Environmental Mine Waste Management:
Strategies for the Prevention, Control, and Treatment of Problematic Drainages
Volume 2 of 2

Advances in Mine Waste Management Project
Final Report to the Minerals Coordinating Committee
June 30, 2001

Minnesota Department of Natural Resources
Division of Lands and Minerals

ENVIRONMENTAL MINE WASTE MANAGEMENT:

STRATEGIES FOR THE PREVENTION, CONTROL, AND TREATMENT OF PROBLEMATIC DRAINAGES

Volume 2 of 2: Appendices

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Emmelyn Leopold
Kim Lapakko

Minnesota Department of Natural Resources
Division of Lands and Minerals
500 Lafayette Road
Saint Paul, MN 55155

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APPENDIX 1

Outlined List of Mitigation References
In progress

A1.1. Alphabetical bibliography of Mitigation References in the MN DNR (Papyrus) Database. Papyrus reference numbers are in [].

Sorted by: Authors, Year, Title
Current Search: keyword='mitigation' or keyword='treatment'
Contains 932 references
Listed with Format HIBBING
Listing Created 4 Jun 2001, at 8:27

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A1.3. Summary of US, Canadian, Australian, and Swiss efforts to compile mitigative strategies for environmental metal mine waste management.

Table A1.3.1. Summary of methods to prevent or control the generation of acidic drainage.

Table A1.3.2. Summary of treatment methods for acidic drainage

APPENDIX 2

Selective Handling of Reactive Mine Wastes

Table A2.1. Bibliographic annotations of literature regarding the backfilling of open pits using tailings and waste rock. Italicized numbers refer to reference numbers for the Reclamation Section's literature database (Papyrus).

Reference	Study Scale[1]	Waste Material		Treatment or Amendment Strategy			
		Waste Rock	Tailings	Alkaline Addition	Flooded	Dry Cover	Paste or HDS
Hill & Benson (1999)	F	U					
Cravotta et al. (1994)	O	U		U			
Hockley et al. (1997)	O	U		U	U		
Chapman et al. (1998)	R, O	U		U	U		
Sevick et al. (1998)	M, L, O	U		U	U		
Dorey et al. (1999)	O	U		U	U		
Morin & Hutt (2000)	O	U		U	U		
Flambeau Mining Co. (2001)	O	U		U	U		
Peterson et al. (1998)	M	U			U		
Wickham et al. (2001)	M	U			U		
Lewis-Russ et al. (1997)	O	U			U	U	
Durkin (1995)	O	U		U		U	
Kerr et al. (1999)	O	U		U		U	
Duex (2000)	O	U		U		U	
Aiken et al. (1999)	M, L, F	U		U	U	U	
Jakubick et al. (1997)	O	U		U	U	U	
Orava et al. (1997)	R	U	U	U	U		
Cincilla et al. (1997)	R		U				U
Dahlstrom (1997)	R, F		U				U
Verburg et al. (2000)	L		U		U		U

HDS = high density solids
[1] R= review,L= laboratory, M= modeling, F= field, O= operation

Table A2.2. Bibliographic annotations of literature regarding the backfilling of underground mine workings. Italicized numbers refer to reference numbers for the Reclamation Section's literature database (Papyrus).

Reference	Study Scale[1]	Issue Addressed	
		Problematic Drainage	Stability
Dahlstrom (1997)	R		
Luppnow & Dorman (1998)	R		
Hockley et al. (1998)	M	U	
Bertrand et al. (2000)	L	U	
Verburg et al. (2000)	L	U	
Walker (1993) *#1174*	R		U
Zou (1997)	L		U
Gay & Constantiner (1998)	L		U
Lord & Liu (1998)	L, F		U
Ouellet et al. (1998)	L, O		U
Bernier et al. (1999)	L		U
Chen et al. (1999)	L		U
Qiu et al. (1999)	L		U
Amaratunga (2000)	R		U
Landriault et al. (2000)	R, O		U
Kump (2001) *#3911*	O		U
Burnett et al. (1995)	O	U	U
Helms & Heinrich (1997)	L, O	U	U
Scheetz & Schantz (1999)	L	U	U

[1] R= review,L= laboratory, M=modeling, F= field, O= operation

Table A2.3. Bibliographic annotations of literature regarding the co-disposal of tailings and waste rock. Italicized numbers refer to reference numbers for the Reclamation Section's literature database (Papyrus).

Reference	Study Scale[1]	Hydrology	Removal Mechanism	Disposal Methods	Economic Aspects
Nicholson et al. (1989) *#1131*		U			
Nichol et al. (2000)	F	U			
Lefebvre & Gelinas (1995)	O	U			
Wilson et al. (2000b)	O	U			
Lapakko & Eger (1981)	L		U		
Iwasaki et al. (1982) *#621*	L		U		
Lapakko et al. (1983)	L,F		U		
Eger et al. (1984)	L,F		U		
Lapakko et al. (1985) *#3875*	L,F		U		
Lapakko et al. (1986a) *#2849*	L,F		U		
Lapakko et al. (1986b) *#2838*			U		
LTV, 1996. Closure Plan	O			U	
Mehling et al. (1997)	O			U	
Williams (1997)				U	U
Sellgren & Addie (1998)				U	U
Johnson et al. (1995)	L				
Fortin et al. (2000)	L				
Lamontagne et al. (2000)	L				
Poulin et al. (1996) *#3896*	L				
Wilson et al. (2000a)	O				

[1] R= review,L= laboratory, M=modeling, F= field, O= operation

Table A2.4. Selected Monitoring Well Data Collected by LTV Steel Mining Company (Clark, personal communication).

HORNFELS WELL MONITORING

H-1	Well at E side of hornfels	right side of hornfels as viewed from dike							
Date	pH (s.u.)	Cu (µg/L)	Ni (µg/L)	Co (µg/L)	Zn (µg/L)	SO4 (mg/L)	Ca (mg/L)	Mg (mg/L)	T. Fe (mg/L)
29-Mar-1995	7.7	1	17	2	20	153	37	44	5.1
17-May-1995	7.6	2	8	2	20	190	60	50	14.0
16-Aug-1995	7.4	4	15	9	20	186	61	45	7.0
14-Nov-1995	7.5	11	25	12	70	177	48	40	5.7
22-Mar-1996	7.6	<1	4	9	27	160	70	45	3.4
20-Jun-1996	7.5	13.7	16.4	20.6	121	138	65.5	40	16.0
15-Aug-1996	9.8	3.8	14.2	9.7	16	152	7.8	12.6	4.4
5-Dec-1996	ns								
18-Mar-1997	ns								
11-Jun-1997	ns								
7-Aug-1997	ns								
20-Nov-1997	ns								
28-May-1998	ns[1]								
2-Sep-1998	7.3	6.2	40.6	10.1	52	162	96.2	67.5	5.3
20-Nov-1998	8.2	26.4	23.4	5.7	<10	85.1	101	98.1	41.1
25-Feb-1999	7.6	24.9	17.6	6.2	68	135	61.4	35.1	23.0
11-May-1999	9.50	96.4	19.6	2.2	41	72.4	6.4	9.7	11.0
16-Aug-1999	7.21	<1	5.1	2.7	23	200	77.5	46.2	4.3
15-Nov-1999	7.20	<1	<2	<1	<10	171	52.2	59.5	0.20
14-Feb-2000	7.50	6.5	9.9	1.2	<10	138	3.5	2.0	7.94

H-2	Well at S end of hornfels	1706.16'	center (south) of hornfels as viewed from dike

HORNFELS WELL MONITORING

Date	pH (s.u.)	T. Cu (µg/L)	T. Ni (µg/L)	T. Co (µg/L)	T. Zn (µg/L)	SO4 (mg/L)	Ca (mg/L)	Mg (mg/L)	T. Fe (mg/L)
29-Mar-1995	8.8	1	13	2	10	66	14	40	3.8
17-May-1995	8.9	5	17	4	30	74	20	40	10.0
16-Aug-1995	7.8	6	12	2	10	112	39	34	14.0
14-Nov-1995	7.8	9	14	2	30	99	28	28	21.0
22-Mar-1996	8.4	2	2	2	20	79	23	33	10.0
20-Jun-1996	7.5	12.4	13.1	2.1	31	166	67.6	36	15.0
15-Aug-1996	9.7	<1	<2	2.6	<10	151	38.6	33.6	1.74
5-Dec-1996	9.1	7.3	10.4	<1	11	155	41.7	27.7	9.25
18-Mar-1997	ns								
11-Jun-1997	8.9	2	6.8	3.8	14.7	89.1	21.0	26.5	16.8
7-Aug-1997	9.2	10.6	243	3.2	15	77.5	6.4	20.6	25.2
3-Dec-1997	9.4	2.5	<2	<1	19	61.0	<1	11.7	30.7
28-May-1998	9.9	<1	8.6	4.4	60	81.6	3.2	6.4	11.7
2-Sep-1998	7.4	2	50.5	17	181	162.0	56.3	47.6	13.2
20-Nov-1998	8.7	23.8	22.3	5.4	<10	83.8	101.0	98.7	30.6
25-Feb-1999	10.0	9.7	5.5	2.3	29	67.7	3.7	12.3	15.3
11-May-1999	9.38	125	25.3	2.5	17	85.8	6.7	5.5	13.5
16-Aug-1999	7.33	<1	<2	<1	<10	95.9	14.0	33.6	0.69
15-Nov-1999	7.60	<1	<2	<1	<10	177.0	50.9	58.4	0.23
14-Feb-2000	7.61	3.4	<2	2.1	<10	176.0	41.8	46.8	0.14

H-3	Well at W side of hornfels				left side of hornfels (as viewed from dike)				
Date	pH (s.u.)	T. Cu (µg/L)	T. Ni (µg/L)	T. Co (µg/L)	T. Zn (µg/L)	SO4 (mg/L)	Ca (mg/L)	Mg (mg/L)	T. Fe (mg/L)
29-Mar-1995	ns								
17-May-1995	7.0	3	19	3	30	729	120	190	25.0
16-Aug-1995	7.1	2	40	7	40	825	225	118	11.0
14-Nov-1995	ns								
22-Mar-1996	7.6	<1	14	3	32	140	90	48	10
20-Jun-1996	ns								
15-Aug-1996	ns								
5-Dec-1996	ns								
18-Mar-1997	ns								
11-Jun-1997	ns								
7-Aug-1997	ns								
20-Nov-1997	ns								
28-May-1998	ns[1]								
2-Sep-1998	9.9	80.8	29.6	2	20	108	15.8	8.8	9.24
20-Nov-1998	8.8	<1	23.0	5.7	41	83.8	13.2	34.6	30.6
25-Feb-1999	10.0	18.8	11.5	1.6	24	114	6.0	4.5	5.1
11-May-1999	9.45	4.3	<2	1.2	17	67	3.4	3.8	3.75
16-Aug-1999	7.95	2.7	<2	<1	<10	102	43.0	21	12.1
15-Nov-1999	7.60	<1	<2	<1	<10	188	50.2	57.4	0.23
14-Feb-2000	7.50	15.9	21.2	<1	<10	54.5	4.9	<0.5	2.78

HORNFELS WELL MONITORING

SE-1	Seep at south end of basin								
Date	pH (s.u.)	T. Cu (µg/L)	T. Ni (µg/L)	T. Co (µg/L)	T. Zn (µg/L)	SO4 (mg/L)	Ca (mg/L)	Mg (mg/L)	T. Fe (mg/L)
29-Mar-1995	7.3	1	1	1	<10	230	76	75	1.6
17-May-1995	7.3	<1	<1	6	10	336	90	90	0.6
16-Aug-1995	7.3	1	6	2	<10	259	75	98	1.0
14-Nov-1995	7.3	1	1	2	<10	220	56	68	1.6
22-Mar-1996	7.3	<1	<1	5	17	296	115	109	2.4
20-Jun-1996	7.2	12.6	<1	4.4	12	277	106	110	2.3
15-Aug-1996	7.3	<1	<2	7.8	<10	279	98.6	101	1.84
5-Dec-1996	7.5	<1	<2	1.3	<10	179	62.0	55.8	1.57
18-Mar-1997	7.6	<2	9	2.2	<10	175	48.4	37.1	0.08
13-May-1997	7.6	<1	<2	2.3	<10	162	45.0	36.5	0.07
7-Aug-1997	7.8	<1	<2	<1	<10	132	32.6	34.4	<0.03
20-Nov-1997	7.7	<1	<2	<1	<10	154	41.6	37.3	0.07
28-May-1998	7.6	3	<2	3.8	12	151	39.4	37.8	0.15
19-Aug-1998	7.6	<1	<2	2.5	13	133	40.7	39.6	0.08
9-Nov-1998	7.7	<1	13.6	2.6	<10	147	39.9	38.8	0.10
17-Mar-1999	8.2	<1	<2	1.5	19	188	38.4	40.5	0.06
14-May-1999	7.95	<1	<2	<1	17	249	90.6	109	1.73
25-Aug-1999	7.46	8.5	<2	3	13	234	88.3	105	1.59
10-Nov-1999	7.10	<1	<2	2.7	<10	225	79.5	89.1	0.98
17-Feb-2000	7.10	<1	<2	6.6	<10	261	80.9	93.7	1.35

SE-2	Seep at south end of basin								
Date	pH (s.u.)	T. Cu (µg/L)	T. Ni (µg/L)	T. Co (µg/L)	T. Zn (µg/L)	SO4 (mg/L)	Ca (mg/L)	Mg (mg/L)	T. Fe (mg/L)
29-Mar-1995	7.9	<1	2	<1	<10	227	61	68	0.1
17-May-1995	8.0	1	<1	1	<10	202	60	50	0.1
16-Aug-1995	7.9	2	7	1	<10	182	53	44	0.4
14-Nov-1995	7.9	2	1	<1	1	159	34	40	0.1
22-Mar-1996	8.1	2	<1	4	15	185	80	60	13.0
20-Jun-1996	7.9	12.0	<1	1.6	<10	203	71	64	0.35
15-Aug-1996	8.2	<1	<2	2.8	<10	234	70.7	71.0	0.23
5-Dec-1996	7.7	<1	4.1	1.3	<10	199	59.1	68.6	0.46
18-Mar-1997	8.2	<2	10	1.4	<10	330	83.0	91.5	5.05
13-May-1997	8.2	<1	<2	<1	<10	273	74.8	95.5	0.23
7-Aug-1997	8.2	<1	3.2	<1	<10	226	65.9	70.8	0.22
20-Nov-1997	8.3	<1	4.4	<1	<10	172	58.9	59.2	<0.03
28-May-1998	8.2	2.5	<2	1	<10	177	52.6	51.9	0.27
19-Aug-1998	8.0	<1	<2	<1	13	170	58.4	57.0	0.55
9-Nov-1998	7.9	<1	<2	<1	<10	164	53.0	53.7	2.38
17-Mar-1999	8.0	<1	<2	1	<10	162	44.3	42.7	0.35
14-May-1999	7.92	<1	<2	<1	14	155	43.7	43.4	0.08
25-Aug-1999	7.64	2.5	<2	<1	<10	156	43.9	49.7	0.14
10-Nov-1999	7.50	<1	<2	<1	<10	149	47.1	51.8	0.13
17-Feb-2000	7.20	<1	<2	2.6	<10	154	44.3	48.0	0.19

HORNFELS WELL MONITORING

SE-3	Seep at south end of basin								
Date	pH (s.u.)	T. Cu (µg/L)	T. Ni (µg/L)	T. Co (µg/L)	T. Zn (µg/L)	T. SO4 (mg/L)	Ca (mg/L)	Mg (mg/L)	T. Fe (mg/L)
29-Mar-1995	8.2	<1	<1	1	<10	230	58	63	<0.1
17-May-1995	8.4	<1	<1	1	<10	192	150	140	<0.1
16-Aug-1995	8.4	1	<1	1	<10	193	54	46	<0.4
14-Nov-1995	8.3	1	1	<1	1	171	38	48	<0.1
22-Mar-1996	8.3	<1	<1	3	14	200	80	65	5.0
20-Jun-1996	8.5	13.6	<1	1.6	<10	213	74	67	2.30
15-Aug-1996	8.4	5.8	<2	3.9	<10	229	68.4	70.0	0.91
5-Dec-1996	7.5	<1	2.7	<1	<10	408	81.8	125.0	0.07
18-Mar-1997	8.3	<2	9.5	2.5	32	260	70.5	75.2	1.41
13-May-1997	8.3	3.2	<2	<1	<10	235	64.1	69.8	0.15
7-Aug-1997	8.2	<1	<2	<1	<10	184	56.3	56.7	0.06
20-Nov-1997	8.2	<1	<2	<1	<10	187	61.2	57.9	<0.03
28-May-1998	8.2	<1	<2	3.8	<10	162	44.8	43.1	2.81
19-Aug-1998	7.8	1.9	<2	<1	15	141	48.2	46.9	4.24
9-Nov-1998	7.9	<1	<2	<1	<10	157	45.9	46.5	0.13
17-Mar-1999	7.9	<1	<2	1	<10	191	46.6	47.4	0.52
14-May-1999	7.90	<1	<2	<1	15	164	48.2	49.1	0.05
25-Aug-1999	7.70	3	<2	<1	<10	174	48.9	57.5	0.15
10-Nov-1999	8.30	<1	<2	<1	<10	176	49.2	56.7	0.11
17-Feb-2000	7.80	<1	<2	2.5	<10	166	44.6	52.5	0.28

HORNFELS WELL MONITORING

NE Seepage Recovery Sump									
Date	pH (s.u.)	T. Cu (μg/L)	T. Ni (μg/L)	T. Co (μg/L)	T. Zn (μg/L)	SO4 (mg/L)	Ca (mg/L)	Mg (mg/L)	T. Fe (mg/L)
9-Jun-1993		2	1	<1	<10				
29-Aug-1993		<1	<1	2	<10				
4-Nov-1993		<1	<1	<1	<10				
8-Feb-1994		1	1	<1	<10				
7-May-1994		1	<1	<1	<10				
16-Aug-1994		7	2	<1	<10				
22-Feb-1995		1	<1	1	<10				
16-May-1995		2	<1	<1	<10				
16-Aug-1995		2	8	<1	10				
31-Oct-1995		1	10	<1	10				
26-Mar-1996		<1	3	1	15				
20-Jun-1996		13	1	<1	<10				
15-Aug-1996		5	<2	3	<10				
12-Nov-1996		<1	6	<1	241				
18-Mar-1997	7.9	6.2	10.3	<1	<10	228	67.3	58.1	0.26
13-May-1997	7.9	2.7	<2	<1	<10	238	69.4	68.3	0.39
7-Aug-1997	7.8	<1	<2	<1	<10	277	79.6	99.9	0.36
20-Nov-1997	7.9	<1	<2	<1	<10	358	94.1	104	<0.03
28-May-1998	8.2	<1	<2	<1	11	241	67.3	71	0.28
19-Aug-1998	7.8	<1	<2	<1	13	200	66.3	71.4	0.25
9-Nov-1998	7.7	<1	<2	<1	<10	211	65.5	62.2	0.30
25-Mar-1999	8.2	<1	<2	<1	19	213	72.3	71.1	0.95
14-May-1999	8.40	<1	<2	<1	19	233	74.4	75.5	0.95
25-Aug-1999	7.79	<1	<2	<1	<10	170	67.7	75.1	0.38
10-Nov-1999	7.8	<1	<2	<1	<10	196	67.9	68.7	0.30
17-Feb-2000	7.3	<1	<2	2.4	<10	207	71.0	76.3	0.48

HORNFELS WELL MONITORING

NW Recovery Sump									
Date	pH (s.u.)	T. Cu (µg/L)	T. Ni (µg/L)	T. Co (µg/L)	T. Zn (µg/L)	SO4 (mg/L)	Ca (mg/L)	Mg (mg/L)	T. Fe (mg/L)
9-Jun-1993		<1	<1	<1	<10				
29-Aug-1993		<1	<1	<1	<10				
4-Nov-1993		2	<1	<1	<10				
8-Feb-1994		4	5	<1	10				
7-May-1994		1	<1	<1	20				
16-Aug-1994		8	2	<1	<10				
22-Feb-1995		4	1	<1	<10				
16-May-1995		<1	<1	<1	<10				
16-Aug-1995		2	6	<1	10				
31-Oct-1995		1	6	<1	10				
26-Mar-1996		2	<2	<1	20				
20-Jun-1996		1	<1	<1	<10				
15-Aug-1996		<1	<2	2	<10				
12-Nov-1996		<1	3	<1	11				
18-Mar-1997	7.4	<2	9	<1	14	235	60.3	71.0	0.68
13-May-1997	7.9	<1	<2	<1	<10	163	44.8	52.2	0.15
7-Aug-1997	8.3	<1	<2	<1	<10	162	50.5	69.6	0.17
20-Nov-1997	7.4	<1	<2	<1	<10	179	48.5	61.0	0.36
28-May-1998	8.1	3	3.1	1.1	10	178	56.8	65.3	0.18
19-Aug-1998	7.8	<1	<2	<1	12	158	45.4	54.0	0.23
9-Nov-1998	7.9	<1	<2	<1	<10	215	67.6	72.2	0.10
25-Mar-1999	8.7	<1	<2	<1	19	261	67.3	88.0	0.44
14-May-1999	8.03	<1	<2	<1	15	196	49.5	59.5	0.30
25-Aug-1999	7.51	<1	<2	<1	<10	188	54.0	70.9	0.19
10-Nov-1999	7.5	<1	<2	<1	<10	219	50.9	70.7	0.31
17-Feb-2000	7.7	<1	<2	4.9	<10	531	98.9	129.0	5.10

“ns” = no sample

[1] new well installed spring 1998

Table A2.5. Selected Tailings Basin Seepage Data Collected at LTV Steel Mining Company (from Berndt et al. 1999).

LTV-Seep Date	06/13/96	09/11/96	01/22/97	04/09/97	07/16/97	10/09/97	02/04/98	04/08/98	07/23/98	10/07/98	02/10/99
Field Parameters											
Temp (EC)	13	14	11	10	11	14	10	10	10	9	9
Conductivity	1430	1200	1300	1050	1100	1200	1150	900	1350	1100	1250
pH	7.23	7.70	7.30	7.41	7.31	7.45	7.38	7.28	7.22	7.16	7.12
Measured Eh	-48	171	-178	25	2.4	-23	32	117	-38	103	25
Corrected Eh	161	380	33	237	213	186	244	329	174	316	237
majors (ppm)											
Al	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.01	0.01	0.00
Si	11.2	9.3	10.0	10.5	9.3	10.2	10.6	11.1	11.1	12.3	11.4
Fe	2.18	0.09	0.73	1.73	1.20	1.50	2.42	2.49	2.50	2.66	2.70
Mn	1.43	1.23	0.82	0.69	1.00	0.94	0.88	0.91	1.00	0.98	0.99
Sr	0.50	0.24	0.28	0.27	0.33	0.35	0.35	0.38	0.43	0.43	0.40
Ba	0.05	0.02	0.04	0.04	0.04	0.05	0.05	0.05	0.06	0.06	0.05
Ca	96.3	57.0	65.9	50.9	67.3	68.7	65.6	70.6	80.5	82.7	80.6
Mg	109.0	65.4	68.7	49.0	70.9	72.3	69.8	74.1	84.8	87.8	86.8
Na	91	107	140	110	114	107	107	107	102	103	100
K	15.4	16.2	20.9	16.4	17.3	16.3	16.3	16.2	16.2	15.6	15.4
F	1.8	4.4	4.6	4.8	4.8	4.3	4.3	4.2	3.5	3.3	3.3
Cl	31.7	31.9	36.2	35.5	35.7	34.6	32.4	31.4	29.1	28.9	29.1
Br	0.12	0.16	0.16	0.18	0.17	0.16	0.18	0.17	0.15	0.14	0.14
NO3-N	0.18	0.19	0.43	0.47	0.09	1.47	0.11	0.16	0.27	0.35	0.86
SO4	284	207	263	169	237	217	206	220	251	243	234
HCO3-*	530	415	427	342	439	462	450	500	490	524	515
trace (ppb)											
B	383	468	515	457	455	338	453	483	526	527	422
Co	5.1	1.9	3.7	1.6	4.0	2.4	2.0	1.6	2.3	2.0	2.2
Ni	1.9	1.5	0.9	1.7	1.3	1.2	1.7	1.5	3.3	2.7	2.0
Cu	1.2	1.1	0.0	0.0	0.8	0.8	0.9	0.6	1.5	0.5	0.6
Zn	7.8	28.9	30.2	23.7	5.0	17.7	18.1	12.5	15.0	13.9	17.7
As	3.7	2.2	4.8	5.8	4.9	3.7	5.9	4.7	6.1	5.8	5.5
Mo	15	129	123	106	102	86	95	84	67	71	74
W	0.11	0.18	0.75	0.15	0.13	0.12	0.22	0.23	0.81	3.81	1.65
Pb	0.03	0.08	0.48	0.08	0.10	0.00	0.07	0.00	0.09	0.07	0.06
Nutrients (ppm)											
N		<.20	0.22	<0.3	0.28	0.29	0.33	0.46			
NH3-N		0.12	0.32	0.24	0.34	0.27	0.31	0.23			
TP							0.01	0.03			
Notes	*reported as ppm CaCO3										

APPENDIX 3

Physical Isolation of Reactive Mine Wastes

Table A3.1. Bibliographic annotations of research related to dry cover systems for reactive mine wastes. Italicized numbers refer to reference numbers for the Reclamation Section's literature database (Papyrus).

Reference	Study Scale[1]	Barrier Type		Barrier Material					
		Simple	Multi-layer	Soil	Clay	Mine Waste	Organic	Hard pan	Syn-thetic
Mannheim (1970).*#2663*	R	U		U					
Shackelford (1991) *#2665*	L	U		U					
Wilson et al. (1991)	L	U		U					
O'Kane (1993) *#1412*	M	U		U					
Swanson et al. (1995)	M	U		U					
Williams et al. (1997)	O	U		U					
McLendon et al., (1997)	O	U		U	U				
Rowe et al. *#2664*	L	U			U				
Bowders et al. (1994)	L	U			U				
Aziz & Ferguson (1997)	F	U			U				
Lindvall et al. (1997)	O	U			U				
Lundgren (1997)	O	U			U				
Kowalewski et al. (1998)	M	U			U				
Woyshner &Swarbrick (1997)	F	U			U	U			
Muller et al. (1998)	M, O	U				U			
Stogran & Wiseman (1995)	M, L	U				U	U		
Elliott et al. (1997) Tailing & Mine Waste	L, F	U					U		
Elliott et al. (1997) ICARD	L, F	U					U		
Chtaini' et al. (1997)	L, F	U					U		
Tasse et al. (1997)	O	U					U		

Reference	Study Scale[1]	Barrier Type		Barrier Material					
		Simple	Multi-layer	Soil	Clay	Mine Waste	Organic	Hard pan	Synthetic
Tasse (1999)	O	U					U		
Tasse (2000)	O	U					U		
Germain et al. (2000)	O	U					U		
Boorman & Watson (1976) *#969*	O	U							U
Stewart & vonMaubeuge (1997)	R	U							U
Miller & Hornaday (1998)	R	U							U
Brennecke & Corser (1998)	O	U							U
Neukirchner & Lord (1998)	O	U							U
Owaidat & Day (1998)	O	U							U
Shuri et al. (1998)	O	U							U
Sevick et al. (1998)	O	U							U
Lewis et al. (2000)	O	U							U
Udoh (1993) *#2866*	M, F	U		U	U				U
Rasmuson & Eriksson (1987) *#912*	R		U	U					
Rasmuson & Collin *#1590*	M		U	U					
Nicholson et al. (1989) *#1131*	M		U	U					
Barbour (1990) Can. Geotech. J. 27, 398-401	R		U	U					
Collin & Rasmuson (1990) *#1504*	M		U	U					
Akindunni et al. (1991)	M		U	U					
Nicholson et al. (1991) *#1395*	M, L		U	U					
Yanful (1991) *#1396*	M, L		U	U					

Reference	Study Scale[1]	Barrier Type		Barrier Material					
		Simple	Multi-layer	Soil	Clay	Mine Waste	Organi c	Hard pan	Syn-thetic
Yanful et al. (1993) 578	M, O		U	U					
Yanful et al. (1993) 588	O		U	U					
Bell et al. (1994)	O		U	U					
O'Kane et al. (1995)	L, F		U	U					
Aubertin et al. (1997)	M		U	U					
Wilson et al. (1997),	M, O		U	U					
Woyshner et al. (1997)	O		U	U					
Bussiere et al. (2000)	M		U	U					
Yanful & St-Arnaud (1991) *#1397*	F		U		U				
Yanful (1993)	L, F		U		U				
Yanful et al. (1994)	M, O		U		U				
Strachan & Raabe (1998)	O		U		U				
Duex (2000)	O		U		U				
Timms & Bennett (2000)	O		U		U				
Aachib et al. (1994)	L		U			U			
Bussiere et al. (1995)	L		U			U			
Puro et al. (1995)	O		U			U			
Bussiere et al. (1997)	L		U			U			
Hanton-Fong et al. (1997)	O		U			U			
Ricard et al. (1997)	M, O		U			U			
Benzaazoua et al. (1998)	L		U			U			
O'Kane et al. (1998) *#2863*	M,L,F		U			U			
Zahn et al. (2001)	M, F		U			U			
Elliott et al. (1997)	L		U			U	U		

Reference	Study Scale[1]	Barrier Type		Barrier Material					
		Simple	Multi-layer	Soil	Clay	Mine Waste	Organi c	Hard pan	Syn-thetic
Cabral et al. (1997) Tailings & Mine Waste	F		U				U		
Cabral et al. (1997) ICARD	F		U				U		
Cabral et al. (1998)	L, O		U				U		
Cabral et al (1999)	O		U				U		
Hanselman & Courtin (1999)	L		U				U		
Blowes et al. (1991) *#3860*	O		U					U	
Ahmed (1994)	L		U					U	
Ahmed (1995)	L		U					U	
Chermak & Runnells (1997) *#2745*	L		U					U	
McGregor et al. (1997)	F		U					U	
Lin & Herbert (1997)	O		U					U	
Tasse et al. (1997)	O		U					U	
Ettner & Brasstad (1999)	O		U					U	
Shay & Cellan (2000 *#3682*	O		U					U	
Aubertin et al. (1997)	F		U	U		U			
Gardiner et al. (1997)	M, O		U	U		U			
Delaney et al. (1997)	R	U	U	U	U		U		U
Swanson et al. (1997)	O	U	U	U	U	U			

[1] R= review, M= model, L= laboratory, F= field, O= operation

Table A3.2. Summary of Operational-Scale Engineered Dry Cover Systems

UNITED STATES

Dunka Mine, Babbitt, Minnesota

Waste Material:	50 million tons of waste rock & gabbro (in 8014 and 8018) covering 320 acres
Cover Systems:	SMR (min. 2' till & vegetation): stockpiles with no known water quality problems EMR (min. 2' uncompacted -6" material w/ min. 10% at -200 mesh & vegetation): stockpiles 8011, 8012, 8031 FML (6" sand, 30 mil LDPE, 12" sand & vegetation): stockpile 8018 EMR-FML: stockpile 8013, 8014
Dates:	pile 8011: deposition from 1965-1985, capped 1995 pile 8013: deposition from 1967-1991, capped 1991 pile 8014: deposition from 1967-1993, capped 1995 pile 8018: deposition from 1979-1985, capped 1995 pile 8031:deposition from 1979-1985, capped 1995
Parameters Monitored:	flow, Cu, Ni, Co, Zn
Water Infiltration:	
Oxygen Flux:	
Effluent Quality (mg/L):	
Cost Estimates:	
References:	MN DNR, various published and unpublished reports

Ridgeway Mine, South Carolina:

Waste Material:	50 million ton, 300-acre gold tailings impoundment with 62 million ton capacity
Cover System:	preferred plan = 36" saprolite and 7" topsoil and vegetation
Dates:	began Mining in 1987, cover system modeling completed in 1998
Water Infiltration:	
Oxygen Flux:	
Effluent Quality (mg/L):	
Problems Identified:	
References:	Kowalewski et al. (1998) Tailings & Mine Waste pg 369

Homestake's Santa Fe Mine, west central Nevada:

Waste Material:	waste rock dump
Cover System:	cap purpose to neutralize existing near surface acidity and block diffusion of acid solutes. 20 t/ac magnesite and brucite applied uniformly over the surface, covered w/ 8-12" topsoil
Dates:	began leaching ore in 1988 and completed ore processing in 1995
Water Infiltration:	
Oxygen Flux:	

Results:	Monitoring data from the site has indicated that the chemical cap has been successful in preventing the diffusion of acidity into cover soil materials and the long-term goal of revegetation success is likely.
Cost Estimates:	cost of grading, purchase, hauling, application, revegetation, & engineering = $776,000
References:	Shay and Cellan (2000) Proc. 2000 Billings Land Reclamation Symp. CD-ROM

Richmond Hill Mine, Lawrence Co, South Dakota:

Waste Material:	waste rock
Cover System:	The capping system= 6"of limestone, 18" clay, 4.5' of thermal barrier/drain layer, and 6" topsoil.
Dates:	Operation 1988-1993, ARD identified 1992, pit backfill 1994, cap completed 1995.
Parameters Monitored:	An extensive program is in place to monitor the effectiveness of the reclamation plan, and includes measurements on moisture content movement, temperature, oxygen content, and groundwater impacts.
Water Infiltration:	water received by the south sump decreased from 1.54 gpd in 1995 to 0.43 pgd in 1998
Oxygen Flux:	generally less than 0.2%, a few readings as high as 4.5%
Effluent Quality (mg/L):	
Cost Estimates:	
References:	Duex, T.A. ICARD, 2, 807-811 (2000)

CANADA

Equity Silver Mine, near Smithers, British Columbia:

Waste Material:	Southern Tail waste dump= approx 17 million tons gabbro waste rock from the Main Zone Pit was backfilled into the Southern Tail Pit
Cover System:	initially covered w/ 1 m uncompacted till, new cover design= 0.5 m of compacted clay/till overlain by 0.3 m of loose till and then revegetated
Dates:	mine operation 1980 - 1994, large field-scale test of a compacted clay cover evaluated in 1990, compacted clay cover constructed between 1990 and 1994 for the Southern Tail, Main, and Bessemer dumps.
Water Infiltration:	average infiltration has remained below 5% (0.2 - 12%)
Oxygen Flux:	Oxygen concentrations below the cover have decreased to a few percent.
Effluent Quality (mg/L):	no direct data provided, lime consumption at the treatment plant peaked in 1990 and decreased through 1996
Cost Estimates:	Estimated average cost at $35,000/hectare.
References:	Aziz and Ferguson (1997) ICARD, 1, 183 Wilson et al. (1997) ICARD, 1, 197 O'Kane et al. (1995) Mining & the Environment, 565

Cominco Sullivan Mine, Kimberly, British Columbia:

Waste Material:	90 mt of reactive tailing containing pyrrhotite and pyrite were placed in ponds that cover 373 hectares
Cover System:	Field test: Plot 2= 20-60 cm float rock, 25 cm compacted till and 30 cm non-compacted till; plot 3= 20-60 cm float rock and 45 cm non compacted till
Dates:	close in 2001 after 90 yrs of continuous operation, tailings disposal began 1923, field cover investigation began 1993
Water Infiltration:	non-compacted = 28%, compacted = 6%
Oxygen Flux:	non-compacted = 14-20%, compacted = 2-3% wet season & 12-15% dry season
Effluent Quality:	no direct data, treatment plant lime consumption for non-compacted=482 t, compacted=270 t
Cost Estimates:	$15 million ($4.00/m^2)
References:	Gardiner et al. ICARD, 1, 47 (1997)

Kidd Creek tailings, near Timmins, Ontario:

Waste Material:	thickened tailings impoundment
Cover System:	Test plot 1: 60 cm nonreactive tailings Test plot 2: 15 cm slag under 45 cm nonreactive tailings Test plot 3: 60 cm native clay
Dates:	installed in 1995, monitoring results for 1995 & 1996
Water Infiltration:	tailings beneath the covers retained saturation, 0.85-0.89

Oxygen Flux:

Cover	Annual O_2 Flux (mol/m^2/yr)	SO_4 Production (kg/m^2/yr)
NR tailings	36.1	2
slag & NR tailings	73.7	4
clay	13.1	1
none after 1 yr (prediction)	729	40
none after 20 yrs (prediction)	91.1	5

Effluent Quality (mg/L):
Cost Estimates:
References: Woyshner and Swarbrick, ICARD, 3, 1075 (1997)

East Mine, Falconbridge, Ontario:

Waste Material: tailings impoundment

Cover System: 75 cm over 100 m^2 was excavated and refilled with 26-30 cm reactive waste, the self-sealing/self-healing barrier, 17-23 cm high Fe-S (mill washings) material and filled to the top with the original oxidized tailings that had been excavated.

Dates: tailings deposited prior to 1985, cover installation took place in 1995

Water Infiltration: Testing of cores taken through the barrier determined an average vertical K of 3.6×10^{-8} cm/s for the barrier compared to an average K of 3.9×10^{-4} cm/s for the surrounding tailings.

Oxygen Flux: Pore gas oxygen measurements within the vadose zone of the tailings revealed a marked decrease in O_2 concentration below the barrier (<3.4%) when compared to the O_2 profile of the shallow tailings beside the barrier test plot.

Other Pore Gases: Pore gas measurements also determined an accumulation of pore gas CO_2, of up to 17% of the pore gas, below the barrier.

Cost Estimates: Estimated cost: $30-50k/ha.

References: McGregor et al. (1997) ICARD, 3, 1435

East Sullivan Mine, near Val d'Or, Quebec:

Waste Material: 136 ha tailings impoundment

Cover System: forestry wastes produced by sawmiill and presswood industries,

Plot	Organic Cover	Tailings
0	none	weathered
1	none	fresh
2	1 m bark	weathered
3	1 m bark	fresh
4	2 m bark	weathered
5	1 m bark and 30 cm sewage sludge	weathered

Dates: tailings accumulation 1949-1966, forestry wastes laid down between 1984-1996 cover almost 75% of the 136 ha tailings impoundment

Oxygen Flux: O_2 < 3-5%

Pore Gases: CO_2 20-30% and CH_4 10--40%

Effluent Quality (mg/L): pH > 5.5-6, > 2000 ppm $CaCO_3$

Cost Estimates:

References: Tasse et al., ICARD, 4, 1627 (1997)

Norebec-Manitou site, near Val d'Or, Quebec:

Waste Material: sulfidic tailings from the Manitou-Barvue site

Cover System:

Field Test Plot	Reactive Tailings (m)	Sand (m)	Fine Layer (m)	Sand (m)
1	1.6	0.4	0.60 tailings	0.3
2	1.58	0.4	0.60 silt	0.3
3	1.52	0.4	0.30 tailings	0.3
4	1.21	0.4	0.15 tailings & 0.15 tails/bentonite	0.3
5	1.55	0.4	0.90 tailings	0.3
6	1.5	---	---	---

Water Infiltration: The coarse layers maintained saturation ratios of 25-40%, while the fine layers were about 85%.

Oxygen Flux: An effective oxygen barrier is considered to be 80-90% saturated, depending on thickness of the layer.

Cost Estimates: Total construction costs for six cells 70k$US, and instruments totaled about 45k$US.

References: Aubertin et al. (1997) ICARD, 2, 715

Millenbach Site near Rouyn-Noranda, Quebec:

Waste Material: moved 17500 m^3 of waste rock and fine material from various locations on site onto 0.66 ha of tailings

Cover System: composite clay cover (30 cm coarse sand, 50 cm compacted clay, 30 cm fine sand, 10-15 cm top soil).

Dates: operation 1971-1980, decommissioned during 1990 and 91

Water Infiltration: detailed calculations given

Oxygen Flux: 0.56 $mol/m^2/yr$

Effluent Quality (mg/L): oxidation products = 184 kgO_2/yr, 89 $kgFe^{++}/yr$, 306 $kgSO_4/yr$, 319 kg-acidity as $CaCO_3/yr$

Cost Estimates:

References: Woyshner et al., ICARD, 4, 1673 (1997)

Waite Amulet site, near Rouyn-Noranda, Quebec:

Waste Material: tailings

Cover System: 20 x 20 m test plots (max slope = 3:1): plot 1= control; plot 2 & 3= 30 cm sand, 60 cm compacted, varved clay, 30 cm sand, 10 cm gravel; plot 4= 30 cm sand, 80 mil HDPE, 30 cm sand, 10 cm gravel

Dates: project initiated 1990,

Water Infiltration: Infiltration through the soil covers = 3.9% of the precipitation during the study.

Oxygen: 2.0% beneath the soil covers, 6% beneath HDPE

Effluent Quality:

Oxidation Product (g/L)	Uncovered tailings	Covered tailings
Fe	5-20	1-1.5
SO_4	16-63	5-5.7

Cost Estimates:

References: Yanful et al. (1994) Pitt94, 2, 138

Les Terains Auriferes Site, Malartic, Quebec:

Waste Material: Goldfield carbonate tailings: about 10 Mt placed in a 5 m deep layer in a 100 ha tailings pond; Bousquet sulfide tailings: about 8 Mt placed on top of Goldfields tailings

Cover System: Zone 1: 0.5 m sand and gravel
Zone 2: 0.5m sand and gravel, 0.8 m compacted tailings, 0.3 m sand

Dates: closed 1994, construction began 1995

Parameters Monitored:

Water Infiltration: 6 months of monitoring show an average of 86% saturation on top of the stack and 84% on the slopes.

Oxygen Flux: Oxygen consumption measurements show reduced oxygen flux by an average factor of 75, up to 1000.

Effluent Quality (mg/L):

Cost Estimates: construction = \$65,000/ha, total = \$93,500/ha

References: Ricard et al. (1997) ICARD, 4, 1515; Tremblay and Bussiere (1999) MEND 2000 Workshop ME.02

Clinton site, near Woburn, Quebec:

Waste Material: waste rock backfill in a 2000 m^2 x 20 m mine shaft

Cover System: 5000 m^2 capillary barrier= 30 cm coarse waste rock, 120 cm deinking residues, 50 cm granular material, 10 cm vegatative cover

Dates: project began 1997

Water Infiltration: $k = 1\text{-}4 \times 10^{-6}$ cm/s

Pore Gases: O_2 <2%, CO_2 - 38%, CH_4 - 38%

Effluent Quality (mg/L): pH=7.2, Cu<0.005, Zn=1
Cost Estimates:
References: Cabral et al. (1999) Tailings & Mine Waste, 405

Eustis Mine site, near Sherbrooke, Quebec:

Waste Material: abandoned tailings, 1.6 ha test plot
Cover System: deinking residue mixed w/ a "superficial crust" of oxidized tailings, fresh compacted residues, 20 cm compost-residue
Dates: mine active from 1865-1939, project began Aug. 1995, cell construction completed spring 1996,
Water Infiltration: degrees of saturation in the range of 76%-85%
Oxygen: diffusion coefficients in the range of 3.5 m^2/yr to 110 m^2/yr
Pore Gases: Oxygen, carbon dioxide, and methane profiles showed that the oxygen concentration 20 cm below the surface was virtually nil and that the concentrations of carbon dioxide and methane increased with time.
Effluent Quality (mg/L):
Cost Estimates:
References: Cabral et al. (1998) Tailings & Mine Waste, 379
Cabral et al. (1997) Tailings & Mine Waste, 257
Cabral et al. (1997) ICARD, 3, 1109

Poirier Site, northern Quebec:

Waste Material: 5 million tonnes acid generating tailings
Cover System: geomembrane clay cover = 0.5 m compacted clay on top of a geomembrane liner, overlain by 0.5 - 1.5 m till for vegetation
Dates: underground Cu/Zn mine operated 1966-1975, property sold in 1985, closure planning began in 1996, implemented in 1998, scheduled for completion in 2000
Water Infiltration:
Oxygen:
Pore Gases:
Effluent Quality (mg/L):
Cost Estimates:
References: Lewis et al. (2000) ICARD, V2, 959; Gallinger (1999) MEND 2000 Workshop ME.02

Heath Steele Pile 7/12, near Newcastle, New Brunswick:

Waste Material: 14000 t of mine waste rock and has been producing acidic seepage (pH = 2.1-2.8, Fe = 3.5-13.5 g/L, SO_4 = 12.7-43.4 g/L)
Cover System: Pile 7/12 was relocated covering 0.25 ha in plan, and has a maximum depth of 5 m (maximum slope of 3:1). The waste rock pile was underlain by a sand layer and an impermeable membrane. Composite cover design= 130

cm thick soil cover (30 cm sand base, 60 cm compacted till, 30 cm granular, 10 cm gravel for erosion protection-58% sand, 40% gravel).

Dates: waste relocated 1989, closed 1994

Water Infiltration: There has been no noticeable change in the moisture content of the glacial till layer.

Oxygen Flux: Reported oxygen concentrations dropped from an initial range of 7.3 - 20.8 % to 8.2 - 14.5% immediately after the cover was in place to 0.2 - 0.7 % after five years.

Effluent Quality (mg/L):

(g/L)	1989-90	1992	1993
pH	2.1-2.8	2.3-2.9	3.0-3.2
acidity (as $CaCO_3$)	15.8 - 76	15.8 - 54	not available
SO_4	12.7 - 43	5.1 - 71	10 - 74
Fe	3.5 - 13.8	15.8 - 54	5.0 - 31

Cost Estimates:

References:
Yanful et al. Can. Geotech. J., 30, (1993) 578-87.
Yanful et al. Can. Geotech. J., 30, (1993) 588-99
Bell et al. Pitt94, 2, 113.

AUSTRALIA

Rum Jungle, Northern Territory:

Waste Material:	Whites uranium and copper waste rock dump
Cover System:	3 layers = 0.225-0.30 m compacted clay, 0.25-0.30 m sandy clay loam, 0.15 m gravely sand
Dates:	mined during 1950's til early 70's, cover was put in place in 1983
Parameters Monitored:	
Water Infiltration:	approx. 2% of precipitation til 1994, increasing to 10% in 1999
Oxidation Rate:	decreased from 0.0466 to 0.137 kg/s
Effluent Quality (mg/L):	SO_4 generation rate decreased from 2520 to 740 t/yr
Cost Estimates:	
References:	Timms and Bennett (2000) ICARD, 2, 813-818.

Mt Whaleback:

Waste Material:	2 billion tons of waste rock over 30 years
Cover System:	The cover system is designed to maximize infiltration during wet periods for subsequent evapotranspiration while minimizing surface runoff.
Dates:	BHP Iron Ore initiated research programs in January of 1995 to develop long term plans for decommissioning of the waste rock material at their Mt. Whaleback operation.
Results:	Monitoring is in progress and will continue for at least two annual wet/dry cycles.
Water Infiltration:	
Oxygen Flux:	
Effluent Quality (mg/L):	
Cost Estimates:	
References:	O'Kane et al. (1998) SME Preprint 98-70

SWEDEN

Bersbo Pilot Project:

Waste Material:	about 700,000 m^3 of waste rock from an abandoned, historic copper mine
Cover System:	The remediation works included concentration of the waste rock to two piles reducing the exposed area to 50% and dumping 1/3 of the waste volume under the groundwater table in old mine shafts. A sealing layer of compacted clay (0.5 m) installed on one of the piles and cement stabilized fly ash (0.25 m) on the other pile, both covered by a protective cover of 2 m glacial till.
Dates:	remediation project 1987-1989, follow-up & monitoring since 1989
Water Infiltration:	The water percolation has been significantly reduced to less than 3% (clay liner) and 12% (cementitious liner) of what it was prior to covering.
Oxygen Flux:	The oxygen transport is reduced substantially through the clay liner. The cementitious liner, however, is a less good oxygen barrier, partly due to the higher permeability, partly due to the not fully saturated sealing layer.
Cost Estimates:	
References:	Lundgren (1997) ICARD, 3, 1419

Saxberget Mine, near Ludvika:

Waste Material:	sulfidic tailings producing Zn
Cover System:	0.3 m sealing layer of compacted clayey till applied in two lifts, 1.5 m protection layer of unsorted till, and vegetated
Dates:	began in summer of 1993, completed in autumn of 1995
Water Infiltration:	permeability = 5×10^{-9} m/s,
Oxygen Flux:	O_2 <0.5%
Effluent Quality (mg/L):	included in follow up studies
Cost Estimates:	\$14 CND/m^2
References:	Lindvall et al. (1997) ICARD, 3, 1389-1400

NORWAY

Storwartz Mine, near Roros:

Waste Material:	tailings impoundment
Cover System:	Four $4m^2$ induced hardpan test plots: cell 1=3cm fine and coarse limestone covered by 2 cm oxidized tailings; cell 2= 3cm fine and coarse limestone covered by 2 cm unoxidized tailings; cell 3= control; cell 4= 3 cm quicklime & crushed limestone covered by 2 cm unoxidized tailings
Dates:	issue related remediation & preserving historic Mining areas
Hardpan Description:	am-FeOOH clumps & coatings, jarosite, gypsum
Water Infiltration:	k - $9x10^{-6}$ m/s compared to $1x10^{-5}$ m/s in control
Oxygen Flux:	top 20 cm of tailings=8-18%, decreased to <1% by 40-50 cm depth
Cost Estimates:	
References:	Ettner and Brasstad (1999) Tailings & Mine Waste, 457

Table A3.3. Summary of Natural Hardpan Formations on Mine Waste Materials

Unknown Location:

Waste Material:	inactive sulfide-containing mill-tailings deposit
Hardpan Description:	The Fe precipitates consist mainly of goethite, and lesser amounts of lepidocrocite and other Fe hydroxides, and traces of Ca, Cu, Fe, and Zn sulfates.
General Results:	The Fe precipitates fill the intergrain pores, and cement the tailings matrices, leading to a reduction of the tailings porosity. In addition, dissolved elements, e.g. As, Cu, Zn, are retained by Fe oxyhydroxides through adsorption and co-precipitation. As a result, the Fe precipitated layers behave as a barrier to restrict the movement of oxygen, and act as an accumulation zone of heavy metals and As. The extent of sulfide oxidation, and the concentration of dissolved metals, is greatly reduced below the Fe precipitated layers.
Effluent Quality:	below the precipitated layers: pH = 6.2 - 6.5; concentration of dissolved metals = a few ppm
Pore Gases:	Carbon dioxide levels below the cemented layers were recorded at 60% compared to 15% above the layers. The oxygen diffusion coefficient was 1.7×10^{-8} m^2/s compared to 2.0×10^{-6} m^2/s in uncemented tailings.
References:	Lin and Herbert (1997) Tailings & Mine Waste, 237

Canadian Malartic Mine, Quebec:

Waste Material:	The Canadian Malartic impoundment features a layer cake structure with 6.9 Mt of mildly reactive tailings at the base, generated by gold Mining, and 1.2 Mt of strongly reactive, sulfurous, tailings at the top, produced from Ni-Cu extraction.
Dates:	first deposition cycle (Malartic) 1942-1965, second (Marbridge) in the 1960's
Hardpan Description:	Near surface reactive sulfides have already reacted, and oxygen access to fresh sulfides is limited by the presence of a fairly continuous hardpan layer. The surface impoundment was leveled and revegetated with minimal disturbance, keeping the natural drainage pattern developed over the site, and avoiding soil disruptions at depths larger than a few tens of centimeters. Where slope reshaping involved hardpan disturbance, reactive tailings were carried and disposed in a safer location on the site.
Results:	Heavy metals show little mobility within and around the impoundment periphery, as a result of precipitation, coprecipitation, or adsorption. Runoff in direct contact with altered tailings devoid of any significant neutralizing capacities, generates the most contaminated waters, but there are no significant accumulations of such waters.
References:	Tasse et al. (1997) ICARD, 4, 1797.

Waite Amulet, Quebec:

Waste Material:	Cu-Zn massive sulfide tailings deposited in a 41 ha elevated impoundment (2-15 m deep), 30 wt% sulfide minerals (pyrite, pyrrhotite, sphalerite, and chalcopyrite; traces of galena)
Dates:	mill operations from 1929 to 1964
Hardpan Description:	1-5 cm thick, laterally discontinuous (10-100 cm), occurs at the depth of active oxidation, and is characterized by cementation of tailings by Fe(III) minerals
Results:	
References:	Blowes et al. (1991) Geochim. Cosmochim. Acta, 55, 965-978.

Heath Steele, New Brunswick:

Waste Material & Dates:	Two tailings impoundments, an older one 40 ha where tailings were deposited between 1957 and 1963, and a newer one 200 ha where tailings deposition began in 1965.
Hardpan Description:	10-15 cm thick, occurs 20-30 cm below the depth of active oxidation, is continuous throughout the tailings impoundment, and is characterized by cementation of tailings by gypsum and Fe(II) solid phases, principally melanterite.
Results:	Accumulation of gas-phase CO_2, of up to 60% of the pore gas, occurs below the hardpan. The calculated diffusivity of the hardpan layer is only about 1/100 that of the overlying uncemented tailings. The pore-water chemistry has changed little over a 10-yr period, suggesting that the cemented layer restricts movement of dissolved metals and acts as a zone of metal accumulation.
References:	Blowes et al. (1991) Geochim. Cosmochim. Acta, 55, 965-978.

Table A3.4. Bibliographic annotations of research related to wet cover systems for reactive mine wastes. Italicized numbers refer to reference numbers for the Reclamation Section's literature database (Papyrus).

Reference	Study Scale[1]	Disposal Setting: Tailings Basin	Constructed Wetland	Open Pit	Under-ground Mine	Natural Lake	Marine
Aubertin et al. (1997)	R						
Dave et al. (1997) *#3450*	L						
Stogran et al. (1997)	L						
Paktunc & Dave (1999)	L						
Nicholson et al. (1999)	L						
Nicholson & Rinker (2000)	M, L						
Amyot & Vezina (1997)	O	U					
Atkins et al. (1997)	M, F	U					
Kam et al. (1997)	O	U					
Li et al. (1997)	M, O	U					
Ljungberg et al. (1997)	F	U					
Robertson et al. (1997)	F	U					
St-Germain et al. (1997)	F	U					
Beckett et al. (1999)	O	U					
Evans (1999)	L, F	U					
Lindvall et al. (1999)	O	U					
Zou & Huang (1999)	L	U					
Catalan et al. (2000)	O	U					
DeVos et al. (2000)	M, O	U					
Li et al. (2000)	M, F, O	U					
Jones et al. (1997)	M, O		U				
Eger et al. (2000)	F		U				
Atkins et al. (1997)	M, F			U			
Bursey et al. (1997)	M, F			U			

Reference	Study Scale[1]	Disposal Setting					
		Tailing s Basin	Constructed Wetland	Open Pit	Under-ground Mine	Natural Lake	Marine
Hamblin et al. (1997)	M, F			U			
Meyer et al. (1997)	M			U			
Warren et al. (1997)	M			U			
Dorey et al. (1999)	O			U			
Day et al. (2000)	O			U			
Crusius, et al. (2001)	M			U			
Arnesen & Iversen (1997)	O				U		
Neukirchner & Hinrichs (1997)	O				U		
Hockley et al. (1998)	M, O				U		
Quarshie & Rowson (1995)	L					U	
Pedersen et al. (1997)	O					U	
Pedersen et al. (1999)	O					U	
Morin & Hutt (2000)	O					U	
Mugo et al. (1997)	L					U	U
Mugo et al. (1999)	M, L					U	U
Ganguli et al. (2001)	R						U
Orava et al. (1997)	R	U		U			
Arnesen et al. (1997)	M, O	U				U	
Robertson et al. (1997)	O	U				U	

[1] R= review, M= model, L= laboratory, F= field, O= operation

Table A3.5. Summary of Operational-Scale Underwater Cover Systems

UNITED STATES

Eagle Mine, Minturn, Colorado

Disposal Setting:	inactive Zn-Pb ore body
Waste Material:	pyrite, sphalerite & galena
Dates:	operations discontinued in 1977, flooding began in 1983, water quality monitoring began in 1989
Parameters Monitored:	pH, Zn, Mn, SO_4 inflow
Results:	pH increased from 3 to 6, Zn decreased from 350 mg/L to 20-50 mg/L,Mn decreased from 314-380 mg/L to 40-54 mg/L, SO_4 decreased from >11,000 mg/L to 2600-2970 mg/L, inflow has decreased by about 1/3
References:	Neukirchner & Hinrichs (1997) ICARD, V4, 1469.

Kennecott Ridgeway Gold Mine, South Carolina

Disposal Setting:	flooding backfilled open pits to create deep water lakes, pits will be backfilled at a pace that allows for rapid innundation (<4 months)
Waste Material:	backfilled waste rock and alkaline (lime) amended water
Dates:	reclamation of the South Pit began around 1998, the North Pit was still in production
Parameters Monitored:	
Results:	Water quality in the South Pit meets drinking water standards with minor occurrences of Zn, Cu, & Fe at ppb levels. Algae discovered in the pit water in 1998 suggests continued development of a viable fresh water lake environment.
References:	Dorey et al. (1999) Tailings & Mine Waste, 701

CANADA

Island Copper Mine, Vancouver Island, British Columbia

Disposal Setting:	flood the pit with seawater from Rupert Inlet
Waste Material:	acidic drainage from waste rock injected at bottom of pit
Dates:	operation 1971-1996?
Parameters Monitored:	
Results:	
References:	MEND Report 2.36.1 (1995)

Eskay Creek Mine, British Columbia

Disposal Setting:	biologically-barren alpine lake, lime was added to each truckload
Waste Material:	waste rock
Dates:	dump construction began 1990, drainage acidified by 1992, waste rock relocated 1994-1995
Parameters Monitored:	pH, acidity, Zn, Pb,
Results:	in the watershed: 1 year pH = 4.7, Zn = 5 mg/L, Pb = 0.2 mg/L; 3 years: pH = 7.0, Zn = 0.2 mg/L, Pb = 0.009 mg/L (all qualified) in the lake: remained near background levels
Costs:	CND$ 500,000 for relocation and submergence (about CDN $5/t)
References:	Morin, K.A.; Hutt, N.M., ICARD 2000 pg 819-825.

Benson Lake, British Columbia

Disposal Setting:	vertical discharge from raft at 30 m depth
Waste Material:	
Dates:	in operation 1962-1973
Parameters Monitored:	
Results:	
References:	Dave (1992)

Location:	Equity Silver, British Columbia
Disposal Setting:	man-made tailings pond
Waste Material:	tailings (Aspy, py, po), 2-10% S
Dates:	operation from 1980 to Jan. 1994
Parameters Monitored:	N, NO_3, PO_4, Cl,CN, Sb, As, Cd, Fe, Pb, Mn, Zn, C_{org}, DO
Results:	Fe, Mn, Cu, Zn, Sb, As & CN in water cover are low, Cu decreasing, pH = 7-8, DO =5 mg/L
References:	Yanful & Simms (1996); Pedersen et al. (1997) ICARD, V3, 989

Gunnar Pit, Lake Athabasca, Saskatchewan

Disposal Setting: flooded open pit
Waste Material: waste rock?
Dates:
Parameters Monitored:
Results:
References: MEND Report 2.36.1 (1995)

Cluff Lake "D" Pit, Saskatchewan

Disposal Setting: flooded open pit
Waste Material: waste rock
Dates: monitored 1982-1994
Parameters Monitored: pH, metals, anions,
Results: 1989-1994: pH=7.36, SO_4=8.1 mg/L, As=18.4 µg/L, Fe=0.892 mg/L, Ni=0.028 mg/L, U=66.75µg/L
References: MEND Report 2.36.1 (1995); Quarshie & Rowson, Sud95, V1, pg 217

Rabbit Lake Mine, Wollaston Lake, Saskatchewan

Disposal Setting: engineered pit disposal w/ bottom rock drain
Waste Material: uranium tailings,
Dates: operation 1975-1984, deposition 1985-?
Parameters Monitored: pH, Al, As, Ca, Fe, K, Mg, Mn, Ni, Pb, SO_4, Ra, U
Results: 1994: pH = 8.4, SO_4 = 1886 mg/L
Costs: estimated to be $2.62/t tailings in 1995$
References: MEND Report 2.36.1 (1995)

Collins Bay B-Zone Pit, Wollaston Lake, Saskatchewan

Disposal Setting: open pit
Waste Material: waste w/ elevated As, Ni, S, or U
Dates: decommissioned 1991,
Parameters Monitored: T, DO, cond, pH, redox, trace metals, radionuclides, nutrients, major constituents
Results: 1994: As=0.359 mg/L, Ni=0.303 mg/L, SO_4=10.5 mg/L, Pb^{210}=0.11 Bq/L, Ra^{226}=0.06 Bq/L, U=0.014 mg/L
References: MEND Report 2.36.1 (1995)

Anderson Lake, Manitoba

Disposal Setting: discharge into lake at 3-6 m depth
Waste Material: tailings (py, po, sphal, chalcopy, gal, and arsenopy), 20-25% S
Dates: disposal began 1979
Parameters Monitored: T, O_2, Cu, Pb, Zn, Ni, Co, Fe/Al, Mn, S_T, C_{org}, TKN, pH

Results: tailings dispersed up to 2 km from discharge, natural deposition of organic-containing sediment consumed O_2, little post-deposition oxidation except in littoral zones, dissolved metal concentrations decreased abruptly in sediments (removal by flora/fauna?), pH decr from 8 to 6, metals and turbidity incr 5-8x, sulfate incr 19-20x,

Problems Identified: turbidity = suspension of fine particles and effects aquatic biota, burial (siltation) of bottom life has ecological effects, toxicity of soluble compounds on aquatic biota

References: Dave (1992); Pedersen et al. (1997) ICARD, V3, 989; Pedersen et al. (1999) Mining & Environment, v1, 165.

Mandy Lake, Manitoba

Disposal Setting: disposal in shallow water along shore

Waste Material: tailings (py) 15-17% S

Dates: in operation 1917-1920 & 1943-1944

Parameters Monitored: pH, cond, SO_4, Fe, Mn, Zn, Cu, Pb, Co, Ni, TKN, C_{org}, O_2, As,

Results: tailings dispersed up to 2 km from discharge, natural deposition of organic-containing sediment consumed O_2, little post-deposition oxidation except in littoral zones, dissolved metal concentrations decreased abruptly in sediments (removal by flora/fauna?)

Problems Identified: turbidity = suspension of fine particles and effects aquatic biota, burial (siltation) of bottom life has ecological effects, toxicity of soluble compounds on aquatic biota

References: Dave (1992)

Quirke Lake Tailings Test Site, Elliot Lake, Ontario

Disposal Setting: 192 ha tailings pond = series of terraced cells, water from a local lake, limestone addition

Waste Material: 45 million tonnes of uranium tailings & waste rock (py = 5.3%)

Dates: operated from 1956-1961 & 1968-1990; dike construction carried out from 1990 to 1995, flooded began in 1992

Parameters Monitored: pH, SO_4, Fe, Zn

Results: pH variable from 4 to 9, sulfate decreasing slightly, Fe = 0.2 mg/L, Zn = 0.01 mg/L

References: Yanful & Simms (1996); Kam et al. (1997) ICARD, V2, 853; Gallinger (1999) MEND 2000 Workshop ME.02

Panel Wetlands, Elliot Lake, Ontario

Disposal Setting: 123 ha wetland basin filled with tailings and minimum water level of 0.6m, lime addition

Waste Material: 14 million tonnes tailings

Dates: operated 1958-1961 & 1979-1990; tailings deposited in the late 1950s

Parameters Monitored: pH, SO_4, Fe, bacteria,

Results: stable low levels of acidity, SO_4, & Fe for 10 years

References: Yanful & Simms (1996)

Owl Creek, Timmins, Ontario

Disposal Setting: flooded open pit
Waste Material: waste rock
Dates: operation 1981-1989, waste moved to pit in 1991
Parameters Monitored: pH,T, DO, ICP metal scan, SO_4, NO_3
Results: June 1994: pH neutral & water quality "continuing to recover"
Costs: relocation and regrading of remaining waste approx. $7,500,000
References: MEND Report 2.36.1 (1995)

Falconbridge Limited, New Tailings Area, Falconbridge, Ontario

Disposal Setting: tailings pond
Waste Material: tailings
Dates: These tailings were relocated underwater by truck and excavator in the fall of 1996, and by dredging in September and October of 1997. Tailings picked up by dredging in these areas were transported through a floating pipeline and re-deposited in areas immediately upstream of Dam 12. Prior to final phase of dredging in 1997, approximately 850 tonnes of lime were spread on the surface of the tailings.
Parameters Monitored: pH, SO_4, Ca, Cu, Fe, Ni, Zn
Results: Peaks of metal concentrations have appeared each spring. The magnitude of these peaks, however, has declined from 6 mg/L Ni in 1996, to slightly more than 1 mg/L in 1999. The resulting increase in the Lower Terrace is correspondingly lower as well, and, since February 1999, nickel has been below provincial effluent quality limits.

In conjunction with NTC, Lorax Environmental studied the sediment and pore water chemistry in 1998. The main conclusions from this work were: surficial sediments are organic-poor, and the organic matter is of aquatic source (plankton); maximum dissolved oxygen penetration into the sediments was only 3 mm; submerged tailings are anoxic at shallow depths, a indicated by the oxygen data, and supported by profiles of dissolved iron and manganese; and minor amounts of arsenic and molybdenum are diffusion from the tailings, although water column concentrations are quite low (1-2 mg/L)
Cost: Total cost of construction is estimated at $1.39 million. It is estimated that the cost of preliminary engineering, research and design totaled $0.18 million. The estimated total cost of the project is $2.61 million, or $29,000 per hectare.
References: MEND: http://mend2000.nrcan.gc.ca/cases/falcon/falcon.htm; DeVos et al. (2000) ICARD, V2, 933; Hall (1999) MEND 2000 Workshop ME.02

Strathcona sites, Sudbury, Ontario

Disposal Setting: subaqueous deposition in a 90 ha arm of a larger lake basin

Waste Material:	sand and silt sized tailings
Dates:	deposition since 1968, > 20 m thick of sand and silt sized tailings
Parameters Monitored:	grain size, permeability, SO_4, water table depth
Results:	Sulfate levels indicate that the bulk of the tailings pore water consists of mill process water unaffected by sulfide oxidation.
Cost:	
References:	Robertson et al. (1997) ICARD, V2, 621

Solbec Open Pit, Eastern Township, Quebec

Disposal Setting:	open pit
Waste Material:	tailings in 1977, waste rock 1988
Dates:	operation 1962-1970, mill operation til 1977, decommission 1977-1979,
Parameters Monitored:	pH, Cu, Pb, Zn, Fe, Mn, Cd, T, cond, alk, DO, TDS
Results:	1992: pH=6.7, TDS=4.6 mg/L, Cu=0.07mg/L, Zn=2.6 mg/L, Fe=1.07 mg/L
Cost:	1988-1992 = $1.069.100 or $3.87/m^3$
References:	MEND Report 2.36.1 (1995)

Solbec Open Pit, Eastern Township, Quebec

Disposal Setting:	flooded tailings pond
Waste Material:	tailings in 1977, waste rock 1988
Dates:	In 1992, a fourth phase was undertaken. To offset the leaching of the oxidation products from the oxidized tailings, limestone was added in the testing basins as an attempt to neutralize and stabilize the previously acidified water. In December 1993, flooding of the pond after addition of lime was recommended as the reclamation method.
Parameters Monitored:	The 18 water cover samples are subjected to the following analysis without being filtered. In the field: pH, electrical conductivity and Fe+2. In the laboratory: Metals (Fe, As, Cu, Ni, Pb, Zn), sulfates, suspended matter, alkalinity and acidity if appropriate.
Results:	(approximates) pH = 6-8.5, Results have demonstrated that the covering waters met all requirements outlined in Regulation 019 (Ministry of Environment and Wildlife) and even those of the drinkable water regulation.
Cost:	
References:	Amyot and Vezina (1997) ICARD, 2, 681; Amyot & Vezina (1999) MEND: http://mend2000.nrcan.gc.ca/cases/flooding-99/flooding-99.htm

East Kemptville Tin Mine, Nova Scotia

Disposal Setting:	250 ha tailings impoundment, submergence of potentially acid generating tailings
Waste Material:	18.8 million tonnes of pyritic tailings
Dates:	operated from 1984 - 1992
Parameters Monitored:	Zn, Al, F, Mn, Cu

Results:
Cost:
References: Gallinger (1999) MEND 2000 Workshop ME.02

SWEDEN

Stekenjokk, Sweden

Disposal Setting: clarification pond

Waste Material: tailings, 20% S

Dates: operation 1976-1988, decommissioned 1991 w/ permanent flooding, geochemical study begun in 1995

Parameters Monitored: Cu, Pb, Fe, Zn, SO_4, DO

Results (mg/L): Cu= 0.002, Cd=0.07, Zn = 0.1 mg/L, SO_4 = 14 mg/L, suspended solids < 5 mg/L

References: Yanful & Simms (1996); Ljungberg et al. (1997) ICARD, V3, 1401

Udden Open Pit, Sweden

Disposal Setting: flooded open pit

Waste Material: weathered sulfidic waste rock

Dates: relocated June 1994

Parameters Monitored: pH, TDS, cond, SO_4, Cu, Pb, Zn, Cd, flow

Results: August 1994: pH=6.0, SO_4=1200 mg/L, Cu=0.1-0.17 mg/L, Zn=81 mg/L, Cd=0.14 mg/L

References: MEND Report 2.36.1 (1995)

Kristineberg Mine, Sweden

Disposal Setting: Tailings area consists of five individual ponds. In order to finally incorporate the tailings area into the surrounding environment the Vormbacken Creek, which had previously been diverted around the tailings ponds will be rediverted into the lower ponds turning them into artificial lakes naturally interacting with the creek.

Waste Material: tailings

Dates: mill closed in 1991, project commissioned in 1995, began in 1996, scheduled to be completed in 2000

Parameters Monitored:

Results:

References: Lindvall et al. (1999) Mining & Environment, V3, pg 855

NORWAY

Hjerkinn tailings pond, Norway

Disposal Setting: tailings pond
Waste Material: tailings, 5-18% S
Dates: deposition 1968-1993
Parameters Monitored: pH, SO_4, Zn, Fe, suspended solids, turbidity
Results: pH =7.2, SO_4 = 600 mg/L, Zn incr to 200 µg/L and decr to 147 µg/L after deposition ceased, Fe incr to 600 µg/L and decr to 70 µg/L after deposition ceased, suspended solids incr from 1.5 to 2.5 mg/L, turbidity incr from 1.8 to 5.1 BTU
References: Yanful & Simms (1996)

Lokken Site, Sør-Trøndelag County

Disposal Setting: underwater disposal in a man-made pond
Waste Material: tailings, 40% S
Dates: deposition from 1974-1987
Parameters Monitored: pH, SO_4, Fe, Cu, Zn
Results: during deposition pH decr from 10.1 to 5.3, SO_4 = 800 mg/L, Fe = 200-500 µg/L, Cu incr from 200 to 800 µg/L, Zn incr from 500 to 1200 µg/L; post-deposition pH = 4-6, SO_4 = 100 mg/L, Cu = 0-600 mg/L, Zn = 2500 µg/L
References: Yanful & Simms (1996); Arnesen & Iversen (1997) ICARD, V3, 1093.

Lokken Site, Wallenberg Mine, Sør-Trøndelag County

Disposal Setting: flooded mine workings
Waste Material: mine workings, pyrite
Dates: began flooding by natural inflow in 1983
Parameters Monitored: pH, SO_4, Fe, Cu, Zn
Results: pH increased from 2.0 to 4, Cu decreased from 530 to 1 mg/L, Zn increased from 1500 to 4000 mg/L
References: Arnesen & Iversen (1997) ICARD, V3, 1093.

APPENDIX 4

Chemical Stabilization of Reactive Mine Waste Materials

A4.1. Overview of Alkaline Amendment Studies

Numerous studies have been conducted on the application of alkaline materials to neutralize acid released from reactive mine wastes. The purpose of this section is not to provide an extensive literature review of alkaline addition case studies. The following provides a brief overview of recent alkaline addition methodology and applications, including results from a few laboratory and field studies.

A4.1.1. Waste Rock

Laboratory studies

Stewart et al. (1994) reported on column studies using mixtures of fly ash, limestone, or rock phosphate with acid-producing coal refuse. The loadings of mitigative solid were varied and the tests were conducted for periods of two to three years. Coal refuse (2.3% S) was mixed with Westvaco ash mix (20% ash with an NP of 5 kg/t $CaCO_3$) in ratios of 4:1 and 2:1. Both ratios produced neutral drainage (pH 8) for three years, as compared to a control pH of 1.9. In a second phase, column experiments were conducted on various mixtures of coal refuse (4% S) with the Westvaco ash (5, 10, 20, and 33% loadings) and Clinch River fly ash (20 and 33% loadings). With the Westvaco ash, only the highest loading remained neutral after two years. The Clinch River ash (which was more alkaline than the Westvaco ash) maintained neutral drainage for two years, as did a 13% limestone mixture.

Limestone requirements to prevent acid generation and metal release from gold deposit mine wastes were evaluated in a five-year laboratory column study (Day, 1994). Each column tested a different blend of waste rock and limestone with NPRs ranging from 0 to approximately 1:1. The results indicated that a ratio of at least 1:1 is required to delay the onset of drainage acidification. The 1:1.11 limestone blend successfully prevented drainage acidification throughout the study, however, did not prevent zinc release. However, increasing sulfate concentrations, slightly decreasing pH, and acidic residues in the upper portions of the column suggested that the drainage will acidify in the future. Day (1994) concluded that the actual limestone requirement to delay acidification indefinitely would be at least 2:1.

Field studies

Waste materials have been blended with acid-producing mine wastes. Evans and Rose (1995) added lime kiln flue dust (an alkaline waste product) to 360 tons pyritic shale (1.9% S) in amounts providing neutralization potential 0 to 1.71 times the acid production potential. All conditions produced acidic drainage throughout the 10-month field test, although drainage pH increased from 1.9 to 2.6 as the flue dust loading increased. Incomplete mixing of the flue dust and shale, as well as preferential flow, were cited as limitations to the effectiveness of the treatment.

With metal mine waste rock, the blending and layering of limestone is reported to be of minimal mitigative success, due to problems such as inadequate homogeneity of mixtures and preferential flow through acid-generating layers (Mehling et al. 1997). The large particle size of the waste rock, the dimensions of which can reach several feet, most likely contributes to these problems. The reactivity of limestone is also limited when particle size is large.

Whereas ratios of NP:AP have been used to determine the amount of alkaline material required for maintaining neutral drainage from waste rock piles, analysis by Kempton et al. (1997) and Morin and Hutt (2000) indicate that preferential flow has a dominant influence on the effectiveness of waste rock blending. The latter publication indicated that waste rock drainage acidity is dependent on the flow path length within acid neutralizing rock separating zones of acid generating rock. The authors' analysis indicated that waste rock with a bulk NP:AP ratio of 300:1 could release acidic drainage if appropriate neutralizing rock flow path length was not attained.

A4.1.2. Tailings and Fine-Grained Waste Rock

With finer mine wastes (e.g. tailings or fine mine waste) and alkaline solids a homogeneous mixture can be more readily attained and, due to more uniform particle sizing, flow tends to be more uniform than in the wide range of particle sizes in waste rock piles. Blending of alkaline solids with tailings is further attractive due to 1) greater practicality in determining tailings composition within standard operating procedures; 2) their potential for a higher degree of compositional homogeneity over extended periods; and 3) the potential to achieve more intimate blends by moderate modification of standard tailings handling procedures. Furthermore, it may be necessary to treat only those tailings above the water table, since the rate of sulfide Mineral oxidation below this level will be limited by oxygen diffusion through water.

For tailings, the operational phase is less problematic since drainage acidification tends to be limited by 1) the subaqueous environment in which the majority of tailings are disposed and 2) the typical addition of alkaline processing reagents to tailings impoundments. Furthermore, any problematic drainage that is generated can be collected and returned to the impoundment. However, after operations cease and before final closure measures are implemented, mitigation measures may be required, particularly in areas in which sulfidic tailings are not submerged. The addition of alkaline solids to tailings as the tailings are generated may be an effective, low-cost method of neutralizing acid produced during this period. It is also possible that such addition would accelerate the formation of precipitate coatings on the iron sulfide minerals responsible for acid production and, thereby, render these minerals much less reactive.

Laboratory studies

Some success has been attained in mixing limestone with finer acid-producing materials in the laboratory. Lapakko et al. (1997) reported that -10-mesh limestone homogeneously mixed with finely-crushed acid producing rock neutralized acid produced by iron sulfide oxidation. Furthermore, sulfide oxidation rates decreased to the extent that host rock Mineral dissolution neutralized acid produced by iron sulfide oxidation after the limestone was depleted.

Lapakko et al. (2000) conducted laboratory tests under conditions similar to those used with the limestone blends, examining the effects of mixing rotary kiln fines (RK fines) with finely-crushed acid producing rock. Five loadings of RK fines were mixed with finely-crushed acid producing rock to produce neutralization potential (NP) added: acid potential (AP) quotients of 0, 0.11, 0.22, 0.66, and 1.1. The mixtures were subjected to wet-dry cycling for 585 weeks. The RK fines elevated pH and inhibited sulfate release for time periods which increased with the mass of RK fines present. The pH of drainage from the 1.1 quotient typically remained above 8.0 throughout the period of record.

Field studies

A lime amendment was incorporated at a historical abandoned tailings deposit (Davis et al. 1999). Site examination four to five years after the amendment revealed that the pH in pore waters of amended tailings was higher than values associated with unamended tailings. The authors also noted that trace metals were removed from pore waters in the amended tailings by sorption to ferrihydrite and/or coprecipitation. The authors concluded that lime amendment "represents a viable long-term alternative to mitigate a historical problem." Whereas this conclusion is generally consistent with the results presented, it may be premature based on the fairly short-term success. In addition, the paste pH of one tailings sample at an amended site was 4.7, despite a net carbonate value of 13 percent.

Table A4.2. Bibliographic annotations of research related to the use alkaline amendments to control generation of acidic drainages. Italicized numbers refer to reference numbers for the Reclamation Section’s literature database (Papyrus).

Reference	Study Scale[1]	Waste Material		Alkaline Material				
		Hard Rock	Coal	Lime-stone/ Lime	Waste Rock	Coal Combustion Waste	Municipal Waste	Paper Mill Waste
MEND (1998)	R	U	U					
Scheetz et al. (1998)	R		U			U		
Ziemkiewicz & Black (2000)	R		U			U		
Morin & Hutt (2000)	M							
Lapakko & Antonson (1991) #*1355*	L	U		U				
MEND (1994) #*3273*	L	U		U				
Day (1994) #*3578*	L, F	U		U				
Lapakko et al. (1997)	L	U		U				
Taboada et al. (1997)	L	U		U				
Iversen & Arnesen (1988)	O	U		U				
Davis et al. (1999) #*3440*	O	U		U				
Denholm & Hallam	F, O	U			U			
Morin & Hutt (1997)	L	U			U			
Mehling et al. (1997)	L, F, O	U			U			
Condon (1999)	O	U			U			
Lapakko. et al. (2000)	L	U				U		
Chtaini et al. (1997)	L, F	U						U
Coleman et al. (1997)	F		U				U	

[1] R= review,M=modeling, L= laboratory, F= field, O= operation

Table A4.3. Bibliographic annotations of research related to chemical stabilization of reactive mine wastes using microencapsulation to control generation of problematic drainages. Italicized numbers refer to reference numbers for the Reclamation Section's literature database (Papyrus).

Reference	Study Scale[1]	Microencapsulation Technique			Objective	
		Phosphate	Silicate	Other	Coating Feasibility	Drainage Quality
Evangelou (1994)	L	U			U	
Evangelou (1995) *#3709*	L	U			U	
Georgopoulou, et al. (1995)	L	U			U	
Evangelou (1996) *#3738*	L	U			U	
Roy & Worral (1999)	L	U			U	
Conca et al. (1999)	F	U				U
Jensen et al. (1999)	O	U				U
US EPA (1999) *#3688*	F	U				U
Littlepage et al. (2000) *#3745*	F	U				U
Fytas & Evangelou (1998) *#3636*	L	U			U	U
Zhang & Evangelou (1998) *#3637*	L		U		U	
Trezek (1992)*#1741*	F		U			U
Chatham & Svee (1996) *#3675*	F		U			U
Mitchell & Anderson (2000) *#3747*	R, L, F		U			U
Fytas et al. (1999)	L		U		U	U
Fytas et al. (2000)	L		U		U	U
KEECO *#3647*	R		U		U	U
Moskalyk (1995)	L			U	U	U
Chen et al. (1999)	L			U	U	
Maki et al. (1995)	L			U		U

Reference	Study Scale[1]	Microencapsulation Technique			Objective	
		Phosphate	Silicate	Other	Coating Feasibility	Drainage Quality
Vandiviere & Evangelou (1998) *#3113*	F	U	U			U
Adams et al. (1994)	L	U		U	U	
Williams et al. (1999)	F	U	U	U		U

[1] R= review,M=modeling, L= laboratory, F= field, O= operation

APPENDIX 5

Biological Stabilization of Reactive Mine Wastes
Bactericides

A5.1. Overview of Bactericide Studies

Since Mineral potential in Minnesota is largely associated with metallic minerals, this presentation focuses on the application of bacterial inhibitors to sulfidic mine wastes rather than coal mine wastes. Very few studies have been conducted on metal mine wastes using bacterial inhibitors. The majority of these studies evaluated the ability of surfactants such as sodium lauryl sulfate (SLS) to inhibit bacterial growth, thereby reducing drainage acidity and dissolved metal concentrations (Patterson, 1987; Sobek, 1987; Watzlaf, 1988; Parisi et al., 1994). Two studies evaluated the effectiveness of heterocyclic mercaptans (Stichbury et al., 1995; Lortie et al., 1999). The purpose of this section is to review the results of these studies in terms of parameters used to indicate bacteria inhibition (i.e. pH, acidity, chemical release, and bacteria counts).

A5.1.1. Surfactants

The type of surfactants used and application methods varied widely in the studies reviewed here. In three instances, the surfactant used was not identified (Patterson, 1987; Sobek, 1987). SLS was the most commonly tested surfactant, however, Watzlaf (1988) also tested potassium benzoate and potassium sorbate.

Surfactant concentrations were not always reported, although laboratory test claimed to be used to determine appropriate dosages. Based on the two studies that did report surfactant dosages, the field test (Parisi et al., 1994) used a dosage that exceeded the dosage used in unrelated laboratory tests (Watzlaf, 1988) by one to three orders of magnitude.

Test duration varied from a set of eight week laboratory experiments (Sobek, 1987) to a fifteen month field trial (Parisi et al., 1994). As a result of these differences, it was difficult to make valid comparisons between studies.

In general, surfactant effectiveness was reported in terms of the percent reduction in acidity. Application of surfactants during laboratory studies reduced acidity by 50% - 70% when adequate surfactant dosages were used (Sobek, 1987; Watzlaf, 1988). Of the two field studies reviewed, only one was considered successful with acidity reduced from 2500 mg/L to 200 mg/L, or approximately 92% (Parisi et al., 1994). Direct measurements of pH were not reported in any of the laboratory studies and rarely reported for field test sites. Parisi et al. (1994) reported that drainage from waste rock at a silver mine had an average pH of 5.6 during the fifteen month study. These reported results suggest that while drainage acidity was greatly reduced, drainage pH was probably not maintained above the environmental standard of 6.0.

In one study, surfactant effectiveness was determined by chemical release in the drainage. Patterson (1987) reported on a laboratory study in which a 50% reduction (500 to 250 mg/L) in copper release to solution was used as an indicator of SLS effectiveness (Patterson, 1987). Similarly, a field test at a silver mine reported decreases in sulfate concentrations from 1000 mg/L to 300 mg/L (Patterson, 1987).

A third method used to determine surfactant effectiveness is to estimate bacteria population size. Watzlaf (1988) used a most probable numbers method to determine that SLS and potassium benzoate inhibited bacterial growth in batch leach tests for 182 and 231 days, respectively.

A5.1.2. Heterocyclic Mercaptans

Two papers dealt with the application of heterocyclic mercaptans to sulfidic wastes (Stichbury et al., 1995; Lortie et al., 1999). Both papers report on the same basic study, using 2,5-dimercapto-1,3,4-thiadiazole (DMT) and 5-amino-1,3,4-thiadiazole-2-thiol (ATT) as bacterial inhibitors. Stichbury et al. (1995) focused on laboratory experiments conducted over a six week period. In the laboratory experiments, 20 g of fresh tailings and 0.5 g oxidized tailings were treated with 85 ml of inhibitor (at 100, 200, or 500 mg/L) and 5 ml of culture medium containing *T. thioparus*.

Field test plots consisted of plastic barrels (d = 53 cm, h = 82 cm) filled with fresh tailings and backfilled into an old tailings area (Lortie et al., 1999). Test plots were treated with 500 mg inhibitor per kilogram of tailings and monitored for approximately one year.

In both cases, pH values were reported as indicators of inhibitor effectiveness rather than acidity. In laboratory experiments, only one dosage of DMT (500 mg/L) resulted in a final pH above 6.0 (Stichbury et al., 1995). In all other cases, pH decreased below 5.0. After one year, cores were taken from the field test plots and cut into two sections, representing 0-25 cm and 25-50 cm depths (Lortie et al., 1999). Pore water pH in these sections decreased to 3.0 and 3.2, respectively, in the treated test plots compared to 3.5 and 4.2, respectively, in the control test plots.

The results of the laboratory experiments in terms of sulfate concentration were similar to the pH results in that sulfate production was inhibited only in the 500 mg/L DMT incubation. However, sulfate production in the field test plots was reduced from approximately 30,000 mg/L in the controls to 14,000 - 18,000 mg/L in the treated barrels. Thus, sufficient quantities of DMT appear to reduce sulfate production in these tailings by approximately 50% compared to untreated tailings.

Tailings pore waters from the field test plots were also analyzed for Fe, Al, Ca, Co, Cu, Mg, Mn, Na, Ni, and Zn (Lortie et al., 1999). Concentrations of five of these elements (Al, Co, Mg, Mn, and Na) were actually higher in the treated samples than in the controls. No significant differences were seen in concentrations of Ca, Cu, Ni, or Zn. Only iron concentrations were reduced below the levels found in the control samples (i.e. approximately 14,500 mg/L to 4000 mg/L).

The number of *T. thioparus* bacteria associated with each core from the field test plots were also estimated using a most probable numbers technique (Lortie et al., 1999). After one year, bacteria counts in the control samples had reached 10^5 to 10^6, whereas bacteria counts in the treated samples did not exceed 380.

Table A5.2. Bibliographic annotations of research related to the use bactericides to control generation of acidic drainages. Italicized numbers refer to reference numbers for the Reclamation Section's literature database (Papyrus).

Reference	Study Scale[1]	Targeted Bacteria		Waste Material	
		Acidophiles	Neutrophiles	Coal	Hard Rock
Rastogi & Sobek (1986) #*772*	R	U		U	
Sobek (1987) #*1526*	R	U		U	
Benedetti et al. (1990) #*1256*	R	U		U	
Rastogi et al. (1990) #*1255*	R	U		U	
Rastogi (1995) #*3515*	R	U		U	
Ziemkiewicz (1995)	R	U		U	
Delaney et al. (1997)	R	U		U	U
Watzlaf (1988) #*1003*	L	U		U	
Kleinmann (1980) #*3514*	L, F	U		U	
Fox & Rastogi (1983) #*3496*	L, F	U		U	
Kleinmann & Erickson (1988) #*1254*	L, F	U		U	
Patterson (1987) #*1529*	L, F	U			U
Stichbury et al. (1995) #*3733*	L		U		U
Lortie et al. (1999) #*3732*	F		U		U
Sobek et al. (1990) #*1257*	O	U		U	
Parisi et al. (1994)	O	U		U	U

[1] R= review,L= laboratory, F= field, O= operation

APPENDIX 6

Treatment Systems for Problematic Drainage

Table A6.1.1. Bibliography of problematic drainage passive treatment studies conducted by the MN DNR.

Strategy	Study Scale	Reference
Wetlands	Natural Wetlands	Eger, P., Lapakko, K., Otterson, P. 1980. Trace metal uptake by peat: Interaction of a white cedar bog and Mining stockpile leachate. In Proc. of the 6th International Peat Congress. Duluth, MN. Aug. 17-23, 1980. p. 542-547.
		Eger, P., Lapakko, K. 1988. Nickel and copper removal from mine drainage by a natural wetland. In Proc. 1988 Mine Drainage and Surface Mine Reclamation Conference, April 19-21, 1988, Pittsburgh, PA., V1, 301-309.
		Eger, P., Lapakko, K. 1989. The use of wetlands to remove nickel and copper from mine drainage. In Constructed Wetlands for Wastewater Treatment (June 13-17, 1988, Chattanooga, Tennessee). Hammer, D. A. (Ed), Lewis Publishers, Chelsea, MI. p. 780-787.
	Laboratory	Lapakko, K., Eger, P. 1981. Trace metal removal from Mining stockpile runoff using peat, wood chips, tailings, till, and zeolite. In Proc. 1981 Symposium on Surface Mining Hydrology, Sedimentology and Reclamation. Lexington, KY. p. 105-116.
		Lapakko, K., Eger, P. 1983. Passive treatment of sulfide stockpile runoff. In Proc. 1983 National Conference on Environmental Engineering, A. Medine and M. Anderson eds. ASCE, New York. p. 643-651.
		Lapakko, K., Strudell, J., Eger, A. 1986. Trace metal sequestration by peat, other organics, tailings, and soils: A literature review. (Final Report BuMines Contract J0205047). U.S.D.I. Bureau of Mines NTIS # PB 87-186144. 45 p.
		Lapakko, K., Eger, P. 1988. Trace metal removal from stockpile drainage by peat. In Proc. 1988 Mine Drainage and Surface Mine Reclamation Conference, April 19-21, 1988, Pittsburgh, PA. V. 1. Mine Water and Mine Waste. U.S.D.I. Bureau of Mines IC9183. p. 291-300.
	Small-scale Field	Eger, P. 1994. Wetland treatment for trace metal removal from mine drainage: the importance of aerobic and anaerobic processes. In Wat. Sci. Tech., Vol. 29, No. 4. p. 249-256.
		Eger, P., Wagner, J., Kassa, Z., Melchert, G. 1994. Metal removal in wetland treatment systems. Proc. International Land Reclamation and Mine Drainage Conference/Third International Conference on the Abatement of Acidic Drainage. Pittsburgh, PA, April 25-29, 1994.
	Operation	Eger, P., Wagner, J., Melchert, G., 1996. The use of overland flow wetland treatment systems to remove nickel from neutral mine drainage. In Successes and Failures: Applying Research Results to Insure Reclamation Success. Proc. 13th National Meeting, Knoxville, TN, May 18-23, 1996. p.580-89

Strategy	Study Scale	Reference
		Eger, P., Wagner, J.,Melchert, G., 1997. The use of a peat/limestone system to treat acid rock drainage. Proc. Fourth International Conference on Acid Rock Drainage. Vancouver, B.C., Canada, May 31-June 6, 1997. p. 1195-1209
		Eger, P., Melchert, G., Wagner, J. 1999. Using passive systems for mine closure–a good approach or a risky alternative. In Mining in a new era. CD-Rom. Society of Mining Engineers Annual Meeting, Denver, CO, March 1-3, 1999. Preprint 99-38.
		Eger, P., Wagner, J., Melchert, G., Antonson, D., Johnson, A. 2000. Long term wetland treatment of mine drainage at LTV Steel Mining Company's Dunka Mine. MN DNR, Saint Paul, MN. 54p. plus appendices.
Sulfate Reduction	Barrel tests	Eger, P. 1992. The use of sulfate reduction to remove metals from acid mine drainage. In Proc. 9th National Meeting ASSMR, Duluth, MN, June 14-18, 1992. p. 563-575.
		Eger, P. 1994. The use of sulfate reduction to remove metals from acid mine drainage. Proc. International Land Reclamation and Mine Drainage Conference / Third International Conference on the Abatement of Acidic Drainage. Pittsburgh, PA, April 25-29, 1994. p. 412.
		Eger, P., Wagner, J. 1995. Sulfate reduction for the treatment of acid mine drainage: Long term solution or short term fix? In Proceedings of Sudbury '95 - Mining and the Environment. May 28-June 1, 1995, Sudbury, Ontario, Canada. p. 515-524
		Eger, P., Wagner, J. 2001. Sulfate reduction - Designing systems for long term treatment. SME Annual Meeting, Denver, CO, Feb 26-28, Preprint 01-115.

Table A6.1.2. Summary of operational-scale passive treatment systems.

UNITED STATES

Dunka Mine Site, Babbitt, Minnesota:

References: Eger et al. (1998) ASSMR; Eger et al. (1997) ICARD, V3, pg 1195-1209. Eger et al. (2000)

Mine Type: Taconite, sulfidic overburden stockpiles

Results (mg/L):

Wetland Treatment System	Parameter of Concern	Influent (mg/L)	Effluent (mg/L)
W2D-3D[1]	pH	7.1	7.0
	Cu	0.06	<0.005
	Ni	1.94	0.06
	Co	0.02	<0.005
	Zn	0.06	<0.005
W1D [2,4]	pH	7.2	7.4
	Cu	0.03	<0.005
	Ni	0.90	0.13
	Co	0.01	0.02
	Zn	0.02	<0.005
Seep 1[2,4]	pH	6.9	7.2
	Cu	0.27	0.06
	Ni	7.24	3.34
	Co	0.19	0.06
	Zn	0.94	0.41
Seep X[2]	pH	7.1	7.2
	Cu	0.41	0.14
	Ni	1.82	0.99
	Co	0.09	0.03
	Zn	0.57	0.35
EM-8[3]	pH	7.2	7.1
	Cu	0.03	0.01
	Ni	1.62	1.04
	Co	0.01	<0.005
	Zn	0.05	0.04

[1]average influent concentrations monitored from 1992-1994, average effluent concentrations for 1992-1998

[2] average concentrations from July 1995 (wetland brought online) through 1999

[3] average concentrations from 1998 (wetland brought online) through 1999

[4] "Influent" monitoring stations represent the effluent from limestone pretreatment cells intended to raise drainage pH prior to entering the W1D and Seep 1 wetlands.

Capacity: flow is generally at 3-840 L/min, but 6000 L/min possible, design capacity was for 75 L/min

Cost estimates: construction=$1.2 million, operation=$40,000

Other notes: Wetland treatment systems at Dunka are part of an integrated system involving stockpile capping, passive, and active treatment.

Dunka Mine Site, Babbitt, Minnesota, continued:

References: Lapakko et al., 1998; Lapakko and Antonson, 1990

Mine Type: Taconite, sulfidic overburden stockpiles

Treatment Process: pilot-scale limestone bed

Results (mg/L): The bed was 100% efficient in neutralizing acidity in 1990, raising the mean pH from 5.0 to 6.7 and the mean net alkalinity from -85 to +22 mg$CaCO_3$/L. Cu reduced by almost 68%, Zn reduced 42%, Ni & Co reduced 13%. After switching to larger sized limestone (to prevent clogging), treatment became ineffective.

Capacity: The bed received 3600 m^3 of flow at rates of 0.276 to 1.356 L/s, averaging 0.73 L/s or about 45 bv/d.

Cost estimates:

Other notes:

Ferris-Haggarty Mine, Carbon County, Wyoming:

References: Reisinger et al. (2000) ICARD pg 1071

Mine Type: inactive underground copper mine

Inflow quality (mg/L): Osceola Tunnel: pH 6.5-7.0, 3-5 mg/L Cu but up to 15-20 mg/L during spring runoff
Chute/shaft water: pH 3.5-4.0, Cu=12-23 mg/L

Effluent quality: (approx values est. from a graph) pH >6.5, Cu <3

Treatment process: Pilot-scale, anaerobic cell (d=15', 4' deep) filled with cattle manure (containing SULFATE REDUCING BACTERIA*), sawdust, limestone, hay, and alfalfa.

Capacity: gravity feed flow less than 5 gpm

Cost estimates: none given

Other notes:

Lilly/Orphan Boy Mine, Elliston, Montana:

References: Canty, M. (2000) ICARD pg 1139; #3518-Canty (1999) Mining Engineering

Mine Type: metal mine

Inflow quality: 2-8 gpm, pH=3, As=1.07mg/L, Al=9.69mg/L, Cd=0.33mg/L, Cu=0.32mg/L, Fe=27.7mg/L, Mn=6.21mg/L, Zn=26.1mg/L, SO_4=277mg/L

Effluent quality: tunnel: pH=7, 70-100% removal of Al, Cd, Cu, Zn; lower for Fe and As; Mn removal efficiency = 30-80%
portal: pH=6,

Treatment process: "in situ biological reactor" in an underground mine. Platforms containing animal by-products were suspended 30' below the static water level and in the main tunnel. Sulfate reduction increases pH (approx. 7) and decreases Eh.

Capacity: up to 8 gpm

Cost estimates: none

Other notes: This was an EPA Mine Waste Technology Program (MWTP), paper covers 4 years of monitoring

Butte, Montana:

References: McCarthy et al. (1999) 5th Internat. Conf. on Tailings and Mine Waste, 725-733.
Gammons et al. (2000) ICARD, 2, 1159-1168.

Mine Type: historic metal Mining district

Inflow quality (µg/L): Metro Storm Drain: As=6, Cd=42, Cu=100-500, Pb=3, Zn=12176
Butte Reduction Works Wetlands: As=4.2, Cd=39.4, Cu=328.8, Pb=3.2, Zn=11273, pH=7.6
Colorado Tailings Wetlands: As=8.4, Cd=34.4, Cu=286.1, Pb=2.0, Zn=9474, pH=7.2

Effluent quality: Metro Storm Drain: As=25-32, Cd#1, Cu=15-36, Pb=3, Zn=242-1516
Butte Reduction Works Wetlands: As=5.1, Cd=25.0, Cu=84.7, Pb=2.7, Zn=6995.2, pH=7.9
Colorado Tailings Wetlands: As=10.3, Cd=1.5, Cu=22.0, Pb=2.1, Zn=68, pH=8.3

Treatment process: Metro Storm Drain: four parallel cells - Cell 1=horizontal flow, anaerobic, gravel-nonorganic substrate; Cell 2=horizontal flow, anaerobic, gravel-20% compost substrate; Cell 3=vertical flow, anaerobic, gravel-50% compost substrate; Cell 4=same as cell 1 but half the size/RT - effluents run through aerobic ponds & back to the drains
Butte Reduction Works Wetlands: three open water cells separated by two treatment walls (one of d_{50}=6" river rock, other of 90% rock/10% compost) which serve as subsurface flow zones where physical, chemical, and biological processes take place.
Colorado Tailings Wetlands: same as BRW, but with lime addition

Capacity: MSD: 10 gpm per cell
BRW: receives approx 50-150 gpm from Missoula Gulch baseflow (urban storm sewers)
CTW: captures & treats contaminated alluvial groundwater at approx 125 gpm

Cost estimates:

Other notes:

Calliope (abandoned) Mine, Butte, Montana:

References: Zaluski et al. (2000) ICARD pg 1169
Zaluski et al. (1999) In: Phytoremediation and Innovative Strategies for Specialized Remedial Applications. 5th Internat. In Situ and On-Site Bioremediation Symp. pg 205-210.

Mine Type: metal mine

Inflow quality: pH=4, Al=5.7mg/L, As=26µg/L, Cd=38µg/L, Cu=1.7mg/L, Fe=8.7mg/L, Mn=3.0mg/L, Zn=7.9mg/L, SO_4=165mg/L, alk=<10mg/L $CaCO_3$

Effluent quality: pH approx 8; Zn, Cd, Cu and Mn decreased significantly but weren't quantified well

Treatment process: Organic carbon pretreatment lowers Eh, organic carbon substrate supplies food for SULFATE REDUCING BACTERIA*, crushed limestone pretreatment provides alkalinity and increases pH, cobbles are used for the stable substrate for SULFATE REDUCING BACTERIA* growth. All cells were 12' wide

Cell II - below grade, 5' organic pretreatment, 5' crushed limestone pretreatment, 5' organic substrate, 50' cobbles
Cell III - below grade, 5' organic substrate, 50' cobbles
Cell IV - above grade, 5' organic pretreatment, 5' crushed limestone pretreatment, 5' organic substrate, 50' cobbles

Capacity: 1 gpm, 5.5 day residence time

Cost estimates:

Other notes: in general, pretreatment and below grade gave the best results

Mike Horse Mine, Lewis & Clark County, Montana:

References: Anderson & Hansen (1999) 5th Internat. Conf. on Tailings and Mine Waste, 715-724.

Inflow quality (mg/L): pH = 5.5-6.0, Fe=60-180, Zn=20->150, alkalinity < 100, Cd=0.25, Cu=2.3-3.0, Pb=0.35-0.37, Mn=76-86, SO_4=2000-3000

Effluent quality: pH = 6.7, Fe=0.31, Zn=14, alkalinity < 100, Cd<0.001, Cu=0.008, Pb<0.002, SO_4=2200

Treatment process: went online in 1996; ALD within the mine (120 m^3 limestone, 15-30 cm) 6 jet pump aeration system 6 lined oxidation/precipitation pond (2300 m^3) 6 rand filter 6 anaerobic gravel wetland (0.8 ha) 6 aerobic wetland

Capacity: 1.95-4.40 L/s

Cost estimates:

Other notes:

Big Five Wetland, Idaho Springs, Colorado:

References: Wildeman et al. (1993) Wetland Design for Mining Operations

Mine Type: abandoned metal mine

Inflow quality (mg/L): pH=2.8-3.2, Mn=29-35, Fe=33-60, Zn=9, Cu=0.5-0.7

Effluent quality: (approx. values, 1 yr of monitoring) pH=6, Mn=20-35, Fe<20, Zn<1, Cu<0.05

Treatment process: Concrete structure (W=10', L=60', D=2') divided into three 20', 30 mil Hypalon-lined, sections separated by 2x6 treated wood. Substrates for cells A, B, and C were mushroom compost (50/50 animal manure/barley mash waste); equal parts peat, aged steer manure, decomposed wood shavings and sawdust; and the same as cell B underlain by 4-6" of 2-3" limestone rock, respectively. Cattail, sedge, and rush species were transplanted into the cells. Cell A was modified to increase the flow path. Cell B was modified into upflow and downflow cells, and cell D (aerobic polishing) and cell E (downflow, subsurface wetland) were constructed using cell B substrate.

Capacity: #2.6 L/min

Cost estimates:

Other notes: Winter operation possible if delivery systems are insulated, cells receive winter sun, top insulated with hay and plastic

West Fork Unit, Reynolds Co, Missouri:

References: Gusek et al. (2000) ICARD, 2, 1133-1137
Gusek et al. (1999) In: Phytoremediation and Innovative Strategies for Specialized Remedial Applications. 5th Internat. In Situ and On-Site Bioremediation Symp.

Mine type: Pb-Zn

Inflow quality (mg/L): Pb=0.4, Zn=0.36, Cd=0.003, Cu=0.037, pH=7.94

Effluent quality: Pb=0.03-0.05, Zn=0.06-0.09, Cd<0.002, Cu<0.008, pH=6.6-7.8, H_2S=0.01-0.03, BOD<1-3

Treatment process: two stage biotreatment; 1) anaerobic, sulfate reduction for Pb removal and 2) aerobic rock filter/wetland to reoxygenate & polish accomplished in five major parts - settling pond (3030 m^2 surface, 1930 m^2 base, 3' deep), two anaerobic cells (1930 m^2 surface, 1390 m^2 base, cow manure sawdust, limestone & alfalfa substrate), rock filter (5900 m^2 by 1' w/ limestone cobbles), and an aeration pond (8000 m^2).

Capacity: 1200 gpm

Cost estimates: construction approx. $500,000

Other notes:

Tennessee Valley Authority:

References: Wildeman et al. (1993) Wetland Design for Mining Operations; Skousen et al. (1998)

Mine Type: Coal power utility

Inflow quality (mg/L): Generally: (approx. values from all 13 systems) pH = 3.5-6.2, Fe = 0.7-150, Mn = 4-17, acidity = 350, alkalinity=0
Nine systems: pH=4-6.5, Fe<70, Mn<17, Al<30, net alkalinity=35-300
Four systems: Fe>170, no net alkalinity
An additional system: Fe<0.7, Mn=5.3

Effluent quality: Generally: pH = 3-7, Fe = 0.4-83, Mn = 0.6-13, acidity=40, alkalinity=100
Nine systems: meet discharge standards
Four systems: two use an ALD, two require NaOH treatment
An additional system: ineffective at Mn removal

Treatment process: Aerobic wetland
In general: ALD (alkalinity) 6 deep pond (Fe/Mn removal) 6 deep marsh (Fe removal) 6 shallow marsh (Mn removal) 6 rock filter (Mn removal) 6 alkaline bed (pH increase) 6 polishing cell
ALD: trench (W=10-15', L=260', D=5') installed upstream from the impoundment in a seepage area, filled with 3/4 - 1-1/2" limestone, covered with two layers of 10 mil plastic, geofabric, and 2' local soil, seeded, mulched, and fertilized

Capacity: maximum flow ranged from 250 - 7700 L/min, Fe load # 21 gdm, Mn load # 2gdm

Cost estimates: TOTAL $50,000 = design $6000 + permitting $1000 + land acquisition $6000 + construction $22,000 + operation & maintenance $9000 + misc. $6000
ALD alone cost $19,000

Other notes: additional references = Brodie et al. 1991, 1992, 1993

Various sites in West Virginia:

References: Faulkner & Skousen (1994), Pitt, 2, 250-257.
Mine type: abandoned coal mines
Results:

Treatment System	Parameter of Concern	Influent (mg/L)	Effluent (mg/L)	% Reduction
Wetlands (N=5)	pH	2.5-4.0	3.1-5.4	---
	Acidity	106-2388	56-1366	3-76
	Alkalinity	---	---	---
	Fe	10-376	2-86	62-80
	Mn	8-51	7-50	0-11
	Al	9-206	7-163	25-63
	SO_4	217-2821	227-2660	---
ALDs (N=19)	pH	2.6-5.7	3.4-6.8	---
	Acidity	170-2210	0-1000	11-100
	Alkalinity	0-23	0-340	---
	Fe	22-570	1-360	---
	Mn	3-200	2-160	---
	Al	0-287	0-100	---
	SO_4	101-2900	45-2000	---

Treatment process: Wetlands: In general, 0.6 to 1 m of organic material composed of peat and hay overlie about 15-30 cm of limestone. Hay bales were usually used as barriers to slow and direct flow. Two wetlands (S. Kelly and Z&F) were designed to encourage subsurface flow.
ALDs: From the bottom, constructed with 10- to 20- mil liners, filter fabric, gravel-sized limestone, and hay bales separated by the filter fabric. Liners were wrapped around the limestone and hay.

Capacity: Wetlands: 17-98 L/min, 408-1417 m^2
ALDs: 4-87 L/min

Cost estimates: Wetlands: max. installation cost for the smallest wetland = $225,000, $110,000 reported for another.

Other notes:

Various sites in Appalachia:

References: Hedin & Watzlaf (1994), Pitt, 1, 185-194.

Mine type: abandoned coal mines

Results:

Treatment System	Parameter of Concern	Influent (mg/L)	Effluent (mg/L)
ALDs (N=21)	pH	2.3-5.9	3.6-6.7
	Alkalinity	0-33	0-469
	Fe	3-1416	0-625
	Mn	4-136	2-132
	Al	0-486	0-152
	SO_4	430-6719	55-5432

Design Criteria: Limestone mass = 35-945 mt, Mass per flow = 1.0-324 mt/L/min

Capacity: Flow = 0.5-218 L/min, detention time = 5-1588 hrs

Cost estimates:

Other notes: mass of limestone required, $M=Q\rho_b t_d/V_v + QCT/x$
where, Q=flow, ρ_b=bulk density of limestone, t_d=detention time, V_v=bulk void volume (decimal), C=expected alkalinity (mg/L), T=design life, x=$CaCO_3$ content of limestone (decimal)

CANADA

Greater Vancouver Area:

References: McGregor et al. (2000) ICARD, pg 1227
McGregor et al. (1999) SUD, pg 645

Inflow quality (µg/L): tidal flow, pH=6.36, Cd=15.9, Cu=4510, Ni=118, Pb=3.8, Zn=2396

Effluent quality: pH=6.57, Cd<0.1, Cu=7.7, Ni=6.5, Pb=0.7, Zn=27.5

Barrier description: reactive material = 85% pea gravel, 15% compost (by vol.) and a trace amount of limestone; dimensions = 6.7m deep, 10m long, 2.5m thick

Capacity: maximum flow estimated at 66 cm/d

Cost estimates:

Other notes: residence time estimated at 3.8 days; measurements taken after 21 months

Bell Copper Mine, near Smithers, British Columbia:

References: Sobolewski et al. (1995) Mining and the Environment, V2, 683-692.

Mine Type: Cu

Inflow quality (mg/L): source #1: pH=6-8, Cu=0.3-1.0;
source #2: pH=3.5, Cu=35-50;
SO_4>2000 for both

Effluent quality: for about 6 weeks after receiving high Cu flow...pH=7.1-7.5, Cu generally less than 1, spikes up to 10

Treatment process: (experimental systems) large (300 m^2) and small (75 m^2) ponds fertilized with manure and planted with floated peat mats

Capacity: large pond: 8 L/min, retention time = 12 days
small pond: 2 L/min, retention time = 22.5 days

Cost estimates:

Other notes:

Makela Test Cell System-INCO Copper Cliff's tailings near Sudbury, Ontario:

References: Fyson et al. (1995) Mining and the Environment, V2, 459-466.

Mine Type:

Inflow quality (mg/L): pH=5.65-6.04, Al<1, Cu<1, Fe=220-420, Ni=17.1-74.4, S=800-869

Effluent quality: pH=6.1-6.3, acidity-28-339 Al-1, Cu-0.01-0.22, Fe-1-7.6, Ni-0.8, S-298-450

Treatment process: ARUM = 'acid reduction using microbiology'; four cells constructed in series (100m x 20m) with two holding ponds between cells 2&3 and cells 3&4; Cell 1= oxidation/precipitation, Cell 2= oxidation/precipitation, Cell 3= ARUM w/ floating cattail rafts, Cell 4=ARUM w/ floating cattail rafts

Capacity: 1 L/min, retention time = 168 days

Cost estimates:

Other notes:

Nickel Rim, Sudbury, Ontario:

References:	Benner et al. (2000) ICARD, pg 1221 Mayer et al. (1999) Sud, pg 145 (modeling) Benner et al. (1997) GW Monitor. and Remed. Benner et al. (1997) ES&T Blowes et al. (1995) Sud, pg 979 (test cell)
Inflow quality:	flow=49ft/yr, pH=4-6, SO_4=2400-4600mg/L, Fe=250-1300mg/L, Ni<500µmol/L, alkalinity=0-50mg/L (as $CaCO_3$)
Effluent quality:	pH=7.0, SO_4=200-3600mg/L, Fe1-40mg/L, Ni<4 µmol/L, alkalinity=600-2000mg/L
Barrier description:	dimensions = 15m long, 3.6m deep, 4m thick in the direction of flow; substrate = 40% municipal compost, 40% leaf compost, 20% wood chips; pea gravel mixed in to maintain hydraulic conductivity (50/50 organic/gravel)
Capacity:	16 m/yr
Cost estimates:	materials and installation = US$ 30000, approx. half for each
Other notes:	controlling factors = ground water temp., residence time, organic carbon reactivity; removal efficiencies (linked to organic carbon reactivity) have been declining over the 38 months of operation

Halifax International Airport:

References:	Bechard et al. (1995) SUD, pg 545 Bechard et al. (1991) ICARD, pg 171
Inflow quality (mg/L):	see note below
Effluent quality:	see note below
Barrier description:	three clay lined cells built in sequence; dimensions of each cell = 16.8m long, 1m wide, 1m deep; substrate = 1 part planer shavings to 3 parts partially decomposed straw, 54kg sucrose, 6kg urea fertilizer, 1kg phosphate rock
Capacity:	23 L/min
Cost estimates:	
Other notes:	The cells were not very effective. Best results were observed between days 252 and 336 (less than three months) out of >500 days of monitoring.

AUSTRALIA

Hilton Mine, northern Australia:

References: Jones et al. (1995) Mining and the Environment, v2, pg 755-763.

Mine Type: Ag, Pb, Zn

Inflow quality (mg/L): Zn>6.5, Mn -3, pH -7

Effluent quality: Zn -0.3 μM, Mn<0.06, pH -8

Treatment process: Aerobic metals removal: mine water is pumped to surface settling ponds, flows freely down a hillside to an earthen channel leading to an evaporation pond, and flows down another channel to the Hilton Pump Weir, which pumps water to the ore processing plant

Capacity: up to 5000 m^3/d

Cost estimates:

Other notes:

EUROPE

Spain:

References: Ordonez et al. (2000) 5th ICARD, V2, pg 1121-1129.

Inflow quality (mg/L): pH=3.4, DO=4.3, NH_4^+=19.1, SO_4^{2-}=7779, Al=135, FeII=4.6, FeIII=2.6, Mn=140

Effluent quality: pH=7.2, DO=0.2, NH_4^+=92, SO_4^{2-}=7193, Al=1.0, FeII=0.2, FeIII=1.8, Mn=55

Treatment process: SAPS (21 m^3 tanks filled w/ 1m limestone, 0.5m of 70% cow manure, 30% straw) 6 ALD (20m x 1m x 1m, isolated w/ liner) 6 oxidation cascade (3m long, 7-10cm high steps) 6 sedimentation pond (6m x 4m x 1m) 6 substrate wetland (20m x 8m x 1.2m, lined, 60% cow manure, 20% straw, 20% sand) 6 limestone filter

Capacity: regulated flow at 200 L/hr

Cost estimates:

Other notes:

Table A6.1.3. Bibliographic annotations of research related to passive constructed wetland systems, including anoxic limestone drains and permeable reactive barriers. Italicized numbers refer to reference numbers for the Reclamation Section's literature database (Papyrus).

Reference	Study Scale[1]	Passive Treatment Components					
		ALD	Anaerobic Wetland	Aerobic Wetlan d	Settlin g Pond	Reactiv e Barrier	Bio-reactor
Brodie et al. (1992) *#702*	O	U					
Hedin & Nairn (1992) *#701*	R	U					
Kleinmann & Hedin (1993) *#1747*	R	U					
Watzlaf & Hedin (1993) *#703*	L	U					
Hedin et al. (1994) *#2911*	R, M	U					
Brant et al. (1995)	L, F	U					
Wildeman et al. (1997)	R, L, F	U					
Cravotta & Trahan (1999)	O	U (oxic)					
Eger et al. (1997)	O	U	U				
Brodie et al. (1991) *#592*	O	U		U			
Brodie (1991) *#597*	O	U		U			
Skousen (1991) *#598*	R, O	U		U			
Brodie et al. (1992) *#704*	O	U		U			
Skousen & Faulkner (1992) *#595*	O	U		U			
Ordonez et al. (2000)	L, F	U	U		U		
Kuyucak & St-Germain (1994)	L	U	U				U
Anderson & Hansen (1999)	O	U	U	U	U		
Kalin & Smith (1991) *#1390*	L, F		U				
Eger (1992) *#2571*	F		U				
Borek et al. (1995)	F		U				

Reference	Study Scale[1]	ALD	Anaerobic Wetland	Aerobic Wetland	Settling Pond	Reactive Barrier	Bio-reactor
Eger & Wagner (1995) *#2879*	F		U				
Fyson et al. (1995)	F		U				
Sobolewski et al. (1995)	F		U				
Willow et al. (1998)	L		U				
Kalin et al. (1999)	L		U				
McCarthy et al. (1999)	F		U				
Tisch et al. (1999)	L		U				
Marchand & Silverstein (2000)	L		U				
Reisinger et al. (2000)	F		U				
Schmiermund (2000)	F		U				
Gammons et al. (2000)	O		U		U		
Eger et al. (2000)	F			U			
Jones et al. (1995)	O				U		
Blowes & Ptacek (1992)	L, M					U	
Bechard et al. (1995)	F					U	
Blowes et al. (1995)	F					U	
Waybrant et al. (1995)	L					U	
Benner et al. (1997)	O					U	
Waybrant et al. (1997)	M, L					U	
Blowes et al. (1998) *#3764*	R					U	
Benner et al. (1999) *#3437*	M					U	
Conca et al. (1999)	L					U	
Martin & Kempton (1999)	M, L					U	
Mayer et al. (1999)	M					U	
McRae et al. (1999)	L					U	
Astrup et al. (2000) *#3852*	L					U	

Reference	Study Scale[1]	Passive Treatment Components					
		ALD	Anaerobic Wetland	Aerobic Wetlan d	Settlin g Pond	Reactiv e Barrier	Bio-reactor
Chilakapati et al. (2000) Environ. Sci. &Tech.	M					U	
Greenwald et al. (1999)	F					U	
McGregor et al. (1999)	O					U	
Benner et al. (2000)	O					U	
McGregor et al. (2000)	O					U	
Phillips et al. (2000) *#3853*	F					U	
Su & Puls (2001)	L					U	
Yabusaki et al. (2001)	F					U	
Kleinmann et al. (1991) *#1284*	F						U
Kuyucak & St-Germain (1994) *#2874*	L						U
Bechard et al. (1995)	F						U
Beaulieu et al. (1999)	L						U
Canty (1999a)	F						U
Canty (1999b)*#3518*	F						U
Gusek et al. (1999)	O				U		U
Kolmert et al. (1999)	L						U
Zaluski et al. (1999)	F						U
Canty (2000)	F						U
Gusek et al. (2000)	O				U		U
Johnson et al. (2000)	L						U
Zaluski et al. (2000)	F						U

[1] R= review/survey,M=modeling, L= laboratory, F= field, O= operation

Table A6.2.1. Active Treatment Systems Treating Hard Rock Mine Drainage

UNITED STATES

LTV Steel Dunka Mine, MN:

References:	1. LTV Steel Mining Company Dunka Mine, Updated Final Closure Plan, Rev. 4, Mar. 15, 1996. 2. Dunka Data Summary: 1976-1993, Draft, MN DNR, Div. of Minerals, March 15, 1995.
Inflow quality:	pH generally neutral, occasional acidic drainage from Seeps 1 & 3, Ni = 1-20 mg/L, occasionally as high as 30 mg/L, Cu = 0-1 mg/L
Treatment process:	Lime (CaO) is added to the water to bring the pH up to at least 10, creating base metal carbonate and/or sulfate precipitates. The precipitate is treated with both cationic and anionic flocculents to enhance settling. The water then passes into a plate type thickener. The flocculated solids settle to the bottom of the thickener and form a sludge which is pumped to a holding tank. From there, the sludge is pumped to a pressure filter where the solids are retained on a filter media forming a filter cake for disposal. The filtrate is pumped back to the system for reprocessing. The clear water overflow from the plate thickener passes through a sand filter which serves as a final polishing step. The water then passes to a pH adjustment tank where a small amount of sulfuric acid is added to bring the pH back to the compliance level range of 6.5 to 8.5. In actuality, the pH is controlled to 7.0.
Capacity:	Annual flow from EM8 (roughly 2/3 of flow) and Seeps 1, 3, & X = approximately 3 x 10^6 L/yr, which flows to a collection basin and into the treatment plant. Seep 1 flow first passes through a limestone/peat pretreatment system.
Cost estimates:	Operation = roughly $200,000/yr, Construction (equalization pond, pump stations, treatment plant) = $1,494,000
Other notes:	LTV contact person - Jason Augenes, 218-225-4364

Hibbing Research Facility, MN:

References:	MN DNR, unpublished data
Inflow quality:	pH 4, Cu=50-150 ppm, Ni=100-300 ppm, Co=10-30 ppm, Zn=10-30 ppm
Effluent quality:	pH 7.25-9.0, Cu=0.001-0.050 ppm, Ni=0.137-2.1 ppm, Co=0.001-0.1 ppm, Zn=0.001-1.34 ppm, SO_4=30.4-421 ppm
Required limits:	pH=6.5-8.5, Cu=0.5 ppm, Ni=3.0 ppm, Co=1.0 ppm, Zn=4.0 ppm
Treatment process:	The treatment system is fully automated using a computerized control system to operate sensors, timers, and other electronic equipment to treat and discharge water generated at the research site. The water is neutralized in a 500 gal. mixing tank using magnesium hydroxide. The sludge produced is allowed to settle and the treated water overflows (i.e. gravity flow) to a settling basin and subsequently is pumped to the Hibbing sewer system. The sludge is pumped to a 500 gal. holding tank and then dewatered with a filter press. The sludge is landfilled after being analyzed and meeting TCLP limits.
Capacity:	*1.5 gpm*
Cost estimates:	Operation =$12,000/yr, Building construction = $65,000; Treatment system = $30,000

Other notes:

Eagle Mine:

Contact: Dan Scheppers, State of Colorado, Dept of Health, 303-692-3398
Inflow quality:
Treatment process: Lime precipitation of heavy metals with recirculation of sludge
Capacity: 160 gpm treated on average
Cost estimates: Treatment plant run by the responsible party, therefore no figures available
Other notes: additional contact = Gene Taylor, Project Manager, EPA, 303-312-6536

Summitville:

Contact: Angus Campbell, State of Colorado, Dept of Health, 303-692-3385
Inflow quality: pH = 3-3.2, Cu up to 40 mg/L (these are the monitoring parameters), also contains Fe, Mn, Zn, and Al
Treatment process: Hydrated lime precipitation process involving sludge recycle. Water enters a mixing tank that also receives lime (midpoint) and recycled sludge (bottom?). The mixture flows via gravity feed to a reactor and onto a 60' wide x 20' high thickener. A polymer is added to the flow just before the thickener to aid flocculation. Thickener underflow, 7-9% solids, is recycled to the mixing tank. The remaining sludge goes through a filter press and is disposed of on site in cells constructed in open pits. The sludge meets TCLP limits and is not considered hazardous. The small amount of NP provided by the sludge is considered beneficial to the entire system.
Capacity: The plant runs 24 hours a day during the summer months (approx. 9 month operating season). It consists of 2 “trains”, each receiving 550 gal/min inflow and 100 gal/min sludge = 650 gal/min per train or 1300 gal/min total flow. From April 27 through November 12, 1999, they treated 262,996,843 gal of water.
Cost estimates: Operating costs approximately 2.5 million $/year. No good information on capital costs because they renovated an existing processing plant from the 1960s. Renovation consisted mainly of modifying existing tanks and plumbing for mixing and aeration capabilities etc.
Other notes: Most of this information is contained in an Annual Report. Also, they will be testing several passive systems at Summitville during summer 2000 (a saturated organic limestone cell, zeolite system -actually done by a contractor-, and Aquafix).
Additional Contacts: Victor Ketalapper, EPA, 303-312-6578

Argo Tunnel:

Contact: Mary Scott, State of Colorado, Dept of Health, 303-692-3413
Inflow quality: pH = 2.8, Fe = 180 mg/L, Mn = 100 mg/L, Zn = 70 mg/L, Al = 25 mg/L, Cu = 7 mg/L, total load = 1200 lb metals/day
Treatment process: Chemical precipitation using 50% NaOH, more expensive but easier to handle except for the fact that the entire system must be heated because the NaOH solution freezes at 54 EF. Originally intended for the plant to be fully automated but that hasn’t worked out so far (since 1998). Crews have worked shifts twenty four hours a day since operation began.

Inflow enters a rapid mixing tank, where it is mixed with a 50% NaOH solution to elevate the pH to 9.9. Water then flows into an Infilco Degremont Process Tank, which is essentially two tanks stuck together. The first unit is the particle building tank where water is circulated to enhance particle interaction. A polymer and about 5% of the sludge (depending on flow, e.g. 200 gpm flow requires approx. 10 gpm sludge) from the clarifier are added here to improve flocculation. This product flows into the second unit, a 12' diameter clarification tank. Sludge from the bottom of this unit is 3-4% solids. Sludge is pumped to liquid sludge storage and onto a filter press (mechanical dewatering). The filter cake is roughly 16% solids and disposed of in a municipal landfill. They are looking at building an on-site sludge repository but that may be a couple years away yet.

Clarified water (top of the clarification unit) has low turbidity (approx. 0.2 ntu) but still contains about 1 mg/L Mn. Therefore it is run through a mixed media gravity filter, which decreases Mn to 0.2 mg/L. The pH of the water is then adjusted to 8 using CO_2.

Capacity: Average inflow is approximately 320 gpm, ranging from roughly 250 gpm during the winter months to 600 gpm during the spring thaw. They can handle up to 700 gpm on two drains (350 gpm each). However they only have about twelve hours worth of storage capacity in the event of a total shut down. On average they use 1000 gal/day of the 50% NaOH solution, producing 17 yd^3/day.

Cost estimates: Design = $1 million in engineering costs, capital = $1 million in equipment, construction = $3.8 million, operation = $1.2 million per year.

Other notes:

California Gulch Superfund Site, Leadville drainage:

Contact: Mike Holm, EPA, 303-312-6607

Inflow quality: pH ranges from 2.5 to 7, typical is 4-4.5

Treatment process: NaOH precipitation followed by pH adjustment using H_2SO_4. Sludge tends to fail TCLP and must be disposed of as hazardous waste. They are doing some testing of fixation compounds to bind metals so that the sludge can be disposed of as an industrial solid rather than hazardous waste.

Capacity: Can treat up to 2000 gpm, typically treats 1500-1700 gpm. However, subsidence in the area is changing the hydrology and the flow from this tunnel is increasing.

Cost estimates: Operation = $900,000 per year, Sludge disposal = $100,000 per year

Other notes: Automated operation run by the Bureau of Reclamation. Treats drainage from one of the tunnels in California Gulch. They are also looking at options to treat contaminated surface water around Leadville. Additional research in the area of phytoremediation is being conducted on a field scale and looks promising (removes approximately 70% of metals if pH >5.5). see EPA Tech Trends paper by Brad Littlepage and Duane Johnson.

Additional Contacts: Brad Littlepage, plant operator at 719-486-2035.

California Gulch Superfund Site, Yak Tunnel (Leadville):

Contact:	Mike Holm, EPA, 303-312-6607
Inflow quality:	pH ranges from 2.5 to 7, typical is 4-4.5
Treatment process:	NaOH precipitation followed by pH adjustment using H_2SO_4.
Capacity:	2000 gpm maximum. However, flow from this tunnel is decreasing. Currently the plant operates 3-4 days per week, treating approximately 400 gpm.
Cost estimates:	
Other notes:	Manual operation run through a joint venture between Asarco and Resurrection Mining Companies. Treats drainage from the other California Gulch tunnel.
Additional Contacts:	Gary Slifica, Asarco, 719-486-1056,

CANADA

Equity Silver:

Reference:	Billings 2000 Conference-CD ROM
Inflow quality:	Cu = 116 mg/L, Fe = 1340 mg/L, Zn = 154 mg/L
Treatment process:	Lime
Capacity:	60 - 150 L/s, approximately 5000 tonnes lime
Cost estimates:	$1.2 million/year
Other notes:	

Reference: Aziz & Ferguson, Van97, V1, pg 181

Inflow quality: average pH = 2.62, acidity = 8180 mg/L, Cu = 116 mg/L, Fe = 1340 mg/L, Zn = 154 mg/L

Treatment process: 'ARD collection ditches flow into two collection ponds. The upper pond was designed for extra storage during high flow periods. The lower pond has a pumphouse that can pump a maximum of 4850 USG/min. The ARD is normally pumped from this pond to a storage pond located beside the treatment plant, but a portion of the flow can also be pumped directly to the plant for processing.

Neutralization is achieved by adding quick lime slurry to a pH between 8.0 to 8.5. The ARD and lime slurry are combined in the first of three reaction tanks in series with a combined retention time of 20 minutes. Constant agitation is required to ensure maximum lime efficiency. The third reaction tank contains pH probes that control the addition of the lime slurry. Maximum plant throughput is 2150 gal/min.

From the treatment plant the neutralized slurry is discharged to settling ponds where the metals drop out as a metal hydroxide sludge and the treated supernatant decants to a holding pond. During the freshet period when regional creeks are at their maximum flows the treated water is released to the environment using dilution ratios that are specified in the Equity water permit. The ARD sludge is pumped to the Main Zone pit for long term storage.'

Capacity: 'Over the last nine years (pre-1997) the average volume of ARD collected and treated at Equity annually has been approximately 880,000 m^3. The lime required to treat this volume of ARD depends on the acidity, but has averaged 5,060 tonnes/yr."

Cost estimates: Altogether the annual cost of running the ARD collection and treatment system averages around $1.1 million of which the lime represents 70% of the cost.'

Other notes: additional contact = Bill Price, 250-847-7389

Table A6.2.2. Bibliographic annotations of research related to active treatment systems. Italicized numbers refer to reference numbers for the Reclamation Section's literature database (Papyrus).

Reference	Active Treatment System				
	Neutralization	Chemical Precipitation	Ion Exchange	Membrane Filtration	Biologica l
US EPA, (1983)	U				
Eger et al., (1993)	U				
Kuyucak et al., (1995)	U				
Murdock et al., (1995)	U				
Orava et al., (1995)	U				
Aube & Payant, (1997)	U				
Aziz & Ferguson, (1997)	U				
Miedecke et al., (1997)	U				
Poirier & Roy, (1997)	U				
Aube, (1999)	U				
Aube & Zinck, (1999)	U				
Ericksen et al., (1999)	U				
Willow & tenBraak, (1999)	U				
Zinck & Aube, (2000)	U				
Zinck & Griffith, (2000)	U				
Robertson & Rohrs, (1995)	U	U			
Berg & Arthur, (1999)	U	U		U	
Smit, (2000) *#3918*	U	U			
Lapakko, (1993)		U			
Mitchell & Atkinson, (1995) Mitchell & Wheaton, (1999) Mitchell et al., (2000)		U			
Rybock et al., (1999) *#3672*		U			
McKinnon et al., (2000)		U			
Aube & Stroiazzo, (2000)		U			

Reference	Active Treatment System				
	Neutralization	Chemical Precipitation	Ion Exchange	Membrane Filtration	Biologica l
Choung et al., (2000)		U			
Sato & Robbins, (2000)		U			
Leppert et al., (1990) *#802*			U		
Gussmann et al., (1991) *#1357*			U		
Vos & O'Hearn, (1993) *#1415*			U		
Schultze et al., (1994)			U		
Riveros, (1995)			U		
Gilbert et al., (1999)			U		
Metre-General Inc.			U (chelation)		
Sastri & Ashbrook, (1976) *#3574*				U	
Blackshaw et al., (1983) *#1185*				U	
Awadalla & Hazlett, (1992) *#2177*				U	
Stewart et al., (1997)				U	
Nakamura, (1988) *#1488*					U
Dvorak et al., (1991) *#1387*					U
Borek et al., (1995)					U
de Vegt et al., (1997; 1998) *#2781; #3129*					U
Diaz et al., (1997)					U
Rowley et al., (1997)					U
Dijkman et al., (1999)					U
Kolmert et al., (1999)					U
Federal Remediation Technology Roundtable		U	U		U
Bowell, (2000) *#3748*		U	U	U	U

APPENDIX 7

Integrated Case Studies

A7.1. Dunka Mine, Minnesota.

Table A7.1.1. Dunka rock stockpile summary (MN DNR, 1996).

Stockpile	Stockpile Size		Chemical Composition		
	Area (ft^2 x 10^6)	Mass (LT x 10^6)	% NiO_2	% CuO	% S
8011	3.73	20.50	0.03	0.06	0.23
8013	3.58	17.75	0.03	0.09	0.24
8015	NA	4.80	0.02	0.11	1.00
8031	3.17	9.50	0.04	0.10	0.35
8027	NA	0.14	<0.01	0.01	0.12
8012	0.35	1.20	0.07	0.27	NA
8014	2.40	7.00	0.09	0.28	0.73
8018	0.76	2.20	0.09	0.28	0.98

A7.2. Flambeau Mine, Wisconsin.

Table A7.2.1. Calculations showed that the flux of backfill pore water to the Flambeau River would be 6.5 x 10^5 times lower than the ten year, seven day, low flow for the river (Flambeau Mining Company, 2001).

Parameter	Magnitude
Permeability (K)	0.028 ft/d
Hydraulic gradient (I) between west end of pit & river	0.03 ft/ft
Cross-sectional area of pit (A) pit width aquifer depth	65000 ft^2 650 ft 100 ft
Pore water flux (Q = KIA)	54.5 cfd or 0.00063 cfs
Flambeau River average flow	1749 cfs
Flambeau River 10-yr, 7-d low flow	412 cfs

Table A7.2.2. Summary of surface water monitoring results between July 1991 and November 2000 (Flambeau Mining Company, 2001).

Parameter	SW-1 (upstream)	SW-2 (downstream)
pH	6.5 - 8.7	5.5 - 8.3
Alkalinity (mg/L)	nd	nd
Cu (µg/L)	0 - 0.012	0 - 0.012
SO_4 (mg/L)	5.2 - 8.6	5.2 - 8.4

Table A7.2.3. Ground water monitoring well specifications in and around the Flambeau Pit (Flambeau Mining Company., 2001).

Monitoring Well	Depth (feet)		Location	Material Contacted in Pit
	Original	Adjusted		
MW-1000 PR	57.37	57.8	Down gradient from backfill	--
MW-1004 S	38.02	27.0	Northwest edge of pit	--
MW-1004 P	89.34	75.9	Northwest edge of pit	--
MW-1002	17.71	--	North of pit	--
MW -1002 G	53.88	--	North of pit	--
MW-1005	20.16	--	East side of operation	--
MW-1005 S	52.43	--	East side of operation	--
MW-1005 P	93.46	--	East side of operation	--
MW-1010 P	115.4	--	Down gradient from backfill	--
MW-1013	24.2	--	Backfill pore water	Till
MW-1013 A	46.4	--	Backfill pore water	Low S rock
MW-1013 B	86.3	--	Backfill pore water	High S rock
MW-1013 C	201.5	--	Backfill pore water	High S rock
MW-1014	33.7	--	Backfill pore water	Sandstone
MW-1014 A	63.9	--	Backfill pore water	Low S rock
MW-1014 B	104.9	--	Backfill pore water	High S rock
MW-1014 C	156.6	--	Backfill pore water	High S rock

Table A7.2.4. Most ground water quality parameters are monitored on a quarterly basis. However, a more extensive analysis is conducted annually (Flambeau Mining Company, 2001).

Parameter	Monitoring Frequency
Field pH	Quarterly
Lab pH	Quarterly
TDS	Quarterly
Hardness	Quarterly
Alkalinity	Quarterly
As	Annually
Ba	Annually
Cd	Annually
Cr	Annually
Cu	Quarterly
Fe	Quarterly
Pb	Annually
Mn	Quarterly
Hg	Annually
Se	Annually
Ag	Annually
Zn	Annually
SO_4	Quarterly
Color	Quarterly
Field Conductivity	Quarterly
Odor	Quarterly
Turbidity	Quarterly

Table 7.2.2.5. Partial summary statistics for ground water monitoring wells in the Flambeau Pit backfill (Flambeau Mining Company, 2001). Monitoring results from February 1999 to April 2000.

<u>Copper (µg/L)</u>

Monitoring Well	Minimum	Maximum	N	Material Contacted in Pit
MW-1013	ND	ND	ND	Till
MW-1013 A	ND	ND	ND	Low S rock
MW-1013 B	<4.7	36	6	High S rock
MW-1013 C	<4.7	100	6	High S rock
MW-1014	ND	ND	ND	Sandstone
MW-1014 A	<6.0	<6.0	1	Low S rock
MW-1014 B	420	810	6	High S rock
MW-1014 C	<0.47	16	6	High S rock

ND = No data, well not recovered sufficiently to collect ground water sample.

<u>Alkalinity (mg/L)</u>

Monitoring Well	Minimum	Maximum	N	Material Contacted in Pit
MW-1013	ND	ND	ND	Till
MW-1013 A	ND	ND	ND	Low S rock
MW-1013 B	520	630	6	High S rock
MW-1013 C	400	510	6	High S rock
MW-1014	ND	ND	ND	Sandstone
MW-1014 A	390	390	1	Low S rock
MW-1014 B	460	570	6	High S rock
MW-1014 C	300	380	6	High S rock

ND = No data, well not recovered sufficiently to collect ground water sample.

Sulfate (mg/L)

Monitoring Well	Minimum	Maximum	N	Material Contacted in Pit
MW-1013	ND	ND	ND	Till
MW-1013 A	ND	ND	ND	Low S rock
MW-1013 B	770	1900	6	High S rock
MW-1013 C	870	2000	6	High S rock
MW-1014	ND	ND	ND	Sandstone
MW-1014 A	970	970	1	Low S rock
MW-1014 B	580	1600	6	High S rock
MW-1014 C	370	700	6	High S rock

ND = No data, well not recovered sufficiently to collect ground water sample.

A7.3. Richmond Hill Mine, South Dakota.

A7.3.1. Excerpts from the Summary of the Mining Industry in South Dakota 1999. Prepared by the Minerals and Mining Program, South Dakota Department of Environment and Natural Resources, April 2000.
http://www.state.sd.us/denr/DES/Mining/1999minesum.html

Acid Mine Drainage Mitigation Update at Richmond Hill Mine, pg 5:

The pit impoundment, backfilled with acid generating rock and covered with a low permeability capping system, continued to exceed expectations in 1999. Four seasons of monitoring data show that only minimum amounts of oxygen and water are being detected in the impoundment. This indicates the cap is effective in limiting oxygen and water infiltration and is preventing acid generation. No signs of settling or slumping were found during several inspections of the pit impoundment by the department and LAC contractors. Only minor erosion was noted on a few portions of the impoundment. A dense, self-sustaining vegetative cover is becoming established on the pit impoundment and waste dump area.

The capped leach pads are also performing well. No signs of settling or slumping were found on the leach pads. Only minor erosion was noted in several areas. A good vegetative cover is becoming established. Monitoring data shows that the capping systems are effective in reducing water infiltration into the spent ore. Because of the low metal concentrations in the pad effluent, LAC believes that passive treatment may be feasible for long-term water treatment. A pilot plant has been constructed to test passive treatment, and test results are so far encouraging.

Ground and surface water quality around the mine site continues to improve. Biological assessments of Squaw Creek below the mine show that the stream is healthy. At the end of 1999, LAC discontinued water treatment for the winter since the volume of water requiring treatment had been reduced to almost zero. Only seasonal water treatment will be conducted, starting in the summer of 2000. Because of decreased water treatment and reclamation requirements at the site, LAC reduced its workforce to three employees at the end of the year.

LAC Minerals (USA), Inc./Richmond Hill Mine, pg 20:

LAC Minerals may make minor changes to the reclamation plan, including changes to roads and final pond configurations. Proposals on final water treatment plans, including passive treatment, may be submitted to the department. They plan to continue monitoring and seasonal water treatment in 2000.

Table A7.3.1. Typical drainage chemistry from the Spruce Gulch waste rock depository between 1992 and 1995 was acidic and contained elevated heavy metals (Durkin, 1995; Deux, 2000). Concentrations are in mg/L unless noted otherwise.

Parameter	Typical Concentration Range
pH	2.6 - 3.6 s.u.
TDS	800 - 5700
SO_4	700 - 3400
Fe	50
Al	100
Cu	8
Zn	7
Flow	100 - 200 gpm

Table A7.3.2. Monitoring results of the Richmond Hill Engineered Pit Backfill Facility indicate that the reclamation plan has been successful to date (Duex, 2000).

Parameter Monitored	Result
Inflow	
Basin Lysimeter Readings	0.43 gpd in 1998
Barrel Lysimeter Readings	four of six have remained dry; cover permeability < 1 x 10^{-7} cm/s
Pore Gases - Oxygen	0 - 4.8%
Temperature	< 20EC
Settlement	total = 0.12 ft or 0.1%
Ground Water	metals generally meet ground water standards
Surface Water (Squaw Creek)	no measurable affects
Revegetation	a good vegetative cover has been established

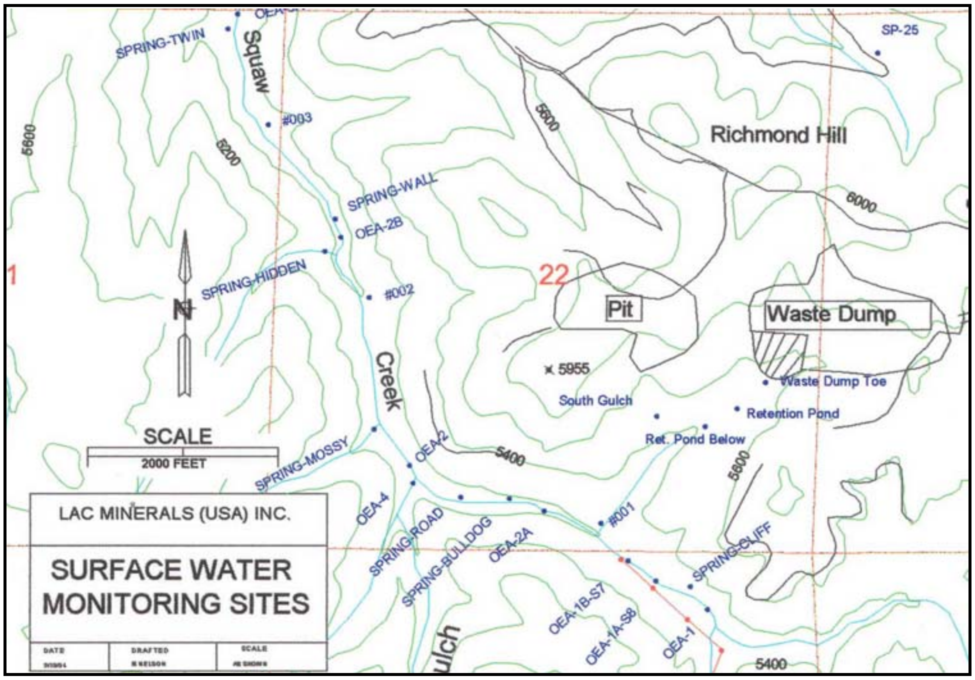

Figure A7.3.1. Partial map of surface water monitoring sites, Richmond Hill Mine, South Dakota (Nelson, 2001).

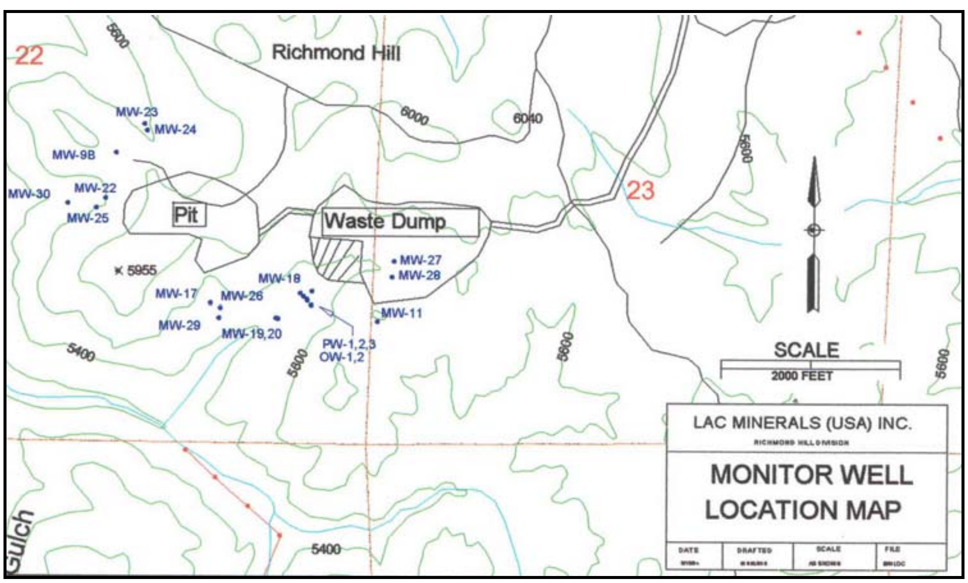

Figure A7.3.2. Partial monitoring well location map at the Richmond Hill Mine, South Dakota (Nelson, 2001).

Richmond Hill Mine, South Dakota

Water Quality from the Waste Dump Toe.

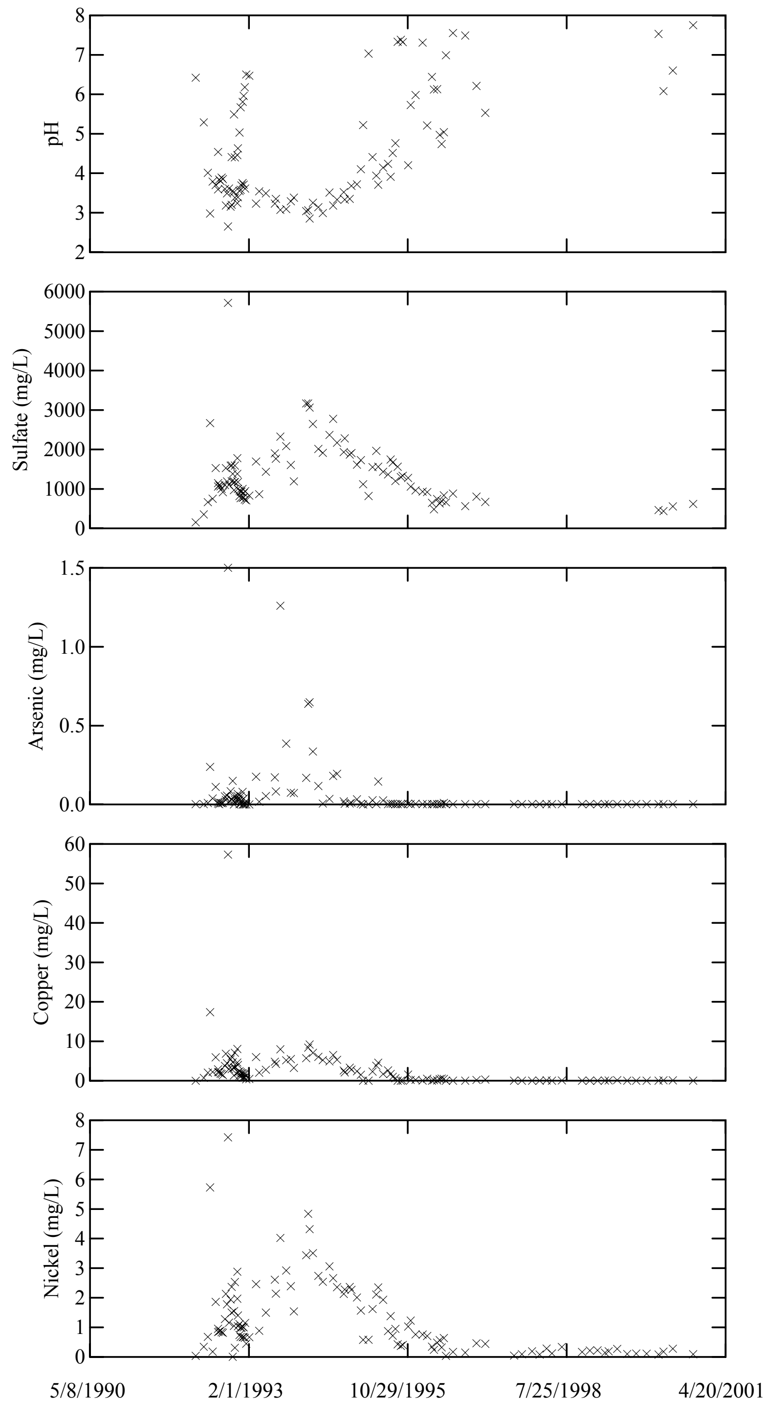

Figure A7.3.3. Surface water quality monitoring results from the toe of the waste dump, Richmond Hill Mine, South Dakota.

Richmond Hill Mine, South Dakota

Ground Water Quality Near the Backfilled Pit and Former Wast

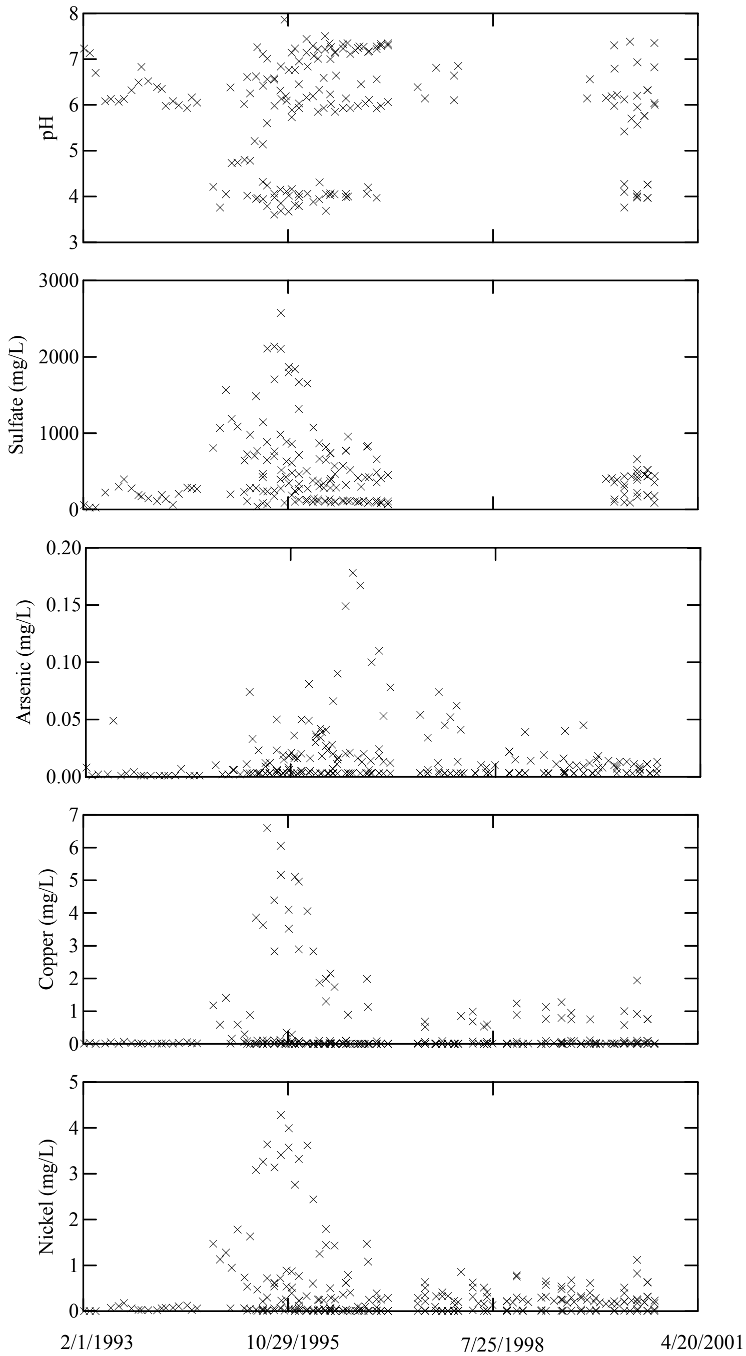

Figure A7.3.4 Ground water quality found in monitoring wells located around the backfilled pit and former waste dump at the Richmond Hill Mine, South Dakota.

A7.4. Additional sources of information regarding integrated environmental mine waste managment system case studies.

Web Sites that Present Case Studies:

www.infomine.com
http://mend2000.nrcan.gc.ca
http://www.inap.com.au/inap/homepage.nsf
www.epa.gov/ORD/NRMRL/

Summitville Mine, CO

Strategies:

- Removal of waste rock piles,
- Backfilling and capping the open pit mine,
- Closure of the heap leach pad,
- Plugging two drainage adits,
- Active treatment of drainages (acid neutralization-precipitation)

References:

- Bigelow, R.C.; Plumlee, G.S.; Edelmann, P. (1995) On-line update of US Geologic Survey OFR 95-23. www.infomine.com.
- Campbell, A.; Gobla, M.J. (2000) Proc. 5th Internat. Conf. on Acid Rock Drainage (ICARD), V2, 1243-1250.
- Ketallapper, V.L.; Christiansen, J.W. (1999) Conf. Proc. Mining and the Environment II, Sudbury, Ontario, September 13-17, V1, 199-21

Equity Silver Mine, British Columbia, Canada

Strategies:

- Water cover over tailings pond,
- Covered waste rock dumps: compacted clay & till cover,
- Southern Tail Pit = backfill with reactive rock beneath ground water table, covered with inert waste rock and till and various clay covers,
- ARD collection and acid neutralization treatment

References:

- Aziz, M.L.; Ferguson, K.D. Equity Silver Mine - Case Study. http://mend2000.nrcan.gc.ca. Contact: Keith D. Ferguson, Manager, Environmental Affairs, Placer Dome North America Ltd. 604-661-1916 or Keith Ferguson@placerdome.com.
- Aziz, M.L.; Ferguson, K.D. (1997) Proc. 4th Internat. Conf. on Acid Rock Drainage (ICARD), Vancouver, B.C., May 31-June 6,. V1, 181-195.
- Pedersen, T.F. et al. (1997) Proc. 4th Internat. Conf. on Acid Rock Drainage (ICARD), Vancouver, B.C., May 31-June 6, V3, 989-1005.
- Wilson, G.W. et al. (1997) Proc. 4th Internat. Conf. on Acid Rock Drainage (ICARD), Vancouver, B.C., May 31-June 6, V1, 197-210.