2.8 Geochemical Rock Characterization

This section summarizes the geochemical rock characterization program for the Minago Project. The program was led by URS and is consistent with widely accepted industrial standards. It occurred between April 2007 and November 2008 (URS, 2009i).

The objectives of the geochemical assessment were to (URS, 2009i):

- Assess major with respect to their Acid Rock Drainage (ARD) and Metal Leaching (ML) potential as waste rock and tailings material;
- Provide information for development of a waste management plan and application for mine development; and
- Determine whether subaqueous tailings storage will be sufficient to prevent ARD/ML from the tailings material.

The reaction of naturally-occurring metal sulphides (primarily iron sulphide) with oxygen and water can produce sulphuric acid or Acid Rock Drainage (ARD) over time. ARD is leachate drainage with a pH less than 4.5. The acidic drainage can dissolve metals in the sulphides and cause metal leaching (ML) by releasing metals to groundwater and/or surface water.

The geochemical program was conducted in two phases to characterize lithologic units that will be encountered, excavated and/or exposed during open pit mining, milling, and concentrating ore on-site by conventional flotation methods. The first phase consisted of static testing to determine the ARD/ML potential of all lithologic units (overburden, Ordovician dolomitic limestone, Ordovician sandstone, altered Precambrian basement, and Precambrian basement) and to design the second phase geochemical assessment program for the Minago site. The second phase involved the assessment of the multiple lithologies encountered within the Precambrian basement, including undifferentiated altered Precambrian basement, granitic rock material, Ultramafic rock that includes ore bearing materials, mafic metavolcanic rock materials, metasedimentary rock materials, and Molson Dike Swarm dikes and sills. The second phase geochemical assessment program consisted of static and kinetic testing and the determination of readily-soluble elements to identify elements that are of potential concern. The reaction rates of acid generating and acid consuming components were also determined (URS, 2009i).

Static testing involves subjecting test specimens to Acid-Base Accounting (ABA) tests (including fizz test, paste pH, inorganic carbonate content, total sulphur, sulphate sulphur, sulphide sulphur, and bulk Acid Neutralization Potential) and total metal content analysis.

In kinetic tests, humidity cell tests are used to simulate the oxidation reactions that would occur upon exposure of sulphidic materials to the environment. Kinetic tests are designed to verify the ARD and ML potential by enhancing and accelerating the rate of acid generation in sulphide-containing material so that results can be obtained in a timely manner to allow prediction of potential future impacts. Humidity cell tests tend to be better than static tests at evaluating the rate of acid production, the availability of acid neutralization, and resultant water quality over natural water pH ranges. Therefore, they are useful for determining whether materials with uncertain acid-generating status are likely to generate acid when exposed to oxidizing conditions.

2.8.1 Geochemical Assessment of Waste Rock

2.8.1.1 Sample Selection for Rock Types

In the Phase I geochemical assessment program for waste rock, a total of forty-nine (49) discrete and composite samples from four (4) drill holes (N-07-27, N-07-28, N-07-29 and N-07-36) at the Minago site were selected by Victory Nickel Inc. (VNI) in April and May 2007 and sent to SGS - Canadian Environmental and Metallurgical Inc., now owned by SGS Lakefield (SGS-CEMI), located in Burnaby, British Columbia, for geochemical analysis. Drillholes N-07-27, N-07-28 and N-07-29 were selected from locations near the ultimate pit outline encountering ultramafic rocks with little to no mineralization. Drillhole N-07-27 represented intersections of low-grade ore zones consisting of insufficient grade thickness, discontinuous lenses of mineralization or dilution of nickel grades due to granite intrusion. Drillholes N-07-28 and N-07-29 were representative of the southern and northwestern portions of the Minago deposit within the ultimate pit outline.

Selected drillhole samples consisted of discrete and composite samples representing the following five main lithologic units at the Minago nickel deposit (in reverse stratigraphic order):

- Overburden (OB);
- Dolomite (LS);
- Sandstone (FS);
- Alteration (AR); and
- Ore Zone (ORE).

In this report, Altered Rock (AR) is defined as the intensely weathered cap at the top of the Precambrian basement rocks, which includes granite and serpentinite. Ore Zone (ORE) is defined as all Precambrian rock types within the ultimate pit limits below alteration, which includes granite, serpentinite, mafic dikes, mafic metavolcanics and amphibolite. Details of the discrete and composite samples used for the Phase I geochemical testing of waste rock are presented in Appendix 2.8.

The Phase II geochemical assessment program of waste rock was conducted with fifty-three (53) drill core samples of Precambrian geologic rock types. These samples were subjected to Acid-Base Accounting (ABA) tests and total metal analysis.

The 53 samples were selected based on a review by URS of (URS, 2009i):

- 2004 borehole lithology logs;
- 2007 sample logs with assay results;
- 2007 borehole lithology logs and corresponding lithology codes (14 lithologies);
- 2007 sample logs with assay results;
- 2007 core photographs;
- estimates of waste rock types (tonnage and volume);
- geological cross-sections with borehole projections; and
- a plan view of the proposed pit outline with borehole locations.

URS selected a total of twenty-eight (28) samples from 2004 Black Hawk Mining (BHK) drill cores representing granite, serpentinite and amphibolite geologic units out of which twenty- one (21) samples were tested. URS also selected sixty four (64) Nuinsco Resources Limited (N) drill core samples, representing granite, serpentinite, amphibolite, metasediment, mafic metavolcanic, mafic dike, regolith geologic units out of which thirty-one (31) samples were tested. Table 2.8-1 provides a summary of the number of samples from each geologic unit of BHK and N drill core samples tested. Details of drill core samples selected and tested during the Phase II geochemical assessment program are given in Appendix 2.8.

Table 2.8-1 Rock Types selected for the Phase II Static Test Program

			# Samples		
Numeric Code	**Alphanum. Code**	**Description**	**BHK holes**	**N holes**	**SUBTOTAL**
1	OVB	Overburden			
2	PZD	Dolomite			
3	PZS	Sandstone			
4	SPT	Serpentinite	7	8	**15**
5	GT	Granite	13	15	**28**
6	AMP	Amphibolite	1	1	**2**
7	PYX	Pyroxenite			
8	PER	Peridotite			
9	SCH	Schist			
10	LC	Lost Core			
11	R	Regolith		1	**1**
12	MD	Mafic Dike		2	**2**
13	MSD	Metasediment		4	**4**
14	MMV	Mafic Metavolcanic		1	**1**
		TOTALS	**21**	**32**	**53**

Source: URS (2009i)

2.8.1.2 Static Testing Program for Waste Rock

Static testing for the Minago Project involved subjecting test specimens to Acid-Base Accounting (ABA) tests and total metal content analysis by inductively-coupled atomic emissions spectrometry (ICP-AES). The static tests were conducted by SGS - Canadian Environmental and Metallurgical Inc. (SGS-CEMI), located in Burnaby, British Columbia. The static testing included the following parameters:

- Fizz Test;
- Paste pH;
- Weight % CO_2, which was converted to Total Inorganic Carbonate (TIC) content expressed as $CaCO_3$ equivalents;
- Total Sulphur content, expressed as weight %;
- Sulphate Sulphur content, expressed as weight %;
- Sulphide Sulphur contents, expressed as weight % and determined from the difference between Total Sulphur and Sulphate Sulphur; and
- ANP by modified Sobek method (results are presented in calcium carbonate equivalent per tonne of rock [kg $CaCO_3$/tonne]).

From the analytical results the following ABA parameters were calculated:

- AGP was calculated from sulphide sulphur content;
- Net-ANP was calculated from the difference between modified Sobek method ANP and AGP calculated from the sulphide sulphur content; and
- NPR was calculated as the ratio of the modified Sobek ANP to AGP.

The criteria used in this study to determine whether sampled materials from the Minago Project are non-acid generating (NAG) are as follows (URS, 2009i):

- If the NPR (the ratio of ANP to AGP) is greater than 4.0, the sample material is considered to be NAG; and
- If the NPR is <1.0, the sample material is considered to be PAG.

2.8.1.2.1 Phase I Acid-Base Accounting (ABA) Results

Results of Phase I Acid-Base Accounting (ABA) test results are presented in Table 2.8-2 and detailed static test results are given in Appendix 2.8. Static test results in Table 2.8-2 are listed with minimum, average and maximum values for each lithology. In addition, minimum, average, and maximum values for all Phase I samples are summarized at the bottom of Table 2.8-2. The results of static tests indicate a natural variability in the geochemical characteristics of lithologic materials that will be encountered during open pit mining of the Minago nickel deposit.

Table 2.8-2 Phase I ABA Test Results for Waste Rock

Sample #	Rock Type	Composite Ratio	Paste pH	Fizz Test	CO_2 (wt%)	$CaCO_3$ Equivalent (kg $CaCO_3$/tonne)	Total Sulphur (wt%)	Sulphate Sulphur (wt%)	Sulphide Sulphur (wt%)*	Maximum Potential Acidity** (kg $CaCO_3$/tonne)	Neutralization Potential (kg $CaCO_3$/tonne)	Net Neutralization Potential (kg $CaCO_3$/tonne)	NPR (NP/MPA)
#8-N-07-27-OB	1	1	7.9	Strong	2.46	205.0	0.04	<0.01	0.03	0.9	198.6	197.7	212
#8-N-07-28-OB	1	1	*7.9*	Strong	2.46	205.0	0.03	<0.01	0.02	0.6	193.6	193.0	310
#8-N-07-29-OB	1	1	8.0	Strong	2.65	220.8	0.04	<0.01	0.03	0.9	229.8	228.9	245
#8-N-07-36-OB	1	1	7.7	Moderate	2.40	200.0	0.12	<0.02	0.10	3.1	194.8	191.7	62.3
		Minimum	7.7		2.40	200.0	0.03		0.02	0.6	193.6	191.7	62.3
		Average	**7.9**		**2.49**	**207.7**	**0.06**	**<0.02**	**0.05**	**1.4**	**204.2**	**202.8**	**207.3**
		Maximum	8.0		2.65	220.8	0.12		0.10	3.1	229.8	228.9	309.8
#9-N-07-27-LS	2	1	8.8	Strong	12.02	1001.7	0.10	<0.01	0.09	2.8	831.7	828.9	296
#9-N-07-28-LS	2	1	8.7	Strong	12.50	1041.7	0.04	<0.01	0.03	0.9	665.7	664.8	710
#9-N-07-29-LS	2	1	8.8	Strong	12.25	1020.8	0.03	<0.01	0.02	0.6	738.1	737.5	1181
#9-N-07-36-LS	2	1	8.9	Moderate	11.90	991.7	0.09	<0.01	0.08	2.5	970.5	968.0	388.2
		Minimum	8.7		11.90	991.7	0.03		0.02	0.6	665.7	664.8	295.7
		Average	**8.8**		**12.17**	**1014.0**	**0.07**	**<0.01**	**0.06**	**1.7**	**801.5**	**799.8**	**643.7**
		Maximum	8.9		12.50	1041.7	0.10		0.09	2.8	970.5	968.0	1181.0
#10-N-07-27-FS	3	1	9.2	Moderate	1.69	140.8	0.13	<0.01	0.12	3.8	141.4	137.7	37.7
#10-N-07-28-FS	3	1	8.9	Moderate	1.04	86.7	0.02	<0.01	0.01	0.3	83.1	82.8	266
#10-N-07-29-FS	3	1	9.0	Moderate	1.52	126.7	0.19	<0.01	0.18	5.6	122.2	116.6	21.7
#10-N-07-36-FS	3	1	8.9	Slight	0.92	76.7	0.22	<0.01	0.21	6.6	68.3	61.7	10.4
		Minimum	8.9		0.92	76.7	0.02		0.01	0.3	68.3	61.7	10.4
		Average	**9.0**		**1.29**	**107.7**	**0.14**	**<0.01**	**0.13**	**4.1**	**103.8**	**99.7**	**83.9**
		Maximum	9.2		1.69	140.8	0.22		0.21	6.6	141.4	137.7	265.9
#11-N-07-27-AR	5,4	1	8.1	None	0.11	9.2	0.34	0.01	0.33	10.3	10.2	-0.1	1.0
#11-N-07-28-AR	4	1	8.0	None	0.34	28.3	0.69	0.04	0.65	20.3	30.8	10.5	1.5
#11-N-07-29-AR	11,5	1	8.6	Slight	1.04	86.7	0.14	<0.01	0.13	4.1	86.3	82.2	21.2
#11-N-07-36-AR	?	1	9.6	None	0.08	6.7	0.19	<0.01	0.18	5.6	17.9	12.3	3.2
		Minimum	8.0		0.08	6.7	0.14	0.01	0.13	4.1	10.2	-0.1	1.0
		Average	**8.6**		**0.39**	**32.7**	**0.34**		**0.32**	**10.1**	**36.3**	**26.2**	**6.7**
		Maximum	9.6		1.04	86.7	0.69	0.04	0.65	20.3	86.3	82.2	21.2
#12-N-07-36-ORE	**?**	**1**	**9.1**	**None**	**0.20**	**16.7**	**4.12**	**<0.03**	**4.09**	**127.8**	**37.9**	**-89.9**	**0.3**
#1-N-07-27-OB/AR	1-4,5	1:1.7	8.1	Moderate	0.91	75.8	0.16	0.01	0.15	4.7	55.1	50.4	11.8
#1-N-07-28-OB/AR	1-4	25:1	8.1	Strong	2.36	196.7	0.04	<0.01	0.03	0.9	196.0	195.1	209
#1-N-07-29-OB/AR	1-5,11	27.5:1	7.9	Strong	2.52	210.0	0.06	<0.01	0.05	1.6	209.5	207.9	134
#1-N-07-36-OB/AR	1-?	0.03:1	9.2	None	0.13	10.8	0.16	<0.01	0.15	4.7	28.4	23.7	6.1
		Minimum	7.9		0.13	10.8	0.04	<0.01	0.03	0.9	28.4	23.7	6.1
		Average	**8.3**		**1.48**	**123.3**	**0.11**		**0.10**	**3.0**	**122.3**	**119.3**	**90.2**
		Maximum	9.2		2.52	210.0	0.16	0.01	0.15	4.7	209.5	207.9	209.1
#2-N-07-27-FS/AR	3-4,5	1:1.25	8.9	Slight	1.09	90.8	0.08	<0.01	0.07	2.2	99.4	97.2	45.4
#2-N-07-28-FS/AR	3-4	25:1	9.1	Slight	1.21	100.8	0.06	<0.01	0.05	1.6	95.8	94.2	61.3
#2-N-07-29-FS/AR	3-5,11	1:2	8.8	Slight	0.71	59.2	0.12	<0.01	0.11	3.4	48.8	45.4	14.2
#2-N-07-36-FS/AR	3-?	0.05:1	9.0	None	0.16	13.3	0.18	<0.01	0.17	5.3	20.5	15.2	3.9
		Minimum	8.8		0.16	13.3	0.06		0.05	1.6	20.5	15.2	3.9
		Average	**8.9**		**0.79**	**66.0**	**0.11**	**<0.01**	**0.10**	**3.1**	**66.1**	**63.0**	**31.2**
		Maximum	9.1		1.21	100.8	0.18		0.17	5.3	99.4	97.2	61.3

Table 2.8-2 (Cont.'d) Phase I ABA Test Results

Sample #	**Rock Type**	**Composite Ratio**	**Paste pH**	**Fizz Test**	**CO_2 (wt%)**	**$CaCO_3$ Equivalent (kg $CaCO_3$/tonne)**	**Total Sulphur (wt%)**	**Sulphate Sulphur (wt%)**	**Sulphide Sulphur (wt%)***	**Maximum Potential Acidity** (kg $CaCO_3$/tonne)**	**Neutralization Potential (kg $CaCO_3$/tonne)**	**Net Neutralization Potential (kg $CaCO_3$/tonne)**	**NPR (NP/MPA)**
#3-N-07-27-FS/LS	3-2	1:7	8.9	Strong	10.49	874.2	0.08	<0.01	0.07	2.2	824.1	821.9	377
#3-N-07-28-FS/LS	3-2	1:7	8.8	Strong	10.77	897.5	0.04	<0.01	0.03	0.9	837.2	836.3	893
#3-N-07-29-FS/LS	3-2	1:7	8.8	Strong	10.63	885.8	0.07	<0.01	0.06	1.9	854.3	852.4	456
#3-N-07-36-FS/LS	3-2	0.15:1	8.9	Moderate	11.70	975.0	0.09	<0.01	0.08	2.5	964.2	961.7	385.7
		Minimum	8.8		10.49	874.2	0.04		0.03	0.9	824.1	821.9	376.7
		Average	**8.8**		**10.90**	**908.1**	**0.07**	**<0.01**	**0.06**	**1.9**	**870.0**	**868.1**	**527.8**
		Maximum	8.9		11.70	975.0	0.09		0.08	2.5	964.2	961.7	893.0
#4-N-07-27-LS/OB	2-1	10:1	8.3	Strong	11.09	924.2	0.07	<0.01	0.06	1.9	903.3	901.4	482
#4-N-07-28-LS/OB	2-1	7:1	8.2	Strong	10.68	890.0	0.04	<0.01	0.03	0.9	864.3	863.4	922
#4-N-07-29-LS/OB	2-1	5.5:1	8.1	Strong	8.06	11.16	0.05	<0.01	0.04	1.3	917.5	916.3	734
#4-N-07-36-LS/OB	2-1	1.0:0.08	8.3	Moderate	11.70	975.0	0.06	<0.01	0.05	1.6	954.8	953.2	611.1
		Minimum	8.1		8.06	11.2	0.04		0.03	0.9	864.3	863.4	481.8
		Average	**8.2**		**10.38**	**700.1**	**0.06**	**<0.01**	**0.05**	**1.4**	**910.0**	**908.6**	**687.2**
		Maximum	8.3		11.70	975.0	0.07		0.06	1.9	954.8	953.2	921.9
#5-N-07-27-ORE/AR	4,5-4,5	11.6:1	9.7	None	0.29	24.2	0.30	0.02	0.28	8.8	59.0	50.3	6.7
#5-N-07-28-ORE/AR	4,5,7,9-4	66:1	9.7	Slight	0.52	43.3	0.08	<0.01	0.07	2.2	41.4	39.2	18.9
#5-N-07-29-ORE/AR	5,6-5,11	12:1	*9.4*	Slight	0.22	18.3	0.12	<0.01	0.11	3.4	40.7	37.3	11.8
#5-N-07-36-ORE/AR	?-?	0.26:1	9.2	None	0.10	8.3	0.33	<0.01	0.32	10.0	20.3	10.3	2.0
		Minimum	9.2		0.10	8.3	0.08	<0.01	0.07	2.2	20.3	10.3	2.0
		Average	**9.5**		**0.28**	**23.5**	**0.21**		**0.20**	**6.1**	**40.4**	**34.3**	**9.9**
		Maximum	9.7		0.52	43.3	0.33	0.02	0.32	10.0	59.0	50.3	18.9
#6-N-07-27-LS/AR	2-4,5	5.6:1	8.7	Strong	10.69	890.8	0.10	<0.01	0.09	2.8	919.0	916.2	327
#6-N-07-28-LS/AR	2-4	18:1	8.5	Strong	12.06	1005.0	0.02	<0.01	0.01	0.3	967.3	967.0	3095
#6-N-07-29-LS/AR	2-5,11	3.6:1	8.5	Strong	9.83	819.2	0.09	<0.01	0.08	2.5	809.7	807.2	324
#6-N-07-36-LS/AR	2-?	0.35:1	9.3	Slight	2.95	245.8	0.12	<0.01	0.11	3.4	231.6	228.2	67.4
		Minimum	8.5		2.95	245.8	0.02		0.01	0.3	231.6	228.2	67.4
		Average	**8.7**		**8.88**	**740.2**	**0.08**	**<0.01**	**0.07**	**2.3**	**731.9**	**729.6**	**953.3**
		Maximum	9.3		12.06	1005.0	0.12		0.11	3.4	967.3	967.0	3095.4
#7-N-07-27-ORE/LS	4,5-2	2.1:1	9.6	Strong	5.25	437.5	0.23	<0.01	0.22	6.9	412.9	406.0	60.1
#7-N-07-28-ORE/LS	4,5,6,7,9-2	3.7:1	9.6	Moderate	3.13	260.8	0.08	<0.01	0.07	2.2	245.1	242.9	112
#7-N-07-29-ORE/LS	5,6-2	3.3:1	9.7	Moderate	2.25	187.5	0.08	<0.01	0.07	2.2	185.5	183.3	84.8
#7-N-07-36-ORE/LS	?-2	0.75:1	8.8	Moderate	7.68	640.0	1.24	<0.01	1.23	38.4	648.9	610.5	16.9
		Minimum	8.8		2.25	187.5	0.08		0.07	2.2	185.5	183.3	16.9
		Average	**9.4**		**4.58**	**381.5**	**0.41**	**<0.01**	**0.40**	**12.4**	**373.1**	**360.7**	**68.4**
		Maximum	9.7		7.68	640.0	1.24		1.23	38.4	648.9	610.5	112.0
#13-N-07-27-OB/LS/FS/AR/ORE	1-2-3-4,5-4,5	0.05:0.49:0.07:0.08:1	9.7	Moderate	3.29	274.2	0.24	0.01	0.23	7.2	231.7	224.5	32.2
#13-N-07-28-OB/LS/FS/AR/ORE	1-2-3-4-4,5,6,7,9	0.03:0.19:0.03:0.01:1	9.7	Moderate	2.18	181.7	0.07	<0.01	0.06	1.9	122.8	120.9	65.5
#13-N-07-29-OB/LS/FS/AR/ORE	1-2-3-5,11-5,6	0.05:0.3:0.04:0.08:1	9.3	Moderate	2.14	178.3	0.13	<0.01	0.12	3.8	179.8	176.1	47.9
#13-N-07-36-OB/LS/FS/AR/ORE	1-2-3-?-?	0.03:0.35:0.05:1:0.26	9.0	Moderate	2.49	207.5	0.62	<0.01	0.61	19.1	200.4	181.3	10.5
		Minimum	9.0		2.14	178.3	0.07	<0.01	0.06	1.9	122.8	120.9	10.5
		Average	**9.4**		**2.53**	**210.4**	**0.27**		**0.26**	**8.0**	**183.7**	**175.7**	**39.0**
		Maximum	9.7		3.29	274.2	0.62	0.01	0.61	19.1	231.7	224.5	65.5
All samples		**Minimum**	7.7		0.1	6.7	0.02	<0.01	0.01	0.3	10.2	-89.9	0.3
All samples		**Average**	**8.8**		**4.6**	**368.9**	**0.24**	**0.01**	**0.23**	**7.1**	**363.5**	**356.3**	**273.4**
All samples		**Maximum**	9.7		12.5	1,041.7	4.12	0.04	4.09	127.8	970.5	968.0	3,095.4
Detection Limits			**0.1**		**0.03**	**---**	**0.02**	**0.01**	**---**	**---**	**0.1**	**0.1**	**---**

Notes:

*** Based on difference between total sulphur and sulphate-sulphur**

**** Based on sulphide-sulphur**

MPA = Maximum Potential Acidity in tonnes $CaCO_3$ equivalent per 1000 tonnes of material.

NP = Bulk Neutralization Potential in tonnes $CaCO_3$ equivalent per 1000 tonnes of material.

NPR = NP / MPA

Lithologies: OB=overburden, LS=dolomite, FS=sandstone, AR=altered Precambrian basement, ORE=Precambrian basement

Rock Types: 1=glacial lacustrine clay, 2=dolomite, 3=sandstone, 4=serpentinite, 5=granite, 6=amphibolite, 7=mafic dike, 9=mafic metavolcanic, 11=regolith

If the concentation was below the dectecion limit, half the detection limit was used to calculate the average.

Source: adapted from URS (2009i)

All samples analyzed in Phase I had alkaline pH values ranging from 7.7 to 9.7 and low sulphate concentrations ranging from <0.01 % to 0.04%. Low sulphate sulphur were expected as drillcores were fresh and the deposit is located at depth where oxygen concentrations are limited. All other Phase I acid-base accounting results varied widely (Table 2.8-2).

To determine whether the tested discrete and composite lithologies are potentially acid generating, the ARD/ML screening criteria of sulphide sulphur greater than 0.3 weight % and a Neutralization Potential Ratio (NPR) of less than 4 were applied to the static geochemical test results. Table 2.8-3 lists the eight Phase I samples that exceeded one or both of these screening criteria in ascending order of the NPR. ARD/ML screening criteria were exceeded by samples containing ore (ORE) and altered rock (AR). The lowest NPR (0.3) and highest sulphide content (4.09%) was measured for the one ORE sample tested. Therefore, ore (ORE) is Potentially Acid Generating (PAG) and AR has an uncertain acid-generating status with NPR values ranging between 1 and 4.

The relationship between Acid Generation Potential based on sulphide concentrations and the modified Sobek bulk Acid Neutralization Potential is shown in Figures 2.8-1 and 2.8-2. Figure 2.8-1 shows the relationship between these parameters for all discrete and composite lithologies tested. Figure 2.8-2 illustrates the relationship between these parameters for altered and Ore Zone Precambrian basement lithologies and composites with overburden and sandstone. Figures 2.8-1 and 2.8-2 serve to illustrate that discrete samples from overburden, sandstone and limestone were non-acid generating as they contained low sulphide sulphur (<0.3 weight %) and low to high carbonate concentrations. Similarly, composites containing combinations of overburden, sandstone and limestone were also non-acid generating. Altered and Ore Zone Precambrian basement lithology discrete samples were likely potentially acid generating or potentially acid generating (PAG). Composite samples containing altered and Ore Zone Precambrian basement lithologies were potentially non-acid generating (PNAG) (URS, 2009i).

2.8.1.2.2 Phase II Acid-Base Accounting (ABA) Results

Results of Phase II Acid-Base Accounting (ABA) test results for waste rock are presented in Table 2.8-4. Table 2.8-4 also lists minimum, average and maximum values for each lithology. Figures 2.8-3 and 2.8-4 illustrate the relationship between Acid Generation Potential based on sulphide concentrations and the modified Sobek bulk Acid Neutralization Potential. Detailed static test results are given in Appendix 2.8.

The results of static tests indicate a natural variability in the geochemical characteristics of lithologic materials that will be encountered during open pit mining of the Minago nickel deposit.

All samples analyzed in Phase II had alkaline pH values ranging from 7.1 to 9.7. Measured sulphate concentrations were almost all <0.01% with the exception of one serpentinite sample (BHK-41-R1-90) taken from 1994 Black Hawk Mining drill core, which had a sulphate sulphur concentration of 0.22%. This value potentially represents oxidation of sulphidic material in that

Table 2.8-3 Phase I Waste Rock Static Samples Exceeding ARD/ML Screening Criteria

Sample #	Rock Type	Composite Ratio	Paste pH	Fizz Test	CO_2 (wt%)	$CaCO_3$ Equivalent (kg $CaCO_3$/tonne)	Total Sulphur (wt%)	Sulphate Sulphur (wt%)	Sulphide Sulphur (wt%)*	Maximum Potential Acidity** (kg $CaCO_3$/tonne)	Neutralization Potential (kg $CaCO_3$/tonne)	Net Neutralization Potential (kg $CaCO_3$/tonne)	NPR (NP/MPA)
#12 N-07-36 ORE	?	1	9.1	None	0.20	16.7	4.12	<0.03	**4.09**	127.8	37.9	-89.9	**0.3**
#11 N-07-27 AR	5,4	1	8.1	None	0.11	9.2	0.34	0.01	**0.33**	10.3	10.2	-0.1	**1.0**
#11 N-07-28 AR	4	1	8.0	None	0.34	28.3	0.69	0.04	**0.65**	20.3	30.8	10.5	**1.5**
#5 N-07-36 ORE/AR	?-?	0.26:1	9.2	None	0.10	8.3	0.33	<0.01	**0.32**	10.0	20.3	10.3	**2.0**
#11 N-07-36 AR	?	1	9.6	None	0.08	6.7	0.19	<0.01	0.18	5.6	17.9	12.3	**3.2**
#2 N-07-36 FS/AR	3-?	0.05:1	9.0	None	0.16	13.3	0.18	<0.01	0.17	5.3	20.5	15.2	**3.9**
#13 N-07-36 OB/LS/FS/AR/ORE	1-2-3-?-?	0.03:0.35:0.05:1:0.26	9.0	Moderate	2.49	207.5	0.62	<0.01	**0.61**	19.1	200.4	181.3	10.5
#7 N-07-36 ORE/LS	?-2	0.75:1	8.8	Moderate	7.68	640.0	1.24	<0.01	**1.23**	38.4	648.9	610.5	16.9

Notes:

* Based on difference between total sulphur and sulphate-sulphur

** Based on sulphide-sulphur

MPA = Maximum Potential Acidity in tonnes $CaCO_3$ equivalent per 1000 tonnes of material.

NP = Bulk Neutralization Potential in tonnes $CaCO_3$ equivalent per 1000 tonnes of material.

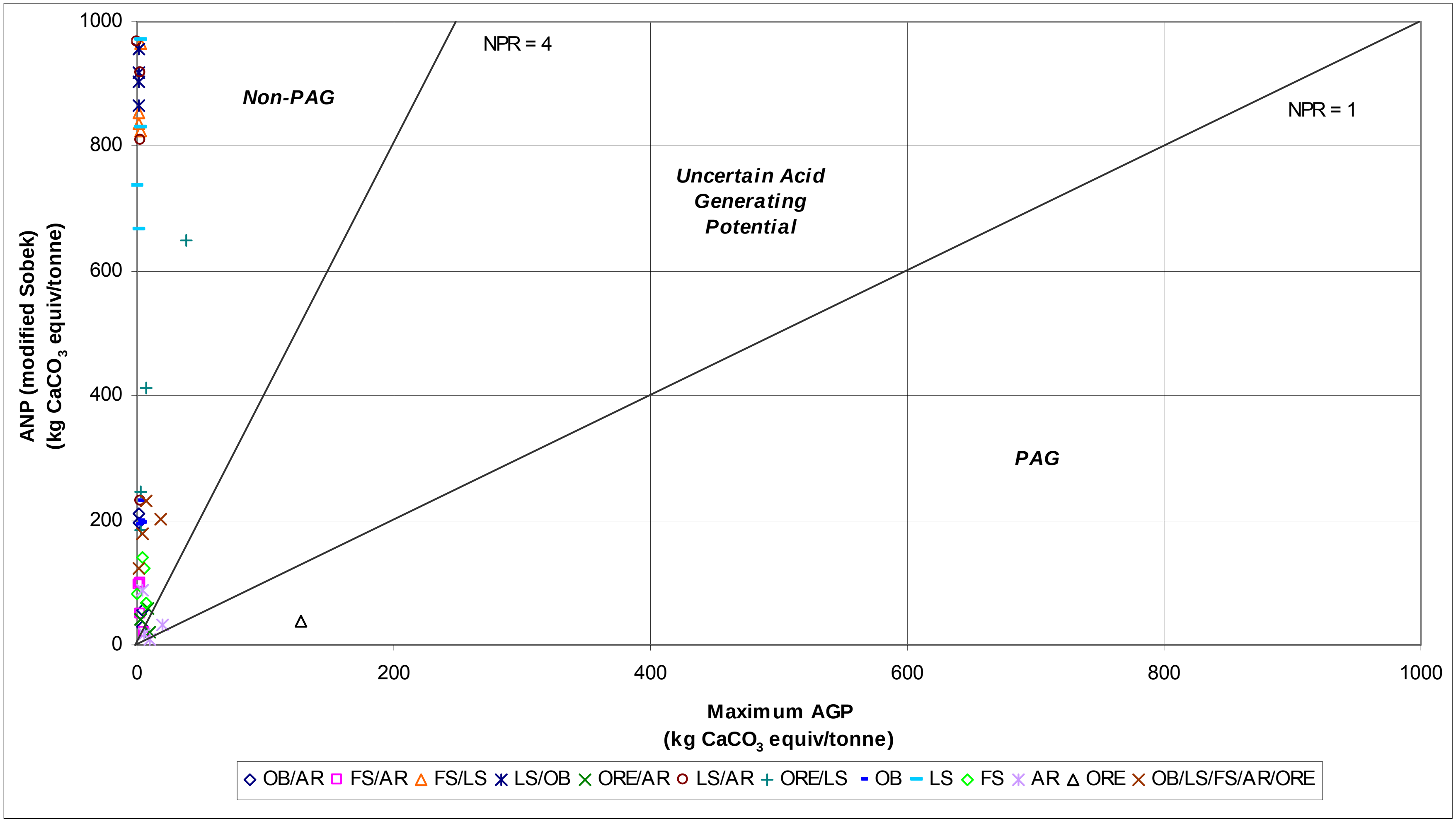
NPR = NP / MPA

1.23 Results highlighted in red and bold and that are underlined exceed the ARD/ML screening criteria (sulphide sulphur > 0.3% and NPR < 4).

Lithologies: OB=overburden, FS=sandstone, AR=altered Precambrian basement, ORE=Precambrian basement

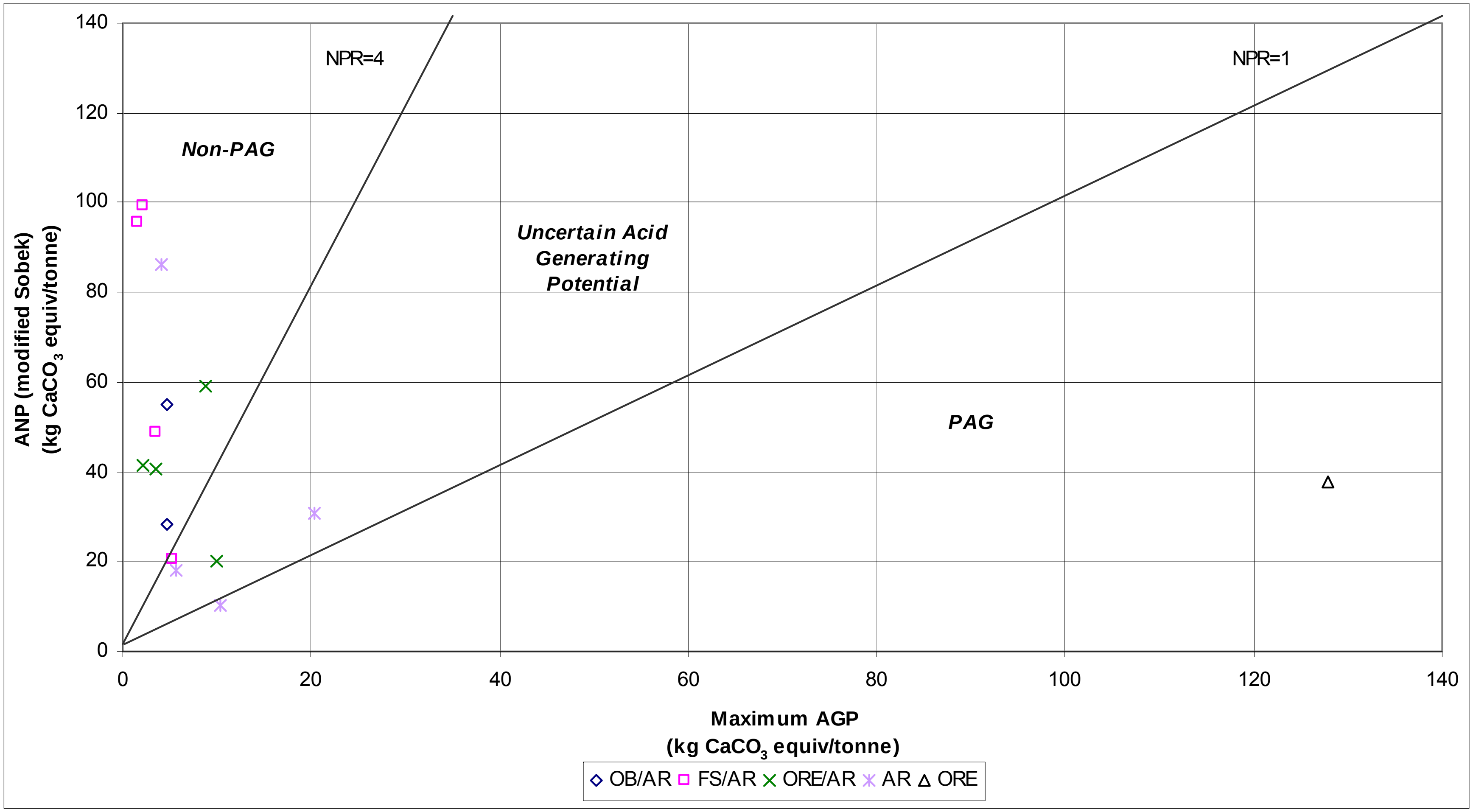
Rock Types: 1 = glacial lacustrine clay, 2 = dolomite, 3 = sandstone, 4 = serpentinite, 5 = granite

Source: adapted from URS (2009i)



Source: URS (2009i)

Figure 2.8-1 Phase I Static Test Results - ANP versus AGP in Major Lithologies



Source: URS (2009i)

Figure 2.8-2 Phase I Static Test Results - ANP versus AGP in Major Lithologies (Detail)

Table 2.8-4 Phase II ABA Test Results for Waste Rock

Sample #	Rock Type	Rock Code	Drill Hole #	From (ft)	To (ft)	Length (ft)	paste pH	Fizz Test	Total Inorganic Carbon (wt%)	CaCO$_3$Equivalent (kg CaCO$_3$/tonne)	Total Sulphur (wt%)	Sulphate Sulphur (wt%)	Sulphide Sulphur (wt%)*	Acid Generation Potential** (kg CaCO$_3$/tonne)	Acid Neutralization Potential (kg CaCO$_3$/tonne)	Net Neutralization Potential (kg CaCO$_3$/tonne)	NPR (ANP/AGP)
49809	Amphibolite	AMP	BHK 52-90	660	665	5.00	9.4	none			0.09		0.09	2.8	28.7	25.9	10.2
929318	Amphibolite	AMP	N0724	196.15	197	0.85	9.7	none			0.12		0.12	3.8	19.0	15.2	5.1
	Minimum						**9.4**				**0.09**		**0.09**	**2.8**	**19.0**	**15.2**	**5.1**
	Maximum						**9.7**				**0.12**		**0.12**	**3.8**	**28.7**	**25.9**	**10.2**
925273	Granite	GT	N0720	249.3	251	1.70	9.7	none	0.07	5.8	0.04	<0.01	0.03	0.9	10.9	10.0	11.7
929488	Granite	GT	N0730	197.7	198.8	1.10	9.2	none	0.03	2.5	0.04	<0.01	0.03	0.9	72.2	71.3	77.1
365663	Granite	GT	BHK 41-R1-90	1080	1083	3.00	9.3	none			0.04		0.04	1.3	47.2	46.0	37.8
49092	Granite	GT	BHK 42-90	767	774	7.00	8.8	none			0.03		0.03	0.9	11.1	10.2	11.9
49125	Granite	GT	BHK 42-90	962	967	5.00	9.4	none			0.02		0.02	0.6	19.2	18.6	30.8
365507	Granite	GT	BHK 42-R1-90	1089	1094	5.00	9.5	none			0.05		0.05	1.6	28.3	26.7	18.1
365525	Granite	GT	BHK 42-R2-90	1156	1172	16.00	9.4	none			0.05		0.05	1.6	17.9	16.3	11.4
49427	Granite	GT	BHK 43-90	927	937	10.00	9.4	none			0.04		0.04	1.3	14.2	12.9	11.4
258551	Granite	GT	BHK 49-R9-90	463.75	467.5	3.75	8.9	none			0.07		0.07	2.2	20.3	18.1	9.3
258637	Granite	GT	BHK 49-R9-90	1015.5	1017	1.50	9.0	none			0.04		0.04	1.3	87.2	85.9	69.7
258779	Granite	GT	BHK 49-R9-90	833.2	835	1.80	8.0	none			0.14		0.14	4.4	28.0	23.6	6.4
49832	Granite	GT	BHK 52-90	817	820	3.00	9.2	none			0.02		0.02	0.6	10.4	9.8	16.7
49842	Granite	GT	BHK 52-90	856.5	865	8.50	9.3	none			0.02		0.02	0.6	11.8	11.2	18.9
49843	Granite	GT	BHK 52-90	865	870	5.00	9.3	none			0.02		0.02	0.6	36.4	35.7	58.2
49907	Granite	GT	BHK 52-90	1126	1133	7.00	9.3	none			0.03		0.03	0.9	15.1	14.2	16.1
924235	Granite	GT	N0702	162.4	163.7	1.30	9.0	none			0.02		0.02	0.6	66.0	65.3	105.5
924558	Granite	GT	N0705	107.35	108.35	1.00	8.4	none			0.16		0.16	5.0	62.9	57.9	12.6
924350	**Granite**	**GT**	**N0706**	**102.5**	**103.25**	**0.75**	**8.8**	**none**			**0.13**		**0.13**	**4.1**	**44.6**	**40.5**	**11.0**
924424	Granite	GT	N0706	178.3	179.7	1.40	8.3	none			0.39		0.39	12.2	9.7	-2.5	0.8
924591	Granite	GT	N0707	109.3	110.6	1.30	9.3	none			0.07		0.07	2.2	29.6	27.5	13.6
924890	Granite	GT	N0710	312.3	312.9	0.60	9.3	none			0.05		0.05	1.6	58.8	57.2	37.6
924964	Granite	GT	N0713	203.2	204.7	1.50	9.5	none			0.05		0.05	1.6	22.6	21.1	14.5
925132	Granite	GT	N0713	436.3	437.27	0.97	9.7	none			0.03		0.03	0.9	10.7	9.7	11.4
929308	Granite	GT	N0724	171.18	171.96	0.78	9.3	slight			0.16		0.16	5.0	48.4	43.4	9.7
925315	Granite	GT	N0725	138.64	139.56	0.92	9.0	none			0.06		0.06	1.9	20.7	18.9	11.1
926297	Granite	GT	N0726	215.81	218	2.19	9.0	none			0.24		0.24	7.5	28.0	20.5	3.7
929497	Granite	GT	N0733	164	165	1.00	9.2	none			0.02		0.02	0.6	47.4	46.7	75.8
258554	Granite	GT	258554	485.00	495.50	10.50	9.0	none			0.03		0.03	0.9	13.0	12.1	13.9
	Minimum						**7.99**				**0.02**		**0.02**	**0.63**	**9.7**	**-2.5**	**0.8**
	Average						**9.10**		**0.05**	**4.17**	**0.07**		**0.07**	**2.28**	**31.9**	**29.6**	**25.9**
	Maximum						**9.72**				**0.39**		**0.39**	**12.19**	**87.2**	**85.9**	**105.5**
924551	Mafic Dike	MD	N0705	101.6	102.6	1.00	8.3	none			0.20		0.20	6.3	28.2		
929437	Mafic Dike	MD	N0730	141	143	2.00	9.4	none			0.05		0.05	1.6	53.9		
	Minimum						**8.30**				**0.05**		**0.05**	**1.6**	**28.2**		
	Maximum						**9.40**				**0.20**		**0.20**	**6.3**	**53.9**		

Source: adapted from URS (2009i)

Table 2.8-4 (Cont.'d) Phase II ABA Test Results for Waste Rock

Sample #	Rock Type	Rock Code	Drill Hole #	From (ft)	To (ft)	Length (ft)	paste pH	Fizz Test	Total Inorganic Carbon (wt%)	$CaCO_3$ Equivalent (kg $CaCO_3$/tonne)	Total Sulphur (wt%)	Sulphate Sulphur (wt%)	Sulphide Sulphur (wt%)*	Acid Generation Potential** (kg $CaCO_3$/tonne)	Acid Neutralization Potential (kg $CaCO_3$/tonne)	Net Neutralization Potential (kg $CaCO_3$/tonne)	NPR (ANP/AGP)
925884	**Mafic Metavolcanic**	**MMV**	**N0712**	**249.5**	**251**	**1.50**	**9.3**	**none**			**0.46**		**0.46**	**14.4**	**21.0**	**6.6**	**1.5**
924159	Metasediment	MSD	N0701	173.45	174.9	1.45	8.6	none			0.17		0.17	5.3	28.3	23.0	5.3
924548	Metasediment	MSD	N0705	99.5	100.8	1.30	7.9	none			0.17		0.17	5.3	6.8	1.5	1.29
924738	Metasediment	MSD	N0710	135.4	136.2	0.80	7.7	none			5.12		5.12	160.0	9.0	-151.0	0.06
925841	Metasediment	MSD	N0712	197	198.5	1.50	8.4	none			0.37		0.37	11.6	89.3	77.8	7.7
	Minimum										**0.17**		**0.17**	**5.3**	**6.8**	**-151.0**	**0.06**
	Average										**1.46**		**1.46**	**45.5**	**33.4**	**-12.2**	**3.60**
	Maximum										**5.12**		**5.12**	**160.0**	**89.3**	**77.8**	**7.73**
926397	**Altered Rock**	**AR**	**N0730**	**94.53**	**95.23**	**0.70**	**9.0**	**moderate**			**0.16**		**0.16**	**5.0**	**549.1**	**544.1**	**109.8**
365627	Serpentinite	SPT	BHK 41-R1-90	1051.5	1056.2	4.70	7.2	none	0.06	5.0	0.80	0.22	0.58	18.1	54.8	36.7	3.0
258612	Serpentinite	SPT	BHK 49-R9-90	746.5	749.9	3.40	8.6	none	1.09	90.8	0.03	<0.01	0.02	0.6	151.1	150.4	241.7
924724	Serpentinite	SPT	N0707	257	258.5	1.50	9.0	slight	3.57	297.5	0.10	<0.01	0.09	2.8	272.4	269.6	96.9
258774	Serpentinite	SPT	BHK 49-R9-90	818.5	821.5	3.00	7.1	none			0.74		0.74	23.1	86.3	63.2	3.7
49816	Serpentinite	SPT	BHK 52-90	744	749	5.00	8.9	none			0.02		0.02	0.6	167.7	167.1	268.3
49828	Serpentinite	SPT	BHK 52-90	801	806	5.00	9.0	slight			0.03		0.03	0.9	154.9	154.0	165.2
49830	Serpentinite	SPT	BHK 52-90	811	815	4.00	9.0	none			0.04		0.04	1.3	153.0	151.8	122.4
49904	Serpentinite	SPT	BHK 52-90	1110	1115.5	5.50	8.8	none			0.06		0.06	1.9	33.4	31.5	17.8
924686	Serpentinite	SPT	N0707	233	243.2	10.20	8.2	none			0.19		0.19	5.9	133.9	127.9	22.5
925856	Serpentinite	SPT	N0712	214.5	215.45	0.95	9.2	none			0.07		0.07	2.2	48.9	46.7	22.3
925017	Serpentinite	SPT	N0713	371.04	371.7	0.66	8.8	none			0.30		0.30	9.4	81.0	71.7	8.6
925276	Serpentinite	SPT	N0720	254.84	256.33	1.49	9.2	none			0.11		0.11	3.4	100.6	97.2	29.3
926243	Serpentinite	SPT	N0726	133.69	134.54	0.85	9.3	none			0.38		0.38	11.9	94.0	82.1	7.9
929407	Serpentinite	SPT	N0730	104.25	105.25	1.00	9.0	none			0.04		0.04	1.3	61.9	60.7	49.5
	Minimum								**0.06**	**5.0**	**0.02**		**0.02**	**0.63**	**33.4**	**31.5**	**3.0**
	Average								**1.57**	**131.1**	**0.21**		**0.19**	**5.96**	**113.9**	**107.9**	**75.7**
	Maximum								**3.57**	**297.5**	**0.80**		**0.74**	**23.13**	**272.4**	**269.6**	**268.3**
924234	**Serp/Gran**	**SPT/GT**	**N0702**	**161.6**	**162.4**	**0.80**	**9.2**	**none**	**<0.01**	**<0.8**	**0.03**	**<0.01**	**0.02**	**0.6**	**56.9**	**56.3**	**91.1**
Detection Limits											0.02	0.01	---	---	0.1	0.1	---

Notes:

* Based on difference between total sulphur and sulphate-sulphur

** Based on sulphide-sulphur

AGP = Maximum Potential Acidity in kilograms $CaCO_3$ equivalent per tonne of material.

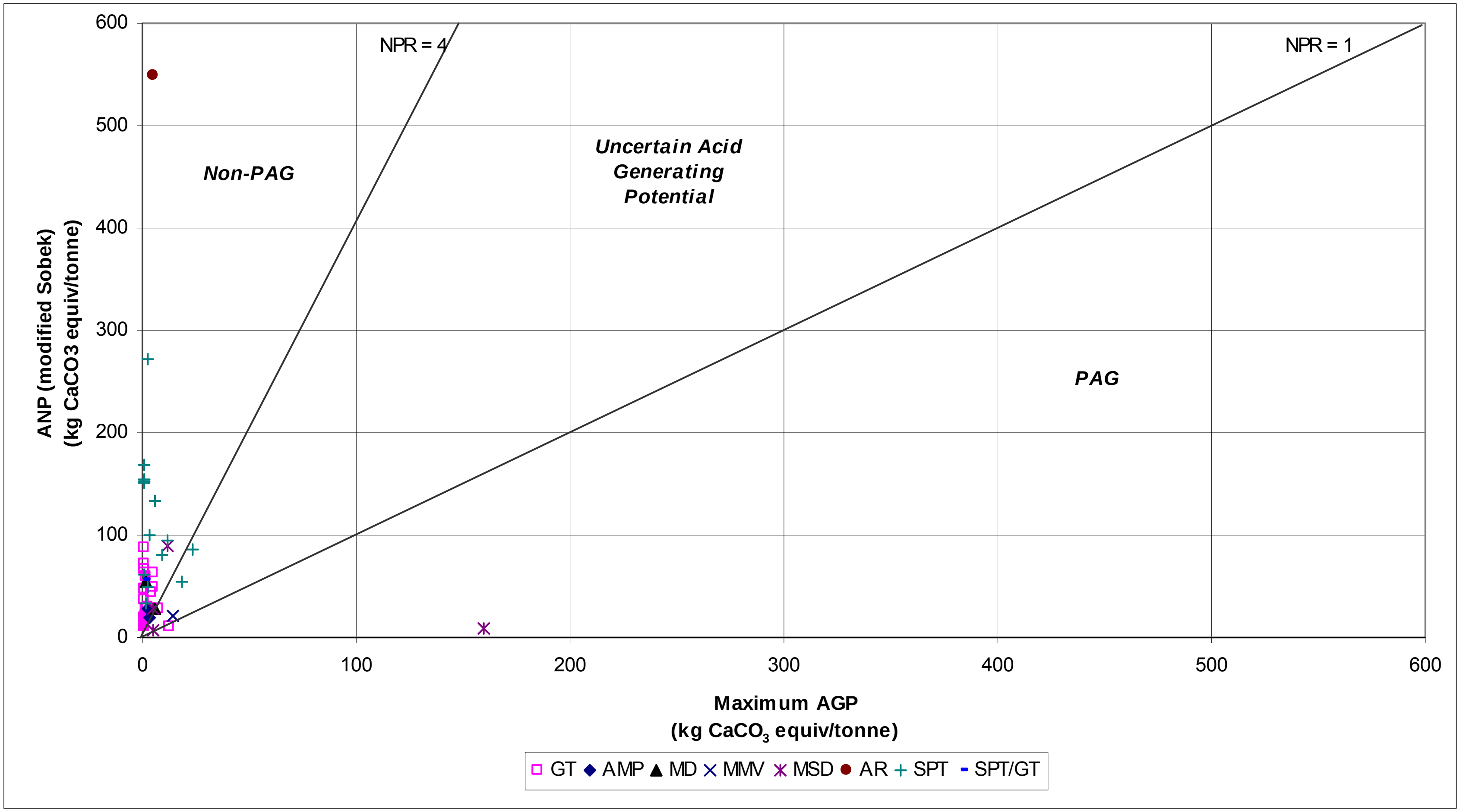
ANP = Modified Sobek Bulk Neutralization Potential in kilograms $CaCO_3$ equivalent per tonne of material.

NPR = ANP / AGP

Rock Types: GT=granite, SPT=serpentinite, MD=mafic dike, MMV=mafic metavolcanic, R=regolith, AMP=amphibolite, MSD=metasediment

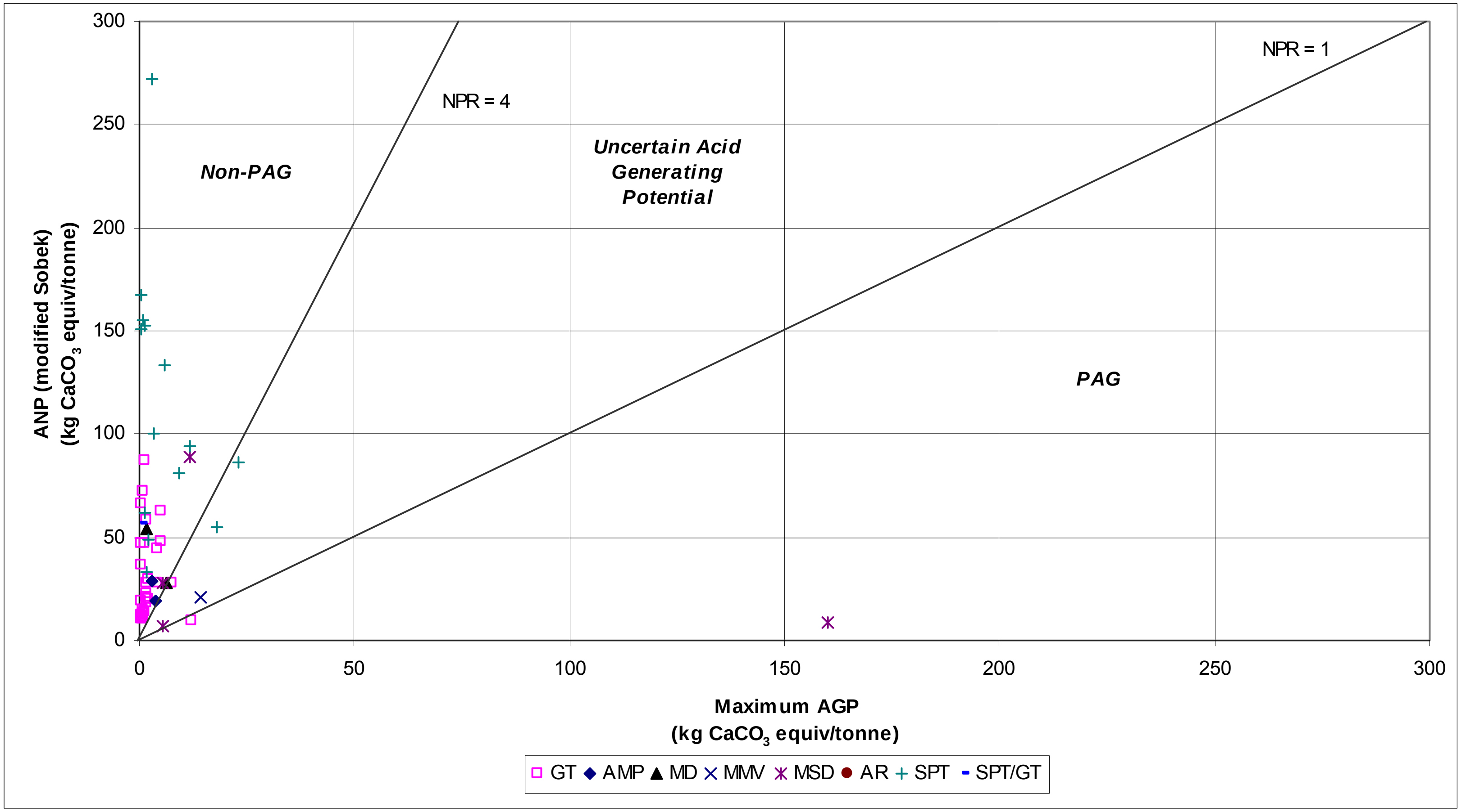
If the concentation was below the dectecion limit, half the detection limit was used to calculate the average.

Source: adapted from URS (2009i)



Source: URS (2009i)

Figure 2.8-3 Phase II Static Test Results – ANP versus AGP in Precambrian Lithologies



Source: URS (2009i)

Figure 2.8-4 Phase II Static Test Results – ANP versus AGP in Precambrian Lithologies (Detail) sample during storage. Low sulphate sulphur were expected as drillcores were fresh and the deposit is located at depth where oxygen concentrations are limited. Phase II results for the other acid-base accounting parameters varied widely (Table 2.8-4).

To determine whether the tested discrete and composite lithologies are potentially acid generating, the ARD/ML screening criteria of sulphide sulphur greater than 0.3 weight % and a Neutralization Potential Ratio (NPR) of less than 4 were applied to the static geochemical test results. Table 2.8-5 lists data for the nine samples tested in Phase II that exceeded one or both of these screening criteria in ascending order of the NPR. These criteria were exceeded by metasediment, mafic metavolcanic, serpentenite, and granite rock types. These results are also illustrated in Figures 2.8-1 and 2.8-4.

2.8.1.2.3 Sulphide Sulphur versus Total Sulphur Concentrations

Almost all sulphate sulphur concentrations were below the laboratory detection limit of 0.01 % by weight (Tables 2.8-2 and 2.8-4). Therefore, for samples where sulphate sulphur was not measured, the total sulphur value was used instead of sulphide sulphur value when calculating AGP. With very few exceptions, total sulphur concentrations were equal to the sulphide sulphur concentrations for the rock types assessed for the Minago Project. One significant exception was the serpentinite sample (BHK-41-R1-90) taken from 1994 Black Hawk Mining drill core. Serpentinite sample BHK-41-R1-90 had a high sulphate sulphur content of 0.22 % by weight (Table 2.8-5). This value potentially represents oxidation of sulphide material in the sample during storage. URS (2009i) recommended that sulphide sulphur analyses be included at 10% as a quality assurance check for additional static testing that may be conducted.

2.8.1.2.4 Carbonate Acid Neutralization Potential versus Modified Sobek Acid Neutralization Potential

Carbonate Acid Neutralization Potential (ANP) is a calculated amount of ANP within a sample that can be attributed to the presence of carbonate minerals. Carbonate ANP is calculated from a sample's % by weight CO_2, which is expressed as TIC in calcite equivalents (kg $CaCO_3$/tonne). By comparing carbonate ANP to ANP measured by the modified Sobek method, one can evaluate the effectiveness of the ABA techniques with respect to errors that may arise due to the presence of non acid neutralizing carbonate minerals (e.g., siderite [$FeCO_3$] and/or the presence of non-carbonate acid buffering minerals (e.g., chlorite and biotite).

The relationship between Carbonate ANP and modified Sobek bulk ANP is shown in Figures 2.8-5 and 2.8-6 for the Phase I and Phase II geochemical assessment program, respectively. The near linear correlation of the data indicates that the modified Sobek bulk method provides a reasonable estimate of the available ANP for all lithologic categories tested. Based on this relationship, URS (2009i) recommended to use the modified Sobek method for additional static testing and that total inorganic carbon (TIC) analyses and Carbonate Acid Neutralization Potential be included at 10% as a quality assurance check for additional static testing that may be conducted.

Table 2.8-5 Phase II Static Waste Rock Samples Exceeding ARD/ML Screening Criteria

Sample #	Rock Type	Rock Code	Drill Hole #	From (ft)	To (ft)	Length (ft)	paste pH	Fizz Test	Total Inorganic Carbon (wt%)	$CaCO_3$ Equivalent (kg $CaCO_3$/tonne)	Total Sulphur (wt%)	Sulphate Sulphur (wt%)	Sulphide Sulphur (wt%)*	Acid Generation Potential** (kg $CaCO_3$/tonne)	Acid Neutralization Potential (kg $CaCO_3$/tonne)	Net Neutralization Potential (kg $CaCO_3$/tonne)	NPR (ANP/AGP)
924738	Metasediment	MSD	N0710	135.4	136.2	0.80	7.7	none			5.12		**5.12**	160.0	9.0	-151.0	**0.06**
924424	Granite	GT	N0706	178.3	179.7	1.40	8.3	none			0.39		**0.39**	12.2	9.7	-2.5	**0.8**
924548	Metasediment	MSD	N0705	99.5	100.8	1.30	7.9	none			0.17		0.17	5.3	6.8	1.5	**1.29**
925884	Mafic Metavolcanic	MMV	N0712	249.5	251	1.50	9.3	none			0.46		**0.46**	14.4	21.0	6.6	**1.5**
365627	Serpentinite	SPT	BHK 41-R1-90	1051.5	1056.2	4.70	7.2	none	0.06	5.0	0.80	0.22	**0.58**	18.1	54.8	36.7	**3.0**
258774	Serpentinite	SPT	BHK 49-R9-90	818.5	821.5	3.00	7.1	none			0.74		**0.74**	23.1	86.3	63.2	**3.7**
926297	Granite	GT	N0726	215.81	218	2.19	9.0	none			0.24		0.24	7.5	28.0	20.5	**3.7**
925841	Metasediment	MSD	N0712	197	198.5	1.50	8.4	none			0.37		**0.37**	11.6	89.3	77.8	7.7
926243	Serpentinite	SPT	N0726	133.69	134.54	0.85	9.3	none			0.38		**0.38**	11.9	94.0	82.1	7.9

Notes:

* Based on difference between total sulphur and sulphate-sulphur

** Based on sulphide-sulphur

AGP = Maximum Potential Acidity in kilograms $CaCO_3$ equivalent per tonne of material.

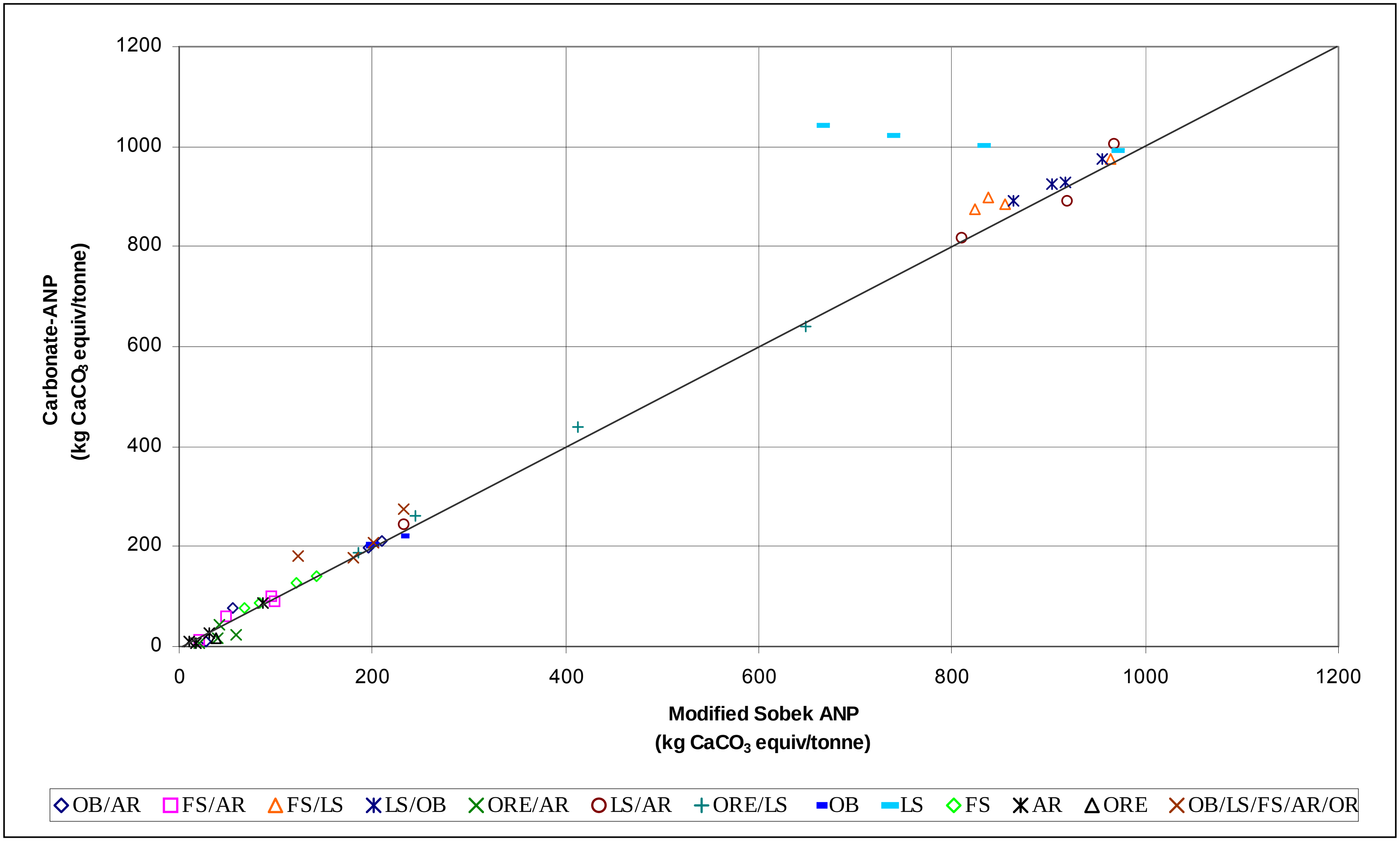
ANP = Modified Sobek Bulk Neutralization Potential in kilograms $CaCO_3$ equivalent per tonne of material.

NPR = ANP / AGP

1.29 Results highlighted in red and bold and that are underlined exceed the ARD/ML screening criteria (sulphide sulphur > 0.3% and NPR < 4).

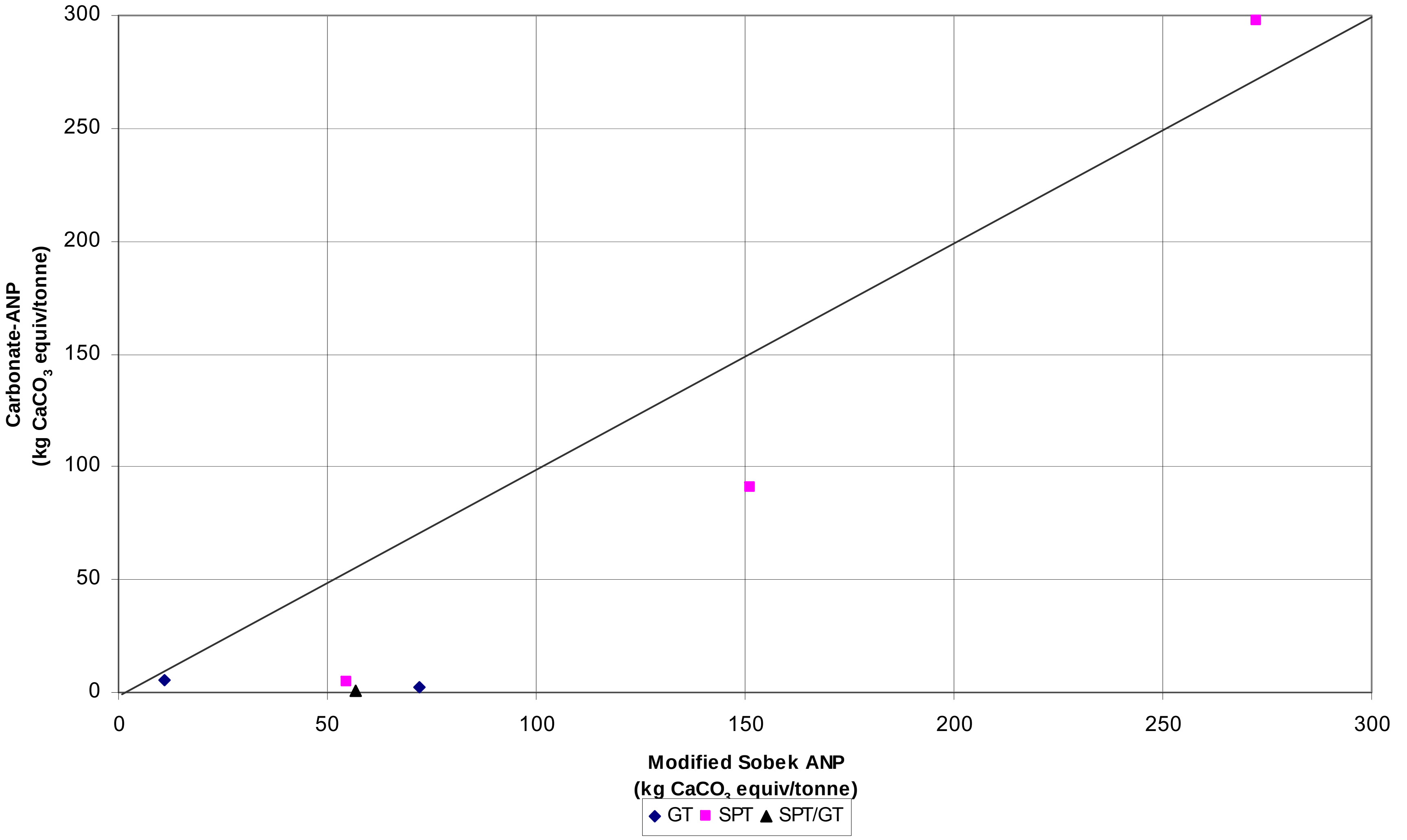
Rock Types: GT=granite, SPT=serpentinite, MD=mafic dike, MMV=mafic metavolcanic, R=regolith, AMP=amphibolite, MSD=metasediment

Source: adapted from URS (2009i)



Source: URS (2009i)

Figure 2.8-5 Phase I Static Test Results - Carbonate ANP versus Modified Sobek ANP



Source: URS (2009i)

Figure 2.8-6 Phase II Static Test Results - Carbonate ANP versus Modified Sobek ANP

2.8.1.2.5 Summary of Static Test Results for Waste Rock

Table 2.8-6 summarizes the NPR and PAG/NAG classifications of the significant lithologies for the Minago Project.

2.8.1.2.6 Metal Concentrations in Phase I and Phase II Samples

Selected average elemental concentrations in Phase I and Phase II static waste rock specimens are summarized in Table 2.8-7 and illustrated in Figures 2.8-7 through 2.8-10. Detailed elemental concentrations are given in Appendix 2.8.

Elemental concentrations in tested rock types were compared to ‘normal’ elemental concentrations in selected rock types for screening purposes (Turekian and Wedepohl, 1961). For screening purposes, levels greater than three (3) times the ‘normal’ elemental concentration were used to identify “elevated” elemental concentrations in the geochemical assessment (URS, 2009i). The matching of rock types between these rock types and those available for ‘normal’ elemental concentrations were a best fit. For example, granitic rocks were compared to low calcium granitic rocks and metasediments were compared to shale, which was assessed to be the closest parent rock type match for comparison (URS, 2009i). Phase I results indicate that overburden, dolomitic limestone and sandstone lithological samples had elevated concentrations of chromium (Cr), nickel (Ni), sulphur (S), antimony (Sb), thorium (Th) and uranium (U) (URS, 2009i). In overburden and limestone concentrations of these elements were slightly elevated and likely represent local and/or regional background. In sandstone elevated chromium, nickel and sulphur concentrations suggest a potential for metal leaching. A preliminary screening of the elemental concentrations of Precambrian basement lithologies indicates elevated barium, cobalt, chromium, copper, iron, nickel and sulphur.

2.8.1.3 Leachate Extractions

Based on the results of the Phase I static test program, four (4) samples were selected for shake flask extractions (SFEs) to determine readily leachable constituents and the likelihood of metal leaching (ML). Two (2) samples were discrete samples (sandstone samples N-07-27-FS and N-07-29-FS) and two (2) samples were composited samples (N-07-27-OB/AR and N-07-29-FS/AR). These samples represent lithological units frac sand (FS) and overburden (OB) and composites containing these lithological units.

Results of the SFE tests on Phase I static test samples are summarized in Table 2.8-8 and complete laboratory analytical results are included in Appendix 2.8. In composite samples N-07-27-OB/AR and N-07-29-FS/AR, aluminum was readily leachable at concentrations greater than the Manitoba Tier III Water Quality Guideline and the CCME Water Quality Guidelines for the protection of freshwater aquatic life (100 µg/L). In these samples, boron was readily leachable with concentrations ranging from 461 to 804 ug/L. In addition, selenium concentration (1.3 µg/L) in leachate from sample N-07-27-OB/AR and the copper concentrtation in sample N-07-29-FS/AR were above the Manitoba Tier III Water Quality Guideline and the CCME Water Quality Guidelines

Table 2.8-6 Summary of NPR and PAG/NAG Classifications by Lithology

MATERIAL TYPE		PAG?	NPR
Overburden		No	62.3 - 310
Ordovician Dolomitic Limestone		No	296 - 1181
Ordovician Sandstone		No	10.4 - 266
Altered Precambrian Basement			
	1) Phase I Static Tests	Uncertain	1.0 – 21.2
	2) HC-1 Static Tests	Uncertain	3.6
	3) HC-1 Kinetic Test	No	-
Precambrian Basement		Yes	0.3
Granite		No, but may have PAG hotspots	0.8 – 105.5
Serpentinite		No, but may have PAG hotspots	3 – 268.3
Amphibolites		No	5.1 – 10.2
Mafic Metavolcanic Rocks		Yes	1.5
Metasedimentary Rock		Yes	0.1 – 7.7
Mafic Dike		No	4.50 – 34.50
Overburden/Altered Rock Composite		No	6.1 – 209
Sandstone/Altered Rock Composite		No	3.9 – 61.3
Sandstone/ Limestone Composite		No	377 - 893
Limestone/Overburden Composite		No	482 - 922
Precambrian Basement/Altered Precambrian Basement Composite		Yes	< 4.0 (?)
Limestone/Altered Precambrian Basement (May be a solution for ARD) Composite		No	67 - 3095
Precambrian Basement/ Limestone (May be a solution for ARD)		No	17 - 112
Overburden/Sandstone/Limestone/Altered Precambrian Basement/Precambrian Basement Composite		No	10.5 – 65.5
Tailings		No	34.1 - 59.8

Table 2.8-7 Average Elemental Concentrations for Major Lithologies

	Sample Type		Number of Samples	Ba (mg/kg)	Co (mg/kg)	Cr (mg/kg)	Cu (mg/kg)	Fe (mg/kg)	Ni (mg/kg)	S (mg/kg)
Phase I Static Testing	**OB**	**Overburden**	4	126	13	52	24	27,200	35	400
	LS	**Limestone**	4	5	1	5	2	3,600	11	950
	FS	**Sandstone**	4	7	4	140	19	6,075	33	1,263
	AR	**Altered Rock**	4	74	54	330	183	26,625	1,230	3,275
	ORE	**Ore**	1	83	38	211	130	63,700	1,899	39,800
	OB/AR	**Overburden/Altered Rock**	4	100	12	99	22	26,050	72	850
	FS/AR	**Sandstone/ Altered Rock**	4	89	10	200	27	18,175	111	925
	FS/LS	**Sandstone / Limestone**	4	1	1	45	4	3,675	7	1,000
	LS/OB	**Limestone Overburden**	4	13	1	13	3	5,150	7	850
	ORE/AR	**Ore / Altered Rock**	4	149	40	254	95	29,275	1,477	1,800
	LS/AR	**Limestone / Altered Rock**	4	44	5	74	12	12,000	54	1,025
	ORE/LS	**Ore / Limestone**	4	165	43	176	56	22,850	1,139	4,200
	OB/LS/FS/AR/ORE	**Overburden / Limestone/ Sandstone/ Altered Rock / Ore**	4	140	28	230	51	24,550	936	2,525
Phase II Static Testing	**AMP**	**Amphibolite**	2	121	24	261	75	24,550	164	1,050
	GT	**Granite**	28	118	8	114	26	16,525	450	621
	MD	**Mafic Dike**	2	64	20	93	103	35,500	127	400
	MMV	**Mafic Metavolcanic**	1	181	17	216	54	22,000	136	4,700
	MSD	**Metasediment**	4	110	45	198	106	48,325	1,070	1,360
	AR	**Altered Precambrian**	1	30	97	170	24	27,300	1,258	1,900
	SPT	**Serpentinite**	14	167	90	376	110	39,100	> 3,266	1,914
	SPT/GT	**Serpentinite / Granite**	1	50	42	317	0.5	29,900	1,731	200
	Three times 'Normal' Concentrations (Turekian and Wedepohl, 1961):									
	3X-Clay			6,900	222	270	750	195,000	675	3,900
	3X-Sandstone			30	0	33	12	11,400	60	0
	3X-Limestone				1	105		29,400		0
	3X-low Ca Granite			2,520	3	12	30	42,600	14	900
	3X-high Ca Granite			1,260	21	66	90	88,800	45	900
	3X-Ultrabasic			1	450	4,800	30	282,900	6,000	900

Note: For concentrations below the detection limit, half the concentration was assumed for calculating the average concentration.

Source: adapted from URS (2009i)

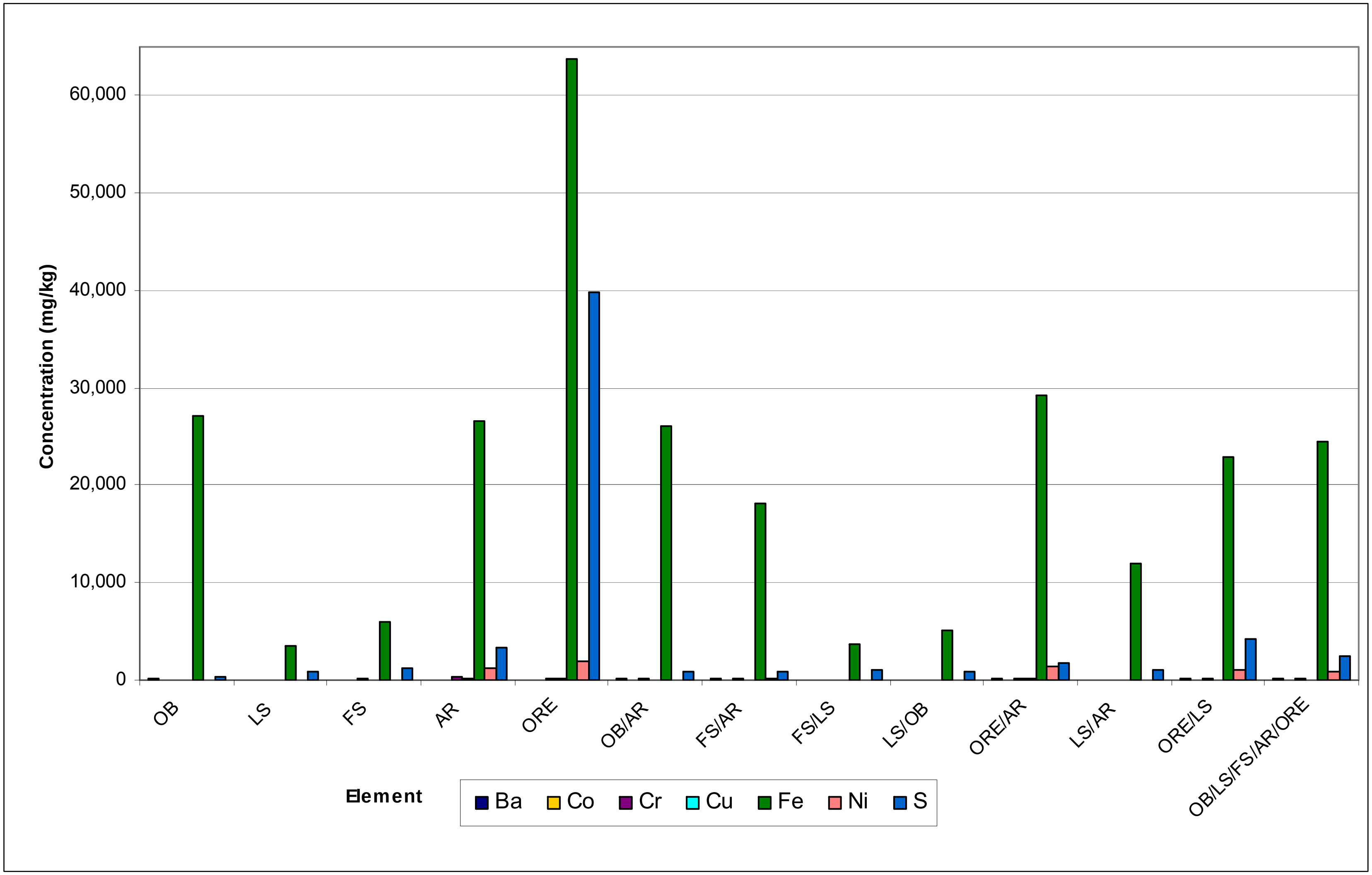


Figure 2.8-7 Phase I Static Test Results - Elemental Concentrations in Major Lithogies

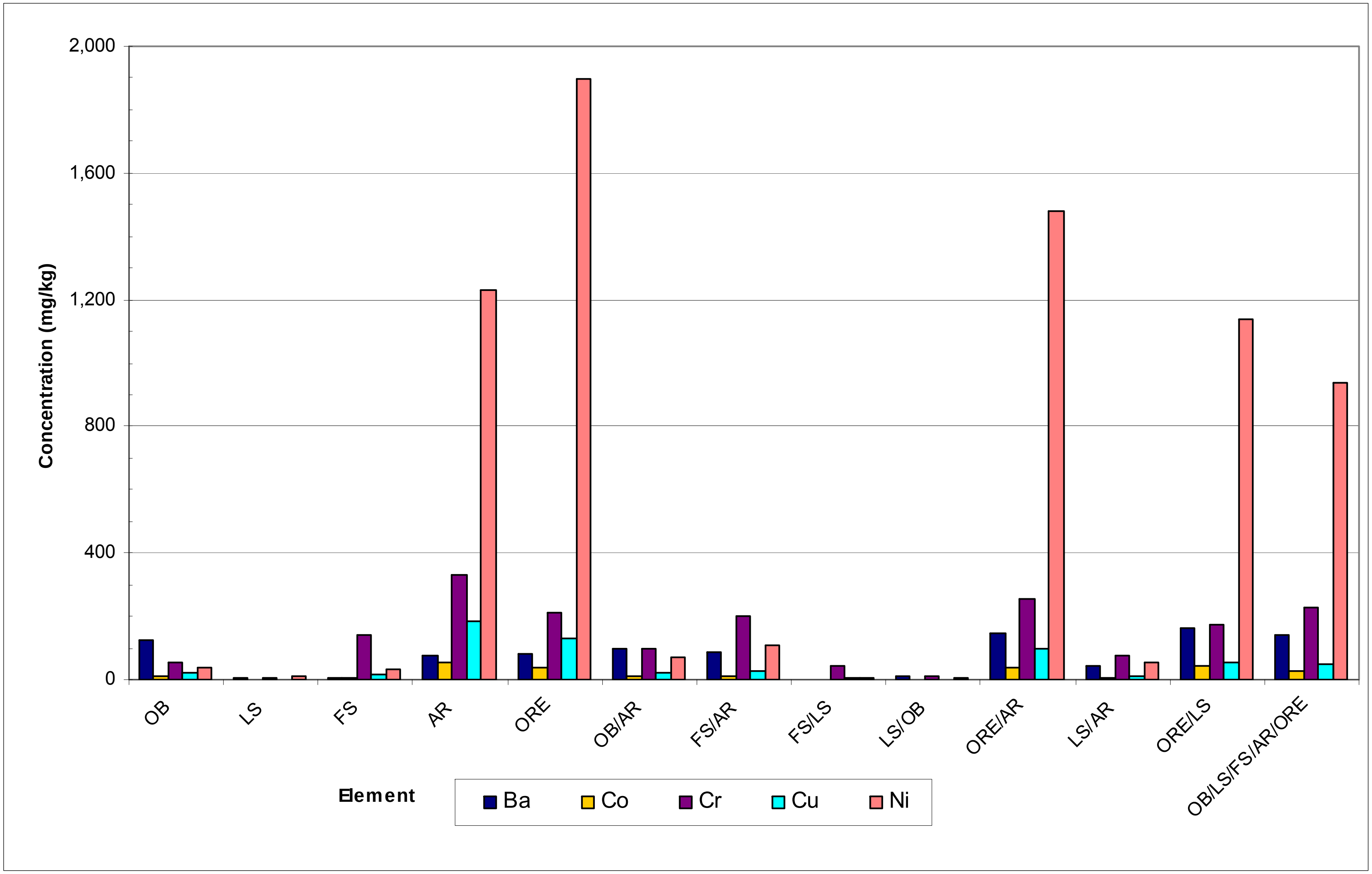


Figure 2.8-8 Phase I Static Test Results – Elemental Concentrations in Major Lithogies (Zoomed in)

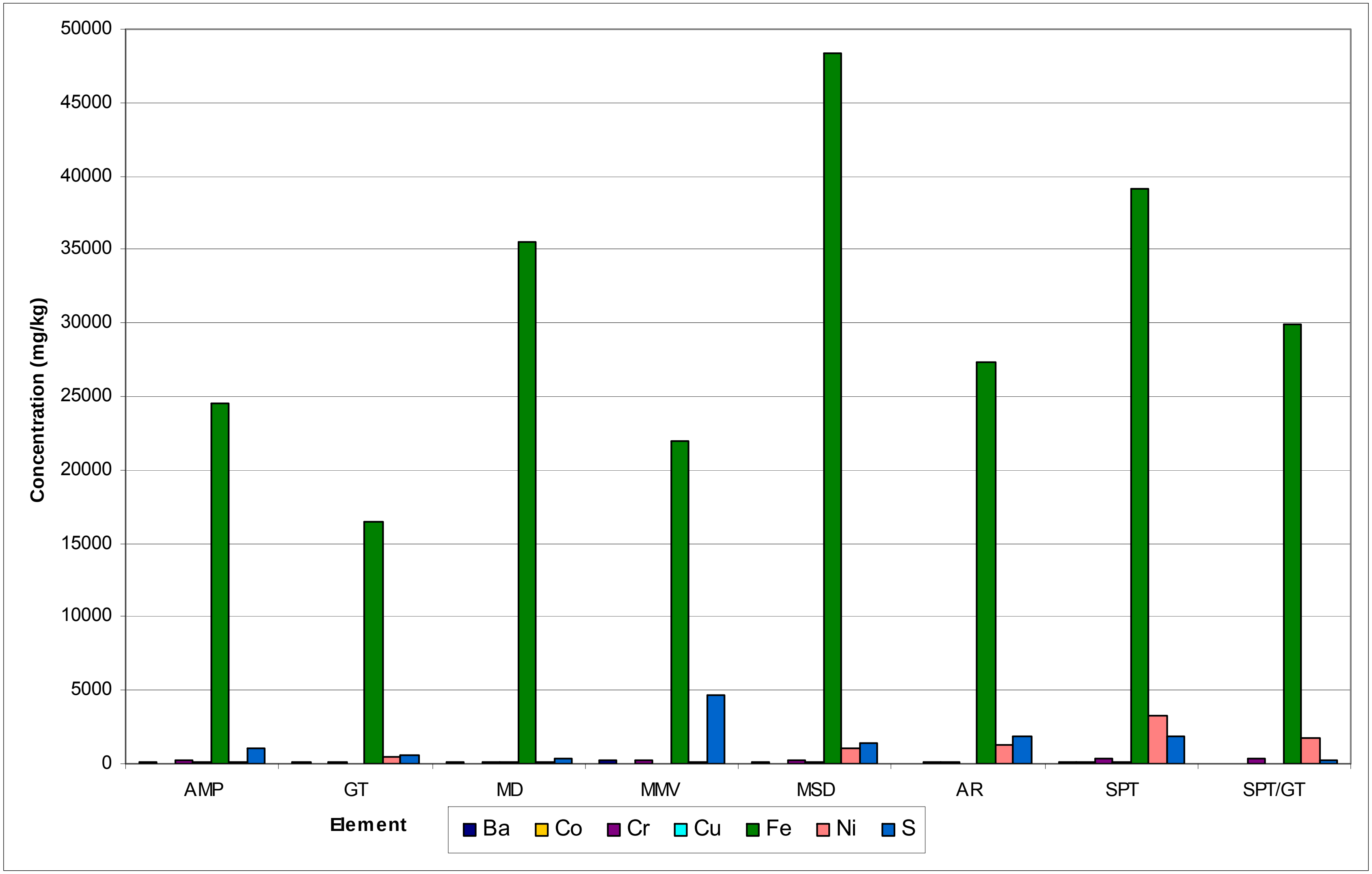


Figure 2.8-9 Phase II Static Test Results – Close-up of Elemental Concentrations in Major Lithogies

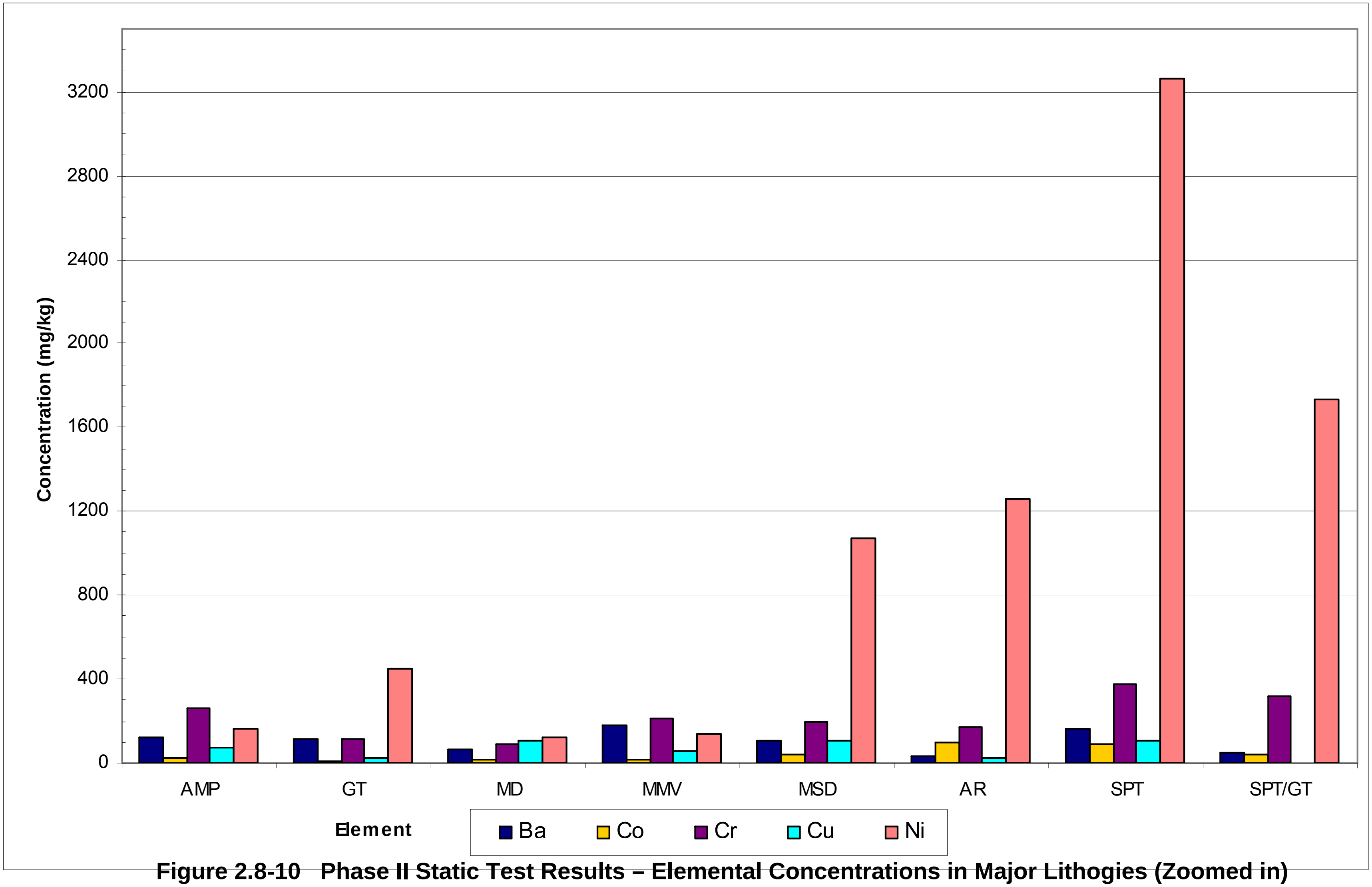


Figure 2.8-10 Phase II Static Test Results – Elemental Concentrations in Major Lithogies (Zoomed in)

Table 2.8-8 Average Elemental Concentrations for Major Lithologies

Sample ID			**#1-NO727-OB/AR**	**#2-NO729-FS/AR**	**#10-NO727-FS**	**#10-NO729-FS**	**REGULATIONS**			
Parameter	**Method**	**Units**					**Manitoba**	**Tier**	**CCME**	**MMER** [1]
Volume Nanopure water		mL	750	750	750	750				
Sample Weight		g	250	250	250	250				
pH	meter		8.15	8.52	7.88	7.90	6.5-8.5	III	6.5-9	6.5-9
Redox	meter	mV	313	292	314	322				
Conductivity	meter	uS/cm	328	266	123	169				
Acidity (to pH 4.5)	titration	mg CaCO3/L	na	na	na	na				
Total Acidity (to pH 8.3)	titration	mg CaCO3/L	1.4	na	1.7	1.8				
Alkalinity	titration	mg CaCO3/L	81.2	100.4	42.5	46.7				
Sulphate	Turbidity	mg/L	125	27	21	37	500	III	--	
Dissolved Metals										
Hardness CaCO3		mg/L	93.4	7	52.2	75.4				
Aluminum Al	ICP-MS	ug/L	115	530	23.5	20.7	100	III	100	
Antimony Sb	ICP-MS	ug/L	0.34	0.14	2.99	0.25	--			
Arsenic As	ICP-MS	ug/L	0.9	1	1.1	0.6	150[A]	II	5	1000
Barium Ba	ICP-MS	ug/L	35.7	2.13	19.3	31.5	--			
Beryllium Be	ICP-MS	ug/L	0.07	0.07	<0.05	<0.05	--			
Bismuth Bi	ICP-MS	ug/L	<0.05	<0.05	<0.05	<0.05	--			
Boron B	ICP-MS	ug/L	461	804	30	47	5000	III		
Cadmium Cd	ICP-MS	ug/L	0.04	0.01	0.03	0.03	here: 0.2-4[B]	II	0.017	
Calcium Ca	ICP-MS	ug/L	24100	1690	10900	16000	--			
Chromium Cr	ICP-MS	ug/L	<0.2	2	<0.2	0.4	here: 8-545[C]	II	8.9 [3]	
Cobalt Co	ICP-MS	ug/L	0.44	0.1	1.29	1.24	--			
Copper Cu	ICP-MS	ug/L	4.4	1.8	2.4	0.9	here: 0.8-12.5[D]	II	2-4[2]	600
Iron Fe	ICP-MS	ug/L	62	128	7	<5	300	III	300	
Lead Pb	ICP-MS	ug/L	0.2	0.11	0.33	0.03	here: 0.1-60[E]	II	here: 1-2[2]	400
Lithium Li	ICP-MS	ug/L	23.5	59.9	6.2	8	--			
Magnesium Mg	ICP-MS	ug/L	8070	680	6040	8620	--			
Manganese Mn	ICP-MS	ug/L	26.8	1.58	6.2	5.62	--			
Mercury Hg	CVAA	ug/L	<0.05	<0.05	<0.05	<0.05	0.1	III	0.026	
Molybdenum Mo	ICP-MS	ug/L	11.4	4.69	9.1	10.2	73	III	73	
Nickel Ni	ICP-MS	ug/L	5	2	28.4	9.6	here: 4.5-430[F]	II	here: 25-65[2]	1000
Phosphorus P	ICP-MS	ug/L	<100	<100	<100	<100				
Potassium K	ICP-MS	ug/L	10600	10900	2840	4030	--			
Selenium Se	ICP-MS	ug/L	1.3	0.6	5.7	0.9	1	III	1	
Silicon Si	ICP-MS	ug/L	1570	4110	620	790	--			
Silver Ag	ICP-MS	ug/L	<0.01	0.03	<0.01	0.06	0.1	III	0.1	
Sodium Na	ICP-MS	ug/L	46400	60700	4590	5830	--			
Strontium Sr	ICP-MS	ug/L	148	24.8	73.3	115	--			
Sulphur (S)	ICP-MS	ug/L	36100	7600	4900	10500	--			
Thallium Tl	ICP-MS	ug/L	<0.05	<0.05	<0.05	<0.05	0.8	III	0.8	
Tin Sn	ICP-MS	ug/L	<0.05	<0.05	<0.05	<0.05	--			
Titanium Ti	ICP-MS	ug/L	5.1	6.2	1.3	2.5	--			
Uranium U	ICP-MS	ug/L	2.61	1.09	4.32	3.46	--			
Vanadium V	ICP-MS	ug/L	1.75	9.04	0.31	0.63	--			
Zinc Zn	ICP-MS	ug/L	0.9	0.7	<0.5	<0.5	here: 10-110[G]	II	30	1000
Zirconium Zr	ICP-MS	ug/L	<5	<5	<5	<5	--	III		

Notes:

[1] monthly mean 2002 Metal Mining Effluent Regulations (MMER) requirements also include cyanide, TSS and acute toxicity.

[2] guideline concentration in CCME Water Quality Guidelines for the protection of freshwater aquatic life (Dec. 2007) depends on hardness.

[3] chromium III

Manitoba Water Quality Standards, Objectives, and Guidelines (Williamson, 2002):

A Arsenic limits: 0.15 mg/L for averaging duration 4 days (4-Day, 3-Year or 7Q10 Design Flow); 0.34 mg/L for averaging duration 1 hr (1-Day, 3-Year or 1Q10 Design Flow)

B Cadmium limits: e{0.7852[ln(Hardness)]-2.715}]×[1.101672-{ln(Hardness)(0.041838)}] for 4 days averaging duration.
e{1.128[ln(Hardness)]-3.6867}]×[1.136672-{ln(Hardness)(0.041838)}] for 1 hour averaging duration.

C Chromium limits: Chromium III: [e{0.8190[ln(Hardness)]+0.6848}]×[0.860] for 4 days averaging duration.
Chromium III: [e{0.8190[ln(Hardness)]+3.7256}]×[0.316] for 1 hour averaging duration.
Chromium VI: 0.011 mg/L for averaging duration 4 days (4-Day, 3-Year or 7Q10 Design Flow);
0.016 mg/L for averaging duration 1 hr (1-Day, 3-Year or 1Q10 Design Flow)

D Copper limits: [e{0.8545[ln(Hardness)]-1.702}]×[0.960] for 4 Days hour averaging duration.
[e{0.9422[ln(Hardness)]-1.700}]×[0.960] for 1 hour averaging duration.

E Lead limits: [e{1.273[ln(Hardness)]-4,705}]×[1.46203 -{ln(Hardness)(0.145712)}] for 4 Days averaging duration.
[e{1.273[ln(Hardness)]-1.460}]×[1.46203 -{ln(Hardness)(0.145712)}] for 1 hour averaging duration.

F Nickel limits: [e{0.8460[ln(Hardness)]+0.0584}]×[0.997] for 4 Days averaging duration.
[e{0.8460[ln(Hardness)]+2.255}]×[0.998] for 1 hour averaging duration.

G Zinc limits: [e{0.8473[ln(Hardness)]+0.884}]×[0.976] for 4 Days averaging duration.
[e{0.8473[ln(Hardness)]+0.884}]×[0.978] for 1 hour averaging duration.

Source: adpated from URS (2009i)

for the protection of freshwater aquatic life. The selenium guideline limit (1.0 µg/L) was also exceeded in leachate collected from the sandstone sample N-07-27-FS (5.7 µg/L).

URS (2009i) predicted that rock that had readily soluble constituents exceeding applicable provincial and/or federal criteria will likely not be exceeding the criteria under field weathering conditions as discussed below. The possible mineralogical source(s) of these readily soluble constituents are also discussed as part of the waste rock kinetic test program.

2.8.1.4 Kinetic Testing Program for Waste Rock

The objectives of the kinetic testing program were to:

- Assess the relative rates of acid generation and acid neutralization of representative material of pit walls, the pit floor, and waste rock material that will be disposed in waste rock dumps;
- Assess the relative timing of complete sulphide oxidation (acid generation) and complete weathering / dissolution of carbonate minerals (acid neutralization) and if acid neutralization is exhausted prior to acid generation, the onset of Acid Rock Drainage and Metal Leaching (ARD / ML);
- Assess the overall effect of mixed waste rock types (e.g., nickel bearing Precambrian rock types and limestone) on the relative rates of acid generation and acid neutralization;
- Predict leachate water quality and loadings from various mine components (e.g., waste rock dumps, pit walls, pit floor, low grade stockpiles); and
- Predict final effluent discharge water quality and, if necessary, the potential requirement for effluent treatment.

URS submitted the four composited waste rock kinetic test samples to SGS-CEMI for analyses, including (URS, 2009i):

- Static testing of humidity cell composites;
- Optical mineralogical analysis;
- Weekly wet/dry cycling with 750 ml deionized water added in week 1 and 500 ml of deionized water in subsequent weeks for 63 weeks;
- Weekly measurement of pH, oxidation reduction potential, specific conductivity, and analysis of acidity, alkalinity, and sulphate;
- Biweekly analysis of total metals by ICP-AES;
- Shake flask extraction tests on humidity cell test samples after 63 weeks; and
- Static testing of humidity cell test residual material after 63 weeks.

2.8.1.4.1 Sample Selection

Based on results of the Phase I static test program, the following four (4) composited Phase I static test samples were selected for laboratory kinetic humidity cell testing (Table 2.8-9) (URS, 2009i):

- Humidity Cell 1 contained four (4) Phase I subsamples of N-07-27-AR, N-07-28-AR, N-07-29-AR and N-07-36-AR. These samples represent a significant portion of the waste rock material that will be generated from the open pit.
- Humidity Cell 2 contained four (4) Phase I subsamples of N-07-27-ORE/AR, N-07-28-ORE/AR, N-07-29-ORE/AR and N-07-36-ORE/AR. These samples represent a significant portion of the waste rock material that will be generated from the open pit.
- Humidity Cell 3 contained four (4) Phase I subsamples of N-07-27-ORE/LS, N-07-28-ORE/LS, N-07-29-ORE/LS and N-07-36-ORE/LS. These samples represent waste rock material that will be generated from the open pit. A portion of the waste rock dumps is expected to contain excess limestone from the open pit and that will not be used in mine development.
- Humidity Cell 4 contained four (4) Phase I subsamples of N-07-27-OB/LS/FS/AR/ORE, N-07-28-OB/LS/FS/AR/ORE, N-07-29-OB/LS/FS/AR/ORE and N-07-36-OB/LS/FS/AR/ORE. Because portions of the waste rock dumps are expected to contain a mix of all lithological units, these samples are representative of mixed waste from the open pit.

2.8.1.4.2 Pre-Kinetic (Humidity Cell) Test Results

Pre-kinetic static test and mineralogical results are summarized below. Detailed results are given in Appendix 2.8 and elsewhere (URS, 2009i).

Types, relative abundances, and modes of occurrence of sulphide and carbonate minerals were assessed by mineralogic analyses. There is limited information on the spatial relationship of sulphides and carbonates within each rock type. However, due to the small size of the rock fragment examined as part of the mineralogical analysis, it can be assumed that within each of the composite samples, acid generating and acid consuming components would be in proximity to one another on a centimeter scale, and in some cases locally on a millimeter scale. In other words, sulphides and carbonates have the same mode of occurrence (e.g., fracture hosted) or are part of the same overall primary or secondary mineral alteration present within the same rock type. The identification of the mineralogy by optical methods is challenging for these sample materials where the grain size is small (i.e., extremely fine grained in many instances) and abundance is low (i.e., trace or <1%).

Humidity Cell 1 – AR Composite

Humidity cell 1 (HC-1) was a composite of altered Precambrian basement material. HC-1 had a total sulphur content of 0.28 weight %, a sulphate sulphur content of <0.01 weight %,

Table 2.8-9 Composition of Waste Rock Kinetic Humidity Cells

HC-1 AR	**Composition Ratio**	Weight (g)	**HC-2 ORE/AR**	**Composition Ratio**	Weight (g)	**HC-3 ORE/LS**	**Composition Ratio**	Weight (g)	**HC-4 OB/LS/FS/AR/ORE**	**Composition Ratio**	Weight (g)
N0727-AR	1	300	N0727-ORE/AR	11.6:1	300	N0727-ORE/LS	2.1:1	300	N0727-OB/LS/FS/AR/ORE	0.05:0.49:0.07:0.08:1	300
N0728-AR	1	300	N0728-ORE/AR	66:1	300	N0728-ORE/LS	3.7:1	300	N0728-OB/LS/FS/AR/ORE	0.03:0.19:0.03:0.01:1	300
N0729-AR	1	300	N0729-ORE/AR	12:1	300	N0729-ORE/LS	3.3:1	300	N0729-OB/LS/FS/AR/ORE	0.05:0.3:0.04:0.08:1	300
N0736-AR	1	300	N0736-ORE/AR	0.26:1	300	N0736-ORE/LS	0.75:1	300	N0736-OB/LS/FS/AR/ORE	0.03:0.35:0.05:1:0.26	300
Total		**1200**			**1200**			**1200**			**1200**

Source: URS (2009i)

and a sulphide sulphur content of 0.28 weight % (Appendix 2.8). The corresponding Acid Generation Potential (AGP) was 8.8 kg $CaCO_3$ per tonne. The total inorganic carbon (TIC) content was 0.35 weight % and the carbonate ANP was 29.2 kg $CaCO_3$ per tonne. This carbonate ANP value correlates strongly with a modified Sobek ANP of 31.9 kg $CaCO_3$ per tonne. The Neutralization Potential Ratio (NPR) was 3.6 (URS, 2009i). Therefore, the sample is considered to have an uncertain AGP, as the NPR is between 1 and 4 (URS, 2009i).

The sulphide minerals identified in trace amounts in the HC-1 sample material were pyrite (FeS_2) and the nickel sulphide species pentlandite ($(Fe,Ni)_9S_8$), millerite (NiS) and violarite ($Fe^{2+}Ni_2^{3+}S_4$). Pyrite occurred as extremely fine clusters of subhedral grains. Pentlandite and millerite occurred as intergrowths, and violarite as corona rims on pentlandite (URS, 2009i).

Up to 15% carbonate was identified occurring in three (3) modes (URS, 2009i):

- Anhedral, strongly foliated masses or individual grains and fragment, likely from veins;
- Very fine granular grains in a microcrystalline groundmass; and
- Aphanitic in patches and fragment.

A complete description of the mineralogical composition of the sample material used for HC-1 is provided in URS (2009i).

Humidity Cell 2 – ORE/AR Composite

Humidity cell 2 (HC-2) was a composite of altered and unaltered Precambrian basement materials. Humidity Cell 2 (HC-2) had a total sulphur content of 0.16 weight %, a sulphate sulphur content of <0.01 weight %, and a sulphide sulphur content of 0.16 weight % (Appendix 2.8). The corresponding Acid Generation Potential (AGP) was 5.0 kg $CaCO_3$ per tonne. The total inorganic carbon (TIC) content was 0.24 weight % and the carbonate ANP was 20.0 kg $CaCO_3$ per tonne. This carbonate ANP value was lower than the modified Sobek ANP of 39.0 kg $CaCO_3$ per tonne. The Neutralization Potential Ratio (NPR) was 7.8 (URS, 2009i). Therefore, the sample is considered to be non-acid generating (NAG), as the NPR is greater than 4.

The sulphide minerals identified in trace amounts in the tested HC-2 sample were pyrrhotite ($Fe_{0.83-1}S$), pyrite (FeS_2) and chalcopyrite ($CuFeS_2$), and the nickel sulphide species pentlandite ($(Fe,Ni)_9S_8$) and millerite (NiS). Pyrrhotite was very fine grained and occurred with pentlandite and millerite in serpentinite (URS, 2009i). Pyrite was extremely fine grained and occurred in dolomite clusters in serpentinite. Pentlandite and millerite were very fine grained and anhedral and occurred in granular clusters in serpentinite. Chalcopyrite occurred as very fine anhedral grains in serpentinite (URS, 2009i).

Up to 3% carbonate, predominantly dolomite, was identified consisting of very fine to fine anhedral grains in serpentinite. A trace amount of brown iron-rich carbonate was also identified and occurred sporadically in a microcrystalline groundmass.

Humidity Cell 3 – ORE/LS Composite

Humidity cell 3 (HC-3) was a composite of Ordovician dolomitic limestone and Precambrian basement material. Humidity Cell 3 (HC-3) had a total sulphur content of 0.35 weight %, a sulphate sulphur content of <0.01 weight %, and a sulphide sulphur content of 0.35 weight % (Appendix 2.8). The corresponding Acid Generation Potential (AGP) was 10.9 kg $CaCO_3$ per tonne. The total inorganic carbon (TIC) content was 4.55 weight % and the carbonate ANP was 379.2 kg $CaCO_3$ per tonne. This carbonate ANP value correlated strongly with the modified Sobek ANP of 443.5 kg $CaCO_3$ per tonne. The Neutralization Potential Ratio (NPR) was 40.7 (URS, 2009i). Therefore, the sample may be considered to be non-acid generating (NAG) as the NPR is greater than 4.

The sulphide minerals identified in trace amounts in HC-3 were pyrrhotite ($Fe_{0.83-1}S$), chalcopyrite ($CuFeS_2$) and possibly cubanite ($(CuFe)_2S_3$), and the nickel sulphide species pentlandite ($(Fe,Ni)_9S_8$), millerite (NiS) and violarite ($Fe^{2+}Ni_2^{3+}S_4$). Pyrrhotite occurred as very fine anhedral grains in granite. Chalcopyrite occurred as extremely fine grains in dolomite and very fine grains in granite and serpentinite. As in other humidity cell samples, nickel sulphides occurred as very fine grained anhedral granular clusters and as intergrown nickel sulphide masses in serpentinite and granite fragments (URS, 2009i).

Carbonates, predominantly dolomite, in limestone fragments comprised approximately 40% of all carbonates. The dolomite consisted of very fine to fine, subhedral to rhombic aggregates. Within serpentinite, carbonates were very fine grained and anhedral (URS, 2009i).

Humidity Cell 4 – OB/LS/FS/AR/ORE Composite

Humidity cell 4 (HC-4) was a composite of material from all the Minago Project rock categories. Humidity Cell 4 (HC-4) had a total sulphur content of 0.73 weight %, a sulphate sulphur content of <0.01 weight %, and a sulphide sulphur content of 0.73 weight % (Appendix 2.8). The corresponding Acid Generation Potential (AGP) was 22.8 kg $CaCO_3$ per tonne. The total inorganic carbon (TIC) content was determined to be 2.62 weight % and the carbonate ANP was 218.3 kg $CaCO_3$ per tonne. This carbonate ANP value correlated strongly with the modified Sobek ANP of 238.1 kg $CaCO_3$ per tonne. The Neutralization Potential Ratio (NPR) was 10.4 (URS, 2009i). Therefore, the sample may be considered to be non-acid generating (NAG), since the NPR is greater than 4.

The sulphide minerals identified in trace amounts in HC-4 were pyrrhotite ($Fe_{0.83-1}S$), pyrite (FeS_2) and chalcopyrite ($CuFeS_2$), and the nickel sulphide species pentlandite ($(Fe,Ni)_9S_8$), millerite (NiS) and violarite ($Fe^{2+}Ni_2^{3+}S_4$) (URS, 2009i). Pyrrhotite occurred as very fine subangular grains in mafic fragments. Chalcopyrite occurred as very fine grains in serpentinite. Pyrite occurred as very fine subangular grains locally intergrown with pyrrhotite and fracture infill in mafic fragments.

Pyrite also occurred as extremely fine grains in altered granite and serpentinite fragments. As in other humidity cell samples, nickel sulphides occurred as very fine-grained anhedral granular clusters and as intergrown nickel sulphide masses in serpentinite and granite fragments (URS, 2009i).

Carbonates, predominantly dolomite, comprised approximately 40% of all carbonates present. The dolomite occurred as very fine to fine, subhedral to rhombic aggregates within the dolomite fragments. Within serpentinite, carbonates were very fine to fine grained and anhedral (URS, 2009i).

2.8.1.4.3 Kinetic (Humidity Cell) Test Results

Chemical loading rates were calculated from kinetic humidity cell test results on a weekly basis by multiplying the volume of leachate extracted by the analytically measured concentration. Loading results were expressed as milligrams constituent per kilogram rock mass per week (mg/kg/wk). Where concentrations of constituents were reported as a detection limit, the detection limit was taken to be the measured value.

Estimated laboratory weekly loading rates are summarized in Table 2.8-10. While loadings were calculated for most constituents or parameters, only those considered most relevant are detailed in the main body of this report. Loading rates for all constituents and parameters can be found in Appendix 2.8, including graphical illustrations of kinetic test results obtained for the waste rock.

Humidity Cell 1

Mineralogical analysis of HC-1 identified mainly granite, serpentinite, and mafic rock fragments. The sulphide content of material in HC-1 was not particularly elevated (0.28 % by weight); however, the carbonate content was low and therefore the NPR was 3.6.

After 63 weeks, the pH of HC-1 was near-neutral at 7.51 and during the kinetic testing, the pH was relatively constant, ranging between 7.30 and 8.53 (Appendix 2.8). The pH decreased very slightly at week 21 and then remained relatively constant thereafter (URS, 2009i).

Sulphate loading rates were initially the highest for HC-1 (Appendix 2.8). The sulphate loading rates decreased from a peak of 138 mg/kg/wk at week 2 to below 17 mg/kg//wk at week 16. This initial sulphate release is likely an artifact of laboratory kinetic testing and the flushing of stored sulphate or quickly generated sulphate caused by rapid sulphide oxidation of sulphides liberated during sample preparation (URS, 2009i). After week 15, sulphate loading rates slowly decreased from a maximum of 20 mg/kg/wk to a minimum of 2 mg/kg/wk.

Nickel loading rates for HC-1 followed a similar pattern to sulphate loading (Appendix 2.8). However, nickel loading rates decreased more rapidly, reaching near steady-state loading rates by week 5. Beyond week 5, nickel loading rates remained relatively constant, ranging from a maximum of 0.0054 mg/kg/wk to a minimum of 0.0009 mg/kg/wk.

In HC-1, the calcium loading rates were initially at a peak of 10.8 mg/kg/wk at week 1 and then rapidly decreased to a minimum of 1.0 mg/kg/wk at week 9 (Appendix 2.8). After week 9, the calcium loading rates steadily increased to 3.8 mg/kg/wk at week 61. The magnesium loading

Table 2.8-10 Laboratory Kinetic Test Results and Loading Rates for Minago Waste Rock

HCT No.	Lithology	Loading Rates (mg/kg/wk)[1]								
		Boron	Chromium	Cobalt	Copper	Iron	Molybdenum	Nickel	Selenium	Strontium
HCT-1	AR	0.04	8.9E-05	6.2E-05	4.6E-04	0.003	4.47E-04	0.0017	1.7E-04	0.035
HCT-2	ORE/AR	0.11	1.4E-04	6.9E-05	2.8E-04	0.017	9.81E-05	0.0060	6.1E-04	0.009
HCT-3	ORE/LS	0.04	9.9E-05	6.0E-05	2.9E-04	0.012	1.00E-04	0.0025	2.4E-04	0.009
HCT-4	OB/LS/FS/AR/ORE	0.09	8.8E-05	5.1E-05	4.6E-04	0.017	2.59E-04	0.0031	8.4E-05	0.017

[1] Loading rates are calculated as the average loading rates during weeks 20-63 when HCTs were in a steady state condition

Source: URS, 2009i

rates in HC-1 showed a similar response as calcium loading rates (Appendix 2.8). However, the minimum magnesium loading of 0.28 mg/kg/wk was not reached until week 19. In the absence of increased sulphate loading rates, the increase in calcium and magnesium loading rates are attributed to non-acid neutralization carbonate dissolution (URS, 2009i).

Aluminum loading rates for HC-1 were high initially at 0.09 mg/kg/wk at week 1. Over 63 weeks of laboratory weathering, aluminum loading rates gradually decreased to 0.006 mg/kg/wk at week 63 (Appendix 2.8). The initial peaks in aluminum loading rates up to week 21 are likely an artifact of the laboratory weathering and due to the flushing of readily soluble aluminosilicate phases (URS, 2009i). The variability in aluminum loading rates after week 21 suggests that there may have been some aluminosilicate mineral solution-dissolution occurring over the remaining period up to 63 weeks.

Selenium loading rates were initially high (0.005 mg/kg/wk) for HC-1 (Appendix 2.8). However, selenium loading rates quickly decreased and reached near steady-state levels by week 19. Between week 19 and kinetic test termination at week 63, the selenium loading rates were extremely low, decreasing from 0.0003 mg/kg/wk to 0.0001 mg/kg/wk. The initially high selenium loading rates suggest that sulphides with trace selenium not detected by optical mineralogical analysis were released during the first 19 weeks of laboratory weathering (URS, 2009i).

Humidity Cell 2

Mineralogical analysis of HC-2 identified mainly granite, serpentinite, and mafic rock fragments in the sample. The sulphide content (0.16 % by weight) of material in HC-2 was lower than HC-1 (0.28 % by weight); however, the carbonate content also was low and therefore the NPR was 7.8.

After 63 weeks, the pH of HC-2 was weakly alkaline at 7.37 and during the kinetic testing the pH was relatively constant, ranging between 7.37 and 9.22 (Appendix 2.8). Beyond week 21, the pH has remained relatively constant, slightly below 8.00. The initial weakly alkaline pH and pH decline to week 21 likely represents an artifact of laboratory kinetic testing and the flushing of an initial release of alkalinity from readily soluble carbonates (URS, 2009i).

Sulphate loading rates were initially the lowest for HC-2 (Appendix 2.8). However, the sulphate loading rates increased to a maximum peak of 79 mg/kg/wk at week 3 and then decreased to 20 mg/kg/wk at week 10. This initial sulphate release is likely an artifact of laboratory kinetic testing and the flushing of stored sulphate or quickly generated sulphate caused by rapid sulphide oxidation of sulphides liberated during sample preparation. Since week 12, sulphate loading rates slowly decreased from a maximum of 18 mg/kg/wk to a minimum of 3 mg/kg/wk.

Nickel loading rates for HC-2 were initially low and have remained low for the 63 week duration of the kinetic tests (Appendix 2.8). Beyond week 5, nickel loading rates increased to 0.0249 mg/kg/wk at week 49, then decreased to 0.0049 mg/kg/wk at week 63.

In HC-2, the calcium loading rates were initially low (0.72 mg/kg/wk), but increased to a peak of 1.1 mg/kg/wk at week 5. After week 10, calcium loading rates stayed relatively constant at 0.45

mg/kg/wk until week 43 (Appendix 2.8). Between week 46 and week 63, the calcium loading rates stayed at or slightly above 1.0 mg/kg/wk. The magnesium loading rates in HC-2 showed a response similar to calcium loading rates (Appendix 2.8). In the absence of increased sulphate loading rates, the increases in calcium and magnesium loading rates are attributed to non-acid neutralization carbonate dissolution.

In HC-2, aluminum loading rates were high initially at 0.17 mg/kg/wk at week 1 and reached a maximum peak of 0.22 mg/kg/wk at week 5 (Appendix 2.8). The initial peaks in aluminum loading rates up to week 21 are likely an artifact of the laboratory weathering and due to the flushing of readily soluble aluminosilicate phases. After week 5, aluminum loading rates decreased overall to 0.005 mg/kg/wk at week 63. The high variability in aluminum loading rates after week 5 suggests that some aluminosilicate mineral dissolution may have occurred (URS, 2009i).

For HC-2, selenium loading rates were initially elevated (0.002 mg/kg/wk), reaching a maximum peak of 0.004 mg/kg/wk at week 5 (Appendix 2.8). After week 5, selenium loading rates decreased and reached near steady-state levels by week 22. Between week 22 and kinetic test termination at week 63, the selenium loading rates were low, decreasing from 0.001 mg/kg/wk to 0.0003 mg/kg/wk. The initially high selenium loading rates suggest that sulphides with trace selenium not detected by optical mineralogical analysis were released, primarily during the first 22 weeks of laboratory weathering. Selenium loading rates were higher for HC-2 (containing fragments of both altered Precambrian basement and Precambrian basement) than for HC-1 (containing only fragments of Precambrian basement material). These results suggest that the sulphide hosting trace selenium may have been more abundant within HC-2 and/or more readily leached from selenium-bearing sulphides in HC-2.

Humidity Cell 3

Mineralogical analysis of HC-3 identified mainly granite, serpentinite, and amphibolite rock fragments within the Precambrian basement material. The sulphide content (0.35 % by weight) of material in HC-3 was moderately high relative to HC-1 and HC-2. However, the carbonate content was high (4.65 % by weight) and therefore the NPR was 40.5.

After 63 weeks, the pH of HC-3 was weakly alkaline at 7.51 and during the kinetic testing the pH was relatively constant, ranging between 7.36 and 9.13 (Appendix 2.8). Beyond week 21, the pH remained relatively constant at slightly below 8.00. The initial weakly alkaline pH and pH decline to week 21 likely represents an artifact of laboratory kinetic testing and the flushing of an initial release of alkalinity from readily soluble carbonates.

For HC-3, the sulphate loading rates were initially 81 mg/kg/wk (Appendix 2.8) and decreased to below 20 mg/kg/wk at week 7. This initial sulphate release is likely an artifact of laboratory kinetic testing and the flushing of stored sulphate or quickly-generated sulphate caused by rapid sulphide oxidation of sulphides liberated during sample preparation (URS, 2009i). After week 7, sulphate loading rates slowly decreased from a maximum of 17 mg/kg/wk to a minimum of 1 mg/kg/wk.

Nickel loading rates for HC-3 were initially low and have remained low for the 63 week duration of the kinetic tests (Appendix 2.8). Between weeks 12 and 63, the nickel loading rates exhibited little variability, ranging from a maximum of 0.0062 mg/kg/wk to a minimum of 0.0013 mg/kg/wk.

In HC-3, the calcium loading rates were initially 1.58 mg/kg/wk and decreased to a minimum of 0.52 mg/kg/wk at week 3 (Appendix 2.8). After week 3, the calcium loading rates increased to near 1.00 mg/kg/wk and stayed near this level until kinetic test termination at week 63. The magnesium loading rates in HC-3 showed a response similar to calcium loading rates and were slightly lower than calcium loading rates (Appendix 2.8). In the absence of increased sulphate loading rates, the increase in calcium and magnesium loading rates are attributed to non-acid neutralization carbonate dissolution (URS, 2009i).

In HC-3, aluminum loading rates were low initially at 0.05 mg/kg/wk at week 1 and reached a maximum of 0.15 mg/kg/wk at week 9 (Appendix 2.8). The initial peaks in aluminum loading rates up to week 9 are likely an artifact of the laboratory weathering and due to the flushing of readily soluble aluminosilicate phases. After week 9, aluminum loading rates decreased overall to 0.008 mg/kg/wk at week 63. The low to moderate variability in aluminum loading rates after week 9 suggests that there may have been a limited amount of alumnosilicate mineral dissolution occurring over the 63 weeks (URS, 2009i).

Selenium loading rates began at 0.002 mg/kg/wk for HC-3, which was significantly lower than for HC-1 and HC-2 (Appendix 2.8). However, selenium loading rates quickly decreased and reached near steady-state levels by week 15. Between week 15 and kinetic test termination at week 63, the selenium loading rates were extremely low, decreasing from 0.0005 mg/kg/wk to 0.0002 mg/kg/wk. The lower selenium loading may initially have been influenced in part by the presence of dolomite rock fragments that provided micro-scale and/or meso-scale pH control on selenium dissolution (URS, 2009i).

Humidity Cell 4

Mineralogical analysis of HC-4 identified mainly granite, serpentinite and mafic rock fragments. The sulphide content (0.73 % by weight) of material in HC-4 was the highest of all four humidity cells. The carbonate content was moderately high (2.62 % by weight) and therefore the NPR was 10.4.

After 63 weeks, the pH of HC-4 was weakly alkaline at 7.36 and during the kinetic testing, the pH was relatively constant, ranging between 7.36 and 8.96 (Appendix 2.8). Beyond week 21, the pH has remained relatively constant at slightly below 8.00. The initial weakly alkaline pH and pH decline to week 21 likely represents an artifact of laboratory kinetic testing and the flushing of an initial release of alkalinity from readily soluble carbonates (URS, 2009i).

For HC-4, the sulphate loading rates were initially 54 mg/kg/wk (Appendix 2.8) and then increased to a maximum peak of 96 mg/kg/wk at week 3. However, after week 3, the sulphate loading rates decreased and were below 20 mg/kg/wk at week 13. This initial sulphate release is likely an artifact of laboratory kinetic testing and the flushing of stored sulphate or quickly generated sulphate caused by rapid sulphide oxidation of sulphides liberated during sample preparation (URS, 2009i). Since week 13, sulphate loading rates have slowly decreased from a maximum of 20 mg/kg/wk to a minimum of 4 mg/kg/wk.

Nickel loading rates for HC-4 were initially low and remained low for the 63 week duration of the kinetic tests (Appendix 2.8). Between weeks 12 and 63, the nickel loading rates were similar to HC-3 and exhibited little variability, ranging from a maximum of 0.0071 mg/kg/wk to a minimum of 0.0021 mg/kg/wk.

In HC-4, the trend of calcium loading rates was similar to HC-1. Calcium loading rates peaked at 2.14 mg/kg/wk at week 2 and then decreased to a minimum of 0.77 mg/kg/wk at week 19 (Appendix 2.8). After week 19, the calcium loading rates steadily increased to 2.5 mg/kg/wk at week 61. The magnesium loading rates in HC-1 showed a similar response as calcium loading rates (Appendix 2.8). Magnesium loading rates were approximately one-half of calcium loading rates. In the absence of increased sulphate loading rates, the increase in calcium and magnesium loading rates are attributed to non-acid neutralization carbonate dissolution (URS, 2009i).

Aluminum loading rates were initially high, at 0.298 mg/kg/wk at week 1. Over 63 weeks of laboratory weathering, aluminum loading rates gradually decreased to 0.009 mg/kg/wk at week 63. The initial peaks in aluminum loading rates up to week 21 are likely an artifact of the laboratory weathering and due to the flushing of readily soluble aluminosilicate phases. The variability in aluminum loading rates after week 21 suggests that aluminosilicate mineral solution-dissolution may have occurred over the remaining period up to 63 weeks (URS, 2009i).

Selenium loading rates began at 0.0005 mg/kg/wk for HC-4, which was significantly lower than was measured for HC-1 and HC-2 and peaked in week 4 (0.0012 mg/kg/wk) (Appendix 2.8). Thereafter, selenium loading rates quickly decreased and reached near steady-state levels by week 18. Between week 18 and kinetic test termination at week 63, the selenium loading rates were extremely low, decreasing from 0.0006 mg/kg/wk to 0.00005 mg/kg/wk. The pattern of selenium loading rates for HC-4 were similar to HC-2, but were approximately three times lower in the first 12 weeks and approximately six times lower near termination of the laboratory weathering. The lower selenium loading initially may have been influenced in part by the presence of dolomite rock fragments that provided micro-scale and/or meso-scale pH control on molybdenum dissolution.

2.8.1.4.4 Post-Kinetic Static Test and Shake Flask Extraction (SFE) Results

Humidity Cell 1 – AR Composite

The post-test ABA results of Humidity Cell 1 (HC-1) are similar to pre-kinetic ABA test results, as previously discussed. The post-kinetic test Humidity Cell 1 (HC-1) had a total sulphur content of 0.31 % by weight, a sulphate sulphur content of <0.01 % by weight, a sulphide sulphur content of 0.29 % by weight, and a insoluble sulphur content of 0.02 % by weight (Appendix 2.8). The corresponding Acid Generation Potential (AGP) was 9.1 kg $CaCO_3$ per tonne. The total inorganic carbon (TIC) content was 0.37 weight % and the carbonate ANP was 30.8 kg $CaCO_3$ per tonne. This carbonate ANP value correlates with a modified Sobek ANP of 31.8 kg $CaCO_3$ per tonne. The Neutralization Potential Ratio (NPR) was 3.5 compared to a pre-kinetic NPR of 3.6. The NPR was 3.5 and therefore the sample is considered be PAG (URS, 2009i).

Humidity Cell 2 – ORE/AR Composite

The post-test ABA results of Humidity Cell 2 (HC-2) are similar to pre-kinetic ABA test results, as previously discussed. The post-kinetic test Humidity Cell 2 (HC-2) had a total sulphur content of 0.37 % by weight, a sulphate sulphur content of <0.01 % by weight, a sulphide sulphur content of 0.31 % by weight, and a insoluble sulphur content of 0.06 % by weight (Appendix 2.8). The corresponding Acid Generation Potential (AGP) was 9.7 kg $CaCO_3$ per tonne. The total inorganic carbon (TIC) content was 0.27 weight % and the carbonate ANP was 22.5 kg $CaCO_3$ per tonne. This carbonate ANP value was lower than the modified Sobek ANP of 45.9 kg $CaCO_3$ per tonne. The Neutralization Potential Ratio (NPR) was 4.7 compared to a pre-kinetic NPR of 7.8. The HC-2 sample is considered to be non-acid generating (NAG), as the NPR is greater than 4.

Humidity Cell 3 – ORE/LS Composite

The post-test ABA results of Humidity Cell 3 (HC-3) are similar to pre-kinetic ABA test results, as previously discussed. The post-kinetic test Humidity Cell 3 (HC-3) had a total sulphur content of 0.56 % by weight, a sulphate sulphur content of <0.01 % by weight, a sulphide sulphur content of 0.56 % by weight, and a insoluble sulphur content of <0.01 % by weight (Appendix 2.8). The corresponding Acid Generation Potential (AGP) was 17.5 kg $CaCO_3$ per tonne. The total inorganic carbon (TIC) content was 4.2 weight % and the carbonate ANP was 350 kg $CaCO_3$ per tonne. This carbonate ANP value correlated strongly with the modified Sobek ANP of 355.8 kg $CaCO_3$ per tonne. The Neutralization Potential Ratio (NPR) was 20.3 compared to a pre-kinetic NPR of 40.7 (URS, 2009i). The HC-3 sample may be considered to be non-acid generating (NAG) as the NPR is greater than 4.

Humidity Cell 4 – OB/LS/FS/AR/ORE Composite

The post-test ABA results of Humidity Cell 4 (HC-4) are similar to pre-kinetic ABA test results, as previously discussed. The post-kinetic test Humidity Cell 4 (HC-4) had a total sulphur content of 0.42 % by weight, a sulphate sulphur content of <0.01 % by weight, a sulphide sulphur content of 0.4 % by weight, and a insoluble sulphur content of 0.02 % by weight (Appendix 2.8). The corresponding Acid Generation Potential (AGP) was 12.5 kg $CaCO_3$ per tonne. The total inorganic carbon (TIC) content was determined to be 2.53 weight % and the carbonate ANP was 210.8 kg $CaCO_3$ per tonne. This carbonate ANP value correlated strongly with the modified Sobek ANP of 211.3 kg $CaCO_3$ per tonne. The Neutralization Potential Ratio (NPR) was 16.9 compared to a pre-kinetic NPR of 10.4 (URS, 2009i). The HC-4 sample may be considered to be non-acid generating (NAG), since the NPR is greater than 4.

Shake Flask Extraction Results

Post-kinetic testing Shake Flask Extraction (SFE) testing was completed to determine what readily soluble residuals remained with the humidity cell rock fragments at termination. Only aluminum in HC-4 (175 µg/L) was detected in SFE leachate at concentrations greater than Manitoba Tier III Water Quality Guidelines and CCME Water Quality Guidelines for the projected of freshwater aquatic life (100 µg/L) (Appendix 2.8).

2.8.1.4.5 Waste Rock Carbonate Molar Ratios, Depletion Rates and Time to Depletion Estimates

Carbonate Molar (Ca + Mg / SO4) Ratios

Carbonate molar ratios (the ratio of (Ca+Mg) to sulphate in the leachates) for the laboratory kinetic humidity cells are shown in Figure 2.8-11. This ratio provides an estimate of the proportion of carbonate that is released (dissolved) in response to sulphide oxidation, and the proportion released due to processes other than acid neutralization when the ratio exceeds 1:1 (URS, 2009i).

After the initial 10 week flushing period, the carbonate molar ratios for all four waste rock humidity cell samples increased over time. Sulphate loading rates (Appendix 2.8) decreased during this span, while calcium and carbonate loading rates increased. Furthermore, by the end of the test the (Ca+Mg)/SO_4 ratio exceeded 1:1 in all the cells. These trends suggest that more carbonate material is being released than can be accounted for solely by the carbonate neutralization of acidity produced by sulphide oxidation. The additional dissolution of carbonate from the humidity cell tests could have resulted from equilibrium dissolution of carbonates in the weekly rinse water in addition to carbonate dissolution due to acid neutralization. Of note, HC-2 showed this trend, yet did not contain limestone material, and the initial TIC content was only 0.24 % by weight. One possible explanation is that because the sulphide content was so low, the available sulphides in the humidity cell were essentially depleted and/or the sulphide oxidation rates had slowed to negligible rates. The release of ANP in the field from the tailings is expected to be significantly slower (URS, 2009i).

Acid Generation Potential Depletion Rates and Timing

The weekly sulphate loading rates determined from humidity cells were used to determine the average rate of sulphide (Acid Generation Potential) depletion in each humidity cell. Based on the humidity cell results, weeks 20 to 63 were considered steady-state or equilibrium conditions and used in acid generation potential depletion rate calculations (Appendix 2.8). In these calculations, the sulphide sulphur values from pre-kinetic static tests of the humidity cell sample materials were used as the initial sulphur concentrations.

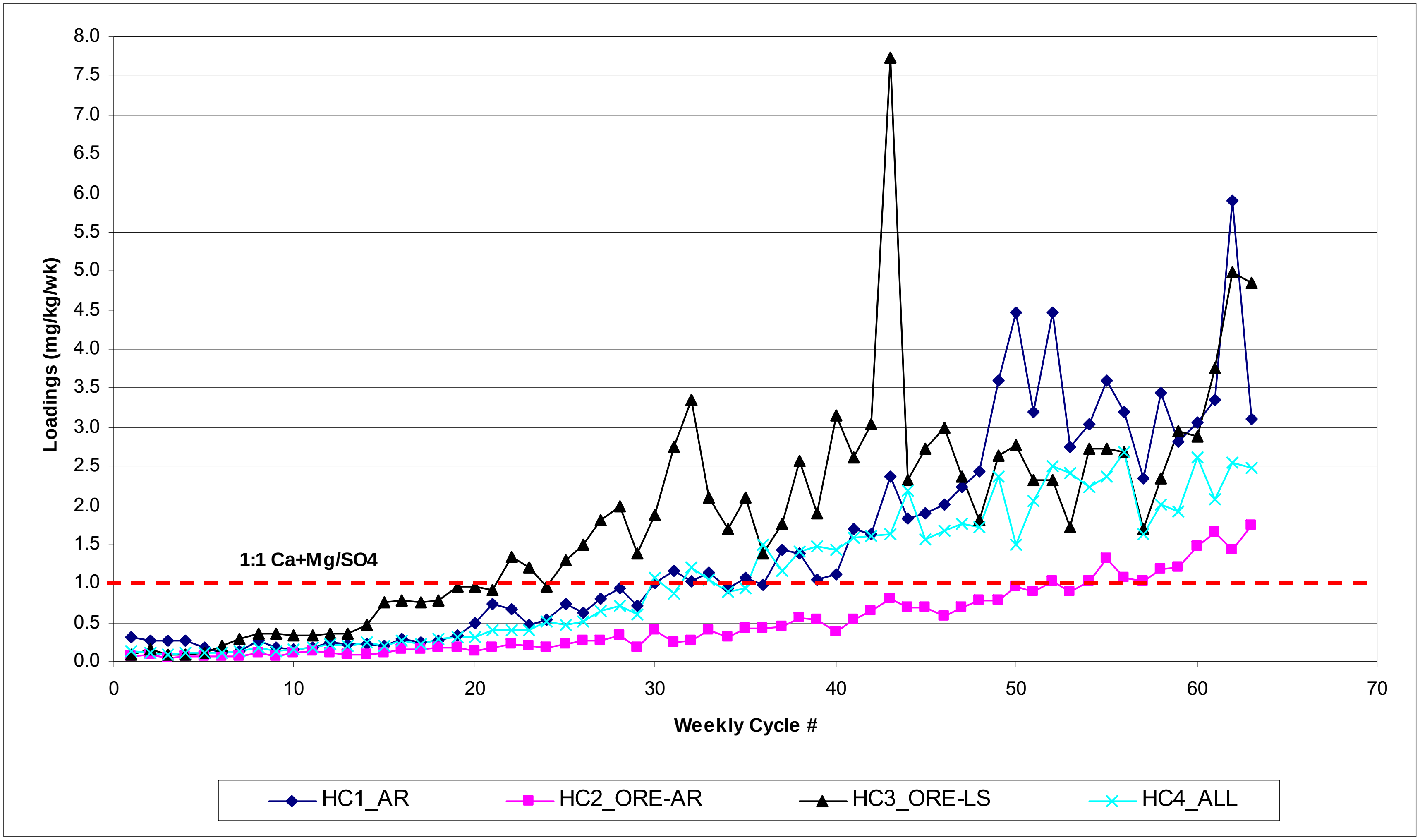


Figure 2.8-11 Ca+Mg/SO_4 Ratios for Minago Phase I Waste Rock Kinetic Tests

The rates of sulphide depletion ranged from a minimum of 0.034 mmol/kg/wk for HC-3 to a maximum of 0.072 mmol/kg/wk for HC-4 (Table 2.8-11). These rates are consistent with the initial sulphide sulphur content. Humidity cell HC-4 contained the highest sulphide content of 0.73 weight % and had an AGP of 22.8 kg $CaCO_3$ per tonne.

Humidity cells HC-1 and HC-2 contained the lowest sulphide sulphur (0.28 weight % and 0.16 weight %, respectively). These humidity cells yielded intermediate sulphide depletion rates of 0.066 mmo/kg/wk (Table 2.8-11). For HC-1, the estimated time to depletion based on the initial sulphide sulphur content is 22 years. For HC-2, the estimated time to depletion based on the initial sulphide sulphur content is 12 years.

Humidity cell HC-3 yielded the lowest sulphide depletion rate of 0.034 mmol/kg/wk. Humidity cell HC-3 contained a mixture of Precambrian basement and Ordovician dolomitic limestone rock fragments. The sulphide sulphur content of this humidity cell was 0.35 weight % and an AGP of 10.9 kg $CaCO_3$ per tonne. However, the total inorganic carbonate (TIC) content of this humidity cell material was 4.55 weight %, with an ANP of 238.1 kg $CaCO_3$ per tonne. Results suggest that the limestone fragments in humidity cell HC-3 likely provided micro-scale neutralization. Alternatively, the availability of the sulphides in this sample may have been lower than in other samples, resulting in lower sulphide exposure to air and water and thus oxidation. In the case of limestone neutralization, this illustrates the potential effectiveness of limestone in waste rock to neutralize and minimize migration of secondary constituents from sulphide oxidation. For HC-3, the estimated time to sulphide depletion based on the initial sulphide sulphur content and the laboratory kinetic rate of 0.034 mmol/kg/wk is 58 years (Table 2.8-11).

For HC-4, the estimated time to depletion based on the initial sulphide sulphur content and the laboratory kinetic rate of 0.072 mmol/kg/wk is 58 years (Table 2.8-11).

Acid Neutralization Potential - Depletion Rates and Timing

The weekly calcium and magnesium loading rates determined from humidity cells were used to determine the average rate of carbonate ANP depletion in each humidity cell. Based on the humidity cell results, weeks 20 to 63 were considered steady-state or equilibrium conditions and used in calculations for the depletion rate of acid neutralization.

The rates of carbonate depletion ranged between 0.05 mmol/kg/wk for HC-2 and 0.11 mmol/kg/wk for HC-1 (Table 2.8-11). The carbonate depletion rates show no apparent correlation with initial TIC content or ANP values. The estimated carbonate depletion rate for HC-1 is 0.11 mmol/kg/wk, and the initial TIC content was relatively low at 0.35 % by weight. Based on this calculated depletion rate from laboratory kinetic tests, the time to carbonate depletion is estimated to be 49 years. For HC-2, the initial TIC content was 0.24 % by weight. Based on the laboratory kinetic humidity cell test results, the carbonate depletion rate is calculated to be 0.05 mmol/kg/wk, and the time to carbonate depletion estimated to be 83 years (URS, 2009i).

Table 2.8-11 Hunidity Cell Depletion Rates for Waste Rock

SULPHIDE DEPLETION CALCULATIONS

Humidity Cell ID	Sample Type			Initial Sulphide-S					Amount of S remaining after 63 weeks (mmol)	Average rate of Sulphide-S depletion per week (based on 44 steady state wks)	Weeks until 0 mmol S	Years until 0 mmol S
			HC mass (g)	(%)	(mg/kg)	(g/kg)	(mol)	(mmol)		(mmol/kg/wk)		
HC-1 AR	Waste Rock (Drill Core)	AR Composite (N0727+N0728+N0729+N036)	1000	0.28	2800	2.8	0.09	87.34	75.39	0.066	1134	22
HC-2 ORE/AR	Waste Rock (Drill Core)	ORE/AR Composite (N0727+N0728+N0729+N036)	1000	0.16	1600	1.6	0.05	49.91	40.66	0.066	617	12
HC-3 ORE/LS	Waste Rock (Drill Core)	ORE/LS Composite (N0727+N0728+N0729+N036)	1000	0.35	3500	3.5	0.11	109.17	103.48	0.034	3022	58
HC-4 OB/LS/FS/AR/ORE	Waste Rock (Drill Core)	OB/LS/FS/AR/ORE Comp. (N0727+N0728+N0729+N0736)	1000	0.73	7300	7.3	0.23	227.70	217.23	0.072	3009	58

CARBONATE DEPLETION CALCULATIONS

Humidity Cell ID	Sample Type			Initial Total Carbonate			Initial Total Ca in Carbonate					Amount of Ca+Mg remaining after 63 weeks (mmol)	Average rate of Ca+Mg depletion per week (based on 44 steady state wks)	Weeks until 0 mmol Ca+Mg	Years until 0 mmol Ca+Mg
				TIC	Carb NP	Carb NP	Total Ca								
			HC mass (g)	(%)	(kg CaCO3/t)	(%)	(%)	(mg/kg)	(g/kg)	(mol)	(mmol)		(mmol/kg/wk)		
HC-1 AR	Waste Rock (Drill Core)	AR Composite (N0727+N0728+N0729+N036)	1000	0.35	29.2	2.9	1.17	11691	11.69	0.29	291.69	284.57	0.11	2523	49
HC-2 ORE/AR	Waste Rock (Drill Core)	ORE/AR Composite (N0727+N0728+N0729+N036)	1000	0.24	20.0	2.0	0.80	8009	8.01	0.20	199.81	197.27	0.05	4292	83
HC-3 ORE/LS	Waste Rock (Drill Core)	ORE/LS Composite (N0727+N0728+N0729+N036)	1000	4.55	379.2	37.9	15.18	151828	151.83	3.79	3788.13	3783.85	0.08	49619	954
HC-4 OB/LS/FS/AR/ORE	Waste Rock (Drill Core)	OB/LS/FS/AR/ORE Comp. (N0727+N0728+N0729+N036)	1000	2.62	218.3	21.8	8.74	87426	87.43	2.18	2181.30	2175.84	0.10	21676	417

Humidity Cell ID	Sample Type		ABA Results				Total Metals	Average Sulphide-S Depletion Rate	Time to Sulphide-S Depletion	Average Carbonate Depletion Rate	Time to Carbonate Depletion	Average Carbonate Molar Ratio	Expected to be Acid Generating?
			ANP	AGP	NNP	NPR	Ni (ppm)	(mmol/kg/wk)	(years)	(mmol/kg/wk)	(years)		
HC-1 AR	Waste Rock (Drill Core)	AR Composite (N0727+N0728+N0729+N036)	31.9	8.8	23.2	3.6	1003	0.07	22	0.11	49	1.70	**No**
HC-2 ORE/AR	Waste Rock (Drill Core)	ORE/AR Composite (N0727+N0728+N0729+N036)	39.0	5.0	34.0	7.8	1428	0.07	12	0.05	83	0.70	**No**
HC-3 ORE/LS	Waste Rock (Drill Core)	ORE/LS Composite (N0727+N0728+N0729+N036)	443.5	10.9	432.6	40.5	913	0.03	58	0.08	954	2.23	**No**
HC-4 OB/LS/FS/AR/ORE	Waste Rock (Drill Core)	OB/LS/FS/AR/ORE Comp. (N0727+N0728+N0729+N036)	238.1	22.8	215.3	10.4	1104	0.07	58	0.10	417	1.39	**No**

Source: adapted from URS (2009i)

The initial TIC content of HC-3 was the highest of all humidity cells at 4.55 weight %. The rate of carbonate depletion calculated from the laboratory kinetic humidity cell test was 0.08 mmol/kg/wk and the estimated time to carbonate depletion is 954 years.

Lastly for HC-4, the initial TIC content was 2.62 weight %. Based on this initial TIC content and calcium and magnesium loading rates measured for HC-4, the rate of carbonate depletion is calculated to be 0.10 mmol/kg/wk and carbonate depletion was estimated to occur in 417 years for HC-4.

2.8.1.5 Preliminary Site-specific NPR Criterion

Non-basement rock materials (e.g., overburden, limestone, and sandstone) appear to contain negligible to low sulphide sulphur concentrations and low to high carbonate concentrations. Thus, these materials do not appear to have significant ARD/ML potential, and these materials can be handled as NAG (URS, 2009i).

With respect to Precambrian rock materials (URS, 2009i):

- altered and ore grade basement lithologies appear to be PAG;
- mafic metavolcanic and metasedimentary material appear to be PAG; and
- granite and serpentinite are NAG in general, but there are localized areas in these lithologies with low NPR that are PAG.

The kinetic tests appear to indicate that combining limestone with PAG material would mitigate ARD in waste rock piles assuming that the two materials are well mixed. If limestone is inadequately mixed with PAG material, ARD could develop in localized areas (URS, 2009i).

Despite elevated concentrations of chromium, nickel, sulphur, antimony, thorium, and uranium throughout the Precambrian basement, the potential metal leaching predicted from these lithologies is very low to low based upon the kinetic test results (URS, 2009i). NPRs in PAG material ranged between 0.1 and 3.7.

Based on the results from HC-1 and HC-2, the carbonate molar ratios indicate a preliminary site-specific NPR of 1.7 is appropriate for segregating PAG from NAG materials. Therefore, URS (2009i) recommended that a preliminary site-specific NPR criterion of 1.7 be used to identify PAG waste materials at the Minago Project.

2.8.2 Geochemical Assessment of Tailings

The tailings assessment was intended to determine the ARD/ML potential of tailings material. The results were used to determine whether subaqueous tailings storage will be sufficient to prevent ARD/ML from the tailings material. The Minago Project tailings geochemical assessment had two parts: a static testing program and a kinetic testing program. Based on discussions with representatives of VNI and Wardrop, the basis of kinetic testing of tailings was that tailings would be contained in a flooded tailings impoundment.

The objectives of the static program were to determine 1) whether representative tailings samples will be PAG or acid-neutralizing, and 2) the total ML potential within those samples. Based on static test results for the tailings samples and the very low sulphur content, it was not considered necessary to calculate primary sulphide oxidation, acid generation, carbonate dissolution, or acid neutralization rates (URS, 2009i). Therefore, the objectives of the tailings kinetic testing program were to assess 1) the geochemical stability of tailings under saturated conditions and 2) potential leachate water quality and chemical loading rates from the tailings.

2.8.2.1 Analytical Methods

In August 2007, after conferring with Victory Nickel and Wardrop about the Minago Project metallurgical testing program, URS requested SGS-CEMI to produce a master tailings composite sample from their 2006 lock cycle metallurgical testing. This sample was called the “2006 Master Lock Cycle Composite” sample.

In 2007, Wardrop completed a second round of bulk metallurgical testing, which was considered to be more representative of the nickel grades within the Minago deposit. The lock cycle test cleaner scavenger and rougher rejects were considered more representative of the potential tailings geochemistry at Minago. The following two samples were produced for static testing by SGS-CEMI (URS, 2009i):

- The “2007 0.3% Ni Lock Cycle Tails” sample contained 0.3 % by weight nickel grade material; and
- The “2007 Master Lock Cycle Composite” sample contained a composite of the master lock cycle material.

2.8.2.1.1 Static Test Program

Static testing for the Minago Project involved subjecting test specimens to Acid-Base Accounting (ABA) tests and total metal content analysis by inductively-coupled atomic emissions spectrometry (ICP-AES). The static tests were conducted by SGS - Canadian Environmental and Metallurgical Inc. (SGS-CEMI), located in Burnaby, British Columbia. The static testing included the following parameters:

- Fizz Test;
- Paste pH;
- Weight % CO_2, which was converted to Total Inorganic Carbonate (TIC) content expressed as $CaCO_3$ equivalents;
- Total Sulphur content, expressed as weight %;

- Sulphate Sulphur content, expressed as weight %;
- Insoluble sulphur content, expressed as % by weight;
- Sulphide sulphur content, expressed as % by weight and determined from the difference between total sulphur and sulphate sulphur plus insoluble sulphur (where sulphate and insoluble sulphur were analyzed); and
- ANP by both modified Sobek and standard Sobek methods.

From the analytical results the following ABA parameters were calculated:

- AGP was calculated from sulphide sulphur content;
- Net-ANP was calculated from the difference between modified Sobek ANP and AGP; and
- NPR was calculated as the ratio of the modified Sobek ANP to AGP.

2.8.2.1.2 Total Metals

The three tailings lock cycle composite samples were submitted to SGS-CEMI for analysis of total metals by ICP-AES following digestion by aqua regia.

2.8.2.1.3 Particle Size Analysis

The 2007 0.3% Ni Lock Cycle Tails sample was submitted for particle size analysis to classify the material based on the Unified Soil Classification System.

2.8.2.1.4 Leachate Extraction Tests

The three tailings lock cycle composite samples were submitted to SGS-CEMI for shake flask extraction tests to determine readily leachable constituents. The shake flask extraction tests were the first step in determining the likelihood of metal leaching from potential tailings material.

2.8.2.1.5 Mineralogical Analysis

A sub-sample of the 2007 0.3% Ni Lock Cycle Tails sample was submitted to the Department of Earth and Ocean Sciences at the University of British Columbia for mineralogic analysis with X-ray diffraction using the Rietveld method. Sub-samples of both the 2006 Master Lock Cycle Composite and 2007 Master Lock Cycle Composite samples were submitted to SGS-CEMI for mineralogical analysis using QEMSCAN and Scanning Electron Microscope equipped with Energy Dispersive Spectrometer (URS, 2009i).

2.8.2.1.6 Kinetic Test Program

Kinetic testing of tailings was carried out under saturated conditions as the tailings are planned to be contained in a flooded tailings impoundment. The objectives of the conducted kinetic testing program were to:

- Assess the geochemical stability of tailings under saturated conditions; and if possible;
- Assess the relative rates of acid generation and acid neutralization of tailings;
- Assess the relative timing of complete sulphide oxidation (acid generation) and complete weathering/dissolution of carbonate minerals (acid neutralization) and if acid neutralization is exhausted prior to acid generation, the potential onset of Acid Rock Drainage and Metal Leaching (ARD / ML);
- Predict leachate water quality and loadings from tailings; and
- Predict final effluent discharge water quality and, if necessary, the potential requirement for effluent treatment.

Due to sample availability, only the 2007 0.3% Ni Lock Cycle Tails sample was submitted to SGS-CEMI for laboratory kinetic subaqueous column tests, including (adapted from URS, 2009i):

- Biweekly cycling with 100 ml of deionized water added on even weeks and 160 ml of deionized water added on odd weeks for 54 weeks;
- Weekly measurement of pH, oxidation reduction potential, specific conductivity and sulphate;
- Biweekly measurement of acidity, alkalinity, and dissolved oxygen on odd weeks; and
- Weekly analysis of total metals by ICP-AES.

2.8.2.2 Results

2.8.2.2.1 Static Test Results for Tailings

Results of the static test program on tailings are summarized below and in Table 2.8-12. Detailed results are provided in Appendix 2.8 and elsewhere (URS, 2009i).

2006 Master Lock Cycle Composite

The 2006 Master Lock Cycle Composite sample had a total sulphur content of 0.12 % by weight, of which 0.03 % by weight was sulphate sulphur and 0.02 % by weight was insoluble sulphur (Table 2.8-12). By difference, the sulphide sulphur content was 0.07 % by weight, equating to an AGP of 2.2 kg $CaCO_3$/tonne. The TIC content was 0.41 % by weight, equating to a carbonate ANP of 34.2 kg $CaCO_3$/tonne. The Sobek ANP was 433.4 kg $CaCO_3$/tonne, and the modified

Sobek ANP was 72.4 kg $CaCO_3$/tonne. The carbonate ANP and modified Sobek ANP values were in reasonable agreement with one another. However, the standard Sobek method significantly overestimated the sample's ANP. URS (2009i) attributed the higher ANP value by the standard Sobek method to dissolution of low soluble carbonate minerals and aluminosilicate minerals. The NPR based on the modified Sobek ANP was 34.1, and the sample material is considered to be NAG.

Table 2.8-12 Static Test Results for Minago Tailings

										Standard Sobek			Modified Sobek		
Sample ID	paste pH	Fizz Test	Total Inorganic Carbon (TIC) (wt%)	Carbonate Acid Neutralization Potential (kg $CaCO_3$/tonne)	Total Sulphur (wt%)	Sulphate Sulphur (wt%)	Sulphide Sulphur (wt%)*	Insoluble Sulphur (wt%)*	AGP** (kg $CaCO_3$/tonne)	ANP (kg $CaCO_3$/tonne)	Net-ANP (kg $CaCO_3$/tonne)	NPR (ANP/AGP)	ANP (kg $CaCO_3$/tonne)	Net-ANP (kg $CaCO_3$/tonne)	NPR (ANP/AGP)
Tails Composite - 2007 [1]	8.38		0.38	31.7	0.12	0.02	0.04	0.06	1.3	455.9	454.7	364.7	74.7	73.5	59.8
Tails Composite - 2007 [2]	8.41	None	0.46	38.3	0.12	0.05	0.07	<0.01	2.2	397.2	395.0	181.6	76.5	74.3	35.0
Tails Composite - 2006 [3]	8.70	Slight	0.41	34.2	0.12	0.03	0.07	0.02	2.2	433.4	431.2	198.1	74.6	72.4	34.1
Detection Limits	0.1		0.03	---	0.02	0.01	---	---	---	0.1	0.1	---	0.1	0.1	---

Notes:

* Based on difference between total sulphur and sulphate-sulphur.

** Based on sulphide-sulphur.

AGP = acid generation potential in kilograms $CaCO_3$ equivalent per tonne of material.

ANP = acid neutralization potential in kilograms $CaCO_3$ equivalent per tonne of material.

NPR = ANP / AGP

[1] = 2007 Master lock cycle composite tailings sample (1st cleaner and rougher tailings).

[2] = 2007 0.3 % Ni lock cycle composite tailings sample (1st cleaner and rougher tailings).

[3] = 2006 Master lock cycle composite tailings sample (1st cleaner and rougher tailings).

Source: URS, 2009i

2007 0.3% Nickel Lock Cycle Composite

The static test results for the 2007 0.3% Ni Lock Cycle Tails sample had a total sulphur content of 0.12 % by weight, of which 0.05 % by weight was sulphate sulphur and <0.01 % by weight was insoluble sulphur (Table 2.8-12). By difference, the sulphide sulphur content was 0.07 % by weight, equating to an AGP of 2.2 kg $CaCO_3$/tonne. The TIC content was 0.46 % by weight, equating to a carbonate ANP of 38.3 kg $CaCO_3$/tonne. ANP by the standard Sobek method was 397.2 kg $CaCO_3$/tonne, and the modified Sobek ANP was 76.5 kg $CaCO_3$/tonne. Again, the standard Sobek method significantly overestimated the sample's ANP. The NPR based on the modified Sobek ANP was 35.0, and the sample material is considered to be NAG.

2007 Master Lock Cycle Composite

The 2007 Master Lock Cycle Composite sample had a total sulphur content of 0.12 % by weight, of which 0.02 % by weight was sulphate sulphur and 0.06 % by weight was insoluble sulphur (Table 2.8-12). By difference, the sulphide sulphur content was 0.04 % by weight equating to an AGP of 1.3 kg $CaCO_3$/tonne. The TIC content was 0.38 % by weight, equating to a carbonate ANP of 31.7 kg $CaCO_3$/tonne. ANP by the standard Sobek method was 455.9 kg $CaCO_3$/tonne, and the modified Sobek ANP was 74.7 kg $CaCO_3$/tonne. Again, the standard Sobek method significantly overestimated the sample's ANP. The NPR based on the modified Sobek ANP was 59.8, and the sample material is considered to be NAG per tonne and the modified Sobek ANP was 59.8 kg $CaCO_3$ per tonne. The Neutralization Potential Ratio based on the modified Sobek ANP was 59.8.

Comparison of Tailings Static Test Results

The static test results from all three samples show a reasonable correlation of both the sulphur species content in the tailings and Acid Generation Potential (AGP), and the TIC and Acid Neutralization Potential ANP. Static test results are also in reasonable agreement with the 2006 tailings lock cycle composite tested by SGS Lakefield (Appendix 2.8). The tailings sample tested by SGS Lakefield had 0.7 weight % total sulphur and <0.04 weight % sulphate sulphur and a modified Sobek ANP of 88.8 kg $CaCO_3$ per tonne.

Based on the static test results, the metallurgical lock cycle testing on two (2) bulk samples from the Minago deposit recovered the majority of sulphide minerals as evidenced by the very low sulphide sulphur content in the cleaner scavenger and rougher tailings tested. Based on the low sulphide sulphur content and high carbonate content, the tested tailings samples are considered to be non-acid generating (NAG).

2.8.2.2.2 Total Metals

The total metal concentrations in the tested tailings are shown in Table 2.8-13. Elemental concentrations were compared to normal elemental concentrations in typical ultramafic rock types

Table 2.8-13 Total Elements Minago Tailings

Sample #	Rock Type	Ag	Al	As	Ba	Be	Bi	Ca	Cd	Co	Cr	Cu	Fe	Hg	K	La	Mg	Mn
		ppm	%	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	ppm	%	ppm	%	ppm
2007 Tails Composite [1]	Tailings	0.1	1.14	1.3	**191**	1	0.4	0.74	0.1	57.8	347	**69.7**	5.27	<0.1	0.57	47	**>10.00**	511
2007 Tails Composite [2]	Tailings	<0.2	0.85	**<5**	**192**	0.6	<5	0.92	2	93	259	**8**	5.44	<1	0.5	59	**>15.00**	524
2006 Tails Composite [3]	Tailings	<0.2	0.89	**7**	**166**	<0.5	<5	0.92	2	48	319	**46**	4.51	<1	0.35	40	**>15.00**	435
Ultrabasic [4]		0.06	2.00	1	0.4	na	na	2.50	na	150	1600	10	9.43	na	40	na	2.04	1620
3X Ultrabasic		0.180	6.00	**3**	**1.2**			7.50		450	4800	**30**	28.3		120		**6.12**	4860

Sample #	Rock Type	Mo	Na	Ni	P	Pb	S	Sb	Sc	Sr	Th	Ti	Tl	U	V	W	Zn	Zr
		ppm	%	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	ppm	ppm
2007 Tails Composite [1]	Tailings	1.2	0.05	>1000.0	0.025	1.6	**0.14**	<0.1	5.4	**53**	**4.7**	0.024	0.1	**3.9**	20	4.3	72	2.7
2007 Tails Composite [2]	Tailings	<2	0.03	2456	65	**8**	**0.15**	**6**	4	**29**	**<5**	0.02	<10	**26**	16	<10	60	6
2006 Tails Composite [3]	Tailings	<2	0.05	2292	111	**6**	**0.13**	**9**	5	**11**	**8**	0.03	<10	**20**	30	<10	22	6
Ultrabasic [4]		0.3	0.42	2000	220	1	0.03	0.10	15	1	0.004	0.03	1	0.001	40	0.7	50	45
3X Ultrabasic		0.9	1.26	6000	660	**3**	**0.09**	**0.30**	45	**3**	**0.012**	0.09	**3**	**0.003**	120	2.1	150	135

Notes:

[1] 2007 Master lock cycle composite tailings sample (1st cleaner and rougher tailings).

[2] 2007 0.3 % Ni lock cycle composite tailings sample (1st cleaner and rougher tailings).

[3] 2006 Master lock cycle composite tailings sample (1st cleaner and rougher tailings).

[4] Source: Turekian and Wedepohl (1961)

Source: URS (2009i)

for screening purposes (Turekian and Wedepohl, 1961). For screening purposes, levels greater than three times the normal concentration was considered to be elevated. The results indicate elevated concentrations of arsenic, barium, copper, lead, antimony, strontium, thallium, and uranium. In general, there was reasonable agreement in concentrations of the same element in all three tailings samples. The full laboratory analytical results are provided in Appendix 2.8.

2.8.2.2.3 Particle Size Analysis

Results of the grainsize analysis of the 2007 0.3% nickel lock cycle composite sample are given in Appendix 2.8. The tailings particle size fell within three general ranges:

- 14%: +60 mesh or 0.25 mm diameter;
- 25%: -140 mesh (0.106 mm) to +270 mesh (0.053 mm); and
- 35%: -325 mesh (0.044 mm).

Based on the USCS soil classification system the tailings are considered to be primarily composed fine sand, silt and clay sized particles.

2.8.2.2.4 Leachate Extraction Results

The results of shake flask extraction tests are shown in Table 2.8-14. The full laboratory analytical results are included in Appendix 2.8. Selenium ranged between 0.9 and 2.08 µg/L; boron ranged between 1,750 and 3,350 µg/L; and nitrite ranged between 0.021 and 0.184 µg/L. The nitrite may have originated from the process chemicals used during the lock cycle testing. Only selenium and nitrite concentrations slightly exceeded Manitoba guideline limits.

Further test work could identify the possible sources of nitrite and assess whether mill process water effluent could contain similar nitrite levels.

2.8.2.2.5 Mineralogical Analysis

The minerals identified using X-ray diffraction in the 2007 0.3% Ni Lock Cycle Tails sample were (in decreasing abundance): antigorite, lizardite, phlogopite, talc, magnetite, dolomite, quartz, vermiculite, and calcite. These minerals reflect mineralogy of altered granite and serpentinite of the Minago deposit. The slower-reacting carbonate mineral dolomite was found to be more abundant than calcite in the tailings sample. The full analytical report is provided in URS (2009i).

The mineralogy identified in both Master Lock Cycle Composite samples using SEM-EDS was consistent with the Rietveld X-ray diffraction analysis. The following non-sulphide minerals were identified (in decreasing abundance): serpentinite, talc, amphibole, phlogopite, carbonate, olivine, chlorite, and quartz. Sulphide minerals identified by Scanning Electron Microscope equipped with Energy Dispersive Spectrometer included millerite, pentlandite, chalcopyrite, pyrite and violarite.

Table 2.8-14 Shake Flask Extraction Test Results for Minago Tailings

Sample ID			**1st Cleaner + Rougher Tails Composite**	**1st Cleaner + Rougher Tails Composite**	**1st Cleaner + Rougher Tails Composite**	**REGULATIONS**			
Parameter	**Method**	**Units**	**2006 - Master**	**2007 - 0.3% Ni**	**2007 - Master**	**Manitoba**	**Tier**	**CCME**	**MMER [1]**
Volume Nanopure water		mL	1800	-	1800				
Sample Weight		g	600	-	600				
pH	meter		8.08	8.3	8.02	6.5-8.5	III	6.5-9	6.5-9
Redox	meter	mV	411	435	374				
Conductivity	meter	uS/cm	590	803	522				
Acidity (to pH 4.5)	titration	mg CaCO3/L	na	na	na				
Total Acidity (to pH 8.3)	titration	mg CaCO3/L	2.5	na	3.2				
Alkalinity	titration	mg CaCO3/L	67.2	94.5	58.4				
Fluoride		mg/L	0.9	0.63	50				
Chloride		mg/L	47.5	114	0.7				
Bromide		mg/L	0.12	1.60	4.1				
Ammonia		mg/L	0.08	0.06	0.04	here: 1.5-8.4	II	19 (as NH_3)	
Nitrite		mg/L	0.184	0.021	<0.5	0.06 (NO2-N)	III	0.06 (NO2-N)	
Nitrate		mg/L	0.07	0.07	<2	10 (as NO3-N)	III		
Sulphate	Turbidity	mg/L	148	176	132	500	III	--	
Dissolved Metals									
Hardness CaCO3		mg/L	165	165	145				
Aluminum Al	ICP-MS	µg/L	2	2.3	8.8	100	III	100	
Antimony Sb	ICP-MS	µg/L	2.21	1.90	0.62	--			
Arsenic As	ICP-MS	µg/L	0.52	0.40	1.30	150[A]	II	5	1000
Barium Ba	ICP-MS	µg/L	37.8	32.0	53.5	--			
Beryllium Be	ICP-MS	µg/L	<0.010	<0.010	0.02	--			
Bismuth Bi	ICP-MS	µg/L	<0.005	<0.005	<0.005	--			
Boron B	ICP-MS	µg/L	1750	3350	2830	5000	III		
Cadmium Cd	ICP-MS	µg/L	0.021	<0.005	0.010	here: 2.9-3.2[B]	II	0.017	
Calcium Ca	ICP-MS	µg/L	40200	16500	17600	--			
Chromium Cr	ICP-MS	µg/L	1.4	0.1	4.8	here: 100.5-111.7[C]	II	8.9 [3]	
Cobalt Co	ICP-MS	µg/L	0.124	0.1	0.287	--			
Copper Cu	ICP-MS	µg/L	1.44	0.3	0.32	here: 12.3-13.7[D]	II	3[2]	600
Iron Fe	ICP-MS	µg/L	3	<1	2	300	III	300	
Lead Pb	ICP-MS	µg/L	0.12	0.018	0.014	here: 3.8-4.3[E]	II	here: 4[2]	400
Lithium Li	ICP-MS	µg/L	26.2	33.5	49.2	--			
Magnesium Mg	ICP-MS	µg/L	15700	22400	24500	--			
Manganese Mn	ICP-MS	µg/L	1.25	1.4	1.96	--			
Mercury Hg	CVAA	µg/L	<0.01	<0.01	<0.01	0.1	III	0.026	
Molybdenum Mo	ICP-MS	µg/L	9.87	10.4	12.3	73	III	73	
Nickel Ni	ICP-MS	µg/L	22.1	8.8	42.5	here: 71.2-79.4[F]	II	here: 110[2]	1000
Potassium K	ICP-MS	µg/L	16400	20100	17300	--			
Selenium Se	ICP-MS	µg/L	1.71	0.9	2.08	1	III	1	
Silicon Si	ICP-MS	µg/L	2090	1650	2690	--			
Silver Ag	ICP-MS	µg/L	0.006	<0.005	0.01	0.1	III	0.1	
Sodium Na	ICP-MS	µg/L	48200	105000	40600	--			
Strontium Sr	ICP-MS	µg/L	307	243	306	--			
Sulphur (S)	ICP-MS	µg/L	57000	46000	58000	--			
Thallium Tl	ICP-MS	µg/L	0.287	0.122	0.327	0.8	III	0.8	
Tin Sn	ICP-MS	µg/L	0.07	0.01	0.02	--			
Titanium Ti	ICP-MS	µg/L	<0.5	<0.5	<0.5	--			
Uranium U	ICP-MS	µg/L	0.049	0.073	0.045	--			
Vanadium V	ICP-MS	µg/L	<0.2	<0.2	<0.2	--			
Zinc Zn	ICP-MS	µg/L	0.8	0.8	0.5	here: 161.9-180.6[G]	II	30	1000
Zirconium Zr	ICP-MS	µg/L	<0.1	<0.1	<0.1	--	III		
Ra-226		Bq/L	na	0.02	0.04	0.6	III		0.37

Notes:

[1] monthly mean 2002 Metal Mining Effluent Regulations (MMER) requirements also include cyanide, TSS and acute toxicity.

[2] guideline concentration in CCME Water Quality Guidelines for the protection of freshwater aquatic life (Dec. 2007) depends on hardness.

[3] chromium III

Manitoba Water Quality Standards, Objectives, and Guidelines (Williamson, 2002):

A Arsenic limits: 0.15 mg/L for averaging duration 4 days (4-Day, 3-Year or 7Q10 Design Flow); 0.34 mg/L for averaging duration 1 hr (1-Day, 3-Year or 1Q10 Design Flow)

B Cadmium limits: [e{0.7852[ln(Hardness)]-2.715}]×[1.101672-{ln(Hardness)(0.041838)}] for 4 days averaging duration.
[e{1.128[ln(Hardness)]-3.6867}]×[1.136672-{ln(Hardness)(0.041838)}] for 1 hour averaging duration.

C Chromium limits: Chromium III: [e{0.8190[ln(Hardness)]+0.6848}]×[0.860] for 4 days averaging duration.
Chromium III: [e{0.8190[ln(Hardness)]+3.7256}]×[0.316] for 1 hour averaging duration.
Chromium VI: 0.011 mg/L for averaging duration 4 days (4-Day, 3-Year or 7Q10 Design Flow);
0.016 mg/L for averaging duration 1 hr (1-Day, 3-Year or 1Q10 Design Flow)

D Copper limits: [e{0.8545[ln(Hardness)]-1.702}]×[0.960] for 4 Days hour averaging duration.
[e{0.9422[ln(Hardness)]-1.700}]×[0.960] for 1 hour averaging duration.

E Lead limits: [e{1.273[ln(Hardness)]-4,705}]×[1.46203 -{ln(Hardness)(0.145712)}] for 4 Days averaging duration.
[e{1.273[ln(Hardness)]-1.460}]×[1.46203 -{ln(Hardness)(0.145712)}] for 1 hour averaging duration.

F Nickel limits: [e{0.8460[ln(Hardness)]+0.0584}]×[0.997] for 4 Days averaging duration.
[e{0.8460[ln(Hardness)]+2.255}]×[0.998] for 1 hour averaging duration.

G Zinc limits: [e{0.8473[ln(Hardness)]+0.884}]×[0.976] for 4 Days averaging duration.
[e{0.8473[ln(Hardness)]+0.884}]×[0.978] for 1 hour averaging duration.

Source: adapted from URS, 2009i

All were less than 1% in abundance. An important note regarding all sulphide minerals identified are their extremely small size, ranging up to 400 µm but typically ranging from 5 to 25 µm.

2.8.2.2.6 Kinetic (Subaqueous Column) Test Results for Tailings

Weekly loading rates, expressed in mg/kg/week, were calculated for the 54 week long kinetic subaqueous column test SAC-1. The volume of extracted leachate was multiplied by the measured concentration and divided by the sample mass. The calculated loading rates, therefore, tended to fluctuate week-to-week since the column was cycled biweekly with 160 ml of water on odd weeks and 100 ml of water on even weeks. Analyses were made on samples of both column surface water and pore water. Where constituents were not detected above laboratory detection limits, the detection limit was taken to be the measured value. While loading rates were calculated for most constituents or parameters, only those considered most relevant are discussed below. These include pH, sulphate, aluminum, nickel, chromium, selenium, calcium, and magnesium (Table 2.8-15). Loading rates for all constituents and parameters can be found in Appendix 2.8.

The pH surface and pore water was similar, near-neutral, and relatively constant, and pH ranged between 6.45 and 8.39 (Table 2.8-15). Overall, there was a very slight increase in pH to week 54 that was likely the result of non-sulphide dissolution of carbonate and/or aluminosilicate minerals in the tailings (URS, 2009i). The pH values in surface water were similar to those in the column pore water (Table 2.8-15).

The sulphate loading rates in pore water were initially half as high as those in surface water, but by week 5 the pore water loading rate exceeded that in surface water and remained higher throughout the test. Surface water loading rates were initially near 4 mg/kg/wk (Appendix 2.8) and likely represented limited carbonate dissolution. After week 11, surface water sulphate loading rates fell off and gradually decreased to approximately 1.5 mg/kg/wk during the last weeks of the test. The pore water sulphate loading rates were initially approximately 2 mg/kg/wk increasing to a maximum peak of 15 mg/kg/wk at week 13 (Appendix 2.8). After week 13, sulphate loading rates gradually decreased to less than 4 mg/kg/wk at week 54. The disconnect between surface and pore water loading rates indicated that these waters were not in equilibrium (URS, 2009i).

Aluminum loading rates were very similar in surface and pore water. Typical loading rates ranged between 0.000046 and 0.00014 mg/kg/wk, and peaks were detected at weeks 16, 22, 27, 31, 45, and 49 (Appendix 2.8). These peaks are interpreted as localized changes in mineral equilibrium due to aluminosilicate weathering and dissolution (URS, 2009i).

Nickel loading rates for surface water were on average approximately five times greater than in pore water (Appendix 2.8); surface water loading rates ranged between 0.00018 and 0.00084 mg/kg/wk, and pore water loading rates ranged between 0.00002 and 0.00023 mg/kg/wk. The increased oxygen content in the surface water samples, and subsequent increased sulphide mineral oxidation, is likely responsible for the difference in nickel loading rates between the surface and pore waters (URS, 2009i).

Table 2.8-15 Laboratory Kinetic Test Results and Loading Rates for Minago Tailings

Subaqueous Column - Surface Water
Sample = 1st Cleaner + Rougher Tails

		Loading Rates (mg/kg/wk) [1]												
	pH	Sulphate mg/kg/wk	Aluminum mg/kg/wk	Antimony mg/kg/wk	Arsenic mg/kg/wk	Cadmium, mg/kg/wk	Chromium mg/kg/wk	Copper mg/kg/wk	Iron mg/kg/wk	Lead mg/kg/wk	Molybdenum mg/kg/wk	Nickel mg/kg/wk	Selenium mg/kg/wk	Zinc mg/kg/wk
Minimum	6.45	0.76	2.00E-05	6.08E-06	2.00E-06	1.60E-07	3.20E-06	1.80E-05	3.20E-05	9.28E-07	6.00E-05	1.80E-04	4.00E-06	4.16E-05
Average	7.55	1.99	2.12E-04	9.29E-06	1.30E-05	7.49E-07	1.21E-05	8.01E-05	1.57E-04	1.62E-05	1.18E-04	4.02E-04	8.72E-06	1.30E-04
Maximum	8.15	4.80	1.44E-03	1.18E-05	6.40E-05	7.68E-06	2.00E-05	2.24E-04	6.20E-04	1.63E-04	1.96E-04	8.42E-04	2.18E-05	7.68E-04

Subaqueous Column - Pore Water
Sample = 1st Cleaner + Rougher Tails

		Loading Rates (mg/kg/wk) [1]												
	pH	Sulphate mg/kg/wk	Aluminum mg/kg/wk	Antimony mg/kg/wk	Arsenic mg/kg/wk	Cadmium, mg/kg/wk	Chromium mg/kg/wk	Copper mg/kg/wk	Iron mg/kg/wk	Lead mg/kg/wk	Molybdenum mg/kg/wk	Nickel mg/kg/wk	Selenium mg/kg/wk	Zinc mg/kg/wk
Minimum	6.97	2.56	2.00E-05	1.00E-05	6.00E-06	1.92E-07	3.20E-06	2.00E-05	1.40E-04	4.16E-07	4.20E-04	2.00E-05	1.28E-06	3.52E-05
Average	7.79	6.95	2.21E-04	3.22E-05	2.39E-05	7.41E-07	1.23E-05	9.39E-05	5.27E-04	9.62E-06	7.44E-04	8.93E-05	3.51E-06	1.15E-04
Maximum	8.39	15.20	1.15E-03	1.63E-04	1.20E-04	4.61E-06	2.00E-05	4.35E-04	1.96E-03	1.06E-04	1.13E-03	2.30E-04	9.28E-06	3.84E-04

1 Loading rates are calculated as the average loading rates during weeks 11-54, when the subaqueous column was in steady state.

Source: adapted from URS, 2009i

Chromium concentrations in surface and pore water were at or below laboratory detection limits throughout the test (Appendix 2.8), and the highest calculated loading rate was 0.00002 mg/kg/wk.

Selenium loading rates decreased during the test (Appendix 2.8) and ranged between 0.000004 and 0.000022 mg/kg/wk in surface water and between 0.0000013 and 0.0000093 mg/kg/wk in pore water.

Calcium and magnesium loading rate profiles were similar to the sulphate loading rate profiles; these rates increased between weeks 1 and 12 in pore water while remaining fairly constant in surface water, and then they declined consistently through the rest of the test (Appendix 2.8). Surface water calcium loading rates peaked at 1.04 mg/kg/wk and dropped to 0.37 mg/kg/wk at test's end. Pore water calcium loading rates peaked at 2.7 mg/kg/wk and dropped to 0.5 mg/kg/wk at test's end. Surface water magnesium loading rates peaked at 0.46 mg/kg/wk and dropped to 0.16 mg/kg/wk at test's end. Pore water magnesium loading rates peaked at 1.20 mg/kg/wk and dropped to 0.4 mg/kg/wk at test's end.

Molar ((Ca + Mg) / SO_4) Ratios and Carbonate Depletion Rates

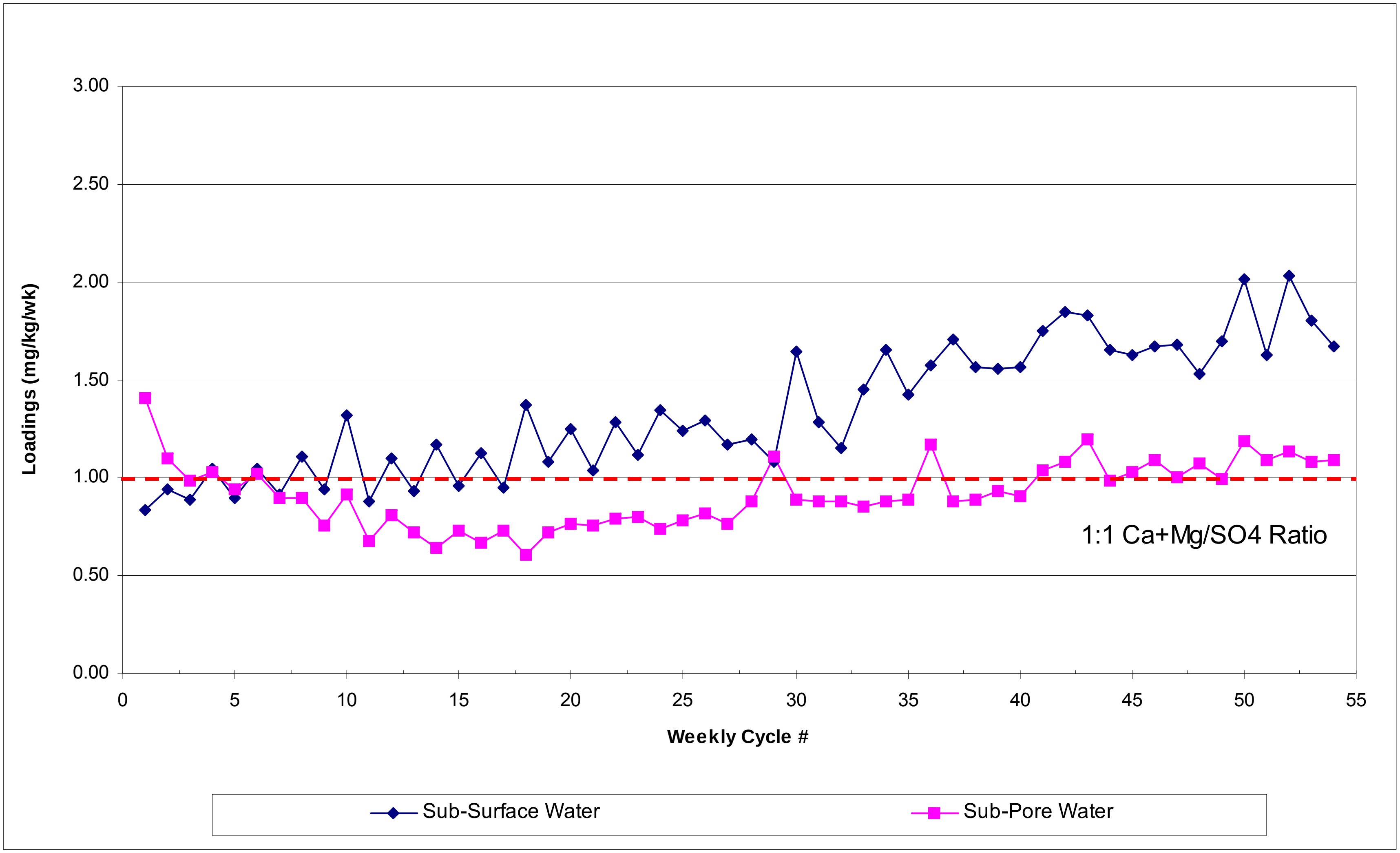
Carbonate molar ratios (the molar ratio of calcium and magnesium to sulphate in the leachates; $(Ca+Mg)/SO_4$) for the subaqueous column test SAC-1 are shown in Figure 2.8-12. This unitless ratio provides an estimate of the proportion of carbonate material that is released (dissolved) in response to both sulphide oxidation and to processes other than acid neutralization.

The molar ratios for the column surface water varied around a value of 1.0 for the first 17 weeks of the test (Figure 2.8-12), indicating that for every molecule of sulphide mineral oxidized to sulphate, one molecule of carbonate was dissolved. After week 17, the ratio increased from approximately 1.0 to 2.0, which appeared to have resulted from increased carbonate dissolution. This shift to higher molar ratios may have occurred as carbonate material maintained chemical equilibrium with the surface water solution because both sulphate and carbonate loading rates were decreasing during this period of the test (Appendix 2.8) (URS, 2009i).

The molar ratios for the column pore water decreased from a peak value of 1.4 to 0.6 over the first 18 weeks of the column, gradually increased to 0.9 by week 40, and then remained at approximately 1 for the rest of the test (Figure 2.8-12). The beginning of the test was a period when both sulphate and carbonate loading rates were increasing in the pore water, and the decrease in the molar ratio appears to be the result of both pore water coming into chemical equilibrium with the minerals and sulphide mineral oxidation. During the last 13 weeks carbonate dissolution and sulphide oxidation appear to be in a 1:1 relationship (URS, 2009i).

Acid Generation Potential Depletion Rates and Timing

The weekly sulphate loading rates determined from the tailings subaqueous column were used to determine the average rate of AGP (sulphide mineral) depletion. Based on these results, weeks



Source: adapted from URS (2009i)

Figure 2.8-12 Carbonate Molar Ratios for Minago Tailings

11 to 54 were considered steady-state or equilibrium conditions and this value was used in rate calculations. It should be noted that subaqueous columns are not intended to provide primary reaction rates of sulphide oxidation, as mineral dissolution and secondary mineral precipitation reactions that mask primary reaction rates can occur in the tailings. Thus, these sulphate loading rates are expected to be lower than primary reaction rates obtained from a humidity cell and must be used with caution. However, these rates may be closer to actual field rates and can be a useful indicator of the relative difference in AGP and ANP rates and the time to their depletion. The sulphide sulphur concentrations from pre-kinetic static tests of the humidity cell sample materials were used as the initial AGP values.

Based on the calculated loading rates from tailings material, the calculated rate of AGP depletion from tailings surface water was 0.021 mmol/kg/wk (Table 2.8-16), and the estimated time to depletion of AGP from the sample was approximately 19 years. The sulphide depletion rate in tailings pore water was 0.072 mmol/kg/wk (Table 2.8-16), and the estimated time to AGP depletion was approximately five years. Details are given in Appendix 2.8.

Acid Neutralization Potential Depletion Rates and Timing

The weekly calcium and magnesium loading rates determined from the tailings subaqueous column were used to determine the average rate of carbonate (ANP) depletion. Based on the humidity cell results, weeks 11 to 54 were considered steady-state or equilibrium conditions and this value was used in rate calculations. The TIC values from pre-kinetic static tests of the humidity cell sample materials were used as the initial carbonate concentrations. Details of the calculations are provided in Appendix 2.8.

Based on the calculated loading rates from tailings material, the calculated rate of carbonate ANP depletion from tailings surface water was 0.027 mmol/kg/wk (Table 2.8-16), and the estimated time to carbonate ANP depletion was 274 years. The calculated rate of carbonate depletion from tailings pore water was 0.060 mmol/kg/wk (Table 2.8-16), and the estimated time to carbonate depletion was 121 years. Note that the AGP and ANP depletion rates are similar in magnitude, which is further evidence that the carbonate mineral depletion occurred in direct response to sulphide mineral oxidation and acid production (URS, 2009i).

2.8.3 Conclusions

The standard Sobek method significantly over-estimated the ANP of material sampled from the Minago Project when compared to ANP measured using carbonate ANP and modified Sobek method, the results of which tended to be in relative agreement (URS, 2009i).

Overburden, Ordovician dolomitic limestone, and Ordovician sandstone material overlying the altered Precambrian basement and Precambrian basement lithologies are considered not potentially acid generating (NAG) and have minor metal leaching potential based on the results of this geochemical characterization program (URS, 2009i).

Table 2.8-16 Subaqueous Tailings Column Depletion Rates

COLUMN	Column Mass (kg)	Initial Sulphide-S					Sulphur remaining (mmol)	Avg. Sulphur depletion rate (mmol/kg/wk)	Weeks to Sulphur depletion	Years to Sulphur depletion
		(%)	(mg/kg)	(g/kg)	(mol)	(mmol)				
SAC-1 SURFACE WATER	5	0.07	700	0.7	0.11	109.17	102.88	0.021	992.4	19.08
SAC-1 PORE WATER	5	0.07	700	0.7	0.11	109.17	90.36	0.072	249.8	4.80

COLUMN	Sample Mass (kg)	Initial Total Carbonate[1]					Remaining Carbonate (mmol)[2]	Avg. Carbonate Depletion rate (mmol/kg/wk)[3]	Weeks to Carbonate Depletion	Years to Carbonate Depletion
		as TIC (%)	(kg $CaCO_3$/t)	(%)	(mmol/kg)	(mmol)				
SAC-1 SURFACE	5	0.46	38.3	3.8	383.014	1915.07	1907.52	0.027	14261	274
SAC-1 PORE	5	0.46	38.3	3.8	383.014	1915.07	1899.14	0.060	6302	121

NOTES:
1 Based on total inorganic carbonate measurements (TIC); assumes all carbonate ANP as calcite.
2 Based on difference between the initial total carbonate and the amount of calcium (Ca) and magnesium (Mg) which has leached from the samples.
3 Based on steady state combined depletion rates of Ca and Mg between weeks 11 and 54.

Cell ID	Sample ID	ABA Results				Total Metals (ppm)	Average Sulphide Depletion Rate [1,2]	Time to Sulphide Depletion [1]	Average Carbonate Depletion Rate [1,3]	Time to Carbonate Depletion [1]	Average Carbonate Molar Ratio [1]	Expected to be acid generating?
		ANP [4,5]	AGP [5]	NNP [5]	NPR	Ni	(mmol/kg/wk)	(years)		(mmol/kg/wk)	(years)	
SAC-1 SURFACE	2007 0.3% Ni Lock CycleTails	76.5	2.2	74.3	35.0	2456	0.021	19.1	1.29	0.027	274	**NO**
SAC-1 PORE	2007 0.3% Ni Lock CycleTails	76.5	2.2	74.3	35.0	2456	0.072	4.8	0.83	0.060	121	**NO**

NOTES:
1 Subaqueous column calculations are based on steady state conditions betwee weeks 11 and 54.
2 Sulphide depletion rates are based on the initial sulphide sulphur content.
3 Carbonate depletion rates are based on the initial total inorganic carbon (TIC) content.
4 NP derived from the modified Sobek method.
5 units are kg $CaCO_3$ per tonne.

Source: adapted from URS (2009i)

A preliminary screening of the elemental concentrations of overburden, Ordovician dolomitic limestone and Ordovician sandstone detected elevated chromium, nickel, sulphur, antimony, thorium and uranium. In overburden and Ordovician dolomitic limestone, concentrations of these elements were slightly elevated and likely represent local and/or regional background. In Ordovician sandstone, elevated chromium, nickel, and sulphur concentrations suggest a potential for metal leaching. The NPRs of composite samples containing Ordovician dolomitic limestone suggest that these materials could provide sufficient neutralization capacity to offset the AGP of Precambrian basement lithologies (URS, 2009i).

Generalized altered Precambrian basement and Precambrian basement samples contained low to high sulphide sulphur concentrations, coupled with low to moderate carbonate concentrations. The fresh material was considered to be PAG, while the altered material was equivocal: five of the eight altered Precambrian basement samples were NAG while three were PAG. Composite samples containing these lithologies and Ordovician sandstone or overburden were considered to be NAG. Screening of undifferentiated Precambrian basement material indicated elevated levels of barium, cobalt, chromium, copper, iron, nickel, and sulphur (URS, 2009i).

Granite is considered to be NAG, based on a low but variable sulphide sulphur content ranging from 0.02 to 0.39 % by weight (AGP values ranging from 0.63 to 12.2 kg $CaCO_3$/tonne) and low to moderate ANP values of 9.7 to 87.2 kg $CaCO_3$/tonne. Higher sulphide sulphur value and low ANP values occurred in one sample, which was considered to be PAG. The NPR value ranged from 0.8 to 105.5. Screening the elemental concentrations in granite indicated elevated levels of silver, arsenic, cadmium, cobalt, chromium, copper, iron, nickel, phosphorus, selenium, sulphur, antimony, and possibly bismuth and mercury (URS, 2009i).

Serpentinite was considered to be NAG based on low but variable sulphide sulphur values ranging from 0.02 to 0.80 % by weight (AGP values ranged from 0.6 to 23.1 kg $CaCO_3$/tonne) and ANP was moderate to high at values of 33.4 to 272.4 kg $CaCO_3$/tonne. The NPR values ranged from 3.0 to 268.3. Screening the elemental concentrations in these rock types indicated elevated levels of arsenic, copper, molybdenum, nickel, lead, selenium, sulphur, antimony (URS, 2009i).

Amphibolite, mafic dike, and altered Precambrian basement rock types contain negligible to low sulphide sulphur concentrations (<0.3 % by weight) and low to high carbonate concentrations. These rock types were considered to be NAG. The NPR values ranged from 5.1 to 10.2. Screening the elemental concentrations indicated elevated levels of silver, arsenic, cadmium, cobalt, chromium, copper, nickel, selenium, sulphur, antimony, and possibly bismuth and mercury (URS, 2009i).

Mafic metavolcanic rock was considered to be PAG based on low sulphide sulphur content (0.5 % by weight or an AGP of 14.4 kg $CaCO_3$/tonne) and an equally low ANP of 21.0 kg $CaCO_3$/tonne. The NPR value was 1.5. Screening the elemental concentrations in this rock type indicated elevated levels of silver, cadmium, selenium, sulphur, antimony, and possibly bismuth (URS, 2009i).

Metasedimentary rock was considered to be PAG based on a variable sulphide sulphur content of 0.2 to 5.1 % by weight (AGP of 5.3 to 160.0 kg $CaCO_3$/tonne) and a low to moderate ANP of 6.8 to 89.3 kg $CaCO_3$/tonne. The NPR value ranged from 0.1 to 7.7. Screening the elemental concentrations indicated elevated levels of silver, cadmium, cobalt, chromium, copper, nickel, selenium, sulphur, antimony, and possibly mercury (URS, 2009i).

The sample population of rock types used to draw these conclusions is small relative to the estimated volume of waste rock expected to be generated by mining activities at the Minago Project, and additional static testing may be required on discrete samples of all lithologies to develop a statistically valid dataset to confirm the conclusions of this geochemical assessment (URS, 2009i).

Waste Rock Kinetic Test Program

The carbonate molar ($Ca+Mg/SO_4$) ratios in conjunction with the sulphate, calcium, and magnesium loading rates indicated that carbonate dissolution in the humidity cells was not solely attributable to sulphide oxidation and acid generation.

Humidity cell NPR values categorized the humidity cells as near PAG (NPR = 3.7) or NAG (NPR ranged between 7.8 and 40.5). The calculated times to depletion of carbonate minerals was greater than for sulphide minerals in all the humidity cell tests, and so all the cell samples were considered NAG.

Humidity cells containing Ordovician dolomitic limestone yielded lower sulphide loading rates from a higher initial sulphide sulphur content, suggesting that limestone may have provided micro-scale neutralization of sulphide oxidation.

The leaching rates from the humidity cells for all metals of concern (nickel, aluminum, molybdenum, selenium, chromium, cobalt, copper, iron, and trace elements such as strontium) were low, indicating that metal leaching from waste rock, pit walls and other waste materials may be low.

Loading rates from kinetic humidity cell tests of samples of altered Precambrian basement and Precambrian basement material, encountered in and adjacent to the pit shell, indicated the time to completely oxidize the acid generating potential (i.e., sulphide material) was 12 to 58 years, while the time calculated to consume the acid neutralization potential (i.e., carbonate material) was a period of 49 to 954 years. These humidity cell test results also suggest that limestone mixed with altered Precambrian basement and Precambrian basement could be effective in providing excess acid neutralization capacity to compensate secondary sulphide oxidation products on a micro-scale or meso-scale in-situ (URS, 2009i).

URS (2009i) recommended an operational program for static testing on blast hole cuttings based on a geologic block model. Based on kinetic test carbonate molar ratios, URS recommended a preliminary neutralization potential ratio of 1.7 for segregatiing PAG from NAG waste rock materials (URS, 2009i).

URS (2009i) recommended the following common method for differentiating PAG from NAG material, used at many operating mines, for the Minago Project:

- Collect samples from blasthole cuttings in PAG waste material – ultramafic and granitic;
- Perform static testing (using ABA and/or other appropriate surrogate methods) and fizz tests of blasthole cuttings at an on-site laboratory;
- Input the static test results into a geologic block model and krig the results;
- Communicate the in-pit PAG/NAG limits to pit operators; and
- Dispose of the material in the appropriate disposal areas, based on the PAG/NAG delineation.

This process has been used successfully at several open pit mines in British Columbia, including the Huckleberry Mine, QR Mine, and Kemess South Mine (URS, 2009i).

Tailings

Static Test Program

Analysis of the 2006 and 2007 Master Lock Composite samples indicated that metallurgical lock cycle testing removed the majority of sulphide minerals. Based on the low sulphide sulphur content and high carbonate content, the tailings samples were considered to be NAG.

Metal concentrations screening found elevated arsenic, barium, copper, nickel, lead, antimony, strontium, thallium, and uranium relative to similar rock types (per Turekian and Wedepohl, 1961).

Kinetic Test Program

The (Ca+Mg)/SO_4 molar ratios, in conjunction with the sulphate, calcium, and magnesium loading rates, indicated that carbonate dissolution is primarily attributable to sulphide oxidation and acid generation.

The tailings are predicted to be NAG in a subaqueous environment, based on the low sulphide sulphur content, and because the time to depletion of carbonate minerals was greater than for sulphide minerals.

The metal loading rates are low, suggesting low leaching potential from tailings material.

2.9 Mining Processes

2.9.1 Overview

The open pit was designed using a two-stage approach. In the first stage, an optimum pit shell using the Lerchs-Grossman pit optimization method was identified. In the second stage, the selected pit shell was refined to a more detailed pit design that included catch berms and haul roads. Subsequently, mine development and production schedules were developed, equipment selections were performed and the capital and operating costs were estimated.

The Minago deposit has potential as a large tonnage, low-grade nickel sulphide deposit suitable for open pit bulk tonnage mining. Wardrop determined that the mining operation is amenable to conventional open pit mining methods.

The mine will provide mill feed of sulphide ore at a rate of 10,000 tonnes/day (t/d) for a total of 25.2 Mt of ore grading at 0.43%, over a period of approximately 8 years (Wardrop, 2009b). Local sandstone, that forms part of the overburden, is of suitable quality to produce frac sand, which is used in the oil and gas industry. The open pit will provide sand feed to a frac sand processing facility at a rate of about 4,100 t/d of sand, for a total of 14.9 Mt of frac sand over a period of about 10 years. The sand will be mined over a period of 3 years at the start of the mining operations, and then stockpiled. The throughput of the sand plant will be maximized to match the ore processing schedule (Wardrop, 2009b).

The estimated overall stripping ratios (waste-to-ore ratio tonne/tonne, t/t) to mine both the nickel sulphide ore and frac sand are given in Table 2.9-1.

Table 2.9-1 Open Pit Design 14 Stripping Ratios

Case	SR (t/t) (No Overburden)	SR (t/t) (With Overburden)
Frac Sand Only	7.48	8.23
Nickel Ore Only	11.27	11.71
Nickel Ore and Frac Sand	6.72	7.00

Source: Wardrop, 2009b

An overall mining sequence was developed in three phases: one initial pit phase and two pushback phases. Mine development will commence with the removal of trees and roots, and then the muskeg and clay overlying the dolomitic limestone will be dredged and removed from the open pit area. The dredging method has been selected for the removal of the muskeg and clay overburden, since mechanical removal using excavators for removal, and trucks for transportation and dumping would be difficult and expensive due to the soft clays.

The dredging is scheduled to commence in the spring of “Year –3” (2011) to prepare for dolomite removal starting at the beginning of “Year -2” (2012). The removal of the dolomite will take approximately 2 years with frac sand being available at the beginning of “Year –1” (2013). Another year later, at the start of “Year 1” (2014), the nickel ore will be available for processing (Wardrop, 2009b).

A general arrangement drawing for the Mine Complex is shown in Figure 2.1-2. The particular features of the layout, which are pertinent to the operation of the open pit mine, are as follows:

- close proximity of the Overburden Disposal Facility to the open pit to minimize the pumping distances for dredging;
- close proximity of the Dolomite and Country Rock Waste Rock Dumps to the open pit to minimize the haul distances for the waste rock; and
- close proximity of the Tailings and Ultramafic Waste Rock Management Facility (TWRMF) to the open pit to minimize the haul distances involved in moving and placing the dolomite etc. for the dam construction and disposal of ultramafic waste rock.

2.9.2 Geotechnical Considerations

2.9.2.1 Open Pit Stability

Wardrop completed a geotechnical stability analysis for the open pit project in August 2008 (Wardrop, 2008a). Based on the collected geotechnical information, analytical, empirical, and numerical methods were used to derive rock strengths from diamond drilling, field mapping programs, an auger drilling program, a site visit, and from previous geotechnical studies. The measured values were compared to the proposed final pit design through the use of rock mass classification and empirically derived rock mass strengths (Wardrop, 2009b).

At Minago, the open pit stability will be maintained by managing the following two major rock strength principles (Wardrop, 2009b):

1) When assessing a rock slope on a large scale, a rock mass behaves as a homogeneous material with a limited strength, within geological domains.
2) The geological structure within the rock mass may provide major planes of weakness that can produce both large and small scale failures.

Geotechnical Domains and Design Sectors

The proposed open pit was broken down into seven geotechnical domains for pit wall design. These domains are based primarily on similar rock types and similar geotechnical data. An overview of the seven geotechnical domains (domains I through VII) is provided in Table 2.9-2.

Table 2.9-2 Geotechnical Domains for Minago Project

Domains	Types	Lithologies	Thickness (m)	Intersects Pit Wall
I	Overburden	Peat	2	Yes
II	Overburden	Clay	13-15	Yes
III	Sedimentary Rock	Dolomite	51-56	Yes
IV	Sedimentary Rock	Sandstone	6-10	Yes
V	Unconformity	Regolith	0-6	Yes
VI	Igneous Rock	Host[1]	Varies[3]	Yes[4]
VII	Igneous Rock	Country[2]	Varies[3]	Yes

Source: Wardrop, 2009b

Notes:

1 Host rock – is primarily composed of ultramafic rock

2 Country rock – is primarily composed of granite, and also contains amphibolites, and ultramafic rock. Due to the heterogeneous nature of country rock, these sub-units were all grouped together until further data collection becomes available during construction.

3 The host intrusive body has a near vertical contact with the country rock. The thickness varies with the intersection of the pit wall.

4 Intersects pit wall at the toe of the slope.

For the geotechnical design, the final pit design was subdivided into four main design sectors with each design sector being composed of the geotechnical domains I through VII, described in Table 2.9-3. The locations of these geotechnical domains within each design sector are illustrated in Figure 2.9-1 and the overall pit slope geometry based on geotechnical concepts is illustrated in Figure 2.9-2.

For Open Pit design, a factor of safety of 1.20 for slope stability is generally considered to be acceptable (Wardrop (2009b). In the pit slope stability analysis, the factors of safety were calculated from numerical modeling for various conditions, including the following four groundwater conditions:

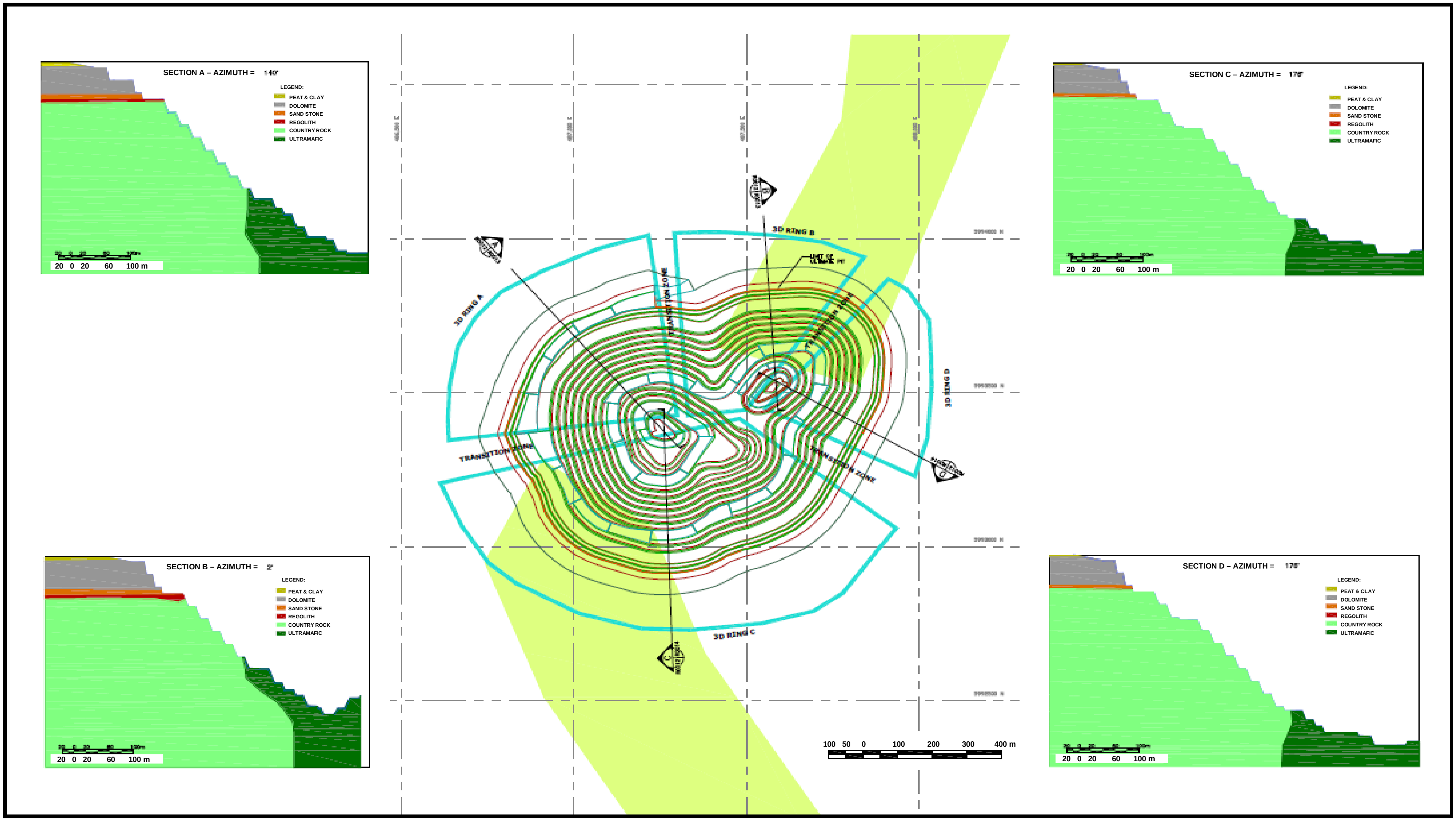
- Condition 1: The pit is dewatered and dry.
- Condition 2: The pit is dewatered, but the sandstone remains saturated.
- Condition 3: The pit has a perched water table above the basement rocks.
- Condition 4: The pit is completely saturated through the basement rocks.

The estimated factors of safety for different design sectors, geotechnical domains, and groundwater conditions of the open pit at Minago are provided in Table 2.9-4. Estimated safety factors ranged from 1.15 to 1.97. Almost all safety factors that were below 1.2 were limited to Groundwater Situation 4 (i.e. case for which the pit was assumed to be completely saturated

Table 2.9-3 Geotechnical Parameters for the Final Design Pit by Sector

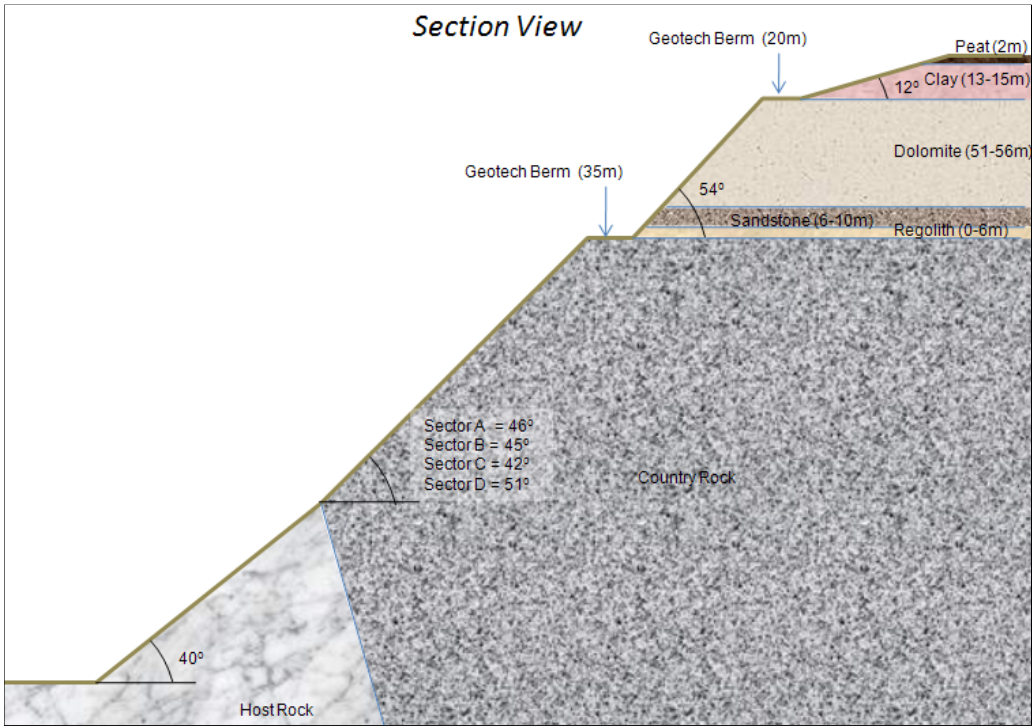
Design Sector	Geotechnical Domain	Interamp Angle (°)	Bench Face Angle (°)	Bench Height (m)	Catch Bench Width (m)
A	I, II	8.9	12.0	12	20
A	III	54.0	80.0	24	10
A	IV,V	12.9	80.0	12	35
A	VI	40.0	70.0	24	12
A	VII	46.0	70.0	24	12
B	I, II	8.9	12.0	12	20
B	III	54.0	80.0	24	10
B	IV,V	12.9	80.0	12	35
B	VI	40.0	70.0	24	12
B	VII	45.0	70.0	24	12
C	I, II	8.9	12.0	12	20
C	III	54.0	80.0	24	10
C	IV,V	12.9	80.0	12	35
C	VI	40.0	70.0	24	12
C	VII	42.0	70.0	24	12
D	I, II	8.9	12.0	12	20
D	III	54.0	80.0	24	10
D	IV,V	12.9	80.0	12	35
D	VI	40.0	70.0	24	12
D	VII	51.0	70.0	24	12

Source: Wardrop, 2009b



Source: adapted from Wardrop, 2009b

Figure 2.9-1 Overall Pit Slope Geometry, Plan View



Source: Wardrop, 2009b

Figure 2.9-2 Overall Pit Slope Geometry based on Geotechnical Concepts

Table 2.9-4 Safety Factors for all Domains

Safety Factor	Domains I, II	Domains I, II, III. IV. V			All Domains	
Groundwater Condition:	1	1	2	3	2	4
Section A	1.28	1.72	1.28	1.42	1.37	1.17
Section B	1.67	1.97	1.64	1.85	1.39	1.19
Section C	1.81	1.39	1.15	1.20	1.33	1.16
Section D	1.90	1.74	1.21	1.33	1.40	1.21

Source: Wardrop, 2009b

Notes on Groundwater Situations:

1) The Minago pit is successfully dewatered and the pit is dry.
2) The Minago pit is successfully dewatered, but the sandstone remains completely saturated.
3) The Minago pit has a perched water table above the basement rocks. The shape of the groundwater profile is parabolic. The water table resumes its original height at a distance of four times the height of sandstone, limestone, and overburden units.
4) The Minago pit is completely saturated through the basement rocks. The shape of the groundwater profile is parabolic. The water table resumes its original height at a distance of four times the height of the slope.

through basement rocks). If the pit is successfully dewatered, the normal design condition will be groundwater condition 3 (i.e., the pit has a perched water table above the basement rocks). Since an open pit safety factor of 1.2 is considered acceptable, the designed final pit was presumed to be stable under normal design conditions (Wardrop, 2009b).

Wardrop (2009b) made the following recommendations with respect to final pit stability analysis:

- a geotechnical berm of 35 m in width at the base of Domains IV and V should be constructed to catch sloughing within those domains and debris from domains above;
- a drainage ditch at the base of Domains IV and V within the geotechnical berm should be constructed to divert groundwater infiltration from the highly conductive sandstone unit, with a hydraulic conductivity of 7×10^{-3} cm/s;
- further geological structural data should be collected to assist in the optimization of the bench geometry;
- the influence of groundwater on the stability of the open pit should be assessed to address pressure build up within the geological structure; and
- groundwater levels from the hydrogeological investigation should be incorporated into the finite element modelling.

A geotechnical berm with a width of 35 m will be required in the sandstone and regolith to catch sloughing material from the dolomite above, as the weaker sandstone material will promote toppling-type failures of the dolomite along critical jointing. The 35 m wide geotechnical berm will provide catchment for the material toppling from the dolomite domain. Since the amount of material toppling from the dolomite cannot be predicted accurately, a worst-case scenario assuming the entire height of the dolomite domain toppling was selected as the criteria for design. The material is assumed to fall on to the geotechnical berm and sit at the rock’s internal angle of repose of 38°. The bench geometry will be further optimized once more structural information becomes available.

2.9.2.2 Mine Optimization

Wardrop completed a 3D geological block model named “LG Final Model 07 Oct 08”, which incorporated the available information from diamond drill holes on the Minago Property. This geologic model was the basis for the pit design and the mine optimization (Wardrop, 2009b).

Work completed in December 2008 indicated that economic recovery of the underground resource at Minago is currently not feasible due to an insufficiently measured and indicated resource. For this reason, mine optimization calculations are based on an “open pit only” option and do not take the effect of breakeven open pit and underground costs into account (Wardrop, 2009b).

Pit optimization calculations were performed to determine the optimum pit limits and produce economically mineable ore reserves at a maximum net present value (NPV). Wardrop's pit optimization work included (Wardrop, 2009b):

- a geotechnical review;
- initial optimization;
- development of a preliminary schedule based on a best case, a worst case and a specified case;
- development of preliminary economics for the schedule; and
- selection of a pit shell that represents the highest present value for the specified case.

Wardrop used the Lerchs-Grossman (LG) algorithm from Gemcom Software International Inc., supplemented by GEMS^{TM} mine planning software, to perform the pit optimization for the project. The LG algorithm progressively manipulates related blocks that are economic when costs of mining ore and waste stripping are taken into account and in accordance with specified variable pit slopes. The resulting pit outline includes all economic blocks (Wardrop, 2009b).

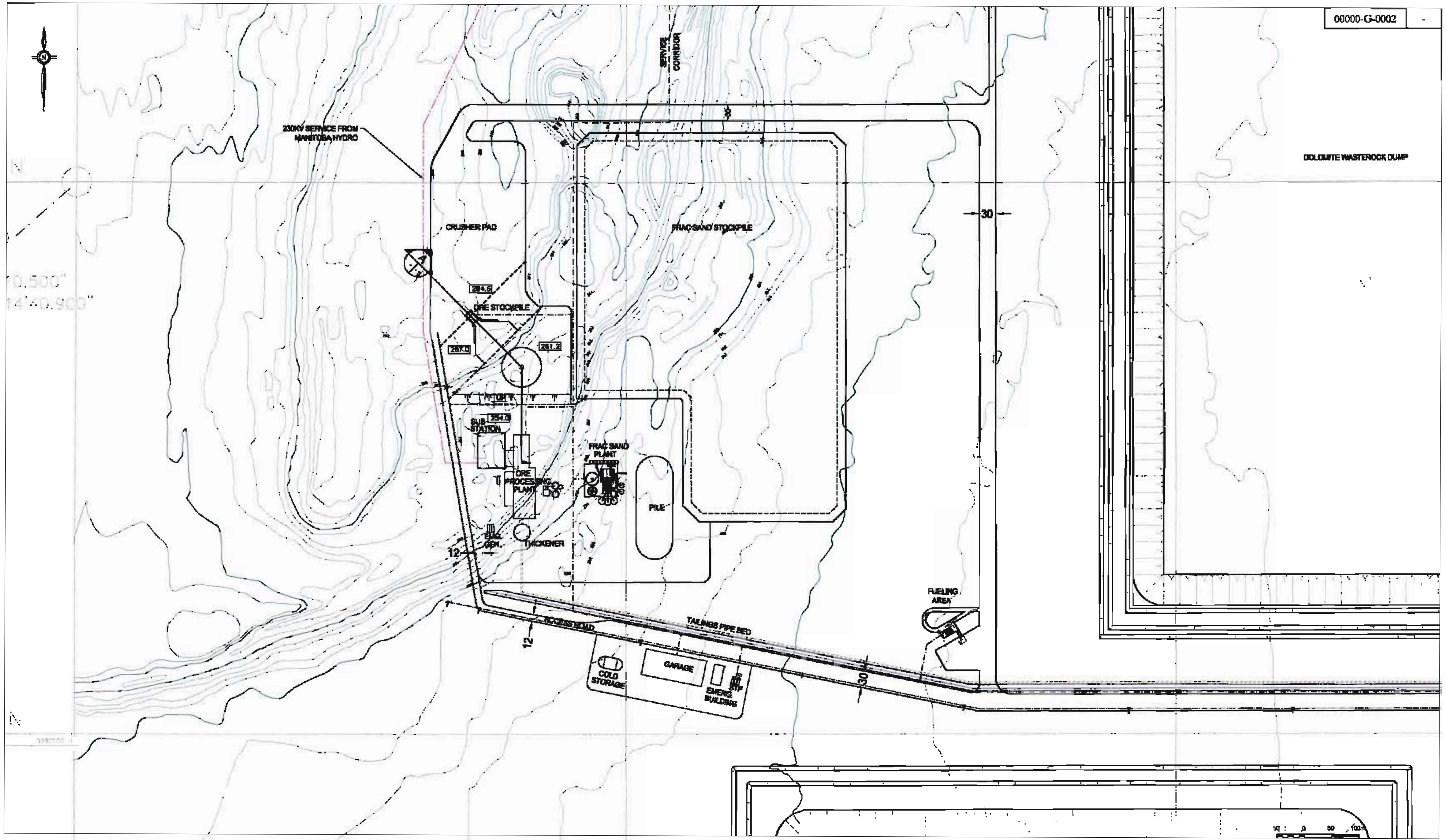
In Wardrop's work, the final pit was sub-divided into four main design sectors based on similar rock types and geotechnical data. The block dimensions for the Lerchs-Grossman algorithm were determined for each sector of the geotechnical domains. A 3D geological block model and other required economic and operational variables were added and manipulated for export to the LG algorithm. These variables included rock classification, mining and milling parameters, and anticipated product grades. Other inputs included costs, metal prices, and smelter terms (Wardrop, 2009b).

Based on sensitivity analyses conducted on various pit configurations, Pit #14 was selected as the optimum pit. The Pit #14 configuration generated the highest discounted cash flow (Present Value) at $461 million; had a Ni(S) grade of 0.3881% and an estimated ore tonnage of 29.4 Mt. However, in case that the pit might be expanded in the future when (or if) drill density and metal prices permit, Wardrop (2009b) recommended to locate surface facilities, such as roads, waste rock dumps and water drainage holes, to accommodate the larger Pit #27 dimensions.

For the detailed open pit design, catchberms and ramps were included and a higher cut off grade of 0.2% Ni(S) was used instead of 0.17% Ni(S) that was used for the initial pit optimization (Wardrop, 2009b).

2.9.3 Project Development

The Minago Project development has been broken down into the stages of pre-production work (stripping) and three mineable phases based on mineralogical, geotechnical and pit optimization work conducted by Wardrop. The general arrangement drawing for the mine, primary concentrator, ancillary structures, waste dumps, and tailings dam are illustrated in Figures 2.1-2 and 2.9-3.



Source: adapted from Wardrop's drawing 0951330400-G0002 (Wardrop, 2009b)

Figure 2.9-3 Minago Plant Area

Pre-production work will begin with initial pushbacks commencing three years prior to the designated mill-start up year. Contractors will strip peat and clay, and limestone and dolomitic waste rock. Once Victory Nickel's mining equipment becomes available, the contractor's stripping equipment will be gradually phased out and replaced by the owner's equipment (Wardrop, 2009b).

Approximately 11.2 Mt of peat and clay will be excavated from the open pit area by dredging in "Year -3" (2011) in preparation for the owner to start stripping 42.7 Mt dolomite/limestone waste rock at the beginning of "Year -2" (2012).

The overburden material will be deposited in a 300 ha Overburden Disposal Facility (ODF), located above an area with thick, low-strength clays (Figure 2.1-2). Keeping the overburden materials separate from the rest of the materials will allow for future reclamation of this material.

A portion of the excavated limestone and dolomitic waste rock will be used for the construction of roads, containment berms, and portions of the base layer of the Tailings and Ultramafic Waste Rock Management Facility (TWRMF) and for the site preparation of a Crusher Pad and a Ore Stockpile Pad while the remainder will be deposited in the 191 ha Dolomite Waste Rock Dump (Figure 2.1-2).

2.9.3.1 Mineable Phases

Project development will involve three mineable phases based on mineralogical, geotechnical and pit optimization work. The mineable phases are based on the measured and indicated mineral resources and the optimized pit. The projected material to be mined in the three phases is summarized in Table 2.9-5 and illustrated in Figure 2.9-4 and Tables 2.9-6 and 2.9-7 provide a breakdown of the materials to be mined from the open pit (Pit #14 configuration). The projected mine production will peak at 51.2 Mt in the year 2013 (Wardrop, 2009b).

For Phase I, the pit was designed from the initial economic shells generated by a Whittle^{TM} optimization run. The initial economic shells prioritize the high grade ore mining at the top portion of the orebody, and at the lowest amount of waste stripping. The objective of this prioritizing was to maximize cash flow and to speed up the capital recovery during the initial years. Phase I will mine 2.47 Mt of sand and 1.70 Mt of Ni(S) ore at 0.387% Ni(S) for a total material of approximately 44.8 Mt (Wardrop, 2009b).

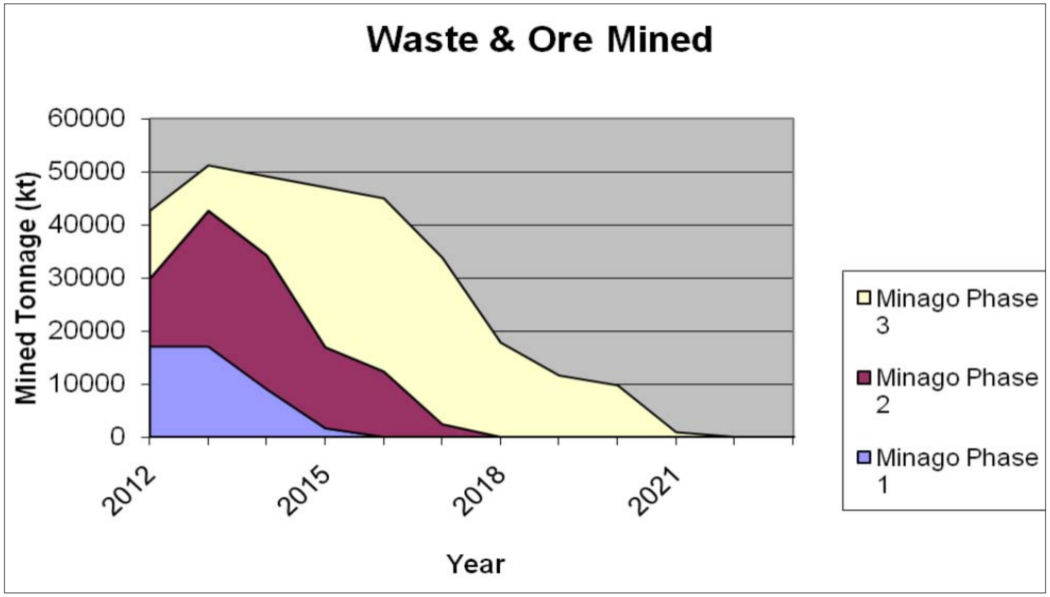
The Phase II geometry expands in all directions from the Phase I geometry to mine the next high grade blocks of the orebody. The final highwalls will be reached in the west and southwest of the ultimate pit shell to achieve the required minimum mining width. Phase II will mine 4.91 Mt of sand, 9.4 Mt of Ni(S) ore at 0.438% Ni(S) for a total material of about 93.6 Mt (Wardrop, 2009b).

In Phase III, the remaining ore inside the ultimate pit shell will be mined to achieve the final highwalls. Phase III will mine 7.47 Mt of sand, 14.03 Mt of Ni(S) ore at 0.429% Ni(S) for a total material of about 170.3 Mt.

Table 2.9-5 Material to be Mined by Mineable Phase (in Kilo Tonnes)

Phase	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	TOTAL
Minago Phase 1	17,063	17,063	9,003	1,642							**44,771**
Minago Phase 2	12,796	25,592	25,213	15,274	12,339	2,391					**93,605**
Minago Phase 3	12,796	8,531	14,890	30,129	32,639	31,400	17,766	11,570	9,729	881	**170,330**
Total	**42,655**	**51,185**	**49,106**	**47,045**	**44,978**	**33,791**	**17,766**	**11,570**	**9,729**	**881**	**308,706**

Source: adapted from Wardrop, 2009b



Source: adapted from Wardrop, 2009b

Figure 2.9-4 Material to be Mined by Mineable Phases

Table 2.9-6 Overall Pit Mining Schedule

	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	TOTAL
Dolomite (kt)	42,655	43,179	15,183	10,015	0	0	0	0	0	0	**111,032**
Granite (kt)	0	1,744	20,890	20,440	35,711	24,459	9,784	4,944	3,832	199	**122,003**
Ultramafic (kt)	0	861	7,941	5,524	5,667	5,732	4,382	3,026	2,297	229	**35,659**
Sand (kt)	0	5,289	2,092	7,466	0	0	0	0	0	0	**14,847**
Total Ni Ore (kt)	0	112	3,000	3,600	3,600	3,600	3,600	3,600	3,600	453	**25,166**
Total Tonnage (kt)	**42,655**	**51,185**	**49,106**	**47,045**	**44,979**	**33,791**	**17,766**	**11,570**	**9,729**	**881**	**308,706**

Source: adpated from Wardrop, 2009b

Table 2.9-7 Projected Material Quantities and Volumes Mined from the Open Pit

Material	Tonnes (kt)	Density (t/m^3)	Volume (in-situ m^3)	Volume (swelled m^3; swell value: 30%)
Ore	25,166	2.612	9,634.697	12,525,106
Sand	14,847	2.400	6,186,065	8,041,885
Granitic Waste Rock	122,005	2.702	45,148,004	58,692,405
Ultramafic Waste Rock	35,659	2.590	13,767,708	17,898,020
Overburden	11,217	1.856	6,044,945	7,858,428
Limestone	111,032	2.790	39,797,437	51,736,668
Total Waste Rock	**268,695**		**98,713,149**	**128,327,093**
Total Mined	**319,924**		**120,578,855**	**156,752,512**

Source: Wardrop, 2009b

The ultimate pit design is summarized in Table 2.9-8 and illustrated in Figure 2.9-5. Overall, the ultimate pit contains 14.8 Mt of sand and 25.17 Mt of Ni(S) ore at 0.43% Ni(S) (Wardrop, 2009b). The total depth of the ultimate pit will be 359 metres and the elevation of the pit bottom will be -112 m.a.s.l. Both the ore and the waste will be mined using 12 m high benches (Wardrop, 2009b).

Table 2.9-8 General Pit Characteristics

Item	Size
Pit Top Elevation	About 247 m
Pit Bottom Elevation	-112 m
Pit Depth	About 359 m
Volume of Pit	156.7 million m^3
Area of Pit Top	1.0 million m^2
Perimeter at the Top of the Pit	3,7 km
Length from East to West	1.2 km
Length from North to South	1.1 km

Source: Wardrop, 2009b.

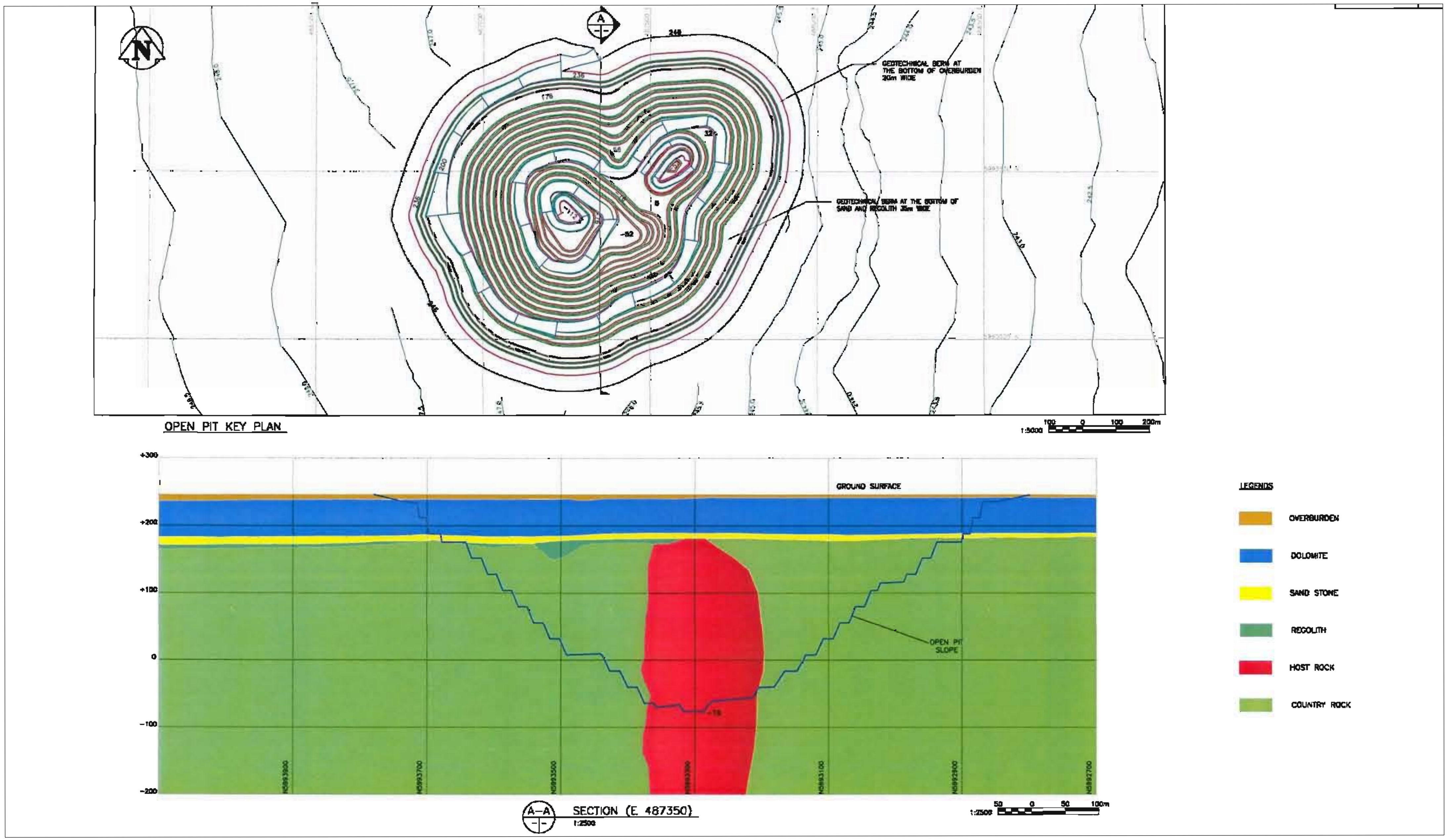
The mine will start delivering frac sand ore in the year just prior to Frac Sand production at the start of 2013. The delivery of nickel sulphide ore is scheduled to begin in late 2013 in preparation for Ore Processing at the start of “Year 1” (2014) and will continue until “Year 8” (2021).

The delivery and placement of overburden, limestone, and basement rock will closely follow the geotechnical parameters governing the construction of the waste rock dumps, tailings dam, and the ODF (Wardrop, 2009b).

Each of the mineable phases or pushbacks is designed at a mining width of about 65 m to accommodate mining equipment that will operate in the benches. The mining width allows for 35 m of double-sided loading if, for example, a Komatsu PD4000 electric hydraulic shovel were to be used. The remaining 30 m road is designed to accommodate two lanes of traffic using typical 218 tonne haul trucks.

2.9.4 Production Rate and Schedule

Wardrop developed a conventional open pit mining operational plan for the Minago Project that will provide mill feed at the rate of 10,000 t/d of nickel sulphide ore, totalling 25.2 Mt of ore over a period of approximately 8 years (Wardrop, 2009b). It was assumed that contractor activities will begin 3 years before mill start up and that 112 kt ore will be stockpiled in 2013, for later milling (Wardrop, 2009b). Table 2.9-9 lists the projected annual nickel ore tonnage (in Kilo Tonnes) and grade.



Source: adapted from Wardrop's drawing 0951330400-R0023 (Wardrop, 2009b)

Figure 2.9-5 Ultimate Pit Design at Minago

Table 2.9-9 Estimated Annual Ore Tonnage (in Kilo Tonnes) and Grade

	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023
Ni Ore		112	3,000	3,600	3,600	3,600	3,600	3,600	3,600	453		
Grade (%)		0.37	0.42	0.43	0.43	0.41	0.44	0.43	0.45	0.47		

Source: adapted from Wardrop, 2009b.

The open pit will also provide sand feed to a frac sand process facility at the rate of about 4,100 t/d of sand feed (1.50 Mt/a), totalling 14.9 Mt of sand feed over a period of about 10 years.

Outotec Physical Separation Division in Jacksonville, FL designed the Frac Sand Plant for Minago, which accommodates year round operations and is capable of producing three saleable products including two types of fracturing sand and a flux sand product. Approximately 612,863 t/a of the final product will be frac sand capable of meeting the American Petroleum Institute (API) specifications, and 529,941 t/a of the final product will be non-API frac sands (which includes 62,500 t/a of flux sand) to be sold to other markets. The throughput of the sand plant will be maximized to match the ore processing schedule.

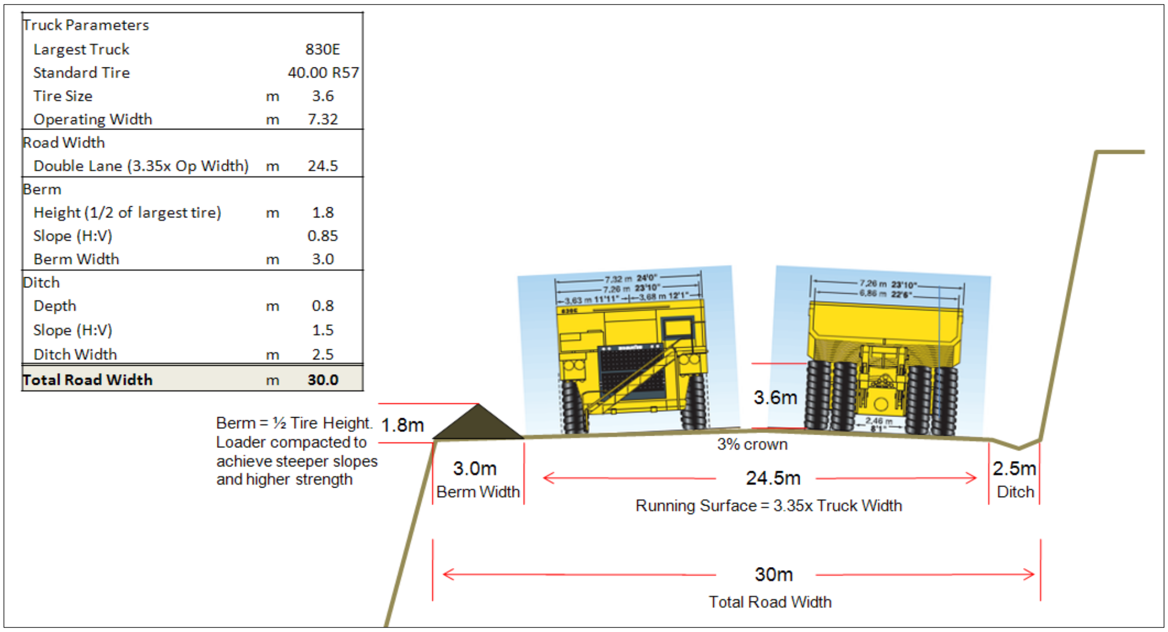
The sand will be mined over a period of 3 years and stockpiled. A Frac Sand Process Plant is projected to be commissioned during 2013 and it is anticipated that delivery of frac sand ore will begin in 2013.

2.9.5 Mine Access and Infrastructure

The Minago Project is located just off Provincial Highway #6 approximately 100 km north of Grand Rapids, MB. Currently, there is no mining related infrastructure on the Property and the site has only been accessed via a winter road in the winter and by Argo or helicopter in the summer due to swampy site conditions in the summer.

A road network will be required to gain access to the proposed Minago Project. In the proposed site layout, illustrated in Figure 2.1-2, there will be two main types of roads - 8 metre wide service roads and 30 metre wide haul roads. All roads inpit and around the waste rock dumps and the haul roads to and in the Tailings and Ultramafic Waste Rock Management Facility (TWRMF) will be 30 metres in width.

The 30 metre wide haul roads will allow the trafficking of 218 tonne trucks. The designed width includes an outside berm at 3.0 m wide and 1.8 m high and ditches at 2.5 m for two lane traffic to accommodate 218 tonne Komatsu 830E haul trucks as shown in Figure 2.9-6. Ramps were designed at a maximum gradient of 10%.



Truck Parameters		
Largest Truck		830E
Standard Tire		40.00 R57
Tire Size	m	3.6
Operating Width	m	7.32
Road Width		
Double Lane (3.35x Op Width)	m	24.5
Berm		
Height (1/2 of largest tire)	m	1.8
Slope (H:V)		0.85
Berm Width	m	3.0
Ditch		
Depth	m	0.8
Slope (H:V)		1.5
Ditch Width	m	2.5
Total Road Width	m	**30.0**

Source: Wardrop, 2009b

Figure 2.9-6 Road Width Design

2.9.6 Mining Method

2.9.6.1 Drilling

The initial drill requirements will consist of two blasthole drills capable of drilling 270 mm (10 5/8 inch) diameter blastholes. A 8.4 m x 8.4 m pattern has been selected for waste, and a 8.0 m x 8.0 m pattern for ore (Wardrop, 2009b). A diesel-powered hydraulic percussion track drill will be used for secondary blasting of oversize material, sinking cut drilling, pre-shearing, etc. Details on anticipated penetration and drilling rates and anticipated yearly drill net operating hours available per unit are given in Appendix 2.9.

2.9.6.2 Blasting

An explosive supplier will be selected to erect an explosive plant and storage facility on site. Under the supervision of the mine blasting foreman, the supplier will be contracted to supply, deliver, and load explosives into the blastholes. The drill blast foreman will oversee the contractor's blasting crew who will prime, stem, and tie-in blastholes (Wardrop, 2009b).

VNI will not be responsible for the manufacturing of explosives and will not own the Explosive Plant. The Explosive Plant will produce ANFO.

Blasting parameters and the expected blasthole productivity are set out in Table 2.9-10. Estimates of the overall explosive consumption are based on using a 70% ANFO and 30% emulsion mix product.

Table 2.9-10 Blasthole Hole Parameters and Drill Productivity

Blast Hole Drill Productivity	**Units**	**Rock Type**		
		Dolomite	**Basement Waste**	**Ore**
Hole Diameter	cm	26.9	26.9	26.9
Bench Height	m	12.0	12.0	12.0
Sub grade	m	1.7	1.7	1.6
Powder Factor	kg/t	0.21	0.21	0.24
Bank Density	t/m^3	2.7	2.7	2.61
Rock Mass per Hole	t	2,286	2,286	2,006
Spacing and Burden	m	8.4	8.4	8.0
Drilling Rate	**m/h**	**45**	**32**	**32**

Source: Wardrop, 2009b

The preservation of rock mass integrity will allow for the development of the steepest wall slope. This will be achieved by applying careful blasting methods. A buffer blasting practice will be implemented adjacent to the final pit walls to minimize damage to them due to blasting (Wardrop, 2009b).

2.9.6.3 Waste and Ore Loading

The initial loading fleet will consist of three 22 m^3 (30 yd^3) electric cable shovels and one 20 m^3 (25 yd^3) front end loader. The shovel size has been matched with 218 tonne trucks to provide a swing cycle of 37 seconds and a total truck load time of 3.9 minutes (Wardrop, 2009b). The loader has been matched with 218 tonne trucks to enable loading in eight passes for handling rock and a digging cycle of 47 seconds for each material (net productive operating time). Sample shovel productivity calculations and the yearly shovel net operating hours available per unit are given in Appendix 2.9.

Material weight in sample calculations was assumed to be the average for all materials ranging from 1.90 t/bank m^3 to 2.70 t/bank m^3 with most being greater than 2.40 t/bank m^3. The base productivity was assumed to occur under normal ideal loading condition. Productivity for both ore and sandstone materials were assumed to be 90% of the base productivity as the benches will be mined at half the height of normal conditions (6 m) to improve selectivity, resulting in increased shovel delays (Wardrop, 2009b).

2.9.6.4 General Hauling Conditions

The 218 tonne haul trucks were selected to match the 22 m^3 (30 yd^3) electric hydraulic shovels and 20 m^3 (25 yd^3) front end loaders in determining the number of trucks required for each operating year.

Anticipated yearly truck net operating hours available per unit are given in Appendix 2.9. Estimated cycle times are based on measured haulage profiles from pit sources by mining phase to destinations based on material types (Wardrop, 2009b). Truck productivities were estimated based on expected operating conditions, haulage profiles, production cycle times. Cycle times were calculated using Caterpillar Inc.'s Fleet, Production and Cost (FPC) software.

Each bench for each phase was assigned a specific cycle time according to its final destination. A table of all the cycle times is given elsewhere (Wardrop, 2009b). All cycle times include an average loading time of 3.9 min, a loader exchange of 0.3 min, and a dump time of 0.5 min.

A rolling resistance of 3% was used on most roads, but the first 200 m in-pit and the last 200 m on the dump were increased to 5% to simulate rougher conditions. All ramps were assigned a grade of 10% in the pit and on the dumps. A maximum speed of 40 km/h was used in most conditions but was reduced to 30 km/h when on the main ramp in the pit for safety (Wardrop, 2009b).

2.9.7 Pushback Width

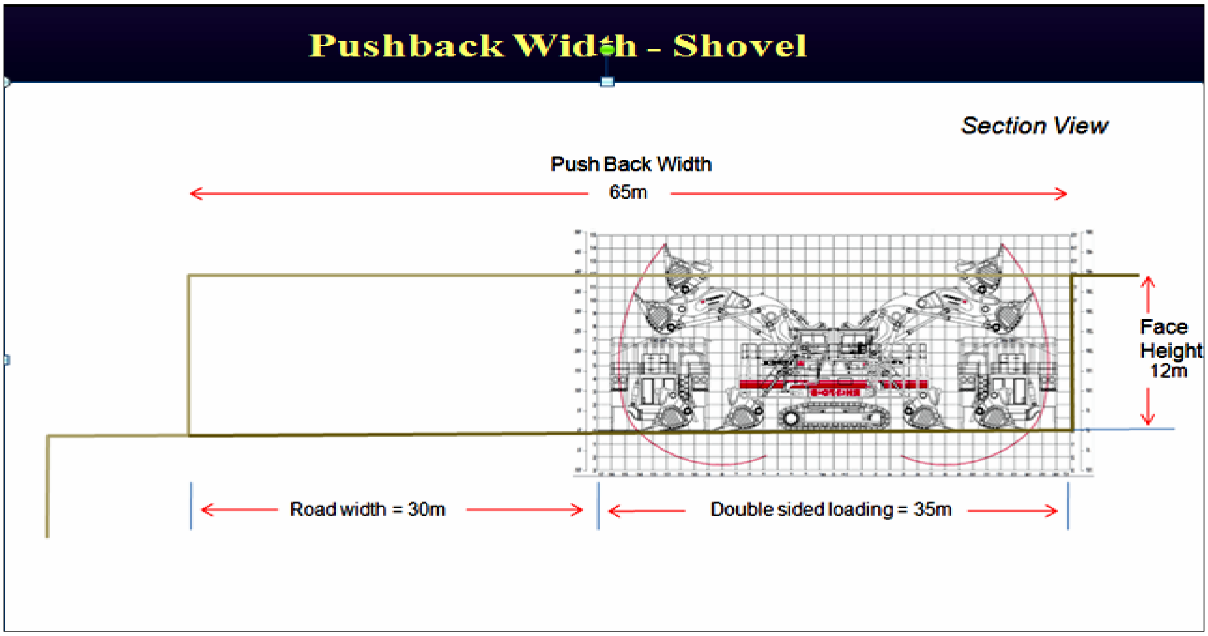
Figure 2.9-7 shows the proposed pushback width. The approximation of the pushback width was determined based on:

- the selection of the Komatsu PC4000, as the electric hydraulic shovel, loading a Komatsu 830E haul truck;
- a minimum double-side loading width of an electric hydraulic shovel at 35.0 m, which will accommodate a turning width of 28.4 m for the Komatsu 830E haul truck; and
- a 30 m haul road width.

The proposed minimum pushback width is the sum of the minimum double-side loading radius at 35 m, and the haul road width at 30 m, for a total width of 65 m.

2.9.8 Mining Equipment Selection

Due to the relatively short mine life, the low capital cost of smaller electric hydraulic shovels and Manitoba's low power costs, a fleet consisting of 22 m^3 (~30 yd^3) electric hydraulic shovels, 20 m^3 (~25 yd^3) loaders and 218 tonne trucks was determined to be the most economic equipment choice in combination with 270 mm (10 5/8″) blasthole drills, supplemented by auxiliary equipment such as tracked dozers, rubber tired dozers, graders and other minor equipment (Wardrop, 2009b).



Source: Wardrop, 2009b

Figure 2.9-7 Pushback Width Showing Shovel

In order to meet a production rate of 10,000 t/d of ore, ten 218-tonne trucks, three 22 m^3 bucket shovels, and one 20 m^3 loader will initially be required. This will ramp up to 19 trucks in "Year 3" (2016), 15 owned, 4 rented/leased The yearly equipment requirements are shown in Table 2.9-11. Yearly shovel and truck net operating hours per unit and sample shovel productivity calculations are provided in Appendix 2.9.

A comprehensive list of equipment for the mine site is given in Table 2.9-12.

2.9.9 Pit Dewatering

The progressive development of the open pit will result in increased water infiltration from precipitation and groundwater inflows. As much as 20% of groundwater flow is expected to (worst case) to seep into the open pit, despite that the dewatering wells will be operating (Wardrop, 2009b).

As the pit deepens and widens, it will be necessary to control water inflow through the construction of in-pit dewatering systems such as drainage ditches, sumps, pipelines and pumps.

To minimize groundwater infiltration and surface run-off, a ring road and berm complete with drainage ditches will be constructed to divert water away from the pit.

Table 2.9-11 Truck, Shovel and Loader Requirements by Year

Equipment	2011 Contractor	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021
Trucks											
Phys. Avail.		95.00%	92.04%	89.19%	86.44%	83.78%	81.23%	78.76%	76.39%	75.00%	75.00%
Utilization		82.2%	79.3%	76.4%	73.7%	71.0%	68.5%	66.0%	63.6%	62.2%	62.2%
Productivity		606	528	460	440	397	364	333	311	289	267
Number Req’d		10.0	15.0	17.0	17.0	19.0	16.0	10.0	7.0	7.0	1.0
Shovels											
Phys. Avail.		92.00%	92.00%	89.00%	86.00%	83.00%	80.00%	77.00%	74.00%	74.00%	74.00%
Utilization		71.6%	71.6%	68.7%	65.8%	62.9%	60.0%	57.1%	54.1%	54.1%	54.1%
Productivity		2,758	2,758	2,758	2,758	2,758	2,758	2,758	2,758	2,758	2,758
Number Req’d		2.5	3	3	3	3	3	2	1	1	1
Loaders											
Availability		90.00%	90.00%	90.00%	89.38%	88.66%	87.94%	87.22%	86.50%	85.78%	85.06%
Utilization		85.0%	85.0%	84.5%	84.0%	83.5%	82.0%	81.5%	81.0%	80.5%	80.0%
Productivity		1,495	1,495	1,486	1,478	1,469	1,442	1,434	1,425	1,416	1,407
Number Req’d		0	1	1	1	1	1	1	1	1	1

Source: Wardrop, 2009b

Table 2.9-12 Site Wide Equipment List

PHASE	OPERATION
EQUIPMENT	Quantity
Hydraulic Backhoe – Caterpillar 385CL (4 Cu.m.)	1
Electric Hydraulic Shovel – Komatsu PC4000E	2
Utility Backhoe – Caterpillar 336DL (2 Cu.m.)	1
218 Tonne Haul Truck – Komatsu 830E – AC	15
Wheel dozer – Caterpillar 854K	1
Grader – Caterpillar 16M	1
Track Dozer c/w Ripper – Caterpillar D10T	3
Blast hole Stemmer – Caterpillar 262C	1
Front end loader – Le Tourneau L-1350	1
Electric bench drill – Atlas Copco PV351E Open Pit	2
Secondary drill – Sandvik Pantera DP 1500	1
Ambulance – Ford E-150 Commercial	1
Fire Truck – Pierce Velocity™ Custom Chassis	1
Vibratory compactor – Caterpillar CS56	1
Bus – ABC TD 925	2
Rough Terrane Forklift – Sellick S160	1
Shop Forklift – Hyster H100FT	1
Pick-up truck – Ford Ranger	9
Pick-up (crew cab) truck – Chevrolet Silverado 2500HD	9
Hiab truck (crane picker) – National 880D	1
Welding truck, Lube/fuel truck, Mechanics truck	6
Tire Handler – Caterpillar 980H	1
Integrated tool carrier – Caterpillar IT38G	1
Water truck – Caterpillar 785D	2
Sanding truck – Komatsu HD325-7	1

In the pit, dewatering sumps will be utilized to contain groundwater and storm water run-off, which will be pumped directly to the diversion ditches and into the Polishing Pond. The in-pit pumping requirements will vary on an annual basis and will increase as the catchment area increases with successive pushbacks heading towards the ultimate highwalls.

Based on pumping tests conducted by Golder Assoicates, a dewatering well system has been designed, which is detailed in Section 7.6. The design consists of 12 dewatering wells located at a distance of approximately 300 m to 400 m along the crest of the ultimate open pit, pumping simultaneously from the limestone and sandstone units. The total pumping rate for the wellfield is predicted to be approximately 40,000 m^3/day (7,300 USgpm), and the average pumping rate for an individual well is estimated to be about 3,300 m^3/day (600 USgpm) (Golder Assoicates, 2008b). The associated drawdown cone, defined using a 1 m drawdown contour, is predicted to extend laterally in the limestone to a distance of approximately 5,000 to 6,000 m from the proposed open pit. Based on a series of sensitivity analyses conducted, Golder Associates (2008b) predicted that the actual dewatering rate for the entire wellfield could vary from 25,000 m^3/day (4,600 USgpm) to 90,000 m^3/day (16,500 USgpm).

For design purposes, it was assumed that pit dewatering would be at a rate of 40,000 m^3/day consisting of 32,000 m^3/day from the dewatering wells and 8,000 m^3/day from the pit pumping system.

2.10 Milling Processes

2.10.1 Summary

The nickel ore processing plant is designed to process nickel ore at a nominal rate of 10,000 t/d. The process will consist of the following conventional operations (Wardrop, 2009b):

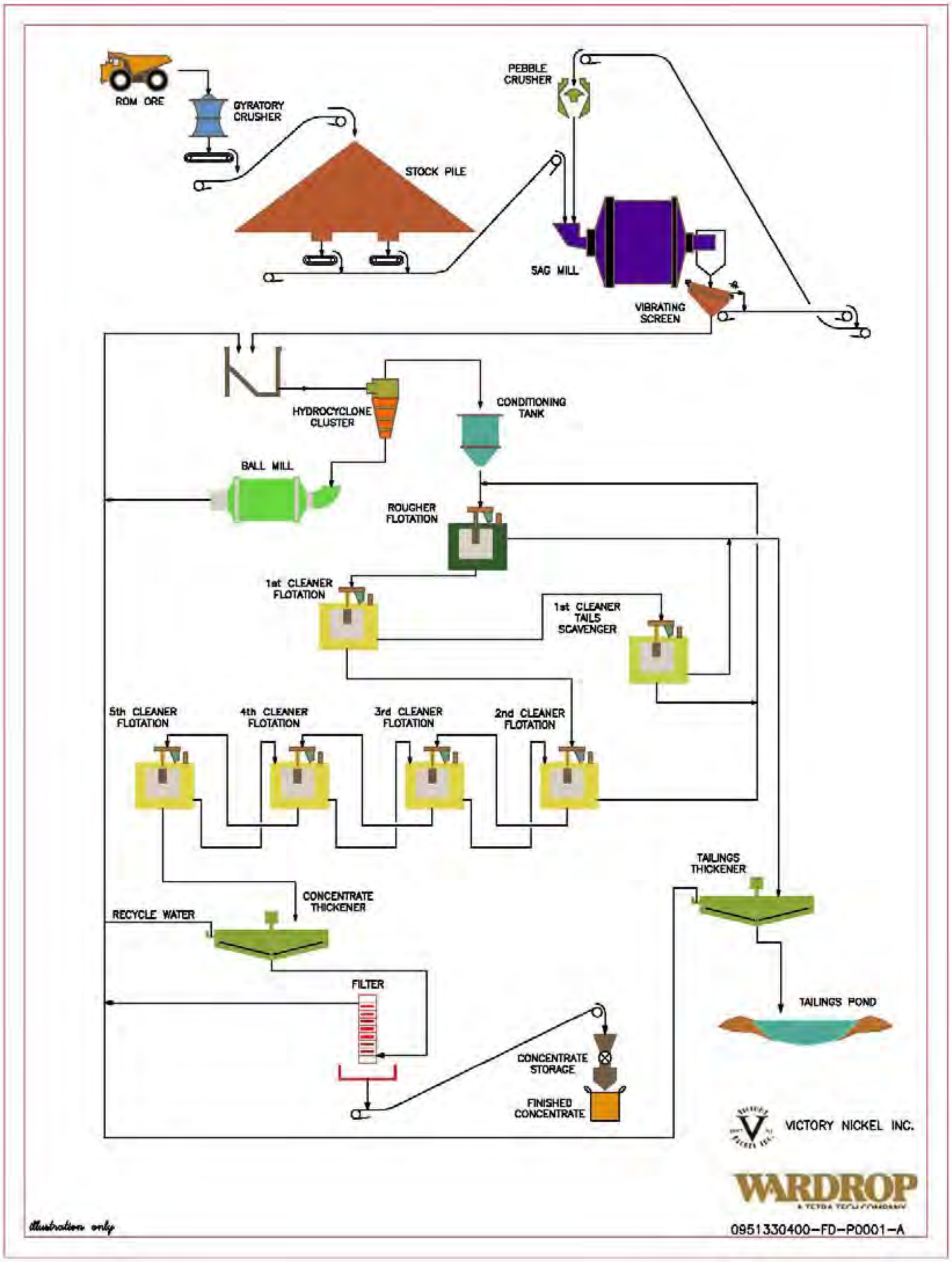
- primary crushing;
- ore stockpile and reclaim;
- grinding circuit and size classification;
- rougher/scavenger/cleaner flotation using reagants;
- concentrate dewatering using filter presses, bagging and load out; and
- tailings thickening.

Major design criteria for the Nickel Ore Processing Plant are outlined in Table 2.10-1 and Figure 2.10-1 gives a simplified process flow sheet. Brief descriptions of the individual process components are given in the next subsections.

Table 2.10-1 Major Design Criteria

Criteria	Qty	Unit
Operating Days per Year	365	d
Overall Plant Availability	95	%
Primary Crushing Rate	502	t/h
Primary Crusher Availability	83	%
Ore Specific Gravity	2.65	
Processing Rate (at 100% availability)	416.7	t/h
SAG Mill Feed Size, 80% Passing	130,000	µm
SAG Mill Product Size, 80% Passing	1,072	µm
SAG Mill Circulating Load	16	%
Ball Mill Circulating Load	250	%
Primary Grind Size, 80% Passing	68	µm
Primary Bond Work Index (BWI)	14.9	kWh/t
Abrasion Index	0.065	
Concentrate Thickener Underflow	70	% Solids
Final Concentrate Moisture Content	8.6	%

Source: Wardrop, 2009b



Source, Wardrop, 2009b

Figure 2.10-1 Simplified Flowsheet of the Nickel Ore Processing Plant

2.10.1.1 Crushing Operations

The ore from the open pit will be transported to the primary crusher by 218 tonne haul trucks. The crushing will be done with a primary gyratory crusher and hydraulic rock breaker capable of crushing the ore to an optimal size for grinding. The primary gyratory crusher facility is designed to crush ore at an average rate of 502 t/h (83% availability). The crusher feed size will be approximately 1,000 mm with a product size of 80% passing 130 mm. The crushing plant will operate on a 24 hour cycle. A primary crusher apron feeder will feed a transfer conveyor to deposit the material to the ore stockpile (Wardrop, 2009b).

A fogging dust suppression system will be incorporated into the primary crusher facility to minimize the amount of dust generated during crushing and handling. This will be an air/water system to minimize the use of fresh water (Wardrop, 2009b).

2.10.1.2 Ore Stockpile

The nickel ore stockpile will have a live capacity of 10,700 t. The ore will be reclaimed from the stockpile by two 1.219 m (48”) x 8.0 m (26’3”) apron feeders. Each reclaim apron feeder will feed a single semi-autogenous grinding (SAG) mill primary feed conveyor at a nominal rate of 250 t/h and a mechanical wheeled loader will trim the stockpile.

The stockpile will be equipped with a fogging dust suppression system to minimize the amount of dust generated during material handling, as will all transfer points along the discharge conveyors (Wardrop, 2009b).

2.10.1.3 Grinding and Classification

The grinding circuit, consisting of one semi-autogenous grinding (SAG) mill and one ball mill, will grind the ore prior to flotation and will reduce the ore to 80% passing 68 µm. The maximum SAG mill feed size is 153 mm based on the maximum product size of the primary crusher. The SAG mill product will be fed to a vibrating screen. The screen oversize will be recycled through a pebble cone crusher for intermediate crushing to 80% passing 16 mm. The crushed product will be conveyed back to the SAG mill feed conveyor. The screen underflow will gravity feed to a hydrocyclone cluster feed pump box, which will feed a hydrocyclone cluster (Wardrop, 2009b).

The hydrocyclone cluster will classify the underflow of the vibrating screen and the ball mill discharge. The hydrocyclones’ underflow will feed an underflow launder, and will then gravity-flow to the ball mill feed chute at a recirculating load of 250%. The hydrocyclone cluster overflow launder will gravity-flow to the conditioning tank at the start of the rougher flotation (Wardrop, 2009b).

Potassium amyl xanthate (PAX) and sodium hexametaphosphate (SHMP or Calgon) will be added to the ore in the grinding stage to enhance the flotation performance downstream. PAX is a collector used in flotation and SHMP is a dispersant which acts to prevent the talc (MgO) in the ore from floating (Wardrop, 2009b).

2.10.1.4 Flotation

The flotation circuit is designed to produce a high-grade nickel concentrate and final tailings. The flotation circuit will be conventional and will consist of one bank of rougher cells, one bank of scavenger cells, and five banks of cleaner cells.

The major equipment in the flotation circuit will include (Wardrop, 2009b):

- one 120 m^3 conditioning tank;
- eight 160 m^3 rougher flotation tank cells;
- eight 50 m^3 first cleaner flotation tank cells;
- two 30 m^3 first cleaner scavenger flotation tank cells;
- six 50 m^3 second cleaner flotation tank cells;
- four 50 m^3 third cleaner flotation tank cells;
- four 10 m^3 fourth cleaner flotation tank cells; and
- four 5 m^3 fifth cleaner flotation tank cells.

PAX and methyl isobutyl carbinol (MIBC), a frother, will be added at five different stages to the rougher flotation circuit. Depramin C (CMC), which is a depressant for MgO, will be added to the cleaner flotation cells to make sure the concentration of MgO in the concentrate is within acceptable limits (Wardrop, 2009b).

Flotation optimization will be provided by 12 on-stream samplers, 2 particle size analyzers and an online X-ray analyzer. An automatic sampling system will collect samples from various product streams for on-line analysis and daily metallurgical accounting. Particle size analyzers will provide main inputs to the control system and monitor equipment production. The online X-ray analyzer will be used to monitor the performance of the flotation process to optimize concentrate grade and nickel recoveries (Wardrop, 2009b).

2.10.1.5 Dewatering and Drying

The final flotation concentrate will be thickened to 70% solids in a 5 m conventional concentrate thickener. The underflow will be stored in a 5.2 m diameter stock tank, which will feed a filter press. The stock tank will have the capacity to accumulate 24 hours of production. The thickener overflow will be recycled and pumped to the process water tank.

The slurry in the stock tank will be fed to a filter press at a solids feed rate of 5.26 t/h (3.7 m^3/h) to dewater the concentrate cake to a moisture content of 8.6% by weight and a thickness of 40 mm. A dryer was not incorporated into the design because the filter press is capable of dewatering the final concentrate to the low moisture content of 8.6%. However, space for a potential dryer was incorporated into the plant layout (Wardrop, 2009b).

The concentrate filter cake will flow by gravity from a hopper to a concentrate belt feeder which will feed a bagging machine. The bagging machine is designed to operate 10 h/d and will bag 2 t concentrate bags. During bagging machine shutdown, the concentrate storage hopper capacity will allow storage of 14 hours of concentrate production.

A 32 m diameter high rate tailings thickener will clarify the final tailings. The thickener underflow of 45% solids will be pumped to the Tailings and Ultramafic Waste Rock Management Facility (TWRMF) and the overflow will be recycled for process water.

2.10.2 Nickel Ore Plant Layout

Figure 2.10-1 illustrates the Nickel Ore Plant Layout. The SAG and ball mill products will discharge into a common pump box. Since the hydrocyclone cluster underflow launder feeds the ball mill feed chute, the hydrocyclone cluster was located on the north side of the ball mill.

The flotation cells will be located in one area, serviced by a single overhead crane. Each bank of flotation cells was laid out linearly to maximize efficient operation of the cells and eliminate short-circuiting. Pumps and pump boxes will be positioned around the exterior of the flotation area for ease of maintenance and access.

The flotation cell banks will be positioned to decrease the length of pipelines and to decrease the amount of pumps and pump boxes. For example, the fourth cleaner bank of cells will be located above the fifth cleaner cells, so concentrate and tailings can flow by gravity and eliminate the need for pumps and pump boxes. The scavenger cells will also be slightly elevated to allow the concentrate and tailings to gravity flow to the desired locations.

The reagent area will be located on the west side of the building to minimize pump head and pipe lengths.

A central control room located between the grinding and flotation areas will allow control room operators to oversee the operations in both areas.

An assay and metallurgical laboratory will also be incorporated into the mill building to perform laboratory tests.

2.10.2.1 Water and Air Supply

Fresh water will be supplied by an 11 m diameter and 10.4 m high storage tank with a total capacity of 757 m^3 (200,000 gal). The lower portion (75%) will be used for fire water while the upper portion will be used for reagent mixing water, grinding mill cooling water, pump gland water, the potable water treatment system, and fresh water supplied to the Frac Sand Plant and Nickel Ore Processing Plant. Dewatering wells will be utilized to supply water to the fresh water tank.

A fresh water supply pump house will supply all fresh water to the plant. The supply will comprise three separate systems. Each of these systems will consist of one pump capable of satisfying the demand and one spare pump of identical size. The capacity of the pump house is shown in Table 2.10-2.

Table 2.10-2 Pump House Capacity

Service	Requirement	To be installed
Potable water	5 m^3/hr (22 gpm)	2 @ 5 hp
Gland water	75.6 m^3/hr (332 gpm)	2 @ 25 hp
All other fresh water	50 m^3/hr (220 gpm)	2 @ 25 hp

Source: Wardrop, 2009b

A secondary fresh water tank will be located in the reagent area and used strictly for reagent mixing. Mill cooling water from the grinding area will be recycled to the reagent water tank and fresh water will be supplied to the reagent tank to maintain a specific level depending on consumption (Wardrop, 2009b).

Process water will be supplied by an 11 m diameter and 10.4 m high storage tank. The process water tank will be supplied from the fresh water tank, concentrate thickener overflow, tailings thickener overflow, and water recycled from the Polishing Pond. Process water will be required for all flotation cells (launders) and mill grinding areas, as well as the concentrate filter press (Wardrop, 2009b).

A raw water supply pump house will supply all raw water to the plant, at a required rate of 1440 m^3/hr (6339 gpm). The water will be pumped with one 300 hp pump rated at 1600 m^3/hr (7000 gpm). A second identical pump will be installed for redundancy (Wardrop, 2009b).

The fresh and raw water pump houses will be insulated and heated and will have crawl-beams and electrical hoists, where needed, to facilitate removal of the pumps and motors.

The mill building air supply will be produced by two plant air compressors (one standby), a dedicated filter press compressor, and three aeration blowers (two operating, one standby). The plant air compressor will supply process air for the mill lubrication system, concentrator utility hoses, reagent area and plant valves and piping leaks. The plant air compressor will also supply air to an instrument air dryer which will produce instrument air for the pneumatic valves, reagent dust collectors, assay laboratory bag house, laboratory equipment, and the mill pneumatic clutches (Wardrop, 2009b).

Low pressure air will be supplied to the flotation circuit by two operating aeration blowers. A standby blower will be utilized to generate enough capacity in the event of a blower failure.

2.10.2.2 Typical Reagent Consumption

Flocculants will be used in each thickener to assist in settling and generating a precipitate from solution. Reagents including potassium amyl xanthate (PAX) and sodium hexametaphosphate (SHMP or Calgon) will be added to the ore in the grinding stage to enhance the flotation performance downstream. Methyl isobutyl carbinol (MIBC) and depramin C (CMC) will be added to the cleaner flotation to increase concentrate quality.

The projected reagent addition rates are given in Table 2.10-3 and the storage and preparation of reagants is outlined below. The Material Safety Data Sheets (MSDS) for these chemicals, including toxicological information, are provided in Appendix 2.10.

All reagant mixing and storage tanks will be equipped with low and high level indicators and instrumentation to ensure that spills do not occur during preparation and normal operation. In the event of a spill, sump pump locations are located throughout the reagent area for proper containment. Shower and eye wash safety stations will also be installed in case of skin or eye contact during a spill. Appropriate ventilation, fire and safety protection and MSDS stations will be provided at the facility.

Each reagent line and addition point will be labelled in accordance with Workplace Hazardous Materials Information Systems (WHMIS) standards and all operation personnel will receive WHMIS training and additional training for the safe handling and use of all reagents.

2.10.2.2.1 Preparation and Storage of Reagants

Figures 2.10-2 through 2.10-5 show reagents flow sheets and Figures 2.10-6 and 2.10-7 show concentrate flocculant and tailings flocculant flow sheets. Handling methods of the various process reagents are discussed below.

Potassium Amyl Xanthate (PAX)

Potassium Amyl Xanthate (PAX) will be shipped to the Minago site in bulk 1,000 kg super sacs. The bulk PAX will be diluted to a 10% solution in a 49.2 m^3 (13,000 gal) mixing tank (Wardrop, 2009b). Each batch process will consume five bulk super sacs and will be performed once per day. Once properly mixed, the PAX solution will be stored in a 60.6 m^3 (16,000 gal) storage tank (Wardrop, 2009b). The PAX solution will be pumped from the holding tank to a distribution trough. The distribution trough will allow for proper calibration and will feed separate metering pumps for each addition point (Wardrop, 2009b).

Table 2.10-3 Reagents and Flocculants in the Mining and Milling Process

					Dosage (g/tonne)	Dosage (kg/day)
CMC	Carboxmethyl Cellulose	wood product (used to make creamy soups)	Depressant	Depressant for Talc(MgO) coats talc particles to make them hydrophilic	700	7000
PAX	Potassium Amyl Xanthate		Collector	Collector for minerals coats mineral particles to render them hydrophobic so that are attracted to air bubbles and reject water	425	4250
SHMP	Sodium hexametaphophate	Calgon (water softener)	Dispersant	Dispersant for Talc keeps talc particles from adhering to mineral particles	500	5000
MIBC	Methyl isobutyl carbinol	similar to dish soap	Frother	Frothing agent to create stable froth bubbles in flotation cells to float metal particles	70	700
Flocculant (Tails)	Anionic polyacrylamide	used in water treatment	Coagulant	used in thickeners and clarifiers to collect particles so that they will agglomerate and sink	23	227
Flocculant (Conc.)	Anionic polyacrylamide	used in water treatment	Coagulant	used in thickeners and clarifiers to collect particles so that they will agglomerate and sink	5	0.63

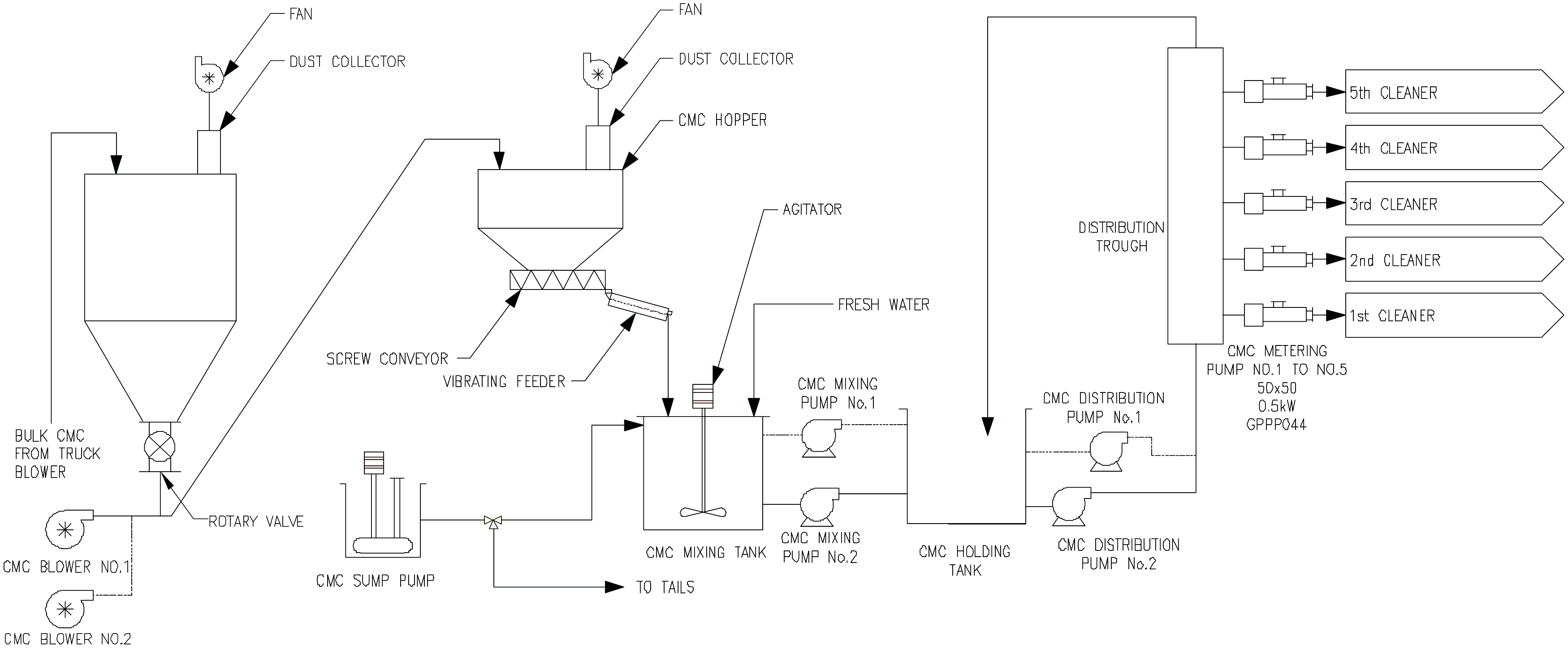


Figure 2.10-2 CMC Reagent Flow Sheet

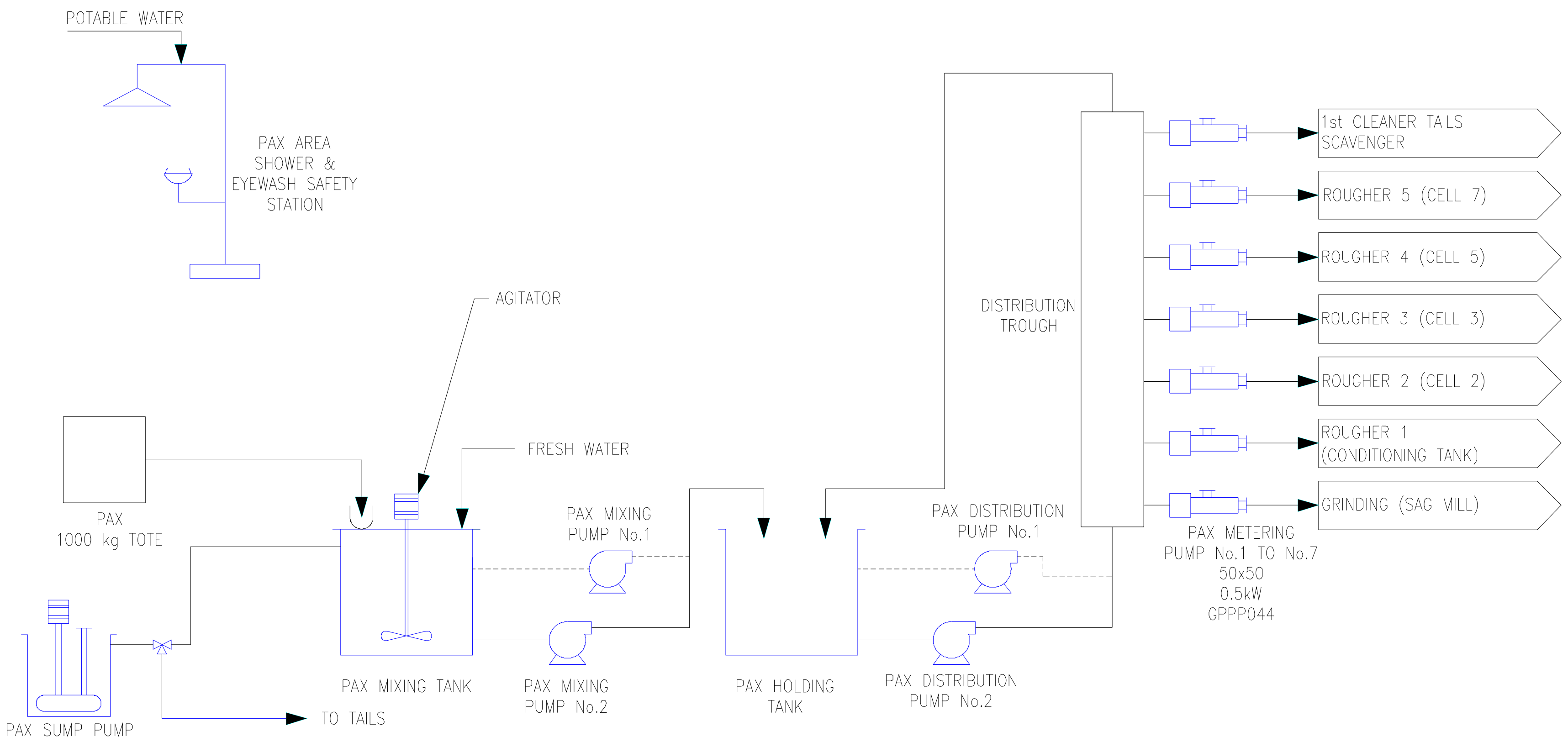


Figure 2.10-3 PAX Reagent Flow Sheet

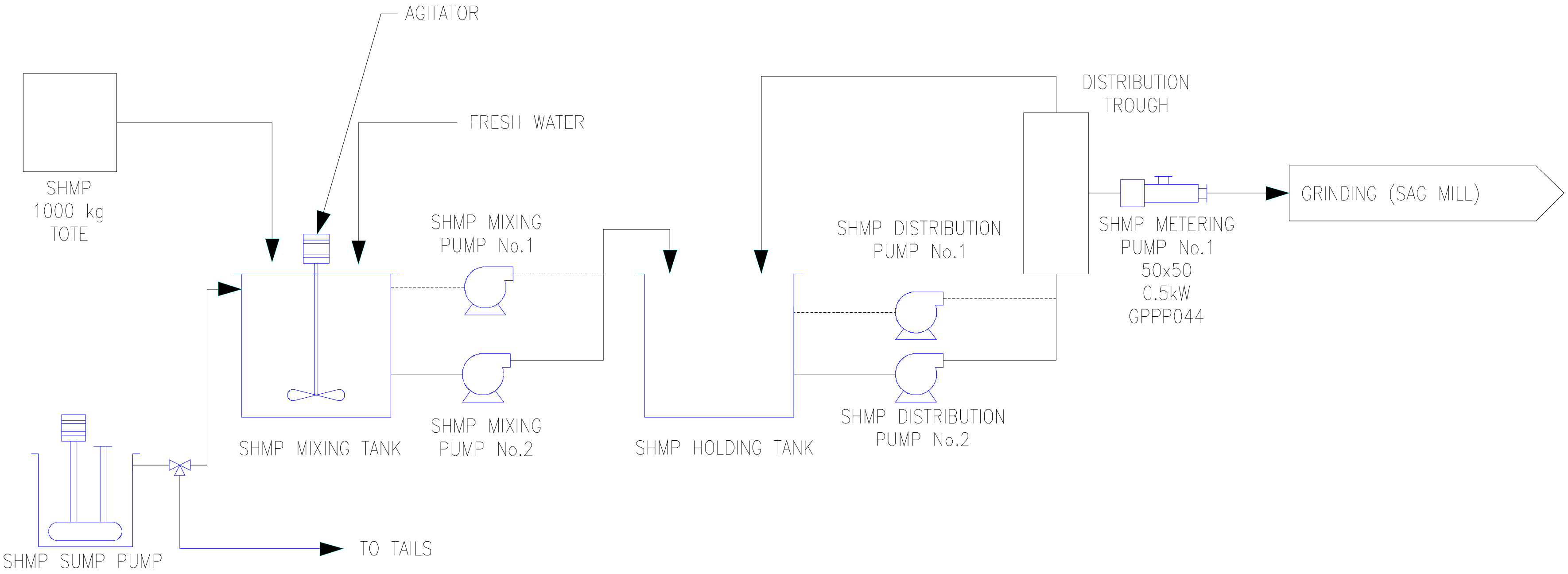


Figure 2.10-4 SHMP Reagent Flow Sheet

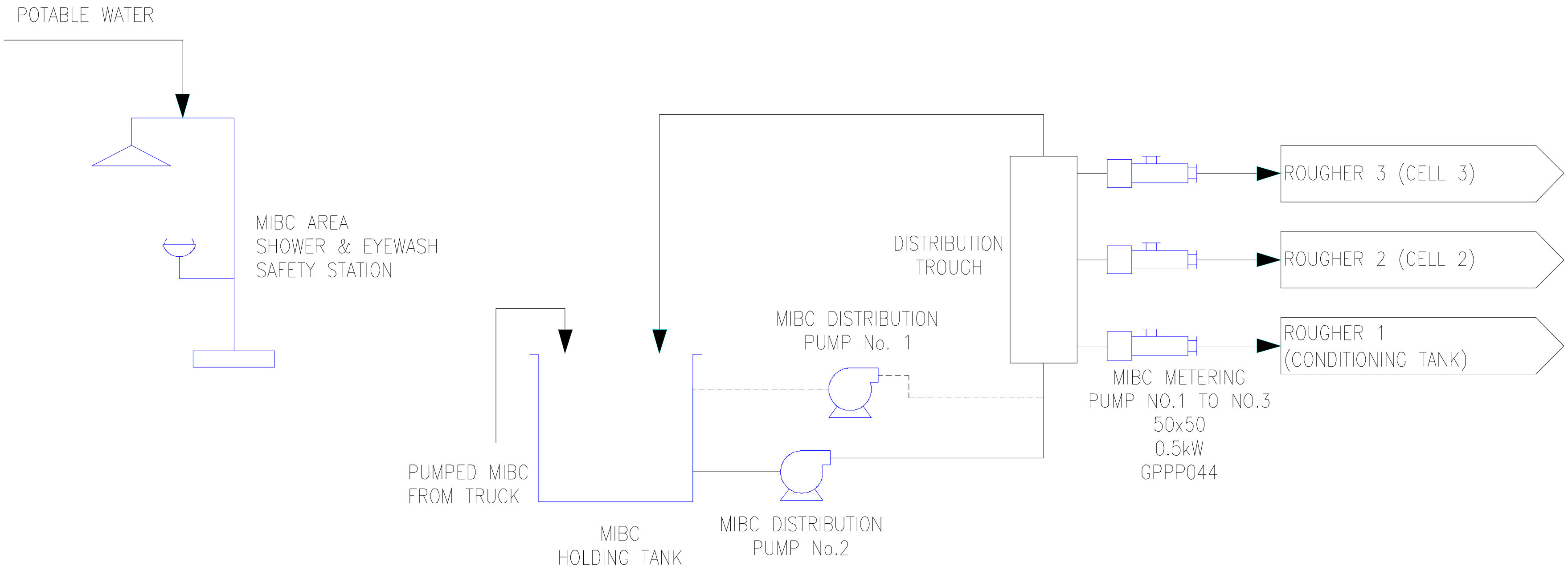


Figure 2.10-5 MIBC Reagent Flow Sheet

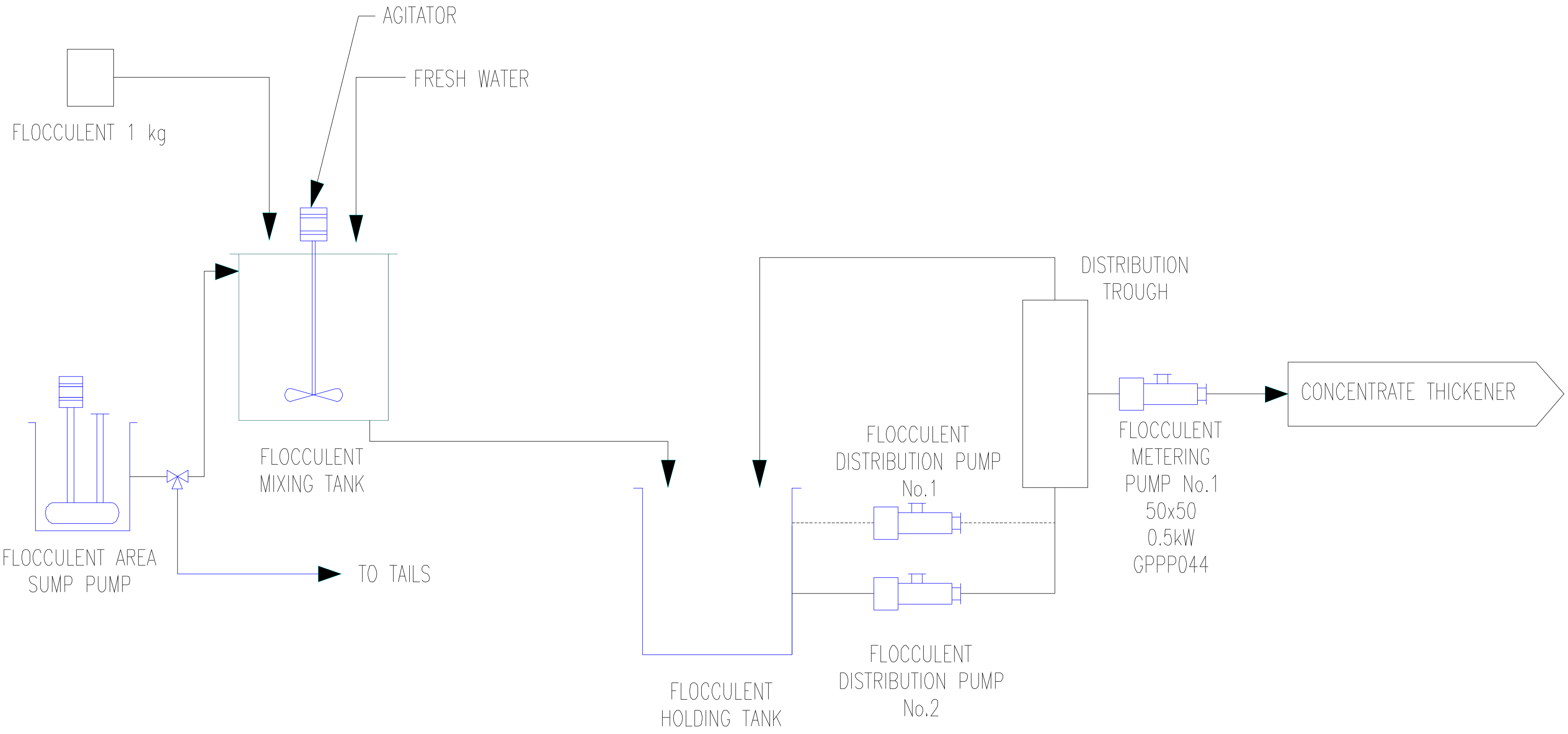


Figure 2.10-6 Concentrate Flocculant Flow Sheet

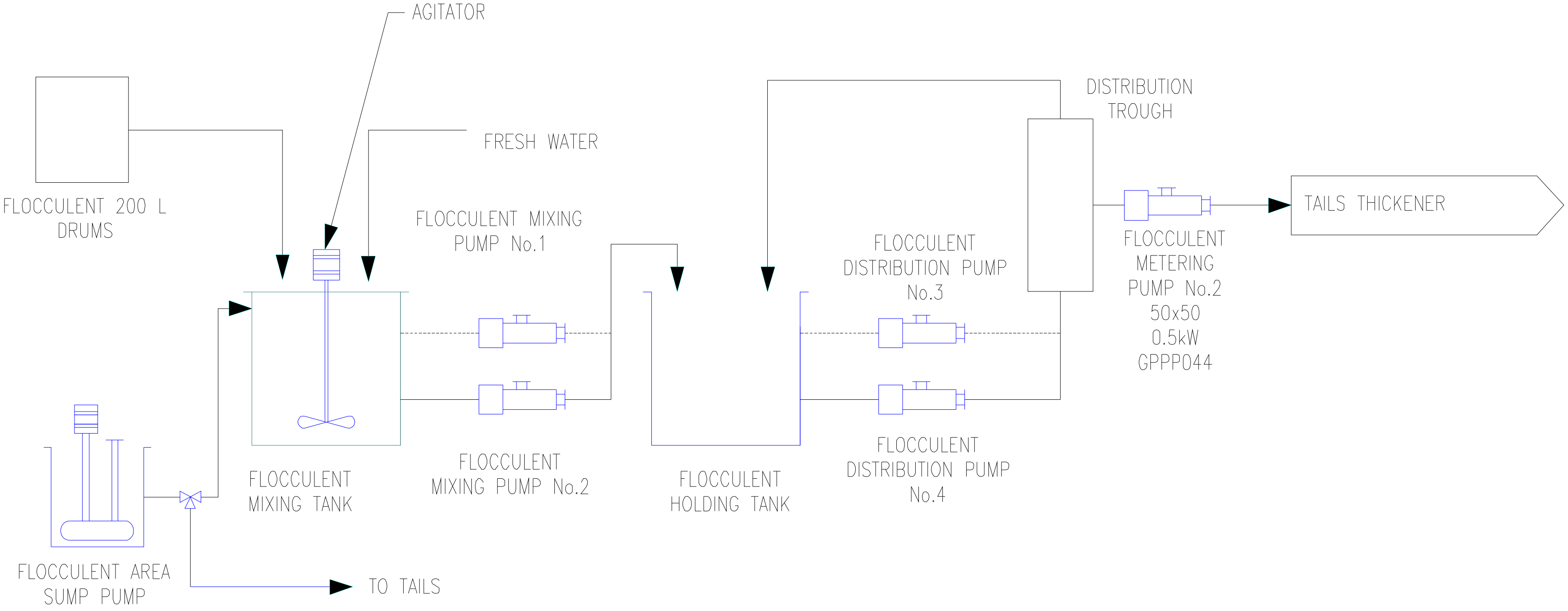


Figure 2.10-7 Tailings Flocculant Flow Sheet

Sodium Hexametaphosphate (SHMP)

Sodium Hexametaphosphate (SHMP) will be shipped in 1,000 kg bulk super sacs. The SHMP will be diluted to a 10% solution in a 56.8 m^3 (15,000 gal) mixing tank. Each batch process will consume six bulk super sacs and will need to be performed once per day. The SHMP will be stored in a 68 m^3 (18,000 gal) storage tank. The 10% SHMP solution will be pumped from the storage tank to a distribution trough by a horizontal centrifugal pump. The flow from the distribution trough will be metered through a progressive cavity pump to the addition point in the SAG mill (Wardrop, 2009b).

Methyl Isobutyl Carbinol (MIBC)

Methyl Isobutyl Carbinol (MIBC) will be shipped at 100% concentration in bulk 20 m^3 (5,280 gal) tankers, stored in a 26.5 m^3 (7,000 gal) storage tank and pumped in undiluted form to a distribution trough (Wardrop, 2009b). The distribution trough will feed separate diaphragm metering pumps, which will distribute the MIBC to each addition location (Wardrop, 2009b).

Carboxymethyl Cellulose (CMC)

Carboxymethyl Cellulose (CMC) will be delivered by 20 t bulk tanker trucks and stored in a 56.6 m^3 (2,000 ft^3) dedicated silo. Bulk CMC will be retrieved from the silo by a roots type blower to a 10 m^3 (350 ft^3) transition hopper located in the reagent preparation area. CMC will be metered from the transition hopper by a screw conveyor and vibrating feeder to an agitated 45.4 m^3 (12,000 gal) mixing tank. The 2% CMC solution will be prepared continuously and pumped to a 208 m^3 (55,000 gal) storage tank. The mixing tank will have a retention time of approximately three hours. The storage tank capacity was based on 14 hours of reagent consumption. This will allow for servicing the mixing tank agitator and pumps without affecting the CMC addition to the process. CMC from the storage tank will be pumped to a distribution trough. The flow will then be metered through separate progressive cavity pumps to each addition location (Wardrop, 2009b).

Flocculants

The concentrate flocculant **Hychem 308** or equivalent, will be shipped in 25 kg bags. The concentrate flocculant will be diluted to a 0.1% solution in a 1.1 m^3 (300 gal) mixing tank (Wardrop, 2009b). This flocculant is a non-toxic inert hydrocarbon polymer, similar to treatment used in drinking water. The polymer attracts the charged solids in the slurry, causing them to clump together - thus gaining enough mass to drop out of solution via gravity.

Each batch process will consume 1 kg of concentrate flocculant and will be performed every second day. After mixing, the 0.1% solution will be pumped to a storage tank with a capacity of 1.5 m^3 (400 gal). Stored concentrate flocculant will be pumped to a distribution trough. A progressive cavity pump will pump the required amount of flocculant from the distribution trough to the concentrate thickener.

The tailings flocculant, **Mag 10**, will be shipped in 200 L drums containing 91% active flocculant. The tails flocculant will be diluted to a 0.5% solution in a 38 m^3 (10,000 gal) mixing tank. Each batch process will consume one drum per day and will be prepared once per day. After mixing, the Mag 10 flocculant will be stored in a 45.4 m^3 (12,000 gal) storage tank. The Mag 10 solution will be pumped from the storage tank to a distribution trough by a low shear progressive cavity pump. A progressive cavity metering pump will meter the flow from the distribution trough to the tails thickener at a precise flow.

2.10.3 Instrumentation and Process Control

Instrumentation and process control systems will be set up to monitor and control various site operations including those related to (Wardrop, 2009b):

- the crusher/stockpile;
- the process plant;
- the tailings pump house; and
- well dewatering.

The Minago Project control system will be comprised of control and control sub-system hardware located in electrical rooms, with a dual redundant Data Communication Network (DCN) providing real time communication between the control sub-systems, remote Operator Interface Systems (OIS) and Engineering Work Stations (EWS). All critical modules of the control system will be implemented in a redundant configuration with dual redundant processing, power distribution and communications to enable uninterruptible automatic control (Wardrop, 2009b).

The central control room located in the process plant will provide site-wide control and monitoring though the use of each interconnected area control system. The crusher/stockpile area will also have the provision for local control through a local control panel.

Alarm annunciation and alarm summary displays with user-configurable alarm limits, alarm enable/disable functions, alarm logging, and acknowledgement facilities will also be provided with the control system. This will include real time and historical trending with a selectable sample time.

The main equipment and associated instrumentation are located on the following Process and Instrumentation Diagrams (P&IDs) that are given elsewhere (Wardrop, 2009b):

- Crusher/Stockpile Area:
  - ➢ Gyratory Crusher - 70000-P-101;

  - Crusher Apron Feeder - 70000-P-101;
  - Stockpile Feed Conveyor - 70000-P-101; and
  - Stockpile Apron Feeders 1 & 2 - 70000-P-101.

- Process Plant Area:
  - SAG & Ball Mill - 70000-P-102;
  - SAG Mill Feed Conveyor - 70000-P-102;
  - Pebble Crusher - 70000-P-102;
  - SAG Mill Discharge Vibratory Screen & Conveyor - 70000-P-102;
  - SAG Mill Flexiwall Conveyor - 70000-P-102;
  - Cyclone Cluster & Pumpbox - 70000-P-103;
  - Ball Mill - 70000-P-103;
  - Rougher/Cleaner/Scavenger Flotation Cells & Pumpboxes - 70000-P-104/105/106/107/108;
  - Tailings & Concentrate Thickeners & Pumpboxes - 70000-P-109/110;
  - Concentrate Filter Press, Feeder, & Bagging Machine - 70000-P-110;
  - CMC/PAX/MIBC/SHMP/ Flocculent Reagents - 70000-P-111/112/113;
  - Sample Pumps & X-Ray/Particle Analyzer - 70000-P-114;
  - Potable Water Plant - 70000-P-115;
  - Sewage Treatment Plant - 70000-P-116;
  - Emergency Genset; and
  - Air Compressors - 70000-P-119.Gyratory Crusher - 70000-P-101.

- Tailings Pump House Area:
  - Transfer Pond - 70000-P-109;
  - Transfer Well - 70000-P-109;
  - Tailings Management Area Pond - 70000-P-109; and
  - Polishing Pond - 70000-P-109.

- Well Dewatering Area:
  - Open Pit Dewatering Wells - 70000-P-117.

Additional systems which will be monitored and controlled through the central control room in the process plant include the potable water plant, the sewage treatment plant, and the backup genset.

2.10.3.1 Process Control System Recommendation

Specifications for a process control system and a Distributed Control System (DCS) system architecture were developed based on required instrumentation, as summarized in the Instrumentation Index (Wardrop, 2009b). Wardrop reoommended to use Invensys's Foxboro DCS as a process control system after a review of Programmable Logic Controller (PLC)/DCS systems available from Allen-Bradley, Emerson, Modicon, and Invensys. The Invensys's Foxboro DCS meets most of the requirements set for the Minago Project. However, since there was some concern with the digital control with this system, a combined system using a DCS system with PLC controls will likely be developed in the detailed engineering phase (Wardrop, 2009b).

2.10.4 Frac Sand Processing Plant

2.10.4.1 Introduction

The Minago Frac Sand Feasibility Study was conducted in parallel to Victory Nickel's Minago Feasibility Study. The Minago Frac Sand Feasibility Study is a result of the Preliminary Economic Assessment (PEA) (Wardrop, 2006), which identified a sandstone horizon (averaging nine metres thick) above the unconformity of the main nickel bearing serpentinite. This sandstone layer will be removed to access the nickel mineralization within the proposed open pit mine. The sandstone unit is amenable for use as a Fracturing Sand (Frac Sand) used in the oil and gas industry as it is typically comprised of small, round, uniformly sized silica sand.

Frac sands are used as part of a process to improve the productivity of petroleum reservoirs. This treatment, known as hydrofracing, is the forcing of a concoction of frac sands, viscous gel and other chemicals down a well to prop open fractures in the subsurface rocks thus creating passageways for fluid from the reservoir to the well. Frac sands function as a proppant: sized particles that hold fractures open after a hydraulic fracturing treatment.

The Minago sandstone will be mined, and then hauled to a temporary stockpile location separate from the waste dumps, where it will be processed. The Minago sandstone is not expected to require drilling and blasting to be removed, but will require additional backhoe cleanup due to the expected undulating contact at the top of the basement rocks. A backhoe will windrow the sand so that a front-end loader can easily load the material while minimizing the loss of sand due to the loaders large bucket size. The sand will be released each time mine development passes through the bedrock contact. These times are outlined in Table 2.10-4 (Wardrop, 2009b).

A separate NI-43-101, document for the Standard Disclosure of Mineral Projects was filed with Sedar to qualify the Sand Resources (Wardrop, 2009b).

Outotec Physical Separation Division (Outotec) in Jacksonville, FL, designed a Frac Sand Plant for Minago, which includes both wet and dry process plants; each containing dedicated processes for friable and non-friable ore types. The plant will be operable year round and accommodates seasonal market demand fluctuations with a capacity of 1.6 times the average production. The in-

Table 2.10-4 Final Pit Contained Sand Resource

Phase	Sand (tonnes)
Starter Pit	5,288,864
Phase 1	2,091,628
Phase 2	7,466,065
Total	**14,846,557**

Source: Wardrop, 2009b

situ sand will be processed at a feed rate of 1.5M t/y, producing different grades of frac sand at a rate of 1,142,805 tonnes of marketable sand annually (Outotec, 2008).

2.10.4.2 Laboratory and Flowsheet Development Test Work

To determine the quality of the sand and to evaluate the feasibility of the project, Wardrop arranged a series of test programs conducted by various independent laboratories. Representative Minago sand samples were tested for different standard quality parameters in accordance with the American Petroleum Institute (API) "Recommended Practice 56 - Recommended Practices for Testing Sand Used in Hydraulic Fracturing Operations, 1995".

The API parameters include (Outotec, 2008):

- Grain size: 90 wt.% of the sand must fall within a specified size range for a particular product. The generally defined frac sand products are 12/20, 20/40, 40/70 and 70/140 (defined in terms of ASTM sieve sizes);
- Sphericity and roundness: The shape of the grains. Spherical, round grains are desired;
- Crush resistance: The amount of fines generated after a product is subjected to a specified pressure;
- Acid solubility: The percentage of the material dissolved in a HCL/HF acid solution;
- Turbidity: The amount of silt and clay-sized particulate matter in the sand; and
- Clusters or agglomerated grains: The presence of clusters or agglomerated grains reduces strength of the overall sand. The API specification is < 1% clusters.

The following three different test programs were conducted between May 2007 and November 2008 (Wardrop, 2009b):

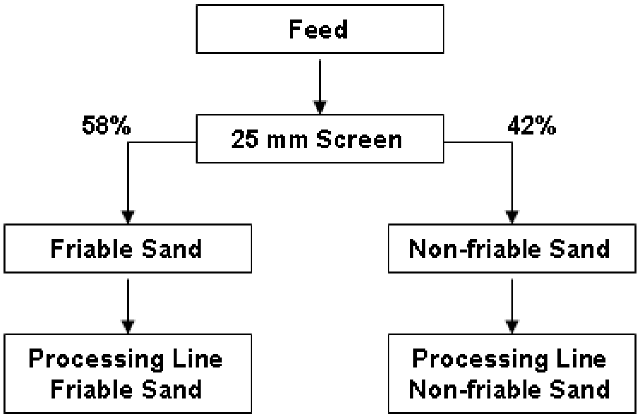
- Program 1: Between May and July 2007, Loring Laboratories Ltd. (Loring) of Calgary, AB performed mineralogical analyses, and EBA Consulting Engineers and Scientists (Material and Pavements Practice) (EBA) of Calgary, AB, performed material analyses.

- Program 2: Between May and September 2007, the Saskatchewan Research Council of Saskatoon, SK (SRC) performed mineralogical analyses, and the University of Saskatchewan performed a material analysis.
- Program 3: between December 2007 and January 2008, and between September and November 2008, Outotec Physical Separation Division (Outotec) in Jacksonville, FL performed mineralogical analyses and a material analysis.

During Program 1, each of four representative drill hole samples was split into two; the first half of each sample was provided to Loring for testing, the second half of each sample was retained. The sample from a fifth hole was split into four samples, which then formed the basis of Program 2 (Wardrop, 2009b). The results from both Programs 1 and 2 indicated low crush resistance parameters.

Outotec initiated test Program 3; wherein the remaining halved cores from the four original samples, plus representative samples from two additional holes, were delivered to Outotec and combined into a blended sample (Wardrop, 2009b). Outotec separated the sandstone into hard (non-friable) sand and consolidated (friable) sand. Using this approach, Outotec was able to improve the crush resistance parameter of the friable sand to meet API standards, thereby increasing the marketable volume. The non-friable sand was then crushed to produce a fine frac sand product suitable for shale gas applications (Wardrop, 2009b).

Subsequently, Outotec developed flowsheets for a Frac Sand Plant to meet API specifications for fracturing sand. Friable and non-friable portions will be processed separately, in two parallel circuits. A screen will be used to classify the friable ore from the non-friable (Figure 2.10-8) and only the non-friable portion of the material will be crushed.



Source: Outotec, 2008

Figure 2.10-8 Outotec Flowsheet, Separating Friable from Non-friable Sand

The parallel process is needed to ensure the non-friable products do not cause cluster related quality problems within the high value friable sand products. This approach ensures that the friable products will meet all of API's standards: sphericity and roundness, turbidity, crush resistance, low impurity level., leading to a higher volume of production of the different marketable products.

2.10.4.2.1 Friable Ore

The friable portion of Minago's sandstone deposit will be used to produce 20/40 and 40/70 frac sand meeting the API RP 56 specifications (API, 1995). The process operations required to successfully beneficiate the friable material are (Outotec, 2008):

- Attrition scrubbing,
- Desliming,
- Pre-classification,
- Drying,
- Screening, and
- Magnetic separation.

Attrition scrubbing (to break down agglomerates), desliming, and pre-classification are important sequential wet processes that will be performed first. Softer grains and coatings must be removed along with the Minus 140 Mesh particles. The presence of the Minus 140 Mesh materials would negatively impact the quality of the final sand products (Outotec, 2008).

Once the scrubbing and desliming have been completed, the sand will then be pre-classified using density separators. The pre-classified sand will be dried before it can be successfully upgraded to API quality frac sand. A fluid bed dryer will be used to remove all moisture from the sand (Outotec, 2008).

Once dried, the sand will be screened to the desired API size fractions of 20/40, 30/50, 40/70, and 70/140. The screened material will then be sent to dedicated magnetic separators for the removal of undesirable magnetic minerals and contaminants that can cause failings in API crush tests. Thereafter, API frac sand products will be ready for storage and sale (Outotec, 2008).

2.10.4.2.2 Non-Friable Ore

The following process steps were identified to successfully beneficiate the hard, non-friable sand (Outotec, 2008):

- Crushing, jaw and impactor;

- Pre-classification;
- Drying; and
- Screening.

The non-friable sand will require crushing to break down the large rocks and agglomerated particles for sufficient liberation. This step will enable upgrading in further processing stages to produce marketable products. Crushing tests were conducted to identify the suitable type and size of crushing required. At Minago, a combination of jaw and impactor crushing will be used. Jaw crushing will be used in advance of the impact crusher to allow for the processing of larger particles since impact crushers of the size needed for the feed rate are limited to approximately 100 mm top size particles (Outotec, 2008).

Following crushing, the non-friable ore will be slurried and then pre-classified using density separators to remove both the very coarse (+ 50 mesh) and very fine (–140 mesh) particles. The pre-classified nominal –50 mesh/+140 mesh sand will be filtered using belt filters and then transferred to the dry process for further upgrading (Outotec, 2008).

The pre-classified, non-friable material will be dried in a fluid bed dryer to remove all remaining moisture. This dry sand will then be screened to produce +50, 20/40, and 50/140 sand products. These products will not meet the API requirements for fracturing sand but can be used as flux sands or in applications where non-API fracturing sand is suitable (Outotec, 2008).

2.10.4.3 Frac Sand Plant Design

The Frac Sand Plant design was completed by Outotec, Physical Separation Division, Jacksonville, Fl, USA. Outotec developed an initial plant design to determine the cost of the proposed plant within an accuracy of -10% to +20%. Key process design considerations included deposit characterization and feed material assumptions, plant area capacities, operating hours for plant sections, and product quantities and grades. The initial design was followed by a Phase II revision, which included improvements to reduce the total costs and improve general plant and process operations.

The Outotec Phase II design takes into account the seasonality fluctuating demands of the frac sand market, the inclement winter weather of Manitoba, Canada, and the need to operate the full plant year-round (Outotec, 2008). The wet and dry plants will operate together in series, and are designed to operate at wet plant feed rate of 265 t/h. The overall plant has been designed to achieve a throughput that is 1.6 times average production rate, allowing plant capacity to meet periods of expected peak demand.

It is estimated that a 16-month schedule for plant completion (detailed design, procurement, construction, and commissioning) is the best-case scenario (Outotec, 2008).

The following key assumptions were made in the design of the Frac Sand Plant (Outotec, 2008):

- Plant capacity of 1,142,805 t/y comprised of 612,863 t/y API frac sand, and 529,941 t/y non-API sand, which includes 62,500 t/y of flux sand;
- Plant feed rate of 265 t/h or 1,500,000 t/y,
- Yearly operating hours – 4,822, 12 months yearly operating window for wet and dry processes;
- Friable and non-friable ores to be processed in separate, dedicated circuits;
- Two wet winter stockpiles (250,000 tonnes each) will be established to allow stockpiling of screened friable and non-friable material, during non-freezing months, for use as feed in the winter months. This is required because the screening stage will not be able to distinguish between a single large rock and a frozen lump of ore during the winter operation. The stockpiles will be built during the periods of low sales demand;
- Plants will be fed using front-end loaders via hopper and feeder systems;
- Marketable products will be held in storage silos (two-day capacity based on average production rates) and be transported via truck to the rail load-out or the marketplace; and
- Waste products will be stored in stockpiles (if solid) or send to the tailings impoundment (if slurry) via the thickener. Solid waste material will be removed by loader and truck.

Simplified block diagrams for the wet and dry Frac Sand Plants are given in Figures 2.10-9 and 2.10-10, whereas detailed material (mass and water) balance diagrams for the wet and dry Frac Sand Plant are provided in Appendix 2.10. These material balance diagrams or Process Flow Diagrams (PFDs) are listed in Table 2.10-5. Detailed Process Design Basis and the Operational Philosophy are provided elsewhere (Outotec, 2008).

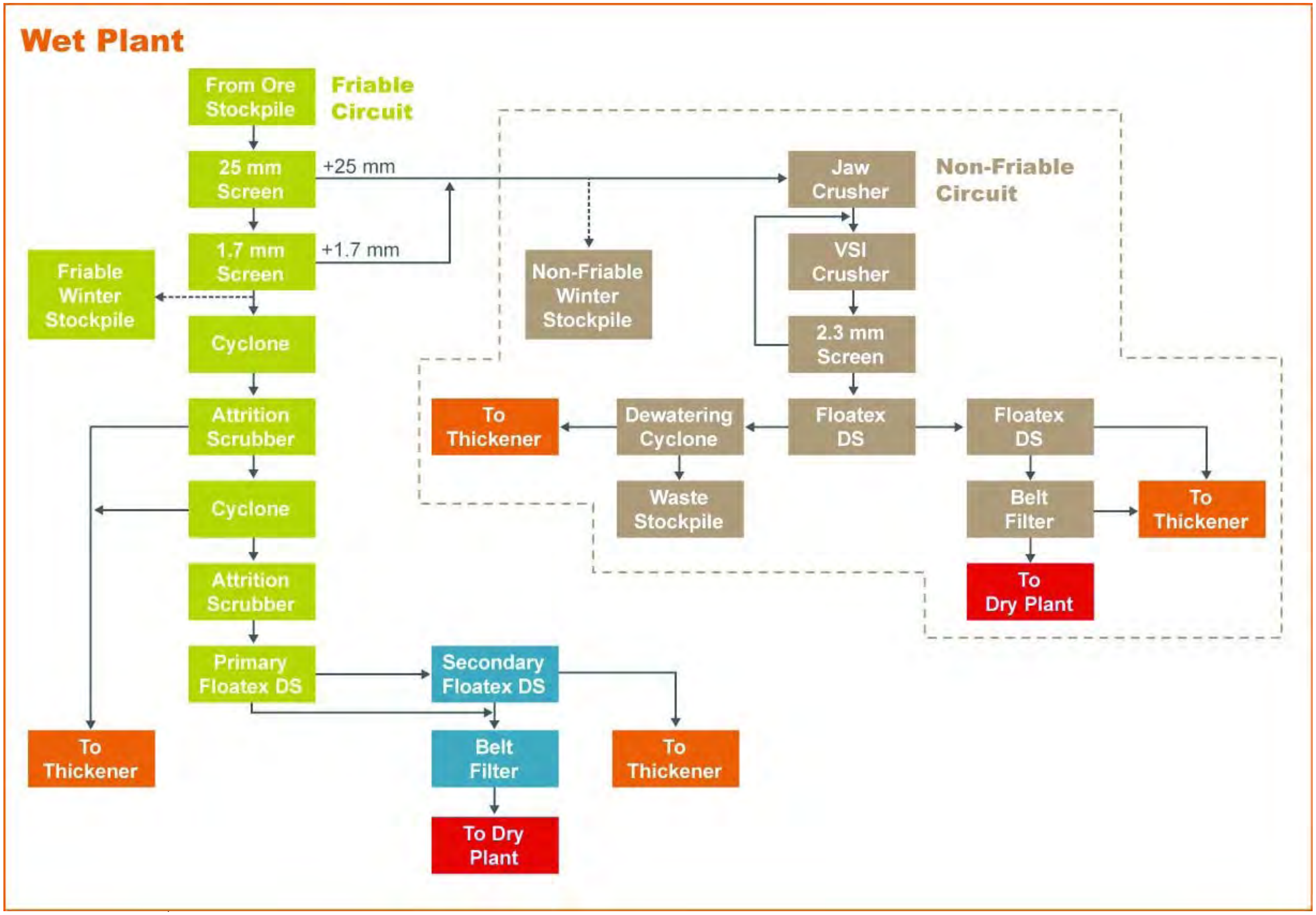


Figure 2.10-9 Flow Sheet for Minago's Wet Frac Sand Plant

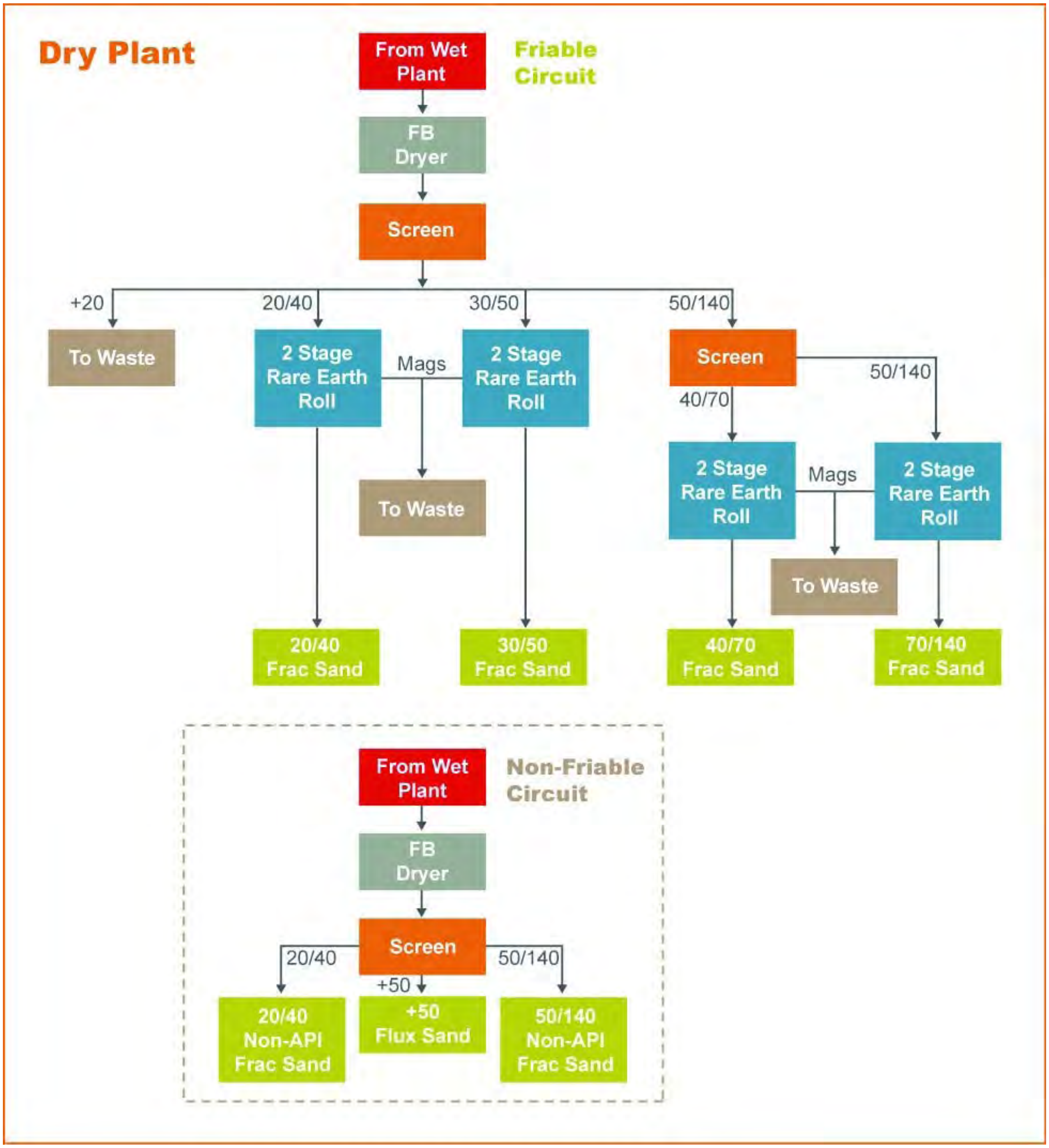


Figure 2.10-10 Flow Sheet for Minago's Dry Frac Sand Plant

2.10.4.3.1 Site Layout

Figures 2.10-11 and 2.10-12 illustrate the conceptual site layout of Minago's Frac Sand Plant. Figure 2.10-11 shows the overall site plan with winter stockpiles while Figure 2.10-12 details the proposed plant area and buildings. The plant site will require approximately 250 m x 250 m.

2.10.4.3.2 Electrical Design

The electrical design for the Frac Sand Plant will interface with the existing electrical infrastructure. The Frac Sand Plant will draw power from Minago's primary transformers and bring it to dedicated motor control centers (MCCs) in the wet and dry plants. The MCCs will include all of the appropriate secondary transformers to provide power to the operation at 600 volts, 240 volts and 120 volts. In addition, MCCs will contain all appropriate switchgear, starters, breakers, etc. for the various pieces of electrical equipment operating in the plant. It was assumed that all starters would be DOL type (Outotec, 2008).

A combination of remote and local start-stops will be used as appropriate throughout the plant, with suitable isolation stations for safe operation and maintenance (Outotec, 2008).

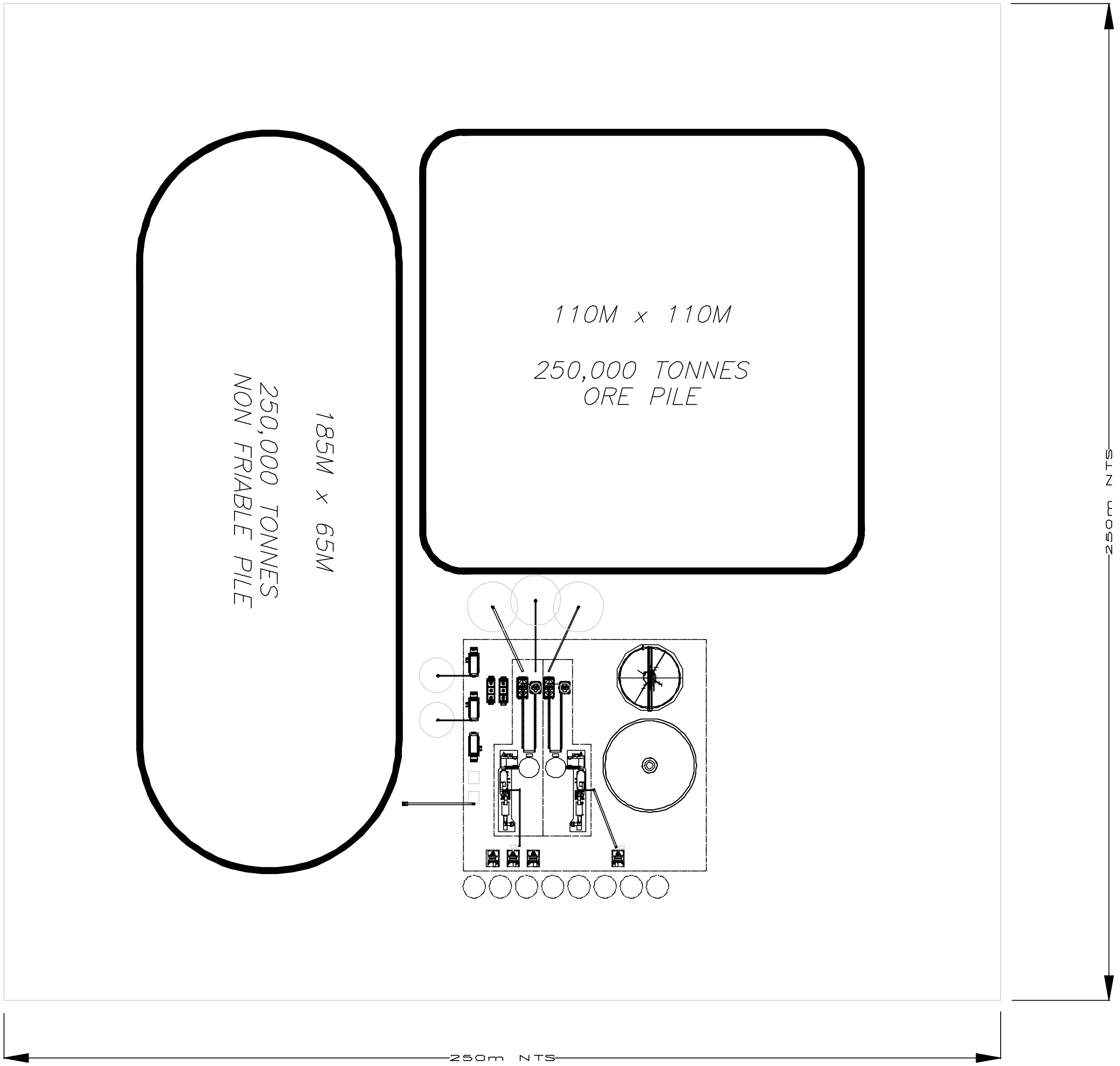
Outotec has been involved in the design and build of several fracturing sand plants. The estimate of bulk electrical and plant automation and control was based upon other similar frac sand plants designed by Outotec. Examples of P&ID diagrams for plants similar to the one envisioned for Minago are given in Outotec (2008).

2.10.4.3.3 Power and Energy Consumption

Based on the current design (Outotec Phase II design), the plant will have 4,145 connected horsepower or 3,091 kW and will operate 4,822 hours/year. Using these hours and the various capacities through the two sections (wet and dry) of the plant, the average electrical consumption will be 12.2 kWh/tonne with production of 1,142,805 tonnes annually assuming 75% of connected horsepower. This power consumption is in-line with typical frac sand plants with installed crushing.

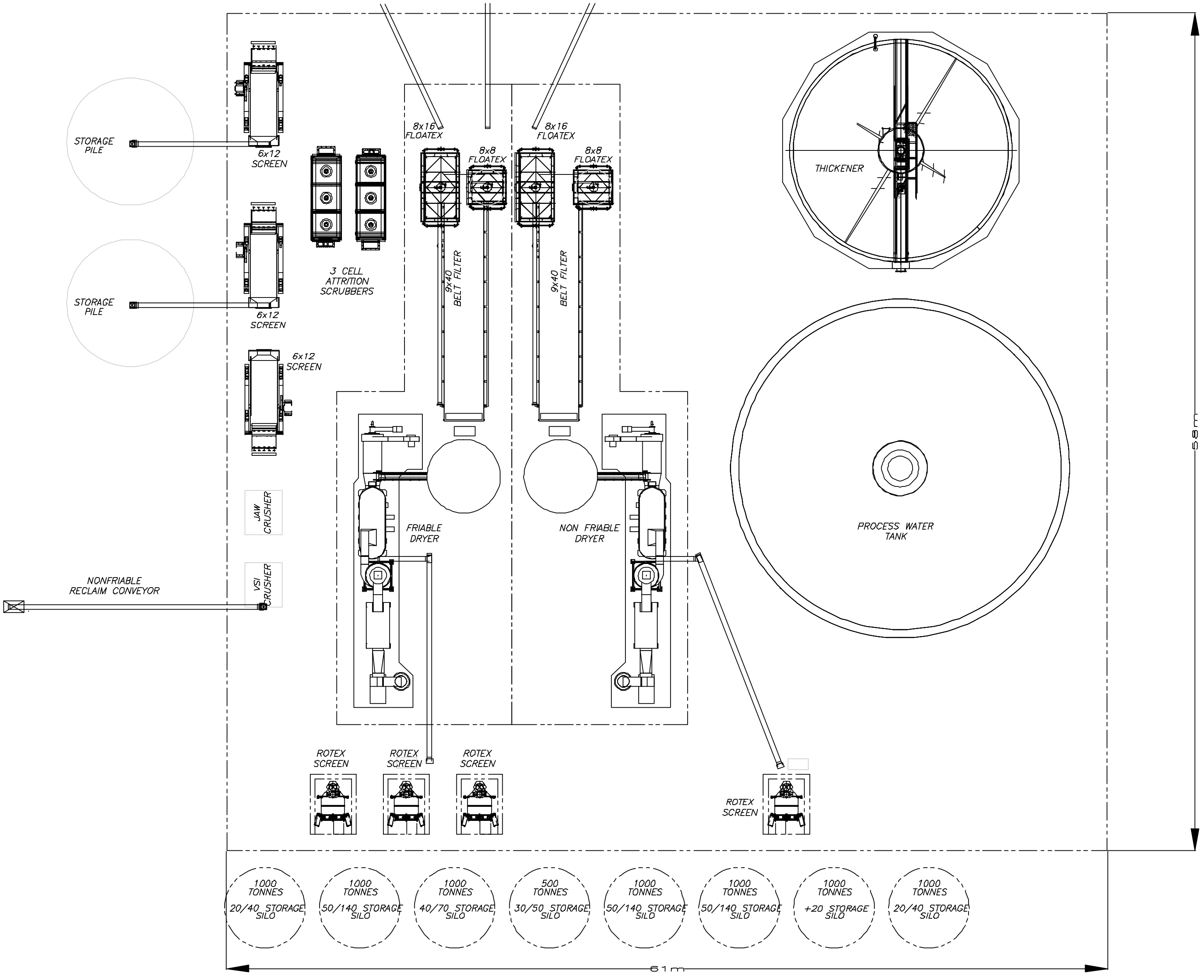
2.10.4.4 Rail Load-out Area

IM&M Consulting, Calgary, Canada designed the Rail Load-out site for the Frac Sand Plant, located at Ponton approximately 60 km from the proposed loading facility at the mine. The complete Rail Load-out design report is given a separate report, entitled 'IM&M Rail Load-Out Design' (IM&M Consulting, 2008). The loadout property will be built and serviced by OmniTrax Rail, the Company with a railhead at Ponton.



Source: Outotec, 2008

Figure 2.10-11 Conceptual Layout of the Frac Sand Plant



Source: Outotec, 2008

Figure 2.10-12 Conceptual Layout of the Frac Sand Plant (Zoomed in)

Table 2.10-5 List of Process Flow Diagrams for Minago's Frac Sand Plant

Drawing No.	Title	Description
WP-PFD-001 revP2	Area 01/Wet Plant	Screening and scrubbing
WP-PFD-002 revP2	Area 02/Wet Plant	Density separator circuit - Friable
WP-PFD-003 revP2	Area 03/Wet Plant	Crushing - Non Friable
WP-PFD-004 revP2	Area 04/Wet Plant	Density separator circuit - Non Friable
WP-PFD-005 revP2	Area 05/Wet Plant	Plant Thickener
DP-PFD-001 revP3	Area 06/Dry Plant	Drying and screening - Friable
DP-PFD-002 revP3	Area 07/Dry Plant	Screening and magnetic separation - Friable
DP-PFD-003 revP2	Area 08/Dry Plant	Drying and Screening - Non Friable
DP-PFD-004 revP3	Area 09/Dry Plant	Storage silos - Friable and Non Friable
DP-PFD-005 revP2	Area 09/Dry Plant	Plant Product load out

Source: Outotec, 2008

It is anticipated that a portion of the sand will be trucked from the mine to a frac sand transload facility, then transloaded into rail cars and shipped to market. Operationally, the rail load-out facility will require two switches per week, of 90 hopper cars each, with an average production of 1 car loaded every 50 minutes. Conceptual plans include 3 - 30 car storage tracks, a 1 - 10 car loading area, and a 30 car pre-loading storage area. Switching, within the facility, is expected to be by car mover. As such, road allowances and set offs will need to be provided to allow for car mover access.

The proposed Rail Load-out Facility will include two buildings (IM&M Consulting, 2008):

1) The first building will be a covered truck unloading building designed to accommodate a Super B tractor/trailer unit. This building will be 30.5 m long x 6 m wide and 6 m clear above the top of rail siding rail, with an 5.5 m high x 5 m wide truck pass opening at both ends.
2) The second building will be a railcar loading building that will contain an 18.2 m car. The building will be 30.5 m long x 12.2 m wide, and 6.1 m clear above the top of rail with an 6.1 m high x 6.1 m wide rail car pass opening at both ends. To accommodate the overhead rail loading equipment, an additional 9.1 m long x 3.7 m wide x 4.5 m high structure will be centered into the roof of the original building.

Building structures will be unheated but will protected workers from wind and precipitation. Other facility design features include the following (Wardrop, 2009b):

- The product will be protected from the elements and remain dry to within 1% moisture content.

- Transloading will be conducted using two modified Super B grain trailer loads per single rail car. The product will be scaled into the trucks at the dry plant; weigh scales for the transload site will not be required.
- The design product load for each rail car is 88 Mt, although the current track maximum is 79.4 Mt. The design product load for the Super B unit is 22 Mt.
- Super B grain trailers will bottom-dump into two under-floor unloading hoppers spaced at 8.5 m on-center. These 3 m x 3 m x 1.2 m hoppers will be contained within a concrete vault that allows for inspection of the tail pulley and conveyor load centering device on each of two 0.5 m conveyors.
- The conveyors will extend 37 m between the center lines of the two buildings. They will extract sand from the unloading hoppers and transfer it into rail cars. The covered conveyors will be 39.6 m long. Dust collection at the filling spouts will be discharged into rail cars.
- The rail cars will be constructed with two 13.7 m compartment-covered hoppers with 0.75 m hatches spaced at 3 m on-center. This design will allow for the use of 18.3 m cars.
- Dust collection at the filling spouts for the rail cars has been included in the design, but is not required for the truck receiving hoppers. The railcar loading building will require a minimum vertical clearance of 2.5 m over the cars for the main portion of the building, increasing to 11.5 m in the overhead loading section.
- Protection from falling will be provided within the railcar unloading building. A stair case will be required to a gantry located 4.5 m above top of rail, for the full length of the building. Workers will be allowed access to the top of the rail cars, within the environmental protection of the building, to open and close hatches. A drawbridge gangway will be required at the loading point, and 15.2 m (50') on center on both sides of the loading point. A continuous lanyard style fall protection system will run the full length of the structure.
- An electrically-heated operations building will be provided at the gantry elevation level near to the filling location. The operations building will be sized as a two-man warm-up area, and will contain the motor control panel for the conveyors, loading spouts, and dust collectors. The rail car filling area and the tops of cars in front and behind the filling area will be viewable from this building.