (2000), an employee of ALS Chemex, summarized the sample preparation history of the Project up to that point, the following is an extract from his summary.

- Pre-1996, Lerch Brothers, and State of Minnesota crushed in a jaw crusher to about 1/4 inch and pulverized about 250g in a Bico type plate pulverizer to about -100 mesh (149 µm). Bondar Clegg also did some work on the Project, crushing about the same, but pulverizing in a ring mill to -106 µm.
- In 1997, samples were sent directly to Acme Laboratories, where they crushed to finer than 1/4 inch and pulverized to about 149 to 106 µm range.
- In 1998, Lerch Bros. crushed and pulverized about 250 g in an older ring mill to finer than 149 µm and sent to Acme.
- In 1999, Lerch Bros. prepped as in 1998, but sent to Chemex for analysis. Early on in the Project, I requested a finer grind out of Lerch Bros, and they accomplished it. (-106 mic). Also in 1999, some drill cuttings and core were directly picked up by ALS Chemex. This is what we did in Thunder Bay:
- 3.5-4 kg of RC or percussion samples were dried and split to obtain two splits of each sample. Core samples of 2.5 to 3 kg were crushed to pass >70% -2 mm, 200 to 300 g were split out. Both r.c. cuttings and crushed core were shipped to Toronto for pulverizing in a ring mill to >95% -106 µm (-150 Tyler mesh).
- We also took selected core samples and crushed to -1/2 inch and put in a poly bottle, purged with nitrogen, and capped and sealed for special met / enviro work.

11.2.2 Sample Preparation Pre-2005

In summary (Gatehouse 2000a), pre-2005 drilling has been prepared in either of two ways depending on drill type or on the work load of Lerch Bros in Hibbing.

- 5' of 6" RC chips
- 1/16 split using an Eklund rotary Splitter (3 to 4 kg)
- Jaw crush >> Gyratory Crusher >> Rolls crusher
- 1/16 split to 200 to 250 g for pulverizing to 109 µm (some data poorly pulped to 150 µm)
- 5' of 1/2 core (1.65" and 2.2"diameter, BTW, NTW) at Chemex
- Rhino (Jaw) Crush to 2 mm

- Split 200 to 250 g for pulverizing to 109 µm
- 5' of 1/2 core (1.65" & 2.2"diameter, BTW, NTW) at Lerch Bros.
- Jaw Crush >> Gyratory Crusher
- Split 200 to 250 g for pulverizing to 149 µm.

11.2.3 Sample Preparation 2005 through 2007

The 2005 and 2007 sample preparation varied at the cutting and sampling stage with ½ core samples used for all NQ2 and NTW drilling and 1/8 core samples used for all four inch and PQ drilling. For smaller diameter core, the field duplicates were ¼ core, for the larger cores the field duplicates were 1/8 core.

All sample preparation after cutting was done at ALS Chemex in Thunder Bay, Ontario, and all analyses at ALS-Chemex in Vancouver, B.C. Transport from Hoyt Lakes to Thunder Bay was by truck driven by ALS-Chemex employees and under ALS-Chemex custody.

Sample preparation methods were as follows:

- A 10 lb to 15 lb sample was crushed in a single stage crusher to 90% -2 mm
- A 500 to 700 g sample was split off and pulverized to -150 mesh in one pass
- 1 in 20 samples also duplicated at the crusher
- Approximately 200 g for each sample were sent to Vancouver
- All samples were analyzed for multi-element ICP package (four acid digestion) and PGE
- Depending on batch size and other factors 1 in 10 to 1 in 20 samples were submitted as pulps for analysis for whole rock major elements, aqua regia digestion, REE and iron oxide (FeO)
- A standard, coarse blank (iron formation) or core (field) duplicate was submitted at a rate of one in every 12 samples
- LECO Corporation (LECO) furnace sulphur was run on 1 in 10 samples.

11.3 Analytical History

The following discussion is derived largely from Patelke and Geerts (2006), an internal company report on the compilation and history of the newly revised PolyMet drilling database.

There are eight generations of sample preparation and analyses that contribute to the overall project assay database:

- Original US Steel core sampling, by US Steel, 1969-1974
- Re-assaying of US Steel pulps and rejects, selection by Fleck and NRRI, 1989-1991

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- Sampling of previously unsampled US Steel core, sample selection by Fleck and NRRI in 1989-1991
- Sampling of two NERCO drill holes in 1991
- Sampling of RC cuttings by PolyMet in 1998-2000
- Sampling of PolyMet core in 2000
- Sampling of previously unsampled US Steel core (sample selection work done by NRRI, done in two phases) in 1999-2001

1. Sampling of PolyMet core from 2005 drilling, continued sampling of previously unsampled US Steel core in 2005-2006, and sampling from 2007 drilling, with continued protocols in place since 2005.

2. Employees of PolyMet (or Fleck Resources) have been either directly or indirectly involved in all sample selection since the original US Steel sampling. Sample cutting and preparation of core for shipping has been done by PolyMet employees or contract employees. Reverse circulation sampling at the rig was done by, or in cooperation with, PolyMet employees and drilling contractor employees.

3. US Steel took about 2,200 samples, mostly ten ft in length, and assayed for copper, nickel, sulphur, and iron. Assays were done at two US Steel laboratories in Minnesota, the Applied Research Laboratory (ARL) in Coleraine (now the NRRI mineral processing laboratory), and the Minnesota Ore Operations Laboratory (MOO) at the MinnTac Mine in Mountain Iron. Most of the original US Steel samples have been superseded by ACME and Chemex re-assays which included many more elements.

4. Analytical method at these US Steel laboratories is uncertain (AAS?). While standards were developed and used (as evidenced by documents in PolyMet files), it is not thought the standards were inserted into the sample stream in a blind manner. It is likely that these were used for calibration or spot checks.

5. There are less than 200 sets of US Steel copper-nickel values that remain in the database.

6. PolyMet used 63 coarse reject US Steel samples, weighing from five to seven pounds each, to create three standards in 2004. The 2004 assay results are consistent with estimates based on original US Steel assays of drill core. The ALS-Chemex results are shown in Table 11-2.

Table 11-2: ALS-Chemex Assays compared with US Steel Assays

	Cu %	Ni%	S %
Standard 1 expected value based on 1969 to 1974 US Steel assays	0.18	0.08	1.04

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	Cu %	Ni %	S %
Standard 1 assayed value-2004 – Chemex	0.20	0.11	1.08
Standard 2 expected value based on 1969 to 1974 US Steel assays	0.36	0.14	0.88
Standard 2 assayed value-2004 – Chemex	0.37	0.15	0.82
Standard 3 expected value based on 1969 to 1974 US Steel assays	0.55	0.18	1.17
Standard 3 assayed value-2004 – Chemex	0.57	0.21	1.04

Averages are based on twenty samples of each standard with 4-acid assays completed in 2004. In all cases, the US Steel results are slightly understated relative to the Chemex values. These standards have been used throughout the 2005 and 2007 programs.

The re-assaying of US Steel pulps and sampling of previously unsampled core completed in 1989-1991 was sponsored by Fleck Resources and partially involved cooperative work with the NRRI in Duluth. A large number of pulps and coarse reject from the original US Steel drilling were re-assayed for copper, nickel, PGE, and a full suite of other elements. The NRRI's contribution was the selection and sampling (and re-logging) of previously unsampled core. This was the first large scale testing for PGE done on the Project.

About 2,600 of these analyses are in the current PolyMet database. All of this analytical work was done at ACME Laboratories by aqua regia with ICP-ES for copper and nickel. Gold, platinum, palladium were by lead-oxide (PbO) collection fire assay/AAS finish. There is uncertainty about the level of standards used at ACME, though it is certain that they used some duplicates. There is agreement between the ACME assays done on pulps and rejects and the original US Steel work. PolyMet is using the US Steel sulphur value for most of these intervals. Sample preparation for all this work is thought to have been done by ACME.

The two NERCO BQ core holes (1991, 162 samples) were analyzed at ACME by the same methods.

There are 5,216 analyses from the RC drilling in the current PolyMet database. The 1998 RC drilling program started with all analyses being sent to ACME and check assays going to Chemex. RC sample collection involved a 1/16 sample representing each five ft run. These were sent to Lerch Brothers of Hibbing Minnesota (Lerch), for preparation, and then sent to ACME for analysis. It is not certain that all samples were prepared at Lerch.

Part of the way through the RC program, PolyMet switched laboratories and sent the samples to Chemex, with ACME undertaking check assays. Analytical methods for the RC samples were aqua regia digestion, fire assay for PGE, and ICP-AES for other elements. LECO furnace sulphur was run on nearly every sample.

Table 11-3 details the distribution and source of the assays for the RC drilling.

Table 11-3: Assaying of RC Samples

	Percent of Samples in Database

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	Percent of Samples in Database
ACME	21
Chemex	41
Chemex Re-run (chosen over ACME or Chemex)	38

The PolyMet core drilling has all been assayed by ALS Chemex. A matrix problem was discovered on some copper and nickel assays in the earlier groups in 2000. The problem was rectified and affected samples were re-assayed (eventually including some RC samples). Sample preparation was done at Chemex, though some may have been done at the Lerch facility — various original Chemex laboratory certificates show both "received as pulp" and give grind directions. ACME ran the check assays on these samples.

Some samples on US Steel in 2000 core were done through ACME.

On pre-2005, post US Steel sampling, intervals were generally five ft, sometimes adjusted for geological breaks. Analyses were aqua regia digestion with fire assay for PGE and ICP-AES for other elements. LECO furnace sulphur was run on most intervals. During this program standards and blanks were inserted into the sample stream.

Table 11-4 details the distribution and source of assays for PolyMet core drilling.

Table 11-4: Assaying of Samples from all Core Drilling on Project

	Percent of Samples in Database
ACME	6%
Chemex	91%
Chemex Re-run	3%
USS	< 1%

Samples (collected by Severson et al., in 1999-2000 and Patelke, in 2000-2001) of previously unsampled US Steel core were assayed by ALS Chemex. These samples were sawn at the Coleraine laboratory by University of Minnesota employees. At various times samples were prepared at the Coleraine laboratory, Lerch, and probably by ALS Chemex.

Assays were by aqua regia digestion with fire assay for PGE and ICP-AES for other elements. LECO furnace sulphur was run on most intervals. During this program, standards and blanks were inserted into the sample stream.

Samples were generally five ft in length, with some adjustments to avoid crossing geologic boundaries. This work was intended to supplement and in-fill the database, primarily in the Unit 1 mineralized zone as well as to provide some geochemical data for waste characterization.

The 2005 drilling and 2005-2006 sampling used four acid digestions on all samples, with aqua regia also done on about 1 in 10 samples. Since 2005, all samples have honored geological contacts.

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PolyMet continued in 2005 and 2006 the process of assaying previously un-sampled US Steel core, adding about 1,700 assays during 2005-2006. The majority of this is in the anticipated 20 year pit.

Table 11-5 shows previously un-sampled intervals of US Steel core that were sampled by Severson et al (1999-2000) and Patelke (2000-2001).

No sieve tests are available for pre 2005 work. These were performed for samples from the 2005 and 2007 drilling programs.

Table 11-5: Details of Sampling of US Steel Core by PolyMet

	Number of Samples in Database from each Laboratory	Minimum Number of Duplicates and/or Re-runs
Chemex (Post Re-run)	5,032	229

11.4 Quality Assurance and Quality Control

A comprehensive QA/QC program involving the use of coarse blanks, standards and duplicates has been instigated under the direction of Hellman and Schofield and Lynda Bloom of ASL, Toronto. This process consisted of the production of three matrix-matched standards from the Duluth complex, sample preparation and homogenization, homogeneity testing, formulation of recommended values based on a round robin and routine insertion of standards on an anonymous basis. The three standards have copper concentrations in the approximate range 0.15 to 0.60% and nickel from 0.1 to 0.2%. Homogeneity of pulps, as determined by coefficients of variation from 20 replicate assays, is excellent with, for example, values less than 2% for copper and nickel and less than 5% for palladium. Analytical method for the matrix match standard was ALS Chemex code ME-ICP61, 4 acid digestion ICP with AES finish for Co, Cu, Mo, Ni and Zn. For the platinum group metals and gold, ALS Chemex code PGM-ICP23, which is a 30 g fire assay with ICP-AES finish. Total sulphur was done by LECO furnaise Code S-IR08. Table 11-6 shows the expected value of the standards.

Table 11-6: Standard Reference Material

	Standard 4-1		Standard 4-2		Standard 4-3	
Element	Average	Std.Dev	Average	Std.Dev	Average	Std.Dev
Co (ppm)	90.1	10.44	95.10	10.64	110.73	11.11
Cu (%)	0.201	0.008	0.378	0.009	0.589	0.019
Mo (ppm)	13.87	1.78	9.61	1.36	12.25	1.40
Ni (%)	0.109	0.007	0.143	0.009	0.197	0.015
Zn (ppm)	174.15	14.62	116.77	12.18	124.76	12.65
Au (ppb)	57.85	12.70	33.32	6.48	54.18	7.36
Pt (ppb)	36.54	9.50	55.76	11.15	125.52	15.55
Pd (ppb)	117.52	10.66	238.95	14.64	518.05	22.18
S (%)	1.17	0.04	0.91	0.04	1.15	0.005

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Reference materials (RMs) and Blanks were inserted at a rate of one blank with every 35 samples and 1 SRM with every 36 samples. Duplicate are submitted with every 36 samples. Typically, there are very few assay failures found in the drill programs with Chemex and they are investigated in batches by PolyMet. Depending on the nature of the failures, samples may be re-run or discarded from the data set.

11.4.1 Linda Bloom Assessment of the QA/QC Program to 2005

AGP observes that Lynda Bloom of Analytical Solutions Ltd is independent from the issuer and specialized in sampling and analytical procedures, QA/QC program design, QC review and laboratory audits. She is very well known in her field and review the PolyMet 2005 quality control program. AGP reviewed the report provided to PolyMet from Lynda Bloom and agree with its findings.

11.4.2 AGP Assessment of the QA/QC Program to 2007

AGP reviewed the data provided by PolyMet for the two main pay elements. Out of 526 RMs submitted between April 2005 through to June 2007, there was 54 copper failures (10.2%) most of which occur with RM 4-2 showing a 21.2 % failure rate. The other two RMs 4-1 and 4-3 indicated a failure rate of 2.9% and 5.0%, respectively. All copper standards showed increase deviation from the expected values for samples submitted after April 30, 2007.

For Nickel, out of 526 RMs submitted during April 2005 through to June 2007, there were six nickel failures (1.14%).

The exact number of batches resubmitted to the laboratory is unknown by the author.

11.5 Databases

It is AGP's opinion that PolyMet staff has made a strong commitment to the geological and assay database and have, as far as is possible, produced a database that is complete, well documented and traceable.

11.6 Core Storage and Sample Security

The US Steel core has been stored, either at the original US Steel warehouse in Virginia, Minnesota during drilling, or more recently at the Coleraine Minerals Research Laboratory (now a part of the University of Minnesota). Core has been secured in locked buildings within a fenced area that is locked at night where a key must be checked out. The NERCO BQ size core is also stored at this facility.

The PolyMet core and RC reference samples were stored in a PolyMet leased warehouse in Aurora, Minnesota during drilling and pre-feasibility. Core and samples were then moved in 2002 to a warehouse in Mountain Iron, Minnesota where they remained until 2004. They were then moved to a warehouse at the Erie Plant site in Hoyt Lakes. Access to this warehouse has been limited to PolyMet employees.

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11.7 Comments on Section 11

AGP is of the opinion that PolyMet went to considerable effort to ensure that the laboratory procedures, QA/QC protocol, the use of a matrix match standards and the continuous sampling of most of the historical holes. There was a weak degradation of precision at the laboratory starting April 30, 2007 to the end of the drill program. This is noticeable in both copper and nickel assays. The degradation was not sufficient to create a material change that would affect the resource model grade.

The distribution of the core drilling versus RC in the database (Table 11-7) shows that 91% of the holes used in the resources are core holes. RC holes amount only to 9% of the database. The reproducibility of the grade between RC and Core holes was investigated and found to be within acceptable limit. A discussion on that subject has been inserted in Section 12 of this report.

Table 11-7: Distribution of Drillhole Types

CORE_RC	No. of Holes	Length of Hole	Percent of Total
CORE	299	227,665.50	80
CORE-SKEL (partial assays)	17	30,745.00	11
RC	52	24,650.00	9
RC-CORE	3	2,696.50	1
		285,757.00	

The QP regards the assay database and analytical procedure to be industry standard and of sufficient precision to be use in the resource estimation.

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12 DATA VERIFICATION

12.1 PolyMet Data Compilation and Verification 2004

Data verification by PolyMet has involved the checking of digital data against that in the paper records and also establishing the quality and source of that data.

In 2004, all tables in the drillhole database (header, survey, lithology, and assay) were reconstructed from digital and paper records and checked by PolyMet staff against the completely re-organized original paper data. Known discrepancies were addressed and corrected. In the assay data file, erroneous or suspect data was not removed, but was flagged to prevent its inclusion in the "accepted values" file used for evaluation.

The 2004 recompilation included a generalized first-pass review list for finding any database errors or suspect assays as well as facilitating further sorting and analysis. This occurred during and after assembly of the current PolyMet drill database and prior to the finalization of an "accepted values" assay data file for project evaluation. Suspect values were either corrected or flagged for exclusion from the final "accepted values" file.

This review by PolyMet included the following quality assurance steps:

- The completeness of paper records was confirmed for each hole and assay certificates were checked to determine if they were the final versions.
- Drill hole numbers were checked for correct formats.
- Drill hole lengths were checked against data in PolyMet database header file. Any assay or lithology depths recorded as below the length of the hole were assessed.
- Depth to overburden were checked against lithological logging, many RC samples, in particular, were shown as having been collected in the overburden, these were then isolated and rejected.
- The master assay file as a whole was sorted by each element in every laboratory group. The data filter in Excel was used to inspect and check the lowest and highest value samples. The highest values were checked against the paper records. The lowest values were checked against detection limits for that period. Any discrepancies found were checked and corrected.
- All assays below detection limit were designated with "less than symbols (<)". All "<" were corrected to the detection limits listed by the laboratories for that time as shown in their "schedules of services". It was found that ACME did not show the "<" values in their older digital data reports, these had to be checked against paper records and entered manually.
- Where LECO Corporation furnace sulphur analyses had been run, these were compared with the ICP scan sulphur, if one or other seemed out of range, the

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possible reason was investigated and corrected if possible. If not reconcilable, the data was flagged as not to be used.

- Copper and nickel parts per million values were converted to percent for the final step before export of data for resource estimation.
- If the original copper value was above the upper detection limit of the method, the determination had always been re-run by a different method; this value was merged into the database as copper percent data.
- Duplicates were noted as field duplicates (two 1/4 core samples), or sample preparation duplicates (laboratory duplicates) where a crushed and/or ground sample was split at the laboratory. These duplicates were considered to have been assayed at about the same time. Copper and nickel values were compared; where these values did not reasonably match both samples were removed from the final data set.
- Where there are multiple "good" assays for copper, nickel, etc, i.e., US Steel and ACME, or ACME and Chemex, (the same intervals, but generally done at different times) the values were compared; for those that did not match, a preferred value was resolved through examination of the data or both samples were removed from consideration for the final data set.;
- Obvious laboratory typographical errors or inconsistent data were checked and either corrected or flagged to not be used. These included simple laboratory errors such as double decimal points or mistyped sample numbers;
- Copper, nickel, sulphur, platinum, palladium and gold were plotted as a function of time to highlight clusters of data well above or below the average for the group, none were found;
- Duplicate results were plotted for US Steel work in the 1970s, to determine any discrepancies;
- All "check assays" were checked as duplicate pairs; if the samples were not in reasonable agreement, then the samples were flagged for possible exclusion.

12.1.1 First Step

The first step was to sort the data into subsets by laboratory and time.

12.1.2 Second Step

The second step was to compare all the "intentional duplicate pairs", i.e., all pulp duplicates and quarter core duplicates done by the same laboratories at (more or less) the same time. PolyMet calculated a copper:copper ratio for these pairs, sorted from lowest to highest, graphed these, and generally discarded pairs where the copper:copper ratio values were beyond the inflection point of the sigmoidal graph. This somewhat depended on the geologist's view of the quality and size of the sample group, but usually this was any difference greater than about 10% to 15% of the pair.

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Experience in the data set, as well as some other ratio tests, were also used to see if numbers were reasonable. Only a single sample from each pair that PolyMet believed matched duplicate and original was used.

12.1.3 Third Step

The third step was to compare pairs or multiple samples on the same interval by different laboratories at different times (US Steel and ACME, ACME vs. Chemex vs. Chemex rerun etc.) The same approach was used, graphing copper:copper ratios and eliminated those pairs outside some range determined by inspection of the graph, which again was group by group dependent. This was more subjective. The goal here was to find mis-numberings or mis-orderings, not to quantify the quality of the data. Other ratio tests were also applied to identify if values were within expected ranges (copper:sulphur, copper:nickel).

As a result of this review, about 1,800 intervals were flagged as suspect and filtered out of the "accepted values" data used for resource evaluation.

An unexpected, but welcome, result of the 2004 data re-compilation was the discovery that about 5,000 samples taken by Severson et al. (2000) and Patelke (2001) on stored US Steel core had not been previously entered into any database. This addition greatly improved the data density within Unit 1, as well as improving the waste characterization data set for the upper units.

12.2 Hellman and Schofield Assessment

Dr. Hellman of Hellman and Shofield Pty Ltd. (H&S) undertook several assessments of the database and advised PolyMet of a number of minor issues which were addressed. Dr. Hellman conducted spot checks of the digital data by comparing it with assay certificates. In addition, Mr. S. Gatehouse, a former North Mining employee, now an employee of Hellman and Schofield Pty Ltd, did a detailed review of sampling and QA/QC aspects whilst in the previous employ of North. Although a number of concerns were identified, these did not relate to the possibility of overstatement of grade but, rather, highlighted the conservative nature of the assays.

A re-study by Hellman and Schofield of PolyMet's work of 205 coarse blanks with drill samples in 2000 shows only three samples exceeding 70 ppm nickel. These three samples appear to have resulted from transcription errors. However, PolyMet has identified some samples that were incorrectly labelled and has deleted these from the database. There is negligible cross contamination for copper, gold and platinum as evidenced by the rest of the data set. Approximately 2% of coarse blanks have palladium in excess of 20 ppb, which may suggest either some cross-contamination during sample preparation or variable background content in the blank. In another sampling program in 2000-2001, there were negligible values above lower detection limits for gold, palladium and platinum for 82 submitted blanks. The use of pulp blanks, as well as the coarse blanks, may help to resolve any future issues regarding higher than expected values.

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12.2.1 Reverse Circulation Drilling Compared to Diamond Drilling

Hellman (2005, 2006) has analyzed duplicate assay sets from RC samples that are closely situated (within 20 ft of each other) to core samples.

Gatehouse (2000) summarizes the sampling and assaying of the RC samples: 6" hole RC drilling conducted by PolyMet in 1998 had assay samples over 5' taken at the rig using a 1/16 split creating (10 to 15 lb) samples. These initially was were sent to Lerch Bros in Hibbing where preparation consisted of jaw and gyratory crushing of entire sample followed by riffle splitting (0.5 lb) for final pulping. Assaying was done by ACME using the same techniques as above. One in ten samples had pulps sent to Chemex in Vancouver for check assaying using the same Fire Assay technique and similar (notionally stronger) aqua regia ICP technique for Co, Ni, Cu and other elements.

In the 1999-2000 drilling and prior to February 2000, PolyMet sampling of 5' intervals of ½ BTW core was prepared at Lerch Bros Hibbing as above and assayed using Acme. One in ten samples were sent to Chemex as the check laboratory. Subsequently, for no apparent technical reason, Chemex were made the primary laboratory and Acme was used as a check. Analytical techniques remained the same.

This analysis is summarized in Table 12-1 for Diamond Drilling-Reverse Circulation (DD-RC) sample pairs that are at a similar elevation. For comparison, Table 12-2 shows pairs of closely situated core samples.

Table 12-1: Summary of Closely Situated RC and DD Samples

Parameter	DD Samples	RC Samples
Cu%	0.25	0.25
Ni%	0.07	0.08
Co (ppm)	62	70
Au (ppb)	32	36
Pd (ppb)	231	223
Pt (ppb)	54	59
Separation distance/number of pairs	15.6 ft/200	

Table 12-2: Summary of Closely Situated DD and RC Samples

Parameter	DD Samples	RC Samples
Cu%	0.22	0.23
Ni%	0.07	0.07
Co (ppm)	60	71
Au (ppb)	97	98
Pd (ppb)	306	238
Pt (ppb)	62	56

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Parameter	DD Samples	RC Samples
Separation distance/number of pairs	31.3 ft/98	

These results show excellent agreement even for gold, palladium and platinum. The differences between the RC and DD samples are of a similar level to those between adjacent pairs of diamond core samples. These results strongly support the integrity of both the RC samples and their assays, especially considering the many generations of sampling at NorthMet.

AGP reviewed the information available and agrees with Hellman and Schofield's conclusion.

12.2.2 Wardrop Assessment (September 2007)

Wardrop carried out an internal validation of the 330 drill holes in the NorthMet database used in the September 2007 resource estimate. Data validation has been done throughout the years by various consultants to PolyMet prior to the 2007 drill campaign and therefore the hole selection for the validation was heavily weighted on the 2007 drilling with spot checks of the US Steel, 1999, 2000 and 2005 drill campaigns. A total of 40 holes were checked amounting to 3,121 individual samples or 9% of the total sample counts in the database.

The error rate was found to be exceptionally low with only one sample (or 0.03%) entered erroneously in the GEMS database. In addition, three samples were found to have a laboratory certificate value available but were entered in GEMS as not sampled because they failed to meet PolyMet's quality standard.

During the validation, the QP found that values from laboratory certificates prior to the 2005 drill campaign were rounded half-up at the 3rd decimal while certificate values from the 2005-2007 drill campaign were truncated to the 3rd decimal during the parts per million (ppm) to percent conversion, thereby slightly understating the actual laboratory value.

The core handling facility at NorthMet is located in the former LTVSMC light duty mechanical shop and warehouses. The facility is large, well lit and equipped with overhead cranes and front-end loaders assisting staff moving palletized core bundles and crates containing sample bags ready for shipment to the ALS Chemex laboratory in Thunder Bay, Canada. The core logging room is very large and well lit and contains three large tables allowing Geologists to lay out in excess of 1,000 ft of core at any one time. Three diamond core cutting saws plus a spare are located in the core cutting room.

Table 12-3 shows a summary of the holes validated by Wardrop.

Table 12-3: Holes Validated by Wardrop

Hole-ID	Source	Elements Checked	Total No. of Samples	Errors	Missing in Gems
26025	Lab cert paper copy	Cu, Ni	176	1	
26093	Lab cert paper copy	Cu	163	0	
99-309B	Lab cert paper copy	Cu	142	0	

Hole-ID	Source	Elements Checked	Total No. of Samples	Errors	Missing in Gems
00-337C	Lab cert paper copy	Cu, Ni, Pd	121	0	1
00-352C	Lab cert paper copy	Cu, Ni	156	0	2
00-352C	Lab cert PDF	Cu, Ni	156	0	
05-406C	Lab cert PDF	Cu	107	0	
05-451C	Lab cert PDF	Cu, Ni, Pd, Pt, Au, Co	150	0	
05-501C	Lab cert PDF	Cu, Ni, Pd, Pt, Au, Co	151	0	
05-502C	Lab cert PDF	Cu, Ni, Pd, Pt, Au, Co	182	0	
07-510C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	44	0	
07-511C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	32	0	
07-512C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	28	0	
07-513C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	42	0	
07-514C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	46	0	
07-515C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	45	0	
07-516C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	70	0	
07-517C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	58	0	
07-518C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	71	0	
07-519C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	60	0	
07-520C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	73	0	
07-521C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	55	0	
07-522C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	49	0	
07-523C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	43	0	
07-524C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	62	0	
07-525C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	41	0	
07-526C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	55	0	
07-527C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	59	0	
07-528C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	24	0	
07-529C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	19	0	
07-530C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	24	0	
07-531C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	27	0	
07-532C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	96	0	
07-533C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	116	0	
07-534C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	35	0	
07-535C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	64	0	
07-536C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	26	0	
07-538C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	44	0	
07-539C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	98	0	

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Hole-ID	Source	Elements Checked	Total No. of Samples	Errors	Missing in Gems
07-540C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	111	0	
		Total checked	3121	1	3
		Total Samples in Database	34641		
		Percent checked	9.0%		
		Percent errors		0.03%	
		Percent missing			0.10%

During the site inspection, 12-drill hole collars were located using a hand held Garmin GPSMap 60CSx global positioning instrument. The average difference between the GPS collar against the database value was 22 ft, which is very good considering that the instrument reported an accuracy of ±17 to 18 ft at most field locations surveyed which is typically influenced by vegetation cover and number of satellites seen by the instrument on the day the survey was taken.

On location, the QP also inspected the core facility, core cutting room and shipping crates, geological logging and collected a limited number of check samples. Figure 12-1 shows a few images taken during the site inspection.

12.2.3 AGP Assessment (October 2007)

AGP data validation for the October 2007 database consisted of comparing an archived copy of the database used in the September 2007 resource estimate for discrepancies. Comparison focused on drill hole collar location and length, down-hole survey data from-to pairs, azimuth and dip differences, assay data from-to pairs, Cu%, Ni%, Pd ppb, Pt ppb, Au ppb and Co ppm differences.

Results indicated that for holes used in the resource model that were common to both databases the collar and survey information was identical. In the assay table, one recorded missing assay results in the September database was now complete and one copper assay had a difference of 0.15%.

An additional 16 holes belonging to the summer 2007 drill program were checked against the electronic copy of the lab certificate. Only one error was found (hole 547C) accounting to less than a 0.1% error rate as shown in Table 12-4.

Table 12-4: Additional Holes Validated by AGP

HOLE-ID	Source	Elements Checked	Total No. of Samples	Errors
07-541C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	71	0
07-542C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	57	0
07-543C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	135	0
07-544C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	72	0
07-545C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	94	0

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HOLE-ID	Source	Elements Checked	Total No. of Samples	Errors
07-546C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	19	0
07-547C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	80	1
07-548C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	67	0
07-549C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	27	0
07-550C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	37	0
07-551C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	67	0
07-552C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	140	0
07-553C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	102	0
07-554C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	63	0
07-555C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	42	0
07-556C	Electronic XLS Lab cert	Cu, Ni, Pd, Pt, Au	38	0
		Total Checked	111	1

12.2.4 Site Visits by AGP

The March 21-23, 2007 and August 27-29, 2007 site visit entailed a review of the following:

- Overview of the geology and exploration history of the geology of the Duluth complex presented by Mr. Patelke
- Current exploration program design (drill hole orientation, depth, number of holes, etc.)
- Surveying (topography and drill collar)
- Field visit to the to review drill procedures
- Visit of the core logging facility
- Discussion of the sample transportation and sample chain of custody and security
- Core recovery
- QA/QC program (insertion of standards, blanks, duplicates, etc.)
- Review of the diamond drill core, core-logging sheets and core logging procedures. This review included commentary on typical lithologies, alteration and mineralization styles, and contact relationships at the various lithological boundaries.
- During the 2007 visit, AGP collected quarter core character samples. AGP retained full custody of the sample from the NorthMet project site to Barrie Ontario where the samples were shipped to Activation Laboratories Ltd., at 1428 Sandhill Drive, Ancaster, Ontario, via Canada Post. This sample analysis allowed an independent laboratory, not previously used by PolyMet, to confirm the presence of the metal of interest. The samples were analysed for platinum group elements by Fire assay with

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a ICP/MS finish. Copper, Nickel and Cobalt were analyse with a 4 acid digestion ICP method (Code 8 – 4 acid ICP-OES).

- Table 12-5 shows the grade comparison between the AGP quarter core character sample and the PolyMet laboratory result for the same sample. From the assay results shown in Table 12-5, AGP confirmed that the general range of values reported by PolyMet correspond well with those reported by character samples collected by AGP.

Table 12-5: Character Sample Results

	AGP	PolyMet	AGP	PolyMet	AGP	PolyMet	AGP	AGP	PolyMet
Elements	11213	261033	11214	114084	11215	114118	11216	11216-split	00-347C-455-460
Cu%	0.438	0.542	0.811	0.926	0.335	0.355	0.280	0.272	0.209
Ni%	0.100	0.123	0.226	0.218	0.097	0.090	0.130	0.124	0.089
Co%	0.010	0.010	0.012	0.010	0.009	0.010	0.010	0.010	0.010
Au (ppb)	64.0	71.0	68.0	80.0	21.0	36.0	46.0	31.0	20.0
Pt (ppb)	151.0	149.0	149.0	172.0	47.4	53.0	90.9	109.0	75.0
Pd (ppb)	381.0	394.0	738.0	753.0	156.0	162.0	430.0	496.0	306.0

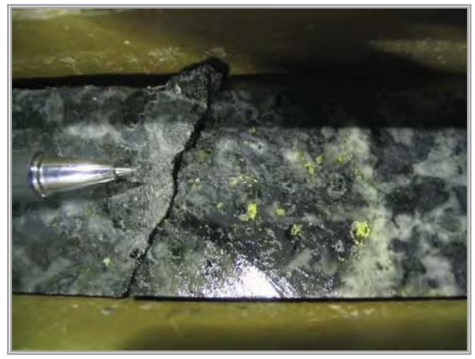
Following the site visit by AGP, the QP regards the sampling, sample preparation, security and assay procedures as adequate to form the basis of resource estimation.

Figure 12-1: Sample Preparation, Security and Assay Procedures

Crate almost ready for shipment to ALS Chemex


Core cutting in progress

Core storage facility



Typical Copper mineralization

US Steel core re-sampled by PolyMet

Collar coordinate hole 98-108B

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13 MINERAL PROCESSING AND METALLURGICAL TESTING

The Pre-feasibility Study of the NorthMet Project, which was completed in 2001 and filed on SEDAR contained a description of metallurgical test work and hydrometallurgical process design work undertaken as an integral part of that Pre-feasibility Study. Further mineral processing developments were described in a report entitled "Technical Update of the NorthMet Project Incorporating the established Cliffs-Erie crushing / milling / concentration facilities with the Hydrometallurgical processes described in the May 2001 Pre-feasibility study" by P. Downey and Associates, dated July 2004 and filed on SEDAR.

Since that time additional mine engineering work has been undertaken along with metallurgical test work by SGS Lakefield Laboratories and extensive process design and engineering work by Bateman Engineering Pty Ltd. as part of the DFS. The results of this DFS were filed on SEDAR September 20, 2006 (Hunter, 2006).

There have been no substantive changes to the processing flowsheet since 2006, however PolyMet has made two relatively minor changes in order to improve the economics, take advantage of its marketing relationship with Glencore, and reduce the environmental impact of the Project. In May 2008, PolyMet modified the process to include an initial stage when it would sell concentrate during completion of construction and commissioning of the hydrometallurgical plant contemplated in the DFS. This approach had the advantage of staging capital costs so that the hydrometallurgical plant could be funded in part from cash flow from sales of concentrate, and reduced reliance on delivery of long lead-time equipment before the start commercial production.

In February 2011, PolyMet made further modifications to its plans, replacing the full hydromet facility with a smaller plant resulting in production and sale of high-grade copper concentrate, value added nickel-cobalt hydroxide, and precious metals precipitate products.

Both of these changes have a positive impact in project economics and, as such, neither is material in terms of the viability of the NorthMet Project.

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14 MINERAL RESOURCE ESTIMATES

14.1 Data

Mineral resource estimates have been completed by AGP for PolyMet's NorthMet polymetallic Deposit. The NorthMet Deposit is located in the St Louis County in north-eastern Minnesota, USA at Latitude 47°36' north, Longitude 91°58' west, approximately 70 miles north of the City of Duluth and 6.5 miles south of the town of Babbitt. PolyMet Mining Corp. (as Fleck Resources), acquired a 20-year renewable mineral lease to the NorthMet Deposit in 1989 from US Steel, which disposed of much of its non-core assets to RGGS Ltd. in 2003 consequently transferring the underlying mineral rights to RGGS Ltd.

Gemcom software GEMS 6.04™ was used for the resource estimate in combination with Sage 2001 for the variography. The metals of interest at NorthMet are copper, nickel, cobalt, platinum, palladium and gold. Minor amounts of rhodium and ruthenium are also present although these elements are not significant. Sulphur was also estimated for process and environmental purposes.

PolyMet provided the digital data files in a GEMS database dated October 13, 2007. The GEMS database consisted of the digital drill hole database containing a complete data set from 673 holes, a triangulation workspace with the upper surfaces of the different units on the NorthMet Deposit, two geological domains for the Virginia Formation inclusions, two grade shell domains and a topographic and ledge surface. Appendix A lists the data that was available for the December 2007 resource evaluation.

As shown in Table 14-1, out of a total of 673 holes, 371 were used for the resource evaluation grade models. None of the holes in the database had pending assays. A total of 47 stratigraphic control drill holes without assays were left out of the resource model along with the 241 vertical electrical soundings (entered into the database as "pseudo" drill holes for ease of use) holes and 15 other holes drilled to assess the bedrock depth.

The PolyMet NorthMet project geology is divided into seven main lithological units and two grade shell domains. A typical cross section Figure 14-1 shows the stratigraphic position of the units in relation to the grade shells DOM1 and Magenta Zone.

The bulk of the mineralization is located within the two grade shells with minor amounts in the remainder of Units 1 through 7. The Virginia Formation typically carries very low copper, nickel, palladium, platinum, gold and cobalt values but has elevated sulphur values and has been modelled for waste characterization purposes. No grades were interpolated in the Iron Formation (Unit 30).

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Table 14-1: Total Number of Holes Used for the December 2007 Resource Estimate

	No. of Holes	Total Length (ft)	Total Number of Assays
Holes with assay results 2007	61	24,530	3,612
Holes with assay results pre-2007	309	261,227	31,790
Holes outside the pit area/hydro holes	47	29,827	0
Vertical electrical bedrock sounding holes	241	3,900	0
Depth to bedrock holes	15	155	0
Total	673	319,639	35,402

14.2 Geological Models

The NorthMet Deposit digital data set consists of seven surfaces provided by PolyMet describing the geological boundaries observed during core logging. The stratigraphy (bottom to top) covers the Iron Formation, the Virginia Formation, Unit 1, Unit 2 and 3 combined into Unit 3, Unit 4 and 5 combined into Unit 5, Unit 6, Unit 7 and the overburden (glacial drift). Topography is a two ft contour derived from air photo work in 1999.

This geological model is overlain by two grade shell models, the DOM1 Zone and the Magenta Zone where the boundaries were drawn based on a US$6.00 per short ton NMV calculated with the formula in Section 17.2.11 of this report. The US$6.00 NMV is currently below the cut-off and is designed to include all areas of mineralization that have the potential to be economically viable. The grade shell model also limits the potential smearing of high grade value into adjoining low grade areas or vice versa.

The DOM1 domain is located near the top of Unit 1 and breaks through the contact to include some of the higher grade material near the bottom of Unit 2 (Unit 2 is merged with Unit 3 in this study). The DOM1 domain spans 14,300 ft east-west and 4,700 ft in the north-south direction between 2895955 E and 2910402 E and 730073 N to 741199 N and is largely unchanged since the September 2007 resource estimate.

The Magenta Zone domain is smaller in size and is mostly contained within Units 5 and 6 but occasionally is seen in Units 3 and 7. The domain is located in the western part of the NorthMet Deposit between 2897383 E and 2902320 E and 732708 N and 737038 N. The Magenta zone was re-interpolated based on the summer drilling program. The domain was extended predominantly in a westerly direction and is now 147,097,310 ft3 larger.

Based on the contact profile, the geological model was re-coded into six distinct grade domains for the purpose of grade interpolation as illustrated in Figure 14-1 which also illustrates the location of the various units and grade shell domains.

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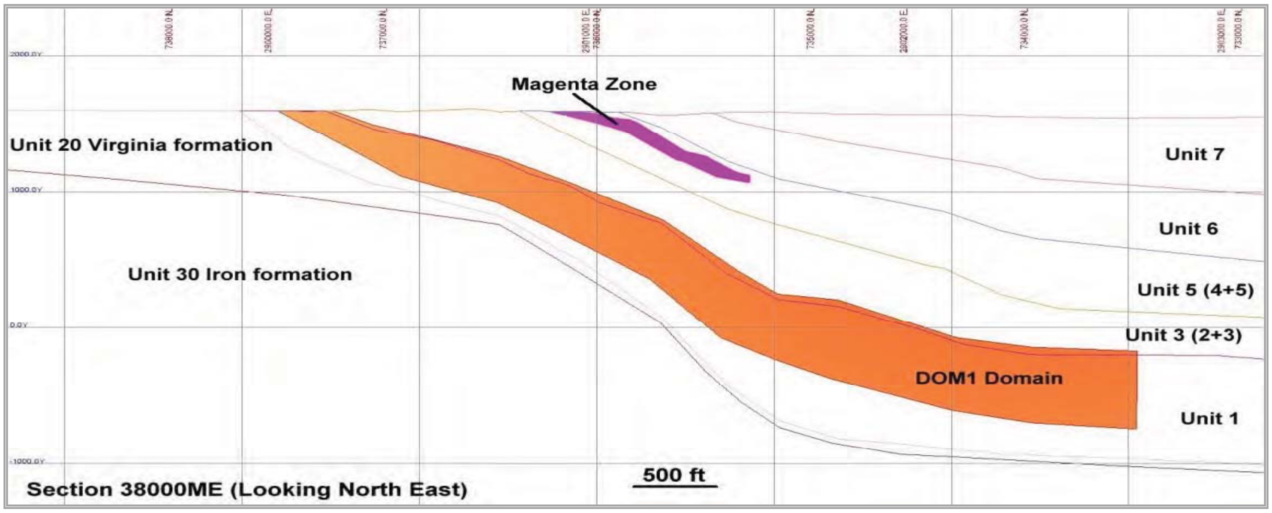
Figure 14-1: Domains and Unit Code



14.3 Exploratory Data Analysis

Exploratory data analysis is the application of various statistical tools to characterize the statistical behaviour or grade distributions of the data set. In this case, the objective is to understand the population distribution of the grade elements in the various units using such tools as histograms, descriptive statistics, probability plots and contact plots.

Statistical analysis of the data was performed on each of the unit codes and also on the grade shell domains.

14.3.1 Assays

Table 14-2 shows the assay mean values for the different unit codes. Units 1, 5, and 6 show elevated metal values, with minor amounts distributed in Unit 7. The complete set of descriptive statistics for the NorthMet Deposit is included in Appendix B.

Table 14-2: NorthMet Raw Assay File by Unit – Mean Grade

Units	30	20	1	2+3 (3)	4+5 (5)	6	7
Number of Samples	76	1370	19819	8164	3351	1596	462
Cu (%)	0.001	0.017	0.211	0.067	0.118	0.142	0.033
Ni (%)	0.001	0.012	0.066	0.034	0.040	0.051	0.038
Co (ppm)	0.22	23.18	66.86	52.83	53.51	63.62	64.55
Pt (ppb)	1	2	45	24	43	59	20
Pd (ppb)	1	7	172	76	113	147	39
Au (ppb)	1	3	24	13	21	25	8
S (%)	0.24	1.74	0.63	0.18	0.26	0.23	0.07

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14.3.2 Contact Profiles

As part of the September 2007 resource model, AGP examined in detail the contact relationship between the individual units and between the units adjacent to the grade shell models. Only copper was used for this study assuming that nickel, cobalt and platinum, palladium and gold would behave similarly since the correlation coefficients (Hellman) are known to be high. No other elements were evaluated and the study was not updated with the October 2007 dataset.

The software calculates the average grade of an element over distance from a boundary between two lithologies, two units/domains or two indicator values. Contact relationships can be used to determine the inclusion or exclusion of sample data points used in the interpolation of one particular grade domain and also to assist in confirming geological interpretations. A gradational contact (or soft boundary) generally allows the interpolation parameters to include a limited number of samples from the adjoining domain while a sharp contact (or hard boundary) will restrict the sample points used in the interpolation to its own domain.

Results from the analysis are as follows with accompanying plots in Figure 14-2 thru to Figure 14-6.

- The expected hard boundary between the Virginia Formation (Unit 20) and Unit 1 is clearly visible in the contact plots with no grade enrichment at the contact and a slight depletion in Cu% grade up to 20 ft from the boundary inside Unit 1.
- Units 1 and 3 (2 + 3) also show a hard boundary with a large variance in grade and no apparent enrichment or depletion at or near the boundary.
- Units 3 (2 + 3) and 5 (4 +5) show a gradational contact with copper enrichment near the boundary.
- Units 5 (4 + 5) and 6 show a gradational contact near the boundary and a slight depletion internal to Unit 6, followed by an enrichment. Note that the data point count for Unit 5 (4 + 5) is 2609 points with 393 points inside the higher grade Magenta Zone. It is therefore normal to expect a higher grade in Unit 6 than Unit 5 (4+ 5).
- Units 6 and 7 both show gradational contacts and even grade distribution. The point count for Unit 7 is low at 358 points.

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Figure 14-2: Contact Profiles Unit 20 and Unit 1 (distance in ft)

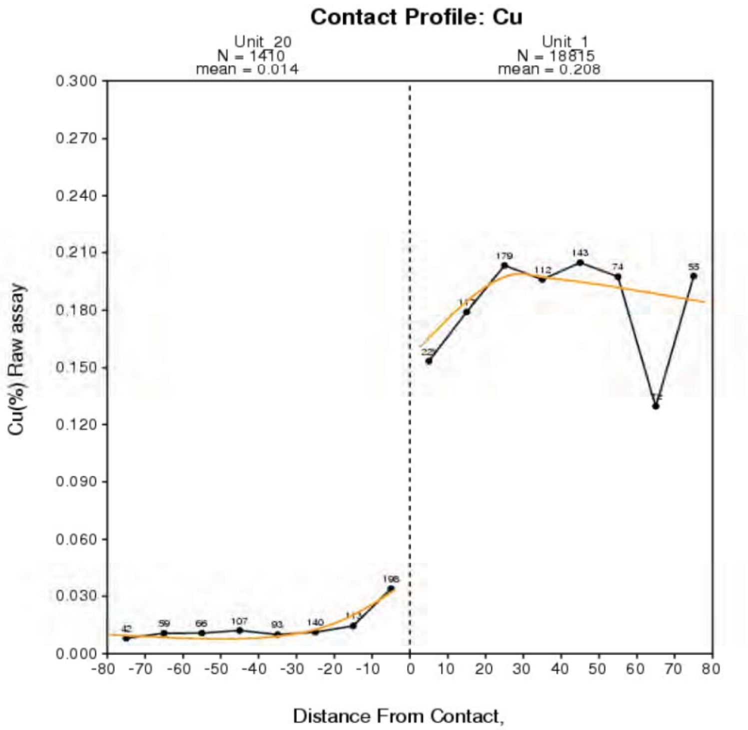

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Figure 14-3: Contact Profile Unit 1 and Unit 3 (Distance in ft)

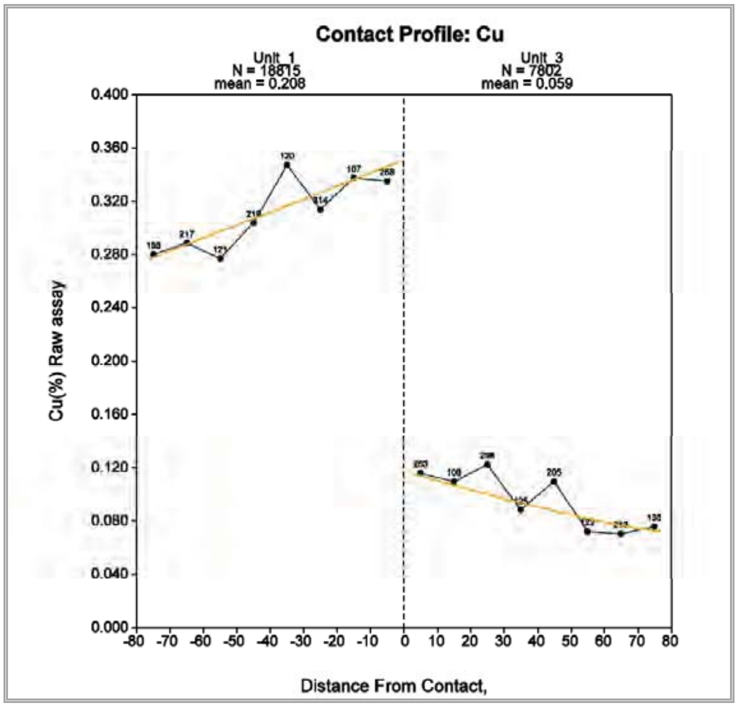

Figure 14-4: Unit Contact Profile Unit 3 and Unit 5 (Distance in ft)

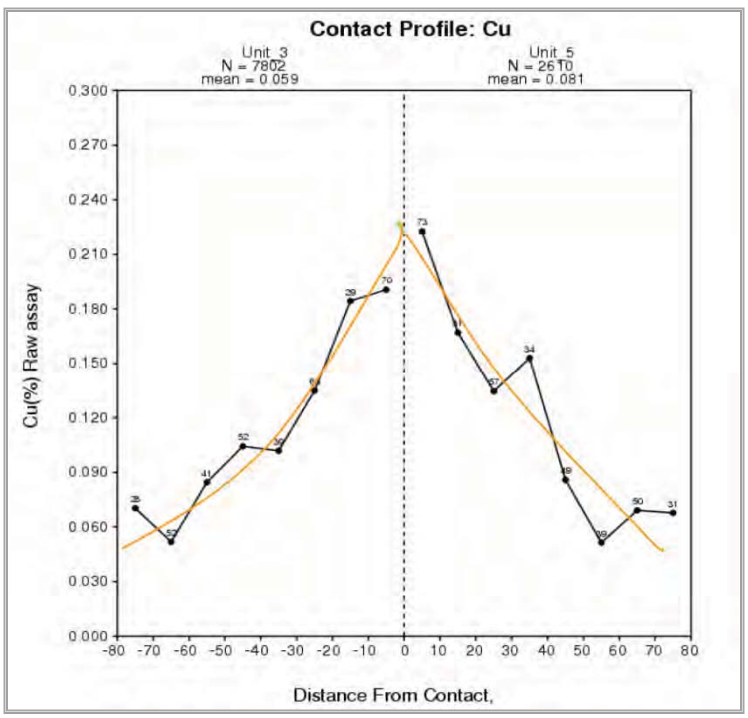

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Figure 14-5: Contact Profile Unit 5 and Unit 6 (Distance ft)

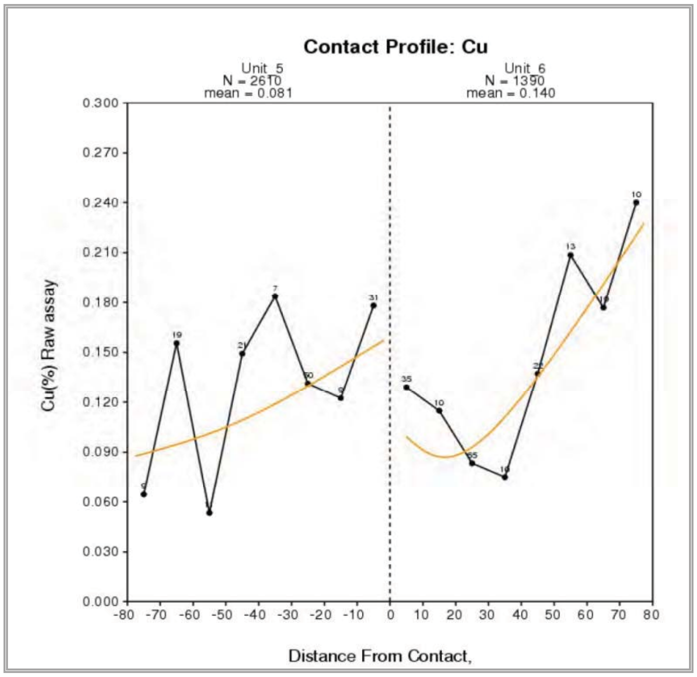
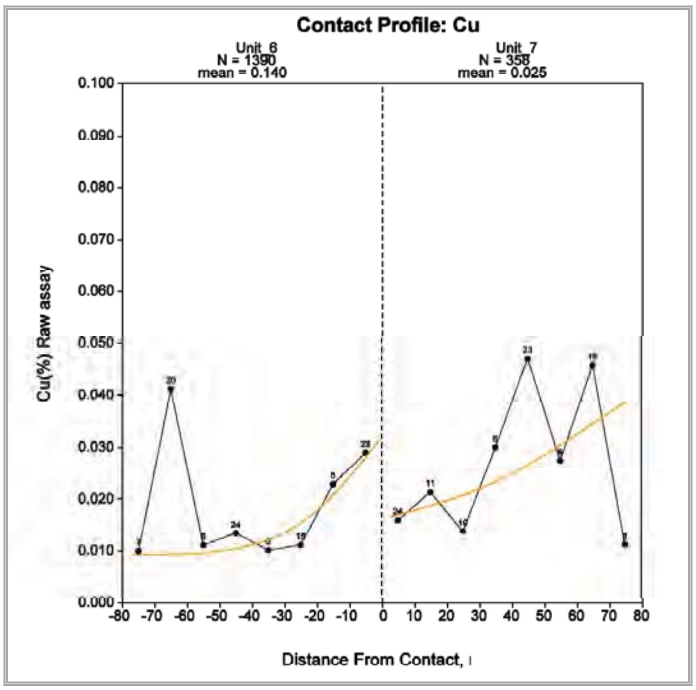


Figure 14-6: Unit Contact Profile Unit 6 and Unit 7 (Distance in ft)



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On the basis of the unit contact profile results, the assay points located in the DOM1 and Magenta Zone grade shell models were grouped by unit code and additional contact profiles were evaluated between the following boundaries as shown in Figure 14-7.

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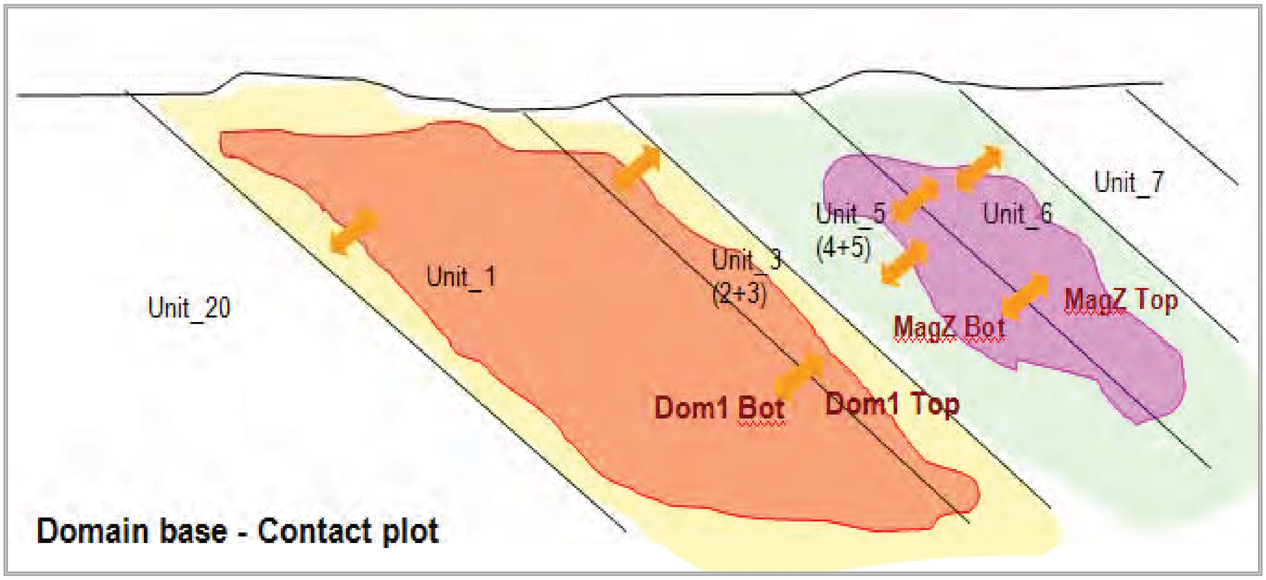
Figure 14-7: Schematic Cross-Section Illustrating Unit and Domain Nomenclature and Contact Profiles



The Magenta Zone overlays Units 3 (2 + 3), 5 (4 + 5), 6 and 7, however, since the Magenta Zone is primarily in contact with Unit 5 (4 + 5) and 6, only the points from these Units were considered for the contact study relating to the Magenta Zone:

- Unit 1 and DOM1 points located in Unit 1
- DOM1 points located in Unit 1 and DOM1 points located in Unit 3 (2 + 3)
- Unit 3 (2 + 3) and DOM1 points located in Unit 3 (2 + 3)
- Unit 5 (4 + 5) and Magenta Zone points located in Unit 5 (4 + 5)
- Magenta Zone points located in Unit 5 (4 + 5) and Magenta Zone points located in Unit 6
- Unit 6 and Magenta Zone points located in Unit 6.

Results for DOM1 grade shell indicate the following with accompanying plots in Figure 14-8:

- Gradational contact across Unit 1 and the DOM1 bottom boundary
- Sharp contact with no enrichment between DOM1 bottom and DOM1 top mimicking the Unit 1 and Unit 3 (2 + 3) contact profiles
- Gradational contact across DOM1 top and Unit 3 (2 + 3)

Contact plots for across the Magenta Zone indicate the following with accompanying plots in Figure 14-9

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- Semi-soft contact between Unit 5 (4 + 5) and the bottom of the Magenta Zone. Grade increases gradually inside the Magenta Zone
- Relatively sharp contact exists between the Magenta top and Unit 6. Grade decreases gradually from the core of the Magenta Zone toward the contact. The copper grade in Unit 6 is consistently low.

Based on the contact profile, the geological model was re-coded into six distinct grade domains for the purpose of grade interpolation as illustrated in Figure 14-10.

Figure 14-8: Grade Shell DOM1 Contact Profiles (distance in ft)

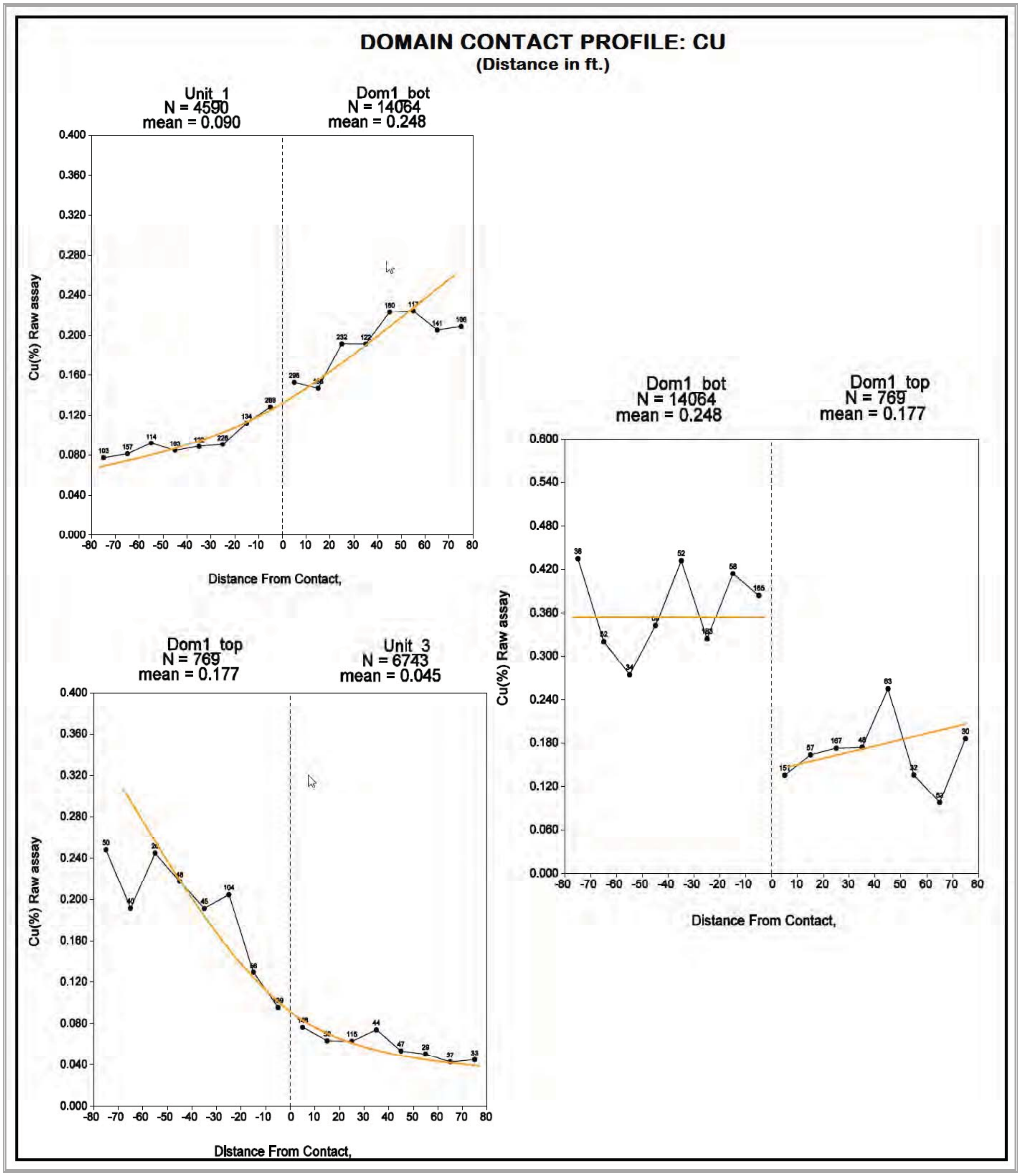

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Figure 14-9: Contact Profile for Magenta Zone Grade Shell (distance in ft)

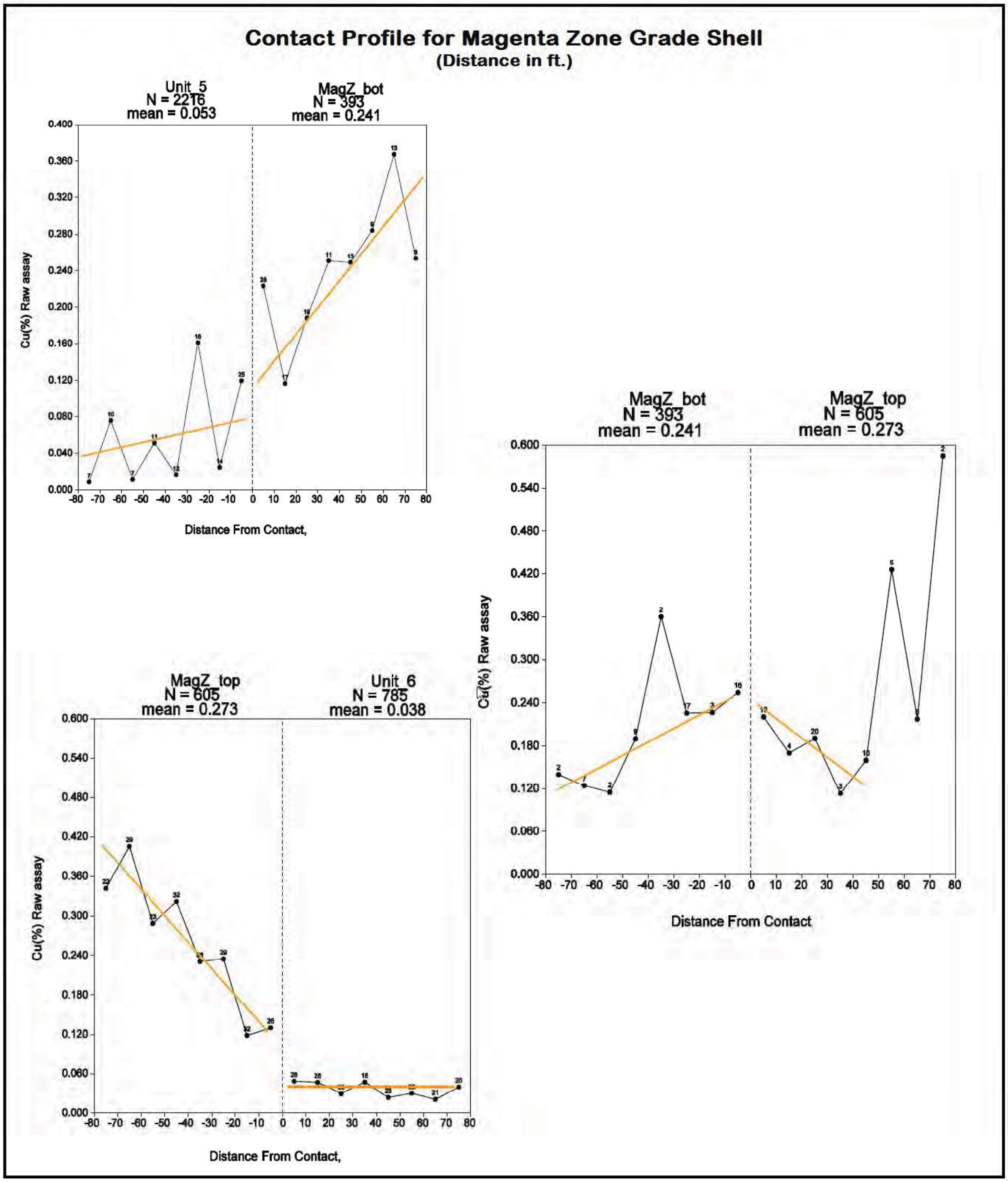

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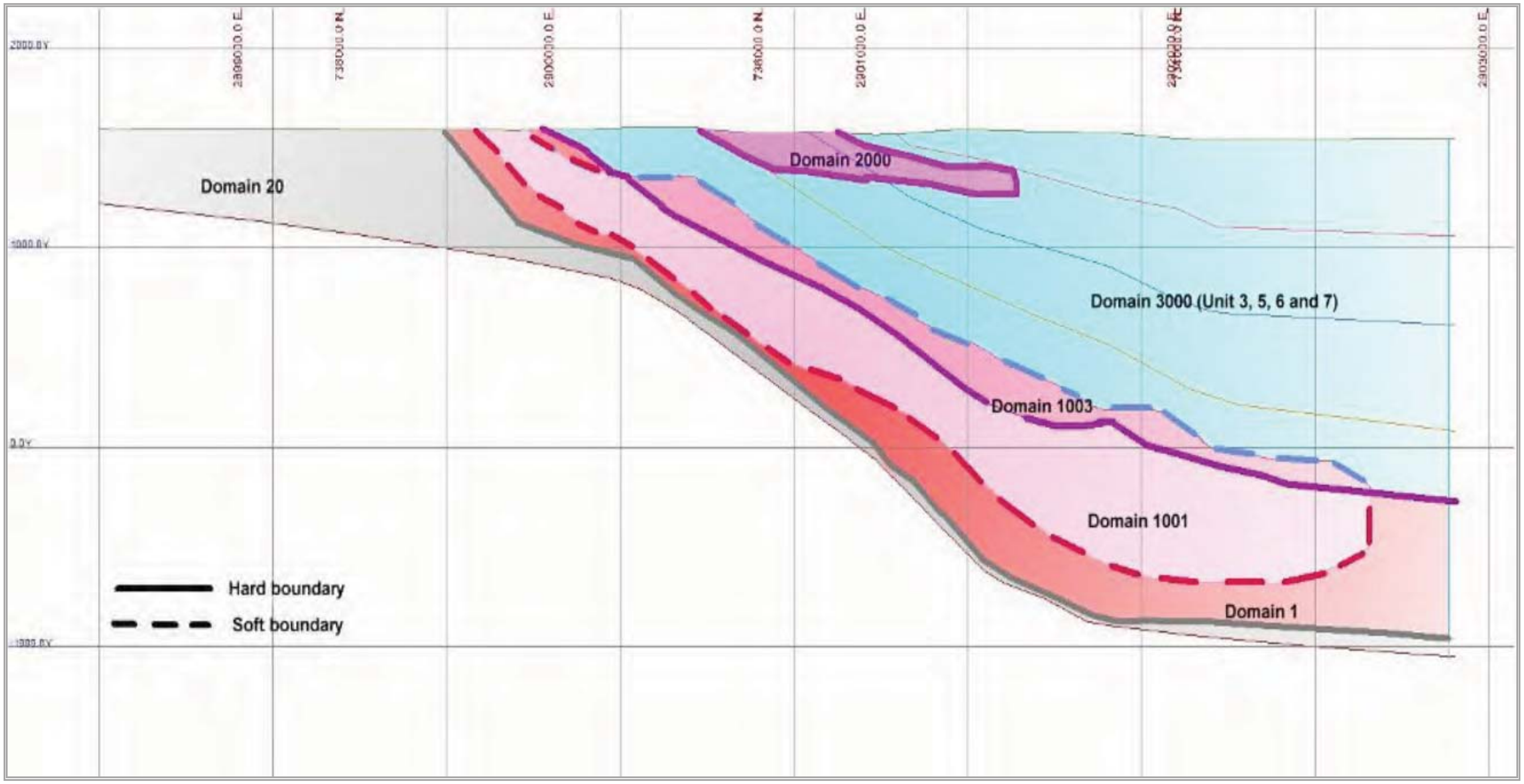
Figure 14-10: Grade Domains Schematic Section Looking North-East



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14.4 Grade Capping/Outlier Restrictions

A combination of decile analysis and review of probability plots were used to determine the potential risk of grade distortion from higher-grade assays. A decile is any of the nine values that divide the sorted data into ten equal parts so that each part represents one tenth of the sample or population. In a mining project, high-grade outliers can contribute excessively to the total metal content of the NorthMet Deposit.

Typically in a decile analysis, capping is warranted if:

- the last decile has more than 40% of metal, or
- the last decile contains more than 2.3 times the metal quantity contained in the one before last, or
- the last centile contains more than 10% of metal, or
- the last centile contains more than 1.75 times the metal quantity contained in the one before last.

The decile analysis performed by the QP for the September 2007 resource model was not updated with the October 2007 dataset as very few additional data points were added. Results shown in Appendix C indicate that no grade capping is warranted for the DOM1 and Magenta Zone grade shell domains. Unit 1, Unit 20 and Units 3, 4, 5, 6 and 7 outside the Magenta Zone show significant high-grade outliers and a high-grade search restriction was considered by the QP as appropriate for the NorthMet Deposit. Table 14-3 compares the analyses and tabulates the implemented level.

Table 14-3: Threshold Value Used for High Grade Search Restriction (May 25, 2007 dataset)

	Cu (%)	Ni (%)	Co (ppm)	Pt (ppb)	Pd (ppb)	Au (ppb)	S (%)
Unit 20	0.7	0.18	n/a	200	1000	80	7.5
Unit 1 outside DOM1 Grade shell	1.8	0.6	n/a	450	1600	500	7.5
DOM1 (in Unit 1)	n/a	n/a	n/a	n/a	n/a	n/a	n/a
DOM1 (in Unit 3)	n/a	n/a	n/a	n/a	n/a	n/a	n/a
Units 2/3, 4/5, 6, and 7 excluding Magenta Zone	2.1	0.4	n/a	700	4000	500	8
Magenta Zone	n/a	n/a	n/a	n/a	n/a	n/a	n/a

The search restriction size was based on a next block, diamond shape pattern with a 75 ft radius from the block center. Essentially, a sample selection ellipsoid is applied to a block during the interpolation process. Points that are above the threshold value and outside the smaller restricted search ellipsoid are eliminated from the set during the interpolation. Grade for the block is calculated and the process is repeated for the next block. The end result is that all high grade samples are used at face value but their range of influence is limited to an area that is more or less 75 ft in diameter

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14.5 Composites

Core length statistics on the October 13 dataset indicate the sampling intervals in the two grade shell domains for the NorthMet Deposit average 5.3 ft in the DOM1 domain and 5.8 ft in the Magenta Zone. The upper third quartile shows 10 ft or less for Units 1, 3, 5, 6, 7, and 20. Based on that information a 10 ft composite length was selected. This length allowed for a few samples of greater length to be broken without affecting the variance and shorter samples to be combined to produce a sample of proper support. Summary statistics are shown in Table 14-4.

Assays were composited in 10 ft intervals starting at the toe of the hole and honouring the geological hard boundaries. Composite remnants, which are composites less than 10 ft in length, are unavoidable if the hard geological boundaries are to be honoured. The compositing methodology used by AGP locates the composite remnant (<10 ft) in Unit 20 and on the wider side of the Unit 1-Unit 3 boundary while minimizing the composite remnants in the remaining units.

Table 14-4: Core Length Summary Statistics (October 15 Dataset)

Unit Code	30	20	1	3	5	6	7	DOM1	Magenta Zone
Number of values	2	982	4698	6857	2189	845	427	15495	1894
Minimum (ft)	0.3	1.5	1.0	0.3	1.0	2.0	2.0	0.3	1.0
Maximum (ft)	17.0	12.5	14.0	12.0	26.0	12.5	12.0	17.0	15.0
Mean (ft)	10.0	5.2	5.2	6.7	8.3	8.6	8.9	5.3	5.8
Median (ft)	10.0	5.0	5.0	5.0	10.0	10.0	10.0	5.0	5.0
First quartile (ft)	-	5.0	5.0	5.0	5.0	7.0	8.0	5.0	5.0
Third quartile (ft)	-	5.0	5.0	10.0	10.0	10.0	10.0	5.0	5.0

Un-sampled intervals, gaps and assays below detection limits were composited at zero grades for copper, nickel, platinum, palladium, gold and cobalt.

For sulphur, the un-sampled intervals were initialized to the domain average value prior to compositing. A total of 1,571 sulphur intervals out of 35,402 (or 4.4% of the assay database) needed initialization. Table 14-5 shows the background value used for this resource estimate.

Table 14-5: Sulphur Background Values for Unsampled Intervals

Domain	Sulphur Background Value
Unit 1 outside Domain 1	0.454
Domain 1	0.668
Unit 3,5,6 or 7 outside Domain 1 or Magenta zone	0.146
Magenta zone	0.420
Virginia formation	1.230

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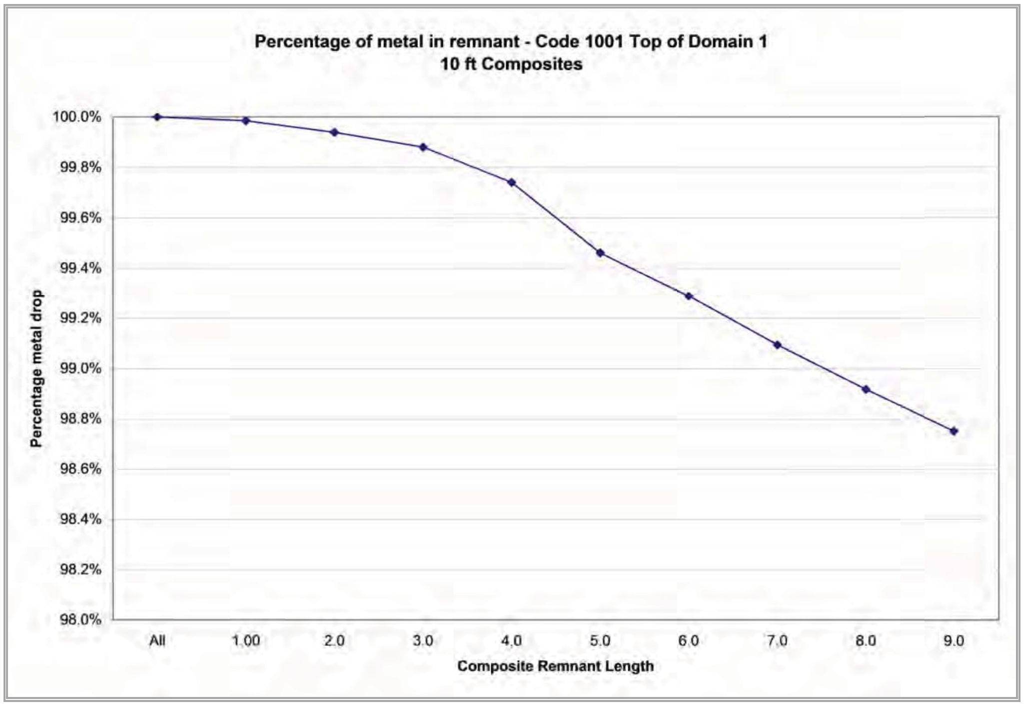
Domain	Sulphur Background Value
Iron formation	0.240

Statistical analysis of the composite remnants indicates that intervals less than 4 ft could be safely deleted from the dataset without introducing a bias in the remaining composites. This ensured that smaller, less representative samples would not be included in the interpolation. Figure 14-11 shows an example graph for the upper DOM1 Zone where deleting composites less than four ft would only affect the metal content by 0.2%. Box plots showing statistical analysis of sample interval lengths are included in Appendix D along with the complete remnant statistical study.

Figure 14-11: DOM1 Composite Remnants



Composite statistics by unit codes are shown in Table 14-6. Complete composite statistics are located in Appendix E. Composite statistics sorted by grade domain code shown in Table 14-6 and Table 14-7.

Table 14-6: Final Composite Statistics by Unit Code (October 2007 Dataset) Mean Grade Compilation

Units	1	2/3	4/5	6	7	20	30
Counts	11,481	6,813	4,054	2,184	847	2,241	374
Cu (%)	0.201	0.047	0.057	0.064	0.015	0.007	0.001
Ni (%)	0.062	0.026	0.022	0.026	0.019	0.006	0.001
Co (ppm)	60.5	40.9	31.6	34.6	31.3	10.1	0.2

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Units	1	2/3	4/5	6	7	20	30
Pt (ppb)	44	17	21	28	9	1	0.5
Pd (ppb)	167	53	54	68	18	3	0.5
Au (ppb)	23	10	11	12	4	2	0.5
S (%)	0.64	0.17	0.18	0.17	0.11	1.40	0.30

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Table 14-7: Final Composites by Domain (October 15 Dataset) – Mean Grade Compilation

Grade Domain	Unit 1 outside DOM1 Zone	Unit 20 Virginia Formation	Code 22 Ramp Area	Code 23	DOM1 Bot (in Unit 1)	DOM1 Top (In Unit 3)	Magenta Zone	Unit 3,4,5,6 and 7 outside Magenta Zone
Domain Code	1	20	22	23	1001	1003	2000	3000
Count	3,192	2,018	498	102	8,158	423	1,132	12,155
Cu%	0.081	0.007	0.028	0.015	0.250	0.176	0.241	0.029
Ni%	0.029	0.005	0.014	0.012	0.075	0.071	0.066	0.019
Cobalt ppm	40.6	9.2	31.4	23.6	68.5	71.9	64.7	32.8
Platinum ppb	14	1	6	3	56	60	95	11
Palladium ppb	50	3	16	3	215	219	252	30
Gold ppb	9	2	5	4	29	33	43	6
S%	0.47	1.41	0.67	2.29	0.70	0.37	0.40	0.14

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14.6 Variography

Geostatisticians use a variety of tools to describe the pattern of spatial continuity, or strength of the spatial similarity of a variable with separation distance and direction. The correlogram measures the correlation between data values as a function of their separation distance and direction. If we compare samples that are close together, it is common to observe that their values are quite similar and the correlation coefficient for closely spaced samples is near 1.0. As the separation between samples increases, there is likely to be less similarity in the values and the correlogram tends to decrease toward 0.0. The distance at which the correlogram reaches zero is called the "range of correlation" or simply the range. The range of the correlogram corresponds roughly to the more qualitative notion of the "range of influence" of a sample; it is the distance over which sample values show some persistence or correlation. The shape of the correlogram describes the pattern of spatial continuity. A very rapid decrease near the origin is indicative of short scale variability. A more gradual decrease moving away from the origin suggests longer scale continuity.

Using Sage 2001 software, directional sample correlograms were calculated for all elements, copper, nickel, platinum, palladium, gold, cobalt and sulphur in each of the six grade domains along horizontal azimuths of 0, 30, 60, 90, 120, 150, 180, 210, 240, 270, 300 and 330 degrees. For each azimuth, sample correlograms were also calculated at dips of 30 and 60 degrees in addition to horizontally. Lastly, a correlogram was calculated in the vertical direction. Using the thirty-seven correlograms an algorithm determined the best-fit model. This model is described by the nugget (C0) which was derived using down hole variograms; two nested structure variance contribution (C1, C2), ranges for the variance contributions and the model type (spherical or exponential). After fitting the variance parameters, the algorithm then fits an ellipsoid to the thirty-seven ranges from the directional models for each structure. The final models of anisotropy are given by the lengths and orientations of the axes of the ellipsoids. Tables 14-8 to 14-10 summarize the results of the variography.

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Table 14-8: Variography DOM1 Top and Bottom (October 15 Dataset)

Domain	Component	Increment	Cumulative	Rotation	Angle1	Angle2	Angle3	Range1	Range2	Range3
DOM1 Bottom – Au Code 1001	Nugget C0	0.036	0.036							
	Exponential C1	0.748	0.784	ZYZ	-82.94	-72	45	14.3	60.8	3.4
	Exponential C2	0.216	1	ZYZ	-101.9	-53	11	108.7	466.1	560.8
DOM1 Bottom – Co Code 1001	Nugget C0	0.044	0.044							
	Exponential C1	0.697	0.741	ZYZ	-99.94	58	4	105.9	221.1	24
	Exponential C2	0.259	1	ZYZ	-135.9	23	93	18	630.2	773.2
DOM1 Bottom – Cu Code 1001	Nugget C0	0.005	0.005							
	Exponential C1	0.605	0.61	ZYZ	-85.94	-75	-4	26.1	74.9	7.9
	Exponential C2	0.39	1	ZYZ	-202.9	72	36	76.1	611.7	473.7
DOM1 Bottom – Ni Code 1001	Nugget C0	0.006	0.006							
	Exponential C1	0.6	0.606	ZYZ	-41.94	21	42	58.3	11	33.3
	Exponential C2	0.394	1	ZYZ	-84.94	-46	-5	67.4	488.4	369.3
DOM1 Bottom – Pd Code 1001	Nugget C0	0.008	0.008							
	Exponential C1	0.671	0.679	ZYZ	-52.94	15	-16	8.2	44.6	22.3
	Exponential C2	0.321	1	ZYZ	-110.9	-51	12	103.9	699.9	441.8
DOM1 Bottom – Pt Code 1001	Nugget C0	0.014	0.014							
	Exponential C1	0.745	0.759	ZYZ	-108.9	21	21	6.5	33.4	24.1
	Exponential C2	0.241	1	ZYZ	-150.9	-71	31	108.3	494.6	895
DOM1 Bottom – S Code 1001	Nugget C0	0.015	0.015							
	Exponential C1	0.558	0.573	ZYZ	-92.94	-56	9	19.4	157.1	8.8
	Exponential C2	0.427	1	ZYZ	-100.9	52	51	162.3	357.3	56.2
DOM1 Top – Au Code 1001	Nugget C0	0.013	0.013							
	Exponential C1	0.817	0.83	ZYZ	-147.9	-33	-39	38.6	20.3	9.5
	Exponential C2	0.17	1	ZYZ	-83.94	-55	11	85.3	201.4	873.1
DOM1 Top – Co Code 1003	Nugget C0	0.006	0.006							
	Exponential C1	0.626	0.632	ZYZ	-4.94	-83	-95	10.7	165.5	19.9
	Exponential C2	0.368	1	ZYZ	-66.94	31	67	12.1	2965.2	491.9
DOM1 Top – Cu Code 1003	Nugget C0	0.028	0.028							
	Exponential C1	0.833	0.861	ZYZ	-90.94	-79	61	17.9	84.7	5.8
	Exponential C2	0.139	1	ZYZ	-58.94	-37	-31	156.8	1250.9	648.6
DOM1 Top – Ni Code 1003	Nugget C0	0.016	0.016							
	Exponential C1	0.559	0.575	ZYZ	-102.9	-9	-4	79.8	104.6	14.2
	Exponential C2	0.425	1	ZYZ	-47.94	-1	-32	40.3	477.2	253.8
DOM1 Top – Pd Code 1003	Nugget C0	0.004	0.004							
	Exponential C1	0.79	0.794	ZYZ	-68.94	-32	6	23.1	89.6	9.7
	Exponential C2	0.206	1	ZYZ	-53.94	-54	-21	81.6	277.2	1041.1
DOM1 Top – Pt Code 1003	Nugget C0	0.416	0.416							
	Exponential C1	0.391	0.807	ZYZ	-88.94	-55	14	49.9	207.8	3.7
	Exponential C2	0.193	1	ZYZ	-73.94	-46	-12	98.1	446.7	640.1
DOM1 Top – S Code 1003	Nugget C0	0.061	0.061							
	Exponential C1	0.819	0.88	ZYZ	-65.94	-69	0	37.3	100.5	9.4
	Exponential C2	0.12	1	ZYZ	-81.94	-9	-11	77.5	1,568.4	352.5

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Table 14-9: Variography Unit 1 and Unit 20 (October 15 Dataset)

Domain	Component	Increment	Cumulative	Rotation	Angle1	Angle2	Angle3	Range1	Range2	Range3
Unit 1 – Au Code 1	Nugget C0	0.784	0.784							
	Spherical C1	0.137	0.921	ZYZ	-57.94	80	-36	143.4	102.9	3
	Spherical C2	0.079	1	ZYZ	-151.9	-3	91	542.1	12688	16,954
Unit 1 – Co Code 1	Nugget C0	0.495	0.495							
	Spherical C1	0.186	0.681	ZYZ	-115.9	64	-50	213.8	80.9	26.7
	Spherical C2	0.319	1	ZYZ	-89.94	-48	97	3002.4	244.7	789.9
Unit 1 – Cu Code 1	Nugget C0	0.48	0.48							
	Spherical C1	0.265	0.745	ZYZ	-100.9	-11	-30	15.6	95.6	118.3
	Spherical C2	0.255	1	ZYZ	-62.94	4	16	52.4	104.2	960.3
Unit 1 – Ni Code 1	Nugget C0	0.647	0.647							
	Spherical C1	0.205	0.852	ZYZ	-128.9	85	48	155.9	181.5	10.1
	Spherical C2	0.148	1	ZYZ	-118.9	3	46	283.3	3019.2	1,094.7
Unit 1 – Pd Code 1	Nugget C0	0.508	0.508							
	Spherical C1	0.296	0.804	ZYZ	-121.9	90	3	306	171.8	7.9
	Spherical C2	0.196	1	ZYZ	-66.94	7	89	5569.9	902.3	599.5
Unit 1 – Pt Code 1	Nugget C0	0.672	0.672							
	Spherical C1	0.234	0.906	ZYZ	-122.9	89	-35	313.8	213.9	8.1
	Spherical C2	0.094	1	ZYZ	29.06	-74	47	1183.8	765.1	2,754.6
Unit 1 – S Code 1	Nugget C0	0.533	0.533							
	Spherical C1	0.3	0.833	ZYZ	119.06	70	-16	316.1	93.5	40.9
	Spherical C2	0.167	1	ZYZ	-101.9	39	8	218.4	2008.7	214.2
Unit 20 – Au Code 20	Nugget C0	0.368	0.368							
	Spherical C1	0.435	0.803	ZYZ	-74.94	90	26	66.6	85.5	6.2
	Spherical C2	0.197	1	ZYZ	-55.94	-12	62	143.8	79.1	546.8
Unit 20 – Co Code 20	Nugget C0	0.398	0.398							
	Spherical C1	0.279	0.677	ZYZ	-124.9	-62	81	48.3	215.9	11.4
	Spherical C2	0.323	1	ZYZ	-106.9	50	33	457	1,859.6	223.2
Unit 20 - Cu Code 20	Nugget C0	0.45	0.45							
	Spherical C1	0.381	0.831	ZYZ	-94.94	87	-49	163.5	152.2	9
	Spherical C2	0.169	1	ZYZ	-60.94	-5	-54	155.5	500	1,200
Unit 20 – Ni Code 20	Nugget C0	0.406	0.406							
	Spherical C1	0.34	0.746	ZYZ	-80.94	90	3	182.4	67.1	7.9
	Spherical C2	0.254	1	ZYZ	-83.94	11	9	78.3	117.5	1,190.4
Unit 20 – Pd Code 20	Nugget C0	0.571	0.571							
	Spherical C1	0.198	0.769	ZYZ	-68.94	61	-55	44.1	140.4	163.5
	Spherical C2	0.231	1	ZYZ	-14.94	0	-24	5.4	50.9	609
Unit 20 – Pt Code 20	Nugget C0	0.434	0.434							
	Spherical C1	0.402	0.836	ZYZ	-47.94	89	-47	81.3	52.1	4.9
	Spherical C2	0.164	1	ZYZ	-39.94	3	82	179.3	76.5	759.2
Unit 20 – S Code 20	Nugget C0	0.227	0.227							
	Spherical C1	0.389	0.616	ZYZ	-150.9	28	3	28.4	60.8	138.8
	Spherical C2	0.384	1	ZYZ	-48.94	0	13	47.9	105.4	1,410.5

Table 14-10: Variography Magenta Zone and Code 3000 (October 15 Dataset)

Domain	Component	Increment	Cumulative	Rotation	Angle1	Angle2	Angle3	Range1	Range2	Range3
Magenta Zone – Au Code 2000	Nugget C0	0.004	0.004							
	Exponential C1	0.796	0.8	ZYZ	-47.94	41	-57	34.7	77.2	13.1
	Exponential C2	0.2	1	ZYZ	-102.9	-69	3	48.5	1609.1	469.9
Magenta Zone – Co Code 2000	Nugget C0	0.003	0.003							
	Exponential C1	0.695	0.698	ZYZ	-68.94	83	-14	16.6	91.5	8.6
	Exponential C2	0.302	1	ZYZ	-91.94	35	48	1415.2	297.2	134.7
Magenta Zone – Cu Code 2000	Nugget C0	0.004	0.004							
	Exponential C1	0.81	0.814	ZYZ	-10.94	20	-54	170.1	67.4	19.9
	Exponential C2	0.186	1	ZYZ	-87.94	-53	-4	26.4	1004.3	911.1
Magenta Zone – Ni Code 2000	Nugget C0	0.006	0.006							
	Exponential C1	0.816	0.822	ZYZ	-12.96	27	-63	156.4	89	19
	Exponential C2	0.178	1	ZYZ	-88.9	-53	-3	28.7	1396.2	424.5
Magenta Zone – Pd Code 2000	Nugget C0	0.003	0.003							
	Exponential C1	0.744	0.747	ZYZ	-63.94	57	11	35.5	79.1	11.5
	Exponential C2	0.253	1	ZYZ	-5.94	-88	-25	60.2	272.8	1068.1
Magenta Zone - Pt Code 2000	Nugget C0	0.004	0.004							
	Exponential C1	0.727	0.731	ZYZ	-59.94	59	8	28.3	103.7	1.9
	Exponential C2	0.269	1	ZYZ	-105.9	-74	2	33.1	937.5	246.1
Magenta Zone – S Code 2000	Nugget C0	0.082	0.082							
	Exponential C1	0.723	0.805	ZYZ	-4.94	21	-97	149.2	87.1	19
	Exponential C2	0.195	1	ZYZ	-88.94	-68	-2	26.5	551.9	332.2
Unit 3,4,5,6,7 – Au Code 3000	Nugget C0	0.3	0.3							
	Exponential C1	0.7	1	ZYZ	5.06	-22	18	210.6	78.5	20.2
Unit 3,4,5,6,7 – Co Code 3000	Nugget C0	0.152	0.152							
	Exponential C1	0.848	1	ZYZ	-5.94	0	7	101.9	17.2	1321.8
Unit 3,4,5,6,7 – Cu Code 3000	Nugget C0	0.006	0.006							
	Exponential C1	0.994	1	ZYZ	69.06	20	-55	410	29.7	21
Unit 3,4,5,6,7 – Ni Code 3000	Nugget C0	0.142	0.142							
	Exponential C1	0.858	1	ZYZ	12.06	-13	-11	318.9	19.4	58.2
Unit 3,4,5,6,7 – Pd Code 3000	Nugget C0	0.4	0.4							
	Exponential C1	0.6	1	ZYZ	-47.94	25	31	216.2	66.1	27.7
Unit 3,4,5,6,7 – Pt Code 3000	Nugget C0	0.133	0.133							
	Exponential C1	0.867	1	ZYZ	-11.94	37	-14	133.4	87.8	9.8
Unit 3,4,5,6,7 – S Code 3000	Nugget C0	0.011	0.011							
	Exponential C1	0.989	1	ZYZ	79.06	18	-55	176.4	56.9	28.2

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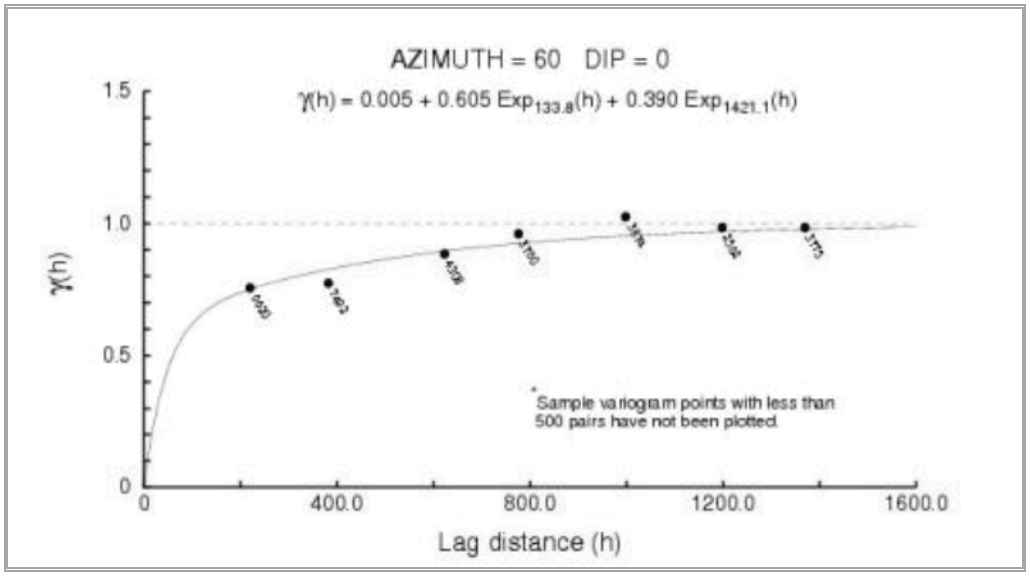
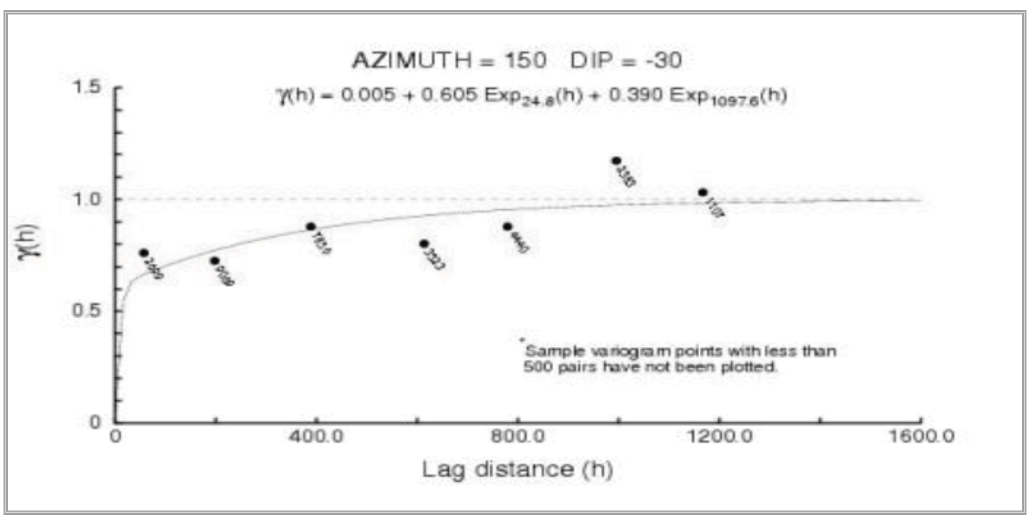
Generally, ranges for the copper correlogram in the main DOM1 grade shell reach 1,000 ft at approximately 96% of the 1.0 sill level in the main strike direction as shown in Figure 14-12.

Figure 14-12: Copper Correlogram for Domain 1001 – Main Strike Direction



In the down dip direction, the range is shorter reaching about 800 ft at about 96% of the sill value as shown in Figure 14-13. The variography indicate good continuity in the grade distribution, the contact profile show a good marker horizon exists between unit 1 and unit 3 which is consistent with PolyMet's NorthMet field geologists being able to predict the location of the high grade horizon with a relatively good degree of accuracy prior to drilling.

Figure 14-13: Copper Correlogram for Domain 1001 - Down Dip Direction



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The Magenta Zone show shorter ranges with a maximum range of 800 ft at the sill in the main strike direction and 500 ft in the down dip direction.

Domain 1003 did not provide enough points to generate a reliable correlogram and AGP elected to use the lithological Unit 3 points for the spatial analysis in lieu of the domain 1003 points.

The complete spatial analysis is attached in Appendix F.

14.7 Density Assignment

PolyMet's October 15, 2007 database contains 6,997 specific gravity/density measurements.

Mark J. Severson et al., Natural Resources Research Institute of the University of Minnesota, Duluth compiled 1,037 comparative specific gravity (SG) determinations in 1999-2000 using Jolly balance determinations on smaller pieces and duplicate measurements of displacement and weight ("graduated cylinder method") on larger core pieces.

From this work, Severson reported the following:

> When compared to the Jolly Balance method, the Graduated Cylinder method is not only faster (about 25 samples per hour, versus the Jolly Balance's 30-40 samples per day), but just as accurate.

and subsequently concluded:

> In most cases, sample variance is smaller for the Graduated Cylinder method than the Jolly Balance method, probably because the Graduated Cylinder method uses a much larger sample. This sheer difference in specimen size makes the Graduated Cylinder samples more robust to minor variations. Furthermore, the relatively simple nature of the Graduated Cylinder method reduces the chance for introducing measurement errors.

PolyMet used primarily the Graduated Cylinder method for subsequent specific gravity determination. The distribution of the data including all determinations in the database is shown in Table 14-11.

Table 14-11: Percentage of Specific Gravity Determination by Method (October 15 Dataset)

Method	Percent of Total Determination	Average SG
PolyMet Graduated Cylinder	82	2.93
PolyMet Weight in Water	3	2.95
Severson/Zanko Data - Graduated Cylinder	14	2.92
Severson/Zanko Data - Jolly Balance	1	2.93
Chemex (average)	0.1	2.91

Density measurements to date have been made on core that has not been oven dried and has not been sealed. This is likely to have resulted in a small (~1%) overstatement due to the inclusion of moisture that would normally be driven off at 105 to 110°C. It is recommended that approximately 50

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samples be selected and the weight loss be determined after drying for the same temperature and duration as used by the assay laboratory.

The QP considered the specific gravity determination using the graduated cylinder method to be accurate enough to use in the resource estimation.

Table 14-12 list the average specific gravity determination including all determination (October 2007 dataset) sorted by unit.

Table 14-12: Specific Gravity Average per Unit (October 15 Dataset)

Unit	Mean	Count
1	2.98	2,381
3 (2+3)	2.92	1,818
5 (4+5)	2.90	1,266
6	2.90	902
7	2.92	326
20	2.77	273
30	3.17	9
All Units	2.93	6,975

14.8 Resource Model Definition

One block model was constructed in Gemcom's GEMS version 6.04™ software. The block size was 50 ft by 50 ft by 20 ft to allow for detailed engineering of the resource model.

The block model matrix was defined using the following coordinates (block edge) based on the Minnesota State Plane Grid (North Zone, NAD83, NAVD 88):

- Easting: 2,896,240.59081
- Northing: 728,838.73616
- Top elevation: 1,620
- Number of blocks in the X direction: 399
- Number of blocks in the Y direction: 122
- Number of blocks in the Z direction: 81

The model is rotated 33.94 degrees counter-clockwise around the origin giving the model X direction an azimuth of 56.06 degree. The block model matrix covers the area bounded by the coordinates listed in Table 14-13.

Table 14-13: Maximum and Minimum Coverage for the Block Model Matrix (edge to edge)

Coordinate	Minimum	Maximum

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Coordinate	Minimum	Maximum
Easting	2892834.810	2912791.563
Northing	728838.736	745038.007
Elevation	0.00	1620

A unit model was assigned a code corresponding to the integer code of the lithological units. Blocks in this model have a value of 30, 20, 1, 3, 5, 6, or 7. A domain model was coded using the DOM1SOL, MAGZONE, and two Virginia Formation inclusions wireframe named CODE21 and RAMP-07 in the database. Blocks in this model have values of 1000 for the DOM1 grade shell, 2000 for the Magenta Zone grade shell, and 21 or 23 for the two major Virginia Formation inclusions. The final grade domain code was calculated in the Rocktype model using a block model manipulation script where the block integer code was assigned according to the matrix in Table 14-14 and illustrated in Figure 14-14 graphically.

Table 14-14: Grade Domain Coding Matrix

Domain Code	Unit Code						
	30	20	1	3 (2+3)	5 (4+5)	6	7
-	30	-	-	-	-	-	-
-	-	20	-	-	-	-	-
23	23	23	23	23	23	23	23
22	22	22	22	22	22	22	22
1000	-	-	1,001	1,003	-	-	-
2000	-	-	2,000	2,000	2,000	2,000	2,000
3000	-	-	-	3,000	3,000	3,000	3,000

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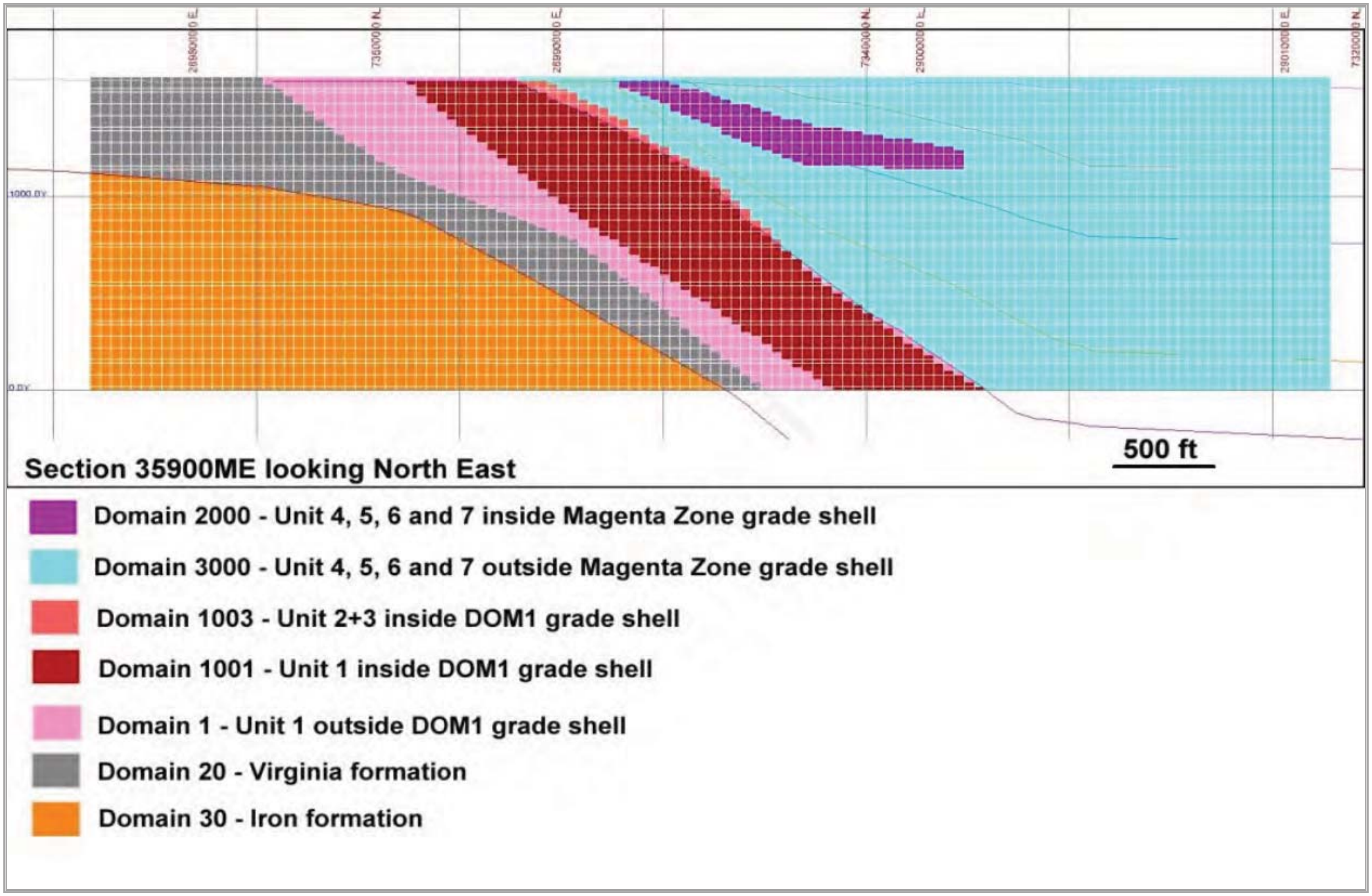
Figure 14-14: Final Grade Domain Code in the Gemcom© Rocktype Model



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14.9 Interpolation Plan

Interpolation was carried out in five passes with an increasing search radius coupled with a decreasing sample density restriction. The interpolation plan used for the NorthMet Deposit allows for a limited soft boundary across the grade shell domain DOM1 and its surrounding unit code. The soft boundary search was limited to the most restrictive Pass 1 search in order to avoid high grade smearing into the lower grade areas or vice versa, as the search ellipsoid becomes larger in the subsequent passes. With the exception of DOM1 grade shell boundary, the remaining grade domains were treated as hard boundaries.

The search ellipsoids orientation and dip were tweaked in this resource estimate to coincide better with the average strike and dip angle of the NorthMet Deposit. Grade shell DOM1 shows an average azimuth of 59.6° and dips towards the southeast at 28.6°. The Magenta Zone is flatter, exhibiting a strike of 51.7° dipping southeast at 14.5°. Units 1 and 20 were kept at the average deposit strike of 56.06° and dipping southeast at 30°.

Search ranges were based on the density of diamond drilling and the two main ore domain copper correlograms. Generally, the ratio between the major and semi-minor axis is 0.56 while the ratio between the semi-minor and minor axis was kept around 0.23 for Pass 1 to Pass 4 inclusively. The incremental ratio of the major axis between passes was 0.5, 0.66 and 0.45 respectively for Pass 1 to Pass 2, Pass 2 to Pass 3 and Pass 3 to Pass 4.

Table 14-15 summarizes the ellipsoid dimensions used in the different passes while Table 14-16 summarizes the search angle and search restriction imposed on the high grade outliers as described in the capping section (Section 14-4) of this report.

A series of model in the block matrix called Nbsamp1, Nbsamp2, Nbsamp3 and Nbsamp4 recorded the number of samples used to interpolate the blocks. These models were used in a block manipulation script to fill a PassNb model with a value of 1, 2, 3 or 4 representing at what pass a given grade was interpolated.

The target domain code and sample code controls the soft/hard boundary of the model. When a block is interpolated with a given target domain code the software will load the point file according to the grid listed in Table 14-17 and Table 14-18.

Table 14-15: Ellipsoid Dimensions

	Ellipsoid dimension (in ft)			Number of Samples Used			
	X	Y	Z	Min	Max	Max per hole	Comment
Pass 1	300	170	40	6	15	5	Minimum of two holes required
Pass 2	600	340	80	6	15	5	Minimum of two holes required
Pass 3	900	500	115	2	15	5	
Pass 4	2,000	1,100	265	2	15	5	
Pass 5	8,000	6,000	1,200	2	15	5	Use to fill un-interpolated blocks

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Table 14-16: Sample Search Parameters (all passes)

	Search Angle			Search Restriction Size and High Grade Threshold Value Used								
	Z	X	Z	Z	X	Z	Au (ppb)	Cu (%)	Ni (%)	Pd (ppb)	Pt (ppb)	S (%)
Dom 20,22,23	0	30	0	75	75	75	80	0.7	0.18	1000	200	7.5
Dom 1	-6	29	0	75	75	75	500	1.8	0.6	1600	450	7.5
Dom 1001	-6	29	0	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
Dom 1003	-6	29	0	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
Dom 3000	-5	18	0	75	75	75	500	2.1	0.4	4000	700	8
Dom 2000	4	15	0	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a

Table 14-17: Pass 1 – Target Domain Code and Sample Code Used

	20	1	1001	1003	3000	2000	22	23
20	x							
1		x	x					
1001		x	x					
1003				x	x			
3000				x	x			
2000						x		
22							x	
23								x

Table 14-18: Pass 2,3,4 and 5 – Target Domain Code Sample Code Used

	20	1	1001	1003	3000	2000	22	23
20	x							
1		x						
1001			x					
1003				x				
3000					x			
2000						X		
22							x	
23								x

The density model was initialized with the unit average density from Table 14-18. The density data collected by PolyMet was interpolated into the model using a simple inverse distance model with a fairly restrictive search ellipse of 300 ft x 300 ft x 75 ft. The minimum number of samples was set to six, the maximum was fifteen and a maximum of five samples per hole was imposed. In total, 3.22% of all the blocks in the model were interpolated by the inverse distance method.

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14.9.1 Minor Elements

AGP carried out a geostatistical study of the elements that may have a measurable effect on stockpile drainage water quality for waste characterization and environmental purposes. The thirteen elements analyzed were silver, arsenic (As), barium (Ba), beryllium (Be), cadmium (Cd), chromium (Cr), manganese (Mn), molybdenum (Mo), phosphorus (P), lead, antimony (Sb), vanadium (V), zinc (Zn). The water quality elements grades were interpolated using an inverse distance square technique in a separate block model in GEMS version 6.04. The model matrix was replicated from the main resource grade model and thus occupies the same space.

14.10 Classification of Mineral Resources

Several factors are considered in the definition of a resource classification:

- Canadian Institute of Mining (CIM) requirements and guidelines
- experience with similar deposits
- spatial continuity
- confidence limit analysis
- geology

No environmental, permitting, legal, title, taxation, socio-economic, marketing or other relevant issues are known to the author that may affect the estimate of mineral resources. Mineral resources tabulated in section 14-11, are not mineral reserves and do not have demonstrated economic viability. Reserves can only be estimated on the basis of an economic evaluation that are used in a Pre-Feasibility or Feasibility Study of a mineral project and are tabulated in section 15-3 of this report.

Four confidence categories exist in the model. The usual CIM guidelines of Measured, Indicated and Inferred classes are coded 1, 2 and 3 respectively. A special code 4 called “Fill” in this report represents what are typically un-interpolated blocks. NorthMet requires that all blocks in the model carry sulphur value in addition to the six primary grade elements for environmental purposes and therefore a fourth and fifth pass was used, with a large search ellipsoid, so that all blocks in the model are populated with a grade value.

Typically, confidence level for a grade in the block model is reduced with the increase in the search ellipsoid size along with the diminishing restriction on the number of samples used for the grade interpolation. This is essentially controlled via the pass number of the interpolation plan describe in the previous section. A common technique is to categorize a model based on the pass number and distance to the closest sample. In numeric models with hard boundaries between grade domains the technique has a tendency to stripe the model with measured category in close proximity with inferred category. If the interpolation uses a minimum number of holes similar to pass 1 and pass 2 in the current model, this effect can be aggravated showing an indicated category in between drill holes where a series of blocks were interpolated with the pass 1 with a minimum of 2 drill holes restriction while the blocks located directly on the drill holes could not see the next hole end up classified as inferred.

For the NorthMet Deposit, AGP elected to classify the mineral resource primarily using the Pass number from the interpolation plan with help from a core area model to minimize having blocks in the measured category in close proximity with blocks in the inferred category.

The core area model represents the density of the drilling in the resource model based on two components; the position of the drillholes and the number of drillholes surrounding the blocks in the matrix. The model was created as follows:

- A model in the block model matrix called DDH175 was first created by assigning the percentage of the blocks inside a 175 ft extruded drillhole trace. The model was then interpolated with a inverse distance methodology using octant search with a round ellipse of 300 ft x 300 ft x 60 ft in order to fill the spaces in the immediate vicinity of the drillhole 175 ft extruded trace. The model contains values from 0 to 100% representing how far a block center is from a 175 ft extruded drillhole trace where 100% means the block is fully within the trace of the drillhole shown in the top right inset image of Figure 14-15.
- A second model called NBHoles was created in the block model matrix containing the same number of drillholes that are visible from a given block in the model within a 300 ft search bubble. The model contains values from 0 to 15 representing the number of drillholes visible within a 300 ft search bubble from the block center shown in the bottom left inset of Figure 14-15.
- A third and final model called Core was constructed in the block model matrix containing the combination of the DDH175 model and the NBHoles model weighted at a 25/75 ratio between the DDH175 and NBHoles model respectively. This procedure essentially eliminated the stripping effect visible in the DDH175 model for holes near the fringe area of the core while giving more weight to the number of drillholes visible from a block center. The resulting model carries an empirical value from 0 to 81.25 (average 7.131) describing more or less the number of drillholes visible to a block center in relation to the proximity to the nearest hole. A high value is well within the core area drilled by PolyMet's NorthMet staff geologists while a low value is near the fringe. The core area values are shown in the main image Figure 14-15.

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Figure 14-15: Core Area with Drillhole Traces

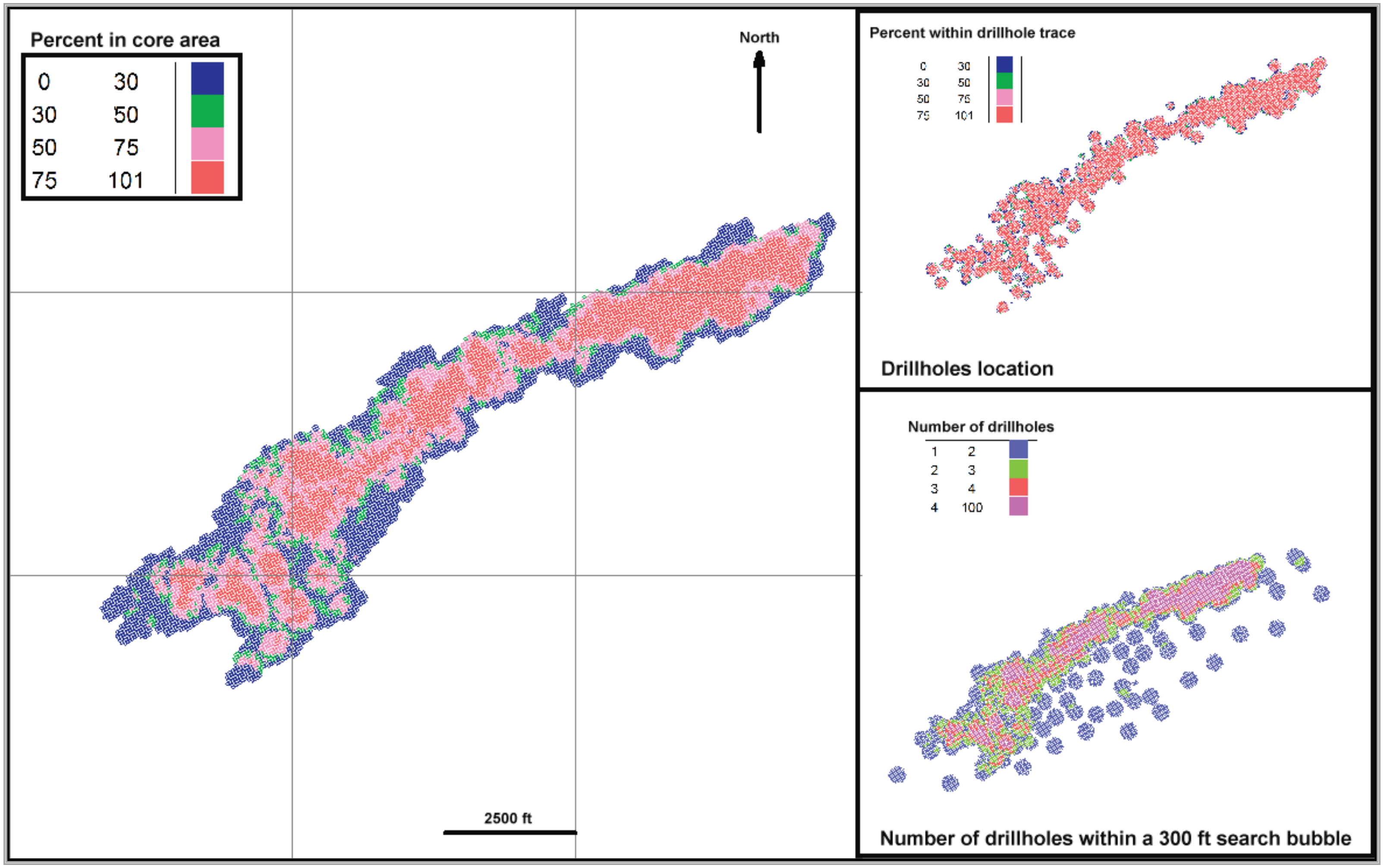

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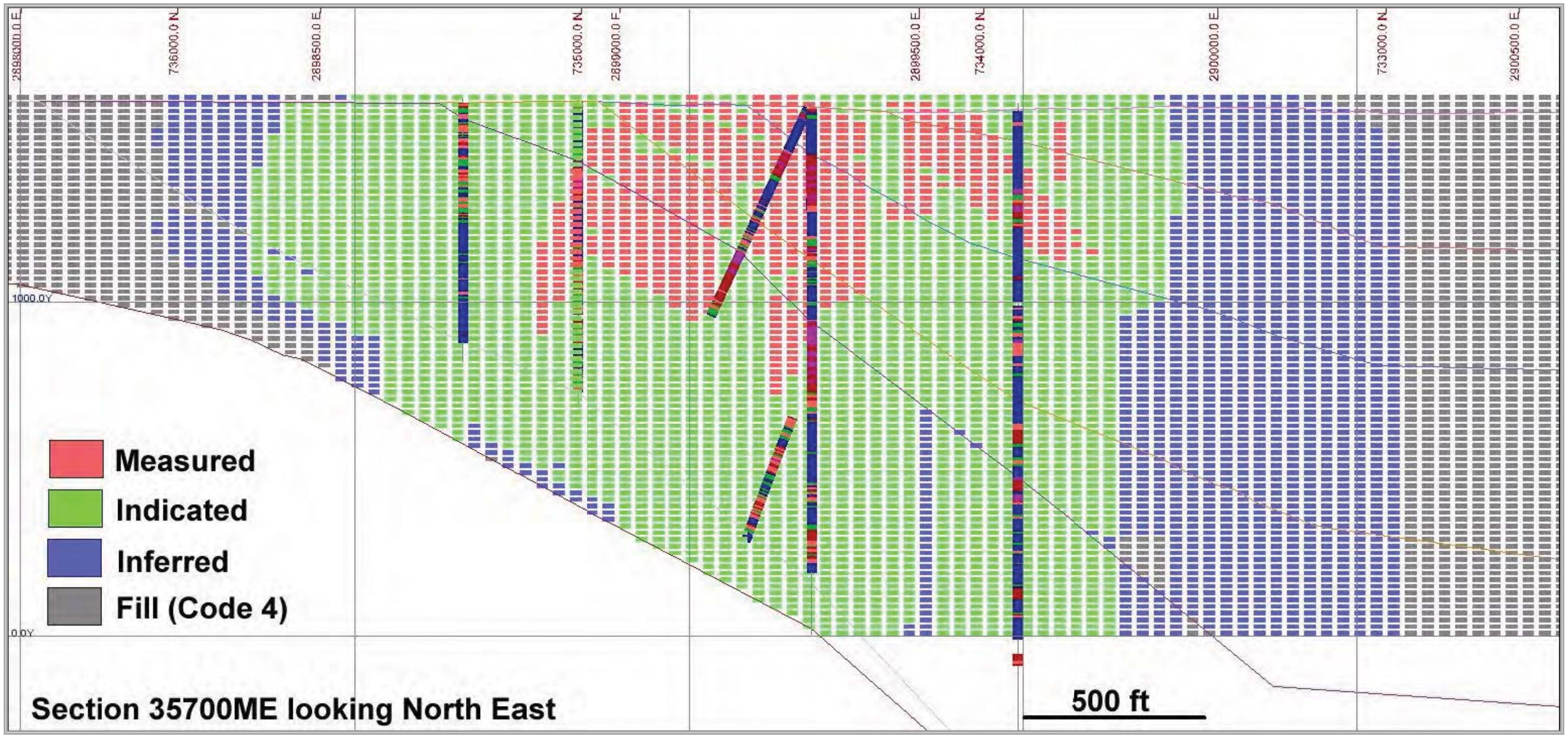
Figure 14-16: Section 35700ME Classification Model



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The category model was coded using the pass number to define the Measured, Indicated and Inferred category in combination with the core area model as per schedule in Table 14-9 where a block located outside the core area was likely to be downgraded in category. The procedure allowed the fine tuning of the measured category.

Table 14-19 summarizes the classification parameters used for the category models. Based on the criteria outlined in

Table 14-20, 3% of the blocks estimated at the NorthMet project are classified as Measured, 14% of the blocks are Indicated and 22% of the blocks are Inferred. The remaining blocks are either non-interpolated, category 4 or "fill." Figure 14-6 shows a representative section of the category model.

Table 14-19: Classification Parameters

Pass Number	Inside Core	Outside Core
Pass 1	Measured if Core value > 75	Indicated
Pass 2	Indicated	Indicated
Pass 3	Indicated	Inferred
Pass 4	Inferred	Fill
Pass 5	Fill	Fill

Table 14-20: NorthMet Project Category Model Tabulation

Unit	Total No. of Blocks	Measured		Indicated		Inferred		Non-Interpolated or Fill	
		No. of Blocks	%	No. of Blocks	%	No. of Blocks	%	No. of Blocks	%
20,1,3,5,6,7	2,829,567	109,992	3	560,643	14	880,740	22	1,278,192	32
30 or Air	1,113,351	-	0	-	0	-	0	1,113,351	28
Total Block	3,942,918								

14.10.1 Net Metal Value Formula

For comparison purposes, AGP was requested by PolyMet to use the same metal price and recovery figures used previously in the report titled "Technical Report on the Results of a Definitive Feasibility Study of the NorthMet Project" authored by D.J. Hunter and dated October 2006 and also used in the September 2007 resource model.

Net Metal Value is calculated as follows:

1) For all elements a net metal price is calculated:

Net Metal Price = (Metal price - Refining, insurance and transport charge)

2) For each element, a factor is calculated:

a) For Copper and Nickel (expressed in %):

Factor = Net Metal Price * Recovery Ore to Conc. * Recovery Conc. To Metal * Conversion % to lbs

b) For Cobalt (expressed in ppm):

Factor = Net Metal Price * Recovery Ore to Conc. * Recovery Conc. To Metal * Conversion ppm to % * Conversion % to lbs

c) For Platinum, Palladium and Gold (expressed in ppb):

Factor = Net Metal Price * Recovery Ore to Conc. * Recovery Conc. To Metal * Conversion ppb to ppm * Conversion ppm to troy oz

3) For all elements, the value per tonne is calculated in US$:

Value/tons = grade * factor

4) Total NMV is the addition of the Value per tons for each element:

NMV = Value/tons Cu + Value/tons Ni + Value/tons Co + Value/tons Pt + Value/tons Pd + Value/tons Au

Table 14-21 lists the price, recoveries, refining, insurance and transportation charge used in the calculation. Conversion factors used are:

- percent to pounds per short ton multiply by 20
- ppm to percent multiply by 0.0001
- ppb to ppm multiply by 0.001
- ppm to troy ounces multiply by 0.02917 or (1/34.285).

Table 14-21: NMV Input Parameters

Metal in Model	Unit	Metal Price ($)	Refining, Insurance and Transport ($)	Recovery Ore – Concentrate	Recovery Concentrate – Metal
Copper (%)	US$/lb	1.25	0.00	0.9420	0.980
Nickel (%)	US$/lb	5.60	1.40	0.7250	0.970
Cobalt (ppm)	US$/lb	15.25	6.10	0.4200	0.970
Platinum (ppb)	US$/troy oz	800.00	18.00	0.7690	0.945
Palladium (ppb)	US$/troy oz	210.00	17.00	0.7960	0.945
Gold (ppb)	US$/troy oz	400.00	9.50	0.7570	0.885

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14.11 Mineral Resource Tabulation

Table 14-22 shows resources below the overburden bottom surface to 0.00 elevation for Unit 20, 1, 3 (2+3), 5 (4+5), 6 and 7. The base case is using a cut-off grade of 0.2% copper.

Table 14-22: Resource Model Summary at 0.2% Cu Cut-off

Cut-off @ 0.2% Cu	Volume (M ft3)	Density (st/ft3)	Tonnage (M st)	Cu (%)	Ni (%)	S (%)	Pt (ppb)	Pd (ppb)	Au (ppb)	Co (ppm)
Measured	1,530.3	0.093	141.9	0.338	0.094	0.81	81	301	42	77
Indicated	3,244.0	0.093	300.2	0.318	0.087	0.78	81	287	41	72
M+I	4,774.3	0.093	442.1	0.325	0.089	0.79	81	292	41	73
Inferred	1,712.8	0.093	158.7	0.329	0.088	0.73	86	315	43	55

Table 14-23 shows the resource sensitivity to changes in cut-off with the base case cut-off highlighted.

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Table 14-23: Cumulative Resource Model Results at Various Cu % Cut-offs (for sensitivity only)

Cut-off	Volume (M ft3)	Density (st/ft3)	Tonnage (M st)	Cu (%)	Ni (%)	S (%)	Pt (ppb)	Pd (ppb)	Au (ppb)	Co (ppm)
Measured										
>0.5	126.1	0.093	11.7	0.574	0.140	1.08	124	485	62	89
>0.4	395.8	0.093	36.7	0.485	0.125	1.01	108	417	55	86
>0.3	852.5	0.093	79.1	0.411	0.110	0.93	95	360	49	82
>0.2	1530.3	0.093	141.9	0.338	0.094	0.81	81	301	42	77
>0.1	2529.3	0.093	234.4	0.263	0.077	0.67	64	232	33	71
Indicated										
>0.5	207.2	0.093	19.2	0.577	0.131	1.04	138	509	69	80
>0.4	629.8	0.093	58.3	0.487	0.117	0.97	119	438	61	77
>0.3	1503.5	0.093	139.2	0.404	0.103	0.89	100	365	51	74
>0.2	3244.0	0.093	300.2	0.318	0.087	0.78	81	287	41	72
>0.1	7078.7	0.092	654.2	0.223	0.066	0.65	54	187	29	66
Inferred										
>0.5	137.4	0.093	12.8	0.607	0.139	1.04	160	635	85	66
>0.4	349.3	0.093	32.4	0.512	0.119	0.91	139	531	72	62
>0.3	875.8	0.093	81.3	0.411	0.105	0.83	108	407	53	58
>0.2	1712.8	0.093	158.7	0.329	0.088	0.73	86	315	43	55
>0.1	3133.6	0.092	289.6	0.246	0.068	0.62	62	221	32	52

Table 14-24 reports resources above an elevation of 0.00 ft using an NMV value of US$7.42 derived from the same metal prices and recoveries used previously in the Hunter, 2006 report and also in the Wardrop resource model dated September 2007.

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Table 14-24: Resource Model Summary at US$7.42 NMV

Cut-off @ US$7.42 NMV	Volume (M ft3)	Density (st/ft3)	Tonnage (M st)	Cu (%)	Ni (%)	S (%)	Pt (ppb)	Pd (ppb)	Au (ppb)	Co (ppm)	NMV (US$)
Measured	2,185.03	0.093	202.5	0.285	0.083	0.71	71	258	36	74	14.58
Indicated	5,319.88	0.093	491.7	0.256	0.075	0.69	66	231	34	70	13.20
M+I	7,504.91	0.093	694.2	0.265	0.077	0.69	68	239	35	71	13.60
Inferred	2,484.53	0.092	229.7	0.273	0.079	0.65	73	263	37	56	13.97

14.12 Block Model Validation

The NorthMet grade models were validated by two methods:

- Visual comparison of colour-coded block model grades with composite grades on section plots.
- Comparison of the global mean block grades for ordinary kriging, inverse distance, nearest neighbour models, composite grades and raw assay grades.

14.12.1 Visual Comparisons

The visual comparisons of block model grades with composite grades show a reasonable correlation between the values. No significant discrepancies were apparent from the sections reviewed.

14.12.2 Global Comparisons

The grade statistics for the raw assay grade, composite grade, ordinary kriging, nearest neighbour and inverse distance models, are tabulated below in Table 14-25. Figures 14-17 and 14-18 show the differences. Grade statistics for composite mean grade compared to raw assay grade indicated a normal reduction in values for all elements. The block model mean grade when compared against the composites also indicated a normal reduction in values for all elements.

Percent changes in metal content shown in Table 14-26 between the nearest neighbour, inverse distance and ordinary kriging model are in very close agreement among all three methods with less than 2.0% difference in all elements except for cobalt showing 3.1% difference between the ordinary krig model and the nearest neighbour model.

Table 14-25: Global Grade Comparison at 0.00 Cu% Cut-off

Source	Cu (%)	Ni (%)	S (%)	Pt (ppb)	Pd (ppb)	Au (ppb)	Co (ppm)
Assay	0.160	0.055	0.44	40	140	21	62

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Source	Cu (%)	Ni (%)	S (%)	Pt (ppb)	Pd (ppb)	Au (ppb)	Co (ppm)
Composite	0.119	0.041	0.38	31	105	16	47
Block NN with MII*	0.059	0.023	0.26	16	51	8	30
Block ID with MII	0.060	0.024	0.26	16	51	8	30
Block OK with MII	0.060	0.024	0.26	16	51	8	31
Block OK with MIIF*	0.052	0.022	0.24	15	45	6	30

Note: * MII – Measured, Indicated and Inferred. MIIF – Measured, Indicated, Inferred and Filled

Figure 14-17: Global Grade Comparison for Unit 1-7, Cu %, Ni% and S%

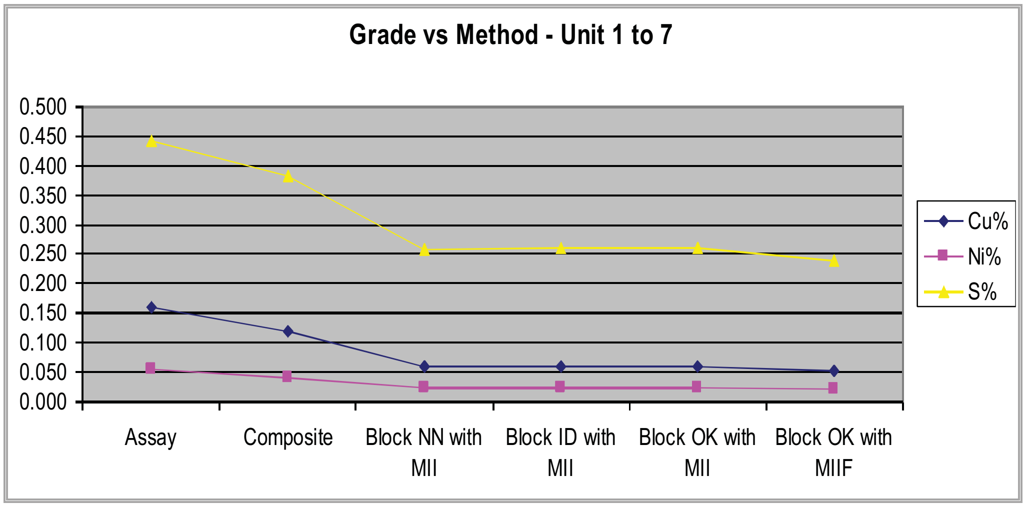

Figure 14-18: Global Grade Comparison for Unit 1-7, Pt (ppb), Pd (ppb), Au (ppb) and Co (ppm)

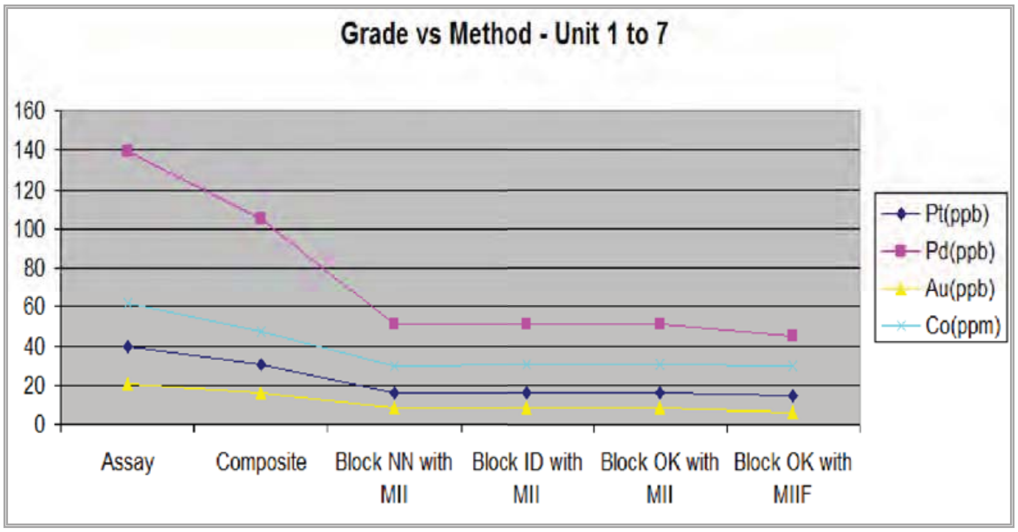

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Table 14-26: Global Comparison at 0.00 Cu% Cut-off (Percent Difference in Metal Content)

Method	Cu % Diff	Ni % Diff	S % Diff	Pt % Diff	Pd % Diff	Au % Diff	Co % Diff
NN - Base case	0	0	0	0	0	0	0
OK – NN	1.9	1.6	0.6	1.4	1.0	1.2	3.1
ID – NN	1.5	0.8	0.6	1.3	0.6	0.8	1.6
OK – ID	0.4	0.8	0.0	0.1	0.4	0.5	1.5

14.12.3 Block Model Comparison with the Previous Resource Estimate

The December 2007 resource estimate was compared with the figure listed in Table 17-23 of the Wardrop, September 2007 report.

Volumes and tonnages were compiled for the December 2007 resource estimate from the overburden surface down to the 0.00 ft elevation. A NMV cut-off of US$7.42 was selected using the same metal price and recoveries used in the previous estimate.

Results shown in Table 14-27 indicated a slight increase of 15.5 million short tons in the Measured category and 40.5 million short ton in the Indicated category for a total of 56 million short tons or 8.1% increased in the Measured plus Indicated category. The Inferred Resource dropped by 21.9 million short tons or 9.5%.

Grades in the Measured and Indicated categories dropped slightly for copper and nickel and increased slightly for platinum, palladium, gold and cobalt grade elements. Copper changed by -0.3%, nickel by -0.5%, platinum by +2.1%, palladium by +1.8%, gold by +2.1% and cobalt by +0.1% as shown in Figure 14-19.

The contained metal value shown in Table 14-28 increased for all elements by about 10% in the Measured and Indicated categories. Copper increased by 8.5%, nickel by 8.2%, platinum by 11.1%, palladium by 10.8%, gold by 11.0% and cobalt by 8.9% as shown in Figure 14-20.

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Figure 14-19: Resource above 0.00 ft Comparison – Grade

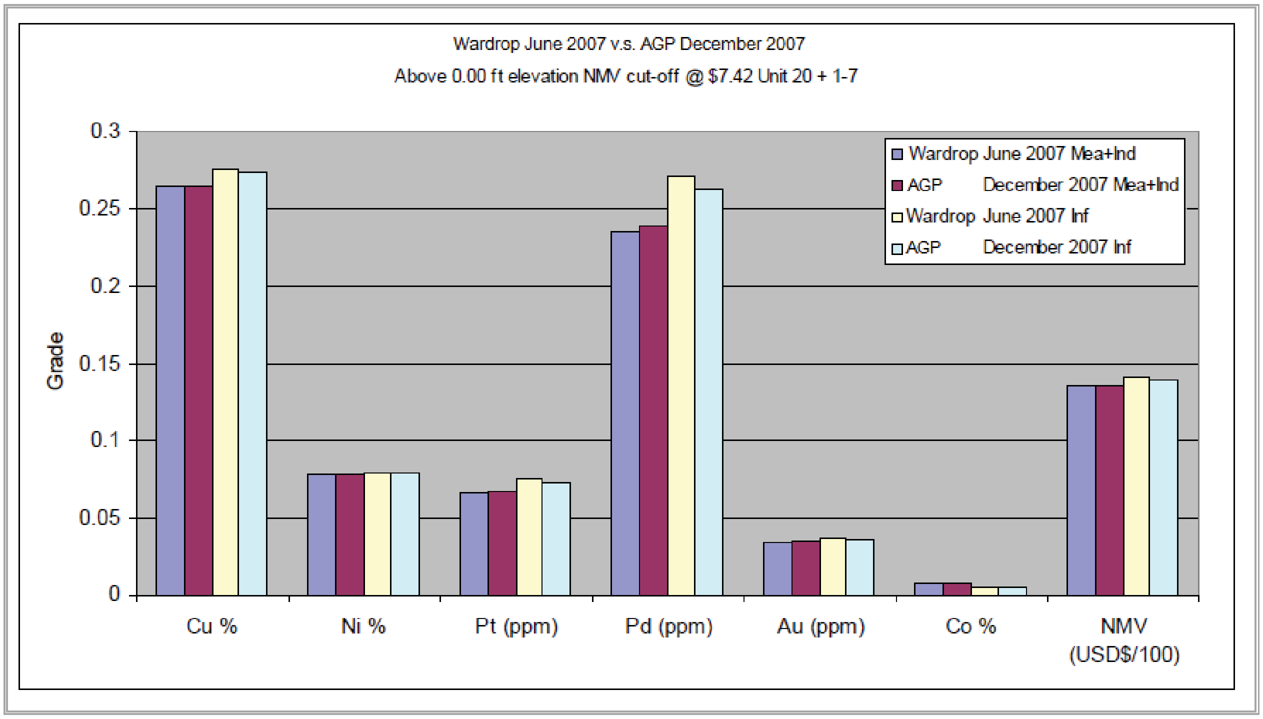

Figure 14-20: Resource above 0.00 ft Comparison – Product

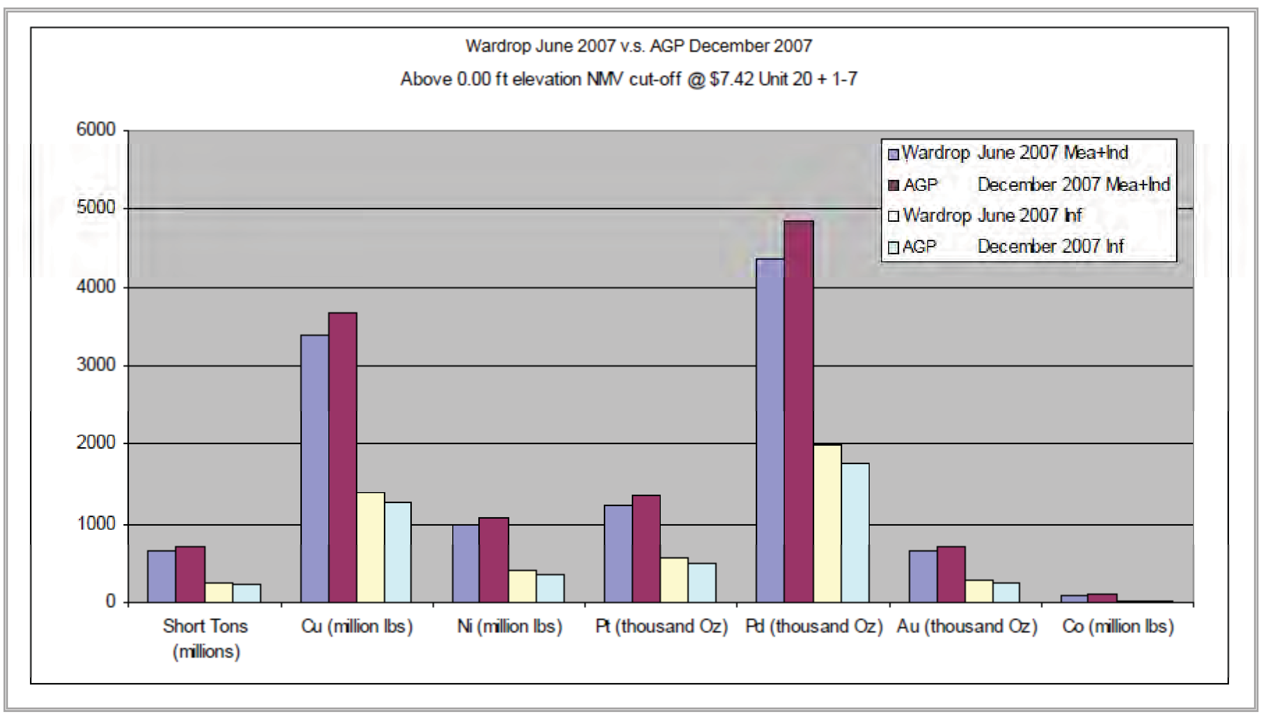

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Table 14-27: Resource above 0.00 ft Comparison – Grade at US$7.42 NMV Cut-off

Source	Tonnage (Mst)	Cu %	Ni%	S %	Pt (ppb)	Pd (ppb)	Au (ppb)	Co (ppm)
Wardrop Jun 2007 - Measured	187.0	0.287	0.084	0.72	68	256	35	73
AGP Dec 2007 - Measured	202.5	0.285	0.083	0.71	71	258	36	74
	15.5	-0.5%	-0.8%	-0.8%	3.9%	1.1%	2.7%	0.3%
Wardrop Jun 2007 - Indicated	451.1	0.256	0.075	0.68	65	226	34	70
AGP Dec 2007 - Indicated	491.7	0.256	0.075	0.69	66	231	34	70
	40.6	-0.1%	-0.4%	0.7%	1.4%	2.2%	1.8%	0.0%
Wardrop Jun 2007 - Mea + Ind	638.2	0.265	0.078	0.69	66	234	34	71
AGP Dec 2007 - Mea + Ind	694.2	0.265	0.077	0.69	68	239	35	71
Difference (Dec - Jun)	56.086	-0.001	0.000	0.002	1.414	4.255	0.701	0.075
% Difference (Dec-Jun)	8.8%	-0.3%	-0.5%	0.2%	2.1%	1.8%	2.1%	0.1%
Wardrop Jun 2007 - Inferred	252	0.275	0.079	0.64	76	272	37	56
AGP Dec 2007 - Inferred	230	0.273	0.079	0.65	73	263	37	56
Difference (Dec - Jun)	-21.921	-0.002	0.000	0.013	-3.450	-8.800	-0.476	0.544
% Difference (Dec-Jun)	-8.7%	-0.6%	-0.2%	2.0%	-4.5%	-3.2%	-1.3%	1.0%

Table 14-28: Resource above 0.00 ft Comparison – Product at US$7.42 NMV Cut-off

Source	Tonnage (Mst)	Cu (M lb)	Ni (M lb)	S (M lb)	Pt (Koz)	Pd (Koz)	Au (Koz)	Co (M lb)
Wardrop Jun 2007 - Measured	187.0	1072	314	2680	372	1394	192	27
AGP Dec 2007 - Measured	202.5	1154	337	2879	418	1526	214	30
	15.5	7.7%	7.5%	7.4%	12.5%	9.4%	11.2%	8.7%
Wardrop Jun 2007 - Indicated	451.1	2314	680	6150	860	2969	442	63
AGP Dec 2007 - Indicated	491.7	2519	738	6749	950	3307	491	68
	40.6	8.8%	8.5%	9.7%	10.5%	11.4%	10.9%	9.0%
Wardrop Jun 2007 - Measured + Indicated	638.2	3,386	994	8,830	1,232	4,363	634	90
AGP Dec 2007 - Measured + Indicated	694.2	3,673	1,075	9,628	1,369	4,833	704	98
Difference (Dec - Jun)	56.1	287.3	81.5	798.0	136.9	469.6	69.9	8.0
% Difference (Dec-Jun)	8.8	8.5	8.2	9.0	11.1	10.8	11.0	8.9
Wardrop Jun 2007 - Inferred	252	1385	397	3204	560	1994	272	28
AGP Dec 2007 - Inferred	230	1257	361	2983	488	1761	245	26
Difference (Dec - Jun)	-21.9	-128.8	-35.5	-221.0	-71.9	-232.7	-26.9	-2.2
% Difference (Dec-Jun)	-8.7	-9.3	-8.9	-6.9	-12.8	-11.7	-9.9	-7.8

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15 MINERAL RESERVE ESTIMATES

15.1 Key Assumptions/Basis of Estimate

Mineral Reserves for Northmet are supported by a LOM plan which was developed using the following key parameters.

15.1.1 Pit Slopes

The June 2006 Golder report provided parameters for the Reserve statement. The Golder report was also used as the basis for the DFS Update.

The Golder report indicated inter-ramp angles of 51.4 degrees for all sectors, except one, were possible. That one sector utilized an inter-ramp angle of 55.1 degrees and was achieved with a bench face angle of 70 degrees versus the other sectors 65 degree face angle. In all cases, a berm width of 32.8 feet (10 metres) was considered.

The area impacted by the increased bench face angle was minimal. To simplify the pit design, all areas were designed with a bench face angle of 65 degrees, 32.8 foot berm width to achieve an inter-ramp angle of 51.4 degrees.

15.1.2 Stope Considerations

The NorthMet Deposit outcrops in the project area. It is lower grade than typical underground deposits and more disseminated, not providing focused areas of higher grade ore. Due to this, AGP considered only an open pit configuration. No underground mining methods were examined for the purposes of stating reserves.

15.1.3 Dilution and Mining Losses

The Mineral Resource estimate for Northmet is considered to be internally diluted. Additional external dilution adjustments were made at the time of ore and waste delineation for mine planning purposes.

To all blocks above cutoff, an examination of contact dilution was completed. The blocks surrounding an individual block being queried were examined to determine if they were below cutoff. If they were, their weighted average grade was estimated. This was applied to block and a diluted grade by element determined. On average, the dilution percentages for the entire model were:

- Copper = 2.2%
- Nickel = 2.5%
- Platinum = 2.4%

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- Palladium = 2.6%
- Gold = 2.3%
- Cobalt = 0.8%

AGP assumed that the ore loss was equal to the dilution tonnage, so the effect of dilution was only a reduction in overall grade but the tonnage remained constant. Considering the bulk nature of mining proposed, AGP deemed this to be appropriate.

15.2 Conversion Factors from Mineral Resources to Mineral Reserves

Mineral Reserves have been determined from Mineral Resources by taking into account geologic, mining, processing, economic parameters and permitting requirements and are therefore classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves.

15.3 Mineral Reserves Statement

The Qualified Person for the Mineral Reserve estimate is Gordon Zurowski, P.Eng, a principal of AGP Mining Consultants Inc.

Mineral Reserves are reported at commodity prices of:

- Copper = $1.25 / lb
- Nickel = $5.60 / lb
- Platinum = $800.00 / troy ounce
- Palladium = $210.00 / troy ounce
- Gold = $400.00 / troy ounce
- Cobalt = $15.25 / lb

These prices were used to generate the DFS pit shell, within which the reserves were contained. This pit shell is the same design as outlined in the DFS study published October 2006 and developed by Australian Mine Design & Development Pty Ltd. (AMDAD). This pit shell was applied to the updated resource model.

A mining cutoff was used by AGP that was determined on a block by block basis with the following formula:

Block Value ($) = Gross Metal Value – Mining Cost – Processing cost – G&A.

Where:

- Block Value = net value of the block in dollars

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- Gross Metal Value = value of metals considering price, recovery and downstream costs
- Mining Cost = cost to mine ore and waste adjusted for haulage path
- Processing Cost = cost to process ore tonnes
- G&A = anticipated General and Administrative costs

The block value was stored in each block and a cutoff where the block value was greater than or equal to $0.01. This implies that the block would make $0.01 or greater of net revenue (not considering capital) to mine the block and process it for the contained metal. Blocks with a value of $0.00 or less were deemed to be waste material.

Table 15-1 Updated Reserve Estimate – September 2007

Class	Tonnage (Mst)	Grades (Diluted)					
		Copper (%)	Nickel (%)	Platinum (ppb)	Palladium (ppb)	Gold (ppb)	Cobalt (ppm)
Proven	118.1	0.30	0.09	75	275	38	75
Probable	156.5	0.27	0.08	75	248	37	72
Total	274.7	0.28	0.08	75	260	37	73

The following notes should be read in conjunction with Table 15-1:
Rounding as required by reporting guidelines may result in apparent summation differences between tons, grade and contained metal.
Tonnage and grade measurements are in Imperial units.
The reserves are bound within the DFS pit shell.

15.4 Factors That May Affect the Mineral Reserve Estimate

The mine reserves are based on the complete DFS pit shell from the 2006 study, using the updated geologic resource as of September 2007. AGP has developed and prepared costing for a larger pit, but restricted the final phase in the detailed work to maintain similar production tonnage to the September 2007 reserve statement. If Polymet were to decide to extend the mine life, the additional phase (32.5 million tons) could readily be brought into the reserve category indicating potential upside to the project with an additional 2.8 years.

A sustained higher metal price regime has the potential to allow expansion of the existing pit phases both laterally and to depth. In addition, higher metal prices may assist in lowering the cutoff grade within each phase if sufficient plant and stockpile capacity exist.

The project is pursuing environmental permitting which may restrict the overall potential of the proposed mine, although the resources outside the current permit plan indicates that further constraint is unlikely. Any conditions from the permitting review may have the potential to reduce the overall size of the project. These would need to be examined in detail to see what impact, if any potential conditions may have.

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16 MINING METHODS

16.1 Background

PolyMet requested an update of the 2006 DFS plan to take into consideration various changes since the release of the DFS report. These included:

1) Additional drill results which resulted in an updated and reinterpreted NorthMet resource model used for this report,

2) Updated capital and operating costs from vendors and suppliers to reflect current market conditions,

3) Change in equipment selection criteria to larger more productive fleets,

4) Implement owner operated mining rather than contractor mining,

5) Altered mining sequence to improve the mine environmental footprint.

The 2007 resource update was the basis for the updated production schedule developed in this DFS Update. This resource update was a collaborative effort between AGP and Polymet team members. The impact of adding new resources to the NorthMet project were to be examined to allow Polymet management understand the full potential of the NorthMet Deposit and its potential for future mining enhancements.

The property was visited by AGP mining personnel in October 2007. This was to become familiar with the deposit, Polymet personnel and their areas of expertise. It was also to better understand what opportunities may exist in the area of the NorthMet Deposit to assist in improving overall project economics and environmental footprint.

Capital and operating cost estimates in U.S. dollars were determined with current parameters from suppliers and vendors. PolyMet and AGP personnel worked together to determine the complete capital requirements and ensure items were not forgotten in the overall cost estimate.

An internal study examined the potential benefit of larger mining equipment to reduce operating cost and mine emissions. This study indicated that bulk mining fleets offered cost savings that needed to be fully quantified. This was examined.

The DFS project economics utilized contract mining for operating costs. PolyMet management felt that costs savings to the overall project could be achieved by operating the mine themselves and limit the contracting to maintenance and other support services. This was considered for the reserves update.

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Subsequent changes that will be incorporated into project proposal to be described in the Supplemental Draft EIS include altering the mining sequence so that the eastern pit becomes available for backfilling.

With this direction, AGP was instructed to create an update of the DFS plan in sufficient detail to allow a new 43-101 report be issued if required. An updated reserve statement was to be developed at the culmination of the work.

16.2 Geotechnical

No update on the geotechnical parameters has been completed since receipt of Golder's June 2006 report. The Golder report was used as the basis for the DFS Update.

The Golder report indicated inter-ramp angles of 51.4 degrees for all sectors, except one, were possible. That one sector utilized an inter-ramp angle of 55.1 degrees and was achieved with a bench face angle of 70 degrees versus the other sectors 65 degree face angle. In all cases, a berm width of 32.8 feet (10 metres) was considered.

The area impacted by the increased bench face angle was minimal. To simplify the pit design, all areas were designed with a bench face angle of 65 degrees, 32.8 foot berm width to achieve an inter-ramp angle of 51.4 degrees.

16.3 Mining Model Development

The geologic block model was constructed in Gemcom© by AGP with the assistance of PolyMet personnel. This model was then imported into Minesight© for use in the pit optimizations and production schedule development. The dimensions of the models remained the same for the mining models. Items that were brought across were:

- Rock Type
- Density
- Classification (Measured, Indicated and Inferred)
- Rock Type
- Unit
- Domain
- Specific Gravity
- Copper grade (%)
- Nickel grade (%)
- Sulphur (%)
- Platinum grade (parts per billion)
- Palladium grade (parts per billion)

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- Gold grade (parts per billion)
- Cobalt grade (parts per million)

PolyMet provided topography and overburden surfaces for use in both the geologic model and mining model.

A recovery item was included in the mining model to consider the impact lower grades would have on recovery. A fixed recovery for all grade items was used in DFS which AGP deemed potentially optimistic for very low grade material without detailed testing at the lower grades.

To examine the impacts of lower grade, a fixed tail recovery formula was applied to each block for each grade item. PolyMet provided the tail grades that had been determined from the previous round of metallurgical testing for the copper and nickel grades. The assumption was made that below this tail grade, the recovery would be zero. The lower limits used for the DFS recoveries were:

- Copper 0.25% Cu
- Nickel 0.101% Ni

It was also assumed that if the copper recovery was zero, the platinum, palladium, gold and cobalt recoveries would also be zero. While practically this would not be the case, with little information to define the recoveries for these elements at the low levels AGP believed this to be a reasonable approach to examine sensitivity of the model to this parameter.

The DFS recoveries used have been shown in the Table 16-1.

Table 16-1 DFS Recoveries and Fixed Tail Grades

Grade Element	DFS Recovery (%)	Fixed Tail Grade (%)	DFS Update Recovery (%)
Copper	92.33	0.025	Variable
Nickel	70.34	0.030	Variable
Platinum	72.69		72.69
Palladium	75.24		75.24
Gold	67.04		67.04
Cobalt	40.75		40.75

The recovery for copper in each block was completed with the logic shown in Table 16-2.

Table 16-2 Recovery Calculation for Copper and Nickel

Grade Element	Recovery %	Formula
Copper		
Copper % < 0.025%	0%	RCu = 0%, RPt, RPd, RAu, RCo = 0%
0.025% < Copper % < 0.25%	variable	RCu = ((Cu% - 0.025)/Cu%) x 100
Copper % > 0.25%	92.33%	RCu = 92.33%

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Grade Element	Recovery %	Formula
Nickel		
Nickel% < 0.03%	0%	RNi = 0%
0.03% < Nickel% < 0.101%	variable	RNi = ((Ni% - 0.03)/Ni%) x 100
Nickel% > 0.101%	70.34%	RNi = 70.34%

The recovery items in the model are:

- RCu = Copper recovery
- RNi = Nickel recovery
- RPt = Platinum recovery
- RPd = Palladium recovery
- RAu = Gold recovery
- RCo = Cobalt recovery

The calculated recoveries were used in the economic pit determination.

16.4 Economic Pit Development

In the determination of the economic pits, various items were required. These included:

- Metal prices
- Mining cost
- Milling cost
- General and Administrative costs
- Geotechnical parameters

Metal prices for use in the design of the economic pit shells were based upon the DFS values. A second price regime was examined to determine the benefit a slight change in metal price would have on the overall pit size.

The three-year average price was examined for the period of October 12th, 2004 to October 12, 2007 for comparison to the DFS values. Those values have been illustrated in Table 16-3 with the other two price regimes.

Both of the metal price scenarios were below the current 3-year average prices highlighting the conservative approach taken to the long-term mine development.

Table 16-3 Metal Price Comparison

	Units	3 Year Average Metal Prices	DFS Metal Prices	Economic Case Metal Price

	Units	3 Year Average Metal Prices	DFS Metal Prices	Economic Case Metal Price
Copper	$/pound	2.52	1.25	1.50
Nickel	$/pound	11.01	5.60	6.50
Platinum	$/ounce	1,076	800	900
Palladium	$/ounce	283	210	225
Gold	$/ounce	555	400	450
Cobalt	$/pound	19.21	15.25	15.25

Table 16-4 shows the metal prices used and the realized values for economic pit determination. The realization values were provided by PolyMet based on the work completed previously for the DFS and represent the net metal price with consideration for transportation, treatment and refining.

Updated cost estimates since the completion of the DFS design have allowed a refining of the mining cost. This included fuel and electricity prices as well as equipment operating cost estimates obtained from vendors. An examination of the processing, general and administrative and rail haulage costs was also completed. These have been compared to the DFS values in Table 16-5.

Table 16-4 Economic Pit Shell Metal Prices and Realized Value

	Units	DFS Metal Prices	Realization Value	Net Price
Copper	$/pound	1.25	0%	1.25
Nickel	$/pound	5.60	25%	4.20
Platinum	$/ounce	800	18.00 $/ounce	782
Palladium	$/ounce	210	17.00 $/ounce	193
Gold	$/ounce	400	9.50 $/ounce	390.50
Cobalt	$/pound	15.25	40%	9.15

Table 16-5 Updated Pit Optimization Costs

Cost Item	Units	DFS	Updated DFS
Mining	$/ton	$1.30	$1.01
Incremental Haulage	$/ton/20 foot bench	$0.02	$0.00
Rail Haulage	$/ton ore	$0.25	$0.16
Processing	$/ton ore	$5.96	$6.97
General & Administrative	$/ton ore	$1.62	$0.51

The total mining cost of $1.01 per ton mined was based on the average cost over the life of the mine. This balances mining at the lower depths with mining at higher elevations as phases would be depleted and new phases initiated. The lower cost was also developed using 240 ton trucks matched to 29 cubic yard hydraulic shovels versus the smaller sized fleet that had been proposed for the DFS.

The total for rail haulage, processing and G&A in the DFS was $7.83 per ton of ore. PolyMet provided the updated costs for rail haulage, processing and G&A which were $7.64 per ton of ore.

A review of the previous design indicated that an overall angle of 48 degrees was suitable for use in the economic pit development as it mimicked the final DFS design with ramps included.

A series of economic pits were developed to examine the impact of:

- Metal Prices
- Recoveries

The economic pit shell routine used in Minesight© incorporated a Lerch-Grossman routine. The first set of economic pits utilized DFS costs with both fixed and variable recovery. The next set used the Economic Case metal prices for both fixed and variable recovery. In both sets, the variable recovery resulted in an ore tonnage reduction when compared to the fixed recovery for the same metal price scenario. For the DFS price case, this was a 20% reduction while the Economic case was a 24% reduction. The Economic case with its higher metal prices included additional lower grade material from a lowering of the internal cutoff versus the DFS price case. This resulted in a greater influence of the lower recoveries for the low grade ore. Further testing of the recovery at low grades would be required prior to development.

The results of that analysis have been included in Table 16-6 and depicted in Figure 16-1.

Table 16-6 Economic Pit Shell Results

		DFS Prices		Economic Case Prices	
Item	Units	Fixed Recovery	Variable Recovery	Fixed Recovery	Variable Recovery
Ore	tons (millions)	461.2	384.1	570.8	460.3
Copper	%	0.29	0.32	0.27	0.30
Nickel	%	0.08	0.09	0.08	0.09
Platinum	ppb	77	82	70	77
Palladium	ppb	270	291	245	272
Gold	ppb	39	41	36	39
Cobalt	ppm	74	76	73	74
Waste	tons (millions)	1,039.0	1,023.3	1,130.4	1,119.1
Total	tons (millions)	1,500.2	1,407.4	1,701.2	1,579.4
Strip Ratio		2.25	2.66	1.98	2.43

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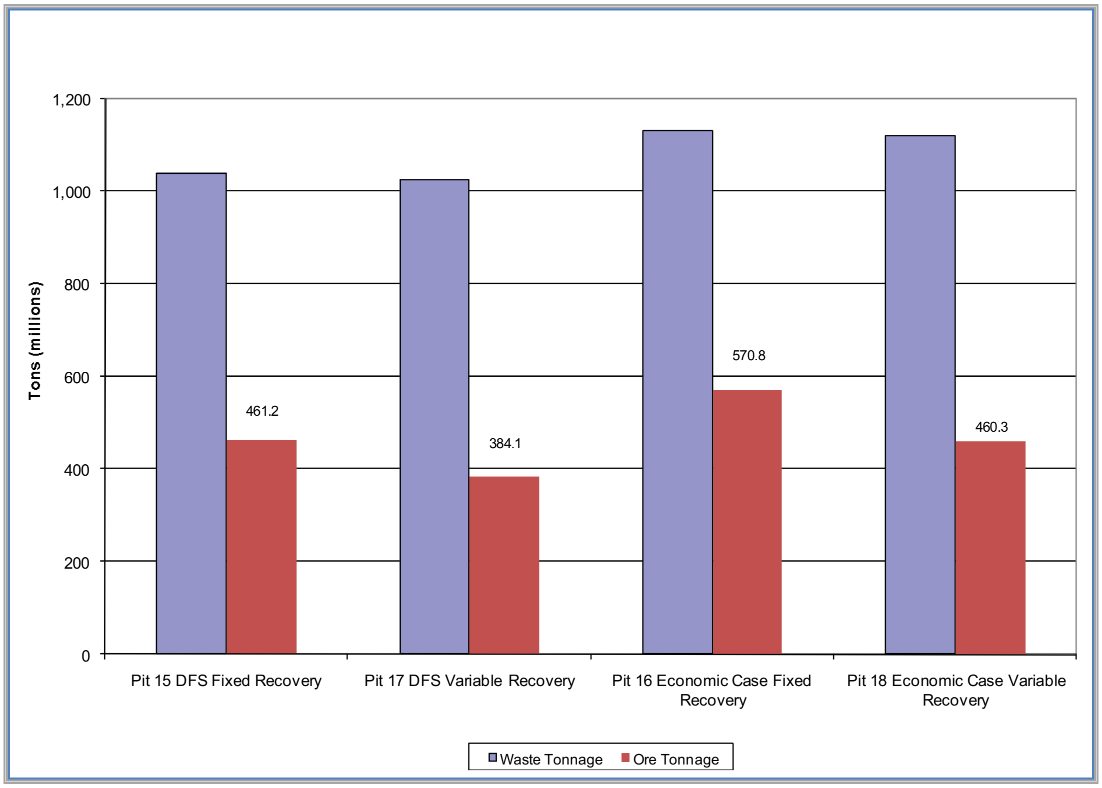
Figure 16-1: Economic Pit Shell Comparison



The economic pit which considered the DFS metal prices and variable recovery was used for the updated NorthMet pit design. This was designated Pit 17 based on the iteration that was examined. This represented a conservative approach to the determination of the economic pit with the inclusion of the variable recovery and the DFS prices.

The economic pit represented the ultimate pit shape. Phasing was required to optimize the mining sequence for production purposes and waste stockpile management. Additional pit optimizations were completed that considered a reduction in metal prices relative to the Base Case pit (Pit 17). The reductions ranged from:

- -10%
- -12%
- -14%
- -16%
- -18%
- -20%

These price reductions were applied to all the metals not just copper and nickel. The realized metal prices used have been shown in Table 16-7. Based on the analysis, shell 26 (-18%) mimicked the DFS pit and was chosen for use in the development of the final design.

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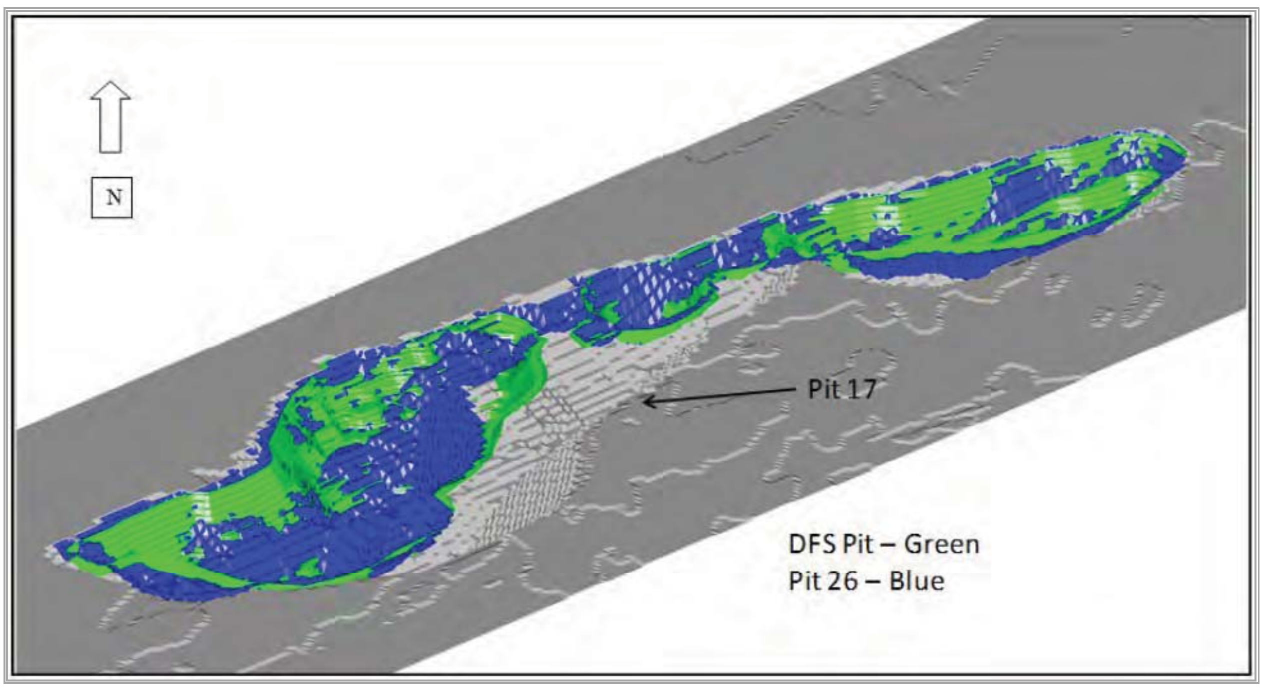
Table 16-7 Realized Metal Price Values

	Copper $/pound	Nickel $/pound	Platinum $/ounce	Palladium $/ounce	Gold $/ounce	Cobalt $/pound
Base Case	1.25	4.20	782.00	193.00	390.50	9.15
-10%	1.13	3.78	703.80	173.70	351.45	8.24
-12%	1.10	3.70	688.16	169.84	343.64	8.05
-14%	1.08	3.61	672.52	165.98	335.83	7.87
-16%	1.05	3.53	656.88	162.12	328.02	7.69
-18%	1.03	3.44	641.24	158.26	320.21	7.50
-20%	1.00	3.36	625.60	154.40	312.40	7.32

This indicated that the DFS pit was well within the metal price regime chosen with much lower prices than used in the Base Case design. This shell has been shown in Figure 16-2 for comparison with the Base Case (Pit 17) as the base topography. There were two distinct lobes mined in the smaller configuration; an eastern and western side. This same arrangement was implemented in the final design. Pit 26 extends further to the west on the western side while it is slightly smaller on the north east side of the western area. This concept was incorporated into the final design.

Figure 16-2: Comparison of DFS Pit against Pit 26 (-18% metal price) with Base Case (Pit 17)



16.5 Final Pit Design

The final design, for the purpose of this technical report, took Pit 26 and broke it into three areas and several phases within each area. The areas were east, west and middle. This was based on economic pit development with prices reduced by 30, 35 and 40 percent. In this way, the most economic material was highlighted and was targeted with an earlier phase. The eastern side had five discrete phases developed, while both the east and west had three phases each. These are shown in Figure 16-3 in an abbreviated form. The values ending in"E" refer to eastern pits, in "W" are west and "M" are the middle pits.

As previously stated, detailed mine planning, waste characterization and waste handling are being updated for the Supplemental Draft EIS.

The eleven phases were developed following the Golder recommendations of:

- Inter-ramp angle = 51.4 degrees
- Bench face angle = 65 degrees
- Safety bench width = 35 feet
- Safety bench interval (vertical) = 100 feet

Mining in the pit was designed for 240 ton trucks. A road width of 122 feet was required to allow 3.5 times the truck width plus berm and ditches. All ramp gradients were at 8%.

Figure 16-3: Mine Areas and Phases

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Reserves for each of the mining phases was calculated and tabulated. The cutoff used was based on a net value calculation. Each block was assigned a mining cost, processing cost, general and administrative cost and revenue. These were then calculated on a block by block basis with the following logic:

- Value per block = Revenue – Mining cost – Processing - G&A cost
- Value per ton = Value per block / block tonnage
- Revenue = grade item recovery x element grade x realized price x block tons
- Mining Cost = block tons x mining cost
- Processing Cost = block tons x processing cost
- G&A Cost = block tons x G&A cost

The cutoff for the reserves was based on the value per ton being greater than zero dollars:

- Cutoff Value per ton > $0.00

The result was a net value per ton mined or net smelter value. The average net value by phase has been shown in Figure 16-4 and Table 16-8.

Figure 16-4: Phase Net Value per Ton

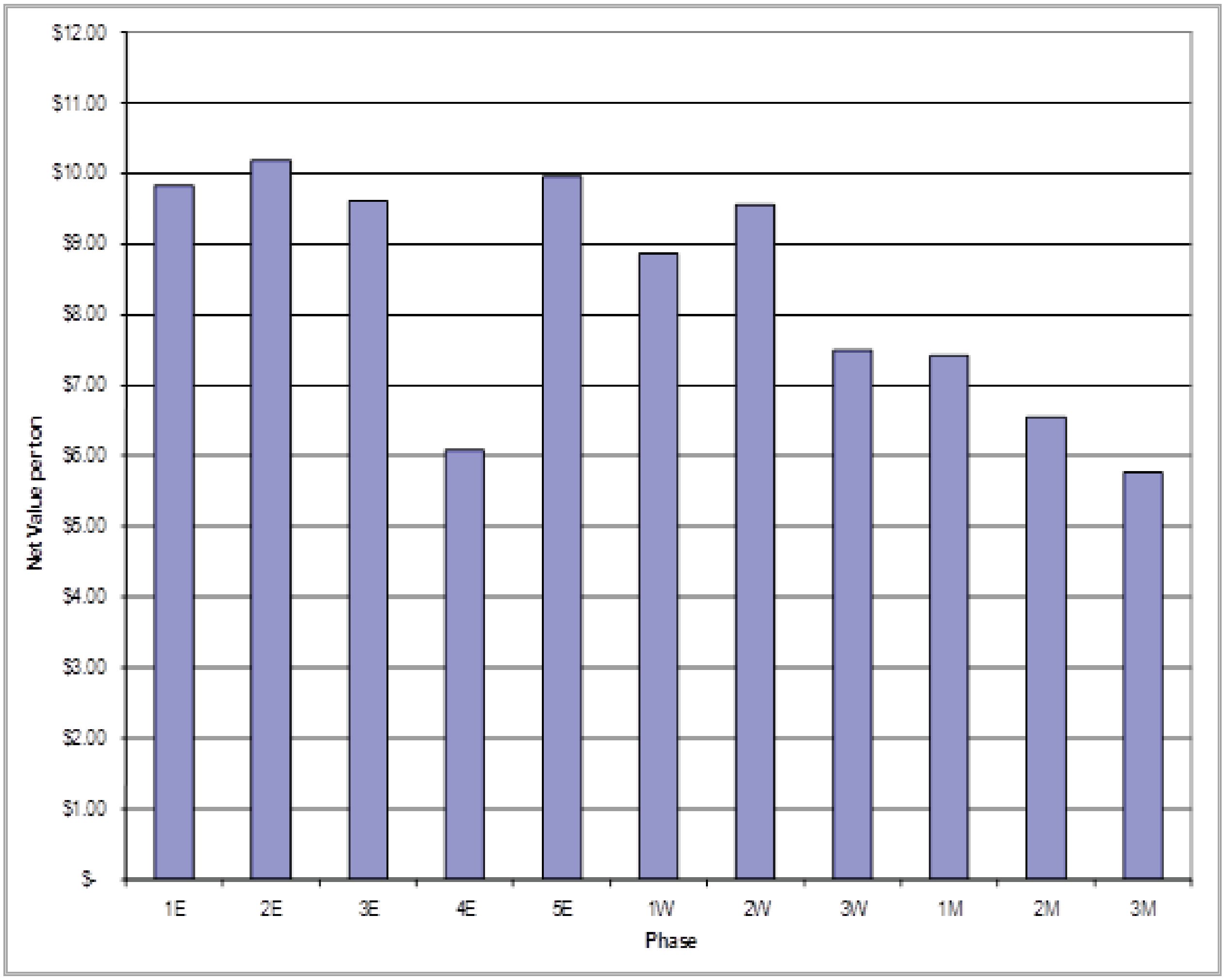

Table 16-8: Net Value per Phase and Area

	East					West			Middle		
Phase	1E	2E	3E	4E	5E	1W	2W	3W	1M	2M	3M
$/ton	9.83	10.18	9.61	6.09	9.97	8.86	9.56	7.50	7.43	6.55	5.77
Average	$9.27/ton					$8.49/ton			$6.50/ton		

This analysis indicated that all the east phases should be mined prior to the west and middle phases, except for phase 4E. Phase 4E extended into the center portion of the deposit where drilling has been limited and the grades lower resulting in a lower net value. This phase would be mined last in any mining sequence to maximize value.

These phases were used in the development of the mine production schedule.

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The metal prices used in the cutoff were the economic model case with the realized values. Fixed recoveries were also used as no laboratory testing at the low grades had been completed at the time of the update. By increasing the metal value, the cutoff dropped which assisted in waste management by directing marginal ore material to the processing plant rather than a lined stockpile. The parameters for the cutoff calculation have been tabulated below in Table 16-9.

Table 16-9: Cutoff Calculation Parameters

	Cutoff Metal Prices	Realization Value	Net Price	Recovery
Copper	$1.50 / pound	0 %	$1.50 / pound	92.33 %
Nickel	$6.50 / pound	25 %	$4.88 / pound	70.34 %
Platinum	$900 / ounce	18.00 $/ounce	$882 / ounce	72.69%
Palladium	$225 / ounce	17.00 $/ounce	$208 / ounce	75.24 %
Gold	$450 / ounce	9.50 $/ounce	$440.50 / ounce	67.04 %
Cobalt	$15.25 / pound	40 %	$9.15 / pound	40.75 %

16.6 Production Schedule

The criteria for the mining schedule provided by PolyMet initially were:

1) 32,000 tons per day mill feed rate

2) 5 million ton limit to low grade stockpile size

These were based on the DFS mine plan reflecting the orginal Environmental Assessment Worksheet and the Draft EIS published in 2009. The key criteria have been honoured in the updated mine schedule.

A difference between the Updated DFS and the DFS mine plan was the development focus for the east pits. The DFS considered a balanced approach to manage strip ratio. The "east side first" approach offered advantages in waste management by allowing backfilling of the eastern pits with waste from the west and middle pits.

While the net value may be lower in some of the west pits than the east, the strip ratio for the initial cut, 1W, is substantially lower than 5E. The practicality of mining and maintaining sufficient feed to the mill required this to be developed prior to the completion of the phase 5E.

Prior to calculating each phase's final resource and developing the production schedule, the dilution grade for each element needed to be determined. Dilution was estimated on a block by block basis rather than as an overall average. In this manner, discrete ore blocks would be properly assessed with higher dilution. Ore blocks surrounded by other ore blocks would not be treated adversely in a grade reduction. For massive deposits such as NorthMet, this approach provided a more realistic estimate of the expected dilution.

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Each block in the model had the following dimensions:

- X = 50 feet
- Y = 50 feet
- Z = 20 feet

Initial estimates for the type of equipment that would be mining the deposit were a 29 cubic yard hydraulic excavator. This class of hydraulic excavator has a bucket 13 feet wide. Considering the hydraulic excavator's capabilities and the nature of the deposit a dilution width of 5 feet was deemed reasonable and applied.

To calculate the dilution for one side of the block being diluted, the following was assumed:

1) Block volume = 50 feet x 50 feet = 2,500 square feet

2) Dilution (one side) = 50 feet x 5 feet = 250 square feet

3) Dilution Percentage = 250 ft2/(250 ft2 + 2,500 ft2) = 250 ft2/(2,750 ft2)= 9.1%

The percentage dilution by the number of diluting sides has been summarized in Table 16-10.

Table 16-10 Dilution Percentages

Block Sides Exposed	Dilution %
0	0.0%
1	9.1%
2	16.7%
3	23.1%
4	28.6%

Each block in the model was queried, and its surrounding blocks examined. The number of below cut-off blocks surrounding each individual block was recorded in the model. This number was then used to determine the dilution percentage in accordance with the calculated values shown in Table 16-11. The appropriate percentage was then stored in the block model.

To determine the grade of the dilution material, the model was queried for the grade of each block below cut-off surrounding the individual block. This information was extracted to an ASCII file then loaded into a drill hole database. This database was used to interpolate the dilution grade based on an inverse distance relationship. The grade of the diluting material was then stored in each individual block.

The diluted grade for resource determination was then estimated with the following formula:

- DCu = (100-Dilution %) x Cu% + (Dilution % x CuWst)

Where:

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- DCu = diluted copper grade
- Dilution % = dilution percentage
- Cu% = undiluted copper grade
- CuWst = copper grade of the diluting material

This same methodology was applied to all of the grade items in each block. The total pit design that matches the DFS pit was examined to determine the diluted and undiluted grades for each element. The variation in the final grade has been shown in Table 16-11.

The overall dilution percentage was very low which was expected due to the massive nature of the NorthMet Deposit.

It was assumed that the ore loss was equal to the dilution as the value was similar to what would be expected for ore loss. This resulted in no increase in tonnage, but a grade reduction of approximately 2.5%, depending on the element considered.

Table 16-11 Diluted Grade Comparison

	Copper (%)	Nickel (%)	Platinum (ppb)	Palladium (ppb)	Gold (ppb)	Cobalt (ppm)
Undiluted Grade	0.280	0.082	76.5	262.2	37.9	73.8
Diluted Grade	0.274	0.080	74.7	255.6	37.1	73.2
Dilution %	2.2	2.5	2.4	2.6	2.3	0.8

Waste management at NorthMet was considered critical to the overall effectiveness of the project. Waste categorization was based on criteria that had been established in the DFS study. Those criteria were not changed for this update.

Overburden material was modelled to be all material that was beneath the topography surface, but overlain on the bedrock. This material was tracked separately as it was not required to be stockpiled in a lined facility.

The rock waste classification was based on the sulphur percentage and the copper sulphur ratios. The waste categorization followed the criteria outlined in Table 16-12.

Table 16-12 Waste Categorization

Waste Category	Element Criteria
Category 1	Block value < 0.12% S
Category 2	Block value < 0.12% S or 0.12% S < Block value < 0.31% S with Cu/S ratio < 0.3
Category 3 (Lean Ore)	0.12% S < Block value < 1.0% S with Cu/S ratio > 0.3
Category 3	0.31% S < Block value < 1.0% S with Cu/S < 0.3

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Waste Category	Element Criteria
Category 4	Block Value > 1.0% S or Virginia Formation rock (including Virginia Formation floaters)

These categories were used in the final scheduling of the material. In the case of the Category 3 (Lean Ore) material, this was combined with the Category 3 waste for scheduling purposes.

Resources for each phase were once again estimated with a cutoff value of $0 per ton and resulting diluted grades output. Economic case realized metal prices were used with fixed recoveries as per the DFS. The waste tonnage by category type was also output at the same time. These resources were stored in the mine scheduling spreadsheet and from this the production schedule developed.

The impact of elevating the cutoff grade and stockpiling material in the initial years was considered. An examination of the cutoff grade indicated that a cutoff of $7.00 per ton or greater yielded a reasonable increase in feed grade while being able to maintain the stockpile level at 5 million tons. Resources were output with a cutoff bin of $0 per ton and $7.00 per ton. The higher grade material was to be processed first while the lower grade material stockpiled until later in the mining sequence.

Eleven phases were designed to fit within the footprint of the previous DFS pit. Total resources for those phases have been shown in Table 16-13.

The waste material classification was reinterpreted from the drilling that was completed to further define the magenta zone. This resulted in the Category 3 and 4 waste tonnage totals exceeding previously established limits from the EIS. For this reason, PolyMet opted to reduce the size of the pit mined to ensure the stated tonnage limits were not exceeded for the current mine permitting process. This was accomplished by excluding Phases 3W and 3M.

Table 16-13: Updated DFS Pit Tonnages

	Tons	Diluted Grades					
		Copper (%)	Nickel (%)	Platinum (ppb)	Palladium (ppb)	Gold (ppb)	Cobalt (ppm)
Ore	306,252,000	0.27	0.08	74.8	256.0	37.1	73.3
Waste							
Overburden	23,420,000						
Category 1&2	297,828,000						
Category 3	113,783,000						
Category 4	12,550,000						
Total Waste	447,580,000						

The resulting tonnage of ore and waste plus their associated grades have been tabulated in Table 16-14. These tonnages were used for the final production schedule in determining an updated operating cost, equipment requirements and capital cost.

Table 16-14: Final Updated DFS Pit Tonnages

	Tons	Diluted Grades					
		Copper (%)	Nickel (%)	Platinum (ppb)	Palladium (ppb)	Gold (ppb)	Cobalt (ppm)
Ore	223,956,000	0.28	0.08	75.1	264.5	37.6	73.3
Waste							
Overburden	18,520,000						
Category 1&2	210,151,000						
Category 3	64,789,000						
Category 4	10,350,000						
Total Waste	303,810,000						

While this tonnage is less than the reported reserve, the larger pit design indicates that with the updated model, more material is possible to convert to reserve. AGP has opted to maintain the existing reserve base and the costing exercise was optimized for a reduced mine tonnage.

At a plant production rate of 32,000 tons per day, 11.67 million tons of ore was required annually. The ore processing ramp up schedule was over a one year period from the start of the concentrator. This resulted in the first year achieving only 8.76 million tons processed. The ore reserves contained within the modified pit design allowed for 20 years of ore processing. The ore tonnage by phase resulting from this schedule has been shown in Figure 16-5. The plant feed grades for copper and nickel has been shown in Figure 16-6. A significant dip in the feed grade occurred in Years 15, 16 and 17 that resulted from the timing of the various phases and location of material. Attempts at smoothing that grade release during this time were made but were unsuccessful.

Only one year of pre-production mining was required to provide sufficient ore material for the processing plant. The ore was located very near surface, which allowed for rapid preparation of ore inventory in the phases. Waste mining was focused on developing the eastern pits first to allow the backfilling of those pits earlier. The result of this action was a significant waste mining requirement in the first five years upfront due to the higher strip ratio of later phases of the eastern pit. From Year 6 to Year 12 the waste mining requirements were more stable, then climb to mine the last phases at a higher strip ratio. The waste mining requirement by phase has been shown graphically in Figure 16-7. Total material mined annually has also been illustrated in Figure 16-8.

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Figure 16-5 Production Schedule Ore Tonnage by Phase

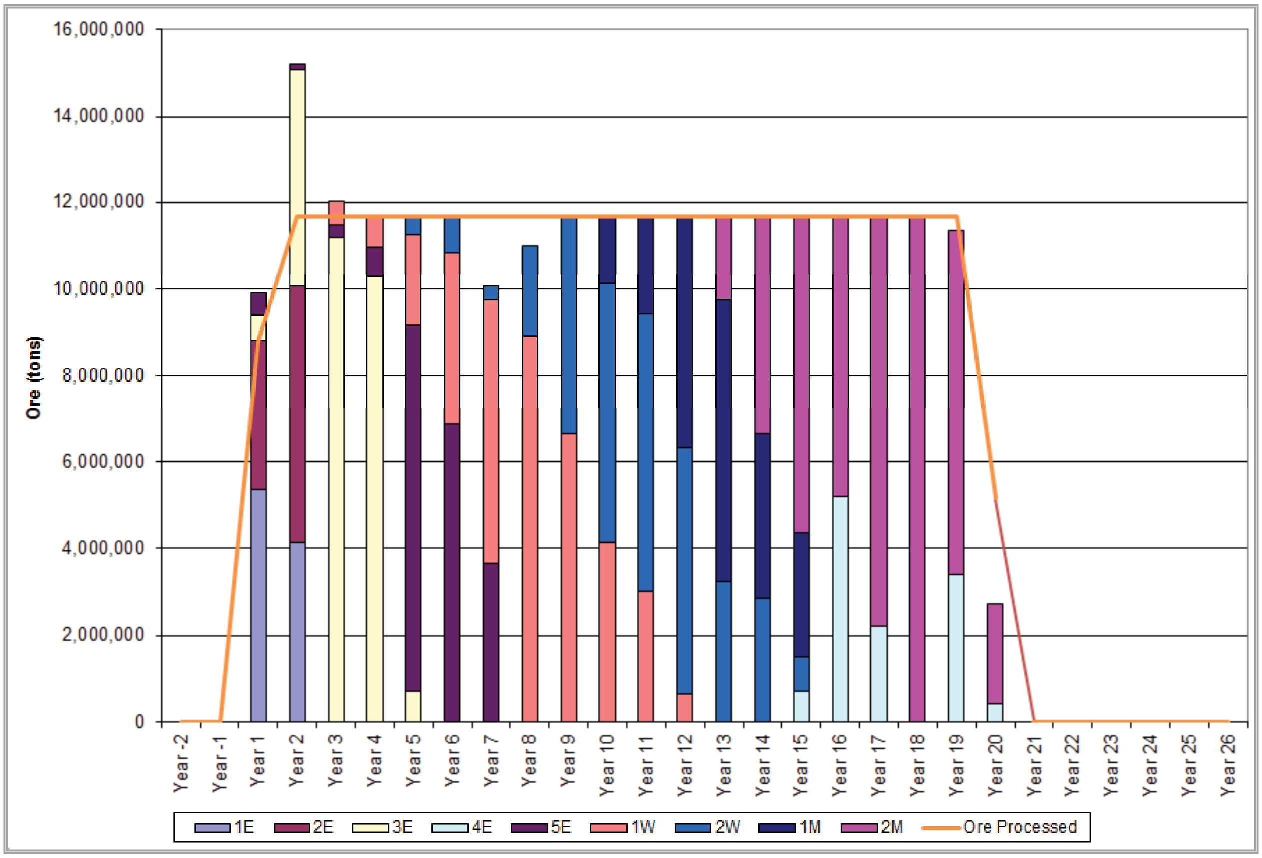

Figure 16-6 Plant Feed Grades – Copper and Nickel

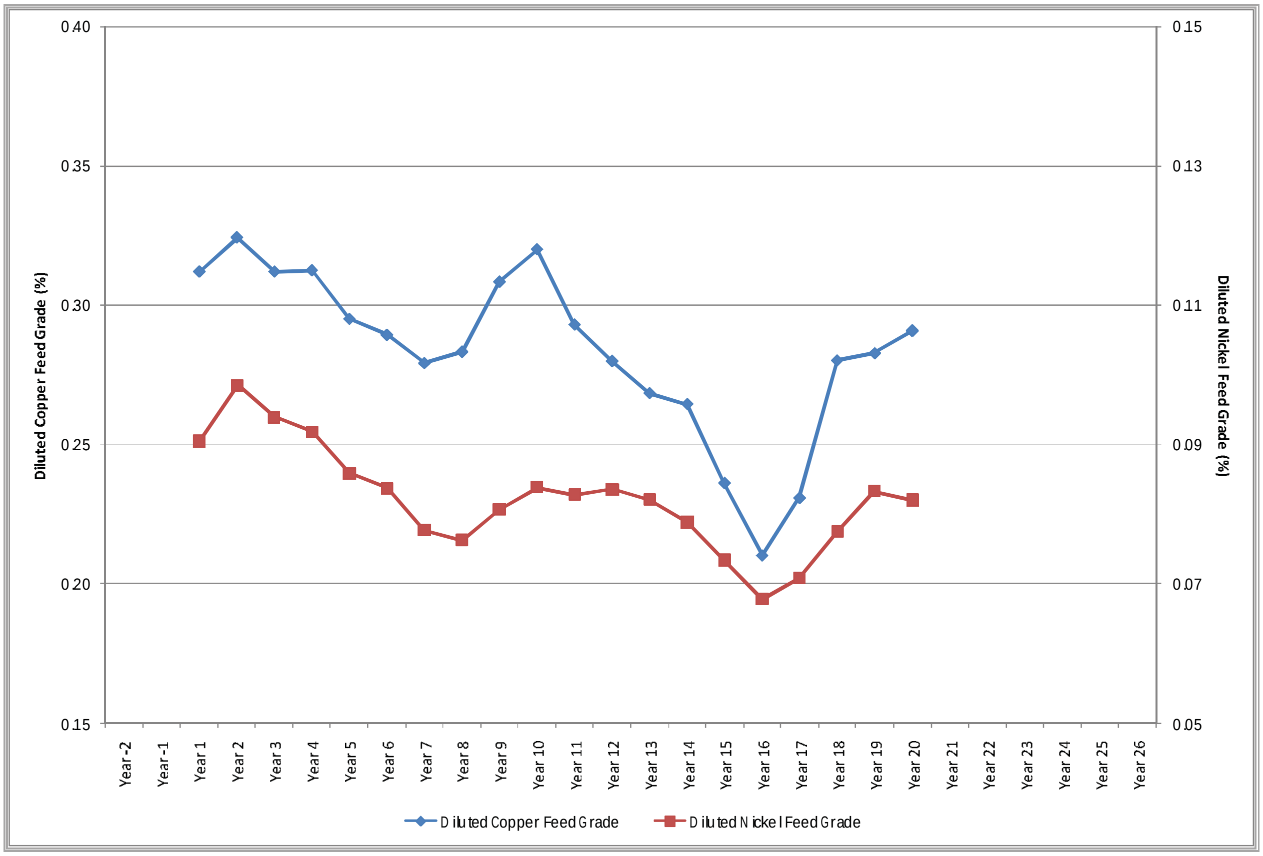

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Figure 16-7 Waste Tonnage By Phase

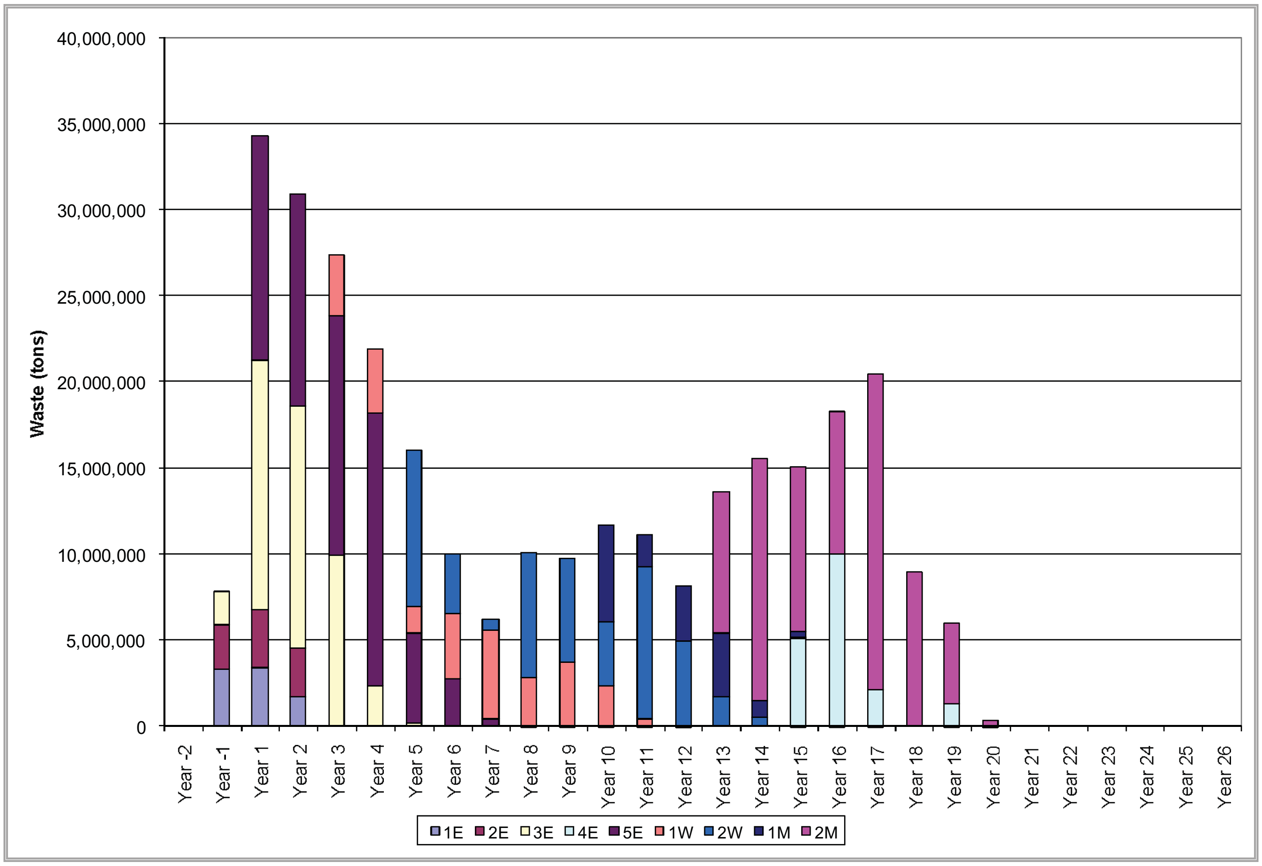

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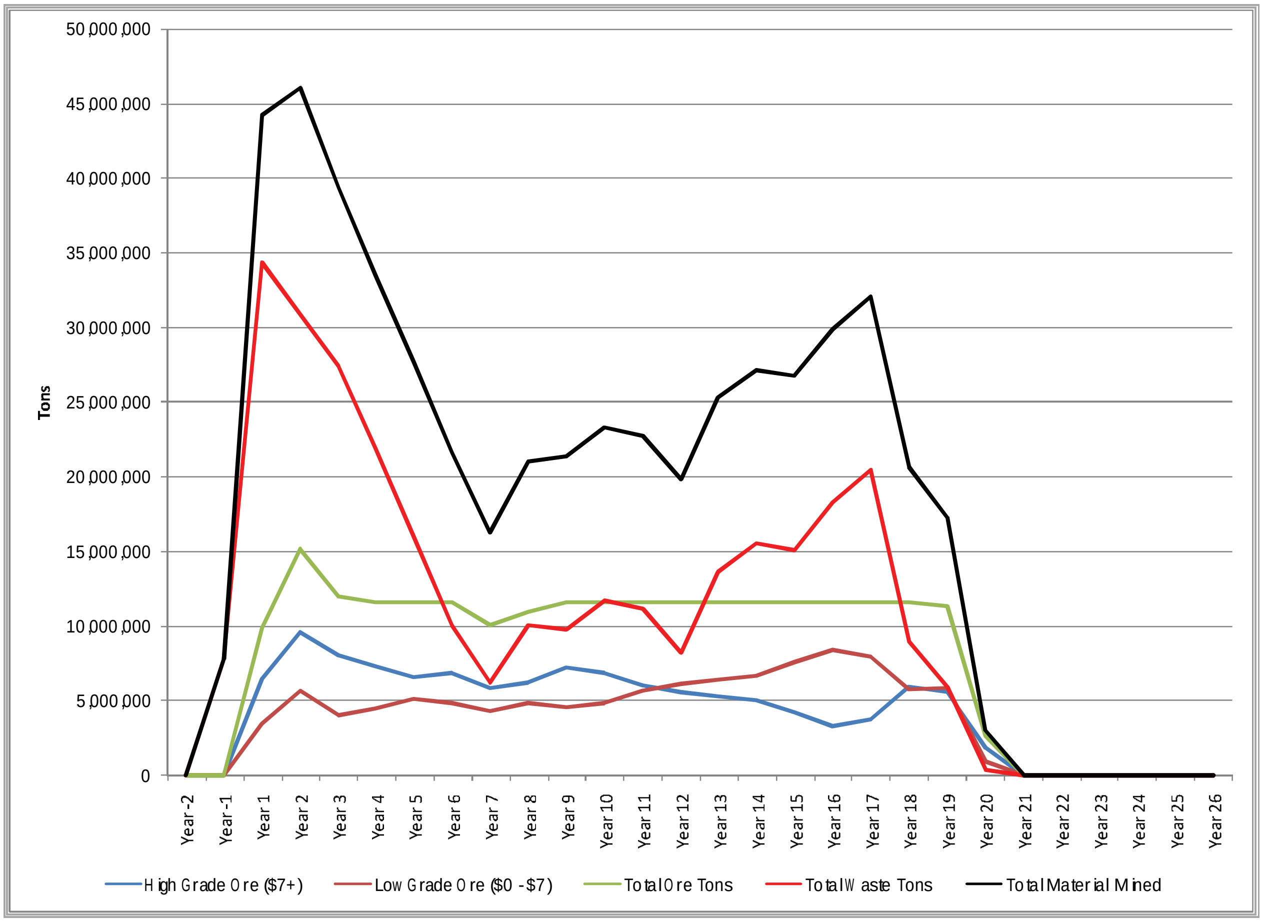
Figure 16-8 Total Material Movement by Year



16.7 Waste Management

This section describes waste management plans at October 15, 2007 as modified for the 2008 DFS Update. Plans being incorporated into the Supplemental Draft EIS that will be incorporated in a new Technical Report once all the details have been finalized include backfilling some waste into the East Pit once it has been mined out in year 11 of operations. A portion of the newmly mined waste will be taken directly to the East Pit, while selected waste material from the first eleven years (stored in stockpile) will be rehandled and moved to the East Pit for final placement.

Minnesota design criteria were used in the slope configuration for the stockpiles as per the DFS. These were:

- Face Slope = 22 degrees (1 Vertical: 2.5 Horizontal)

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- Lift Height = 40 feet
- Berm Separation = 40 feet vertically
- Berm Width = 30 feet

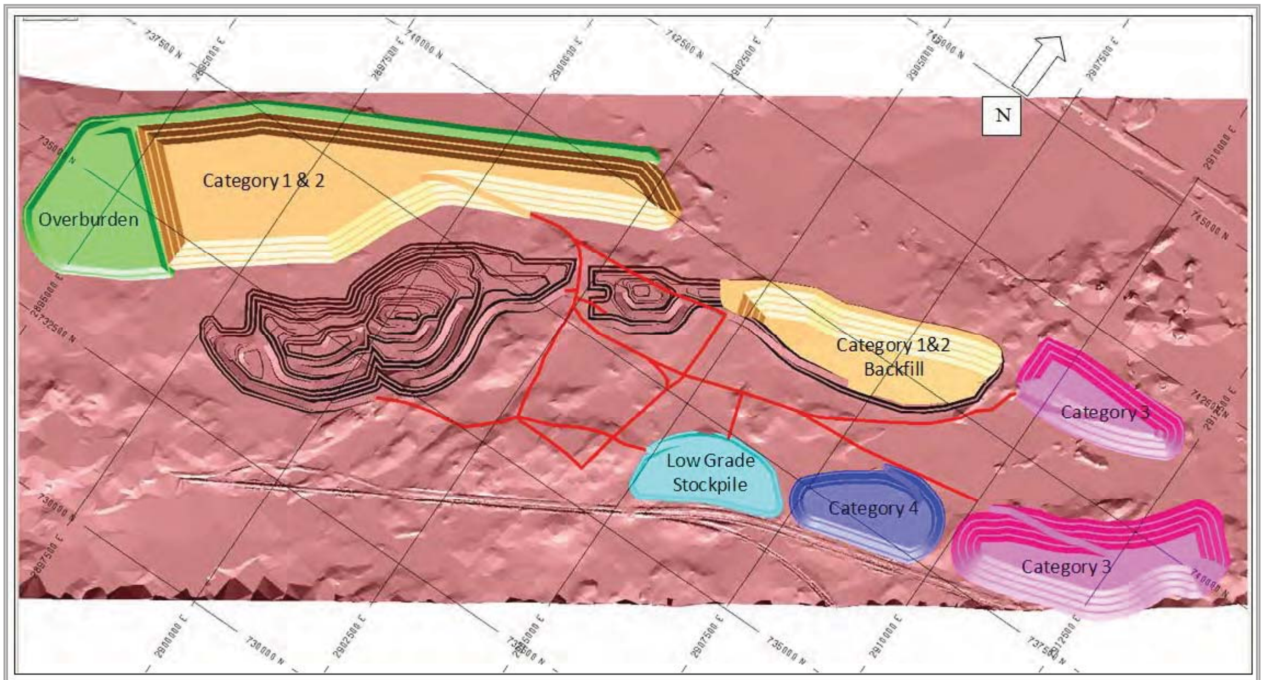
The location, footprint and height of the dumps set in the DFS and in the preparation of the 2009 Draft EIS were used as outlines for the DFS Update, but have subsequently been altered to include East Pit backfill, the elimination of some dumps, and redesign of the remaining permanent dump.

In the DFS Update, overburden material was the first material mined and stockpiled in the northwest corner of the property, on the footwall side of the NorthMet Deposit. A finger dyke was built at the 1640 level along the northwest edge extending to the south in a shape similar to a golf club. Once that design had been completed, additional material was placed on top of the southern end until required for reclamation purposes. A total of 10.3 million loose cubic yards of storage were required and designed. The design for the overburden stockpile has been shown in Figure 16-9..

Material was stockpiled in the northwest area adjacent to the overburden stockpile until Year 8. The overburden toe dyke assisted in controlling drainage from this stockpile by containing and redirecting to the water control systems present on the property. A total of 61.2 million loose cubic yards of material were stored in this stockpile.

East pit mining in this study had been accelerated to permit backfilling of waste material from the western and middle sections of the NorthMet Deposit. This was intended to allow selected waste rock to be placed in the mined out east pits as shown in Figure 16-9 and stored sub-aqueously after Year 8.

Figure 16-9 Waste Stockpile Locations with Final Pit Outline – DFS Update not SDEIS Plan



Backfilling in the east pit was not available until Year 8, after phase 5E had been completed. For this evaluation, backfilling of the east pit followed this protocol:

1) Material would be backfilled first along the footwall in a 140 foot wide finger dump at the 1590 elevation,

2) Backfill of the remainder of the east pits would be from the bottom up,

3) Water would be allowed to rise and stabilize at the predicted level of 1592 elevation.

Selective rock types were to be used in the backfill and, under the DFS Update plan, the entire east pit was not backfilled. Subsequent changes to the plan that will be incorporated in the Supplemental Draft EIS include filling the East Pit to create a wetland environment.

All of the material mined in the current production schedule was stored in the stockpiles as outlined in Figure 16-9.

16.8 Mine Operations and Equipment

Mining of the NorthMet Deposit and the economic pit cost parameters were based on bulk mining methods 365 days per year. The waste rock and ore would be drilled, blasted, loaded and hauled with conventional drills, trucks and hydraulic shovels.

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Ore will be hauled directly to the truck dump/feeder facility for loading on the rail cars. Stockpile ore would be placed in a lined storage area to the east of the truck dump. The direct ship ore would be hauled to the NorthMet processing facility and discharged above the primary crusher. Rehandle of the stockpile ore would be accomplished by a loader and truck hauling to the truck dump/feeder from the stockpile or direct load into railcars.

Waste material will be categorized from the drill and blast results and modelling. This would then be dispatched at the shovel face to the appropriate stockpile location. As much as possible, reclamation of the waste stockpiles will occur concurrently with the mining. Overburden material will be reclaimed from the stockpiles (if no direct ship material was available) and placed on the dump. This will allow for revegetation of the waste stockpile current with mining.

The original DFS had envisaged the use of 100 ton trucks, but with the scale of the deposit, much larger trucks were considered. A review was made of various fleet configurations that varied truck capacity and shovel types. The result of that analysis indicated that the 240 ton truck configuration matched with a hydraulic shovel provided the most cost effective method of developing the NorthMet Deposit.

The hydraulic shovel offered several benefits to the particular needs of NorthMet. Selective mining of the ore was planned to be achieved by mining ore on a 20 foot bench while bulk mining waste on a 40 foot bench. Hydraulic shovels, due to their unique operating configuration, were better suited to this task without sacrificing productivity. Cable shovels require a higher bench face to be consistently more productive.

In the current mining environment of equipment supplier shortages, cable shovels were difficult to obtain in a shorter time frame at a reasonable cost. New hydraulic shovels were available in one year while cable shovels were at least two years. From a cost perspective, cable shovels were approximately 2.5 times the cost of a comparable sized hydraulic shovel. For these reasons, the economics favoured the use of hydraulic shovels at NorthMet.

To meet the production needs for the NorthMet project, it was anticipated that two 31 cubic yard hydraulic excavators would be required with a 21.5 cubic yard front end loader as backup. The hydraulic shovels would be in an electric powered configuration for operating cost reasons and also reduction of site emissions.

Typical blasting in the Iron Range of Minnesota has been with a 16” diameter holes. Golder completed an evaluation of the rock at NorthMet to determine what would be correct. Utilizing the KuzRam model, Golder recommended a smaller bit diameter of 12 ¼”. This recommendation was examined in detail with updated drill operating costs and the recommendation remained to use a 12 ¼” diameter borehole. The drills specified though were capable of drilling the more locally common 16” diameter hole, should it prove more cost effective once mining progressed.

PolyMet had already purchased a used electric drill with the capability that was going to be refurbished prior to mining commencing. A second drill of comparable size was considered but in a diesel configuration to provide flexibility with multiple phases. This would allow for rapid drill deployment.

Support equipment included tracked and rubber-tired dozers with graders and small front end loaders. Two large and two small water trucks were envisaged for use to control dust and water the drills for dust suppression. A large rubber tired dozer was included in the fleet of dozers. This was to provide flexibility either at the shovel face or on the dumps without the excessive travel time concerns raised with conventional track dozers. A smaller track dozer was planned for use to manage the tailings facility.

Stockpile turnover rehandle annually was estimated to be in the order of 320,000 tons per year. This was based on the assumption that 5 weather days would affect the pit that would require ore to be loaded from the stockpile and replenished. An additional 5 days of maintenance for the feeder was planned which required direct loading of ore from the stockpile to the train. This was also included in the 320,000 tons per year requirement.

16.9 Reserves in DFS Update Plan

A portion of the total reserves outlined in Section 15 are to be mined. That portion in the current plan has been shown in Table 16-15 and Table 16-12.

Table 16-15 Diluted Mineral Reserves in DFS Update Plan

Category	Tons	Copper (%)	Nickel (%)	Platinum (ppb)	Palladium (ppb)	Gold (ppb)	Cobalt (ppm)
Proven	116,430,500	0.30	0.09	77	279	39	74
Probable	107,548,000	0.27	0.08	73	249	37	73
Total	223,978,500	0.28	0.08	75	265	38	73

Table 16-16 Diluted Mineral Reserves by Category and Phase in DFS Update Plan

Category	Tons	Copper (%)	Nickel (%)	Platinum (ppb)	Palladium (ppb)	Gold (ppb)	Cobalt (ppm)
Proven							
1E	8,271,800	0.31	0.09	74	293	37	69
2E	5,820,100	0.31	0.09	75	316	37	76
3E	22,085,400	0.31	0.09	67	279	37	77
4E	5,880,400	0.22	0.08	71	272	35	75
5E	17,720,300	0.32	0.09	69	303	36	72
1W	16,321,500	0.29	0.08	96	286	48	73
2W	13,952,300	0.32	0.09	97	289	46	77
1M	11,827,100	0.29	0.08	68	249	35	74
2M	14,551,600	0.28	0.08	70	240	35	72
Sub-total	116,430,500	0.30	0.09	77	280	39	74
Probable							

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Category	Tons	Copper (%)	Nickel (%)	Platinum (ppb)	Palladium (ppb)	Gold (ppb)	Cobalt (ppm)
1E	1,222,400	0.27	0.08	66	245	31	66
2E	3,547,000	0.30	0.09	77	325	38	72
3E	5,711,900	0.28	0.08	63	265	35	73
4E	6,075,900	0.21	0.07	65	285	35	69
5E	2,860,400	0.27	0.08	58	250	30	73
1W	20,439,500	0.30	0.08	82	258	41	77
2W	19,673,400	0.31	0.08	96	282	44	76
1M	10,494,800	0.24	0.08	60	212	31	72
2M	37,522,700	0.24	0.07	65	223	33	69
Sub-total	107,548,000	0.27	0.08	73	249	37	73
Total	223,978,500	0.28	0.08	75	265	38	73

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17 RECOVERY METHODS

The 2006 Technical Report described in detail the recovery methods contained in the DFS. Since then, PolyMet has simplified the proposed metallurgical process that will be used to process the ore to recover base metals, gold and PGE metals. Previous plans included two autoclaves and a copper solvent extraction/electro-wining ("SX-EW") circuit to produce copper metal along with value added nickel-cobalt hydroxide and precious metals precipitate products.

PolyMet now plans to build the Project in two phases, comprising:

- Phase I: produce and market concentrates containing copper, nickel, cobalt and precious metals
- Phase II: process the nickel concentrate through a single autoclave, resulting in production and sale of high grade copper concentrate, value added nickel-cobalt hydroxide, and precious metals precipitate products.

The changes reflect continued metallurgical process and other project improvements as well as improved environmental controls that are being incorporated into the Project. The advantages, compared with the earlier plan, include a better return on capital investment, reduced financial risk, lower energy consumption, and reduced waste disposal and emissions at site.

The Process Plant will consist of a Beneficiation Plant and Hydrometallurgical Plant. The processing steps that would be involved in each operation are described below. The Process Plant would also include a Tailings Basin, Hydrometallurgical Residue Facility and a railcar maintenance shop.

17.1 DFS Metallurgical Testwork

The aim of the DFS testwork program was to develop and demonstrate a complete process flowsheet for treatment of polymetallic sulphide material from the NorthMet Deposit with an average head grade of approximately 0.31 % copper, 0.09 % nickel, 0.08 g/t platinum, 0.28 g/t palladium and 0.04 g/t gold. The flowsheet arising from this testwork subsequently served as the basis on which the plant was designed to process 32,000 short tons (29,030 metric tonnes) per day or 11.68 million short tons (10.6 million metric tonnes) per year of run of mine (ROM) ore.

The process route selected for recovering the base metals and AuPGMs is based on the mineralogy and involves an initial concentration step to recover the sulphide minerals and AuPGMs by crushing, grinding and bulk sulphide flotation. The bulk sulphide concentrate is then treated by a hydrometallurgical process that includes chloride-assisted pressure oxidation leaching (POX) with subsequent metal recovery. Copper is recovered as LME grade cathode. Nickel and cobalt are recovered together as a mixed hydroxide precipitate. The gold and PGM are collected in a precipitate with some copper and sulphur. The mixed hydroxide precipitate and gold-PGM precipitate are refined off-site by off-take parties. The advantage of this hydrometallurgical method is that all the base metals and AuPGMs are extracted in a single step (the chloride assisted POX) and can be subsequently separated and recovered onsite.

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There are two waste streams from the ore processing plant. Flotation tailings are pumped directly to a separate storage facility. The hydrometallurgical plant residue is formed by mixing the final POX residue with gypsum, iron/aluminium hydroxide and magnesium hydroxide residues. This combined residue is placed in the lined hydrometallurgical residue facility.

The development and demonstration of this process has taken place via several integrated pilot plant testwork campaigns from as early as 1999. A thorough review of the most recent 2005 and 2006 testwork has been presented in the DFS with findings and conclusions from all pilot campaigns incorporated into the current plant design.

17.1.1 Testwork History

Testwork in 1997 and 1999-2000

PolyMet launched an intensive testwork program in 1998 and 1999-2001 to examine the potential for hydrometallurgical processing of the NorthMet ore. After extensive analysis, flotation of Cu, Ni, and AuPGM to a bulk concentrate followed by a high temperature, chloride-assisted POX approach was selected, and the process was fully demonstrated at the bench scale.

Pilot Plant Campaigns 1999-2001

Pilot plant campaigns were completed in 1999-2001 at Lakefield to produce a bulk concentrate from the NorthMet ore and to investigate the recovery of Cu, Ni, and AuPGMs from the bulk concentrate. The flotation process to produce the bulk concentrate included rougher, scavenger and cleaner unit operations. The final bulk concentrate contained 14.7% Cu, 3.05% Ni, 32.9% Fe, 0.14% Co, 26.7% S, 1.41 g/t Au, 2.22 g/t Pt and 9.9 g/t Pd.

Bulk concentrate was ground to P80 of 15 µm, a fine grind being important for complete extraction of AuPGM, and re-pulped to approximately 10% solids in an agitated vessel prior to injection into a six-compartment autoclave. The autoclave operated at conditions identified in earlier batch scale testwork to be optimum: 225° C, 690 kPa oxygen gas overpressure and 120 minutes residence time. The discharge residue was filtered and the pressure leach solution treated in a number of ways to recover AuPGMs from the PLS. The AuPGM depleted liquor was then stage neutralised to pH 2.0, using limestone, and copper cathode was produced via conventional SX/EW. A portion of the raffinate was bled from the circuit and set aside with the balance of the raffinate recycled as a cooling solution to the autoclave. The main autoclave pilot plant operated successfully for 14 days including a 10-day integrated run with Cu SX raffinate recycled back to the autoclave.

A further pilot plant was used to demonstrated a process for treatment of raffinate that included rejection of Al and Fe, and production of high purity nickel and cobalt metals by a solvent extraction and electrowinning process.

Testwork in 2005-2006

The 2005-2006 pilot plant program was overseen by Bateman and undertaken to confirm the entire metallurgical flowsheet feasibility from ore processing to final product recovery, to provide the design basis for the process plant, to collect extensive environmental data and to optimise aspects of the process, in particular:

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- Increasing sulphide recovery from the ore to the bulk flotation concentrate (to minimise environmental impacts of sulphide in tailings).
- Recycling of a portion of the leach residue to the autoclave for improved AuPGM extraction and autoclave design optimisation (reduced autoclave sizing).
- Precipitant selection and optimisation for iron reduction and AuPGM recovery.
- Investigation of an option to separate Co and Zn via solvent extraction prior to Ni hydroxide precipitation, as an alternative to precipitation of a mixed Ni-Co-Zn hydroxide product.
- The pilot-scale testwork program evaluated continuous and fully integrated testing of the proposed flowsheet in several phases, accompanied by bench scale variability and optimisation testwork:
- Phase 1 – Comminution and Flotation
- Phase 2 – Leaching and Metal Recovery (Cu Cathode, AuPGM Precipitate and Ni-Co Mixed Hydroxide Precipitate) from the Phase 1 Flotation Concentrates
- Phase 3 – Testing of Solvent Extraction and Electrowinning for Cu, Ni and Co and Precipitation for Separate Nickel, Cobalt and Zinc Product Recovery (Hydroxides)
- Phase 4 – Optimization Flotation and Autoclave Bench and Pilot Plant Testing in March–April 2006.

In 2005, a 44 short ton bulk sample of large diameter diamond drill core was delivered to Lakefield for flotation testwork and subsequent production of concentrate for hydrometallurgical pilot plant program. Another nine short tons of drill core sample was provided in April 2006 for additional pilot scale testwork.

17.1.2 Comminution and Flotation Testwork

Each composite was tested separately using optimised comminution and flotation parameters established in previous testwork.

The flotation pilot testwork provided bulk concentrate products for further hydrometallurgical testing.

Comminution Testwork

Comminution parameters were determined for the composites and show a high level of consistency. The ore can be broadly categorised as mildly abrasive and towards the higher end of the hardness scale. A review of the specifications of the existing crushing and grinding equipment has confirmed that it is more than capable of reducing the particle size to suit flotation at the required throughput.

Average values determined from Bond tests for the rod and ball mill work indices were 13.4 and 15.5 kWh/t respectively, and 0.40 for the abrasion index.

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17.1.3 Flotation Laboratory Batch and Locked Cycle Testwork Outcomes

This work mimicked the flowsheet derived from past testing and confirmed that the metallurgical behaviour of all composites was consistent. The flowsheet adopted a standard rougher scavenger circuit followed by two stages of cleaning. A regrind mill on the combined scavenger concentrate and first cleaner tailing was also included to ensure middling particles (particles containing both sulphides and gangue) underwent further size reduction.

The testwork confirmed optimum flotation parameters for maximum sulphide and base metals recovery to the concentrate, and determined:

- a reagent regime, including:
- flotation collector (potassium amyl-xanthate, PAX) dosage rate;
- copper sulphate addition as an activator to enhance metal and sulphide recovery; and
- combined frother of 3:1 MIBC:DF250.
- a selected grind size of 125 µm for flotation pilot ing feed.
- total rougher and scavenger time of 15 minutes.
- flotation pilot plant outcomes

A total of 53 short tons were processed, in four composite groupings. Flotation performance was similar for all composites and circuit changes were introduced to enhance sulphide recovery to concentrate and thus reduce sulphur content of the tailings.

The bench and pilot plant work confirmed the importance of copper sulphate ($CuSO_4$) as an activator for sulphide mineral flotation. The addition of copper sulphate to a conditioning step prior to the scavenger flotation step successfully reduced the sulphur grade of the final tailings to ≤0.15%, thus meeting PolyMet's objective of minimising possible environmental impacts of sulphur in tailings.

Table 17-1 shows flotation circuit performance for non-activated versus activated pilot-scale tests in 2005.

Table 17-1: Impact of Copper Sulphate on Pilot Plant Recovery

Description	Distribution, %					
	Cu	Ni	S	Pt	Pd	Au
Non-activated	94.3	69.3	72.4	69.1	75.8	58.5
Activated	94.2	72.5	82.2	67.5	83.1	57.9

The additional flotation pilot testwork undertaken in 2006 was able to confirm the reagent regime, provide a reduction in overall residence time (and hence circuit size), and attain similar metals recoveries and reduced sulphide in tailings. The additional work also led to refinement of the circuit to include a scavenger conditioning stage and splitting scavenger concentrate (first scavenger

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concentrate directly to the cleaner circuit and the second scavenger concentrate to the regrind mill before returning to the rougher circuit) to reduce the solids loading in the regrind circuit.

The range of grade for concentrates produced in the 2006 testwork is shown below in Table 17-2.

Table 17-2: Concentrate Composition from 2006 Piloting

Grade	Assays					
	Cu (%)	Ni (%)	S (%)	Au (g/t)	Pt (g/t)	Pd (g/t)
Concentrate	7.16-10.1	1.66-2.20	18.4-21.5	0.65-1.28	1.17-1.59	5.76-6.71

Flotation pilot testing covered a range of samples with head grades from 0.27% to 0.41% Cu and from 0.094% to 0.122% Ni. In the grade range tested, flotation recovery did not appear to change with head grade hence a constant flotation recovery was used.

Flotation tailings and concentrate were tested by Outokumpu Technology (solid-liquid separation equipment vendors) in a continuous, high rate thickening rig to determine flocculant and thickening design parameters.

A detailed mineralogical analysis was made on flotation tailings.

17.1.4 Hydrometallurgical Bench Testwork – 2005

Pre-Piloting

Hydrometallurgical pre-piloting bench testwork was conducted to optimise circuit conditions for the pilot-scale testwork, in particular temperature, residence time and reagent additions for a number of unit operations. The results of this testing were then incorporated into the pilot plant design and operating philosophy.

During and Post-Piloting Bench Testwork

A number of bench programs were undertaken to provide important information for final design. These included:

- AuPGM stability studies - The stability of the leached Au and PGM species in the autoclave discharge were tested by timed sampling of slurry taken from the pilot plant discharge. This was important to confirm that the Au and PGM would not be re-precipitated and lost during the post autoclave solid-liquid separation steps. The stability of Au and PGM in solution was proven to be independent of agitation and temperature within the range of conditions tested.
- Rheology - Rheology tests were carried out on slurry samples recovered from the Pilot Plant operation.

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- AuPGM concentrate upgrading - Autoclave testwork was conducted to upgrade the Au and PGM content of the AuPGM concentrate. This was done by selective re-leaching of base metals and sulphur from the AuPGM concentrate product, at both high and low temperatures. This work confirmed a window of temperature to upgrade the AuPGM precipitate (with an optimum average temperature at 195 °C). It was possible to upgrade the AuPGM precipitate from approximately 1,000 g/t (Au+Pt+Pd) to 16,000 g/t (Au+Pt+Pd).
- Co and Ni recovery from SX strip liquor - The separation of cobalt and nickel by solvent extraction (Phase 3 of the piloting referred above) was successful in producing separate and pure products. Cobalt was recovered in bench scale testwork as a cobalt hydroxide by treating the cobalt strip liquor with magnesium hydroxide slurry. Nickel precipitation was performed as part of the pilot plant continuous operation.

17.1.5 Hydrometallurgical Pilot-Scale Test Campaigns

The flowsheet tested during the August-September 2005 pilot campaign covered POX through to recovery of Cu, AuPGMs, Ni, Co, and Zn. The autoclave feed material consisted primarily of the concentrates produced in the flotation piloting described above, as well as some concentrate remaining from year 2000 testwork. This concentrate, which had been carefully stored in a freezer, was used to extend the circuit running time and provide additional product for characterisation.

A separate pilot campaign was conducted in October 2005 to test an option for separate recovery of Ni, Co and Zn hydroxide products via a Co/Zn SX circuit.

An additional autoclave pilot program was performed in April 2006. This short program was designed to confirm the viability of recycling a portion of the autoclave leach residue for improved AuPGM recovery and shorter autoclave residence time (1.1 hours of residence time instead of the 2 hours used in the "non-recycle" configuration.

The pilot plant design was developed by Bateman using a metallurgical flowsheet produced by METSIM modelling software. METSIM is an industry standard metallurgical simulation and design computer software package and METSIM models developed by Bateman were delivered to the Lakefield staff for design and operation of the pilot plant facilities.

As part of the hydrometallurgical pilot plant design, corrosion coupons were strategically placed in various parts of the circuit to obtain information on materials selection for the commercial plant.

Outcomes and conclusions from hydrometallurgical pilot plant work are summarized below in Table 17-3.

Table 17-3: Pilot Plant Test Outcomes and Conclusions

Flowsheet Area	Pilot Plant Conditions and Outcomes
POX	Optimum autoclave operating parameters included: operating at 225°C, ~3,100 kPag Total

Flowsheet Area	Pilot Plant Conditions and Outcomes
	Pressure, ~800 kPa O2, 10 g/L chloride and a 1.1 hour first pass residence time. Metals extractions were shown to improve by the introduction of a 200% residue recycle stream (i.e., a 2:1 ratio of leach residue to fresh feed). Average extractions for metals at optimum conditions were: Cu 99%, Ni 99%, Co 98%, Au 89%, Pt 93% and Pd 94%.
AuPGM Recovery	Au and PGM were precipitated from solution by adding CuS, recycled from the residual Cu recovery circuit. Recoveries were excellent with below detection limit values for AuPGM remaining in solution, and corresponded to a minimum precipitation efficiency of Au 88%, Pt 98% and Pd 99.5%. Further testwork led to the reintroduction of sulphur dioxide (SO_2) in the final flowsheet as a reductant for iron prior to CuS addition. This reduces the consumption of CuS and limits the elemental S content of the concentrate. The SO_2 pre-reduction system was tested and piloted in the year 2000 pilot plant at Lakefield.
Solution Neutralisation	This circuit operated to a pH of 1.3-1.4 using ground limestone addition while gypsum thickener underflow was recycled as seed to the first reactor. Analysis of the gypsum residue reported insignificant base metal content and low residual carbonate (0.07%).
Copper Solvent Extraction	Copper was extracted at 40°C in 3 counter current stages, scrubbed in 1 stage (to prevent chloride transfer to Cu electrowinning) and stripped in 2 stages. Two organic extractants, Acorga® M5640 and LIX® 973NS LV, were pilot tested. Orfom® CX80CT diluent was used in each case. Recovery of Cu to the strip liquor averaged 95.5% for both extractants, producing raffinate with Cu <1.0 g/L from PLS ranging 18-25 g/L Cu. No evidence of crud formation during testing was noted.
Copper Electrowinning	A total of 69 kg of copper metal was produced. Cathodes were harvested twice during the campaign. Four cathodes were sampled for purity – 2 from each extractant cycle. Cathodes from Cycle 2 met LME grade A specifications while cathodes from Cycle 1 showed minor contamination of Pb and S attributed to erratic temperature control during test start-up.
Raffinate Neutralisation	Raffinate is neutralized prior to recycle of raffinate as cooling solution back to the autoclave. This is necessary to reduce the free acid level in the autoclave product solutions and prevent the formation of basic ferric sulphate (BFS). The pH setpoints for raffinate neutralisation varied between 1.2-1.5 and were controlled via limestone slurry addition. Loss of Ni and Co to the residue was minimal.
Iron Removal	A portion of neutralized raffinate solution was directed to nickel and cobalt recovery. The first step in the Ni and Co recovery circuit is iron removal by oxidation and neutralization. Ferrous iron was oxidised to ferric iron by addition of gaseous oxygen and were removed from solution (along with aluminium) by hydroxide precipitation. Limestone was added to achieve the target pH of 4.2. Iron removal residue consisted predominantly of gypsum with low levels of iron and aluminium hydroxides. Ni and Co losses in the residue were minimal. Iron and aluminium removal efficiencies were 99.9% and 94.1% for this circuit.
Aluminum Removal	A separate stage of aluminium removal was included in the pilot plant circuit. In practice, this circuit did not consume limestone, as pH naturally rose to 4.6-4.7 due to an excess of

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Flowsheet Area	Pilot Plant Conditions and Outcomes
	alkalinity from the iron removal stage. Iron and aluminium removal efficiencies were 71% and 96% respectively (to give overall precipitation efficiencies of nearly 100% after two stages).
Residual Copper Recovery	Residual copper was precipitated as copper sulphide (CuS) using sodium hydrosulphide (NaSH), and was collected for use in AuPGM recovery. Stoichiometric addition of NaSH was required for copper precipitation. Solution analysis confirmed precipitation of 92% of the Cu for this circuit, with insignificant co-precipitation of Ni and Co.
Mixed Hydroxide Precipitation Stage 1 (HP1)	Ni and Co were precipitated as a mixed hydroxide using magnesium hydroxide slurry to a target efficiency of 85%. The mixed hydroxide precipitates collected during the pilot plant analysed 31.5-36.3% Ni, 1.67-1.92% Co, 0.31-0.37% Cu, 0.51-0.59% Fe, 4.27-4.84% Zn and 0.62-1.04% Mg.
Mixed Hydroxide Precipitation Stage 2 (HP2)	This circuit recovered residual nickel and cobalt from solution by precipitation with hydrated lime slurry at pH 8. Precipitate was thickened and recycled to the neutralisation circuit (where the residual metal hydroxides redissolved). Removal efficiency of residual Ni and Co from the feed solution averaged 93% and 92% respectively giving overall precipitation efficiencies through the two stages of hydroxide precipitation of nearly 100% for both Ni and Co.
Magnesium Removal	Magnesium was removed from the barren solutions after Ni and Co recovery by addition of hydrated lime slurry to pH 9. Mg precipitation was close to the target 50%. The magnesium hydroxide – gypsum product slurry was thickened, with overflow used as process water and underflow directed to tails. The absence of pay metals in the feed to magnesium precipitation resulted in negligible Ni and Co losses (0.14% and <0.02% respectively).
Co/Zn Solvent Extraction	The cobalt and zinc solvent extraction circuit was run as part of the campaign to produce purified metal hydroxides (rather than mixed hydroxide precipitation). Bulk Co/Zn extraction was achieved in 4 stages at pH 5.0-5.5 and 55°C, using 5% v/v Cyanex® 272 extractant in Orfom® SX80CT diluent. The higher temperature favoured Co extraction and displacement of co-extracted Mg. Co stripping then proceeded in 3 stages at pH 3 and 45oC, before Zn stripping in 2 stages at pH <1 and 40°C. Co extraction rates greater than 96% were achieved, with raffinate grades of below 10 ppm Co. Zinc extraction was greater than 99.9%. No evidence of crud formation during testing was noted and the circuit operated smoothly.

A variety of specialist vendors for thickening, filtration and flocculant selection were present during piloting to perform bench tests on slurry samples withdrawn from the operating pilot plant. The results of this testing have been used to provide equipment design parameters.

Flotation and hydrometallurgical piloting provided data for the development of a flowsheet generating maximum overall base and precious metal recoveries to final marketable products. The DFS

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engineering design incorporates the data from the various pilot campaigns that provides confidence for the capital cost and operating cost estimates.

17.2 Design Criteria and Process Overview

Key Design Criteria:

- Ore Feed 32,000 st/d (1,333 st/h)
- Plant Availability 90.0%

Crushing:

- Number of stages 4
- Feed to crushers F80 740 mm
- Primary crusher discharge P80 83 mm
- Secondary crusher discharge P80 39 mm
- Tertiary crusher discharge P80 11.4 mm
- Quaternary crusher discharge P80 8 mm

Milling:

- Rod Mill Work Index 13.4 kWh/t
- Ball Mill Work Index 15.6 kWh/t
- Abrasion Index 0.403
- Feed to Rod Mills F80 – 8 mm
- Milled product P80 – 120 µm

Flotation (Residence Time / Number of Stages):

- Rougher Flotation 7 min, 1 stage
- Scavenger Flotation 38 min, 1 stage
- Cleaner Flotation 15 min, 2 stages
- Concentrate Grind P80 – 15 µm
- Pressure Oxidation
- Temperature 225 °C
- Pressure 3,380 kPag
- Retention Time 1.1 h
- Solids Recycle Ratio 200% (residue recycle to fresh feed ratio)

Tailings:

- Flotation 34,300 dry st/d
- Hydrometallurgical 2,430 dry st/d (now reduced)
- Power 69 MW (at steady state draw, now reduced)

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The process plant design has been reviewed by PolyMet representatives plus external and Bateman process auditors and has subsequently been used as the basis for the capital and operating costs presented.

17.2.2 Process Overview

Existing equipment will be reinstated and used for both coarse and fine ore crushing (including gyratory and cone crushers), and in the ore milling circuit (including rod and ball mills). The flotation plant is a new circuit that will be housed in the existing Concentrator building.

Coarse Ore Crushing

In the coarse crushing area, ROM ore (with a top feed size of 48 inch /1,200 mm) is reduced in two stages to 100% passing 3 inch (75 mm) prior to further size reduction in the fine crushing circuit at an average feed rate of 1,666 st/h (1,512 mt/h). ROM ore is delivered by rail from the open pit and dumped sequentially from 100 short ton side tipping railcars into a 110 short ton dump pocket above the 60 inch (1200 hp/900 kW) gyratory primary crusher. Following primary crushing, the ore gravitates to a second stage of crushing in three parallel 36-inch (540 hp/400 kW) gyratory crushers. The discharge from the secondary crushers is conveyed to a coarse ore bin above the fine crushers, which has a live capacity of approximately 2,200 tons, which is equivalent to approximately 80 minutes of continuous feed.

Fine ore crushing – The coarse crushed product is further reduced in two stages to 8 mm suitable for feed to the milling circuit. Coarse crushed ore is delivered to a coarse ore storage bin that extends the length of the Fine Crushing Building. Since only three fine crushing lines will be reactivated only a portion of the total live storage capacity will be used. From the coarse ore bin material gravitates to three parallel fine crushing lines, each line consisting of a 7 ft (470 hp/350 kW) standard cone tertiary crusher discharging onto two double deck vibrating screens from where oversize discharges to two 7 ft (470 hp/350 kW) short head quaternary cone crushers. The screen undersize material passes directly to the conveyor below, which also collects quaternary crusher products. The final crushed product is conveyed to a fine ore bin in the Concentrator Building, which has a live mill feed storage capacity of approximately 17 hours.

Ore Grinding

The milling circuits liberate sulphide minerals contained in the ore through a process of particle size reduction. The milling circuit comprises twelve parallel circuits each consisting of a 12 ft diameter and 15 ft long 800 hp (600 kW) rod mill feeding a 1,250 hp (930 kW) ball mill operating in closed circuit with a cyclone, with a circulating load of 250%. Each rod mill receives a proportion of finely crushed ore, approximately 128 st/h (116 mt/h) at P80 of 8 mm, and discharges product to a ball mill, which produces milled product at P80 120 of µm.

Sulphide Flotation

The objective of the flotation circuit is to recover a bulk sulphide concentrate containing the base and precious metals whilst rejecting largely siliceous tailings. The concentrate produced is then fed to the POX in the hydrometallurgical plant.

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Milled primary cyclone overflow along with flotation regrind cyclone overflow is split to two parallel trains of rougher/scavenger flotation. Rougher concentrate from both trains is combined and undergoes two stages of cleaner flotation to reduce mass and increase sulphide grade ahead of POX. Scavenger concentrate is combined with Cleaner One tailings and is fed to a regrind circuit, which includes one regrind mill operating in closed circuit with a regrind cyclone. The regrind cyclone overflow is directed back to the head of flotation.

Scavenger tailings are pumped to the flotation tailings facility.

Flotation requires a number of reagents and make-up storage tanks and dosing pumps are provided within a nearby dedicated flotation reagents area.

Concentrate Fine Grind

The final cleaner concentrate is mixed with flocculant and thickened, and the resulting underflow is pumped to a fine grinding ISA Mill to produce the POX feed at P80 of 15 µm.

Flotation Tailings

Flotation tailings are pumped to the established Tailings Basin. Existing seepage collection systems will be augmented and upgraded to more efficiently capture seepage and return it to the basin.

17.3 Phase I Plant Design

As set out in the introduction to this section, PolyMet now plans to build the Project in two phases:

- Phase I: produce and market concentrates containing copper, nickel, cobalt and precious metals
- Phase II: process the nickel concentrate through a single autoclave, resulting in production and sale of high grade copper concentrate, value added nickel-cobalt hydroxide, and precious metals precipitate products.

The changes reflect continued metallurgical process and other project improvements as well as improved environmental controls that are being incorporated into the Project. The advantages, compared with the earlier plan, include a better return on capital investment, reduced financial risk, lower energy consumption, and reduced waste disposal and emissions at site.

The purpose of the beneficiation process (Figure 17-1) is to produce final separate concentrates. One of the separate concentrates will be a copper concentrate. The other separate concentrates would be differing grades of nickel concentrate. The nickel concentrates can be blended in various combinations. The concentrates could be shipped to customers, used as a feedstock to the hydrometallurgical process, or divided for both uses. PolyMet expects that the Beneficiation Plant would be operational two years before the Hydrometallurgical Plant and during that period, all concentrates would be shipped to customers. Once the Hydrometallurgical Plant becomes operational some or all of the nickel concentrates would be feedstock to the hydrometallurgical process. The decision to ship or process concentrates would be based on equipment maintenance schedules, customer requirements and overall Project economics.

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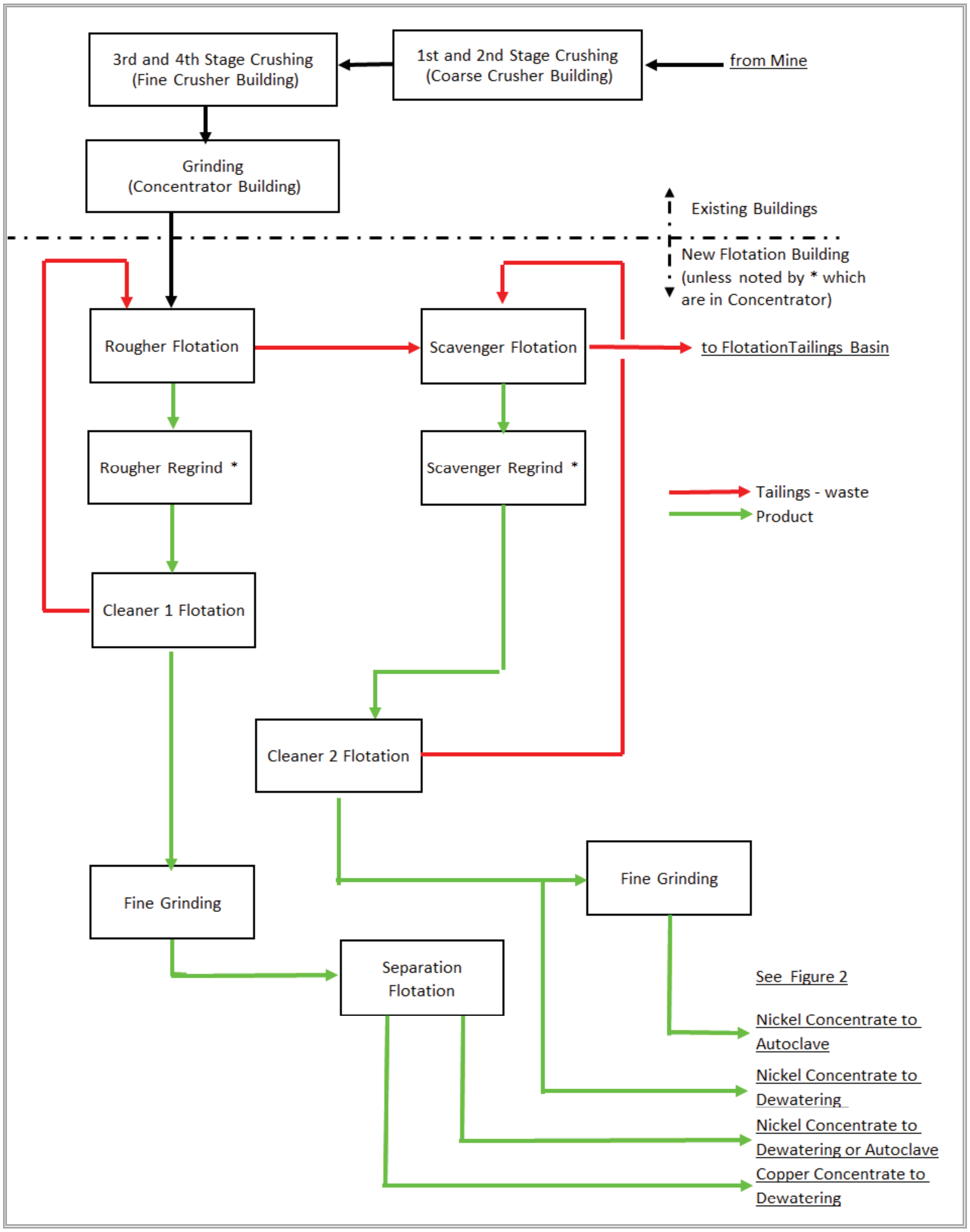
Figure 17-1: Beneficiation Plant Simplified Process Flow Diagram



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The Beneficiation Plant processes include ore crushing, grinding, flotation, dewatering, storage and shipping. Crushing and grinding would occur in the existing Coarse Crusher Building, Fine Crusher Building and Concentration Building, all of which remain from the LTVSMC operations. Flotation would occur in a new Flotation Building. Dewatering, storage and shipping would occur in a new Concentrate Dewatering/Storage Building.

17.3.1 Ore Crushing

In Ore Crushing, ore as large as 48 inches in diameter would be delivered by rail from the mine to the Coarse Crusher Building where each car would be emptied into a primary crusher at an average feed rate of 1,667 tons/hour (t/h). From the primary crusher, ore would move by gravity to four parallel secondary crushers. A conveyor system would move the ore, 80% of which would now be smaller than 2.5 inches, to the coarse ore bin located in the Fine Crusher Building.

The coarse crushed ore would be fed into parallel fine crushing lines. Each line would consist of a tertiary crusher, two quaternary screens and two quaternary crushers. The crushed ore would be transferred to the fine ore bin located in the Concentrator Building. At this stage, approximately 80% of the ore in the fine ore bin would be smaller than 0.315 inch.

17.3.2 Ore Grinding

Ore Grinding, which occurs in the Concentrator Building, would reduce the ore particle size to the point at which 80% would be less than 120 µm (4.7 x 10-3 inches). In Ore Grinding, the fine ore bin would feed into parallel mill lines. Each line would consist of a rod mill in series with a ball mill. The ore would pass through the rod mill once and the ground ore would be delivered to the ball mill. The ground ore would re-circulate through the ball mill until the particle size would be small enough for flotation.

The existing Coarse and Fine Crushing Building and Ore Grinding emission control systems will be replaced with components that meet or exceed the particulate emission standard required of new sources at taconite plants. To reduce space-heating requirements, emission control system exhaust would be able to be recycled to the buildings. The material collected would be mixed with water and added to the milling circuit. This means that the solids removed from the air stream would be recycled to the process and no solid waste management would be required and no water would be lost. Because water would be added to the mill lines and the beneficiation process would be wet from that point on, there would be no need for particulate emission control systems downstream of the fine ore bin.

In the event of a power failure, all process fluids would be contained within the Concentrator Building and recycled to the process when power has been restored. This same containment and recycle system would contain and control any minor spills.

17.3.3 Flotation

Once at a size of 120 µm, the ore would be processed in Flotation to recover the base and precious metal sulfide minerals. Flotation would consist of rougher and scavenger flotation lines followed by

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cleaner stages in a new Flotation Building and would produce separate nickel and copper concentrates.

In Flotation, separation of the sulfide minerals would be achieved using a collector/frother combination. Air would be injected into each flotation cell and the cell would be mechanically agitated to create air bubbles that would pass upward through the slurry in the cell. The frother (methyl isobutyl carbinol and polyglycol ether, or MIBC/DF250), would provide strength to the bubbles and the collector (potassium amyl xanthate, or PAX) would cause the sulfide minerals to attach to the air bubbles. The material attached to the bubbles would be concentrate and the material remaining in the slurry would be tailings.

The Rougher Flotation tailings would go to Scavenger Flotation where collector and frother would be added, along with copper sulphate as a flotation activator. The activator would ensure that the particles that would be difficult to float (i.e., contain minor amounts of sulfide) would be recovered in the concentrate, which reduces the total sulphur content of the tailings. The concentrate from Scavenger Flotation would go through Scavenger Regrind to Cleaner 2 Flotation. Cleaner 2 Flotation tailings would go back to Scavenger Flotation feed, while the nickel rich Cleaner 2 Flotation concentrate would be sent through Fine Grinding 2 to the Hydrometallurgical Plant or directly to Concentrate Dewatering. The tailings from Scavenger Flotation would be sent to the Flotation Tailings Basin. Rougher Flotation concentrate would be fed through Rougher Regrind to Cleaner 1 Flotation. Cleaner 1 Flotation tailings would go back to Rougher Flotation feed, while the concentrate would be sent through Fine Grinding 1 to Separation Flotation. Separation Flotation would produce a copper concentrate and two nickel concentrates. The copper concentrate would go to Concentrate Dewatering. The nickel concentrates would go to Concentrate Dewatering or to the Hydrometallurgical Plant.

The Scavenger Flotation tailings would be pumped to the Flotation Tailings Basin where the solids would settle and be stored permanently. The clear water would be re-circulated to the mill process water system.

In the event of a power failure, all process fluids would be contained within the Flotation Building and recycled to the process when power has been restored. This same containment and recycle system would contain and control any minor spills.

17.3.4 Concentrate Dewatering/Storage

Concentrate Dewatering/Storage would be used to dewater and store copper and nickel concentrates and to load those concentrates into covered rail cars. Concentrate Dewatering/Storage would be within the new Concentrate Dewatering/Storage Building.

The copper and nickel concentrates would each be delivered to separate dewatering lines each with a filter that would reduce concentrate moisture content to approximately 8 to 10%. The water removed by the filter would be returned to the Beneficiation Plant.

Each filtered concentrate would be conveyed to separate stockpiles within an enclosed 10,000 ton storage facility for loading into covered rail cars. The storage facility would store about 7 to 10 days of

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production capacity when flotation concentrate would be directed to Concentrate Dewatering/Storage. The storage facility would have a concrete floor and provisions to wash wheeled equipment leaving the facility to prevent concentrates from being tracked out of the facility.

In the event of a power failure, all process fluids would be contained within the Concentrate Dewatering/Storage Building and recycled to the process when power has been restored. This same containment and recycle system would contain and control any minor spills.

17.3.5 Processing Parameters

Table 17-4 shows PolyMet's estimates for daily production rates and size reduction through the processing steps in the beneficiation process. The rates and sizes provided are the values PolyMet would use to design plant piping and equipment.

Table 17-4: Design Processing Parameters

Step	Input			Output		
	Material	Rate (st/d)	Size (")	Material	Rate (st/d)	Size (")
Ore Crushing	Ore	32,000	48	Ore	32,000	0.315
Ore Grinding	Ore	32,000	0.315	Ore	32,000	4.7 x 10-3
Flotation	Ore	32,000	4.7 x 10-3	Conc.	374 to Hydrometallurgical Plant and 286 to Concentrate Dewatering, or 660 to Concentrate Dewatering	1.8 x 10-3
				Tailings	31,340	4.7 x 10-3
Conc. Dewatering	Conc.	660	7.1 x 10-4	Dried Ni and Cu Conc.	286 copper, and 374 nickel	7.1 x 10-4

Water needed for the milling and flotation circuits would primarily be return water from the Tailings Basin, which would include treated Mine Site process water. Any shortfall in water requirements would be made up by raw water from Colby Lake using an existing pump station and pipeline.

17.3.6 Process Consumables

PolyMet anticipates the raw materials shown in Table 17-5 would be consumed by the Beneficiation Plant processes.

Table 17-5: Beneficiation Plant Consumables

Consumable	Quantity	Mode of Delivery	Delivery Condition	Storage Location	Containment
Grinding Media (metal alloy grinding rods and balls)	15,600 t/a	Rail (13 rail cars/mo)	Bulk	Concentrator Building	None required

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Consumable	Quantity	Mode of Delivery	Delivery Condition	Storage Location	Containment
Flotation Collector (PAX)	1,171 t/a	Truck (2-3 trucks/mo)	Bulk bags	Reagents Building	None required
Flotation Frother (MIBC and DF250)	1,007 t/a	Tank truck (2-3 trucks/mo) 1	Bulk	Reagents Building	Separate 13,200 gal storage tanks
Flotation Activators (copper sulphate)	592 t/a	Truck (1-2 trucks/mo)	Bulk bags	Reagents Building	9,200 gal Activator Storage Tank
Flocculant (MagnaFlox 10)	16.5 t/a	Truck (1 truck/2 mo)	1,875 lb bulk bags	Reagents Building	None required
Gangue Depressant (CMC)	1073 t/a	Truck (2-3 trucks/mo)	Bulk bags	Reagents Building	None required
pH Modifier (hydrated lime)	10,279 t/a	Tank Truck (1-2 trucks/day)	Bulk	Reagents Building	Storage Silo

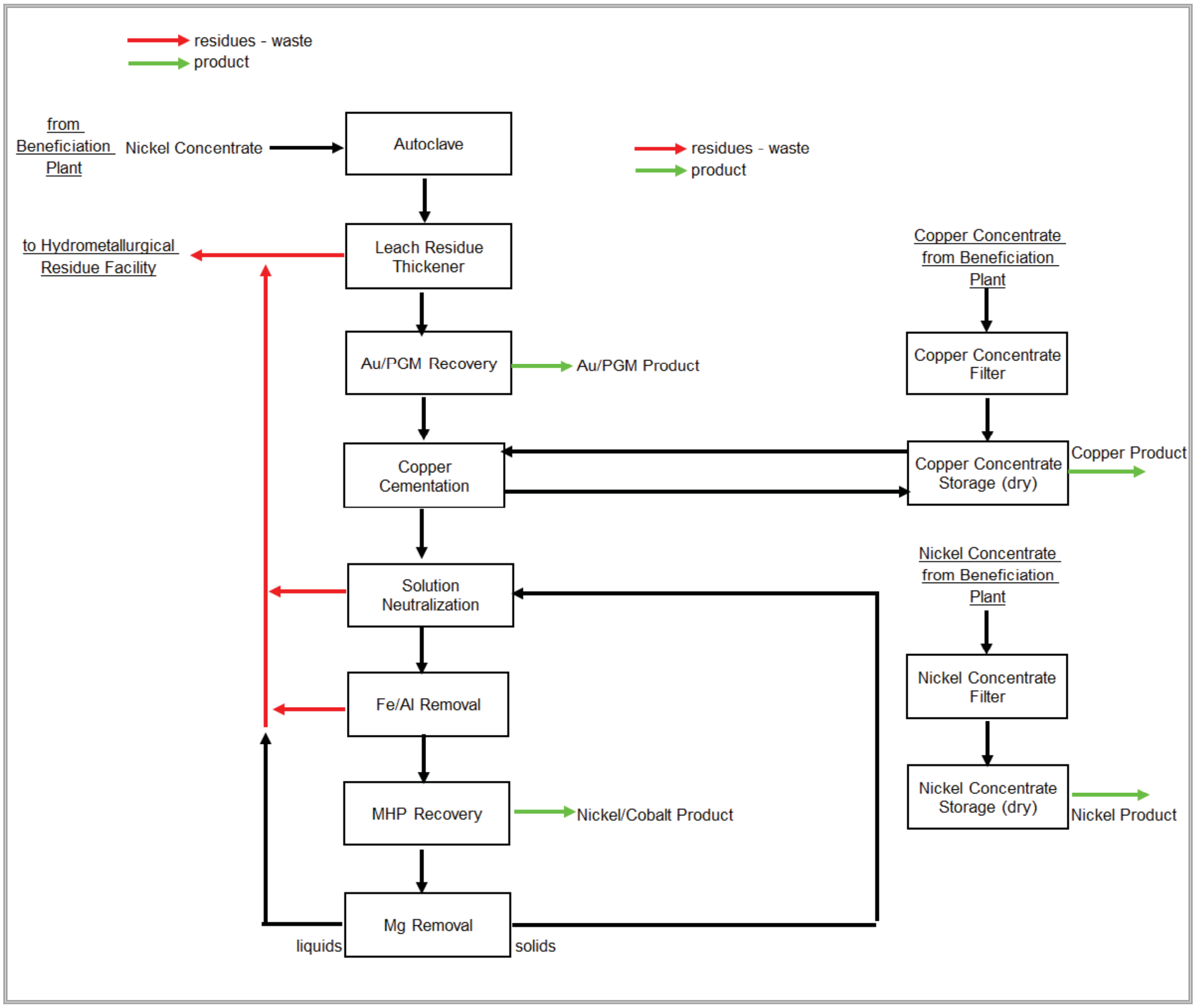
17.4 Phase II – Hydrometallurgical Plant

Hydrometallurgical processing technology would be used for the treatment of concentrates. This process would involve high pressure and temperature autoclave leaching followed by solution purification steps to extract and isolate platinum group, precious metals and base metals. All equipment used in the hydrometallurgical process would be located in a new Hydrometallurgical Plant Building.

Once the Hydrometallurgical Plant becomes operational some of the concentrates produced in the Beneficiation Plant would be feedstock to the hydrometallurgical process. The feedstock would be a combination of the separate nickel concentrates produced by the Beneficiation Plant. The decision to ship or process concentrates would be based on equipment maintenance schedules, customer requirements and overall Project economics.

PolyMet expects that the autoclave would be operational two years after the Beneficiation Plant becomes operational. A simplified process flow diagram for the hydrometallurgical process is shown on Figure 17-2.

Figure 17-2: Hydrometallurgical Plant Simplified Process Flow Diagram



17.4.1 Autoclave

In the Autoclave, the sulfide minerals in the concentrate would be oxidized and dissolved in a solution. Gold and platinum group metals would dissolve as soluble chloride salts. The solid residue produced would contain iron oxide, jarosite and any insoluble gangue (non-ore silicate and oxide minerals) from the concentrate. Generation of acid from the oxidation of major sulfide minerals would result in leaching of the silicate, hydroxide and carbonate minerals present in the concentrate.

Mine Waste Water Treatment Facility sludge (to recover metals and provide disposal of remaining solids) and hydrochloric acid (to maintain the proper chloride concentration in the solution to enable leaching of the gold and platinum group metals) would be added to the concentrate before the Autoclave. The Autoclave would be injected with oxygen gas supplied by a cryogenic oxygen plant at a rate that would be controlled to ensure complete oxidation of all sulfide sulphur in the concentrate.

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Slurry discharging from the Autoclave would be sent to the Leach Residue Thickener where solids would be settled with the aid of a flocculant. The Leach Residue Thickener underflow would be filtered to produce a filter cake, which would be washed, re-pulped, combined with other hydrometallurgical residues and pumped to the Hydrometallurgical Residue Facility. The Leach Residue Thickener overflow would go to Gold and Platinum Group Metals (Au/PGM) Precipitation.

17.4.2 Gold and Platinum Group Metals (Au/PGM) Recovery

The product produced by Au/PGM Recovery would be a filter cake made up of a mixed gold and platinum group metals sulfide precipitate. The filter cake would be put into either bulk bags or drums for sale to a third party refinery. The remaining solution would go to Copper Cementation.

17.4.3 Copper Cementation

Copper concentrate from dry concentrate storage would be re-pulped and the solution from Au/PGM Recovery would be contacted with the re-pulped copper concentrate. Copper would precipitate mostly in the form of copper sulfide. The enriched copper concentrate would be filtered and bled back into the copper concentrate stream ahead of filtration. All solutions would remain in the hydrometallurgical process. The remaining solution would then go Solution Neutralization.

17.4.4 Solution Neutralization

Solution Neutralization would be used to neutralize acids formed as a result of the upstream process. Solution from Copper Cementation would go to Solution Neutralization. Calcium in the form of either limestone or lime would be added. The result of the calcium addition would be the formation of gypsum that would be filtered to produce a gypsum filter cake. This filter cake would be washed, re-pulped, combined with other hydrometallurgical residues and pumped to the Hydrometallurgical Residue Facility. The solution remaining after neutralization would go to Iron and Aluminum Removal.

17.4.5 Iron and Aluminum Removal

Solution Neutralization would feed Iron and Aluminum Removal. Limestone, steam and air would be added to cause the aluminum and iron to precipitate. The precipitated metals would be filtered to produce a filter cake, which would be washed, re-pulped, combined with other hydrometallurgical residues and pumped to the Hydrometallurgical Residue Facility. The remaining solution would be sent to Mixed Hydroxide Precipitation.

17.4.6 Mixed Hydroxide Product (MHP) Recovery

Copper-free solution from Iron and Aluminum Removal would be reacted with magnesium hydroxide to produce nickel and cobalt precipitate. The precipitated metals would be filtered to produce a filter cake. The final mixed hydroxide product would have an approximate composition of 97% nickel and cobalt hydroxides with the remainder as magnesium hydroxide. The high quality mixed hydroxide filter cake would be packaged for shipment to a third party refiner. The remaining solution would go to Magnesium Removal.

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17.4.7 Magnesium Removal

Lime slurry would be added to the solution from MHP Recovery to facilitate magnesium precipitation. The resulting slurry would be pumped to the Hydrometallurgical Residue Facility along with other residues. The solids would settle in the residue cell to be stored permanently while the clear water would be reclaimed continuously to the Hydrometallurgical Plant process water system.

17.4.8 Process Consumables

The raw materials described below as well as those summarized in Table 17-5 would be consumed by the Hydrometallurgical Plant processes. Table 17-6 provides additional information regarding processing reagents deliveries, capacity and nominal use at the site.

Table 17-6: Materials Consumed by the Hydrometallurgical Plant Processes

Consumable	Quantity[1]	Mode of Delivery	Delivery Condition	Storage Location	Containment
Sulphuric acid3	1,500 t/a	Tanker (2 tank cars/mo)	Bulk	Adjacent to General Shop Building	31,965 gal storage tank with secondary containment
Hydrochloric acid	3,590 t/a	Tanker (3 tank cars/mo)	Bulk	Adjacent to General Shop Building	36,120 gal storage tank with secondary containment
Cobalt Sulphate3	18 t/a	Freight (1 delivery/mo)	67 lb bags in powder form	General Shop Building	In bags and batch mixed when needed
Guar Gum (Galactosol)3	6.5 t/a	Freight (1 delivery/mo)	70 lb bags in powder form	General Shop Building	Batch mixed on a daily basis (0.5% solution w/w)
Liquid Sulphur Dioxide	1,433 t/a	Tanker (2 tank cars/mo)	Bulk	Adjacent to General Shop Building	30,000 gal pressurized storage tank with secondary containment
Sodium Hydrosulphide3	513 t/a	Tanker Truck (2-3 tankers/mo)	Bulk as a 45% solution with water (w/w)	Adjacent to General Shop Building	25,750 gal storage tank
Limestone	125,000 t/a	Rail (1 100-car trains/week from April to October)	Bulk	Stockpiled on site	Berms/ditches around outdoor stockpile with water that has contacted limestone collected and added to the plant process water.

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Consumable	Quantity[1]	Mode of Delivery	Delivery Condition	Storage Location	Containment
Lime	4,344 t/a	Freight (75 loads/mo)	Bulk	Adjacent to General Shop Building	Lime Silo and 21,000 gal storage tank
Magnesium Hydroxide	4,866 t/a	Tanker (7 tank cars/mo)	60% w/w magnesium hydroxide slurry	Adjacent to General Shop Building	Magnesium Hydroxide 270,000 gallon Storage Tank
Caustic (NaOH)	33 t/a	Tanker Truck (1 load/mo)	50% w/w solution	General Shop Building	1,300 gal storage tank
Flocculant (MagnaFloc 342)	14 t/a	Freight	1,543 lb bulk bags of powder	Main Warehouse	In bags and batch mixed regularly as 0.3% w/w solution
Flocculant (MagnaFloc 351)	90 t/a	Freight	1,543 lb bulk bags of powder	Main Warehouse	In bags and batch mixed regularly as 0.3% w/w solution
Nitrogen (used in Hydrometallurgical Plant)2	19,113 t/a	NA	NA	NA	NA

Note: [1]Nitrogen used in the Hydrometallurgical Plant would be produced as a byproduct in the Oxygen Plant and no shipping or storage would be required

17.4.9 Hydrometallurgical Process Water

A separate Hydrometallurgical Plant process water system would be required due to the different nature of the process solutions involved in the hydrometallurgical and beneficiation processes. Hydrometallurgical process water would contain significant levels of chloride relative to the water in the milling and flotation circuits. The system would distribute water to various water addition points throughout the Hydrometallurgical Plant and would receive water from the Hydrometallurgical Residue Facility (water that was used to transport hydrometallurgical residue to the facility). Make-up water would come from flotation concentrate water and raw water.

17.5 Required Process Services

The Plant Site would require various services to perform its functions. These services would be in addition to site infrastructure needs. These services are summarized in Table 17-7.

Table 17-7: Plant Site Services

Service	Source	Source Location	Needed for
Compressed Air	Duty/standby arrangement of rotary screw type compressors	General Shop Building	Provide air at a pressure of 100 psig for plant services
Instrument Air	Air withdrawn from the plant air receiver	General Shop	Provide air for instruments

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Service	Source	Source Location	Needed for
	to an instrument air accumulator and dried in a duty/standby arrangement of driers and air filters	Building	
Steam	Natural gas-fired boiler	Hydrometallurgical Plant	Generates heat needed for start up of the autoclaves
Diesel Fuel Storage	Existing Locomotive Fuel Oil facility (storage is discussed in more detail in Section 3.1.2.8)	Area 2 Shop	Diesel for locomotives
Gasoline Storage	Existing storage facility – two 6,000 gal tanks	Main Gate	Gasoline for vehicles
Raw Water	Water from Colby Lake via an existing pumping station and pipeline (see Section 4.1)	Stored in the Plant Reservoir	Plant fire protections systems, plant potable water systems, make up water for grinding and flotation process water and hydrometallurgical plant process water
Potable Water	Existing Processing Plant potable water treatment plant would be refurbished and reactivated	Near the Plant Reservoir	Potable water distribution system includes the Area 1 and Area 2 Shops
Fire Protection	Existing fire protection system would be refurbished, reactivated and extended to new buildings	Plant Reservoir	Area 1 and Area 2 Shops have independent fire protection systems
Oxygen	440 t/d Oxygen Plant. Plant process takes in ambient air, compresses it and separates the oxygen from nitrogen and other trace atmospheric gases. Oxygen would be transported via pipeline to plant processes and nitrogen and trace gases would be returned to the atmosphere.	Adjacent to Concentrator	Plant processes

17.6 Plant Site Air Quality Management

All active areas at the Plant Site, including the Tailings Basin, would be subject to a Fugitive Dust Control Plan approved by MPCA for managing fugitive dust generated at material handling locations, unpaved roads and areas potentially subject to wind erosion. The emission control systems on plant processes would have automated monitoring and alarming of operating parameters that indicate off-spec performance with auditable procedures to track the actions taken by operating and maintenance personnel in response to the alarm. Periodic stack testing would demonstrate compliance and confirm the proper alarm points.

17.7 Comments on Section 17

The modifications to the flowsheet since the DFS was completed in September 2006 reduce the technical risks during start up (because initial production of concentrates use very established technology). The permitting delays have provided PolyMet with an unusual opportunity to review and analyze its plans, resulting in a technically and economically stronger project.

The biggest technical risk in the DFS was the start-up of the hydromet circuit – fine-tuning the process chemistry to achieve expected recoveries and commercial product standards. With the revised schedule, PolyMet will have commercial sales of copper and nickel concentrates during ramp-up of the hydromet circuit.

18 PROJECT INFRASTRUCTURE

As reported in the DFS, one of the key elements of this project is that infrastructure is well established, generally in good condition and, in most cases, requires only minor modification to accommodate new installation. Existing infrastructure and services include:

- incoming HV power (138 kV) from the Minnesota Power grid
- power distribution within and around the existing facilities
- water supply and distribution
- sewage collection system (though treatment plant must be replaced)
- guard house and related security facilities
- offices, changing rooms, meeting rooms, lunch rooms
- sample preparation and analytical laboratories
- warehouses and storage facilities
- road and on-site railroad system
- railroad connection to common carrier rail network
- workshops
- natural gas supply
- communications
- mine railroad and locomotive services and refuelling facilities
- tailings disposal facilities.

All the above were evaluated in detail to determine their suitability and cost effectiveness of re-use and cost estimates have been included to refurbish the existing facilities and return them to a condition suitable for safe re-use by PolyMet. In 2010 and 2011 a program of testing and refurbishing the major electrical equipment of switchgear, transformers and MCCs was initiated which has confirmed that most of these assets can be reactivated for the NorthMet project.

In addition to the various and extensive offices available at the Coarse Crushing facility, the Fine Crushing building, in the Concentrator, associated with the General Workshop, the Unit Rebuild Workshop and the warehouse complex, PolyMet has also acquired the former LTVSMC Administration Building located away from the main industrial area on the public road from Hoyt Lakes. This building previously housed 150-200 administration staff. PolyMet intends to use this building during the construction phase to accommodate engineering and construction management staff. Existing telecommunications, networking and fibre optic connections within the building are functional and can be fully reactivated at minimal cost.

Historically the Mesabi Iron Range has been the centre of a very large and extensive iron ore mining industry with six world-class taconite iron ore mining operations in production at this time. To support this mining activity, the area has a very well developed infrastructure, which includes excellent roads, extensive railroads, access to ocean shipping via the nearby ports of Duluth/Superior, reliable grid

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power, engineering support services and service providers as well as a significant pool of skilled labour for construction as well as operations. PolyMet will benefit from the existence of this infrastructure, which will facilitate construction and provide simplified and reliable shipping logistics for equipment, parts, consumables and product export.

18.1 Road and Logistics

18.1.1 Ore Haulage

The LTVSMC taconite mining operation depended entirely on rail transport of ore to the Primary Crusher. To minimise capital cost, PolyMet plans to re-use large parts of the former LTVSMC railroad system, which will be refurbished to transport run of mine ore approximately nine miles from the NorthMet open pit to the Primary Crusher at a planned rate of 32,000 st/d, 365 d/a.

Ore will be mined conventionally and transported by mine haul trucks to a rail transfer hopper located near the pit rim. With a live storage capacity of 3,600 tons, the rail transfer hopper will allow for rapid and efficient loading of rail cars while effectively separating and de-coupling the mining and rail haulage systems. Storage capacity provided by the rail transfer hopper plus the adjacent ore stockpile will allow a degree of independence between the mining and the rail haulage systems; however, limitations on ore storage capacity in the crushing system and at the Concentrator will require railroad haulage to operate 7 days per week, year round to ensure concentrator feed can be maintained.

The rail transfer hopper will be constructed from reclaimed and refurbished components of two approximately similar structures, which PolyMet has acquired from Cliffs. Built in the latter part of the 1990s and known as "Super Pockets", the two former LTVSMC rail transfer hoppers transferred taconite ore very efficiently from mine haul trucks to rail cars until closure in 2001. PolyMet has already recovered for re-use the mechanical, hydraulic and electrical components of these two hoppers and proposes to build a single, purpose-built structure, similar to the original LTVSMC hoppers, on the south side of the NorthMet pit. The newer equipment will be refurbishment for reactivation while the second, older, set will be retained and refurbished in due course as operating spares.

Figure 18-1 shows one of the two LTVSMC transfer hoppers operating with taconite. Equipment condition is good and estimates for its refurbishment have been obtained from original equipment manufacturers. PolyMet is confident that the re-built system will work efficiently and cost-effectively.

To connect the rail transfer hopper to the Primary Crusher, a total of 10,600 ft of new track will be constructed along with installation of 1,600 new ties and 3,000 ft of new rail in existing track. The existing Main Line will not require upgrading as it has remained in irregular service since closure of the LTVSMC facilities. The new sections of track construction will include 5,000 ft of spur line to connect the transfer hopper to the existing main line, and 5,600 ft of new track to connect the main line and existing track running to the Primary Crusher. Much of the latter will utilise former rail bed from which ties and rail were removed prior to acquisition by PolyMet. Design includes provision adjacent to the rail transfer hopper for direct loading of railcars using front-end loaders in the event of hopper breakdown or non-availability.

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Figure 18-1: LTVSMC Rail Transfer Hopper in Operation

To connect the rail transfer hopper to the Primary Crusher, a total of 10,600 ft of new track will be constructed along with installation of 1,600 new ties and 3,000 ft of new rail in existing track. The existing Main Line will not require upgrading as it has remained in irregular service since closure of the LTVSMC facilities. The new sections of track construction will include 5,000 ft of spur line to connect the transfer hopper to the existing main line, and 5,600 ft of new track to connect the main line and existing track running to the Primary Crusher. Much of the latter will utilise former rail bed from which ties and rail were removed prior to acquisition by PolyMet. Design includes provision adjacent to the rail transfer hopper for direct loading of railcars using front end loaders in the event of hopper breakdown or non-availability.

The rail infrastructure will be refurbished to safely meet operational requirements at minimal capital cost with periodic rail and tie replacement during mine life to maintain serviceability.

PolyMet has acquired from Cliffs 120 side dumping, 100-ton capacity DIFCO railcars formerly used by LTVSMC to transport run of mine taconite to the Primary Crusher. These rail cars, which are not self-dumping, are very robust and have been inspected by KOA who have developed an estimate to restore the fleet to operational condition. The strategy is to initially restore the fleet to safe and reliable operating condition at minimal cost. Once the mine is operational and generating cash flow, rolling stock will undergo progressive restoration/rebuilding as required to minimise operating costs for the remaining mine life.

18.1.2 Minesite Infrastructure

Mine Site Facilities

Apart from the rail-loading hopper, facilities at the NorthMet mine site will be kept to a minimum. A covered field service and refuelling facility with temporary storage tanks will be set up near the rail transfer hopper. As is common at taconite mining operations in the area, fuel oil will be supplied direct to the end-user by a local supplier who will also be responsible for its storage and distribution.

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In much the same way, a local supplier of explosives and blasting accessories will provide an 'in-hole' service delivering and placing explosives directly into blast holes. The supplier will be responsible for storing and delivering explosives and hence no onsite explosives magazine will be required.

Mine and Railroad Offices and Staff Facilities – Area 2 Workshops & Offices

Offices and change-house facilities for mine and railroad operating and technical personnel will be provided by refurbishing existing facilities located adjacent to the railroad and about two miles east of the Primary Crusher. Known as the Area 2 Shop, this facility includes a large building which will house the refurbished offices and personnel facilities as well as a workshop, complete with overhead crane that will be set up for railroad rolling stock maintenance.

Mine Mobile Equipment Maintenance Facility – Area 1 Truck Shop

This study assumes the mining contractor will be responsible for equipment fleet maintenance and that all associated costs are included in the contract rates used to develop mine operating costs. PolyMet now owns the former LTVSMC mine mobile equipment maintenance complex known as the Area 1 Truck Shop (Figure 18-2). This will be refurbished and reactivated for use by the mining contractor. Area 1 Truck Shop is a purpose-built, fully enclosed, winterised, heavy mobile equipment maintenance facility located about one mile west of the process plant site. Comprising six truck bays (capable of accommodating haul trucks up to 240 ton payload class), three miscellaneous heavy equipment bays, a two-stall, enclosed truck wash down bay and associated shops, lunch room, offices, storage capacity, change house and ablution amenities, this facility is ideal for maintaining the mining equipment fleet. Although it is located about nine miles from the mine site, access between the two will be in part via the existing, upgraded Dunka Road and in part through former LTVSMC mine areas (now inactive) to avoid mixing light and heavy vehicular traffic in the vicinity of the Area 2 Offices. The minor inconvenience of having to move equipment between the mine and the workshops is offset by having a ready-made, comprehensive maintenance facility available at very low capital cost.

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Figure 18-2: Area 1 Truck Shop viewed from the southeast showing the tracked equipment bays and tyre shop

Mine Site Electrical Power Distribution

Electrical power for the major items of mining equipment (excavators, blast hole drills, dewatering pumps, powering the rail transfer hopper facility and for ancillary services) will come from the nearby 138 kV transmission line owned and operated by local power utility, Minnesota Power (MP). For cost estimation purposes, it has been assumed that the power utility will provide the main step down transformer at the mine site as well as the connection from the 138 kV transmission line. From there power will be distributed around the open pit by means of a single circuit line suspended from wooden poles. This supply line will be extended periodically as required by the changing nature of the ongoing mining operation. PolyMet has already acquired sufficient 4,160 V, skid-mounted substations to meet the start-up requirements of the mining fleet though it is anticipated that additional substations and extension of the in-pit power line will be required in years 6 and 12.

18.1.3 Existing Beneficiation Plant & Equipment

Assessment Methodology & Engineering Philosophy

At closure, the former LTVSMC facilities were a fully operational, well maintained, going concern. Shut down had been systematic and there was an expectation that the plant would be re-started at some point in the future. Prior to the start of the DFS preliminary engineering studies by Optimum Project Services Ltd., Penguin Automated Systems, Inc. and Bateman assessed the major elements of the crushing plant, milling and tailings disposal facilities and determined they were fit for the purpose of crushing and milling NorthMet ore. The exception was the original taconite flotation equipment, which is to be removed and replaced with larger capacity, state-of-the-art flotation equipment engineered specifically for NorthMet ore. It was PolyMet's expectation, therefore, that much of the

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plant could be reactivated at minimal cost, with up-grades restricted to areas such as environmental controls and dust extraction where stringent compliance standards are expected.

To assess the condition of existing equipment and hence to determine the risks and costs associated with re-starting it, detailed site inspections were carried out by qualified individuals who had previously worked at and knew the plant intimately. In addition to drawing on the personal experience and knowledge of former LTVSMC employees, detailed and pertinent operating data, maintenance records and reports, and supervisors' shift logs were reviewed to provide a detailed picture of the condition of the plant at closure. During July and August 2006, a number of motors including those for a crusher, a rod mill, a ball mill, feeders and various drives were successfully test-started to confirm reactivation assumptions. Existing instrumentation was also reviewed to confirm the extent to which it could be reactivated. The number of test failures was minimal thereby adding confidence that the selected plant can be re-started with limited refurbishment. Appropriate allowances are made in the Capital and Operating cost estimate for refurbishment prior to restarting equipment and subsequent staged maintenance.

Because the original LTVSMC plant had a capacity (90,000 lt/d) nearly three times larger than that required by PolyMet, part of the design and commissioning philosophy assumed reactivation of sufficient plant and equipment to meet the expected ramp-up schedule with subsequent reactivation of additional equipment to provide spare capacity when major scheduled overhauls or maintenance work is required.

Another aspect of design philosophy relates to the use of spare equipment. There is a large amount of equipment available to PolyMet, which does not need to be immediately reactivated. Therefore, PolyMet intends to refurbish some of this surplus equipment progressively to provide spares in the event of breakdown, or additional capacity in the event that some existing equipment does not perform as expected.

Requirements for Re-commissioning Existing Plant Facilities

Based on detailed plant condition assessments, the following activities will be necessary to refurbish and reactivate the ore beneficiation facilities.

- The existing plant facilities will be cleaned up and made safe ahead of refurbishment work. This work will include removal of debris as well as asbestos removal and mitigation.
- Buildings are structurally in very good condition and need only minor repairs including some minor roof patching and drain pipe replacement due to freezing damage.
- Crusher maintenance records were used to determine remaining wear life and to plan and schedule subsequent maintenance. Liners and wear materials will be replaced where remaining life was identified as less than 25% original or where obviously required. Other items needing attention in the Coarse Crushing facility include the rebuild of an existing Pioneer feeder and replacement of one METSO apron feeder.

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- In the Fine Crushing facility, equipment from four of the original seven lines was sold and removed prior to acquisition by PolyMet. The planned production rate requires only three fine crushing lines, each line consisting of one 7 ft standard tertiary cone crusher in series with two 7 ft quaternary shortheads. These will be arranged so as to maximise live storage capacity in the overhead coarse ore bin. There are also a variety of spare crusher frames, bowls, mantles, drive motors, conveyors and feeders, which will be refurbished for use as spares. Using LTVSMC maintenance records verified by field inspection, it was determined that only one of the three tertiary crushers requires new liners and a frame repair. The six quaternary crushers have good liners in place and will only require servicing prior to start-up. The six existing single deck screens between the tertiary and quaternary crushers will be replaced with new double deck screens for increased screening efficiency.
- Conveyors 3A, 4B and 5N will be reactivated to transport fine crushed ore to the ore beneficiation building storage bins. As elsewhere, maintenance records were used to determine the condition of conveyor drives, bearings, trippers, feeders and related components. Visual inspection of conveyor idlers indicated about 10% would need replacement prior to start-up. Chute work will be replaced where worn.

Of the 34 original rod/ball mill grinding lines (Figure 18-3), only twelve will be needed for 32,000 t/d capacity. Mill lines 1-N to 12-N inclusive will be reactivated, though it is proposed to use and relocate the mills with the most remaining liner life.

There are also three 12 ft 2" by 23 ft 4", 1500 hp regrind mills, one of which will be reactivated to regrind scavenger concentrate, while regrind mill 3S will be used to produce a limestone slurry for acid neutralisation in the hydrometallurgical plant. The third mill will be available as stand-by.

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Figure 18-3: Rod Mill - Detail

The concentrator upper bay is equipped with two overhead cranes, one 200 ton capacity and one 25 ton capacity, which range over the full length and breadth of the milling level. These cranes are functional and will require only inspection and re-certification before reactivating. These cranes also provide tremendous operational and maintenance flexibility as they have sufficient lifting capacity to pick up and move a mill shell (rod or ball) complete with media charge to a central maintenance area.

Based on mill throughput records and maintenance records, a liner replacement schedule was developed which optimises remaining liner life and forms the basis of mill capital and operating cost estimates.

The large number of redundant mills and associated feed equipment will allow PolyMet to progressively refurbish units as required for spares. Moreover, in the unlikely event that existing equipment does not perform as expected, additional milling capacity can be brought on line quickly and cheaply.

A new sulphide flotation circuit will be installed. A feature of mill building design was the use of gravity feed wherever possible to minimise pumping.

The existing raw, domestic, mill, service and fire water systems will be reactivated with only limited refurbishment necessary. The original facilities made extensive use of pumped hot water and steam for plant heating; however, to avoid costly overhaul of this system, new gas-fired heating equipment will be installed and, where necessary, existing gas-fired equipment will be reconditioned. (The plant site is served by a natural gas pipeline with up to 13,000 M cu ft/day of natural gas at 125 psi, which far in excess of PolyMet's consumption estimates.

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The primary substation was operated continuously with a power draw of 130 MW and since LTVSMC closure parts of this substation have been kept in operation, albeit at reduced load. PolyMet will re-commission it to service the existing plant site facilities, the new hydrometallurgical plant facilities and the new mine service area.

Included in the acquisition of the Erie Plant were large numbers of spare electric motors of all sizes, MG sets, electrical switching gear, starters, motor controls and associated electrical gear.

18.2 Waste Storage Facilities

18.2.1 Flotation Tailings Management

Flotation tailings would be placed on the former LTVSMC tailings basin. The existing former LTVSMC tailings basin is unlined and was constructed in stages beginning in the 1950s. It was configured as a combination of three adjacent cells, identified as Cell 1E, Cell 2E and Cell 2W and was developed by first constructing perimeter starter dams and placing tailings from the iron-ore process directly on native material. Perimeter dams were initially constructed from rock and subsequent perimeter dams were constructed of coarse tailings using upstream construction methods. The LTVSMC tailings basin operations were shut down in January 2001 and have been inactive since then except for reclamation activities consistent with a MDNR approved Closure Plan.

The NorthMet flotation tailings would be deposited in slurry form through a system of pumps and moveable pipelines. Tailings would go into Cell 2E for the first seven years of operation, then into both Cells 1E and 2E. Tailings would be deposited by gravity flow over discharge beaches when necessary and otherwise subaqueously via movable diffusers throughout the pond. The small and fairly uniform grind size of the tailings would allow for a fairly consistent particle size distribution to be achieved, minimizing segregation of coarse and fine portions. The dam would be raised using the LTVSMC bulk tailings. Tailings beaches would exist along the northern and northeastern dams of Cell 2E and the southern and eastern dams of Cell 1E.

The tailings would settle out of the slurry and the decanted water would be allowed to pond and be collected using a barge pump back system. The barge system would consist of a primary pump barge in Cell 1E, an auxiliary pump barge in Cell 2E, piping from the primary pump barge to the Beneficiation Plant and piping from the auxiliary pump barge to Cell 1E. The auxiliary pump barge would not be needed once the cells combine to form one cell. The return water pipelines would be moved as dams are raised (up to the maximum of 1,732 ft amsl to keep the pipeline at or near the top of the dam. The return water pipes would be fitted with a relief drain valve to allow for water to be drained back to ponds in case of shutdown during winter operations to avoid damage to the pipes from freezing or suction. Pumps would also be fitted with deicing mechanisms to avoid freezing.

18.2.2 Hydrometallurgical Residue Management

The hydrometallurgical process would generate residues from five sources:

- autoclave residue from the leach residue filter

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- high purity gypsum from the solution neutralizing filter (depending on the market, this may become a saleable product, but is currently planned to be managed as a waste)
- gypsum, iron and aluminum hydroxide from the iron and aluminum filter
- magnesium hydroxide precipitate from the magnesium removal tank
- other minor plant spillage sources.

In addition to the above listed sources, solid wastes from the wastewater treatment facility at the mine sire (WWTF) would be recycled directly into the Hydrometallurgical Plant to recover metals. The WWTF solids would be similar to the Hydrometallurgical Residue Facility materials, consisting primarily of gypsum, metal hydroxides and calcite. These hydrometallurgical residues, which would include the non-recoverable metal portion of the solid wastes from the WWTF, would be combined and disposed of in the Hydrometallurgical Residue Facility as described below.

18.2.3 Hydrometallurgical Residue Cell Design and Operations

The Hydrometallurgical Residue Facility would consist of a lined cell located adjacent the southwest corner of Cell 2W of the former LTVSMC tailings basin. The cell would be developed incrementally as needed, expanding vertically and horizontally from the initial construction and would initially be designed to accommodate approximately 2,000,000 tons or six years of operations. The cell would be filled by pumping the combined hydrometallurgical residue as slurry from the Hydrometallurgical Plant. A pond would be maintained within the cell so that the solids in the slurry would settle out, while the majority of the liquid would be recovered by a pump system and returned to the plant for reuse. The residue discharge point into the cell would be relocated as needed to distribute the residue evenly throughout the cell.

18.3 Water Management

Water would be consumed at the Plant Site in both the Beneficiation Plant and the Hydrometallurgical Plant. For the most part, water operations within these two plants would operate independently. The only exceptions would be the transfer of flotation concentrate from the Beneficiation Plant to the Hydrometallurgical Plant and the combining of filtered copper concentrate and solution from Au/PGM Recovery in the Copper Cementation process step.

18.3.1 Hydrometallurgical Plant

All water that enters the Hydrometallurgical Plant would be consumed within the hydrometallurgical process, exiting as steam or becoming entrained within the solid waste residues or products generated through the hydrometallurgical process. The average annual water demand rate for the Hydrometallurgical Plant is estimated at 240 gpm, but varying from 114 to 406 gpm monthly as operating and climatological variations occur. At the same time, hydrometallurgical process residues would be disposed in the lined Hydrometallurgical Residue Facility, where the solids would settle out and the water would pond on the cell. To the extent possible, water that would be used to transport residue to the facility would be returned to the Hydrometallurgical Plant; however, some losses would occur through evaporation, storage within the pores of the deposited residue, or liner leakage to

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groundwater. In addition, water that would be contained in process fluids, should spillage of these fluids occur, would remain within the Hydrometallurgical Plant buildings and be returned to the appropriate process streams.

18.3.2 Beneficiation Plant

Within the Beneficiation Plant, water would be used to carry the ore through the grinding, flotation and separation steps, then to transport the tailings to the Tailings Basin. To the extent possible, water that would be used to transport tailings to the basin would be returned to the Beneficiation Plant, however some losses would occur through evaporation, storage within the pores of the deposited tailings, or seepage to groundwater under the Tailings Basin.

In addition, water that would be contained in process fluids, should spillage of these fluids occur, would remain within the Beneficiation Plant buildings and be returned to the appropriate process streams.

18.3.3 Tailings Basin

The primary source of process water for the Beneficiation Plant and the Hydrometallurgical Plant would be the Tailings Basin, which includes treated water piped from the Mine Site. Process water needs above and beyond that would be pumped from Colby Lake.

The Tailings Basin would be the final collection for process water that flows through the Beneficiation Plant and process water pumped from the Mine Site. Direct precipitation and run-off from the process areas at the Plant Site would also be directed to the Tailings Basin. Water that seeps from the toe around the perimeter of the Tailings Basin and emerges as surface seepage would be collected and returned to the Tailings Basin. Current surface seepage as well as any new surface seepage that develops during NorthMet operations will be collected. During times of high water flow from the Mine Site, which could result in excess water in the tailings basin, the recovered groundwater seepage would be pumped to a new Waste Water Treatment Plant located south of the Tailings Basin. These water management methods would result in no new direct surface discharge of process water at the Plant Site or Mine Site during operations and would minimize water needed via water appropriation from Colby Lake.

18.4 Camps and Accommodation

The LTVSTC operations employed approximately 1,400 people when they were shut in 2001. Hoyt Lakes was originally built to provide homes and a community for people working at the operations. Several other cities near the NorthMet Project are well equipped with schools, hospitals and other services.

18.5 Comments on Section 18

The existing plant and associated infrastructure immediately related to the plant and within the community are key attributes of the NorthMet Project. In view of the slower permitting process than

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what was originally expected, PolyMet plans to update its assessment of work needed at the existing facilities. However, the basic infrastructure remains in good shape, even if more electrical and other work needs to be done than was contemplated in the DFS.

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19 MARKET STUDIES AND CONTRACTS

In the 2006 DFS Technical Report, PolyMet set out an analysis of the markets for the three products it then contemplated. As described elsewhere, PolyMet now plans to produce copper and nickel concentrates initially, then upgrade the nickel concentrate into a nickel-cobalt hydroxide and a precious metals precipitate.

An essential part of these revised plans is PolyMet's ability to market these products. In September 2008, PolyMet announced that it had entered into a long-term marketing agreement with Glencore whereby Glencore will purchase all of PolyMet's products (metals, concentrates or intermediate products). Pricing is based on London Metal Exchange with market terms for processing – in the case of copper concentrates, the benchmark is annual Japanese smelter contracts.

19.1 Market Studies

Since most of PolyMet's products are actively traded on terminal markets with active forward pricing, PolyMet has not conducted any specific market studies. Metal prices used in mineral resource and reserve calculations are substantially below recent levels and PolyMet.

19.2 Commodity Price Projections

Resource and reserve estimates have been based on prices substantially below recent market levels.

Table 14-21 summarizes metal prices used in resource and reserve estimation, prices used is the economic analysis in the DFS, prices used in the May 2008 DFS update, and three-year trailing average prices to June 31, 2012.

Copper and nickel are the most important metals for PolyMet. In the DFS, PolyMet estimate that copper would contribute 46% of net revenues, nickel-cobalt 38% and precious metals 16%.

19.3 Contracts

In 2008, PolyMet entered into an agreement with Glencore whereby Glencore will purchase all of PolyMet's products (metal, intermediate products, or concentrates) on independent commercial terms at the time of the sale. Glencore will take possession of the products at site and be responsible for transportation and ultimate sale.

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19.4 Comments on Section 19

In view of Glencore's position as the world's largest trader of commodities, with especially strong positions in copper and nickel, there are no material risks associated with PolyMet's product marketing.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

PolyMet commenced the environmental review and permitting process in early 2004. In October 2005, the DNR published its Environmental Assessment Worksheet Decision Document establishing the DNR as the lead state agency and the USACE as the lead federal agency for preparation of an EIS for the Project.

In 2006 the Lead Agencies selected Environmental Resources Management, a leading global provider of environmental, health and safety, risk, and social consulting services, as independent environmental contractor (the "EIS contractor") to prepare the EIS. The EIS Contractor team included members with expertise and experience in mining sulfidic ores. Several other government agencies (including the USFS, the Bois Forte Band of Chippewa and the Fond Du Lac Band of Lake Superior Chippewa) joined the EIS preparation team as cooperating Agencies, which brought their special expertise to the process.

In January 2007, PolyMet submitted a detailed project description (DPD) to state and federal regulators. The DPD laid out development plans and proposed environmental safeguards including a mine plan, a wetland mitigation plan, air and water quality monitoring plans and a closure plan with closure estimate. Since then, PolyMet has submitted a supplemental DPD as well as more than 100 supporting research studies, including comprehensive mine waste characterization studies, water quality modelling and air quality modelling.

Under state and federal guidelines and regulations, a Draft EIS identifies the environmental impact of a proposed project as well as evaluating alternatives and ways to mitigate potential impacts. PolyMet was involved in the process of alternative/mitigation development and had input into the technical and economical feasibility of potential alternatives and mitigations. The EIS Contractor prepared a series of preliminary versions of the Draft EIS that were reviewed and commented on by the Lead Agencies, other governmental agencies, and PolyMet.

In November 2009, the Lead Agencies published the PolyMet Draft EIS with formal notification of publication in the Minnesota Environmental Quality Board (EQB) Monitor and the Federal Register, which started a 90-day period for public review and comment, which ended on February 3, 2010. During this period, the lead Agencies held two public meetings – one in the town of Aurora, MN near the Project location and one in Blaine, MN in the metropolitan Minneapolis-St. Paul area.

The Lead Agencies received approximately 3,800 submissions containing approximately 22,000 separate comments, including an extensive comment letter from the EPA in its role as reviewer of projects that could impact the environment. Several other governmental agencies including the United State Forest Service (USFS) and Tribal cooperating agencies took part in the environmental review process.

On June 25, 2010, the Lead Agencies announced that they intended to complete the EIS process by preparing a Supplemental Draft EIS (SDEIS) that incorporates the land exchange proposed with the

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USFS Superior National Forest and expands government agency cooperation. The USFS joined the USACE as a federal co-lead agency through the completion of the EIS process. In addition, the EPA joined as a cooperating agency. The DNR remains the state co-lead agency.

On October 13, 2010, the USACE and the USFS published a Notice of Intent to complete the SDEIS, which will:

- supplement and supersede the Draft EIS and respond to concerns identified by the EPA and other comments on the Draft EIS
- incorporate potential effects from the proposed land exchange between the USFS Superior National Forest and PolyMet

Public review of the scope of the land exchange ended on November 29, 2010. The Notice of Intent stated that the proposed land exchange would eliminate conflicts between the United States and private mineral ownership and consolidate land ownership to improve Superior National Forest management effectiveness and public access to federal lands. The proposed exchange is in accordance with Forest Service Strategic Plan Goals to provide and sustain long-term socioeconomic benefits to the American people, conserve open space, and sustain and enhance outdoor recreation activities.

The NorthMet mine site encompasses approximately 2,840 of the 6,650 acres of land proposed for exchange to private ownership. From a public use perspective, the remaining federal property consists of intermingled and inefficient ownership patterns.

The lands that would be received by the Superior National Forest consist of forest and wetland habitat as well as lake frontage. These lands would enhance public recreation opportunities and complement existing federal ownership by eliminating or reducing private holdings surrounded by Superior National Forest land.

The EIS Contractor and the Lead Agencies are making continued progress toward completion of the SDEIS. The SDEIS follows the Council on Environmental Quality (CEQ) recommended organization under the US National Environmental Policy Act and the Minnesota Environmental Policy Act content requirements:

- Chapter 1.0 – introduction
- Chapter 2.0 – describes the SDEIS development and scoping process
- Chapter 3.0 – describes the Proposed Action and alternatives
- Chapter 4.0 – summarizes the existing conditions
- Chapter 5.0 – presents the direct and indirect environmental consequences
- Chapter 6.0 – describes the cumulative effects on the surrounding environment
- Chapter 7.0 – compares alternatives
- Chapter 8.0 – lists other considerations
- Chapter 9.0 – is the list of preparers.

Once all aspects of environmental modelling, including quality assurance/quality control have been completed, the results will be incorporated into a preliminary SDEIS that will be available for review by

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the cooperating Agencies (including the EPA). Comments from the cooperating Agencies will be incorporated as appropriate, which will then be published for public review and comment. A final EIS will consider those comments.

20.1 Policy, Legal, and Regulatory Framework

The Policy, Legal and Regulatory Framework was described in Chapter 1 of the Draft EIS dated October 2009. This will be updated in the same Chapter of the Supplemental Draft EIS when it is published for public review.

The primary regulatory framework comprises the National Environmental Policy Act and the Minnesota Environmental Policy Act.

20.1.1 National Environmental Policy Act (NEPA)

NEPA requires that federal agencies consider the potential environmental consequences of proposed actions in their decision-making process. The law's intent is to protect, restore, or enhance the environment through well-informed federal decisions. The Council on Environmental Quality (CEQ) was established under NEPA for the purpose of implementing and overseeing federal policies as they relate to this process.

In 1978, the CEQ issued Regulations for Implementing the Procedural Provisions of NEPA. Section 102(2)(c) of NEPA mandates that the lead federal agency must prepare a "detailed statement for legislation and other major federal actions significantly affecting the quality of the human environment." Such projects include any actions under the jurisdiction of the federal government or subject to federal permits; actions requiring partial or complete federal funding; actions on federal lands or affecting federal facilities; continuing federal actions with effects on land or facilities; and new or revised federal rules, regulations, plans, or procedures. Any significant action with the potential for significant impacts requires the preparation of an EIS and a record of decision (ROD).

The USACE determined that the Project would require the preparation of an EIS in accordance with the requirements of NEPA and the CEQ regulations. To comply with other relevant environmental statutes, the decision-making process for the Proposed Action involves a thorough examination of pertinent environmental issues.

The USACE will use the Final EIS to develop the ROD for intent to issue a Section 404 Wetland Permit as needed for the Project to proceed.

Likewise, the USFS will use the Final EIS to develop the ROD for the proposed land exchange action.

20.1.2 Minnesota Environmental Policy Act (MEPA)

In addition to the NEPA process, Minnesota Statutes also require an environmental review of the Project. The MEPA environmental review process is a decision-making tool for state agencies. It informs the subsequent permitting and approval processes and describes mitigation measures that may be available. The MEPA process operates according to rules adopted by the EQB. However, the

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actual reviews are usually conducted by a local governmental unit or a state agency. The organization responsible for conducting the review is referred to as the Responsible Governmental Unit (RGU). The primary role of the EQB is to advise RGUs and state agencies on the proper procedures for environmental review and to monitor the effectiveness of the process in general. Because of its responsibility under Minnesota Rules for the review of all proposed mine projects, the MDNR is the RGU for the Project.

Minnesota Rules dictate that an EIS shall be prepared because the Project exceeds the threshold listed for construction of a new metallic mineral mining and processing facility. Under MEPA, the DEIS must be consistent with Minnesota Rules and the scoping determination.

The DNR will make an adequacy decision on the Final EIS, after which the Final EIS can be used to inform state permitting actions.

20.1.3 Land Exchange Requirements

Most of the public lands involved in the NorthMet Project were acquired by the United States under the authority of the Weeks Act of 1911. Other authorities that govern the land exchange between PolyMet and the United States include the the Federal Land Policy and Management Act of 1976, and the Federal Land Exchange Facilitation Act of 1988.

PolyMet plans to exchange surface rights with the United States under the Federal Land Policy and Management Act, which requires that a land exchange involves the transfer of equal valued land (if land values are not equal, the balance can be paid up to an amount of 25% of the land exchange value) and must also provide that the exchange preserves wetland functions with no net loss to the Federal estate and no increase in flood hazards to the non-Federal estate.

The proposed land exchange will leverage the 2004 Superior National Forest Land and Resource Management Plan (Forest Plan). The land exchange and associated current and future land use must be consistent with the conditions, goals, and guidelines outlined in the Forest Plan. Additionally, the USFS must analyze whether the land exchange meets the goals set forth in the USDA Forest Service Strategic Plan FY 2007-2012 Goals (Strategic Plan). The proposed land exchange would strive to meet four of the seven Strategic Goals: provide and sustain benefits to the American people; conserve open space; sustain and enhance outdoor recreation opportunities; and maintain basic management capabilities of the Forest Service by reducing land lines and mineral conflicts.

The proposed land exchange would be designed to be consistent with the remaining goals and objectives of the Forest Plan, in light of specific land classifications. The proposed non-federal lands for land exchange would need to be incorporated within the adjacent federal ownership and managed in accordance with the Forest Plan direction for the particular Management Area.

The Forest Supervisor, as the Responsible Official for the Superior National Forest, will decide in a ROD whether to proceed with the proposed land exchange. The EIS will serve as the basis for the ROD.

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20.2 Baseline Studies

Extensive baseline studies were described in Section 4 of the Draft EIS. This will be updated in the same Chapter of the Supplemental Draft EIS when it is published for public review.

These studies (Table 20-1) include data on local lakes and rivers that extend to the 1930s in some cases and cover: meteorological conditions, ground and surface water, wetlands, hydrology, vegetation (types, invasive non-native plants, and threatened and endangered species), wildlife (listed species and species of special concern, species of greatest conservation need and regionally sensitive species), aquatic species (surface water habitat, special status fish and macroinvertebrates), air quality, noise, socioeconomics, recreational and visual resources, and wilderness and other special designation areas (established and candidate research natural areas, unique biological areas, national historic landmarks, scenic byway, national recreation trail).

Table 20-1: Baseline Environmental and Environmental Engineering Studies

Winter Wildlife & Wildlife Habitat Survey	Completed
Summer Wildlife & Wildlife Survey	Completed
Wetland Delineation and Classification Survey	Completed
Threatened & Endangered Plant Species Surveys	Completed
Canada Lynx Study	Completed
Stream and Wetland Biological Surveys (fish and aquatic marco-invertebrates)	Completed
Stream Classification of Partridge River and Trimble Creek	Completed
Freshwater Mussel Survey in Trimble Creek and Embarrass Rivers	Completed
Soil Mapping	Completed
Background Surface Water Quality Monitoring in Partridge and Embarrass Rivers	Phase I Completed; Phase II ongoing indefinitely
Compilation of Existing Surface Water Quality Data	Completed
Hydrogeologic Investigation for the PolyMet – NorthMet Mine Site	Completed
Scoping Cultural Resources Assessment	Completed
Phase I Archaeological Survey	Completed
Wetland Hydrology Study	Indefinite monitoring

20.3 Environmental Issues

20.3.1 Comments on the Draft EIS

Public and agency comments on the Draft EIS were collected during the 90-day comment period. Submissions came from stakeholders including government agencies (federal, state, and local), the Bands, local businesses, non-government organizations, private individuals, and the Project proponent. A total of approximately 3,800 comment submissions were received.

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On February 18, 2010, the Co-lead Agencies received a comment letter from the EPA. In the absence of an Agency Preferred Alternative that described a specific project plan that met applicable state and federal regulations, the EPA reviewed the least environmentally acceptable plans and determined that the Project could result in detrimental impacts to water resources, including wetlands. The EPA also believed that impacts to water resources were underestimated and that the Project could have long-term discharges.

The EPA recommended preparation of a Supplemental Draft EIS to assess the impact of a specific project plan and respond to comments on the Draft EIS. The EPA became a co-operating Agency engaged in the preparation and review of the SDEIS.

20.3.2 MPCA Guidance Regarding Wild Rice

In June 2010, the MPCA issued staff recommendations on the site-specific application of a Minnesota standard for wild rice in the Partridge and Embarrass River systems. This guidance applies a water quality standard of 10 mg/L of sulphate to waters used for the production of wild rice during periods when rice may be susceptible to damage by high sulphate levels. The recommendations were updated in March and June 2011, to discuss the variations in conditions from year to year and the travel and residence time of sulphate releases. The MPCA guidance also included tailings basin performance requirements regarding seepage discharges, limitations to sulphate contributions in surface waters, and monitoring requirements. Also addressed were comments and concerns, which MPCA received from interested parties.

PolyMet has undertaken extensive testwork to demonstrate that the NorthMet Project can meet these standards, which will be reflected in the Supplemental Draft EIS.

20.3.3 Other Issues

During the scoping for the proposed project, several issues were identified as possibly resulting in significant impacts, which would require information beyond what was included in the scoping EAW. Of specific interest was additional information related to fish and wildlife resources, threatened and endangered species, physical impacts on water resources, water appropriations, surface water runoff and erosion/sedimentation, waste water, solid waste, cumulative impacts, stockpile cover types, point and non-point source air emissions, noise, archaeology, visibility, compatibility with land use plans and regulations, infrastructure, asbestiform fibers, and the 1854 Ceded Territory.

Subsequent to publication of the Draft EIS, additional issues were identified for further development and discussion in the SDEIS. These included air impacts, wetland impacts, geotechnical stability of the tailings basin, socioeconomics, and water resources impacts. As previously discussed, topic-focused workgroups were assembled from members of the Co-lead and Cooperating Agencies to further explore these issues.

In addition to addressing issues identified during scoping, the SDEIS will also address issues that have been identified as the understanding of the potential impacts of the Project has evolved.

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20.4 Closure Plan

Closure plans for the NorthMet Project, including both the mine site and reclamation of the Erie Plant site were described in Chapter 3 of the Draft EIS and will be updated in the same Chapter of the Supplemental Draft EIS when it is published for public review.

PolyMet plans to build and operate the NorthMet project in a manner that will facilitate concurrent reclamation, in order to minimize the portion of the Project that will need to be reclaimed at closure. In addition to a detailed closure plan, Minnesota Rules require the Company to submit an annual plan that identifies reclamation activities if operations ceased in the following year.

All buildings and structures will be removed and foundations razed, covered with soil and vegetated. Most demolition waste will be disposed in the existing landfill on site, but some that may have elevated contaminants will be handled and disposed separately.

During the last ten years of operations, the East Pit will be backfilled concurrently with mining of the West Pit. At the end of operations, the backfilled East Pit will be flooded, overflowing into the West Pit.

The mine walls will be sloped and revegetated and selective areas of the pit walls will be covered. Pit perimeter fencing will be installed and stockpiles will be covered.

These items are covered in detailed plans covering:

- Demolition of structures (buildings, sanitary systems, wells, power lines, pipelines and tanks) including waste disposal.
- Reclamation of the Mine Site – mine pit reclamation, stockpile reclamation, reclamation of water management systems, building areas, roads and parking lots, and removal of railroad tracks and culverts.
- Reclamation of the Plant Site – FTB reclamation, HRF reclamation, reclamation of water management systems, building areas, roads and parking lots, and removal of railroad tracks and culverts.
- Remediation of legacy Areas of Concern (AOCs) and ongoing mitigation of water quality at the Mining Area 5N and the Tailings Basin as well as plans to investigate for potential releases at the conclusion of operations.
- Ongoing monitoring and maintenance for the existing solid waste disposal facilities, reclamation maintenance.

20.4.1 Financial Assurance

Minnesota Rules require financial assurance instruments to cover the estimated cost of reclamation be submitted and approved by the DNR before a Permit to Mine can be issued.

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Financial assurance must cover the reclamation and post reclamation activities. The plan and the amount are updated each year to reflect the work completed and the plan in the event that the Project closed during the following year. The instruments must be bankruptcy proof.

20.5 Permitting

Prior to construction and operation of the NorthMet Project, PolyMet will require permits from several federal and state agencies. The final EIS will incorporate comments, after which a subsequent Adequacy Decision by the MDNR and Record of Decision by the federal co-lead agencies are necessary before the land exchange can occur and various permits required to construct and operate the Project can be issued. Including:

20.5.1 Government Permits and Approvals for the Project

US Army Corps of Engineers

- Section 404 Individual Permit
- Section 106 Consultation

US Fish and Wildlife Service

- Section 7 Endangered Species Act (ESA) Consultation
- US Forest Service
- Land Exchange Approval

Minnesota Department of Natural Resources

- Permit to Mine
- Water Appropriations Permit
- Dam Safety Permit
- Permit for Work in Public Waters
- Wetland Replacement Plan approval under Wetland Conservation Act

Minnesota Pollution Control Agency

- Section 401 Water Quality Certification/Waiver
- National Pollutant Discharge Elimination System and State Disposal System (NPDES/SDS) Permits
- Solid Waste Permit
- Air Emissions Permit
- General Storage Tank Permit

Minnesota Department of Health

- Radioactive Material Registration (for measuring instruments)
- Permit for Non-Community Public Water Supply System
- Permit for Public On-site Sewage Disposal System

City of Hoyt Lakes

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- Zoning Permit

City of Babbitt

- Building Permit

St Louis County

- Zoning Permit

Zoning PermitMinnesota has extensive experience of permitting and overseeing operation of large-scale iron ore mines. However, PolyMet is the first company to seek permits to construct and operate a copper-nickel mine. As such, the NorthMet Project is defining how established state and federal regulations will be applied to non-ferrous mines.

20.6 Considerations of Social and Community Impacts

Chapter 4.10 of the Draft EIS included extensive discussion of social and community impacts, which will be updated in the Supplemental Draft EIS when it is published for public review.

The Draft EIS observes that the NorthMet Project would have some effect throughout the eastern portion of the Mesabi Iron Range, including the cities of Aurora, Babbit, Hoyt Lakes, Tower, Ely, and Soudan. It also projects some indirect impacts on urban centers such as Duluth and Minneapolis.

St Louis County in general, and the Eastern Range in particular, have seen declining and aging populations – between 1980 and 2004, the population of the County declined by 11% to 199,000 and the population of Hoyt Lakes declined by 38% to 1,961. In the 2000 US Census, the average age of the Eastern Range cities was 44.2 years, compared with 39 for all of St Louis County and an average of 35 years in Minnesota.

Median family income in the Eastern Range cities was $37,443 compared with $47,134 in St Louis County and $56,874 in the state as a whole. Of those over 16 in the Eastern Range, 55.3% were in the Labour force, compared with 62.7% in St Louis County and 71.2% in Minnesota.

According to the Draft EIS, employment in mining declined from 10,973, or 15% of the total 75,104 in St Louis County in 1980 to 5,326, or 7% of the total of 79,650 in 1990. By 2004, mining had declined further, to 2,752 or just 3% of the total of 92,668, ranking twelfth behind health care and social assistance (22%), retail (13%), accommodation and food (10%), education (8%), public administration (6%), manufacturing (6%), construction (4%), finance and insurance (4%), transportation and warehousing (4%), administrative waste services (3%), and other services (3%).

While St Louis Country accounted for just 3.6% of all jobs in Minnesota in 2004, it accounted for 53.6% of the mining jobs.

The Draft EIS also reported that, based on the 2000 US Census, there were 95,800 housing units in the Eastern Range Cities of which 10% were vacant.

Local infrastructure was designed to support these communities when they were larger. For example, the waste water treatment facility in Babbitt has a capacity of 500,000 gal/d with a daily load of

200,000 – 300,000 gal/d. The similar facility in Hoyt Lakes has the capacity to treat 1.2 Mgal/d, with maximum daily load of 670,000 gal/d and average daily loads of 250,000 to 300,000 gal/d.

As part of its input to the Supplemental Draft EIS, PolyMet engaged the University of Minnesota Duluth Labovitz School of Business and Economics' Bureau of Business and Economic Research (BBER) to assess the economic impact of the NorthMet Project on St Louis County, MN.

The BBER study used IMPLAN version 3.0 economic modelling and impact software created by MIG, Inc. The report estimates that, in addition to the 360 direct, full-time jobs, the NorthMet Project will create 631 indirect and induced jobs and contribute approximately $515 million directly and indirectly into the local economy each year.

While the local communities will be able to absorb likely levels of inward migration, the impact on employment levels and the overall local economies could be significant.

20.7 Discussion on Risks to Mineral Resources and Mineral Reserves

The mine plan being considered in the SDEIS contemplates mining approximately 234 million tons of ore over a twenty-year mine life. Any material change to that plan will require environmental review and any change resulting in a material change in the environmental impact will require further permitting.

Economic development of any mineral resources outside the mine plan will be dependent on additional environmental review and permitting.

20.8 Comments on Section 20

Environmental review and permitting is, perhaps, the biggest challenge facing any mining project in the US. PolyMet is well advanced in the process and actively engaged with relevant state and federal agencies. The project is well supported in the local community and will have important socio-economic benefits.

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21 CAPITAL AND OPERATING COSTS

The Technical Report on the Results of a Definitive Feasibility Study of the NorthMet Project that was published in October 2006 detailed the capital costs for the Project to produce copper cathode as well as a mixed Ni/Co hydroxide and PGM precipitate. The process changes described in Section 17 of this update to the Technical Report reflect continued metallurgical process and other project improvements as well as improved environmental controls that are being incorporated into the Project.

PolyMet's last formal update of project scope and costs was in a press release in May 2008 – when total project costs for the two-autoclave plus SX-EW circuit were estimated to be $602 million. Of that total, approximately $127 million was attributed to the second autoclave and the copper SX-EW circuit.

In February 2011, PolyMet reported further refinement of the Project plans, which the Company plans to build the Project in two phases:

- Phase I: produce and market concentrates containing copper, nickel, cobalt and precious metals
- Phase II: process the nickel concentrate through a single autoclave, resulting in production and sale of high grade copper concentrate, value added nickel-cobalt hydroxide, and precious metals precipitate products.

The changes reflect continued metallurgical process and other project improvements as well as improved environmental controls that are being incorporated into the SDEIS. The analysis is based on likely metal market conditions. The advantages, compared with the earlier plan, include a better return on capital investment, reduced financial risk, lower energy consumption, and reduced waste disposal and emissions at site.

Of the total $602 million capital cost estimated in the DFS Update, approximately $127 million was attributed to the second autoclave and the copper circuit eliminated in the 2011 revision.

PolyMet plans to provide a detailed project update when the Project development plans now being analyzed in the SDEIS are finalized. This detailed project update will include revised mine plans, process and project improvements, and will incorporate the latest environmental controls and will conform to the Project that is being analyzed in the Supplemental Draft EIS and which PolyMet is permitting.

21.1 DFS Capital Cost Estimates

Capital cost estimates for the 2006 DFS were generated to an overall level of accuracy of -5% to +15% in order to provide a confident basis for project financing decisions. The following section summarises the basis and methodology for developing capital cost estimates to the required level of accuracy and

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confidence. Capital cost estimates are prepared with an April 2006 cost base without application of escalation and exclude Minnesota state sales tax.

21.1.1 Basis of Capital Cost Estimate

The capital cost estimate was developed on the basis of frozen design criteria and flowsheets and includes an initial and sustaining life of mine capital schedule. Components of the capital cost include:

- Initial capital is that required during the pre-production construction period necessary to bring the operation into production and includes EPCM, owner's costs, first fills, insurance, and commissioning costs.
- Sustaining capital includes replacement of capital plant and equipment and expansion or extension of facilities required to maintain operations, e.g., progressive construction of additional hydrometallurgical residue cells, major rail replacement programs, extension of the impermeable base of waste rock stockpiles, etc.
- The capital estimate is broken down by facilities, equipment items, freight, direct labour, construction, contractors' costs, and spares. Most of the equipment, services and materials will be sourced within the USA and therefore foreign exchange rate variations are unlikely to be significant.
- Contingency was assessed by Bateman using a sophisticated Monte Carlo risk assessment method that analysed key areas of the cost estimate separately and allocated contingency according to assessed risk and commensurate with estimate accuracy.
- State sales tax was excluded on the assumption that it would be recoverable.

The following summarises the basis on which the major components of the capital cost estimate were prepared.

- Mine Pre-production Costs: An estimate was developed by PolyMet from written quotes from four prospective mining contractors. Pre-production mining costs included mobilisation, preparation of site access and construction of initial haul roads, pre-stripping and initial waste removal in preparation for ramping-up to full mill production during Year 1. Material movement quantities were based on a production schedule developed by AMDAD.
- Waste Rock Stockpile Construction: In the absence of close spaced overburden drilling and sampling, excavation and fill volumes were estimated from an overburden thickness model based on drill hole logs, geophysical soundings and a limited number of test pits which provided the basis for assumptions relating to soil types and characterisation. For environmental reasons waste rock stockpiles are required to be constructed with impermeable bases the construction costs of which were estimated from a combination of local contract earthmoving rates and recent project experience elsewhere.
- Mine Power Supply: For costing purposes it was assumed the power utility will provide at no cost the tap and connection to 138 kV transmission line and the main mine site step

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down transformer. The cost of constructing and periodically extending the 4160 V mine site wooden pole mounted, power reticulation line was based on a written quote from local power utility, Minnesota Power.

- Railroad: Railroad costs were estimated by Duluth-based KOA who specialise in railroad engineering and, therefore, were able to call upon reliable, recent local costs of services, construction and materials (rail, ties, etc.) Refurbishment costs for existing track were based on a detailed survey of its condition using recent local rates for similar work elsewhere.
- Rail Transfer Hopper: Design by KOA was closely based on two approximately similar loading hoppers built for LTVSMC in the mid- to late-1990s. Current Iron Range construction labour rates were used with materials costs estimated against an engineered materials take-off. Costs for overhauling and refurbishing salvaged mechanical and hydraulic equipment were provided by original equipment manufacturers.
- Mine and Railroad Infrastructure: Refurbishment costs were based on preliminary architectural and engineering drawings with application of standard unit rates for refurbishment of offices, change houses and personnel facilities. Reactivation costs of Area 1 (mine equipment) and Area 2 Shops (railroad rolling stock maintenance) workshops were estimated from a combination of vendor/supplier quotes, allowances and standard rates for similar work elsewhere.
- Mine to Waste Water Treatment Plant (WWTP) Pipeline: Capital cost was developed from a quote for spiral-wound, steel pipe laid above ground with a factored allowance for installation. Costs for refurbishing existing pumps were supplied by a pump vendor.
- HV Electrical Sub-Station: Although parts of the sub-station remained active since closure, re-activation costs were based on LTVSMC operating and maintenance records, inspections by heavy current electrical contractors and engineers of the local electrical supply utility, Minnesota Power.

Ore Beneficiation Plant: Reactivation costs were based on:

- detailed plant condition surveys
- assessment of operating and maintenance records to determine remaining life in crusher and mill wear materials and liners
- vendor assessment of process control system hardware and I/O points
- vendor quotes for dust extraction system equipment
- vendor quotes for flotation equipment
- test starting of selected, representative electric motors to confirm re-start and start-up failure assumptions.

Hydrometallurgical Plant: Table 21-1 summarizes the basis of new plant capital cost estimates.

Flotation Tailings Basin – Seepage Recovery System Upgrade: Capital estimate was developed by Barr Engineering and based on recent, similar project experience and standard unit costs for pipe and earthworks. Tailings piping will consist of a combination of new and salvaged steel pipe and refurbishment costs of existing tailings pumps were provided by a local pump vendor`.

Hydromet Residue Cells: Excavation costs were developed by PolyMet from local earthmoving contractor unit rates with liner acquisition and placement costs derived from recent, local experience of constructing land fill and taconite tailings disposal facilities.

Table 21-1: Basis of New Plant DFS Capital Estimates

Process	Requirements
Process flowsheets	Optimized
Bench scale tests	Essential
Pilot scale tests	Recommended and completed
Energy and material balances	Optimized
Equipment List	Finalized
Facilities Design	
Plant capacity	Optimised
Equipment selection	Optimised
General arrangements - mechanical	Preliminary
General arrangements - structural	Preliminary
General arrangements - other	Outline
Piping	Based on single line drawings
Electrical	Based on single line drawings
Specifications	General
Basis for Capital Cost Estimate	
Vendor quotations	Multiple, preferably written
Civils	Derived from drawings
Mechanical and piping	Approximate quantities
Structural work	Derived from material take-off
Instrumentation	Derived from material take-off
Electrical work	Derived from material take-off
Indirect costs	Calculated
Project program/schedule	Critical path network
Expected contingency range	10-15%

Limestone Stockpiling and Handling System: cost based on preliminary engineering and materials take-off. Allowances for re-use of some components were also included.

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Fresh Water Reticulation System: Costs were based on field examination and engineered estimates of refurbishment requirements. In the case of the fresh water pipeline from Colby Lake, historical maintenance records were used to estimate the amount of plastic, internal re-sleeving required to return the pipeline to operable condition.

Plant Site Infrastructure: Costs were based on field inspections and an assessment of historical maintenance and operating records. Where equipment or component refurbishment or replacement was necessary costs were derived from vendor and original equipment manufacturers quotes.

The capital cost schedule contains estimates for all environmental aspects of the study that resulted from technical evaluations and studies undertaken by Barr Engineering, SRK, Golder Associates and others.

The overall estimated initial and sustaining capital cost for developing the Project is shown in Table 21-2. Costs are estimated at a base date of April 2006 and exclude escalation. Equipment import duties, freight and insurance are included where appropriate but state sales tax (at 6%), which is recoverable, is excluded.

Table 21-2: Summary of Initial and Sustaining Capital Costs

	Initial **(US$'000s)**	Sustaining **(US$'000s)**
Direct Costs		
Mining & Mine site Infrastructure	18,489	24,354
Railroad	8,464	33,344
Erie Plant Beneficiation Plant	62,992	0
Hydrometallurgical Process Plant	191,996	3,170
Tailings & Residue Disposal	3,134	7,949
Total Direct Costs	285,075	68,817
Contingency	27,070	
Indirect Costs	67,495	2,970
Total Project Capital	379,640	71,787

21.2 DFS Update

In May 2008, PolyMet reported the results of an update to the DFS, which included:

- the sale of concentrate during the construction and commissioning of new metallurgical facilities resulting in a shorter pre-production construction period (under twelve months) and reduced capital costs prior to first revenues ($312 million vs. $380 million)

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- mine plans (based on copper at $1.25/lb) reflect the increase in reserves and decrease in stripping ratio reported on September 26, 2007, the use of 240-ton trucks, and owner versus contract mine operations.

21.2.1 Basis of Estimate

The updated capital cost estimate is based on the original DFS, which was base-dated April 2006. For the updated DFS, capital costs have been captured, generally as follows:

- As part of project set-up and baselining, a work breakdown structure (WBS) was established for the Project. The project WBS is similar to the WBS used in the DFS but has some differences including the differentiation between Phase 1 and Phase 2 costs.
- Around 5,000 DFS cost records were entered into PolyMet's project cost control system
- (PRISM) to establish the DFS baseline budget.
- Costs for the complete scope of the DFS were escalated against relevant industry cost indices to bring them from April 2006 costs to February 2008, that being the most recent month for which cost indices were published at the time.
- Where the scope has changed since the DFS or where there had been developments that provided better scope, quantity or cost definition, costs were re-estimated.
- The revised estimate aims to be as complete as possible. As well as escalating DFS costs and capturing scope changes and growth, it includes:

Some costs that were not captured in the DFS captures costs that have been expended on the Project since the DFS as well as "costs to come."

21.2.2 Labour Assumptions

The original estimate for the DFS included approximately $75 million for construction labour costs. While the stated base date of the DFS is April 2006, most of the craft labour rates for northern Minnesota were renegotiated in May and June 2006 and it has been determined that Bateman used the updated June 2006 rates in preparing the DFS costs.

Table 21-3 compares hourly labour rates (including fringes) for various crafts between June 2006 and for February 2008. Jamar Company submitted yhe rates for June 2006 for use in the DFS on October 27, 2006. The craft labour rates for February 2008 were extracted from the website for "Davis-Bacon wage determinations" for St Louis County, Minnesota.

Table 21-3: Comparison of Hourly Labour Rates

Craft	Jun 2006 ($)	Feb 2008 ($)	Change (%)
Asbestos Worker/Insulator	44.36	45.91	3.49

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Craft	Jun 2006 ($)	Feb 2008 ($)	Change (%)
Boilermaker	46.47	48.32	3.98
Carpenter	35.44	36.79	3.81
Cement Finisher	35.05	36.45	3.99
Electrician (API)	46.19	48.03	3.98
Ironworker	43.13	45.11	4.59
Laborer	31.78	33.33	4.88
Millwright	38.30	39.70	3.66
Operator	43.19	45.03	4.26
Painter	37.68	39.08	3.72
Plumber/Pipefitter	44.82	46.62	4.02
Roofer	31.59	32.75	3.67
Sheet Metal Worker	42.72	44.41	3.96

Table 21-3 shows most of the craft rates have increased by approximately 4% from June 2006 to February 2008. Where possible, the escalation rates for each craft was applied directly to the labour estimate for the corresponding discipline (i.e., the rate for "cement mason" is applied to concrete accounts, the rate for "pipefitters" is applied to piping accounts, etc.). Due to the relative equality of the escalation rates for each craft, it has been determined that there is no appreciable change to the final escalation calculation by weighting the effects of typical crew mixes for each discipline.

21.2.3 Material Costs

Using the Bureau of Labour and Statistics Producer Prices Index Industry database, cost indices for the period between April 2006 and February 2008 were extracted for numerous materials and equipment groups. The cost indices were linked to the cost elements in the PolyMet cost database (PRISM) by Commodity Code. On the basis of the method used, the average escalation of materials and equipment for the original DFS scope is 10.6%.

21.2.4 Contingency

Contingency is an estimate provision to account for items, conditions or events for which the state, occurrence or effect is uncertain and that experience shows will likely result in additional cost. Contingency provisions are sometimes supported by statistical analysis using Monte Carlos simulations. In the case of this estimate:

Contingency means Estimating Accuracy Allowance (EAA), which is a provision to account for uncertainty related to estimated quantities and cost (rates) that have been used in the estimate.

There is no provision for unplanned (future) risks, sometimes referred to as Risk Contingency. That is, there is no provision in the estimate for any deviation from the Project as currently planned, including changes in scope, timing, quantities or costs.

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A Monte Carlos has not been run on the data.

Estimating Accuracy Allowance has been applied at 10% of all other costs in the estimate except Owners Additional Costs.

21.2.5 Mine Capital Costs

Mine Pre-development (including Rail, Flotation Tailings & Hydromet Residues)

The following notes describe the basis upon which capital cost estimate for the mine, railroad tailings and residue facilities were updated for the Updated DFS. The notes refer to Table 21-4, which shows the initial capital costs for the areas listed.

The timing of costs reflects the year in which an action, piece of equipment or construction is required; it does not reflect when commitment is required to ensure timely delivery.

Table 21-4: Initial Capital Costs (US$'000)

Estimate Item	Year 1	Year 2
Haul road construction*	10,559	
Dunka Road upgrade incl. 2 road/rail crossings	1,132	
Stockpile construction - base, liners & sumps	16,582	3,865
Dikes, Perimeter Ditches, Stormwater Pond	4,763	
Process/contact water collection piping	3,523	1,174
Mine Area pre-stripping - overburden	8,738	
Construction Quality Testing	392	78
Site geotechnical drilling*	669	317
Pre-production drilling - grid & blasthole correlation*	1,230	
Pre-production drilling - East Pit footwall definition	512	
Royalty on State taconite waste rock @ $0.50/cu yd	510	
Total: Mine Pre-production Development	48,637	5,435
Waste Water Treatment Facility	4,553	
Central Pumping Station	1,781	
Treated Water Pipeline	2,303	
Total: WWTF, CPS & Treated Water Pipeline	8,637	
Railroad Construction & Refurbishment	9,892	216
Mine Site Infrastructure & Facilities	734	
Mining & Railroad Maintenance & Engineering	2,973	
Mine Site Power Supply & Reticulation	4,705	
Mine Lands Acquisition	3,300	
Tailings Basin	3,097	
Hydromet Residue Cell Construction	8,751	7,151

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21.2.6 Mine Pre-production Development

All pre-production earthmoving, construction of stockpile foundations and liners, construction of ditches, dikes, run-off collection sumps, water conveyance and treatment arrangements at the mine site will be carried out by contractor. Costs are based on the March 2008 revised proposal by Ames Construction, Inc. Ames' proposal was submitted in response to a detailed scope of work prepared on behalf of PolyMet by Barr Engineering,

Inc. and unless otherwise noted, the Ames' estimates have been used as presented. Geotechnical drilling which will be required ahead of construction and most earthmoving

will be carried out by a specialist drilling contractor in conjunction with the principal earthmoving contractor. Barr prepared estimates of geotechnical drilling costs based on recent actual drilling costs and an estimate of the number of holes required for final design of stockpile liner foundations, access and haul roads, dikes and other mine site structures.

Contractors' scope for costing purposes has been limited to clearing and stripping overburden from the East Pit only. Site clearance and overburden removal from the West and Central areas of the mine may ultimately be carried out by the same contractor as used for East Pit pre-production development. For costing purposes West and Central area overburden removal costs have been included as mine sustaining capital. Similar unit rates were assumed for both contractor pre-production stripping and sustaining capital overburden stripping. (Note it is likely that any overburden stripping at NorthMet will be carried out by specialist contractor rather than by the owner because of the unsuitability of the mining fleet for operating on overburden).

Ames estimates reflect a diesel fuel price of US$3.00/gal. This is consistent with other areas of the Updated DFS.

Dunka Road upgrading costs include provision for the construction of two ground level rail track crossings suitable for use by heavy mine equipment.

The cost of stockpile foundations and liners has been distributed over the pre-production period and the first five years of operations to reflect the progressive manner in which stockpiles will be constructed and then extended.

Provision for the cost of stockpile covers is included in Closure Costs.

Costs for process and contact water collection piping connecting the various stockpile sumps, run-off collection sumps and settling ponds with the WWTF have been distributed over Years -1 and +1.

Although the majority of construction quality testing will be required during the pre- production development period, provision has been made to distribute testing to cover all construction and development work to the end of Year 6.

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Heavy mine equipment will have to cross County Road 666 in order to access the Area 1 workshops. Barr (Hibbing) has estimated the cost of constructing a crossing from information provided by Mesabi Bituminous with additional provision for traffic control lights, area lighting and appropriate signage.

Geotechnical drilling and testwork will be required prior to construction of stockpile foundations and other mine infrastructure and facilities. Barr has recommended 430 holes be drilled during Years -1 (315 holes) and +1 (115 holes). Rotosonic and standard penetration test (SPT) drilling have been recommended as suitable methods of collecting overburden samples for testing and characterization though Rotosonic is approximately twice as costly per ft than SPT. Based on field experience with a Rotosonic rig during January 2008 it was assumed that SPT would be adequate for the majority of drilling and sampling purposes. For the purposes of this estimate it has been assumed that 80% of the required holes can be completed using SPT with the balance by Rotosonic.

Two campaigns of pre-production diamond drilling are required during mine pre-production development. A major campaign, consisting of 128 holes to an average depth of 150 ft on the same spacing as blast hole drilling, is required for ore grade control and comparison of diamond drill and blasthole sampling. A second campaign comprising 32 holes is required to better define the East Pit footwall location to minimize the amount of Virginia Formation to be mined. Drilling costs of US$50/ft are un-escalated, based on recent exploration drilling performance and include all drilling costs plus core logging, sample preparation activities, sample transport and laboratory analysis.

Capital cost of a mine site water treatment facility is based on the use of a portable, modular, treatment facility during the first three years of mine life during which time the characteristics of a permanent treatment facility will be determined. Installation and operation of this portable facility during the period up to delivery of first ore to the primary crusher have been treated as initial capital; thereafter its cost of operation has been treated as an operating cost. Barr Engineering developed costs for renting and operating the temporary facility from written estimates provided by several suppliers of portable water treatment plants of which GE (Water) is the preferred supplier. The cost of the temporary facility includes an allowance for providing temporary diesel generated power until the mine power is installed along with construction of a temporary hard standing pad and access track. Use of a portable, temporary facility ensures that availability of water treatment facilities does not delay start of hard rock mining. In addition, use of a temporary, portable plant allows the cost of constructing the permanent facility to be deferred.

The permanent WWTF will be constructed in Year 4 and the cost is based on the Ames quote together with an estimate by Barr of the cost of furnishing and installing all water treatment equipment and appropriate control systems.

Both the temporary and the permanent WWTF will utilize the same water collection system and flow equalization ponds. These will be constructed by the mine pre-production development contractor during the early part of Year -1 (2009) such that they are operational before the start of hard rock drilling and blasting.

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21.2.7 Railroad

Track construction and refurbishment costs are based on the original DFS estimates prepared by Krech Ojard and subsequently updated by them in February 2008.

Not contemplated in the DFS is an additional rail spur at the mine site to allow trains to access to the ore surge pile loading ramp without having to pass under the transfer hopper loading chute. An estimator's allowance has been used for this cost.

As was the case for the DFS, this estimate assumes that Owners crews will perform minor, routine track maintenance and that major maintenance such as rail grinding, tie replacement campaigns and rail replacement will be outsourced and treated as sustaining capital.

For the DFS Krech Ojard estimated a cost of US$10,000 to return each ore car to initial service. Thereafter each car would be rebuilt at a KOA estimated cost of US$25,000 which was treated as sustaining capital. These estimates were based on visual inspection of the rail car fleet. Subsequent (post-DFS) inspection of ore cars by another group of railroad specialists produced an unlikely revised estimate of US$1,700 per car for the initial return to service. For costing purposes, a return to service cost of US$6,000 per car was assumed. There was no change to the DFS estimate of US$25,000 for car re-build.

Krech Ojard updated the rail transfer facility construction cost estimate in March 2008 to include price escalation and modification of the DFS design to reduce the height of the main retaining wall that parallels the rail track.

For the DFS Krech Ojard determined that 30-car trains each pulled by two conventional 3,000hp mainline locomotives would be required. Subsequent re-evaluation has recommended the use of three trains comprising one 2,100hp multiple generator set locomotive pulling between 15 and 18 ore cars. Capital and operating costs are now based on maintaining 4 unit trains of 18 cars each (one unit being held as spare).

DFS costs were based on 30-car trains and assumed all 120-ore cars owned by PolyMet would be returned to service and subsequently rebuilt. Introduction of reduced length trains requires only 72 ore cars to be returned to service with significant cost saving.

Locomotive leasing costs are based on quotes for multiple generator set units, which offer significant emissions and fuel consumption reductions compared with the SD40-2 or -3 standard units used for the DFS.

21.2.8 Mine Site Infrastructure & Facilities

Costs for the relocation and erection of structures to serve as a field service facility and a field re-fuelling facility are based on DFS estimates updated in March 2008.

Not included in the DFS is provision of a fibre optic data link between the mine site and the Area 2 mine operations offices and Area 2 and the plant site. The current estimate assumes shared use of a fibre optic link to be installed by Minnesota Power between their 138kV mine site sub-station and the

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process plant with the cost of an extra 3,500 ft to connect the Area 2 office and the mine WWTF to the main fibre optic cable. The cost of installation is based on an estimate of US$9.40/ft by Minnesota Power and includes installation and appropriate hardware at each end of the cable.

The cost of re-surfacing the asphalt road between the Main Gate and Area 2 offices was estimated by Barr from recent actual costs for similar work by Mesabi Bituminous, Inc.

21.2.9 Mining & Railroad Maintenance & Engineering

Area 1 Shop refurbishment costs are based on updated DFS estimates by Krech Ojard. Area 2 facility upgrade costs are based on updated DFS estimates by Krech Ojard.

Cost of refurbishing the existing Area 2 locomotive refuelling and service facility is based on the use of outsourced third party refuelling direct from road tanker without the use of fixed, diesel fuel storage tanks.

The DFS assumed that Area 2 Shop would be refurbished and equipped to allow maintenance of ore cars. By re-arranging a part of the hydromet reagent storage facilities the original locomotive maintenance shops located within the main General Shop at the plant site became available and will now be refurbished for ore car maintenance and repair.

Because the General Shop is in good condition an unsupported cost provision of US$60,000 has been allowed for the minimal work required.

Estimates for the mine dispatch system range from US$2.5 million for a Modular Mining system to about US$900,000 for a Wenco system. For purposes of this estimate a value of US$1,500,000 has been used and is assumed to include hardware, software and interfacing units mounted on mining equipment.

The DFS estimate for a mine radio communication system has been updated and used herein.

21.2.10 Mine Site Power Supply and Reticulation

The cost of constructing a single circuit, wooden pole mounted conductor is based on a recent quote from Lake County Construction, a subsidiary of Lake County Power, of US$30/ft including placement of wooden poles, aluminium conductor and insulators.

In terms of an agreement, PolyMet is required to make periodic payments to Minnesota Power (MP) for the design and construction of the main 138 kV – 13.8 kV step down sub- station near the mine site. It is assumed an advance payment will be required during the per- production period (year -1) to enable MP to complete design and ordering of equipment for this sub-station.

21.2.11 Mine Equipment – Lease Costs

For costing purposes, it was assumed major items of mining equipment will be acquired under operating or "tax" leases. Estimates of operating lease costs were obtained from all manufacturers or vendors of major equipment and are current for the first quarter 2008.

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Using vendor/manufacturer quotes Wardrop prepared a detailed, life of mine equipment leasing schedule, which reflected the probable reality of an operating mine. Lease terms offered are generally for a 60-month term with the option to replace the equipment at the expiry of the lease or to purchase the equipment at a residual value. Thus, in the case of haul trucks and excavators which would probably have economic life remaining after expiry of the standard lease term, it was assumed that these items were purchased and operated to retirement after which new units would be leased. Thus, in the cost summary tables presented elsewhere equipment lease costs are actually a combination of lease and buy-out costs.

Capital purchase costs were obtained for equipment and items that would not normally be leased. These have been separated from equipment leasing costs and are accounted as capital. Examples of capital equipment include haul truck and front end loader (FEL) tires, blast hole drill strings, trailing power cables, spare truck trays/boxes, spare excavator and FEL buckets and small equipment such as skid steer loaders, small FELs and some service vehicles.

21.2.12 Post Production Start

Provision has been made for diamond drilling for further definition of the Magenta Zone and ongoing reserve replacement once in full operation. Drilling costs are based on unescalated all-up, exploration diamond drilling costs of US$50/ft and are considered sustaining capital.

21.2.13 Tailings Basin

The cost of installing the proposed seepage collection system designed by Barr Engineering is based on the revised bid prepared by Ames Construction.

21.2.14 Hydromet Residue Cells

Based on the assumption that a market will be found within the first three years for the synthetic gypsum component of the residue stream, it will only be necessary to construct two residue storage cells. The first will operate for five years and take the full residue stream while the second will have capacity to accommodate the reduced residue stream over the remainder of mine life. Because two construction seasons will be required earthworks for the first cell will start in Year -1.

Cost estimates to construct hydromet cells are based on a recently updated quote prepared by Ames Construction. The Ames quote covers construction of the first cell and the initial lift of the second cell. Subsequent costs for constructing the remaining cells are based on Ames' quote.

It was assumed that construction of a second lift will occur during the third year of the first cell's operating life.

An estimate developed by Barr Engineering of $11 million has been included as sustaining capital for covering the first residue cell in Years 6 and 7. Cell closure will require supernatant to be decanted off the surface as residues settle. Once sufficiently dewatered, a layer of coarse taconite tailings will be placed over the top of residues on which a double membrane, synthetic cover will be placed. A

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further layer of tailings will be placed over the membrane followed by a layer of topsoil. Finally, the whole area will then be re-vegetated.

21.2.15 Process Plant

Outside of escalation since the DFS, the greater bulk of cost changes in the process plant arise in Phase 1. The main changes are summarized in the Table 21-5 below. The costs were estimated on the following general basis:

21.2.16 Crushing and Milling Equipment

The DFS estimate was based on a limited approach to refurbishment of the crushing and milling equipment assuming a year to ramp-up to the throughput of 32,000 tons of ore a day. This was based on an expected yearlong ramp-up of the hydrometallurgical plant. This period of ramp-up would allow considerable downtime in duplicate streams of the comminution circuit to rectify equipment failures. Only single stream items put the feed to the hydromet plant at risk. The change to production of concentrates as an interim product to generate revenue has revised the period of ramp- up to six months with consequent reassessment of the scope of work for refurbishment. For the DFS update, the following approach was taken:

- All of the equipment was classified in terms of criticality: High criticality was given to equipment that is a single item or in a single stream that will stop feed; Interim criticality was given to equipment that has a standby unit or a second stream is available where production will be impacted but not stopped; Low criticality was given to equipment that is located in multiple streams (e.g. the mills) and failure of a single unit will have minimum impact.
- For all equipment in the High and Intermediate categories, the condition of the equipment is in the process of being assessed as follows: External Inspection from which a condition report will be written that also identifies the requirement for additional investigation by disassembly; Internal Inspection as indicated by the condition report to inspect components that are not accessible from an external inspection; Maintenance Records are being accessed from a database maintained by Cliffs that indicates expected remaining life of wear components and turnaround time for replacement of parts.
- From these sources, an assessment of the most likely failure modes in operation (e.g., by wear or incident) is made based on past operating experience. For the High criticality equipment, the risk of failure is then assessed and a strategy for refurbishment and purchase of spares prior to start-up developed.

The assessment of the scope of work and budget is a "work in progress" at this time. For the DFS update, a consensus assessment on the above principles was made for all of the mechanical equipment based on the current level of understanding of PolyMet's personnel. Changes to the DFS estimate include the following:

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- An allowance of $3.5 million for refurbishing the North 60" gyratory crusher and all associated equipment.
- Recondition the spare bearing assembly for the 60" crusher before start-up and purchase a replacement bearing at start-up.
- Replace the central crusher lube system in the fine crushing building with individual lube units to each crusher (past operating experience of these crushers was unsuccessful at identifying major lube oil losses).
- The primary and secondary drives to conveyor 4B and a spare from 4A will be reconditioned for the single stream conveyor that transfers the crushed or to the north fine ore bins in the concentrator building.
- Fully disassemble and recondition all components of the tripper conveyors in the fine crushing building and the north side of the concentrator building.
- Relocate the mill lube oil rooms to provide space for the flotation equipment.
- Rebuild the mill sumps to provide additional freeboard (operating experience was that sumps overflowed or pumps sucked air).
- The refurbishment of elevators in the coarse and fine crushers was evaluated by the vendor that increased the DFS estimate by $300,000.
- Labour and materials was estimated for all platework refurbishment.
- Dust Collection.

PolyMet has accepted the recommendation of the MDNR to upgrade all of the wet scrubbers in the comminution buildings to bag-houses. This represents a significant change from the existing installation. The DFS estimates were based on refurbishing the existing equipment on the assumption that this would meet permitting requirements. Quotes have been obtained for 17 bag-houses and appropriate allowances added to rework the ducting and provide power and control equipment.

21.2.17 Flotation and Concentrates Handling

The original DFS cost estimate was effectively replaced by the split concentrates estimate for Area 25 – Flotation and Regrind. Areas 27 and 28, Nickel Concentrates Handling respectively, were new.

As part of FEED, Bateman produced a revised equipment list, which they used as the basis for estimating revised costs, and cost estimates for mechanical equipment, concrete, structural steel, and pipework. Costs for electrical/instrumentation were factored on mechanical equipment.

Table 21-5: Phase 1 Budget – Variance from DFS from Scope Changes (excludes tailings facilities)

Equipment & Facilities	Variance $'000
Crushing & Milling	8,663
Flotation & Regrind	15,224
Flotation/Reagent Annex Building	4,005

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Equipment & Facilities	Variance $'000
Reagent Area Additions	1,587
HVAC (duplicate allowance in 2 areas of DFS budget)	-3,439
Copper Concentrate Filtration and Loadout	6,316
Nickel Concentrate Filtration and Loadout	6,732
In Plant Rail Facilities for Concentrate Transport	2,500
Utilities Re-estimate (increased allowance for reinstatement)	2,865
Total Variance – Phase 1	44,454

21.2.18 Owner (Corporate) Capital Costs

Owners Project Team

Costs totaling $6 million are included for PolyMet's project team in Denver and at Hoyt Lakes.

Mobile Equipment & Computing

The estimate includes provision for the purchase of an Enterprise Management System (Ellipse by Mincom) at $155,000 initial purchase price plus one year of "annual costs" at $94,000 as per a quotation to PolyMet.

There are no capital cost provisions for motor vehicles, computer hardware, software or network upgrades.

Commissioning spares, transport, vendor assistance and first fills

The estimate includes provisions for the following:

- commissioning spares - $2.066 million, factored on DFCs
- transport to site - $5.524 million factored on equipment costs
- vendor assistance - $1.586 million, factored on equipment costs
- first fill lubricants – $0.548 million, factored on equipment costs
- first fill reagents - $5.348 million
- insurance.

The estimate includes a provision for project insurance of $6,500,000, based on a proposal submitted to PolyMet by Willis of Minnesota (insurance brokers). $2.5 million has been allocated to Phase 1 and $4.0 million to Phase 2.

Owner's Additional Costs

Estimate provisions for Owner's "below the line" costs include:

- process and EPCM Fees remain unchanged from the DFS at $5 million and $7 million respectively

- USFS land exchange: $3.3 million
- wetlands mitigation costs: total of $7.1 million for land acquisition costs including option costs and the cost of developing wetland credits
- site closure liability: $23,600,000.

Closure Costs

Closure costs were estimated by Jim Scott and Kevin Pylka of PolyMet. The Contingency Closure Estimate assumes that the facility is closed the second year of operation and is the basis for financial assurance and will be updated annually. The End of Mine Life Closure Estimate assumes that the facility is closed at the end of the 20-year proposed mine life. Both estimates include all remediation obligations assumed with the acquisition of the Cliffs Erie property, even though PolyMet plans to complete many of those tasks prior to the end of mine life. All costs are in present day dollars.

- Contingency Closure Estimate 04-17-08: $45.4 million for the total scope (full hydromet)
- and $40.7 million for the concentrates only (i.e., Phase 1) scope
- The amounts included in the Project cost report for the Closure Estimates have been reduced by $23.6 million 'Owners Additional' costs as Current (Closure) Liability.

21.3 Operating Cost Estimates

Table 21-6 summarizes operating costs for the two steady state production scenarios: Production of copper and nickel rich concentrates only (split concentrates only);

- Production of copper concentrates with nickel, cobalt and zinc precipitate produced in a single autoclave and reduced hydrometallurgical circuit (Hybrid)
- Full hydrometallurgical plant producing copper cathode, nickel/cobalt hydroxide and AuPGM precipitate (Hydromet).

For comparison purposes, Table 21-6 includes estimates in the DFS, which included hydrometallurgical treatment of all concentrates and a copper extraction process to produce copper cathode.

Table 21-6: Distribution of Costs between Operating Modes

	Split Conc. Only ($'000)	Hybrid – Split Conc. plus one Autoclave ($'000)	Oct. 2006 DFS Full Hydromet ($'000)
Mine & Railroad	50,356	50,356	44,431
Beneficiation Plant	25,230	25,230	31,419
Flotation, Load Out & Tails	16,165	16,166	8,344
Hydromet		15,658	33,758
Plant Utilities	399	399	2,155

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	Split Conc. Only ($'000)	Hybrid – Split Conc. plus one Autoclave ($'000)	Oct. 2006 DFS Full Hydromet ($'000)
Reagents	408	551	620
Laboratory	115	115	
Process Plant Labour	10,452	16,631	5,400
G&A	11,007	11,007	2,587
Total	114,133	136,113	128,714
Average Operating Costs "Steady State" Operation			
US$/st Ore Milled	9.77	11.0065	11.02
US$/st Total Material Mined	4.22	5.03	4.01

21.3.1 Basis of DFS Estimate

Organization Structure & Human Resources

The process control philosophy and the philosophy upon which the organizational structure is based are closely related and together will govern the structure of the organization, the level and type of skills required and manning levels. As such these philosophies are central to how the operation will be run and hence the costs of running it.

The same broad philosophy applies to mine and railroad, the process plant and administrative services. In general, the organisational structure is intended to minimize the number of management layers while keeping the number of direct reports in each layer to a level that suits the activities involved and maximizes operational efficiency.

Staff and labour costs are based on the following;

- Operations will function 365 d/a, 24 h/d with three 8-hour shifts.
- Operations management and essential support services will be provided round the clock on a continuous basis with technical and general support, and general management services operating on day shift only Monday to Friday, excluding statutory holidays
- Laboratory services will be provided on a continuous basis.
- In the determination of labour rates there was no presumption regarding the use of union or non-union labour.

DFS labour rates and staff wages were based on then current base rates applicable at a nearby taconite mining and processing operation. Cost of employment burden (insurances, medical benefits, social security etc) was determined as a fixed percentage of base rate. Current estimates are based a recent evaluation of current local labour conditions. The cost of employment burden is based on a specified employment and benefits package costed based on actual quotes for provision of those

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benefits. Social security, employment tax and other statutory costs of employment were calculated according the appropriate legislated rates. On average, the value of the benefits and burden package amounted to 30% of base rate for management, technical and supervisory staff while that for equipment and plant operators was 37% of basic. The remuneration package will include a discretionary profit sharing component which varies with position in the organization but which is not included in these operating cost estimates. Table 21-7 summarizes base and benefit rates used.

Table 21-7: Labour Costs

Position	Base Rate (US$/a)	Benefit Rate (%)	Benefit Amount (US$/a)	Rate used for Costing (US$/a)
General Manager	150,000	30	45,000	195,000
GM Admin Assistant	50,000	30	15,000	65,000
Division Manager	120,000	30	36,000	156,000
Clerk	50,000	30	15,000	65,000
Area Manager	100,000	30	30,000	130,000
Manager – operations (shift)	70,000	30	21,000	91,000
Manager – operations Support	70,000	30	21,000	91,000
Manager – dispatch/control room	70,000	30	21,000	91,000
Technical Staff – assigned - shift	70,000	30	21,000	91,000
Technical Staff – assigned – support	70,000	30	21,000	91,000
Manager – Technical/Administrative	100,000	30	30,000	130,000
Technical Staff - Engineer	70,000	30	21,000	91,000
Technical Staff – Technician	60,000	30	18,000	78,000
Administrative Staff	60,000	30	18,000	78,000
Equipment Operator	62,000	37	22,940	84,940
Process Technician	60,000	37	22,200	82,200
Maintenance Technician	62,000	37	22,940	84,940
Electrical/Instrumentation Technician	66,000	37	24,420	90,420

Reagents & Consumables

Mine Site Water Treatment Facility: the cost and consumption of reagents required for the mine site water treatment plant were determined by Barr Engineering from quotes obtained from specialist providers of water treatment technologies and from comparable costs at other treatment facilities. Dosage and consumption rates will only be determined with confidence once the treatment facility is operational so there is some risk that actual reagent costs may be different from those assumed for this exercise.

Mine Operations: The mining operation will require few chemicals or reagents though principal among these will be dust suppression agents for haul and access roads and de-icing chemicals for winter use. Costs for this exercise were based on comparable use at nearby taconite mines. Explosives and

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blasting accessory costs are based on current vendor quotes. Ground engaging tool (GET) costs were estimated from vendor quotes for such items as drill bits and drill rods with useful life assumptions based on experience and typical usage rates at local taconite mines.

Process Plant: Reagent and oxygen consumption rates were determined from Metsim modelling and were optimised during the various pilot-scale test programs carried out at SGS Lakefield Research. Wear materials and grinding media consumption rates were estimated from Bond work and abrasion indices calculated from standard laboratory tests of NorthMet material derived from drilling.

Reagents and consumable quantities are defined in terms of steady state operations. During the detailed design, excursion limits will be further investigated to allow for start-up, commissioning and normal plant variations that sometimes occur as a result of operating practises or changes in plant feed characteristics. First fill reagents are not considered in the operating cost model summary as these are considered as capital cost items.

An allowance in each plant area has been included for consumables such as lubricants, greases, rags, welding electrodes and other miscellaneous items.

In most cases, the same reagent consumption rates used in the DFS were used in this exercise because, with the exception of flotation testwork designed to better define the mixed concentrates only option, no other testwork has been performed since the DFS that would lead to a significant change in the estimates of reagent consumption rates. The unit costs of most reagents were updated based on vendor or manufacturer written quotes with appropriate allowances for transport to site where necessary. Local (within the USA) sources of reagents were selected. Most quotes are current for the 1st Quarter 2008 and needed no escalation.

The late arrival of an updated quote for high purity magnesium hydroxide slurry Mg(OH)2 prevented the inclusion of a more reliable unit price than that provided originally by Bateman. The impact of this omission may be in the order of US$0.01/ton milled and further investigation is recommended during the next project phase.

Maintenance & Repairs

The underlying philosophy is that for mine, railroad and process plant routine inspections, routine service and minor repairs will be carried out by PolyMet staff and technicians whereas major repairs, major scheduled maintenance, major component change-out and unit rebuilds will be outsourced to specialists of whom there are several on the Iron Range and its environs.

Mine Equipment: Maintenance costs are principally based on manufacturers recommendations and typical, comparable practice usually on the basis of a factored percentage (factored for location) of initial cost.

Plant Equipment: Process plant equipment repair and maintenance costs are based on a weighted factored approach. In the case of existing crushing, milling, bulk material transport, pumping equipment and existing infrastructure facilities known, historical costs were taken into consideration. While the factoring approach to cost estimation is reasonably reliable for flotation equipment, filters,

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thickeners etc. maintenance on the autoclaves and the SX-EW plant operating under PolyMet conditions is largely unknown.

In the same way as for the DFS, the current operating cost model has allowed for maintenance costs as a percentage of the direct capital cost for the Project. This equates to approximately 4% to 5% per annum and is based on known maintenance requirements for similar processing facilities of this type. These costs do not include the purchase of recommended spare parts prior to commissioning and ramp-up. Operating and commissioning spares are assumed to be capitalised for the first year of operation. Maintenance costs are expected to increase over the life of the mine as equipment ages due to normal and wear and tear though this is not reflected in the current estimates.

Outsourced Service

As described above, major mine equipment and process plant maintenance will be outsourced. Other outsourced activities may include site security, janitorial services, certain environmental monitoring and sampling activities and periodic tailings dam safety inspection, testing and reporting.

In the mine, transport to site and placement of explosives in blastholes will be carried out by a local vendor of explosives products and blasting costs are based on quotes for the provision of such a service. Similarly, a local vendor will transport fuel oil and lubricants to site and will be responsible for operating day storage tanks and re-fueling equipment and locomotives directly from mobile tankers.

Electric Power

Power costs are based on PolyMet's agreement with Minnesota Power (MP) with provision for escalation due to environmental upgrades and renewable energy initiatives. A flat unit rate of US$0.06/kWh has been used, based "large" customer rates.

Mine power consumption was based on installed motor power with application of a utilization factor based on expected hours of equipment use.

Beneficiation, flotation and hydromet plant power consumption was calculated from the detailed electric motor list and application of a similar utilization factor as used for mine equipment. Power consumption for offices, workshops and support facilities was generally based on an allowance where specific information on installed power was not available.

Power consumption for the Hybrid option was estimated as a function of the amount of nickel rich concentrate that requires treating. For motors such a sump pumps where the utilisation is expected to be less than 85%, the power consumption is assumed to be the same as the Hydromet option. For all other motors, the power consumption is lower by a factor of copper consumption mass divided by the nickel concentrate mass.

Fuel Oil

The unit cost of fuel oil is assumed to be US$3.00/gal though discussions with prospective vendors indicate that hedging and other commercial arrangements may be used to minimize the effects of variable crude oil prices.

The majority of fuel oil is consumed in the mine area and vehicle and equipment consumption rates are based on manufacturers' estimates of typical consumption rates in comparable applications elsewhere. Fuel consumption by ore haulage locomotives was derived from manufacturers' haulage simulations using the planned track profile and proposed pulled load parameters.

General and Administrative Cost

The major G&A cost component is staff and labour (including plant and mine technical support services and the laboratory).

The annual cost of running an administrative organization was developed from experience at Iron Range taconite mines and covers such things as security, office equipment, heat and lighting, communications, overtime, property insurance, office supplies, computer system license fees, admin building maintenance, janitorial services, and allowances for travel and meetings.

Note that while laboratory staff are part of the Technical Services and Support Division and hence fall under the general heading of G&A, the costs of laboratory equipment maintenance, power, reagents and consumables are included in Plant operating costs.

21.3.2 Mine Operating Costs

A significant difference between this estimate and the DFS is the change from contractor to Owner mining. While pre-production mine development will remain a contracted activity it is now intended that PolyMet will acquire and operate its own mining fleet (Table 21-8). To minimize up-front capital costs the majority of the mining fleet will be leased. Leases can be of two types, each with its own specific tax implications though for the purposes of this costing exercise it has been assumed that all leased equipment will be acquired on operating or "tax" leases. Because of potentially significant tax and cashflow implications further financial analysis isrecommended before selection of the specific type of leasing or purchasing instrument.

Table 21-8: Mine Equipment Capital Costs

Equipment	Model	No. Required	Purchase Price	Monthly LeasePayment (based on 60 month lease, except locos)
Electric Hydraulic Shovel	Komatsu PC5500	2	$10,566,000 ea. With spare bucket, power cable, switch house and dispatch system	$153,542 ea.
240 ton Haul Truck	Caterpillar 793 C	9	$3,050,500 ea. With tires, one third cost of a spare box and dispatch system	$44,856.78 ea. w/o tires
Large FEL	Caterpillar 994	1	$4,127,392 with tires, chains , spare bucket and dispatch system	$57,491.48 w/o tires
Electric Rotary Blasthole Drill	Bucyrus 59R (used and rebuilt)	1	$3,707,000 ($1,075,000 for the used drill, $2,632,000 to	NA

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Equipment	Model	No. Required	Purchase Price	Monthly Lease Payment (based on 60 month lease, except locos)
			rebuild)	
Diesel Rotary Blasthole Drill	Atlas-Copco PV351	1	$4,199,679 including freight, drill string, power cable and dispatch system	$72,700.00
Large Bulldozer	Caterpillar D11T with ripper	2	$2,015,277 with one half cost of a spare blade and dispatch system	$29,971.39 ea.
Large Rubber Tired Dozer	Caterpillar 854G	1	$1,990,840 with tires, spare blade and dispatch system	$27,893.96 w/o tires
Rubber Tired Dozer	Caterpillar 834H	1	$1,127,800 with tires, cable reel system and dispatch system	$15,999.65 w/o tires
Front End Loader	Caterpillar 988	1	$894,655 with tires, spare bucket and dispatch system	$12,631.30 w/o tires
Road Grader	Caterpillar 16M	1	$716,600 with tires and dispatch system	$9,963.47 w/o tires
Road Grader	Komatsu GD675	1	$271,500 with tires and dispatch system	$3,687 with tires
Bulldozer (Tailings Basin)	Caterpillar D8TLGP	1	$738,610 with dispatch system	$11,850.00
Utility Excavator with Hammer	Caterpillar 345CL	1	$780,000 with breaker hammer and dispatch system	$6,859.34
Utility Haul Trucks	Caterpillar 777	2	$1,551,112 with tires, spare rock box and dispatch system	$24,961.47 ea. w/o tires
Water/Sand Truck	Kenworth/Sterling	2	$350,000 ea. With sand spreading box	$2,025 ea. (rate for an International chassis instead of Kenworth/Sterling
Tool Carrier	Caterpillar IT38GII	1	$375,000 with forks, bucket and snow plow attachments	$5,473.57
Skid Steer Loader	Komatsu SK1026-5N	2	$39,900	$716 ea.
Lowboy and Tractor	125 ton Load King Trailer with International Tractor	1	$331,500	$5,649
Tire Handler with Front End	Komatsu WA 500	1	$535,000	$8,474

Equipment	Model	No. Required	Purchase Price	Monthly Lease Payment (based on 60 month lease, except locos)
Loader				
Crane 90-ton	Grove RT890E	1	$709,500	$8,988
Light Plants	Almand Maxi-Lite ML-6	5	$14,500	$263 ea.
Pumps	Gorman-Rupp S8C1 Submersible	3	$45,000 ea.	$573 ea.
Pump/Service		1	$250,000	$2,518 (rate for an
Truck				International chassis)
Pickup Trucks		5	$35,000 ea.	$400.00 estimated each
Crew Cab Pickup Trucks		3	$40,000 ea.	$500.00 estimated each
Crew bus	Used, re-conditioned	2	$30,000 ea.	
Fire Truck	Used, re-conditioned	1	$290,000	
Ambulance	Used, re-conditioned	1	$100,000	
Locomotives	NREC 3GS-21C N-Viromotive	4	$1,789,000 ea. With remote control	$650.00 ea. per day

Note: * "All lease payments shown above, except for locomotives are monthly and based on a standard 60 month lease term. Locomotive lease terms are quoted as a daily rate with not fixed term.

The total cost of leasing mining equipment and mine railroad locomotives over a 20-year mine life is US$193.8 million dollars of which US$19.9 million will be required during operating years 1 and 2.

21.4 Comments on Section 21

PolyMet plans to complete a full update of both capital and operating costs when the detailed design is finanalized as a result of the environmental review and permitting process. It estimates that capital costs (other than for mine equipment) have been increasing at approximately 3% a year since 2008, mine equipment costs are more volatile, reflecting shorter-term demand. PolyMet anticipates some expansion of scope of environmental protection measures, which may result in a more substantive change in capital costs.

Operating costs reflect the cost of labour and consumables, especially power. PolyMet's long-term power contract with Minnesota Power is an important factor in stablizing its operating costs.

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22 ECONOMIC ANALYSIS

The following economic analysis reflects the DFS. The impact of the DFS Update in 2008 is described in Section 22.2. In February 2011, PolyMet announced that it planned to build the Project in two phases:

- Phase I: produce and market concentrates containing copper, nickel, cobalt and precious metals
- Phase II: process the nickel concentrate through a single autoclave, resulting in production and sale of high grade copper concentrate, value added nickel-cobalt hydroxide, and precious metals precipitate products.

The changes reflect continued metallurgical process and other project improvements as well as improved environmental controls that are being incorporated into the Supplemental Draft EIS. The advantages, compared with the earlier plan, include a better return on capital investment, reduced financial risk, lower energy consumption, and reduced waste disposal and emissions at site.

This revised plan reduces the DFS Update capital cost estimate by approximately $127 million. PolyMet did not report detailed economic impact of these project changes but the impact will have been positive owing to reduced capital and operating costs. This analysis will be included in the full project update once all of the details of environmental mitigation measures have been finalized in the Supplemental Draft EIS.

22.1 DFS Economic Analysis

The DFA economic evaluation is based on proven and probable reserves of 181.7 million tons and a mining rate of 32,000 tons per day (11.68 million tons per annum). Changes to the basic assumptions and their impacts are discussed later in this section.

All resource and reserve analysis and mine modelling have been based on the following metal prices, reflecting prices that were relevant during the preparation of the DFS, namely: copper - $1.25/lb, nickel - $5.60 per pound, cobalt - $15.25/lb, palladium - $210 per ounce, platinum - $800 per ounce and gold - $400 per ounce. This price scenario equates to a NMV of $16.09 per ton. Key DFS statistics are shown in Table 22-1.

Table 22-1: Key DFS Statistics

Reserves and Resources			
Measured & Indicated (M+I) Resources 1	422.1 M tons	Copper equivalent grade	0.86% Cu
Inferred Resources	120.6 M tons	Copper equivalent grade	0.80% Cu
Proved and Probable Reserves	181.7 M tons	Copper equivalent grade	0.96% Cu
Mining			
Life of Mine average total mining rate	81,070 t/d	Plant feed rate	32,000 t/d

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Reserves and Resources			
Initial mine life (permit application)	20 years		
Production – annual average in 1st five years			
Copper cathode (high grade)	72,057 Mlb	Precious metals (Pt, Pd, Au)	105,984 oz
Nickel in hydroxide	15,400 Mlb	Cobalt in hydroxide	0.727 Mlb
Life-of-Mine operating costs per ton			
Mining cost per ton of rock mined	US$1.14	Processing cost per ton milled	US$6.99
Mining cost per ton of ore mined	US$3.13	General, Admin & other per ton milled	US$0.66
Capital Costs			
Initial Direct Cost	US$285.1 M		
Contingency	US$27.1 M		
Total	US$312.1 M		
Indirect Costs	US$67.5 M		
Total Initial capital	US$379.6 M		
Sustaining capital (20-year project)	US$71.8		
Economic Summary – NI 43-101 Base Case			
IRR after tax	26.7%		
After tax NPV @ 7.5%	US$595.4 M		
Average annual EBITDA in first 5 years	US$175.3 M		

22.1.1 Economic Assumptions

Metal price assumptions for reserve analysis and pit design are deliberately conservative. The U.S. Securities and Exchange Commission (SEC) allows reserves to be estimated using three-year trailing average prices to the date of the reserve report, namely $1.61/lb for copper, 6.52/lb for nickel and $234, $896, and $597 per ounce respectively for palladium, platinum and gold. This price scenario equates to a NMV of $19.55 per ton.

The Base Case for economic modelling in the DFS uses metal prices that are slightly lower than those allowed by the SEC, namely: copper - $1.50/lb, nickel - $ 6.50/lb, palladium - $225/oz, platinum - $900/oz, and gold - $450/oz for a NMV of $18.67 per ton.

These prices are substantially lower than the average in July 2006 of $3.50/lb for copper, $12.06/lb for nickel, and $322, $1,241 and $634 per ounce respectively for palladium, platinum, and gold with a NMV of $36.61 per ton.

As a middle ground, we have used a market-related formula taking the weighted average of the three-year trailing average price at the end of July 31, 2006 (60%) and the average two-year forward price in July, 2006 (40%.) These prices are: $2.25/lb for copper, $7.80/lb for nickel and $274, $1,040, and $540

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per ounce respectively for palladium, platinum and gold with a combined NMV of $24.82. This is the price scenario that has been applied to the case referred to herein as the NI 43-101 case.

22.1.2 Key Data and Economic Analysis

The economics reported in the DFS reflect the initial mine plan which in turn is based on the 2004 Environmental Assessment Worksheet for an ore processing rate of 32,000 tons per day for an initial period of 20 years. As previously described, the pit plan is not fully optimized and the 20-year permit application covers significantly less than half of the measured and indicated resources already defined.

Table 22-2 sets out DFS Base Case metal price assumptions and process recovery and key operating data for the average of the first five years of full-scale production. These data comprise metal content of the three products described above, the contribution to net revenue after third-party processing costs, estimates of cash costs for each metal using a co-product basis whereby total costs are allocated to each metal according to that metal's contribution to the net revenue, cash costs on a by-product basis whereby revenues from other metals are offset against total costs and those costs divided by production – this analysis is included for copper and for nickel. The final columns show the increase or decrease in the EBITDA with a change in the price of each metal.

Table 22-2: Base Case Price and Operating Assumptions and Key Production Numbers

		Assuptions		Average of First Five Years					
		Base Case	Metal Recovery	Production	Contribution to net revenue	Cash Costs co-product	by product	Sensitivity Δ Price	Δ EBITDA
		$lb or Oz	%	mlbs or oz	%	$/lb or $/oz	$/lb or $/oz	$/lb or $/oz	$'000
Copper	lb	1.5	92.3%	72.058	46.0%	0.81	0.06	0.10	6,990
Nickel	lb	6.5	70.3%	15.401	34.1%	2.84	-1.46	0.10	1,195
Cobalt	lb	15.25	40.7%	0.727	3.8%	6.67	n/a	0.10	56
Palladium	oz	225	75.2%	75,995	6.7%	113	n/a	10	737
Platinum	oz	900	72.7%	20,531	7.8%	477	n/a	10	199
Gold	oz	450	67.0%	9,459	1.8%	239	n/a	10	92
Total precious	oz			105,984	16.3%		n/a	10	1,028

The price assumptions included July 2006 average prices (shortly before publication of the DFS), the Base Case and the NI 43-101 case described previously. The table shows a sensitivity analysis of a ± 10% change in the Base Case metal price assumptions.

Table 22-3 sets out key financial statistics – the internal rate of return on the future capital investment and the present value of the future cash flow (including capital costs) using a 5% and 7.5% discount rate on both a pre-tax and an after-tax basis. The bottom section of the table shows the average over the first five years of full-scale production for gross revenue (before royalties and third-party processing fees), net revenues (after those costs) and EBITDA.

The price assumptions included July 2006 average prices (shortly before publication of the DFS), the Base Case and the NI 43-101 case described previously. The table shows a sensitivity analysis of a ± 10% change in the Base Case metal price assumptions.

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Table 22-3: Economic Projections on a Range of Metal Price Assumptions

		Average July 2006	Price Assumptions			
			Main Cases		Sensitivity	
			Market Case 3-year trailing plus 2-year forward	Base Case	Base -10%	Base +10%
Metal Prices						
Copper	$/lb	3.50	2.25	1.50	1.35	1.65
Nickel	$/lb	12.06	7.80	6.50	5.85	7.15
Cobalt	$/lb	14.52	16.34	15.25	13.73	16.78
Palladium	$/oz	322	274	225	203	248
Platinum	$/oz	1,241	1,040	900	810	990
Gold	$/oz	634	540	450	405	495
Financial Summary						
Pre-tax						
IRR	%	61.0%	34.2%	17.4%	11.4%	22.9%
PV discounted at 5%	$'000	2,606,279	1,210,792	450,643	217,282	684,003
PV discounted at 7.5%	$'000	2,034,062	910,978	298,807	110,911	486,702
Post-tax						
IRR	%	47.4%	26.7%	13.4%	8.6%	17.8%
PV discounted at 5%	$'000	1,931,367	873,022	295,515	117,455	472,983
PV discounted at 7.5%	$'000	1,388,430	595,358	161,924	28,036	295,167
First 5 years:						
Average gross revenue	$'000	504,438	341,417	259,111	233,200	285,022
Average net revenue	$'000	440,257	303,147	228,067	205,091	251,044
Average EBITDA	$'000	312,382	175,273	100,193	77,216	123,169

During the first five years of full-scale production, cash costs of production (excluding amortization of capital) on a co-product basis (allocating costs to each metal according to its contribution to revenue) and using Base Case metal price assumptions are projected at $0.81/lb for copper, $2.84/lb for nickel, and $113, $477, and $239 per ounce respectively for palladium, platinum, and gold.

Alternatively, using the by-product method whereby revenues from other metals are offset against costs of a primary metal, the five-year average cash cost of copper would be $0.06/lb or, if NorthMet were viewed as a nickel mine, nickel costs would be minus $1.46/lb.

After state and federal taxes, the Base Case rate of return is 13.4% and the present value of the future cash flow discounted at 7.5% per annum is $162 million. During the first five years of full-scale operation, EBITDA (Earnings before Interest, Taxation, Depreciation, and Amortization, or operating cash flow) is projected to average $100 million a year.

A $0.10/lb change in the copper or nickel price would increase or decrease average annual EBITDA during the first five years of full-scale operation by $7.0 million and $1.2 million respectively and a $10/oz change in all of the precious metal prices (palladium, platinum, and gold) would increase or decrease the five-year average annual EBITDA by $1.0 million.

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22.1.3 2008 DFS Update

Capital Costs

Since the September 2006 DFS, and on a like-for-like basis, the total capital cost has increased by 36% to $516.8 million. This increase reflects both cost inflation and design scope changes since the DFS, including facilities needed to ship concentrate during the construction and commissioning of the new hydrometallurgical plant.

In addition, PolyMet anticipated $85.1 million of expenditures on measures to protect the environment, over and above the measures contemplated in the DFS. $76.6 million for mining equipment that was assumed to be provided by a mining contract in the DFS has been incorporated as an operating lease in updated operating costs.

PolyMet has previously stated that it has been reviewing the possibility of selling concentrate during the construction and commissioning of new metallurgical facilities. This staged approach shortens the initial construction period, makes the Project less sensitive to the delivery schedule for long lead-time equipment such as autoclave vessels, and means that PolyMet can commence operations of the mine, the existing crushing and milling plant, the existing tailings disposal facilities, and the new flotation circuit, before starting the new hydrometallurgical plant.

As a result of the staged approach, the total capital required prior to initial production and sales declines to $312.3 million, which includes $64.7 million of additional environmental safeguards for this level of activity (Table 22-4).

Table 22-4: Capital Costs (US$ M)

	Full Project	*Change from DFS*	Initial Concentrate Sales
Definitive Feasibility Study	379.8		138.7
Escalation and other scope changes	137.0	*36%*	108.9
Total	516.8		247.6
Environmental measures	85.1		64.7
Total change	222.1	*58%*	173.6
TOTAL	601.9		312.3

Operating Plans and Costs

The overall mining and operating plan remains the same as that defined in the DFS and which forms the basis of the plan being analyzed in the environmental impact statement. PolyMet intends to mine 32,000 tons of ore per day for an operating life of twenty years, processing a total of 224 million tons of ore.

The mine plan continues to be based on the following metal prices: copper - $1.25/lb, nickel - $5.60 per pound, cobalt - $15.25/lb, palladium - $210 per ounce, platinum - $800 per ounce, and gold - $400 per ounce.

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Operating costs per ton of ore processed have increased to $13.33 from $11.02 in the DFS reflecting higher fuel, mine equipment, and other consumable costs, as well as general inflation. The cost of mining and delivering ore to the plant is now estimated at $4.31 per ton compared with $3.80 per ton in the DFS. The increase in mining costs has been partially offset by the lower strip ratio, larger mining equipment, and owner versus contractor operation.

The economic analysis is based on SEC-reserve standards, namely the three-year trailing average, which we calculated at April 30, 2008 (the end of our first fiscal quarter). This price deck is copper - $2.90/lb, nickel - $12.20/lb, cobalt - $23.50/lb, palladium - $320/oz, platinum - $1,230/oz, and gold - $635/oz. While these prices are somewhat higher than those used on the economic analysis in the DFS, each price is well below current market levels – in the first quarter of 2008, the following prices prevailed: copper - $3.52/lb, nickel - $13.09/lb, cobalt - $46.37/lb, palladium - $441/oz, platinum - $1,867/oz, and gold - $925/oz.

This translates into copper cash costs of $1.05 per pound using a co-product basis to calculate costs, compared with the DFS estimate of $0.81/lb. Taking revenues from the other metals as a deduction against costs, the co-product basis shows a cost of $(0.28) per pound compared with $0.06 per pound in the DFS.

Economic Summary

Key economic metrics include earnings before interest, tax, depreciation, and amortization (EBITDA) which is projected to increase to $217.3 million on average over the first five years of operations from $175.3 million estimated in the DFS. The net present value of future cash flow (after tax) discounted at 7.5% is estimated to be $649.4 million compared with $595.4 million in the DFS, and the after tax internal rate of return is now estimated at 30.6% compared with 26.7% in the DFS. The table below also sets out the affect on EBITDA of a 10% change in each metal price.

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Table 22-5: Key Economic Highlights

		Update May-08	DFS Sep-06
Operating plan			
Proven and probable reserves	million t	274.7	181.7
Ore mined - life of operation	million t	224.0	181.7
Overburden removed (capitalized under site preparation)	million t	18.5	-
Waste	million t	285.3	302.3
Operating costs per ton processed			
Mining and delivery to plant	$/t	4.31	3.80
Processing	$/t	8.07	6.75
G&A	$/t	0.94	0.46
Total	$/t	13.33	11.02
Metal price assumptions (SEC-standard)			
Copper	$/lb	2.90	2.25
Nickel	$/lb	12.20	7.80
Cobalt	$/lb	23.50	16.34
Palladium	$/oz	320	274
Platinum	$/oz	1,230	1,040
Gold	$/oz	635	540
Economic summary			
Annual earnings before interest, tax, depreciation and amortization (EBITDA) - average first five years	$ million	217.3	175.3
Net present value of future after tax cash flow discounted at 7.5%	$ million	649.4	595.4
Internal rate of return (after tax)		30.6%	26.7%
Sensitivity: 10% ± price = $Δ million in EBITDA			
Copper	$ million	18.6	15.7
Nickel	$ million	13.3	9.3
Cobalt	$ million	0.9	0.9
Palladium	$ million	1.7	2.0
Platinum	$ million	1.7	2.1
Gold	$ million	0.3	0.5
Copper costs			
cash - co-product method	$/lb	1.05	0.81
cash - by-product method	$/lb	(0.28)	0.06

Table 22-6: Metal Prices

		DFS Base Case	DFS Market Case	DFS Update 3-year trailing average	06/30/12 3-year trailing average
Metal Price					
Copper	$/lb	1.50	2.25	2.90	3.56
Nickel	$/lb	6.50	7.80	12.20	9.47
Cobalt	$/lb	15.25	16.34	23.50	17.69
Palladium	$/oz	225	274	320	684
Platinum	$/oz	900	1,040	1,230	1,689
Gold	$/oz	450	540	635	1,485
After tax:					
Internal rate of return	%	13.4%	26.7%	30.6%	
PV dicounted at 7.5%	$ millions	161.9	595.4	649.4	

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22.2 Comments on Section 22

PolyMet plans to complete a full update of both capital and operating costs when the detailed design is finanalized as a result of the environmental review and permitting process.

In addition to reflecting the scope and cost of this design, the update will also reflect current metal market conditions.

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23 ADJACENT PROPERTIES

There are no adjacent properties that PolyMet is proposing to explore or drill as part of any drilling program or other evaluation. There are several other deposits in the Duluth Complex, including the Mesaba project owned by Teck Resources, Serpentine owned by Encampment Resources, and the Nokomis project owned by Twin Metals, a join venture between Duluth Metals and Antofagasta.

Twin Metals has retained Bechtel Corporation to conduct a prefeasibility study on the Nokomis project. Teck completed an internal prefeasibility study on Mesaba when it was seeking to acquire the Erie Plant.

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24 OTHER RELEVANT DATA AND INFORMATION

24.1 US Steel Assays (1960s and 1970s)

US Steel assays are derived from old records which are incomplete in terms of QA/QC details. There are, however, less than ~200 US Steel assays remaining in the database that have not been replaced by more recent assays.

Gatehouse (2000a) summarizes the US Steel sampling and assaying:

USX 'bx' diameter drilling and 10 ft intervals (late60s-70s) was sampled using anvil splitting and prepared and analysed by the central USX laboratory. Sample rejects were kept as –6# and –20# material produced by gyratory and rolls crushers respectively. The precise techniques are not available but given the era, the style of analyses done at that time, and nature of the company it is highly probable that total copper and nickel assays were produced using AAS. No Au or PGMs were analysed. No quality control has been found for this work.

There are 1,790 ACME aqua regia re-assays of samples previously assayed by US Steel. Averages for US Steel and ACME, respectively are copper 0.39% and 0.39%; nickel 0.14% and 0.09%. Two-hundred and seventeen check assays by Chemex are available. Averages for US Steel and Acme, respectively, are copper 0.25% and 0.25%; nickel 0.11% and 0.08%. Thus, US Steel copper assays match, on average, both those by ACME and Chemex. Nickel appears high in the US Steel assays, which may partly be a result of a more total digestion used. Acme's acid digestion was weaker than that used by Chemex.

24.1.1 Status of Nickel Assays

Gatehouse (2000b) summarizes the status of the Ni assays:

- Against Genalysis ICP (4B), Chemex partial aqua regia assays are strongly biased as should be expected. On average, the Chemex preferred assays used for the resource calculation are biased low by 5-6% against Genalysis totals. The clear conditional bias in this data is also as expected and consistent with Lakefield metallurgical reports of a proportion of the nickel resident in silicates. Bias changes from about 20% at 500-600 ppm to no recognizable bias at greater than about 0.3% Ni. This pattern is consistent with higher proportions of Ni being resident in sulfide at higher grades. Lakefield metallurgical reports suggest that Ni in silicates is variable between 200 and 700 ppm. This is also consistent with Co results.
- In summary, the NorthMet Ni resource is based on partial digest results. At worst, the average bias would be 5% lower than total results. This does not necessarily alter the economics of the Project as it may eventuate that Lakefield head assays on which recoveries have been predicated may prove themselves similarly biased.

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24.1.2 Status of Copper Assays

Gatehouse (2000b) summarizes the status of the copper assays:

- On average, preferred Chemex aqua regia assays are biased low by about 2% against Lakefield XRF results (2A) , by 5% against Genalysis total acid digest ICP (2B) and by 1-2% against Chemex total digest ICP(2C). Such results are consistent with the low partitioning of Cu into silicates and represent a limit of a tolerable assay outcome. Biases of much greater than 5% are not acceptable and require improved assay.
- Given the notionally total nature of Genalysis and Lakefield assays it is probable the Chemex aqua regia used in the resource data is low biased from an accurate result by less than 5% on average. This bias is conservative and would have no negative impact on resource figures.

24.1.3 Status of Cobalt Assays

Gatehouse (2000b) summarizes the status of the cobalt assays:

- The Chemex aqua regia digestions are significantly low biased, on average about 20%, against Genalysis total assays. The bias is conditional and significantly increases with lower grade. Though the number of samples is smaller, the same effect can be seen between Chemex aqua regia and Chemex total digest ICP.
- Cobalt forms a very small portion of the value of the resource and, for economic purposes and factoring through metallurgical recoveries, its resource value is likely to be currently underestimated by around 20%. A small upside exists on the value of the resource by virtue of underestimated resource cobalt being related to total cobalt used in metallurgical calculations.

24.1.4 Status of the Palladium Assays

Gatehouse (2000b) summarizes the status of the palladium assays:

- On average, Chemex is biased about 2% high against both Genalysis and Lakefield. Bias is not conditional against Lakefield. Chemex bias is conditional against Genalysis' NiS assay and increases with grade. It is not considered significant given the nugget imprecision between assay types due to sub-sampling and signified by the large dispersion in the …scatter points. However, this situation should be monitored with ongoing quality control in the event that it might become significant with changing mineralized domain.

24.1.5 Status of the Platinum Assays

Gatehouse (2000b) summarizes the status of the platinum assays:

- On average, Chemex is biased low against both Genalysis NiS assays(6B) and Lakefield lead oxide fire assays(6A). Further a conditional bias against Genalysis is similar to that of palladium and similar ongoing monitoring is recommended.

24.1.6 Status of the Gold Assays

Gatehouse (2000b) summarizes the status of the gold assays:

- As with Platinum, gold by virtue of its low abundance is subject to significant sub-sampling nugget effects. Though biases are apparent, the low contribution of Au to economic value means they are not significant at this time. However, quality control monitoring should be continued.
- Against Becquerel NAA (7C), a very good reference technique for gold analyses, Chemex gold is biased low by 20%. The low levels (50 ppb) and severe nugget effects render this insignificant. On average, Chemex is biased low against both Genalysis NiS assays and Lakefield lead oxide fire assays. Further, a conditional bias against Genalysis is similar to that of palladium.
- Extraction of Au into NiS during fire assay is inefficient. The low bias of Genalysis against Chemex (7B) is expected and not relevant.
- The low bias of Lakefield against Chemex is largely a function of assay imprecision at very low grades and is not significant...

24.1.7 Summary – Copper, Nickel, Cobalt

Gatehouse (2000b) summarizes the status of the copper, nickel and cobalt assays:

- Chemex aqua regia assays, on which the Cu Ni Co resources are based, are biased low by a small amount. The total economic impact will be less than 5%, which is acceptable for resource assays. Never the less, it is highly probable that there remains an inherent bias.
- Initial results for a limited number (54) of samples from the recent metallurgical drilling program support Gatehouse's prediction. Cobalt and nickel assays from 4-acid digestions being 14% and 5%, respectively, higher than assays based on aqua regia. Copper values are similar.
- A number of batches assayed in 2000 had included PolyMet standards (N1-3). Some of these have nickel assays that report approximately 10 to 20% above the recommended value though significantly more batches understate nickel. Copper values were largely accurate.

24.1.8 Summary – Platinum Group Elements and Gold

Gatehouse (2000b) summarizes the status of the platinum group element and gold assays:

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- Though some evidence for conditional biases exist between lead oxide and NiS fire assay for PGEs the low level is acceptable for lead oxide fire assay to be used for ongoing resource assessment. However, of lesser economic significance, the strong negative bias of gold in NiS analyses and its greater cost and expertise required for good assays, strongly mitigates against the NiS technique. However, NiS fire assay for PGEs should be used for quality control monitoring as an ongoing precaution against the potential for significant bias in different mineralized domains at NorthMet.
- It is well recognized that nickel-sulphide (NiS) assays underestimate gold. The only good reason to select NiS assaying is for the determination of rhodium, rhenium, etc (Bloom, pers comm).

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25 INTERPRETATION AND CONCLUSIONS

AGP estimated a mineral resource for the NorthMet Deposit using data supplied by PolyMet. This data incorporates the 2007 summer drilling results that were available as of October 15, 2007. The model used the same interpolation methodology used in the Wardrop September 2007 report.

The pre-2007 dataset used by Wardrop was extensively verified by previous authors and the QP spot checked selected holes from the US Steel era and the PolyMet 1999, 2000 and 2005 drill campaign against the paper copies of the laboratory certificates. The 2007 drilling was verified by AGP using the electronic version of the laboratory certificate.

Model was interpolated using Ordinary Kriging with Inverse Distance Squared and Nearest Neighbour interpolation methods used for validation. No significant discrepancies exist between these methods.

Based on the review of the QA/QC, Data validation and statistical analysis of the data, AGP draws the following conclusions:

- AGP has reviewed the methods and procedures to collect and compile geological and assaying information for the NorthMet Deposit and found them meeting accepted industry standards and suitable for the style of mineralization found on the property.
- A mix of data type was use to generate the resource on the property including historical drill results drilled by US Steel data. Fourteen percent (14%) of the assayed footage is by Reverse Circulation (six inch) drilling; with the remainder by diamond coring was use in the resource estimate. PolyMet validated the RC drill results against twin (or near twin) drill hole and found them to be satisfactory. AGP's Principal Resource Geologist visited the site, reviewed some of the historical drill core and interviewed PolyMet staff. AGP believes that the information supplied for the resource estimate and used in this report is accurate.
- A QA/QC program comprising industry standard blank, standard and duplicate samples has been used on the Project since the 2005 drill program. QA/QC submission rates meet industry-accepted standards.
- Data verification was performed by AGP through site visits, collection of independent character samples and a database audit prior to mineral resource estimation. AGP found the database to be exceptionally well maintained and error free and usable in mineral resource estimation. AGP also believes that the information supplied for the resource estimate and used in this report is accurate.
- The specific gravity determinations are representative of the in-situ bulk density of the rock types.

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- Sampling and analysis programs using standard practices provided acceptable results. AGP believes that the resulting data can effectively be used in the estimation of resources.
- Core handling, core storage and chain of custody are consistent with industry standards.
- In AGP's opinion the current drill hole database is adequate for interpolating grade models for use in resource estimation.
- Mineral resources were classified using logic consistent with the CIM definitions referred to in NI 43-101.
- AGP estimate the NorthMet resources (above a US$7.42 NMV cut-off) to contain 694.2 million short tons (629.8 million tonnes) in the Measured and Indicated categories grading at 0.265% copper, 0.077% nickel, 68 parts per billion (ppb) platinum, 239 ppb palladium, 35 ppb gold and 71 parts per million (ppm) cobalt. The Inferred category (above a US$7.42 NMV cut-off) totals 229.7 million short tons (208.4 million tonnes) grading at 0.273% copper, 0.079% nickel, 73 ppb platinum, 263 ppb palladium, 37 ppb gold and 56 ppm cobalt.
- The NMV formula used and described in Section 17.2.12 of this report includes gross metal price multiplied by the processing recovery minus refining, insurance and transportation charges and is the same formula used in the Hunter 2006 report.
- Above the 0.2% copper cut-off the NorthMet Deposit contains 442.1 million short tons (401.0 million tonnes) in the Measured and Indicated categories grading at 0.325% copper, 0.089% nickel, 81 ppb platinum, 292 ppb palladium, 41 ppb gold and 73 ppm cobalt. The Inferred category totals 158.7 million short tons (144.0 million tonnes) grading at 0.329% copper, 0.088% nickel, 86 ppb platinum, 315 ppb palladium, 43 ppb gold and 55 ppm cobalt.
- Comparing the AGP model with the previously published estimate, Table 17.23 of the Wardrop, September 2007 report, results show an increase of 15.5 million short tons (14.1 million tonnes) in the Measured category and 40.5 million short tons (36.7 million tonnes) in the Indicated category for a total of 56 million short tons (50.8 million tonnes) or 8.1% increase in the Measured plus Indicated category. The Inferred Resource tonnage dropped by 21.9 million short tons (26.4 million tonnes) or 9.5%. The comparison includes resources above a US$7.42 Net Metal Value (NMV) cut-off from surface down to the 0.00 ft elevation level.
- Compared with the Wardrop September 2007 estimate, grades in the Measured and Indicated categories dropped slightly for copper and nickel and increased slightly for platinum, palladium, gold and cobalt grade elements. Copper changed by -0.3%, nickel by -0.5%, platinum by +2.1%, palladium by +1.8%, gold by +2.1% and cobalt by +0.1%. However, the contained metal value increased for all elements by about 10% in the Measured and Indicated categories. Copper increased by 8.5%, nickel by 8.2%, platinum by 11.1%, palladium by 10.8%, gold by 11.0% and cobalt by 8.9%.

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- The work carried out during the Summer 2007 drill program has met the primary objectives relating to the in-fill drilling.
- Reserves for the Northmet project contained within the DFS pit shell amounted to:

  Proven = 118.1 million tons Grading 0.30% copper, 0.09% nickel, 75 ppb platinum, 275 ppb palladium, 38 ppb gold and 75 ppm cobalt.

  Probable = 156.5 million tons Grading 0.27% copper, 0.08% nickel, 75 ppb platinum, 248 ppb palladium, 37 ppb gold and 72 ppm cobalt.

  Total Proven and Probable = 274.7 million tons Grading 0.28% copper, 0.08% nickel, 75 ppb platinum, 260 ppb palladium, 37 ppb gold and 73 ppm cobalt.
- Further increases in reserves are dependent upon the conditions outlined in the ongoing environmental review and permitting process.

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26 RECOMMENDATIONS

AGP offers the following recommendations:

PolyMet should proceed with final design engineering and construction of the NorthMet Project as soon as permitting allows. Prior to construction, PolyMet should:

Review and update the scope of the Project design to reflect any changes resulting from the environmental review process and other project enhancements update the capital and operating cost estimates based on the scope review and current prices

Prior to detailed, pre-production planning a limited program of close spaced drilling is recommended. This program will have two objectives:

- To determine the optimum drill hole spacing for grade control and scheduling and,
- To acquire sufficient data to increase confidence in grade affecting the initial open pit production.

Budget for 625 large diameter (5 1/2") reverse circulation drill holes averaging 30 ft for a total of 19,050 ft is estimated at $40 /ft for an all in cost of $782,000 including a $20,000 mobilization charge. Cost is less if using a 3 1/2 " diameter.

All of these items are in PolyMet's budgets for activities before the start of construction, for a total of $3.0 million.

Various recommendations for further work resulted from this Updated DFS, which have subsequently been completed. These included:

1) Various recommendations for further work resulted from the Updated DFS. Some of this work has been completed as of October 2012.

1) Development of a low-grade recovery relationship for copper and nickel and the other metals
Development of a low-grade recovery relationship for copper, nickel and the other metals needs to be completed on low grade samples using a consistent metallurgical protocol. As the cutoff grade is dropped, the impact of lower grades becomes greater and also its impact on overall project economics.

2) Updating of metal payment pricing and terms
Metal prices and terms for mining planning purposes have not been updated since the DFS. With the introduction of concentrate sales, long-term marketing with Glencore, and changes to metal markets, the current cut-off is likely to exclude mineralization that would be economic to mine and process.

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3) Stockpiling options possible to increase initial mill feed grade
Current low grade ore stockpile limit is for 5 million tons of material. If the limit is increased to a higher value, the initial years mill feed grade can be increased improving overall project economics.

4) Potential for daily mine ore production increase
The NorthMet resource base and the geometry of the deposits could allow for an increase in ore tonnage.

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27 CERTIFICATES OF QUALIFIED PERSONS

27.1 Pierre Desautels, P.Geo.

I, Pierre Desautels, P.Geo, of Barrie, Ontario, do hereby certify that as one of the qualified persons (QP) of this technical report, Updated Technical Report on the NorthMet Deposit dated October 12, 2012, amended January 14, 2013; I hereby make the following statements:

- I am a Principal Geologist with AGP Mining Consultants Inc., with a business address at 92 Caplan Avenue, Suite 246, Barrie, Ontario, L4N 0Z7.
- I am a graduate of Ottawa University (B.Sc. Hons., 1978).
- I am a member in good standing of the Association of Professional Geoscientists of Ontario, Registration #1362.
- I have practiced my profession in the mining industry continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience with respect to resource modeling includes 30 years experience in the mining sector covering database, mine geology, grade control, and resource modeling. I was involved in numerous projects around the world in both base metals and precious metals deposits.
- I visited the project site from March 21 to March 23, 2007, and again from August 27 to August 29, 2007, for a period of six days in total.
- I am responsible for Sections 1.2, 1.3, the resource portion of Section 1.4 and the geology, exploration and resource portion of Section 1.8, and complete Sections 2, 3, 4.1 and complete Sections 5 through 12, Section 14, 23, 24 and the portions of Section 25 and 26 related to geology, exploration and resources of the technical report titled "Updated Technical Report on the NorthMet Deposit".
- As of the date of this Certificate, to the best of my knowledge, information, and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the issuer, PolyMet Mining Corp. as defined by Section 1.5 of the Instrument.
- I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 14th day of January 2013, at Barrie, Ontario.

"Original Signed and Sealed"

Pierre Desautels, P.Geo.

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27.2 Gordon Zurowski, P.Eng.

I, Gordon Zurowski, P.Eng, of Stoufville, Ontario, do hereby certify that as one of the qualified person (QP) of this technical report, Updated Technical Report on the NorthMet Deposit dated October 12, 2012, amended on January 14, 2013; I hereby make the following statements:

- I am a Principal Mine Engineer with AGP Mining Consultants Inc., with a business address at 92 Caplan Avenue, Suite 246, Barrie, Ontario, L4N 0Z7.
- I am a graduate of the University of Saskatchewan, B.Sc. Geological Engineering 1989.
- I am a member in good standing of the Association of Professional Engineers of Ontario, Registration #100077750.
- I have practiced my profession in the mining industry continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience includes the design and evaluation of open pit mines for the last 24 years.
- I visited the project site from October 9 to October 11, 2007 for a period of three days in total.
- I am responsible for the reserve portion of Section 1.4, the mining portion of Section 1.5, the reserves and mining portions of Section 1.8, the complete Sections 15 and 16 and the portions of Sections 25 and 26 related to mining and reserves of the technical report titled "Updated Technical Report on the NorthMet Deposit."
- As of the date of this Certificate, to the best of my knowledge, information, and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the issuer, PolyMet Mining Corp. as defined by Section 1.5 of the Instrument.
- I have read NI 43-101 and the technical report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 14th day of January 2013, at Stoufville, Ontario.

"Original Signed and Sealed"

Gordon Zurowski, P.Eng.

27.3 Karl D. Everett, P.E.

I, Karl Everett, P.E. of Duluth, Minnesota, do hereby certify that as one of the qualified person (QP) of this technical report, Updated Technical Report on the NorthMet Deposit, dated October 12, 2012, amended January 14, 2013; I hereby make the following statements:

- I am a Mining Engineer employed by Foth Infrastructure & Environment LLC, with a business address at 8550 Hudson Boulevard, Lake Elmo, MN 55042.
- I am a graduate of the Univerity of Minnesota, Duluth, Minnesota, USA, B.S. Geology 1975 and University of Idaho, Moscow, Idaho, USA, M.S. Mining Engineering , 1981
- I am a licensed Professional Engineer in Minnesota #17616 and a Professional Geologist in Wisconsin #1041.
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience includes mine planning, geology, environmental planning, permitting, reclamation and environmental compliance planning for various companies including BNI Coal, Vulcan Materials, Oglebay Norton and Barr Engineering. I have extensive experience of projects in northeastern Minnesota.
- I visited the Project site on numerous occasions, most recently on April 19, 2012.
- I am responsible for Sections 1.6, 4.7, 4.8, and the complete Section 20 of the technical report titled "Updated Technical Report on the NorthMet Deposit."
- As of the date of this Certificate, to the best of my knowledge, information, and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am independent of the issuer, PolyMet Mining Corp. as defined by Section 1.5 of the Instrument. I have read NI 43-101 and the technical report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 14th day of January 2013, at Elmo, MN.

"Original Signed and Sealed"

Karl Everett, P.E.

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27.4 David Dreisinger, Ph.D., P. Eng., F.C.I.M., F.C.A.E.

I, David Dreisinger, Ph.D., P.Eng., F.C.I.M., F.C.A.E. of Delta, British Columbia, do hereby certify that as one of the qualified person (QP) of this technical report, Updated Technical Report on the NorthMet Deposit, dated October 12, 2012, amended on January 14, 2013; I hereby make the following statements:

- I am the President of Dreisinger Consulting Inc. with a business address at 5233 Bentley Crescent, Delta British Columbia.
- I am a graduate of Queen's University of Kingston, Canada, B.Sc. Metallurgical Engineering 1980 and Ph.D. Metallurgical Engineering, 1984.
- I am a Fellow of the Canadian Academy of Engineering and am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (Registration Number 15803).
- I have practiced my profession continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience includes being employed in research and teaching at the University of British Columbia since 1984, currently holding the title of Professor and Chairholder, Industrial Research Chair in Hydrometallurgy in the Department of Materials Engineering. I have provided consulting services to the global metallurgical industry since 1987.
- I visited the Project site on numerous occasions starting in January 2004. Additionally, I have made visits to the SGS Minerals Laboratory in Lakefield, Canada to observe metallurgical testing of the Project ore since 2004. My most recent vist to site was January 21, 2009.
- I am responsible for the mineral processing portion of Section 1.5, and the complete Sections 13 and 17 of the technical report titled "Updated Technical Report on the NorthMet Deposit."
- As of the date of this Certificate, to the best of my knowledge, information, and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am not independent of the issuer, PolyMet Mining Corp. as defined by Section 1.5 of the Instrument. I currently serve as a director of PolyMet Mining Corp.
- I have read NI 43-101 and the technical report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 14th day of January 2013, at San Fransico, California.

"Original Signed and Sealed"

David Dreisinger, Ph.D., P.Eng.

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27.5 William Murray, P.Eng.

I, William Murray, P.Eng, of Richmond, British Columbia, do hereby certify that as one of the qualified person (QP) of this technical report, Updated Technical Report on the NorthMet Deposit dated October 12, 2012, amended on January 14, 2013; I hereby make the following statements:

- I am President of Optimum Project Services Ltd. with a business address at 6640 Gibbons Dr., Richmond, British Columbia.
- I am a graduate of the Strathclyde University of Glasgow, Scotland, B.Sc. Electrical Engineering 1971.
- I am a registered Professional Engineer in the Province of British Columbia, Registration #14055 and a member in good standing of the Chartered and Electrical Engineers Royal Certificate of the United Kingdom #14/14708207.
- I have practiced my profession in the mining industry continuously since graduation.
- I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
- My relevant experience includes a 40-year career with involvement at progressive stages of seniority in the evaluation and building of projects around the world in coal, iron, base metals and gold. My initial jobs with Anglo American of South Africa allowed exposure to all aspects of mine development including resource estimates; mine planning; process development; design engineering; and cost estimates. My work in recent years has also included economic valuations and market related aspects of mine development.
- I have visited the project site numerous times since the fall of 2003, have been deeply involved in its development ever since, and visited most recently from September 25-27, 2011.
- I am responsible for Sections 1.1, 1.7, 4.2 through 4.6, 4.9, 4.10, and complete Sections 18, 19, 21, and 22 of the technical report titled "Updated Technical Report on the NorthMet Deposit."
- As of the date of this Certificate, to the best of my knowledge, information, and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
- I am not independent of the issuer, PolyMet Mining Corp. as defined by Section 1.5 of the Instrument. I currently serve as a director of PolyMet Mining Corp.
- I have read NI 43-101 and the technical report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 14th day of January 2013, at Richmond, British Columbia.

"Original Signed and Sealed"

William Murray, P.Eng.

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